



Technical Report
Tocantinzinho Project
Brazil

Centered on Latitude 06° 03' S and Longitude 56° 18' W

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TABLE OF CONTENTS

SECTION • 1	SUMMARY.....	1-1
1.1	Summary.....	1-1
1.2	Property Description and Location.....	1-2
1.3	Accessibility, Climate, Local Resources, Infrastructure and Physiography.....	1-3
1.4	History	1-3
1.5	Geological Setting and Mineralization.....	1-4
1.6	Exploration	1-5
1.7	Drilling	1-5
1.8	Sample Preparation, Analyses and Security	1-6
1.9	Data Verification.....	1-6
1.10	Mineral Processing and Metallurgical Testing	1-6
1.11	Mineral Resource Estimates	1-7
1.12	Mineral Reserve Estimates	1-8
1.13	Mining Methods.....	1-8
1.14	Recovery Methods	1-9
1.15	Project Infrastructure.....	1-10
1.16	Market Studies and Contracts.....	1-11
1.17	Environmental Studies, Permitting and Social or Community Impact	1-11
1.18	Capital and Operating Costs.....	1-13
1.19	Economic Analysis	1-16
1.20	Adjacent Properties.....	1-17
1.21	Interpretation and Conclusions	1-18
1.22	Recommendations	1-19
SECTION • 2	INTRODUCTION.....	2-1
2.1	General	2-1
2.2	Report Authors	2-1
SECTION • 3	RELIANCE ON OTHER EXPERTS.....	3-1
SECTION • 4	PROPERTY DESCRIPTION AND LOCATION	4-1
4.1	Project Area and Location.....	4-1
4.2	Land Tenure and Mining Rights in Brazil.....	4-2
4.3	Royalties	4-4
4.4	Water Rights	4-5
4.5	Environmental Liabilities	4-5
4.6	Permitting.....	4-5
SECTION • 5	ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY	5-1
5.1	Site Topography.....	5-1
5.2	Site Access	5-1
5.3	Physiography and Climate	5-2
5.4	Infrastructure	5-8
SECTION • 6	HISTORY	6-1

SECTION • 7	GEOLOGICAL SETTING AND MINERALIZATION.....	7-1
7.1	Regional Geology.....	7-1
7.2	Local Geology	7-3
7.3	Mineralization	7-6
SECTION • 8	DEPOSIT TYPES.....	8-1
8.1	Tocantinzinho Deposit	8-1
8.2	Santa Patricia.....	8-2
SECTION • 9	EXPLORATION	9-1
9.1	Exploration History	9-1
9.2	Soil Geochemistry.....	9-2
9.3	Geophysics	9-5
SECTION • 10	DRILLING	10-1
10.1	Diamond Drilling.....	10-1
10.2	Reverse Circulation Drilling.....	10-7
10.3	Power Auger Drilling	10-7
10.4	Diamond Drilling Logging and Sampling.....	10-8
SECTION • 11	SAMPLE PREPARATION, ANALYSES AND SECURITY	11-1
11.1	Samples Supporting the Mineral Resource Estimate	11-1
11.2	Samples Supporting the Exploration Results	11-6
11.3	Adequacy Statement.....	11-6
SECTION • 12	DATA VERIFICATION	12-1
SECTION • 13	MINERAL PROCESSING AND METALLURGICAL TESTING	13-1
13.1	Introduction	13-1
13.2	Metallurgical Test Work	13-1
13.3	Composite and Sample Selection.....	13-2
13.4	Ore Composition	13-5
13.5	Mineralization	13-6
13.6	Comminution	13-7
13.7	Gravity Concentration	13-10
13.8	Flotation	13-12
13.9	Composition of Flotation Concentrate.....	13-23
13.10	Cyanide Leaching	13-23
13.11	Cyanide Destruction.....	13-32
13.12	Environmental Test Work.....	13-37
13.13	Thickener Test Work.....	13-38
13.14	Other Test Work Completed	13-41
13.15	Risks and Opportunities.....	13-41
SECTION • 14	MINERAL RESOURCE ESTIMATES.....	14-1
14.1	Geologic and Mineralization Models	14-1
14.2	Data Analysis and Estimation Domain.....	14-2
14.3	Evaluation of Extreme Grades	14-3
14.4	Variography.....	14-4
14.5	Model Setup	14-5
14.6	Estimation	14-5
14.7	Validation	14-6

	14.8	Mineral Resource Classification.....	14-7
	14.9	Mineral Resource Summary	14-8
SECTION • 15		MINERAL RESERVE ESTIMATES	15-1
	15.1	Mineral Reserves	15-1
	15.2	Pit Optimization	15-1
SECTION • 16		MINING METHODS	16-1
	16.1	Introduction	16-1
	16.2	Geotechnical and Hydrogeological analysis	16-1
	16.3	Mining.....	16-12
	16.4	Mine Operating Costs	16-32
SECTION • 17		RECOVERY METHODS	17-1
	17.1	Introduction	17-1
	17.2	Process Design Criteria	17-5
	17.3	Process Plant Description	17-6
	17.4	Reagents	17-14
	17.5	Plant Services	17-15
	17.6	Risks and Opportunities	17-17
SECTION • 18		PROJECT INFRASTRUCTURE	18-1
	18.1	Roads and Drainage	18-1
	18.2	Mine	18-2
	18.3	Process Plant	18-2
	18.4	Tailings disposal system	18-2
	18.5	Camp Accommodations	18-13
	18.6	Site infrastructure	18-13
	18.7	Ancillary Buildings	18-15
	18.8	Off-site Infrastructure	18-15
	18.9	Environmental	18-16
SECTION • 19		MARKET STUDIES AND CONTRACTS.....	19-1
	19.1	Market	19-1
	19.2	Contracts	19-1
SECTION • 20		ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT	20-1
	20.1	Environmental Studies	20-1
	20.2	Environmental Systems	20-4
	20.3	Permitting	20-5
	20.4	Social and Community Impact	20-7
SECTION • 21		CAPITAL AND OPERATING COSTS	21-1
	21.1	Capital Costs	21-1
	21.2	Operating Costs	21-3
SECTION • 22		ECONOMIC ANALYSIS	22-1
	22.1	Principle assumptions	22-1
	22.2	Cash Flow Forecasts	22-1
	22.3	Financial Analysis	22-3
	22.4	Taxes and Royalties.....	22-3
	22.5	Sensitivity Analysis.....	22-5

	22.6	Financial Projections	22-8
SECTION • 23		ADJACENT PROPERTIES	23-1
	23.1	Cabral Gold Inc.	23-2
	23.2	Serabi Gold Plc	23-2
	23.3	Gold Mining Inc.	23-2
	23.4	Anglo American plc	23-3
	23.5	Nexa Resources S.A.	23-3
SECTION • 24		OTHER RELEVANT DATA AND INFORMATION	24-1
SECTION • 25		INTERPRETATION AND CONCLUSIONS	25-1
	25.1	Summary	25-1
	25.1	Geology, Deposit, Exploration, Drilling, Sample Preparation and data verification	25-1
	25.2	Mineral Processing and Metallurgical Testing	25-1
	25.3	Mineral Resource Estimates	25-2
	25.4	Mineral Reserve Estimates	25-2
	25.5	Mining Methods	25-2
	25.6	Recovery Methods	25-2
	25.7	Project Infrastructure	25-3
	25.8	Market Studies and Contracts	25-3
	25.9	Environmental Studies, Permitting and Social or Community Impact	25-3
	25.10	Capital and Operating Costs	25-3
	25.11	Economic Analysis	25-3
SECTION • 26		RECOMMENDATIONS	26-1
	26.1	Exploration	26-1
	26.2	Mineral Processing and Metallurgical Testing	26-1
	26.3	Mineral Reserve Estimates	26-1
	26.4	Mining Methods	26-1
	26.5	Recovery Methods	26-2
	26.6	Project Infrastructure	26-2
	26.7	Environmental Studies, Permitting and Social or Community Impact	26-2
SECTION • 27		REFERENCES	27-1
SECTION • 28		CERTIFICATES OF AUTHORS AND DATE AND SIGNATURE PAGE	28-1
		Date and Signature Page	28-1

LIST OF FIGURES

Figure 1-1: Tocantinzinho Project Location	1-2
Figure 1-2: Tenement Area Expansion 2012 to 2018.....	1-5
Figure 1-3: Operating Cost Summary	1-16
Figure 1-4: Cash Flow Forecast.....	1-17
Figure 1-5: Adjacent Properties	1-18
Figure 4-1: Tocantinzinho Project Location	4-1
Figure 4-2: Tocantinzinho Project Mineral Rights	4-3
Figure 5-1: Characteristic Distribution of Monthly Temperature Range at Itaituba Station	5-5
Figure 5-2: Characteristic Distribution of Air Relative Humidity at Itaituba Station.....	5-5
Figure 5-3: Characteristic Distribution of Total Insolation at Itaituba Station.....	5-6
Figure 5-4: Characteristic Distribution of Precipitation and Evaporation at Itaituba Station	5-6
Figure 5-5: Wind Directions History	5-8
Figure 7-1: Tapajós Gold Province - Location and Regional Geology Map	7-1
Figure 7-2: Tocantinzinho Age in the Context of Regional Magmatism and Tectonism (modified from Santos et al., 2001).....	7-2
Figure 7-3: Tocantinzinho Geological Map	7-4
Figure 7-4: Tocantinzinho Deposit –Section 525 showing Mineralization being bound by a Structural Corridor and generally absent inside the Andesitic Bodies	7-6
Figure 7-5: Detail of Gold Grade Distribution inside Mineralized Granite. Section 450 looking SE	7-7
Figure 7-6: Tocantinzinho Deposit “Smoky” and “Salami” Granite	7-8
Figure 7-7: Quartz + Chlorite + Pyrite Sheeted Veins	7-9
Figure 7-8: Quartz Monzonite with well developed White Quartz-K-feldspar Stockwork cut by the main Mineralization Stage. Drill Hole TOC275, 340.5 m	7-10
Figure 9-1: Tenement Area Expansion 2012 to 2018.....	9-2
Figure 9-2: Location of Soil Sampling Campaigns by Eldorado	9-3
Figure 9-3: Gold Anomalies in Soil Sampling Campaigns by Eldorado.....	9-4
Figure 9-4: Santa Patricia coherent Copper and Molybdenum Soil Anomaly	9-4
Figure 9-5: KRB Gold Soil Anomaly.....	9-5
Figure 9-6: IP Geophysical Surveyed Lines.....	9-6
Figure 9-7: Reduced to the Pole Magnetic Survey	9-7
Figure 10-1: Tocantinzinho Deposit - Drill Plan	10-2
Figure 10-2: Santa Patricia Drill Plan with respect to the larger copper soil anomaly	10-4
Figure 10-3: KRB Drill Plan with respect to the larger gold soil anomaly	10-6
Figure 11-1: Tocantinzinho Blank Data – 2008 to 2010	11-2
Figure 11-2: Standard Reference Material Chart, SRM G907-2.....	11-3
Figure 11-3: Standard Reference Material Chart, G901-13.....	11-3
Figure 11-4: Standard Reference Material Chart, SRM Si42	11-4
Figure 11-5: Standard Reference Material Charts, G907-6.....	11-4
Figure 11-6: Relative Difference Plot of Tocantinzinho Field Duplicate Data, 2008 to 2010.....	11-5
Figure 11-7: Relative Difference Plot of Tocantinzinho Pulp Duplicate Data, 2008 to 2010	11-5
Figure 13-1: Drill Holes for Metallurgical Testing-Longitudinal View	13-4
Figure 13-2: Drill Holes for Metallurgical Testing-Cross Section	13-5
Figure 13-3: Sample Preparation Diagram for Comminution Test Work	13-7
Figure 13-4: Gold Recovery vs Concentrate Mass Pull for Reagent Variability Test Work.....	13-14
Figure 13-5: Gold Recovery vs Concentrate Mass Pull for Ore Variability Test Work	13-15
Figure 13-6: Gold Recovery vs Concentrate Mass Pull for Cleaner Flotation Test Work	13-18

Figure 13-7: Gold Recovery and Water Test Conditions	13-22
Figure 13-8: Leach Kinetics of Pilot Plant Concentrate Leach Test Work at Coarser Feed Size ($P_{80} = 125 \mu\text{m}$)	13-26
Figure 13-9: Leach Kinetics of Pilot Plant Concentrate Leach Test Work at Finer Grind Size ($P_{80} = 85 \mu\text{m}$)	13-26
Figure 13-10: Equilibrium Loading of Gold on Carbon	13-31
Figure 13-11: Equilibrium Loading of Silver on Carbon	13-31
Figure 13-12: Flocculant Test Work for Fine Tailings	13-38
Figure 13-13: Flocculant Test Work for Coarse Tailings	13-39
Figure 13-14: Flocculant Test Work for TOP Tailings	13-39
Figure 13-15: Flocculant Test Work for BOT Tailings	13-40
Figure 14-1: Main 3D Geological Models	14-1
Figure 14-2: Relationship between PACK or Mineralized Shell and Main 3D Geological Models	14-2
Figure 14-3: Cumulative Probability Plot for Gold Composites Samples inside the Estimation Domain	14-4
Figure 14-4: Example of Swath Plot (easting) comparing OK and NN Estimates	14-7
Figure 14-5: Mineral Resource Classification with respect to Supporting Drilling	14-8
Figure 15-1: Laser Topography and Surface	15-3
Figure 15-2: Topography Gridded Surface	15-4
Figure 15-3: Rock Type Section	15-4
Figure 15-4: Section 9330706 - Gold Distribution	15-5
Figure 15-5: Bench Plan 60 - Gold Distribution	15-5
Figure 15-6: Slope Design Sectors	15-7
Figure 15-7: Bench Plan 60 Lerchs Grossmann Pit Limits	15-8
Figure 15-8: Section 5782657 East Lerchs Grossmann Pit Limits	15-8
Figure 15-9: Pit Design	15-11
Figure 16-1: Locations of Oriented Geotechnical Drill Holes	16-2
Figure 16-2: Fracture Orientations Measured in all Geotechnical Drill Holes (3980 poles)	16-3
Figure 16-3: Geomechanical Model of Tocantinzinho	16-4
Figure 16-4: Example of Kinematic Failure Analysis (G-TOC 004 – Planar Failure Analysis)	16-5
Figure 16-5: Example of Kinematic Failure Analysis (G-TOC 004 – Wedge Failure Analysis)	16-5
Figure 16-6: Results of Overall Slope Failure Analysis ($FS=1.41$)	16-6
Figure 16-7: Results of Overall Slope Failure Analysis ($FS=1.33$)	16-7
Figure 16-8: Groundwater Contour Map	16-10
Figure 16-9: Section 9330606 North Pit Designs	16-14
Figure 16-10: Section 578265 East Pit Designs	16-14
Figure 16-11: Bench Plan 120 Pit Designs	16-15
Figure 16-12: Material Movement Summary	16-17
Figure 16-13: Mine Development Year -2	16-18
Figure 16-14: Mine Development Year -1	16-19
Figure 16-15: Mine Development Year 2	16-19
Figure 16-16: Mine Development Year 5	16-20
Figure 16-17: Mine Development Year 9	16-20
Figure 16-18: Waste Rock and Stockpile Disposal Layout	16-22
Figure 16-19: Waste Storage Facility Development Year -1	16-23
Figure 16-20: Inclined Berms	16-24
Figure 16-21: Section Line Locations - Stability Analyses	16-25
Figure 16-22: Blast Design Layout Ore	16-26
Figure 16-23: Blast Design Layout Waste	16-27

Figure 16-24: Blast Design Wall Control.....	16-27
Figure 16-25: Phase 1 Cycle Times by Elevation.....	16-29
Figure 16-26: Phase 2 Cycle Times by Elevation.....	16-30
Figure 17-1: Overall Flowsheet.....	17-3
Figure 17-2: Overall Layout for Processing Facility.....	17-4
Figure 18-1: Site Layout.....	18-1
Figure 18-2: Typical section for the final Flotation Tailings Dam.....	18-4
Figure 18-3: Scenario D Impacts.....	18-9
Figure 18-4: Typical Section of Pond # 1.....	18-10
Figure 21-1: Operating Cost Summary.....	21-4
Figure 21-2: Mining Summary.....	21-6
Figure 21-3: Process Summary.....	21-8
Figure 21-4: General and Administration Summary.....	21-11
Figure 22-1: Cash Flow Forecast.....	22-2
Figure 22-2: Gold Price Sensitivity.....	22-6
Figure 22-3: Exchange Rate Sensitivity.....	22-7
Figure 22-4: Sensitivity Analysis - Post-Tax NPV@5%.....	22-7
Figure 22-5: Sensitivity Analysis – Post-Tax IRR %.....	22-8
Figure 23-1: Adjacent Properties.....	23-1

LIST OF TABLES

Table 1-1: Tocantinzinho Mineral Resources as of September 30, 2018.....	1-8
Table 1-2: Tocantinzinho Mineral Reserve Estimate as of March 31, 2019	1-8
Table 1-3: Permits with Expiration Dates	1-12
Table 1-4: Capital Cost Summary by Area	1-14
Table 1-5: Sustaining Capital Cost Summary by Area	1-15
Table 1-6: Life of Mine Costs	1-15
Table 1-7: Financial Analysis	1-16
Table 1-8: Gold Price Sensitivity	1-17
Table 1-9: Exchange Rate Sensitivity	1-17
Table 2-1: Technical Report Matrix.....	2-1
Table 2-2: List of Qualified Persons	2-2
Table 2-3: Project Site Visits	2-3
Table 4-1 Tocantinzinho Project Mineral Rights	4-3
Table 5-1: Estimated Distance of Main Access – Roads.....	5-1
Table 5-2: Estimated Distance of Main Access - Air.....	5-2
Table 5-3: Itaituba Climate Normals Station Data	5-4
Table 9-1: Summary of Soil Sampling Campaigns by Eldorado.....	9-2
Table 10-1: Project Core Drilling Summary	10-1
Table 10-2: Santa Patricia Drill Location and Orientation.....	10-4
Table 10-3: Santa Patricia Significant Intercepts	10-5
Table 10-4: KRB Drill Location and Orientation	10-7
Table 10-5: KRB Significant Intercepts	10-7
Table 10-6: Project Power Auger Drilling Summary	10-8
Table 13-1: Description of Samples in Test Work Program.....	13-2
Table 13-2: Drill Hole and Composite Description	13-3
Table 13-3: Distribution of Ore	13-5
Table 13-4: Ore Composition	13-6
Table 13-5: Comminution Test Work Summary.....	13-8
Table 13-6: Design Criteria – Comminution Characteristics	13-9
Table 13-7: Gravity Test Work on Composites by Wardell Armstrong (2009).....	13-10
Table 13-8: Summary of Gravity Test Work Results	13-11
Table 13-9: Flotation Test Work Results from Ralph Meyertons (2007)	13-12
Table 13-10: Reagent Variability Flotation Test Work Summary.....	13-13
Table 13-11: Ore Variability Flotation Test Work Summary	13-15
Table 13-12: Rougher Flotation Test Work for the ALL (Global Composite) Ore Sample	13-17
Table 13-13: Cleaner Flotation Test Work for ALL (Global Composite) Ore Sample.....	13-18
Table 13-14: Design Criteria - Flotation Retention Time and Reagent Addition.....	13-19
Table 13-15: Locked Cycle Flotation Test Work Summary	13-20
Table 13-16: Pilot Plant Test Work Summary.....	13-21
Table 13-17: Design Criteria - Flotation Recovery.....	13-21
Table 13-18: Water Test Conditions and Results	13-22
Table 13-19: Design Criteria – Flotation Concentrate Grade	13-23
Table 13-20: Cyanide Leach on Flotation Concentrate Test Work (Batch and Pilot Test)	13-25
Table 13-21: SGS Leach Kinetics Test Work Summary.....	13-28
Table 13-22: Design Criteria – Cyanide Leaching	13-29
Table 13-23: Cycle Test Summary	13-30

Table 13-24: Cyanide Destruction Test Work Results by SGS (2013).....	13-33
Table 13-25: Aging Test Work by SGS (2013)	13-34
Table 13-26: Confirmatory Aging Test Work by SGS (2017).....	13-35
Table 13-27: Confirmatory Cyanide Destruction Test Work Results by SGS (2017)	13-36
Table 13-28: Cyanide Destruction Test Work Summary	13-37
Table 13-29: Design Criteria – Reagent Addition for Cyanide Destruction	13-37
Table 13-30: Thickener Settling Test Summary on Flotation Concentrate	13-40
Table 13-31: Thickener Settling Test Summary on Leached Residue	13-40
Table 13-32: Design Criteria – Thickener	13-41
Table 14-1: Descriptive Statistics for Samples inside the Estimation Domain – Au g/t data.....	14-3
Table 14-2: Au Variogram Parameters	14-4
Table 14-3: Azimuth and Dip Angles of Rotated Variogram Axes	14-5
Table 14-4: Tocantinzinho Mineral Resources as of September 30, 2018.....	14-9
Table 15-1: Block Model Extents	15-2
Table 15-2: Rock Type and Density used in Block Model	15-3
Table 15-3: Slope Design Criteria Applied for Pit Optimization	15-6
Table 15-4: Lerchs Grossmann Pit Shell Summary.....	15-9
Table 15-5: Tocantinzinho Mineral Reserve Estimate as of March 31, 2019	15-11
Table 16-1: Input Parameters for Slope Failure Analysis	16-6
Table 16-2: Golder Recommendations for Slope Design	16-8
Table 16-3: Production Schedule.....	16-16
Table 16-4: Stability Analyses Results.....	16-25
Table 16-5: Major Equipment Requirements by Year.....	16-31
Table 17-1: Key Process Design Criteria.....	17-5
Table 18-1: Flotation Tailings Dam features	18-3
Table 18-2: Geotechnical Parameters considered for the Starter Dam Stability Analysis	18-5
Table 18-3: Results of the Tailings Dam Stability Analyses	18-6
Table 18-4: Leaching Effluent Ponds features.....	18-10
Table 18-5: Geotechnical Parameters considered for Pond #1 Stability Analyses	18-12
Table 20-1: Permits with Expiration Dates.....	20-5
Table 21-1: Capital Cost Summary by Area	21-2
Table 21-2: Sustaining Capital Cost Summary by Area	21-3
Table 21-3: Life of Mine Operating Costs	21-4
Table 21-4: Manpower Operating Costs	21-5
Table 21-5: Mine Operating Unit Costs.....	21-7
Table 21-6: Process Plant Operating Cost	21-7
Table 21-7: Process Plant Power Costs	21-8
Table 21-8: Process Plant Reagents and Consumables Costs.....	21-9
Table 21-9: Process Plant Maintenance	21-10
Table 21-10: General and Administration	21-10
Table 22-1: Capital and Operating Costs Summary.....	22-2
Table 22-2: Financial Analysis	22-3
Table 22-3: Gold Price Sensitivity.....	22-5
Table 22-4: Exchange Rate Sensitivity	22-6
Table 22-5: Profit & Loss.....	22-9
Table 22-6: Cash Flow	22-10
Table 22-7: Income Taxes and Offsets.....	22-11

LIST OF ABBREVIATIONS

Abbreviation	Description or Unit
°	Degrees of longitude, latitude, compass bearing or gradient
%	Percent sign
°C	Degree Celsius
µm	micrometre
AACE	American Association of Cost Engineers
ADL	Analytical Detection Limit
Ag	Silver
ANFO	Ammonium Nitrate / Fuel Oil
ANM	National Mining Agency
ARD	Acid Rock Drainage
As	Arsenic
Ascii	(a standard digital data format)
asl	Above Sea Level
Au	Gold
Bi	bismuth
BRL	Brazilian Real
BTW	Drill core size (4.20 cm diameter)
BWI	Bond Ball Mill Work Index
Ca	Calcium
CIM	Canadian Institute of Mining
CIP	Carbon in Pulp
cm	Centimetre
CN	Cyanide
Cu	Copper
CV	Coefficient of variation
DCF	Discounted Cash flow
DNPM	National Department of Mineral Production
E	East
EIA	Environmental Impact Assessment
El.	Elevation
Fe	Iron
g	Grams
g/t	Grams per tonne
G&A	General and Administration
h	Hour
ha	Hectare
HDPE	High Density Polyethylene
HQ	Drill core size (6.3 cm diameter)
HRT	High Rate Thickener
ICP	Inductively Coupled Plasma
ICU	Intensive Oxidation Unit
IP	Induced Polarization
IRR	Internal Rate of Return
JV	Joint Venture

Abbreviation	Description or Unit
K	Potassium
kg	Kilogram
kg/t	Kilogram per tonne
km	Kilometre
Km/h	Kilometres per hour
kPa	Kilopascal
kt	Thousand tonnes
ktpd	Thousand tons per day
ktpy	Thousand tonnes per year
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per tonne
L	Litre
L/s	Litres per second
LOM	Life of Mine
m	Metre
M	Million
Ma	<i>Mega-annum</i> (10 ⁶ years)
mm	Millimetres
Mo	Molybdenum
Mt	Million tonnes
Mtpa	Million tonnes per annum
Mtpy	Million tonnes per year
MW	Megawatt
m ²	Square metre
m ³	Cubic metre
m ³ /h	Cubic metres per hour
m/h	Metres per hour
N	North
Na	Sodium
Nb	Number
NN	Nearest Neighbour
NPV	Net Present Value
NSR	Net Smelter Return
NTW	Drill core size (5.71 cm diameter)
OK	Ordinary Kriging
oz	Troy ounce
oz/t	Ounce per tonne
Pb	Lead
PCA	Environmental Control Plan
ppb	Parts per billion
ppm	Parts per million
QA/QC	Quality Assurance/Quality Control
RC	Reverse Circulation Drill Hole

Abbreviation	Description or Unit
RL	Relative level
ROM	Run Of Mine
RQD	Rock Quality Designation
S	South
s	Second
Sb	Antimony
SG	Specific gravity
SMC	Sag Mill Comminution Index
SMU	Selective Mining Unit
SRM	Standard Reference Material
st	Short ton
t	Ton, tonnes (metric)
t/m ³	Tonnes per cubic metres
t/h	Tonnes per hour
tph	Tonnes per hour
tpy	Tonnes per year
Te	Tellurium
U	Uranium
US\$	US dollars (American currency)
US\$/kWh	American dollars per kilowatt-hour
US\$/oz	American dollars per ounce
US\$/t	American dollars per tonne
UTM	Universal Transverse Mercator
W	Tungsten
W	Watt
WTP	Water Treatment Plant
X X	Coordinate (E-W)
y	Year
Y Y	Coordinate (N-S)
Z Z	Coordinate (depth or elevation)
Zn	Zinc

SECTION • 1 SUMMARY

1.1 SUMMARY

Eldorado Gold Corporation (Eldorado) is an international gold mining company based in Vancouver, British Columbia. Eldorado owns the Tocantinzinho gold Project (Project) through its wholly owned subsidiary Brazauro Recursos Minerais S/A (Brazauro). The Project is located in the State of Pará, northern Brazil.

The information and data included in this report were prepared in accordance with the requirements as defined in the National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects. The previous technical report on the Tocantinzinho Project was filed May 11, 2011.

The updated technical report of the Tocantinzinho Project was prepared using the results of the recent optimization studies carried out in 2018.

Information and data contained in, or used in the preparation of mineral resource and mineral reserve updates was obtained during drilling and sampling programs completed from 2004 to 2015 and analyzed by ALS Chemex Laboratories (ALS). Metallurgical data and process designs were based on various programs completed between 2004 and 2017, reviewed and validated by Hatch Ltd. The 2018 mine designs and mining methods, tailings management and water management were designed from first principles to a feasibility level.

When preparing reserves for any of its projects, Eldorado uses a consistent prevailing gold price methodology that is in line with the 2015 CIM Guidance on Commodity Pricing used in Resource and Reserve Estimation and Reporting. These are the lesser of the three-year moving average and the current spot price. A gold price of US\$1,200/oz Au was set for the current mineral reserve work. All cut-off grade determinations, mine designs and economic tests of economic extraction used this pricing for the Tocantinzinho Project and the mineral reserves work discussed in this technical report.

To demonstrate the potential economics of a Project, Eldorado elected to use metal pricing closer to the current prevailing spot price and then provide some sensitivity around this price. Metal prices used for this evaluation of the Project were US\$1,300/oz Au. This analysis provides a better snapshot of the project value at prevailing prices rather than limiting it to reserve prices that might vary somewhat from prevailing spot prices. Eldorado stresses that only material that satisfies the mineral reserve criteria is subjected to further economic assessments at varied metal pricing.

Qualified persons responsible for preparing this technical report as defined in National Instrument 43-101 (NI 43-101), Standards of Disclosure for Mineral Projects and in compliance with 43-101 F1 Technical Report, are David Sutherland, P.Eng., Rafael Gradim, P.Geo., Persio Rosario, P.Eng., John Nilsson, P.Eng. William McKenzie, P.Eng. and Paulo Franca, AusIMM

In compiling this technical report, Eldorado relied upon L&M Advisory of Sao Paulo, Brazil for the tax calculations and the financial analysis.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Project comprises an area of 68,803.6 ha and is located in the State of Pará in northern Brazil. It is located in the State of Pará in northern Brazil, in the Tapajós Gold province, approximately 200 km south-southwest of the city of Itaituba; 108 km from the district of Morais Almeida, and approximately 1,150 km in S60°W bearing from Belém, the capital of Pará State located along the north seacoast of Brazil, at the mouth of the Amazon River (Figure 1-1).

The Project's location can be found on the Vila Riozinho topographic map sheet (SB.21-Z-A, MIR 194; 1:250,000) at the central northern part. The site is situated at an elevation of 120 m above the sea level.

Approximate coordinates of the center of the Project area are as follows:

- Geographic: S= 06°03'; W= 56°18'
- UTM (Zone 21M): N= 9,330,700; E= 578,200

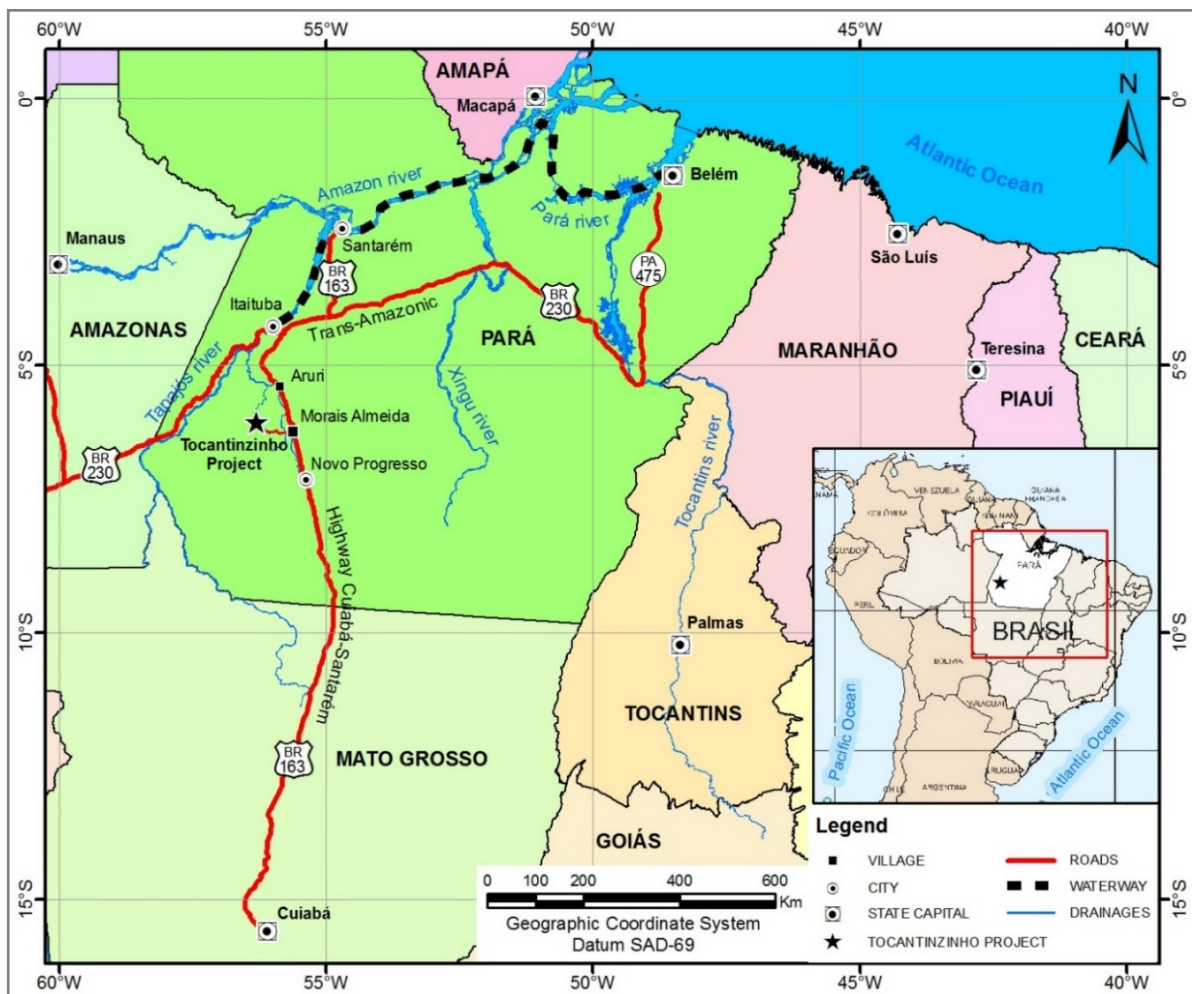


Figure 1-1: Tocantinzinho Project Location

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Project is located to the south of Itaituba city in the state of Pará. Itaituba city, the local centre for services and supplies. Itaituba is accessible from the Project by the Cuiabá-Santarém highway (BR-163). The BR-163 links Cuiaba city located in the south of the Project in the state of Mato Grosso to Itaituba and Santarém cities to the north. In itaituba the BR-163 highway connects with Trans-Amazonic highway (BR-230) which ends in the east in the port of Belem on the Atlantic Ocean shore.

Belem can also be reached by barge via Tapajós River-Amazon River waterway linking Itaituba and Santarém to the Atlantic Ocean.

A 72 km road connects Tocantinzinho site to the Transgarimpeira road. The Transgarimpeira road is accessed from Moraes de Almeida on the BR-163, 21 km east of the Jamanxim River barge crossing at Jardim do Ouro.

There is a 775 m long airstrip at the site.

The Project property is located in a region which is humid tropical forest, experiences significant rainfall during most months of the year and has a mean annual temperature of 28°C. The Project is part of the sub basin of the Jamanxim River that integrates with the Tapajos River basin. The area is hilly with areas that are rocky or saprolite.

1.4 HISTORY

At Tocantinzinho, the gold production was initiated in 1970 with intense garimpo activity in the mid-eighties to mid-nineties; however, there are no published records to support the timing and amount of production. In 1979 Mineração Aurífera Limitada obtained an exploration license with the Departamento Nacional de Produção Mineral (DNPM) over the Tocantinzinho Project area which expired in 1986. The property files were archived by the DNPM in 1992.

In 1997, Renison Goldfields (Australia) and Altoro formed a joint venture (JV) to explore Brazil for major gold deposits and Tocantinzinho was brought to the JV's attention by an air charter pilot. Management and operation of the JV were executed by Altoro. The project was acquired after a visit to the property by the company geologist who collected channel samples from different garimpeiros pits and returned with good results. In 1998 the JV with Renison Goldfields was terminated due to a corporate decision and as a consequence, all properties, projects and data acquired during the joint venture were passed to Altoro.

Altoro's exploration program was carried out from 1998 to early 2000 and consisted of soil geochemistry, ground magnetic survey, auger drilling and geological mapping. Solitario Resources Corporation acquired Altoro in 2000 and terminated the Tocantinzinho Project a year later due to the low gold price.

In 2003, Brazauro, through its Brazilian subsidiary Jaguar Resources do Brazil Ltda., acquired the properties covering the Tocantinzinho mineralization. Based on the results of geochemical soil

sampling, Brazauro initiated a drilling program that lasted until 2008 with the total of 25,635 m from 97 holes.

In July 2008, Eldorado reached an agreement with Brazauro which ensured that Tocantinzinho Project would be explored and developed in a timely manner through the access to Eldorado's exploration and project development expertise in Brazil. After Eldorado took over the Project, in September of the same year, the exploration works were continued in Tocantinzinho with a further 62 drill holes for 19,431 m, reverse circulation and auger drilling, soil geochemistry and geological mapping in surroundings.

In July, 2010 Eldorado completed the arrangement to acquire all the issued and outstanding securities that it did not already own of Brazauro.

In 2012, the Preliminary Environment License was approved and the majority of permits were obtained by 2017. Various positive studies were completed during this stage however the project was deferred pending permits, improved gold outlook and other company projects.

The two major permits required for construction have been granted;

- Para State Department of Environment and Sustainability granted the Installation License in April 2017 with the modified Installation License (LI no. 2771) approved August 9, 2017.
- National Department of Mineral Production (DNPM) (now ANM – National Mining Agency) issued the mining concessions (Ordinance no 87/SGM and 88/SGM) May 18, 2018.

1.5 GEOLOGICAL SETTING AND MINERALIZATION

The Tocantinzinho deposit is best classified as a granite-hosted, intrusion-related gold deposit. It is underlain by igneous rocks of older magmatic arcs of the Tapajós (Cuiú-Cuiú/Creporizão). Textural evidence and contact relationships suggest that the host granitic rocks at Tocantinzinho intruded as elongate bodies along a northwest-striking fault zone that cut through more regionally extensive quartz monzonites and granites. The granitoids were likely emplaced synchronous with faulting, and both intrusive contact and vein orientations suggest the host fault zone was active during this period as a sinistral, dominantly strike-slip feature. The presence of abundant aplites, miarolitic cavities, and blebby quartz textures implies that the host granitic intrusions represent late, volatile-rich components of the parent magma. Vein textures suggest that at least some of the veins, and possibly gold mineralization, were introduced during or just after solidification of the host rocks.

Mineralized granites at Tocantinzinho are visually divided into two sub-units by granite type, alteration mineralogy and colour: these are locally named smoky and salami. The smoky unit is a true granite with quartz, alkali feldspar and plagioclase whereas salami is an alkali feldspar granite comprising quartz, K-feldspar and albite. Contacts are diffuse and a complete gradation exists between the two units. The mineralized granites are intruded by an andesitic body outcropping along the axis of the deposit.

The Tocantinzinho deposit forms a sub-vertical, northwest-trending elongate body approximately 900 m long by 150-200 m wide. It has been drilled to approximately 450 m depth and remains open

below this depth. Within the mineralized granite, gold grades are remarkably consistent and are associated primarily with pyrite in sheeted veins and veinlets.

1.6 EXPLORATION

The exploration work at the Tocantinzinho Project completed to date includes geological mapping, channel and chip sampling, soil and stream sediment geochemical surveys, a detailed topography survey, auger drilling, geophysical investigations, limited reverse circulation drilling, and core drilling. Petrographic and metallurgical studies were conducted on drill core by contracted consulting firms.

A topographic aerial laser survey of the project site was carried out in September, 2010 by Geoid Ltda. A total area of 53 km² was surveyed including the deposit, possible tailings dam areas and the future plant site. The contour interval was 1 m with an accuracy of approximately 0.15 cm in both the horizontal and vertical coordinates.

In late 2010, Eldorado completed an induced polarization (IP) geophysical survey of 45 km line, covering the areas along the Tocantinzinho trend to the northwest and southeast of the deposit. In 2011, aerial and ground magnetic survey data collected in 2005 were re-interpreted.

Eldorado expanded the tenement from 33,979 ha to 68,803.6 ha between 2012 and 2018 (Figure 1-2). Black outlines represent exploration claims. Red outlines represent mining concession. Soil samples were collected at 50 m intervals, using a hand auger with half-metre depth. Line spacing varied by area and campaign. Soil sampling covers approximately 40% of the total exploration package and has been an effective method to delineate zones of gold and copper anomalies and the primary tool to generate drill targets.

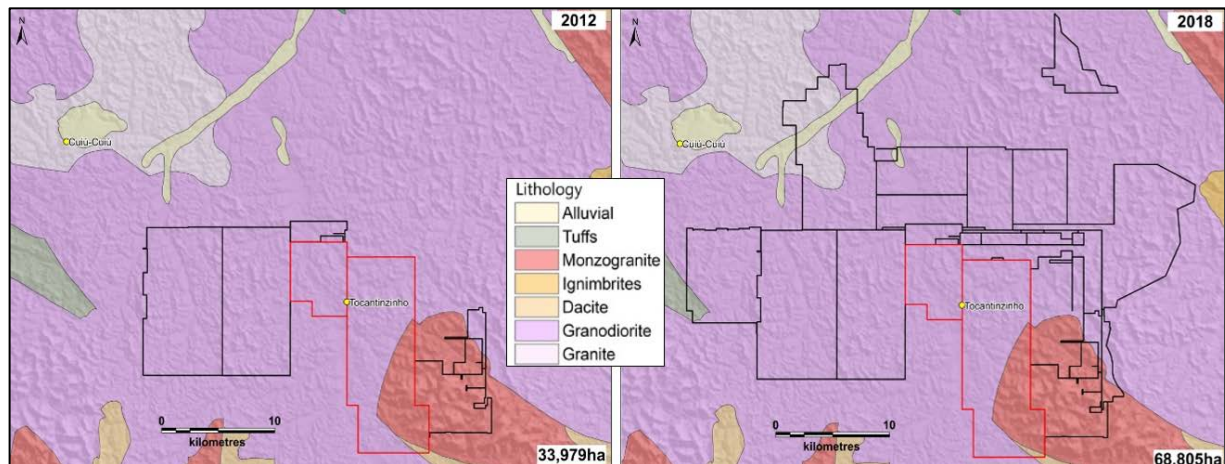


Figure 1-2: Tenement Area Expansion 2012 to 2018

1.7 DRILLING

Diamond drill holes are the principal source of geological and grade data for the Tocantinzinho Project. A total of 82,805 diamond drill metres in 296 drill holes were completed inside the broader tenement in various phases between 2004 and 2015. The mineral resource estimate for the

Tocantinzinho Project is directly supported by 45,039 m drilled between 2004 and 2010. Metallurgical drilling was carried out in 2009 (1,490 m), and drilling for geotechnical purposes was executed by Eldorado in 2010 (1,785 m). Exploration drilling inside the broader package amounts to 34,492 m drilled between 2004 and 2015.

1.8 SAMPLE PREPARATION, ANALYSES AND SECURITY

Diamond drill holes are the principal source of geological and grade data for the Tocantinzinho Project. Drilling at Tocantinzinho has been carried out in multiple phases from 2004 to 2015. In total, 80,805 m in 296 diamond drill holes have been completed at the Tocantinzinho Project, of which 45,039 m were drilled in and immediately around the Tocantinzinho deposit.

All diamond drilling in Tocantinzinho was done with wire line core rigs and mostly of HQ size. The entire lengths of the diamond drill holes were sampled with sample intervals ranging from 0.5 to 2.0 m, usually at two metre long intervals. Core was logged and samples handled in accordance with industry best practices. Sample analyses have been performed by SGS Geosol, ALS Chemex, and ACME laboratories at different points in the projects history.

Sample batches were arranged to contain regularly inserted control samples. A standard reference material (SRM), a duplicate and a blank sample were inserted into the sample stream at every 10th to 15th sample. The duplicates are used to monitor precision, the blank sample can indicate sample contamination or sample mix-ups, and the SRM is used to monitor accuracy of the assay results. Monitoring of the quality control samples showed all data were within control throughout the preparation and analytical processes. In Eldorado's opinion, the QA/QC results demonstrate that the Tocantinzinho gold Project assay database is sufficiently accurate and precise for resource estimation and disclosure of exploration results.

1.9 DATA VERIFICATION

Checks to the entire drill hole database were undertaken. Comparisons of the digital database were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado concluded that the data supporting the Tocantinzinho Project resource work is sufficiently free of error to be adequate for estimation and disclosure of exploration results.

1.10 MINERAL PROCESSING AND METALLURGICAL TESTING

The metallurgical test work completed for the Tocantinzinho Project included the following tests:

- Ore variability in terms of lithology, gold head grade, sulfur head grade, depth, and sample blending
- Metallurgical test work for primary sulfide ore, gold bearing soil, saprolite, transitional and artisanal mining (garimpeiros) tailings
- Detailed chemical analyses of ore feeds, flotation concentrates and flotation tailings
- Ore mineralogy and characteristics assessment

- Comminution testing including Bond crushing, rod milling, and ball milling indices; SMC index, and abrasion index
- Whole ore cyanide leach and cyanide leach of flotation concentrates
- Flotation including batch rougher and cleaner, locked cycle, and pilot plant
- Gravity recoverable gold
- Thickening testing of ore feed, flotation concentrate, leached residue and flotation tailing
- Cyanide detoxification (several methods) and aging test work on tailings and effluent
- Environmental and geotechnical testing of residue

The pilot plant flotation and cyanide test work were completed by Wardell Armstrong International in UK. The gravity test work completed by FLS Knelson in Canada and the tailings and cyanide destruction test work was latest carried out by SGS Mineral Services in Canada.

The Tocantinzinho mill feed during the life of mine (LOM) plan will come from three main sources. Granite representing 93% of the mine, saprolite representing 4% of the mine, and the garimpeiros tailings that were produced from previous artisanal mining, representing the remaining 3%.

The average annual plant head grade is 1.43 g/t Au for granitic ore, 1.21 g/t Au for saprolite, and 1.03 g/t Au for garimpeiros tailings. The combined average annual plant feed grade is 1.41 g/t Au with a maximum peak of 1.75 g/t Au in Year 4.

1.11 MINERAL RESOURCE ESTIMATES

The mineral resource estimate is supported by 3D geological and mineralization models. To constrain gold grade interpolation for the Tocantinzinho deposit, Eldorado created 3D mineralized envelopes, or shells. These were based on initial outlines derived by a method of Probability Assisted Constrained Kriging (PACK). The threshold value of 0.30 g/t Au was determined by inspection of histograms and probability curves as well as indicator variography. The mineralized shell or domain was constrained by a corridor of mineralized granites and an internal barren andesitic intrusion.

A hard cap of 25.0 g/t Au was applied to the assay data to mitigate estimation risk associated with extreme grades. Capping reduces the global gold mean grade by 3.9%. The assays were then composited into 2.0 m fixed-length down-hole composites.

The block size used for the Tocantinzinho model is 10 m east x 10 m north x 10 m high. Modelling consisted of grade interpolation by ordinary kriging (OK) inside the mineralized shell. The search ellipsoids were oriented preferentially to the orientation of the mineralized shell for within shell runs. The model was validated by visual inspection, checks for bias and for appropriate grade smoothing.

Bulk density data were assigned by rock type. The ore hosting granite value equalled 2.62 t/m³ in the primary region and 1.80 t/m³ in the weathered portion (saprolite).

The mineral resources of the Tocantinzinho deposit were classified using logic consistent with the CIM definitions referred to in NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified into measured, indicated, and inferred mineral resource categories.

The Tocantinzinho mineral resources as of September 30, 2018 are shown in Table 1-1. The mineral resource is reported at a 0.3 g/t Au cutoff grade.

Table 1-1: Tocantinzinho Mineral Resources as of September 30, 2018

Mineral Resource Category	Resource t x 1,000	Grade Au g/t	Contained Au oz x 1,000
Measured	17,530	1.51	851
Indicated	31,202	1.26	1,264
Measured & Indicated	48,732	1.35	2,115
Inferred	2,395	0.90	69

1.12 MINERAL RESERVE ESTIMATES

Mineral reserves were calculated in accordance with CIM standards using the mineral resource block model within an engineered pit design. The pit design was based on an optimized pit shell using a US\$1,200/oz gold price. Blocks above a 0.365 g/t cut-off grade are considered mineral reserves. Those mineral resource blocks with a measured resource class were converted into proven reserves, while the indicated resource blocks were converted into probable reserves. Mineral resource blocks classed as inferred were treated as waste. No additional modifying factors were used in the reserve estimate. Table 1-2 presents the mineral reserve estimate for the Tocantinzinho Project as of March 31, 2019.

Table 1-2: Tocantinzinho Mineral Reserve Estimate as of March 31, 2019

Category	Tonnes t x 1,000	Grade Au g/t	Contained Au oz x 1,000
Proven	17,007	1.52	834
Probable	21,898	1.35	949
Proven and Probable	38,906	1.42	1,779

1.13 MINING METHODS

The Tocantinzinho mine will be developed as a conventional open pit using excavators and trucks with wheel loader support. Near surface saprolite material and old tailings will be mined using a fleet of articulated 40 t capacity trucks and 6 m³ excavators in pioneering areas. Some saprolite and fresh granite, quartz monzonite and andesite will be drilled, blasted and loaded by 17 m³ excavators and an 11.6 m³ capacity wheel loader into 90 t rigid frame offroad haulage trucks.

The mine will be developed in two phases. Pre-stripping will be undertaken in Phase 1 over a two year period. Road and drainage development will be undertaken initially as well as stripping of a borrow pit which will provide fresh rock for construction and site preparation. Stockpile and waste dump preparation will take place west of the open pit area. A total of 22.7 Mt will be excavated in the mine during the pre-production period. The mine will deliver 4.336 Mt of ore annually to the processing facility. Peak production years will be Year 1 and Year 2 with a total mining rate of

26.7 Mt. The open pit will operate to Year 9 of the plan. Stockpile recovery and processing will continue into Year 10.

1.14 RECOVERY METHODS

The process selected for the Tocantinzinho Project is a flotation concentrate cyanide leach and carbon adsorption flowsheet comprising crushing, grinding, gravity concentration, flotation, cyanide leaching, carbon adsorption, cyanide detoxification, carbon elution and regeneration, gold refining, and tailings disposal.

The Tocantinzinho process plant will process run of mine (ROM) granite ore, along with minor amounts of saprolite and garimpeiros tailings, and produce gold doré bars and tailings.

The mill is designed with a nominal capacity of 4.3 MTPA at a planned average feed grade of 1.41 g/t Au, producing 174,000 oz of Au annually with a life of mine of 9 years. The plant is designed to treat ore with a maximum head grade of 1.76 g/t Au.

The plant will consist of the following unit operations:

- **Primary crushing** – A vibrating grizzly and jaw crusher in open circuit producing a final product of 80% passing (P80) 148 mm
- **Coarse ore stockpile and reclaim** – A 12 h live storage crushed ore stockpile with two reclaim apron feeders feeding the SAG Mill feed conveyor
- **Grinding** – A SAG mill equipped with a water-jet system for pebble recirculation producing a transfer product P80 of 1000µm to secondary grinding with a ball mill in closed circuit with hydrocyclones producing a final product P80 of 125µm
- **Gravity concentration** – Gravity concentration of hydrocyclone underflow from the secondary grinding circuit to produce a gold-rich concentrate for intensive leaching
- **Intensive cyanidation** – Gravity gold dissolution within the intensive cyanidation reactor for subsequent gold recovery in electrowinning
- **Sulphide flotation** - 2-stage flotation circuit to produce sulphide concentrate for cyanide leaching
- **Pre-aeration, cyanide leaching, and carbon adsorption** – Pre-aeration of feed followed by gold leaching by cyanidation, facilitated by oxygen, followed by adsorption of solution gold onto carbon particles via a Carbon in Pulp (CIP) carousel pump-cell configuration
- **Cyanide detoxification** – Detoxification of cyanide slurry via sodium metabisulphite for SO₂, oxygen and copper sulphate to achieve < 0.2 ppm for CN_{TOT} (total) and for CN_{WAD} (weak acid dissociable)
- **Carbon elution and regeneration** – Acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold rich solution, and thermal regeneration of carbon to remove organic foulants
- **Gold refining** – Gold electrowinning (sludge production), filtration, drying, and smelting to produce gold doré
- **Tailings** – Flotation tailings and concentrate cyanidation tailings (i.e. CIP tailings) are stored in separate tailings storage facilities

- **Water treatment plant (WTP) Future** – Excess water from the CIP tailings is treated for the removal of metals in solution (i.e. Cu) prior to being released to the environment

1.15 PROJECT INFRASTRUCTURE

1.15.1 Infrastructure

The Infrastructure on site will include:

- Roads and drainage
- Open pit, waste rock dumps, stockpiles for high grade saprolite and garimpeiro tailings, pit dewatering
- Secured explosives facility with emulsion plant and storage
- Process plant complete with reagent storage buildings and process control
- Camp accommodations in the existing geology camp and the permanent 500 man camp
- Electrical power distribution from the site substation
- Fresh/fire water system fed from Veados Creek
- Sewage treatment and oil/water separation
- Landfill facility
- Existing airstrip
- Communications – radio and telecom
- Mobile equipment fleet
- Maintenance facility housing truck shop, warehouse, dry and offices
- Truck wash building
- Fuel storage and station
- Assay laboratory
- Gatehouse and site security
- Site access road
- 200 km long 138 kV power line from Novo Progresso and related switching station
- Greenhouse and nursery
- Deforestation storage for reclamation

1.15.2 Tailings

The Tocantinzinho Project tailings disposal system will consist of a flotation tailings dam and two leaching effluent ponds. The dam will have two purposes, tailings disposal and water catchment. These designs were elaborated according to the criteria established by Brazilian Standards NBR 13.028: 2006 and NBR 10.157: 1987, whenever applicable). The design also complies with the most recent version of the Brazilian Standards for tailings dam design (NBR 13.028 - ABNT, 2017). The tailings classification followed the criteria specified by Brazilian Standard NBR 10.004: 2004.

The flotation tailings dam will be built in two phases, a starter dike and one downstream raising. The reservoir capacity for the starter dam is 7.68 Mm³, which will meet the first three years of mine production. The final dam can hold 29.8 Mm³, comprising eleven years of useful life, meeting the

demand for the entire mine predicted operation. The starter dam will be built with compacted soil from borrow areas within the project site. The final dam will be built with compacted rockfill from the pit, having an upstream impervious face of compacted clay.

The tailings from the detox plant will be delivered to two ponds, which will be excavated in soil down to its design defined levels and the excavated material used for building compacted perimeter dikes. Each pond is expected to address 5 years of mine operation. Ponds will be lined with a layer of HDPE geomembrane, having a leakage detection system underneath. Ponds are designed not to overflow, with sufficient free board. Supernatant water will be removed by floating pumps.

Extensive investigation was carried out to determine characteristics of the foundation and construction material for the flotation tailings dam and leaching ponds. Laboratory tests in disturbed and undisturbed samples were carried out to determine the strength and permeability parameters. Stability and seepage analyses resulted in adequate factors of safety.

1.16 MARKET STUDIES AND CONTRACTS

There has been no formal market study completed for the Project. Gold will be produced as doré and sold to final refiners.

1.17 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Environmental baseline studies have been completed on the Project site and preparations are underway to complete the studies on the powerline corridor. The fieldwork including forest and fauna studies, hydrology and hydrogeology monitoring, geochemistry analysis and geotechnical analysis are complete. The archeological surveys have also been conducted.

The following studies have been completed:

- Flora and fauna
- Hydrology and hydrogeology monitoring
- Air quality monitoring
- Geochemistry and geotechnical analysis
- Archaeological

The closure plan was established to identify environmental, social and economical risks after production ceases and to determine measures to be implemented during construction, operation and closure. It will be continuously updated and implemented prior to the shutdown of the operation.

During the Project construction, deforestation materials and topsoil will be stored in various locations on the site to be used for reclamation. A progressive rehabilitation approach will be used to reduce the long-term closure liability. Actively rehabilitating areas during the operational stage will provide the opportunity to develop and test the most effective methodologies. Area drainage will be modified as required prior to the reclamation process.

A seedling nursery will be built on site to grow native plants that will eventually be planted in reclamation areas.

The environmental licensing process for Tocantinzinho Project is complete. The Environmental Impact Assessment (EIA) was submitted to the Environment Department of Para State (SEMA-PA) in January 2012 and was approved in September 2012, with the granting of the Preliminary Environmental License, which includes two main structures: the site, including activities related to mining and ore processing and the access road to the Project.

In January 2016, an installation license for the mining project was requested, which was granted in April 2017 and modifications granted in August 2017. Table 1-3 summarizes the permits status.

Table 1-3: Permits with Expiration Dates

Permit	Related License	Number	Expiration Date	Comments
Tocantinzinho Site	Installation License	2771/2017	2020-04-18	
	Permit for deforestation	3383/2017	2020-04-18	
	Permit to capture, collection, rescue, transportation and release of Wildlife	3384/2017	2018-04-18	A new permit was requested to environmental agency
	Permit to Wildlife Monitoring	3381/2017	2020-04-19	
Tailings Dam and CIP Pond	Installation License	2796/2017	2020-11-20	
	Preliminary Water Permit	710/2016	2018-10-07	
	Final Water Permit	3103/2018	2023-02-01	
Fuel Station	Installation License	2816/2018	2021-01-09	
Concrete Batch Plant	Installation License	2830/2018	2021-04-11	
Effluent Release	Final Water Permit	Process 40870/2017	-	
	Preliminary Water Permit	740/2017	2019-01-09	An extension in validity was requested to environmental agency
Supply wells	Preliminary Water Permit	655/2016	2018-04-29	
	Preliminary Water Permit	877/2018	2020-01-29	
	Final Water Permit	2575/2016	2020-07-25	
Crossings (drainage)	Final Water Permit	2772/2017	2022-02-03	
Supply Industrial water	Preliminary Water Permit	693/2016	2018-09-08	A new permit was requested to environmental agency
Pit Mine Dewatering	Final Water Permit	2481/2016	2020-05-03	

Permit	Related License	Number	Expiration Date	Comments
Transmission Line	Preliminary License	1692/2017	2018-12-28	
	Installation License	2797/2017	2020-12-27	
	Permit for deforestation	3642/2017	2018-12-28	A new permit was requested to environmental agency.
	Permit to capture, collection, rescue, transportation and release of Wildlife	3643/2017	2018-04-18	A new permit was requested to environmental agency
	Permit to Wildlife Monitoring	3644/2017	2018-04-18	A new permit was requested to environmental agency
Access Road	Permit to capture, collection, rescue, transportation and release of Wildlife	9181/2017		Pending
	Permit for Deforestation	39318/2016		Pending
	Installation License	2862/2018	2020-08-19	
	Final Water Permit	2764/2017	2022-03-02	

The pending licenses include:

- Deforestation permit for the new works on the access road - Process n° 2016/39318
- Authorization for capture, collection, rescue, transportation and release of wildlife for access road - Process n° 2017/9181

1.18 CAPITAL AND OPERATING COSTS

During the second quarter of 2017, basic engineering was completed by Eldorado and Ausenco which produced a level 3 capital cost estimate as defined by the American Association of Cost Engineers (AACE) Estimate Classification.

In the second quarter of 2018, a Project optimization exercise was undertaken to investigate cost savings to streamline the process flow sheet, consolidate the site infrastructure into one central location, optimize the facilities for a nine year mine life and update the costs. During the optimization process, the estimate was updated with final design quantities and contractor proposals for the earthworks, deforestation and the high voltage overhead power line.

In 2019, the estimate was updated with new cost information for selected items, particularly the SAG mill, mine equipment and mine preproduction costs; escalation and exchange rates. The 2019 review also added additional costs for electrical supply compensation and changes to selected material takeoff quantities. The capital cost is estimated at US\$441.8 M, costs in the estimate are as of Q1 2019.

A summary of the capital cost in US dollars is shown in Table 1-4.

Table 1-4: Capital Cost Summary by Area

WBS	Description	Total US\$ M	% of Total
A	Overall Site	15.13	4
B	Mine	117.20	31
C	Crushing	9.04	2
D	Process Plant	59.19	16
E	Tailings	9.77	3
F	CIP Ponds	6.31	2
G	Camp	3.05	1
H	Infrastructure	24.35	6
J	Ancillary Facilities	7.87	2
K	Off Site Infrastructure	35.55	9
N	Geology	1.63	0
P	Environmental	0.24	0
Total Direct Cost		289.31	76
Q	Indirects	55.26	15
R	Capital Spares	7.30	2
S	EPCM	17.62	5
T	Owner's Cost	9.80	3
Total Indirect Cost		89.97	24
Total Project Cost		379.28	100
U	Contingency	62.50	16
Total Project /w Contingency		441.78	116

The level of accuracy for an AACE Class 3 estimate is -10% to -20% on the low side and +10% to +30% on the high side with a contingency level for a 50% probability of an overrun or underrun in the range of 5% to 15%.

The current capital cost estimate has an estimated accuracy of – 11% to +21% of the installed cost (before taxes and contingency). Contingency is 16.5% of Project cost before contingency or 14% of total Project cost including contingency. Therefore, the capital cost estimate is within the accuracy level of a Class 3 estimate.

Sustaining capital cost for the Project largely includes mining equipment additions, replacements and rebuilds; tailings dam expansion; an additional CIP pond; access and site road maintenance; construction indirects for the projects; additional spares; and allowances for ongoing projects.

The sustaining capital cost is estimated at US\$151.2 M, a summary of the sustaining capital cost in US dollars is shown in Table 1-5.

Table 1-5: Sustaining Capital Cost Summary by Area

WBS	Sustaining Capital	Total US\$ M	% of Total
A	Land Preparations and Aggregate	7.9	5%
B	Mining	50.8	34%
D	Process Plant	2.4	2%
E	Tailings Pond	25.3	17%
F	CIP Pond	4.6	3%
J	Ancillary Facilities	1.6	1%
K	Access and Site Roads	15.7	10%
Q	Construction Indirects	14.7	10%
R	Spares	4.2	3%
T	Owners Costs	1.4	1%
	Non Recoverable Taxes	22.7	15%
	Total Sustaining Costs	151.2	100%

The life of mine overall operation cost for the project is US\$23.41 per tonne of ore processed. As shown in Table 1-6 and Figure 1-3, the operating cost includes the mine, process plant, and general and administration (G&A). All costs are as of Q1 2019. Mining costs include development in Year - 2 and Year -1.

Table 1-6: Life of Mine Costs

Area	Type of Cost	Unit Cost US\$/t Processed	Total Cost US\$ M/y
Mining	Manpower	2.14	9.28
	Diesel	2.61	11.31
	Consumables	6.63	28.75
	Other	0.05	0.23
	Total Mining costs	11.43	49.57
Processing	Manpower	1.17	5.06
	Power	2.14	9.26
	Consumables	5.21	22.59
	Maintenance	0.50	2.17
	Total Processing Costs	9.02	39.09
G&A	Manpower	1.20	5.19
	Camp	0.79	3.42
	Other	0.98	4.23
	Total G&A costs	2.96	12.83
Total	Total costs	23.41	101.49

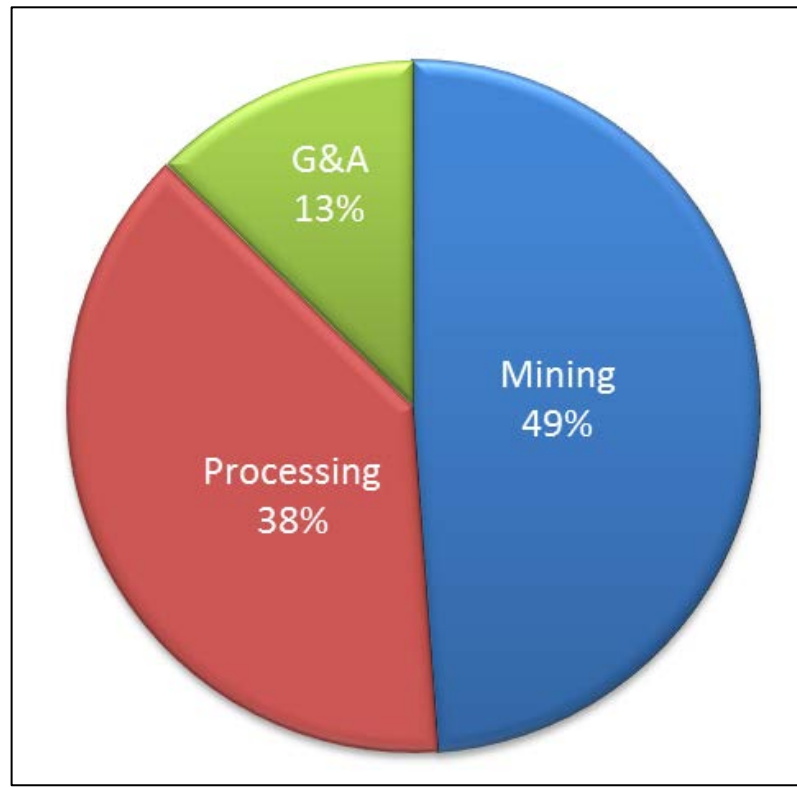


Figure 1-3: Operating Cost Summary

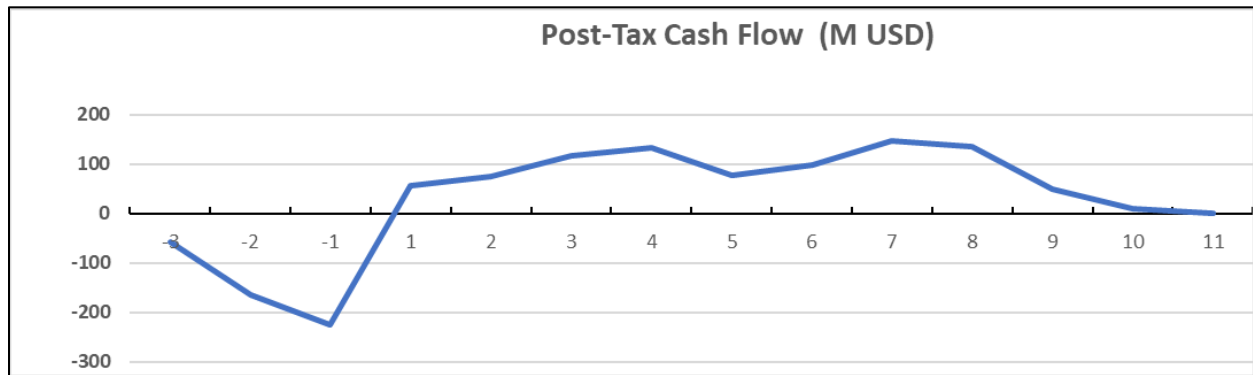
1.19 ECONOMIC ANALYSIS

The financial analysis at a gold price of US\$1,300 per ounce yields an NPV of US\$216.3 M at a 5% discount rate and an IRR of 13.4% post tax, the results are summarized in Table 1-7.

Table 1-7: Financial Analysis

Financial Analysis	Unit	Post-Tax	Pre-Tax
Project NPV@5%	US\$ M	216.30	255.80
Total Cash Flow (NPV@0%)	US\$ M	451.70	511.60
Internal Rate of Return	%	13.4	14.5
EBITDA (annual average)	US\$ M	109.60	109.60
Payback	Years	4.4	4.3

Cash flow forecasts are summarized in Figure 1-4.

**Figure 1-4: Cash Flow Forecast**

Taxes are included in the economic model and includes payments and rebates during operation. Royalties are to be paid to the federal government and private. The economic model included a buydown of the private royalty for a corresponding lower royalty over the life of mine.

The Table 1-8 and Table 1-9 summarizes the sensitivities to gold price and exchange rate.

Table 1-8: Gold Price Sensitivity

Gold Price Cash Flow Analysis		Gold Price US\$/oz				
		1,100	1,200	1,300	1,400	1,500
Post-tax NPV@5%	US\$ M	20.00	119.60	216.30	312.60	408.70
IRR	%	5.8	9.8	13.4	16.6	19.7
EBITDA	US\$ M	78.10	93.80	109.60	125.30	141.00
Payback	Years	6.4	5.4	4.4	3.8	3.4

Table 1-9: Exchange Rate Sensitivity

Exchange Rate Cash Flow Analysis		Exchange rate (BRL/US\$)				
		3.50	3.75	4.00	4.25	4.50
Post-tax NPV@5%	US\$ M	124.40	173.50	216.30	254.00	287.50
IRR	%	9.6	11.6	13.4	15.0	16.5
EBITDA	US\$ M	101.70	105.90	109.60	112.80	115.60
Payback	Years	5.4	4.9	4.4	4.0	3.8

1.20 ADJACENT PROPERTIES

The Tocantinzinho deposit is located in the Tapajós Gold Province and there are a number of gold-focused international exploration and mining companies within a 100 km radius from the deposit. Additionally, the interest in the exploration potential for copper mineralization has recently increased in the region as evidenced by the staking of large exploration claims by base metal mining companies as shown in Figure 1-5.

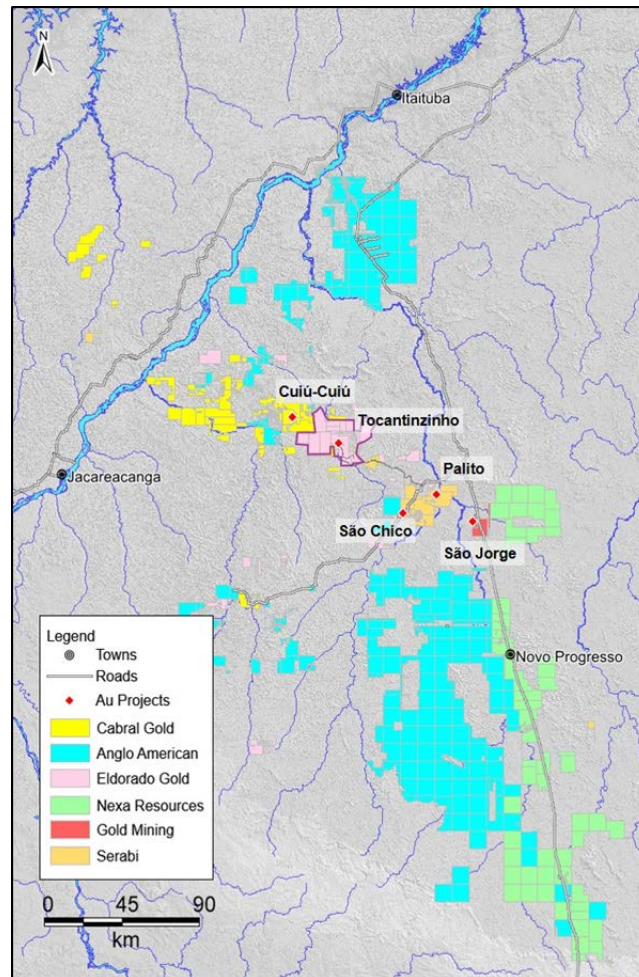


Figure 1-5: Adjacent Properties

1.21 INTERPRETATION AND CONCLUSIONS

The Project has been reviewed thoroughly and it is concluded that it is viable from a technical and economic standpoint. There are some opportunities to fine tune some of the design in the next phase to reduce expenditure.

Regulation related to the tailings impoundment may change in Brazil however the current design is considered to be thorough.

The economy and politics of Brazil could impact the exchange rate of the real so this should be monitored during the execution of the project.

As shown in the metallurgical tests, there is a high content of silver which has not been quantified in the resource. There is an opportunity to improve the economics of the project with the silver that will be extracted with the gold.

1.22 RECOMMENDATIONS

The recommendation from this technical report is that the project is viable and should proceed to detailed engineering.

SECTION • 2 INTRODUCTION

2.1 GENERAL

Eldorado Gold Corporation (Eldorado), an international gold mining company based in Vancouver, British Columbia, through its wholly owned subsidiary Brazauro Recursos Minerais S/A (Brazauro) owns the Tocantinzinho Project (Project), located in the State of Pará, northern Brazil.

Eldorado has prepared this updated technical report of the Tocantinzinho Project which resulted from optimization studies.

2.2 REPORT AUTHORS

The contributors of the report are listed as follows:

Table 2-1: Technical Report Matrix

Section	Title	Company
1	Summary	Eldorado
2	Introduction	Eldorado
3	Reliance on Other Experts	Eldorado
4	Property Description and Location	Eldorado
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Eldorado
6	History	Eldorado
7	Geological Setting and Mineralization	Eldorado
8	Deposit Types	Eldorado
9	Exploration	Eldorado
10	Drilling	Eldorado
11	Sample Preparation, Analyses and Security	Eldorado
12	Data Verification	Eldorado
13	Mineral Processing and Metallurgical Testing	Hatch
14	Mineral Resource Estimates	Eldorado
15	Mineral Reserve Estimates	Nilsson Mine Services
16	Mining Methods	Nilsson Mine Services
17	Recovery Methods	Hatch
18	Project Infrastructure	Eldorado F&Z Consultoria e Projetos
19	Market Studies and Contracts	Eldorado
20	Environmental Studies, Permitting and Social or Community Impact	Eldorado
21	Capital and Operating Costs	Global Project Management Nilsson Mine Services Hatch

Section	Title	Company
22	Economical Analysis	Eldorado
23	Adjacent Properties	Eldorado
24	Other Relevant Data and Information	Eldorado
25	Interpretation and Conclusions	Eldorado
26	Recommendations	Eldorado
27	References	Eldorado
28	Certificates of Authors and Date and Signature Page	Various

2.2.1 Terms of Reference

Companies that provided input to the study include:

- L&M Assessoria Empresarial provided tax expertise to the capital, operating and sustaining costs as well as financial analysis with the tax credits incorporated.
- TEC 3 Geotecnia e Recursos Hídricos designed the tailings impoundment facilities.
- Ausenco Solutions Canada designed the process plant and civil site works outside of the mine and tailings in 2017.
- Hatch updated the process plant design and civil site works outside of the mine and tailings in 2019 based on the optimization activity done in 2018.
- Nilsson Mine Services designed the mine and construction quarry.
- Golder provided the pit slope parameters based on geotechnical work completed in 2012.
- VOGBR completed hydrogeological studies of the pit area in 2010.

2.2.2 Qualified Persons (QP)

Table 2-2: List of Qualified Persons

Qualified Person	Professional Designation	Title	Company	Sections Responsible for
David Sutherland	P. Eng. (BC)	Project Manager	Eldorado	1,2,3,4,5,6,18.1 to 18.3, 18.5 to 18.9,19,20,22,24, 25,26,27
Rafael Gradim	P. Geo (BC)	Corporate Development Manager - Technical Evaluations	Eldorado	7,8,9,10,11,12,14,23
Persio Rosario	P.Eng (BC)	Communion Director	Hatch	13,17,21.2.3
John Nilsson	P.Eng (BC)	President	Nilsson Mine Services	15,16,21.2.2
Paulo Franca	AusIMM	Director	F&Z Consultoria e Projetos	18.4
William McKenzie	P.Eng (BC)	President	Global Project Management	21.1,21.2.1, 21.2.4

2.2.3 Site Visits

Table 2-3: Project Site Visits

Qualified Person	Site Visits	Date of Visit
David Sutherland	4	March 29-30, 2017 (latest)
Rafael Gradim	1	February 21-23, 2019
Persio Rosario	1	February 21-23, 2019
John Nilsson	1	February 21-23, 2019
Paulo Franca	1	February 21-23, 2019
William McKenzie	1	March 22-23, 2018

SECTION • 3

RELIANCE ON OTHER EXPERTS

Eldorado prepared this document with input from the Eldorado's Brazilian operations, other well qualified individuals and third party experts. The qualified persons did not rely on a report, opinion or statement of another expert who is not a qualified person, concerning legal, political, or environmental matters relevant to the technical report.

In compiling this technical report, the taxes were calculated and the financial analysis was done by L&M Advisory of Sao Paulo, Brazil.

SECTION • 4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROJECT AREA AND LOCATION

The Tocantinzinho Project comprises an area of 68,803.6 ha and is located in the State of Pará in northern Brazil.

The Tocantinzinho Project is located in the State of Pará in northern Brazil, in the Tapajós Gold province, approximately 200 km south-southwest of the city of Itaituba; 108 km from the district of Morais Almeida, and approximately 1,150 km in S60°W bearing from Belém, the capital of Pará State located along the north seacoast of Brazil, at the mouth of the Amazon River (Figure 4-1).

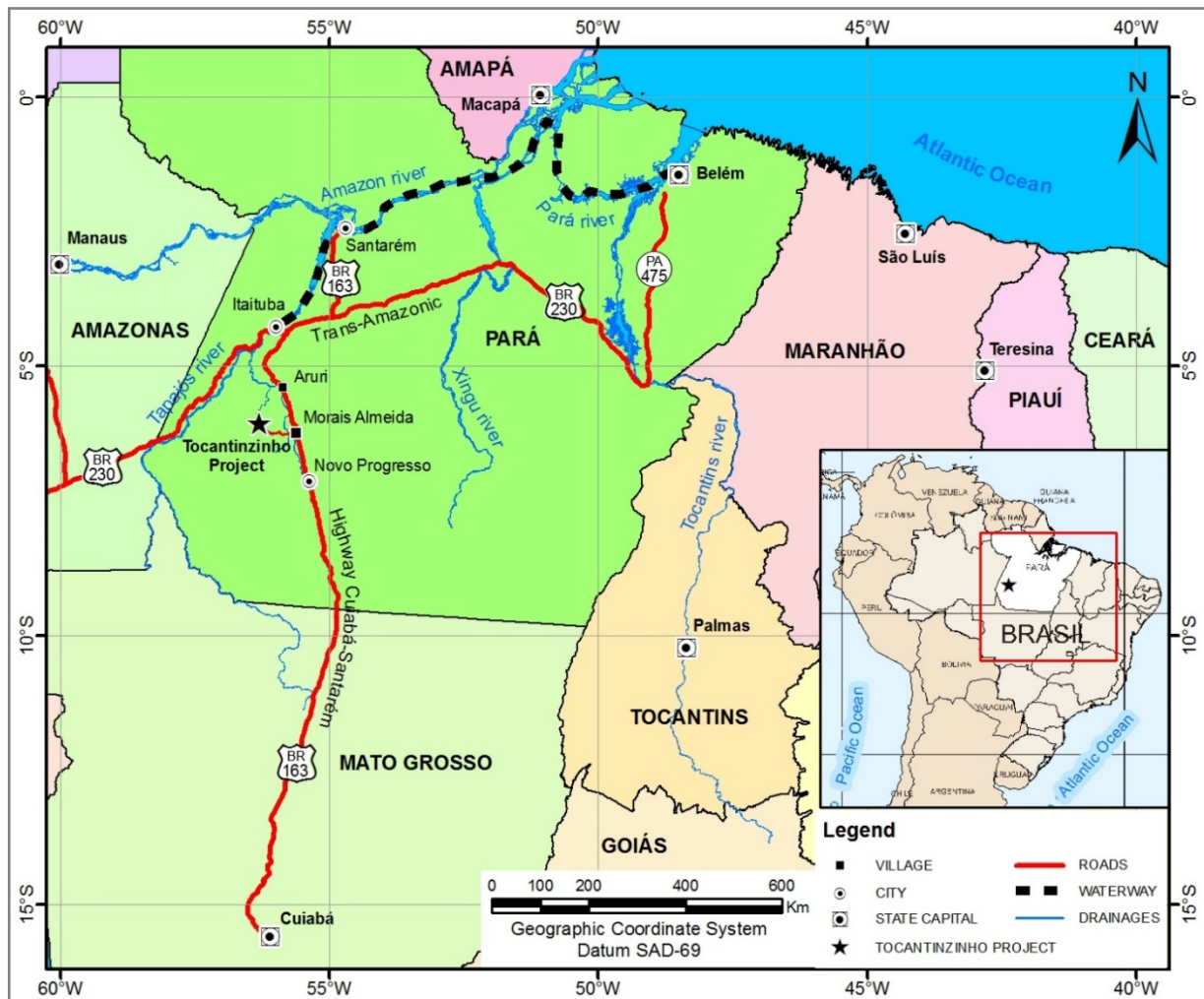


Figure 4-1: Tocantinzinho Project Location

The Project's location can be found on the Vila Riozinho topographic map sheet (SB.21-Z-A, MIR 194; 1:250,000) at the central northern part. The site is situated at an elevation of 120 m above the sea level.

Approximate coordinates of the center of the Tocantinzinho Project area are as follows:

- Geographic: S= 06°03'; W= 56°18'
- UTM (Zone 21M): N= 9,330,700; E= 578,200

4.2 LAND TENURE AND MINING RIGHTS IN BRAZIL

4.2.1 Mining Rights

Brazilian Mining Code (CM) recently has changed by the Decree no. 9,406 of June 12, 2018 that included the extinction of the National Department of Mineral Production (DNPM) and creation of the National Mining Agency (ANM). However the rights of owners of mineral licenses remain unchanged.

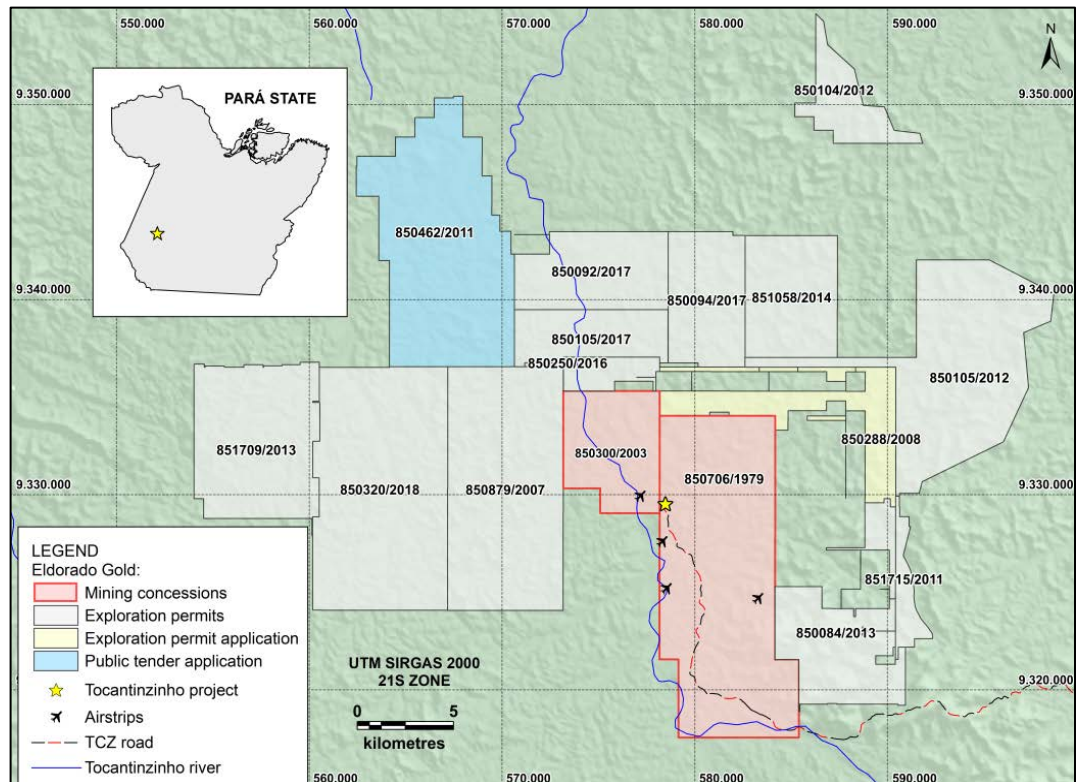
The Tocantinzinho Project comprises of 16 claims over an area of 68,803.6 ha. The mining concession is within the area consisting of two core claims, ANM no. 850,706/1979 and 850,300/2003, covering 12,888.7 ha. These two core claims under mining lease status held 100% by Brazauro Recursos Minerais S.A Brazilian subsidiary of Eldorado Gold Corporation.

The 14 peripheral claims have an additional area of 55,914.9 ha (ANM no. 851,709/2013, 850,320/2018, 850,879/2007, 850,288/2016, 850,105/2017, 850,092/2017, 850,094/2017, 851,058/2014, 850,105/2012, 850,288/2008, 851,715/2011, 850,084/2013, 850,462/2011 and 850,104/2012).

All claims are embedded in the municipality of Itaituba/PA. Mineral rights/claims related to the Tocantinzinho Project are summarized in Table 4-1 and can be seen in the map of Figure 4-2.

Table 4-1 Tocantinzinho Project Mineral Rights

ANM no.	Area (ha)	Status	Owner
850,300/2003	2,888.69	Mining Concession	Brazauro Recursos Minerais S.A.
850,706/1979	10,000.00	Mining Concession	Brazauro Recursos Minerais S.A.
851,709/2013	5,001.13	Exploration Permit	Brazauro Recursos Minerais S.A.
850,320/2018	8,537.29	Exploration Permit	Brazauro Recursos Minerais S.A.
850,879/2007	7,497.75	Exploration Permit	Brazauro Recursos Minerais S.A.
850,250/2016	748.12	Exploration Permit	Brazauro Recursos Minerais S.A.
850,105/2017	2,043.52	Exploration Permit	Brazauro Recursos Minerais S.A.
850,092/2017	2,979.17	Exploration Permit	Brazauro Recursos Minerais S.A.
850,094/2017	2,734.56	Exploration Permit	Brazauro Recursos Minerais S.A.
851,058/2014	2,988.53	Exploration Permit	Brazauro Recursos Minerais S.A.
850,288/2008	2,673.37	Exploration Permit Application	Brazauro Recursos Minerais S.A.
850,105/2012	7,003.77	Exploration Permit	Brazauro Recursos Minerais S.A.
851,715/2011	661.58	Exploration Permit	Brazauro Recursos Minerais S.A.
850,084/2013	3,645.74	Exploration Permit	Brazauro Recursos Minerais S.A.
850,462/2011	7,895.87	Public Tender Application	André Luiz de Deus Maciel
850,104/2012	1,504.54	Exploration Permit	Brazauro Recursos Minerais S.A.

**Figure 4-2: Tocantinzinho Project Mineral Rights**

4.2.2 Surface Rights

The surface area of the Project is located in a property of the Federal Government known as Gleba Sumaúma. The Federal Government issued a certificate stating that there are no indigenous reserves, traditional communities nor small agricultural settlements in the area. Brazauro identified owners established in the area, who had requested recognition of possession rights before the Federal Government. In 2017, Brazauro concluded negotiations with the owners and paid the respective indemnifications, as set forth in the Mining Code. The negotiations covered an area of 6,670 ha, sufficient for the Project, including all areas required for the pit, waste dump, process plant, tailings dam and ponds, camping and administrative buildings. In consideration for the payment, the owners renounced to their requests for possession rights before the Federal Government.

In addition to the above, Brazauro concluded settlements with other squatters, artisanal miners (garimpeiros), small merchants and other occupants without title in activity in the area, with objective to relocate and/or indemnify them. Currently, there are five remaining squatters/garimpeiros who have filed lawsuits to claim additional indemnification payments. Brazauro is pursuing a court settlement or a final court decision to said cases to complete this relocation/indemnification process.

The site project can be accessed by 30 km of the Transgarimpeira state road and 71 km of the Tocantinzinho municipal road.

To construct the power line, Brazauro has been in negotiation with 148 owners along a strip land of 192 km length per 25 m width. As of February 2019, 117 agreements were completed and the remaining are in negotiation.

4.3 ROYALTIES

Consideration has been given to the Federal Royalty, CFEM (Compensação Financeira pela Exploração de Recursos Minerais) and a net smelter royalty (NSR) based on an agreement with Sailfish Royalty Corp.

4.3.1 Royalty Payable to the Federal Government – CFEM

The Federal Constitution of Brazil has established that the states, municipalities, Federal districts and certain agencies of the federal administration are entitled to receive royalties for the extraction of mineral resources by holders of mining concessions (including extraction permits). The royalty rate for gold is 1.5% of gross sales of the mineral product, less sales taxes on the mineral product, transportation and insurance costs.

4.3.2 Private Royalty - NSR

A contractual royalty of 3.5% on Au produced is payable to Sailfish Royalty Corp. Eldorado retains the right to buy-back an undivided 2% of the royalty for US\$5.5 M upon a positive construction decision.

4.4 WATER RIGHTS

A preliminary water permit was granted to Brazauro to reserve the right to use surface water from Veados Creek, to supply the water demand in the beneficiation plant. In August 2018 Brazauro requested an extension in the license validity.

There is a potable water well that has been permitted for the existing exploration camp and there are preliminary permits for eleven future potable water wells. More details can be found in Section 20.

4.5 ENVIRONMENTAL LIABILITIES

4.5.1 Past Mining Activities

Garimpeiros have been working within the project areas for decades with continued activity at the site and it would be expected to find environment contamination related to the past activities, since the primary method of gold extraction used by the garimpeiros was using mercury. However, although there is wide dissemination of mercury in areas of the site, the magnitude found is not enough cause for concern. However during prestripping of the mine, monitoring will take place to discover any potential areas with higher than expected concentrations.

4.5.2 Environmental Studies

Environmental studies have been completed to support the environmental assessment and enable the application of the Installation License. This work was carried out by environmental engineering groups familiar with the Federal and State Government regulations for mining projects. Environmental studies along the power transmission line corridor between Novo Progresso and project site are ongoing; the baseline studies were stipulated as a condition when the Installation License for the transmission line was granted.

4.5.3 Closure and Site Remediation Planning

A high level closure strategy, consistent with industry practice, has been developed and will be further defined upon the start of project execution. Additional details of this plan can be found in Section 20.

4.6 PERMITTING

The majority of the permitting has been completed and is further detailed in Section 20.

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

5.1 SITE TOPOGRAPHY

The Project site is located in the central area of the Tapajós River basin, approximately 100 km to the southeast of the Tapajós River. The regional drainage is to the north. Topography within the project area is irregular with moderate relief. Elevations within the project area vary from approximately 120 m at the Tocantinzinho River to over 200 m at local topographic highs.

5.2 SITE ACCESS

5.2.1 Road

Itaituba is a city located in the state of Pará and is the local centre for services and supplies. The Cuiabá-Santarém highway (BR-163) extends south to Cuiaba in Mato Grosso state which links to national highways reaching industries in southern Brazil. The BR-163 extends north to ports in Itaituba and Santarém on the Tapajós River which connect to the Amazonas River by barge or the nation highway system to Belém the capital of Pará State.

A 72 km road connects Tocantinzinho site to the Transgarimpeira road. The Transgarimpeira road is accessed from Moraes de Almeida, 21 km east of the Jamanxim River barge crossing at Jardim do Ouro.

Road transportation routes and distances are summarized in Table 5-1.

Table 5-1: Estimated Distance of Main Access – Roads

Route	Distance	Access	Travel Time
	km		hours
São Paulo - Moraes Almeida	2,696	BR-050/BR-158/BR-163	37
Belo Horizonte - Moraes Almeida	2,634	BR-040/BR-163	38
Belém - Moraes Almeida	1,532	BR-230/BR-163	23
Novo Progresso - Moraes Almeida	100	BR-163	4
Itaituba - Moraes Almeida	303	BR-163	5.5
Cuiabá - Moraes Almeida	1,172	BR-163	18
Tocantinzinho Project - Deposit	2	Unpaved Road	0.25

5.2.2 Airport

The Pista Nações Unidas airstrip serves the Tocantinzinho Project. The Pista Nações Unida airstrip is 775 m long and is situated 2.0 km south of the camp. This airstrip will primarily be used to supply the camp with personnel and supplies. Utilization is limited to daylight as the runway has no signals or lighting. An allowance to widen and resurface the airstrip is included in the capital cost estimate.

Air transportation routes and distances are summarized in Table 5-2.

Table 5-2: Estimated Distance of Main Access - Air

Route	Distance	Access	Travel Time
	km		hours
São Paulo – Belém	2,450	Commercial flight	4
Belo Horizonte – Belém	2,100	Commercial flight	3
Belém – Itaituba	900	Commercial Flight	3
Itaituba – Tocantinzinho Project	200	Charter Plane	1
Manaus – Tocantinzinho Project	600	Charter Plane	2

5.2.3 Railways

Site access via railway is not applicable for the Tocantinzinho Project.

5.2.4 Water

The Tocantinzinho River and other small streams transect the region allowing access to the property by small boats. Access to the site with larger vessels is not practical.

5.3 PHYSIOGRAPHY AND CLIMATE

5.3.1 Physiography

The Project property is located in the north region of Brazil, which is humid tropical forest. In the study area three forest typologies were observed: secondary forest, dense alluvial forest, and submontane dense ombrophilous forest.

The area is hilly with areas that are rocky or saprolite. The local geology is represented by three lithostratigraphic units: Parauari Intrusive Suite, Maloquinha Intrusive Suite and Quaternary Alluvial Reservoirs. The most significant rocky outcrops occur in the sector where artisanal mining activities were concentrated. The rocks are altered and the different saprolites present a distinction of color and lithological material.

The site area is part of the hydrographic sub-basin of the Jamanxim River that integrates the Tapajós River basin. This basin has the most of its extension protected by conservation units. The site area is influenced by Jamanxim and Tocantinzinho rivers, besides its main tributaries, Veados and Teodorão creeks.

5.3.2 Climate Characterization

Brazil's Northern Region has a tropical climate. During most months of the year, there is significant rainfall. The climate zone in the Northern Region has very distinct areas that lead to three climate subzones: extremely wet, wet, and semi-wet.

Rain is generated by tropical instability lines where converging air leads sometimes to the formation of rain and thunderstorms, and sometimes the formation of hail and moderate to strong winds with gusts reaching 60 to 90 km/h.

The mean annual temperature in the region is approximately 28°C. In general, the temperature amplitudes are small with a gradual increase during winter. The mean absolute values are in range of 23°C to 37°C. The relative humidity averages above 70% throughout the year.

The average annual precipitation is approximately 2,300 mm. The rainiest trimester contributes about 40% of total annual rainfall, corresponding to the months of February, March and April. The driest trimester, corresponding to the months of July, August and September, contribute less than 15% of total annual rainfall.

A weather station operated by the National Meteorological Institute, close to the property is the Itaituba station, with coordinates 4°10'S and 55°21'W.

The Project site is located in the tropical zone of the Southern Hemisphere at the geographic coordinates 6°04'04.02 "S and 56°17'00.17"W, southwestern portion of the Pará State, municipality of Itaituba and the climate in the region can be classified as tropical, hot, humid, with 1 to 2 months of drought in the year.

To verify the climatic characteristics of the region, data was collected in the Climatological Station of Itaituba municipality. The climate normals and averages were used to summarize and describe the average climatic conditions of the project area. The data were compiled and are presented in Table 5-3.

Table 5-3: Itaituba Climate Normals Station Data

Month	Minimum Range	Average Range	Maximum Range	Maximum Absolute	Minimum Absolute	Relative Humidity	Total Insolation	Total Rainfall	Average Evaporation
	Temperature °C					%	hours	mm	mm
Jan	24.51	26.48	28.60	34.52	19.21	86.33	139	226	58
Feb	24.25	26.20	28.42	33.48	19.35	89.76	109	291	47
Mar	24.80	26.34	28.14	33.20	19.21	89.71	119	302	52
Apr	25.05	26.55	28.15	33.76	19.22	89.44	148	243	49
May	25.16	26.66	27.78	34.10	19.80	89.68	170	186	51
Jun	24.65	26.74	28.09	34.27	18.82	87.81	213	93	61
Jul	23.95	26.66	28.02	35.33	16.96	85.48	246	65	76
Aug	25.01	27.28	29.12	35.94	17.18	82.97	241	57	91
Sep	25.19	27.86	29.47	36.58	19.68	82.07	218	79	95
Oct	24.90	27.97	29.50	36.25	20.17	81.63	193	99	97
Nov	24.82	27.71	30.22	36.34	19.55	81.21	171	144	84
Dec	25.07	27.20	29.21	34.86	19.35	86.22	145	173	73
Annual	24.78	26.97	28.73	34.88	19.04	86.03	176	1,957	834

The graphs presented in Figure 5-1 to Figure 5-4 show, respectively, the temporal distributions of temperature, air relative humidity, total insolation, in addition to the total rainfall and evaporation of Itaituba station, considered representative to the Project site.

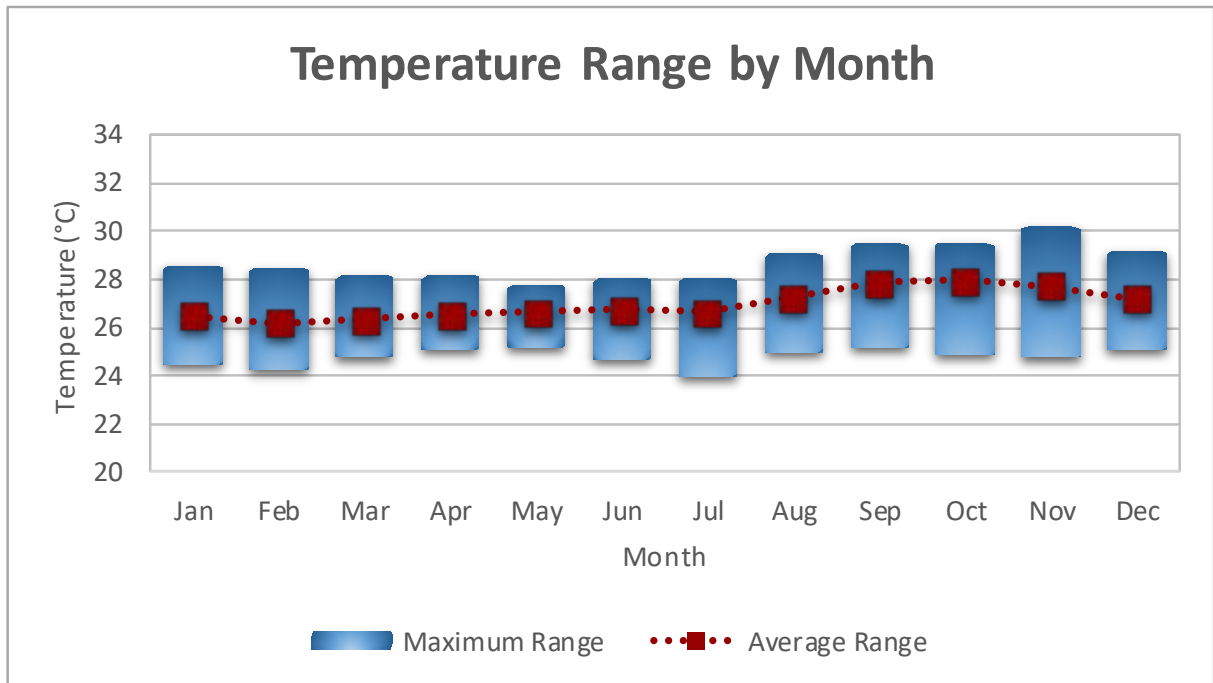


Figure 5-1: Characteristic Distribution of Monthly Temperature Range at Itaituba Station

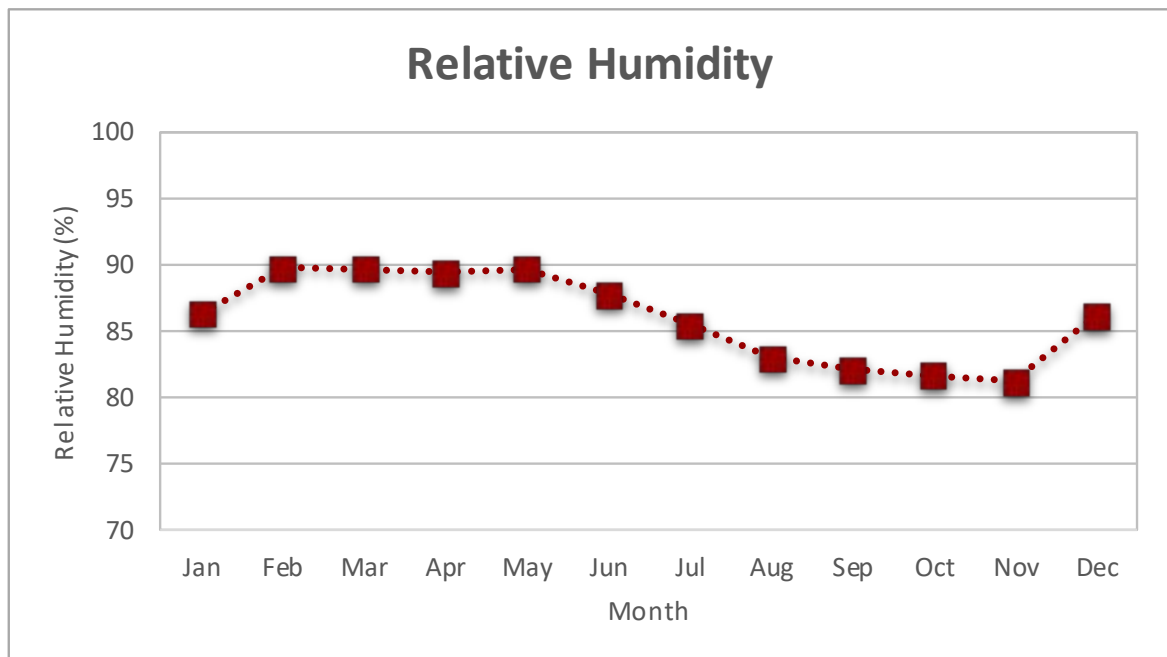


Figure 5-2: Characteristic Distribution of Air Relative Humidity at Itaituba Station

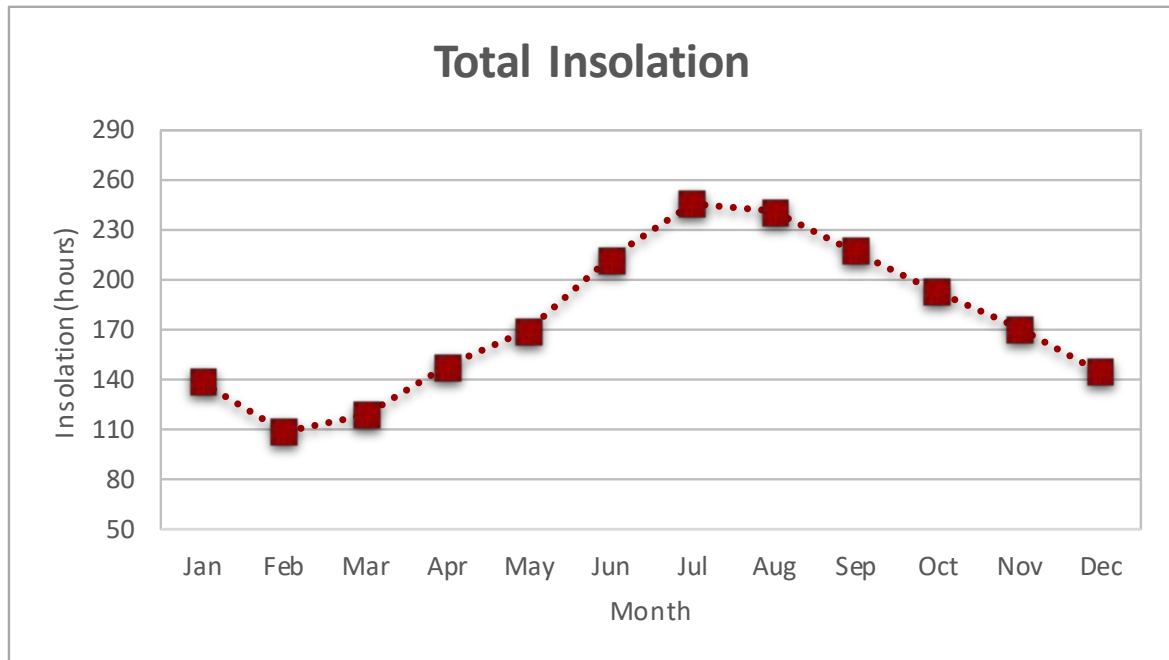


Figure 5-3: Characteristic Distribution of Total Insolation at Itaituba Station

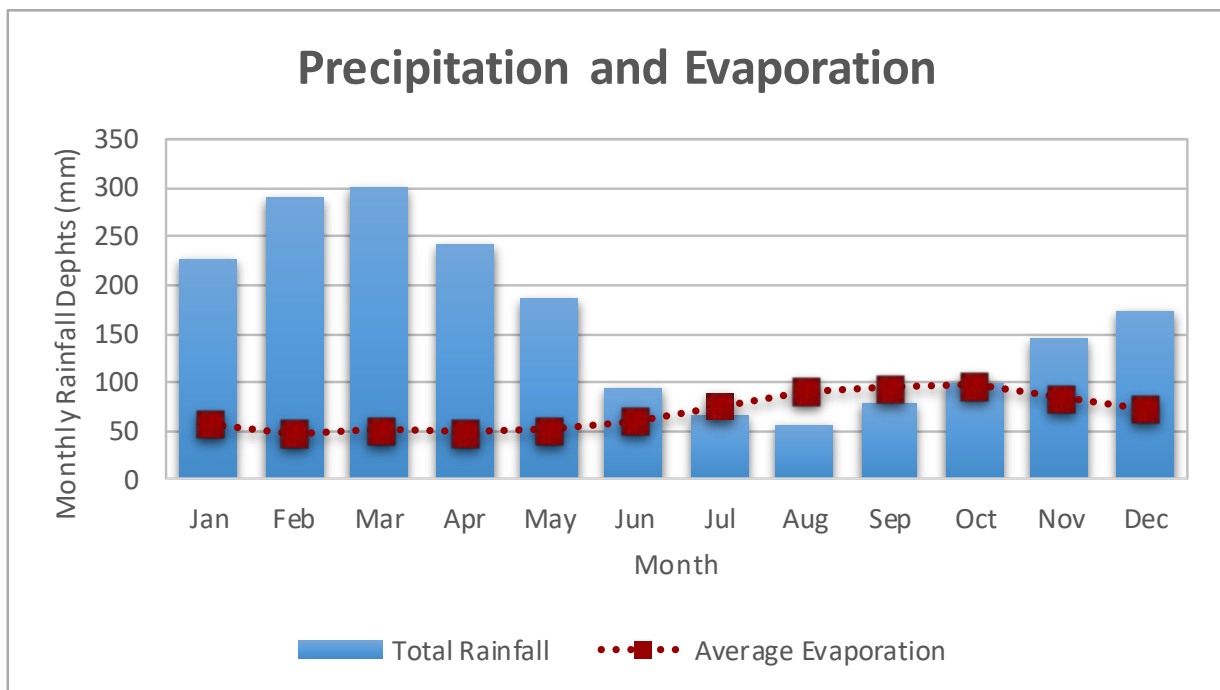


Figure 5-4: Characteristic Distribution of Precipitation and Evaporation at Itaituba Station

The analysis of the Itaituba station data show that the average annual temperature corresponds to 27°C with an amplitude of 3.9°C and the maximum average temperatures occur in October and November, with values around of 30°C. In addition, the records of the lowest temperatures occur in July, with a mean temperature of 23°C. The low temperature amplitude associated with median rates of relative humidity and low rainfall depths characterize the regional climate, being evident the seasonality in the distribution of the rainy season. Peak precipitation usually occurs between January and May.

5.3.2.1 Humidity

Relative humidity ranges from 27% (dry) to 99% (very humid). The air is driest in August, dropping to approximately 29%, and is most humid in January, exceeding 96% relative humidity.

5.3.2.2 Barometric Pressure

The average barometric pressure has a median of 1,011 mbar, time-weighted average of 1,010 mbar and standard deviation of 1.35 mbar.

5.3.2.3 Evaporation

The total evaporation is reported at 900 mm/year for the region.

5.3.2.4 Wind

Annual average wind speeds vary from 0 m/s to 7 m/s (light to moderate breeze), rarely exceeding 7 m/s (moderate breeze). The highest average wind speed is 4 m/s, and lowest average 2 m/s. Maximum gust speed recorded in 2015 was 37 m/s.

The wind is most often out of the east (15% of the time) as shown on Figure 5-5. The sum does not sum to 100% as it accounts for duration where wind speed is zero.

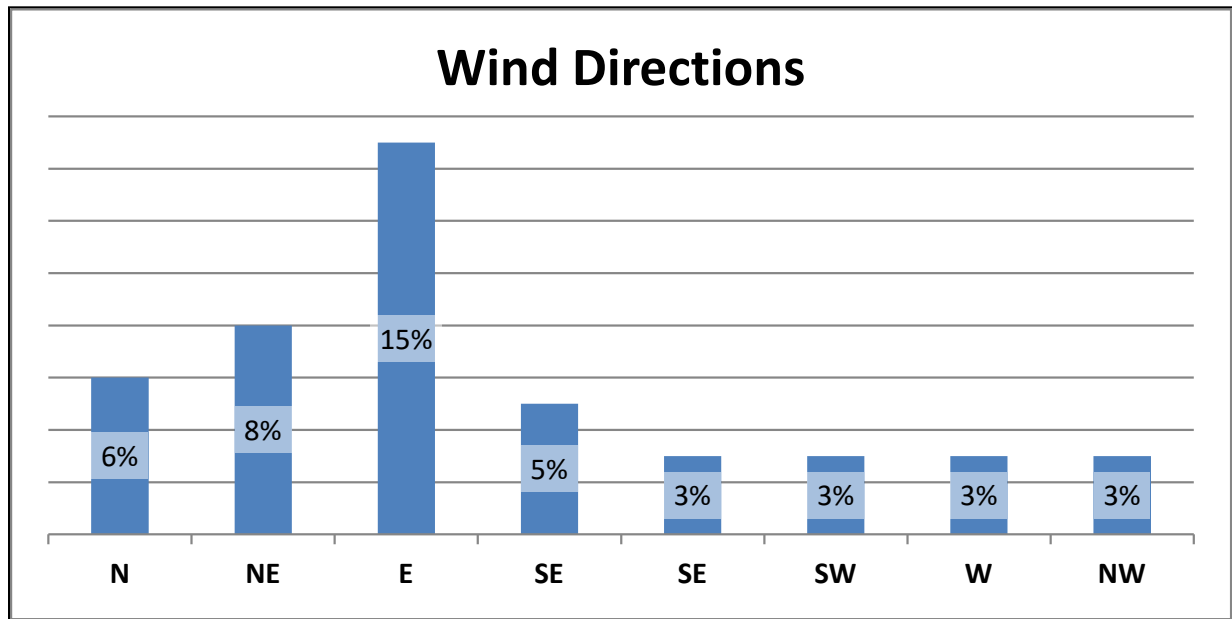


Figure 5-5: Wind Directions History

5.3.3 Air Quality

Air quality for the region is unknown as the region has no industrial activity and minimal habitation within 100 km radius of the site. No air quality information is available for the Itaituba region.

5.4 INFRASTRUCTURE

5.4.1 Road

Itaituba is the local center for services and supplies. The Cuiabá-Santarém Highway BR-163, extending northward from the state of Mato Grosso, reaches Itaituba via a ferry crossing of the Tapajós River. Most heavy equipment and supplies reach Itaituba by smaller ships which move along the Amazon River and Tapajós River.

Highways are in place and maintained within 72 km of the site. The Transgarimpeira road is accessed from Moraes de Almeida, 22 km east of the Jamanxim River barge crossing at Jardim do Ouro. The remaining section of the road has been built to connect the Tocantinzinho site to the Transgarimpeira road. This road is used by logging trucks and requires ongoing maintenance and some work to facilitate traffic for the construction of the Project.

5.4.2 Power

The Tocantinzinho Project will be supplied from the Novo Progresso substation to the south, which will require the construction of approximately 190 km of transmission line and a 138 kV substation at the site. Power infrastructure to upgrade and supply additional power to Novo Progresso is planned to be constructed by the Brazilian utility by later in 2020.

5.4.3 Water

Fresh water will be supplied from Veados Creek and will be stored in a water tank that will supply make-up water to the process; a portion of the water tank will be dedicated as fire water storage.

Potable water will be sourced from wells and will be treated prior to use.

Water reclaimed from ponds will be recycled for use in the process plant.

SECTION • 6 HISTORY

In Tapajós Province region, the mining activity is historically related to gold mineralization. Gold is reported to have been discovered in the region through garimpeiros activities in the 1950s but the area became a significant producer by the 1980s. Unofficial data indicates that in the late 1980s, historical production, by primitive artisanal methods, amounted to between 200,000 and 1 million ounces of gold per year. By the 1990s, the gold production was estimated at 16 million ounces, but the real numbers are unknown.

At Tocantinzinho, the gold production was initiated in 1970 with intense garimpo activity in the mid-eighties to mid-nineties; however, there are no published records to support the timing and amount of production. In 1979 Mineração Aurífera Limitada obtained an exploration license with the Departamento Nacional de Produção Mineral (DNPM) over the Tocantinzinho Project area which expired in 1986. The property files were archived by the DNPM in 1992.

In 1997, Renison Goldfields (Australia) and Altoro formed a joint venture (JV) to explore Brazil for major gold deposits and Tocantinzinho was brought to the JV's attention by an air charter pilot. Management and operation of the JV were executed by Altoro. The project was acquired after a visit to the property by the company geologist who collected channel samples from different garimpeiros pits and returned with good results. In 1998 the JV with Renison Goldfields was terminated due to a corporate decision and as a consequence, all properties, projects and data acquired during the joint venture were passed to Altoro.

Altoro's exploration program was carried out from 1998 to early 2000 and consisted of soil geochemistry, ground magnetic survey, auger drilling and geological mapping. Solitario Resources Corporation acquired Altoro in 2000 and terminated the Tocantinzinho Project a year later due to the low gold price.

In 2003, Brazauro, through its Brazilian subsidiary Jaguar Resources do Brazil Ltda., acquired the properties covering the Tocantinzinho mineralization. Based on the results of geochemical soil sampling, Brazauro initiated a drilling program that lasted until 2008 with a total of 25,635 m on 97 holes.

In July 2008, Eldorado reached an agreement with Brazauro which ensured that Tocantinzinho Project would be explored and developed in a timely manner through the access to Eldorado's exploration and project development expertise in Brazil. After Eldorado took over the project, in September of the same year, the exploration works were continued in Tocantinzinho with a further 62 drill holes for 19,431 m, reverse circulation and auger drilling, soil geochemistry and geological mapping in surroundings.

In July, 2010 Eldorado completed the arrangement to acquire all the issued and outstanding securities that it did not already owned of Brazauro.

In 2012, the Preliminary Environment License was approved and the majority of permits were obtained by 2017. Various positive studies were completed during this stage however the project was deferred pending permits, improved gold outlook and other company projects.

The two major permits required for construction have been granted:

- Para State Department of Environment and Sustainability granted the Installation License in April 2017 with the modified Installation License (LI no. 2771) approved August 9, 2017.
- National Department of Mineral Production (DNPM) (now ANM – National Mining Agency) issued the mining concessions (Ordinance no 87/SGM and 88/SGM) May 18, 2018.

SECTION • 7

GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Tapajós Gold Province is an important metallogenic province located in the central southern portion of the Amazon craton and part of the Venturi-Tapajós (Tassinari and Macambira, 1999) or Tapajós-Parima (Santos et al., 2001) geochronological-tectonic province (Figure 7-1).

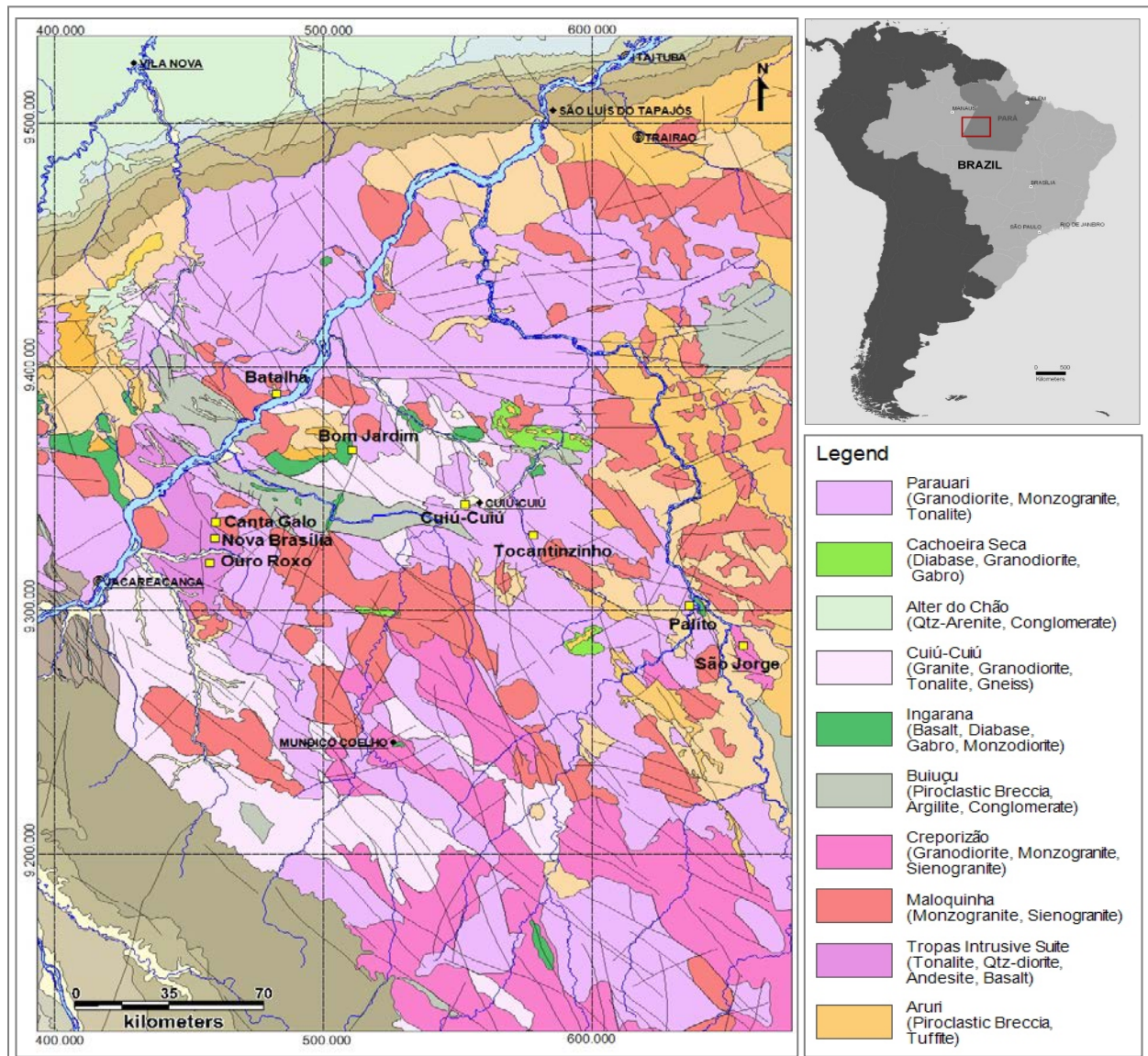


Figure 7-1: Tapajós Gold Province - Location and Regional Geology Map

The oldest rocks found in the Tapajós district are gneisses, schists, and metagranites of the Cuiú-Cuiú complex (2,033 – 2,011 Ma) which is the local basement for all units present in the region (Santos et al., 2001). The Cuiú-Cuiú complex is intruded by granites and granodiorites of the Creporizão Suite (1,974 ± 6 to 1,957 ± 6 Ma), tonalites, diorites and granodiorites of the Tropas Suite (1,909–1,895 Ma) and granites and granodiorites of the Parauari Suite (1,898 – 1,880 Ma) (Figure 7-2); Santos et al., 2001). The rocks of the Parauari, Tropas and Creporizão suites are interpreted to represent the roots of magmatic arcs. Coeval intrusive and extrusive rhyolite, dacite, and andesite of the Bom Jardim and Salustiano formations (1,900–1,853 Ma) and volcanoclastic rocks of the Aruri Formation (1,893 – 1,853 Ma) cut or overlie all units above. Maloquinha Suite alkaline granites (1,882–1,870 Ma) are considered to be anorogenic and intrude all other intrusive units.

In general, the central-northwest portion of the Tapajós district is dominated by the Parauari granites, the southeastern portion is dominated by the Creporizão granites, and the eastern portion is dominated by the Salustiano and Aruri volcanic sequences. The Maloquinha granite is widespread throughout the district.

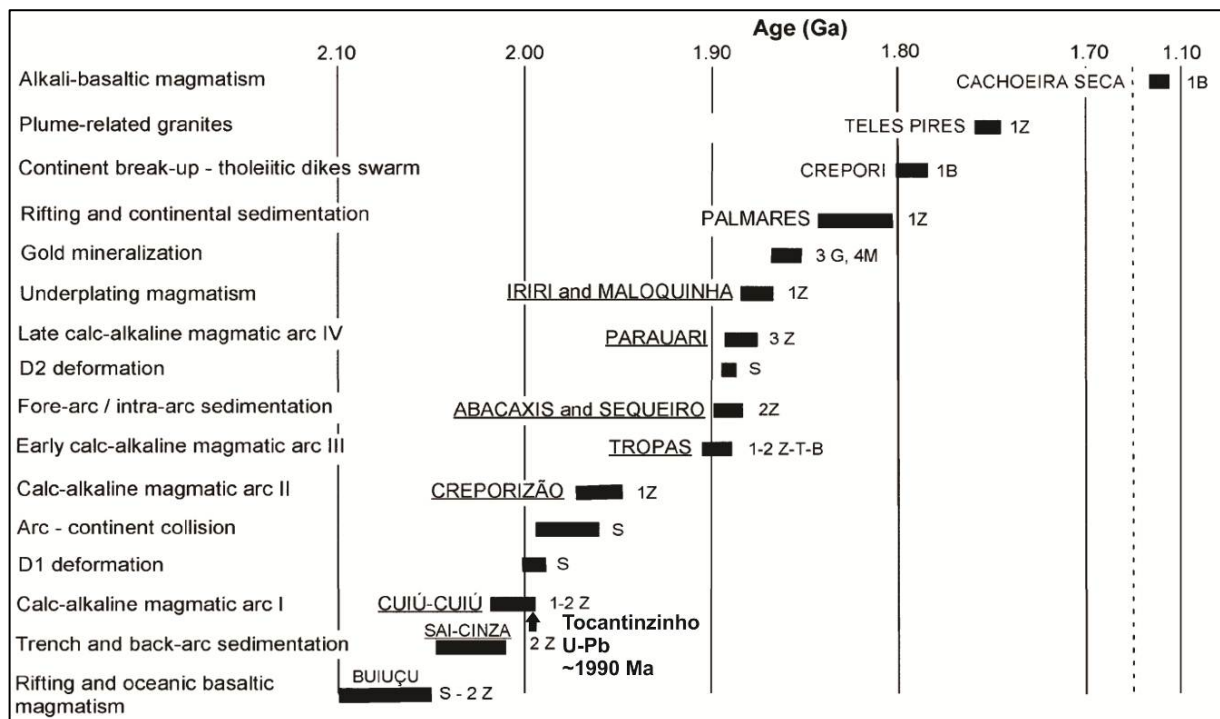


Figure 7-2: Tocantinzinho Age in the Context of Regional Magmatism and Tectonism (modified from Santos et al., 2001)

Gold occurrences are known in almost all rocks types. The main occurrences are in Parauari Suite (Palito), the Cuiú-Cuiú Complex (Cuiú-Cuiú), Tropas Suite (Ouro Roxo), Creporizão Suite (São Jorge and Sucuri), Salustiano and Bom Jardim Formations (V3-Botica, Bom Jardim and Doze de Outubro) and Maloquinha Suite (Mamoal).

In the Tapajós district it has been proposed that most of the intrusions associated with significant artisanal mining, including the Tocantinzinho Project, align along a north-northwest trending

lineament known as the Chico Torres Megashear or Tocantinzinho Trend (Santiago et al., 2013; Borgo et al., 2017; Biondi et al., 2018). This interpreted fault zone appears as a distinct topographic lineament on satellite images and is visible on regional aeromagnetic maps as a linear magnetic anomaly.

7.2 LOCAL GEOLOGY

The Tocantinzinho Project is underlain by granitic igneous rocks ranging in age from ~2007 to 1979 Ma (Borgo et al. 2017). This age range suggests they are more likely part of the older magmatic arcs of the Cuiú-Cuiú Complex (~2010 Ma) or Creporizão Suite (~1,970 Ma; Santos et al. 2001) rather than the younger Parauari and Maloquinha magmatic suites (~1,880 Ma; Figure 7-2). Textural evidence and contact relationships suggest that the host granitic rocks at Tocantinzinho intruded as dyke-like bodies along a northwest-striking fault zone that cut through more regionally extensive quartz monzonites (Figure 7-3). Age data support this observation with the surrounding host rocks having an age range of 2,007 to 1,997 Ma compared to the immediate granitic host rocks that range from 1,993 to 1,979 Ma (Borgo et al., 2017). The granitic rocks were likely emplaced synchronous with faulting, and both intrusive contacts and vein orientations suggest the host fault zone was active during this period as a sinistral, dominantly strike-slip feature (Juras et al., 2011; Borgo et al., 2017; Biondi et al., 2018).

7.2.1 Lithologies

7.2.1.1 Tocantinzinho

The Tocantinzinho deposit is hosted within complexly textured granitic rocks that are strongly veined and altered, and localized within the NW-striking fault zone that crosscuts the earlier quartz monzonite country rocks. The majority of gold mineralization occurs within two distinctly textured, fractured and hydrothermally altered granites (locally termed salami and smoky granites). The salami granite is an alkali feldspar granite (dominantly K feldspar, quartz and albite) whereas the smoky granite is compositionally different with plagioclase present and is a granite *sensu stricto*. The plagioclase commonly displays zoning that was susceptible to alteration (sericite with lesser chlorite and calcite) particularly in the feldspar core. This likely reflects original Na-Ca zoning (Na-rims, Ca-cores) in the plagioclase. The salami granite (alkali feldspar granite) and smoky granite have gradational contacts and locally transition into syenite and quartz syenite. The mineralized granitic rocks contain highly variable granitic textures including pegmatite and aplite, and textures indicative of the magmatic-hydrothermal transition such as miarolitic cavities, interconnected miarolitic cavities and unidirectional solidification textures. The complexly textured mineralized granite host rocks are in contact with a barren hematized granite and an outer quartz monzonite. The pink to red hematized granite is a medium- to coarse-grained equigranular phase. The plagioclase feldspar is commonly altered to sericite, primary biotite is generally strongly chloritized and the hematite replaced primary magnetite. The granite displays no sign of penetrative deformation or brecciation and is unmineralized. The ore body is bounded on both sides by a fine to medium grained, gray-green to reddish quartz-monzonite. This unit is generally magnetic and epidote often occurs filling millimetre-

scale fractures. Fine grained, disseminated pyrite is common but is not associated with gold mineralization.

A large andesite body intrudes the mineralized zone (Figure 7-3). This unit forms an upward flaring cap over the main mineralized zone. At surface it varies in widths from 50 to 80 metres and has a vertical dimension of approximately 50 metres, below which there are a series of narrow andesite dykes which are interpreted as feeder dykes to the larger andesite body. The rock is strongly altered, with intense carbonate, chlorite and sericite. Millimetre-scale fractures are commonly filled by carbonate $\pm$ chlorite. Gold values in andesite are generally below the detection limit with the exception of scattered ore-grade gold values associated with quartz-sulphide veins along contacts.

Rhyolite dykes are exposed in the central portion of the deposit on surface and in drill core. Outcrop patterns and exposed contacts show that they cut across the andesite, making the rhyolite the youngest intrusive rock found to date. It is cream to light green colored with rare millimetre-scale quartz grains and potassium-feldspar phenocrysts in an aphanitic groundmass. Rhyolite dykes are generally 1 to 5 metres wide and typically contains less than 5 ppb Au, but values ranging from 100 to 200 ppb Au can be present where veining is intense.

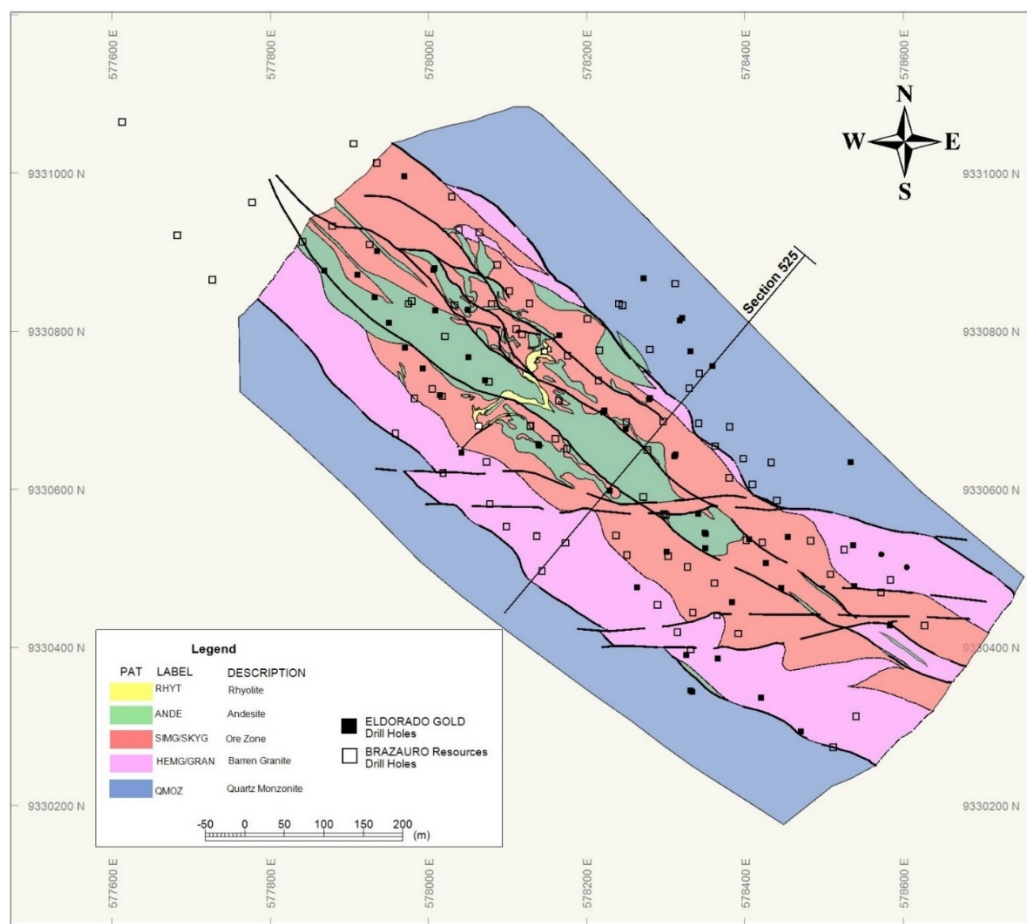


Figure 7-3: Tocantinzinho Geological Map

7.2.1.2 Santa Patricia

The Santa Patricia project occurs 2.5 km west of Tocantinzinho and is defined by > 8 km NW striking Cu-in-soils (> 20 ppm, Section 9) anomaly that coincides with a regional magnetic lineament. Sixteen widely spaced diamond drill holes tested this extensive anomaly to depths of between approximately 200 and 300 m.

The geology of the Santa Patricia is dominated by intrusive rocks that includes a mafic to intermediate plutonic suite and an alkali-granitoid plutonic suite both of which are cut by late stage fine grained andesite and rhyolite dykes. The mafic to intermediate suite comprises medium grained equigranular to porphyritic granodiorite and quartz monzonite, and lesser diorite to monzodiorite and localized gabbro. The alkali-granitoid suite consists of granite to alkali feldspar granite transitional to salami-textured syenite and quartz syenite. Late phases of this suite include micro-granite, aplite and pegmatite. The contacts between the two suites vary from gradational to crosscutting with fractionated aplite-pegmatite phases typically the youngest. However, it is likely that the two suites were initially coeval due to the occurrence of magma mingling and mixing textures between granite, granodiorite and diorite. All of the above igneous rocks host vein-related Cu-Mo mineralization including the late andesite indicating that mineralization was the youngest event.

7.2.1.3 KRB

Rock types at KRB are comprised of a variety of intrusive rocks from alkali feldspar granite, syenogranite and monzogranite to quartz monzonite, granodiorite and monzonite. These sequences are intruded by microgranite, pegmatite, aplite, as well as late dykes of felsic to intermediate composition.

7.2.2 Structural Geology

The Tocantinzinho ore body is localized along a major NW-trending structure which is most clearly portrayed in aeromagnetic data. This structure is interpreted as a regional fault zone which controlled emplacement of the igneous rocks and related mineralization at Tocantinzinho and at several other occurrences and deposits in the Tapajos region.

Most of the mineralized rock encountered in drilling displays a planar fabric defined by thin sheeted veinlets and fractures filled with chlorite-silica. These veinlets usually show a moderate to strong preferred orientation with east to northeast strike and subvertical dip. However, the intensity of this fabric can be quite variable even within a single drill hole. The mineralized granite at Tocantinzinho shows no penetrative foliation except for localized occurrences where strings of blebby quartz grains exhibit a weak parallelism. However, this texture was likely the result of magmatic crystallization and hydrothermal degassing synchronous with active faulting rather than a purely ductile deformation fabric.

Northwest-trending, steeply dipping faults commonly define the contact between the hematized granite and the surrounding quartz-monzonite. These structures are interpreted as portions of, or splays, from the controlling NW-structure. In addition, surface mapping has identified a series of

east-west trending faults within the Tocantinzinho deposit which generally show several to tens of metres of sinistral offset.

7.3 MINERALIZATION

7.3.1 Tocantinzinho

The Tocantinzinho deposit forms a sub-vertical, northwest-trending elongate body approximately 900 metres long by 150-200 metres wide. It has been drilled to approximately 450 metres depth and remains open below this depth.

The gold mineralization is bound on the hangingwall and footwall sides by structural zones which mark the contact with the surrounding barren granite/monzonite. This structural corridor is an outer geological constraint on the estimation domain. The andesitic body close to surface is largely unmineralized and is used as an internal constraint on the estimation domain. Examples of mineralized intercepts in relation to the lithological and structural elements discussed above are shown in a cross section in Figure 7-4. Gold grades are remarkably consistent within the mineralized granite (Figure 7-5).

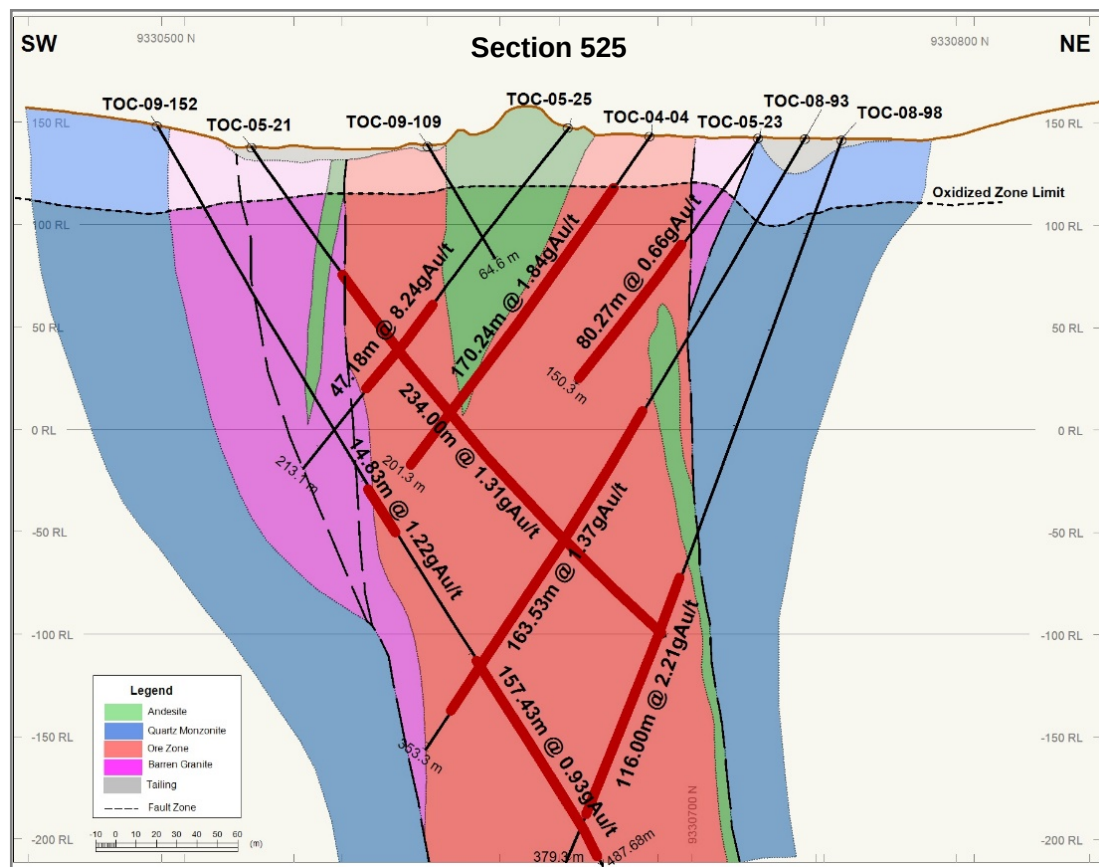


Figure 7-4: Tocantinzinho Deposit –Section 525 showing Mineralization being bound by a Structural Corridor and generally absent inside the Andesitic Bodies

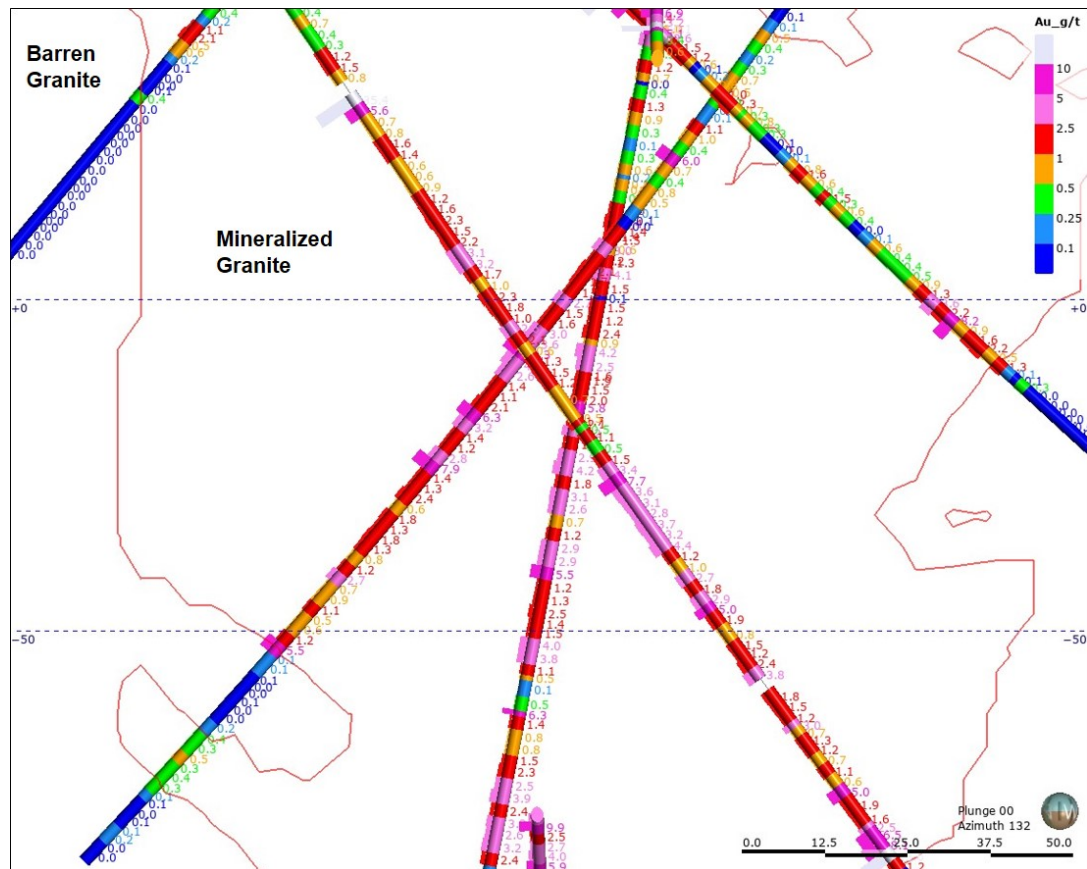


Figure 7-5: Detail of Gold Grade Distribution inside Mineralized Granite. Section 450 looking SE

Mineralized granites at Tocantinzinho are divided into two sub-units, smoky and salami granite, based on granite mineralogy and texture, and alteration mineralogy and colour. Grade distribution is similar in both units and therefore they have been grouped for resource estimation purposes.

The green-grey colour of the smoky granite is due the alteration of plagioclase to sericite with lesser chlorite, calcite and pyrite particularly in the core of the feldspar. Salami mineralized granites are distinctively bright red due to hematite dusting on the feldspars. Staining for potassium feldspar using cobaltinitrate has revealed that both potassic and sodic feldspar are host to the hematitic dusting in feldspar and therefore the red-pink colour is not indicative of potassic alteration. Anastomosing veinlets are common and similar in scale to those in the smoky granites but are generally filled with a distinct black chlorite and lesser sericite-calcite-pyrite-quartz. Quartz textures in the salami and smoky granites commonly have curvilinear grain boundaries with feldspar. Chlorite-sericite-calcite veins are typically wormy rather than planar and appear to cut feldspar but develop along the margins of the quartz grains suggesting this was a relatively late magmatic-hydrothermal phase. Contacts are diffuse between smoky and salami granites and a complete gradation exists between the two units.

Pyrite is the main sulfide phase and commonly contains inclusions of minor chalcopyrite and pyrrhotite. The presence of pyrrhotite is genetically significant because it indicates that the ore at least locally formed under reduced conditions despite the abundance of hematite-dusting on feldspar and hematite-alteration of magnetite in the outer red granite. Gold grains were observed in close association with pyrite along grain boundaries, within fractures and locally as inclusions within pyrite. A white-grey mineral was also observed associated with gold and is likely a bismuth mineral (bismuthinite or bismuth). Multi-element data for samples with anomalous gold ($> 0.1\text{ppm}$) show a strong correlation pair between gold and bismuth (0.87).

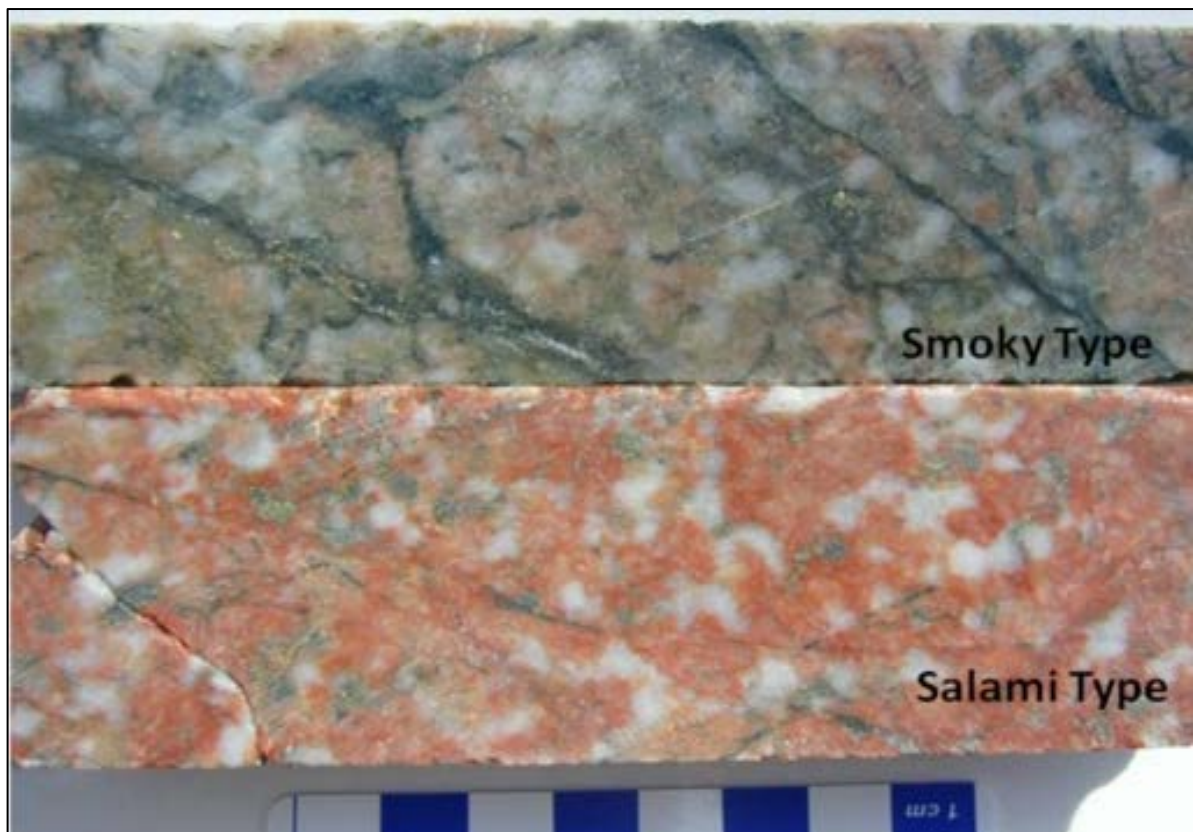


Figure 7-6: Tocantinzinho Deposit “Smoky” and “Salami” Granite

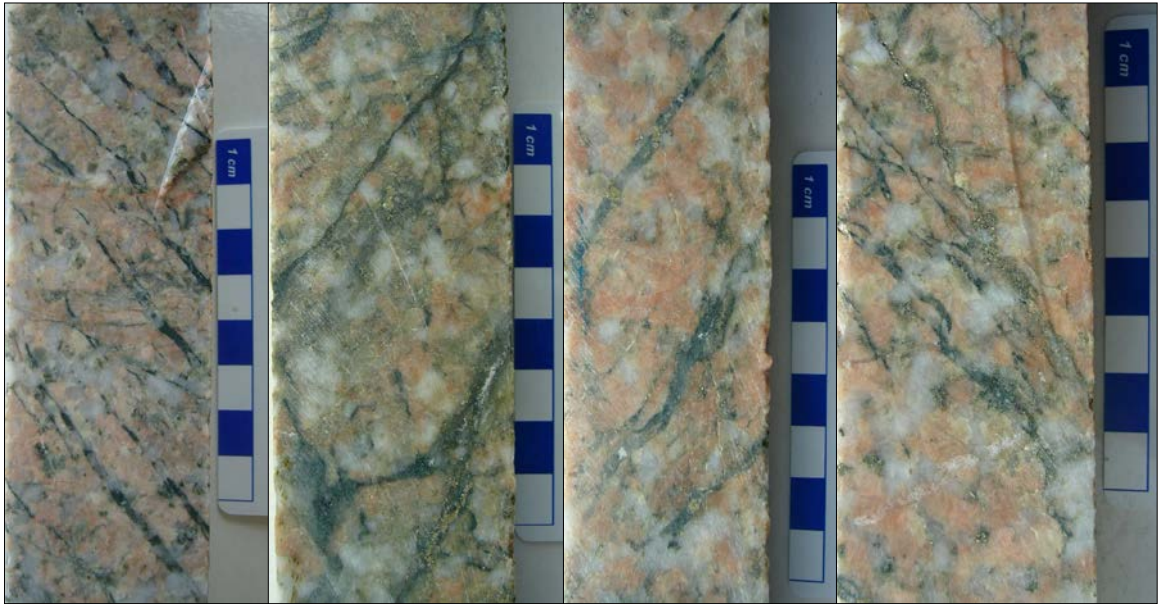


Figure 7-7: Quartz + Chlorite + Pyrite Sheeted Veins

7.3.2 Santa Patricia

The highest grades of Cu-Mo mineralization occur in zones of highest stockwork vein and veinlet intensity, and Cu grade appears to improve with depth in the best drill holes (TOC272 and TOC275). A paragenetic framework has been established that comprises six main vein and alteration stages. The earliest phase is a white quartz±K-feldspar stockwork vein stage that are typically planar, and less commonly irregular, with alteration comprising pink-red K-feldspar as vein envelopes to wider pervasive zones. The main Cu-Mo stage is associated with the second generation of veins that consist of magnetite-muscovite-quartz-chalcopyrite-pyrite-molybdenite±hematite within a strong pervasive coarse muscovite alteration (Figure 7-8), which in places is greisen-like. Localized laminated quartz-molybdenite veins with silicification post-date the main mineralized magnetite-sulfide vein stage. Alteration and vein styles zone away from the main Cu-Mo stockwork to pyrite-sericite-calcite veinlets and alteration through to distal propylitic epidote-calcite-chlorite-pyrite veins and alteration. Late K-feldspar-calcite veinlets crosscut all vein stages.

The Santa Patricia Cu-Mo system is mostly devoid of Au. Where localized Au grades (~0.15 ppm Au) occur, the mineralization style is more akin to the salami-textured style of Tocantinzinho than the stockwork style Cu-Mo mineralization.



Figure 7-8: Quartz Monzonite with well developed White Quartz-K-feldspar Stockwork cut by the main Mineralization Stage. Drill Hole TOC275, 340.5 m

7.3.3 KRB

The understanding of mineralization controls at KRB is at an early stage in light of the limited exploration campaign carried out to date. Gold appears to be associated with quartz veining and pyrite along the margins of felsic to intermediate dykes.

SECTION • 8 DEPOSIT TYPES

8.1 TOCANTINZINHO DEPOSIT

The Tocantinzinho deposit is best classified as a granite-hosted, intrusion-related gold deposit. Intrusion related gold deposits were defined by Thompson et al. (1998) and have the following characteristics:

- The deposits are hosted within or zoned proximal to intermediate to felsic granitic rocks.
- The intrusions are typically moderately reduced to moderately oxidized (ilmenite through to magnetite series) I-type granitic rocks.
- The associated pathfinder elements are typically Bi, Te, Mo, W in the core of the intrusion system, zoning outward to distal As, Sb, Pb and Zn.
- The deposits have a range in styles including sheeted, breccia, stockwork, flat-vein and disseminated to greisen that is controlled by proximity to intrusions, depth of emplacement and structural controls on intrusions.
- Mineralization is coeval with the related intrusions demonstrated through zonation with respect to the causative intrusion and mineralized magmatic-hydrothermal transition textures. These may include miarolitic cavities, vein dykes, unidirectional solidification textures, brain rock, and granite-facies control on Au distribution. In addition age dating in global examples has confirmed synchronicity between intrusions and mineralization.
- Mineralization is typically characterized by reduced, low sulfide (< 2%) ore assemblages including pyrite, pyrrhotite and arsenopyrite with magnetite less common and hematite rare. Proximal gold is typically high fineness and paragenetically related to Bi ± Te.
- Alteration is usually more limited than in typical porphyry environments, and characterized by early, high temperature quartz-feldspar (both potassic and sodic) alteration intimately associated with magmatic-hydrothermal transition textures that evolve to lower temperature white mica/sericite-carbonate-chlorite-quartz alteration and veining typically associated with the main mineralization stage.
- The ore formed from $H_2O-CO_2 \pm$ salt magmatic fluids
- The majority of known deposits are Phanerozoic in age and formed in continental arc to back arc settings.

Tocantinzinho shares many of these characteristics including:

- Fractionated granite host rock package (quartz monzonite, syenite, alkali feldspar granite, granite and aplite).
- Mineralized magmatic-hydrothermal transition textures including unidirectional solidification textures, interconnected miarolitic textures, rapid grain size variations from pegmatite to aplite and vein dykes, and granite facies control on Au-distribution (salami and smoky).

- Alteration assemblages with early (transitional magmatic-hydrothermal) potassic-sodic feldspar through to silicification and pervasive to vein controlled quartz-sericite-chlorite-calcite.
- Fluid inclusions contain $\text{H}_2\text{O}-\text{CO}_2 \pm \text{salt}$.

However, some features of the deposit are not typical of intrusion related gold systems including:

- Intrusion oxidation state: the distinctly red hematitic dusting to the feldspar suggests that the intrusions are more oxidized than typical intrusion related gold systems. However, the exact timing of the hematite is debatable. Late pink-red feldspar veins are common and strong hematitic alteration is associated with late stage breccias and fractures. Primary magnetite was clearly replaced by hematite in the hematitic granite on the margins of the deposit, and primary titanite-ilmenite is commonly observed in the Tocantinzinho granitic rocks indicative of moderately oxidized to moderately reduced primary magma compositions.
- Oxidation state of the mineralization: again the red-hematitic nature of the feldspar suggests more oxidized conditions for ore formation, however, the presence of pyrrhotite inclusions within pyrite and the Au-Bi association is not compatible with a strongly oxidized ore fluid.
- Age and timing: Borgo et al. (2017) presented U-Pb zircon dating on the Tocantinzinho intrusions and surrounding host rocks. The former are older (~2007 to 1997 Ma) than the main Tocantinzinho granitic suite, although in part overlap with error. The Tocantinzinho granitic rocks and mineralization are cut by late andesite and rhyolite dykes, and age data on the dykes (~1996 to 1992 Ma) overlap and are within error of the main host granite phases (~1993 to 1979 Ma). The age of the intrusions, and by inference the mineralization, is Paleoproterozoic which is unusual for intrusion related gold deposits and suggests that Tocantinzinho may be one of the oldest examples of this deposit type.

Biondi et al. (2018) further discuss the classification of Tocantinzinho comparing it to both reduced intrusion related gold and oxidized intrusion related gold (cf. Robert et al. 2007), and to Au-rich porphyry Cu deposits and orogenic Au deposits. The lack of Cu and extensive alteration associated with porphyry systems is inconsistent with that deposit group, however, Santa Patricia certainly has characteristics of porphyry Cu-Mo systems.

There has been much debate regarding the orogenic and intrusion related Au classification (Goldfarb et al., 2005). Tocantinzinho does not appear to fit an orogenic classification for several reasons despite the deposit and related intrusion occurring in a major regional structure. At a regional scale the deposit is not hosted in a metamorphic terrane and at the deposit scale the mineralization is controlled by granite-facies development and related veins and alteration rather than fault-related features. Furthermore, textural timing relationships and age data support a coeval timing for intrusions and mineralization.

8.2 SANTA PATRICIA

Santa Patricia is best described as a porphyry Cu-Mo system based on the following features:

- Highest Cu-Mo grades are associated with the most intense, multi-stage stockwork vein zones.
- The veins and alteration evolve from high (potassium K-feldspar-muscovite) to low temperature (sericite-calcite).
- The alteration is broadly zoned from K-feldspar and muscovite-rich alteration associated with the most intense stockwork centers through to an outer pyrite-sericite alteration to a distal propylitic alteration.

A porphyry exploration model offers potential upside to Santa Patricia. It is likely that the centre(s) of the potential porphyry system(s) have not been discovered. Evidence for this includes the lack of coeval porphyry related intrusions and the absence of abundant, early, irregular A-veins. Additionally, in the well mineralized zones, Cu grades appear to increase with depth and drilling to date has been limited to the upper 300 m.

SECTION • 9 EXPLORATION

9.1 EXPLORATION HISTORY

The exploration work at the Tocantinzinho Project completed to date includes geological mapping, channel and chip sampling, soil and stream sediment geochemical surveys, a detailed topography survey, auger drilling, geophysical investigations, limited reverse circulation drilling, and core drilling. Petrographic and metallurgical studies were conducted on drill core by contracted consulting firms.

Earliest systematic exploration in the project area was conducted by Altoro between 1997 and 1999. Altoro's work included soil sampling around the main garimpeiro (artisanal mining) works (over 700 samples), and collection of 476 channel samples of weathered bedrock from the garimpeiro pits. All of the sampled areas are now covered by either sandy tailings or water. Altoro also collected 6 km of magnetic data with one magnetometer along ten established grid lines, spaced 50 m apart. They drilled a total of 87 power auger holes (1,318 m) in 1998, and 58 additional holes (503 m) in 1999. The average saprolite intersection in all of the 145 holes was 9.1 m with an average grade of 1.0 g/t gold Au.

In early 2004, based on results of geochemical sampling, Brazauro (through its Brazilian subsidiary Jaguar Resources do Brazil Ltda.), initiated an exploratory core drilling program of 20 holes with an average length of 227 m per hole. Brazauro continued drilling until 2008, as described in Section 10. In addition to the drilling program, 106 power auger holes (934 m) were completed and over 500 channel/chip and soil samples were collected.

In September 2008, Eldorado became the operator of a joint venture with Brazauro and continued exploration with a focus on the main Tocantinzinho area until 2010. Soil (2,604), channel/chip (46) and dump (100) samples were also collected during this period.

Exploration activities between 2004 and 2010 established Tocantinzinho as a significant gold discovery and prompted the acquisition of Brazauro by Eldorado finalized on July, 2010.

A detailed topographic survey of the Tocantinzinho area was completed by Eldorado, corresponding to an area of 2.5 km², which covered the main pit area and other areas adjacent to the main mineralized zone. The coordinate system was based on one official and five implemented geodesic points at Tocantinzinho Project.

A topographic aerial laser survey of the project site was carried out in September, 2010 by Geoid Ltda. A total area of 53 km² was surveyed including the deposit, possible tailings dam areas and the future plant site. The contour interval was 1 m with an accuracy of approximately 0.15 cm in both the horizontal and vertical coordinates.

In late 2010, Eldorado completed an induced polarization (IP) geophysical survey of 45 km line, covering the areas along the Tocantinzinho trend to the northwest and southeast of the deposit. In 2011, aerial and ground magnetic survey data collected in 2005 were re-interpreted.

Eldorado expanded the tenement from 33,979 ha to 68,803.6 ha between 2012 and 2018 (Figure 9-1) as well as its soil sampling coverage as discussed below. Black outlines represent exploration claims. Red outlines represent mining concession.

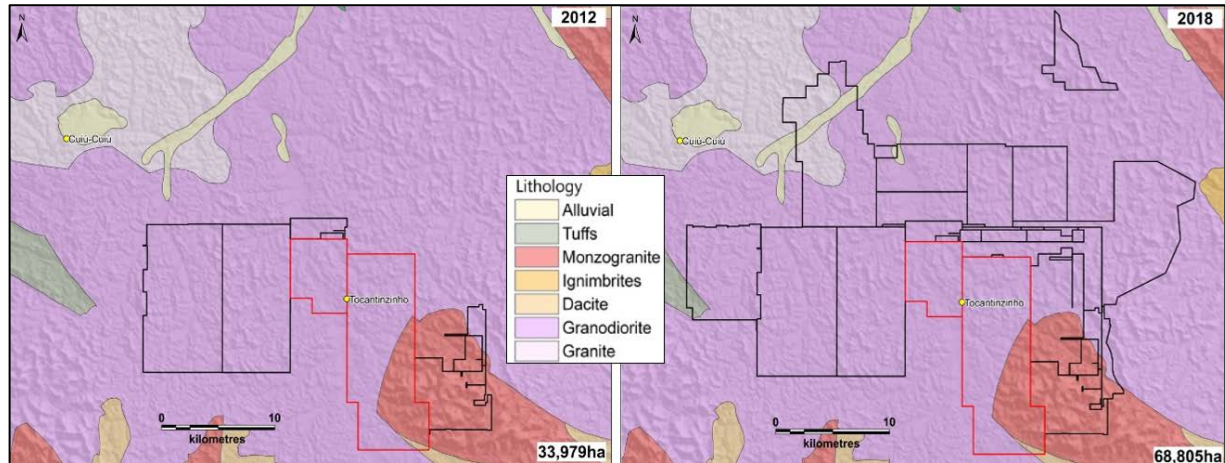


Figure 9-1: Tenement Area Expansion 2012 to 2018

9.2 SOIL GEOCHEMISTRY

Soil sampling was conducted at the Tocantinzinho Project by previous operators (Altora 1997 to 1999; Brazauro 2005 to 2007) mainly around the main garimpeiro pits with the main grid placed to cover the obviously mineralized zone. The surficial volume tested by these early surveys has now been almost completely excavated by the garimpeiros.

Eldorado carried out soil sampling campaigns from mid-2009 to early 2015 (Table 9-1). Initial focus between 2009 and 2011 was on the extensions of the main deposit area trend and adjacent areas. Further extensions of the main mineralized trend as well as testing of parallel trends were carried out between 2012 and 2015 (Figure 9-2).

Table 9-1: Summary of Soil Sampling Campaigns by Eldorado

Time Period	Samples
2009	2,109
2010	326
2011	2,241
2012	1,323
2013	1,578
2014	4,744
Project Total	12,321

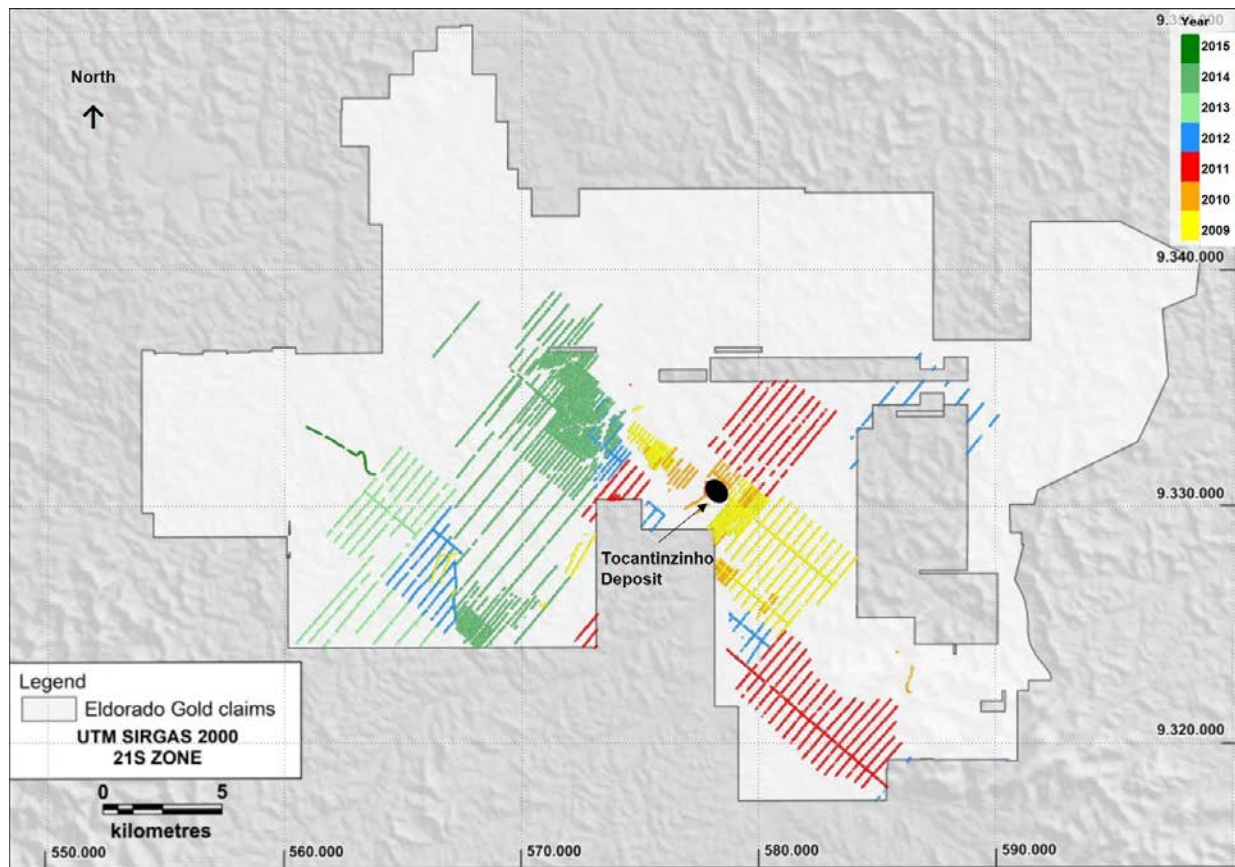


Figure 9-2: Location of Soil Sampling Campaigns by Eldorado

Soil samples were collected at 50 m intervals, using a hand auger with half-metre depth. Line spacing varied by area and campaign. Targets along-strike southeast of the Tocantinzinho deposit were tested with 100 m spaced lines. Targets along-strike northwest of the deposit was covered with 200 m spaced lines. The far southeast extension was tested with 400 m spaced lines. Other areas were initially sampled on 800 m spaced lines, progressively infilled to 100 m in case of positive results. The latter were generally considered to be spatially consistent values above 100 ppb gold (Figure 9-3). The Santa Patricia gold and copper anomaly to the northwest and the KRB anomaly to the southeast stand out. Outer border of exploration claims shown in black.

Soil sampling has been an effective method to delineate zones of gold and copper anomalies and the primary tool to generate drill targets. Approximately 40% of the total exploration package has been covered by soil sampling. Beyond the Tocantinzinho main anomaly and its strike extensions, two areas of soil anomalies stand out and are discussed separately below.

9.2.1 Santa Patricia

The trend of gold anomalies continues to the northwest of Tocantinzinho where it coincides with an 8 km long trend of copper and molybdenum anomalies marked by values >20 ppm (Figure 9-3 and

Figure 9-4). This consistent soil anomaly aligned with the regional mineralization trend defines the Santa Patricia exploration target and justified a scout drilling program described in Section 10. This soil anomaly is open to the northwest where there is no soil sampling coverage.

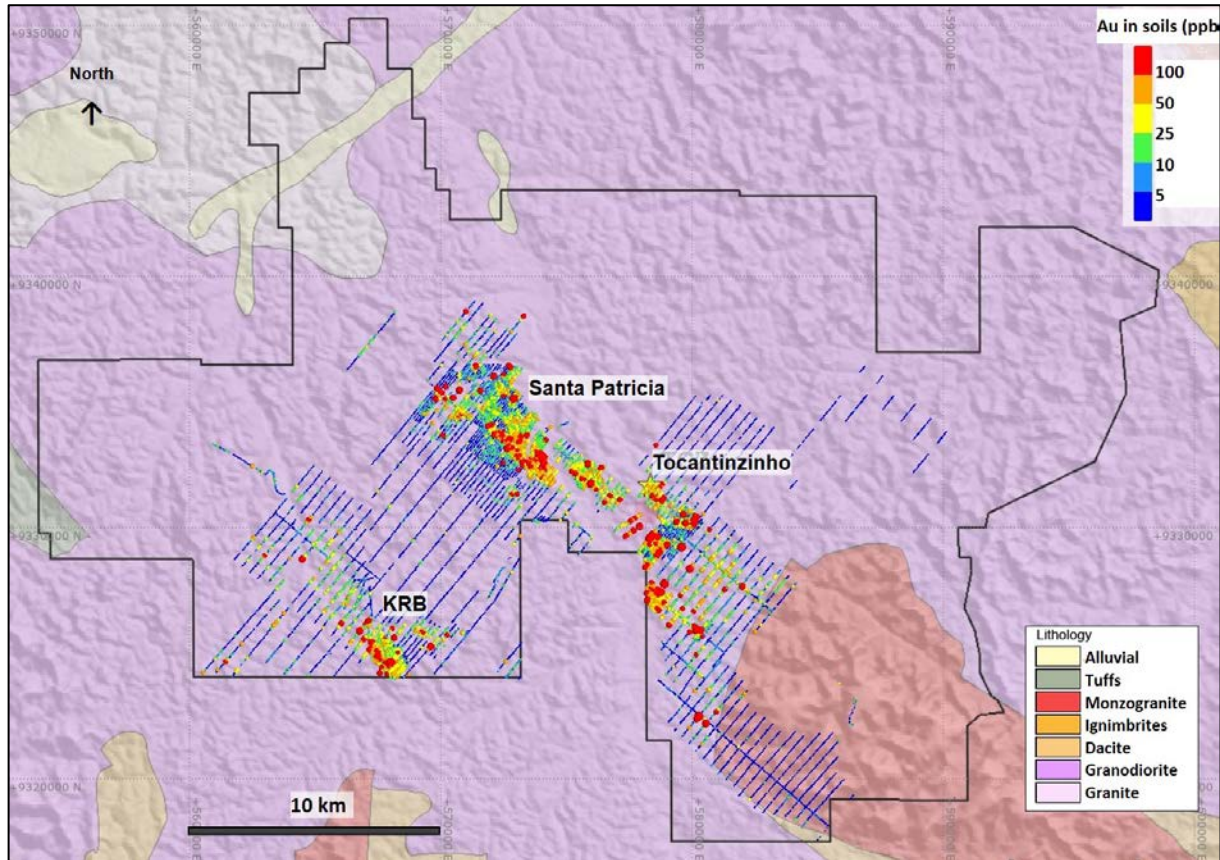


Figure 9-3: Gold Anomalies in Soil Sampling Campaigns by Eldorado

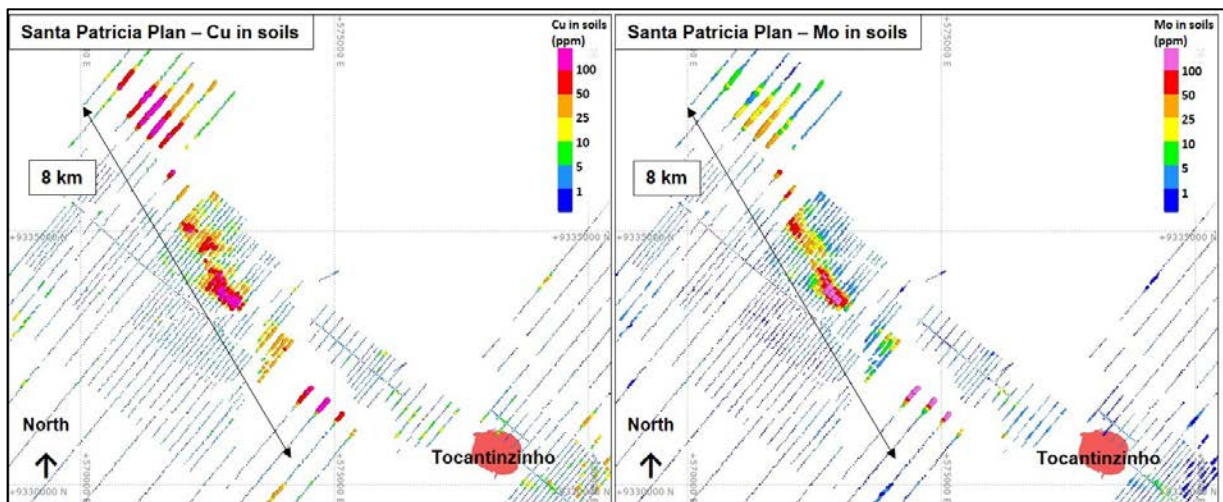


Figure 9-4: Santa Patricia coherent Copper and Molybdenum Soil Anomaly

9.2.2 KRB

The KRB exploration target is part of an 8.5 km long gold in soil anomaly to the southwest of Tocantinzinho, along a parallel northwest-southeast trend (Figure 9-3). This broader trend is marked by > 10 ppb values and includes a 2 km long core with values in excess of 80 ppb (Figure 9-5). Chip sampling in this area returned highly variable gold assays with most values close to detection limit but localized assays of up to 141 ppm. The KRB target was the focus of a scout drilling campaign described in Section 10.

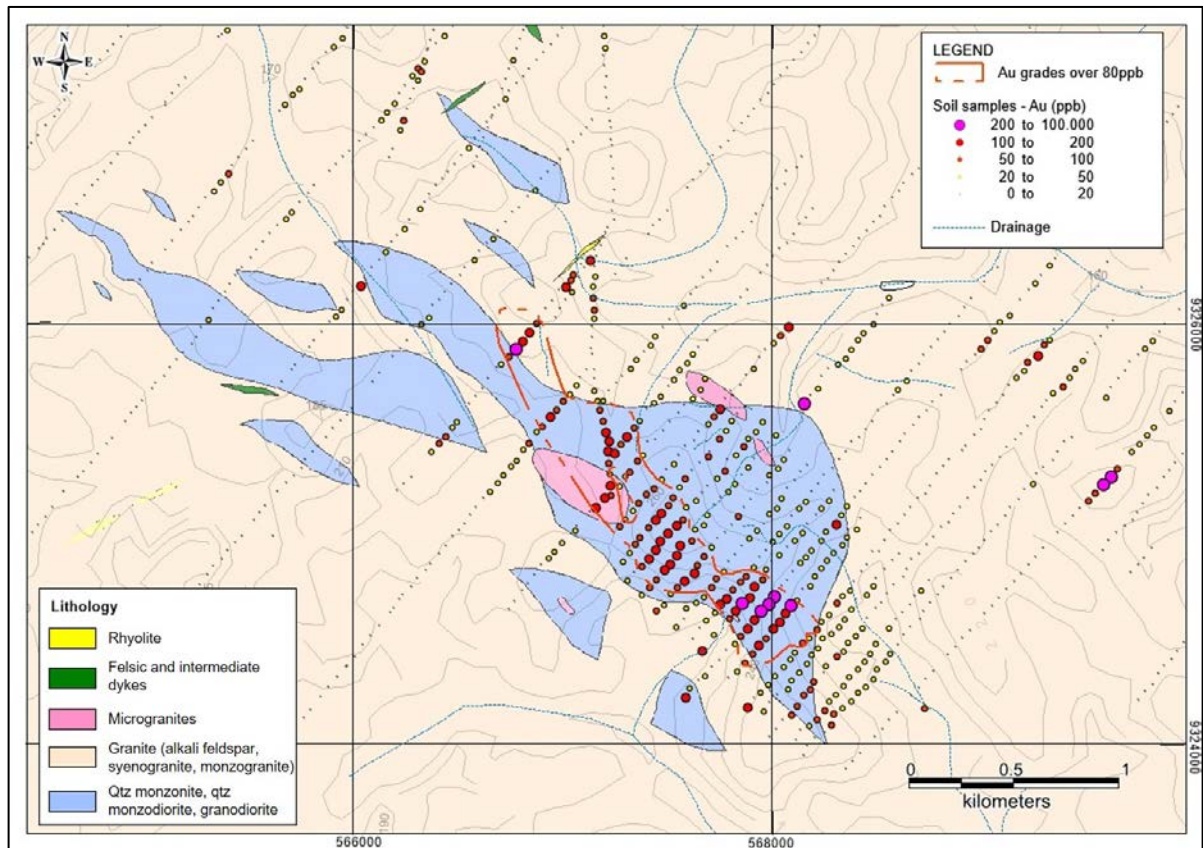


Figure 9-5: KRB Gold Soil Anomaly

9.3 GEOPHYSICS

At Tocantinzinho, geophysical surveys were carried out by the previous operators, Altoro and Brazauro. Altoro collected 6 km of magnetic data with a ground magnetometer survey along 50 m spaced grid lines.

In 2005 Brazauro hired Reconsult Geofísica to process and interpret the raw ground magnetic data and also to interpret geophysical airborne data collected by FUGRO covering the Tocantinzinho area. Reconsult concluded that the mineralization is probably related to the main Tocantinzinho trend structure (oriented N60W); and that mineralization is truncated to the SW by magnetic rock. Based

on the magnetic and radiometric data, Reconsult considered that there exists a strong potential for continuation of mineralization to the NW.

In late 2010, IP lines were completed at the Tocantinzinho Project. Figure 9-6 shows the lines superimposed on gridded gold in soil data. Results of the IP survey show that both chargeability and resistivity map out geology and structural breaks in the underlying rocks but are not effective in directly detecting Tocantinzinho-style gold mineralization.

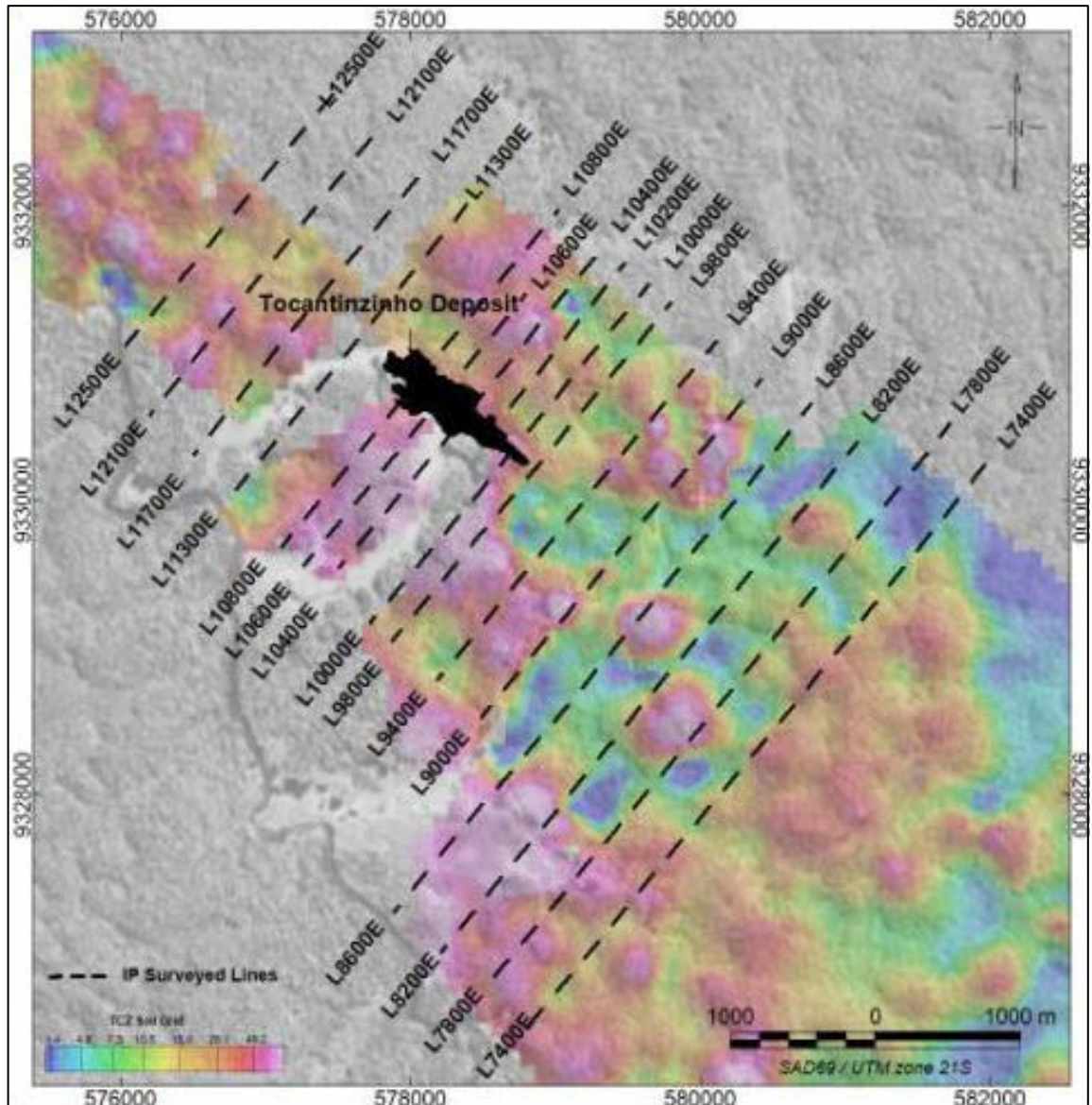


Figure 9-6: IP Geophysical Surveyed Lines

In 2011, S.J. Geophysics reviewed the magnetic airborne data collected in 2005. The original survey included a regional pass with survey lines spaced 400 m apart in the broader exploration land

package and an infill pass spaced 100 m apart centred on Tocantinzinho. The interpretation focused on establishing northwest and northeast structural trends and positioned Tocantinzinho inside a zone of low magnetic susceptibility (Figure 9-7).

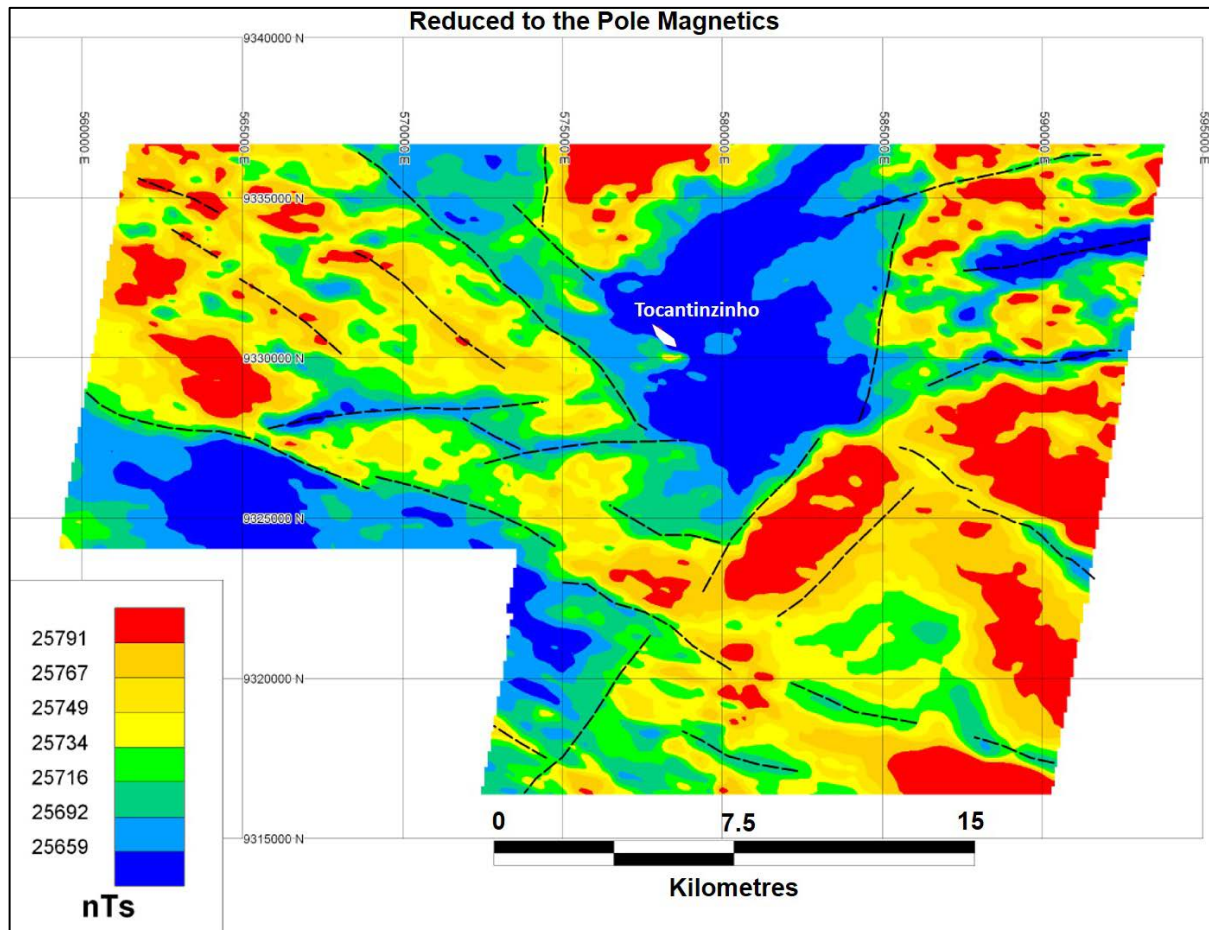


Figure 9-7: Reduced to the Pole Magnetic Survey

SECTION • 10 DRILLING

10.1 DIAMOND DRILLING

Diamond drill holes are the principal source of geological and grade data for the Tocantinzinho Project. A total of 82,805.50 diamond drill metres in 296 drill holes were completed inside the broader tenement in various phases between 2004 and 2015. The mineral resource estimate for the Tocantinzinho Project is directly supported by 45,039.30 metres drilled between 2004 and 2010. Metallurgical drilling was carried out in 2009 (1,490 metres), and drilling for geotechnical purposes was executed by Eldorado in 2010 (1,784.5 metres). Exploration drilling inside the broader package amounts to 34,491.80 metres drilled between 2004 and 2015.

The predecessor company Brazauro (through its Brazilian subsidiary Jaguar Resources do Brazil Ltda.) was the operator between 2004 and August 2008 completing a total of 25,635 metres (approximately 31% of total diamond drilling at the project). In September 2008, Eldorado became the operator of the project and started its first drilling program at Tocantinzinho. Table 10-1 summarizes the various drilling campaigns completed on the property. The locations of the drill holes for all phases are shown on a drill plan in Figure 10-1. A representative cross section through the mineral resource estimate volume is shown in Section 7.

Table 10-1: Project Core Drilling Summary

Operator	Purpose	Time Period	Drill Holes	Metres
Previous Operator	Exploration/ Resource Drilling	2004	TOC 04-01 to TOC 04-20	4,692.90
	Exploration/ Resource Drilling	2005	TOC 05-21 to TOC 05-34	3,758.90
	Exploration/ Resource Drilling	2006	TOC 06-35 to TOC 06-46	3,022.40
	Exploration/ Resource Drilling	2007	TOC 07-47 to TOC 07-71	5,763.20
	Exploration/ Resource Drilling	2008	TOC 08-72 to TOC 08-96	8,397.60
Eldorado	Exploration/ Resource Drilling	2008	TOC 08-97 to TOC 08-107	3,518.00
	Exploration/ Resource Drilling	2009	TOC 09-108 to TOC 09-152	4,633.20
	Metallurgy (twin drilling program)	2009	TOC 09-11TW, 35TW, 36TW, 48TW, 75TW and 102TW	1,490.00
	Exploration Drilling	2010	TOC171 to TOC179	2,016.40
	Geotechnical Drilling	2010	G-TOC001 to G-TOC006	1,784.50
	Resource Conversion Drilling	2010	TOC180 to TOC194	3,581.50
	Exploration Drilling	2011-2012	TOC195 to TOC269	3,488.40
	Exploration Drilling	2014-2015	TOC270 to TOC281; KRB01 to KRB14	6,658.70
TOTAL				82,805.50

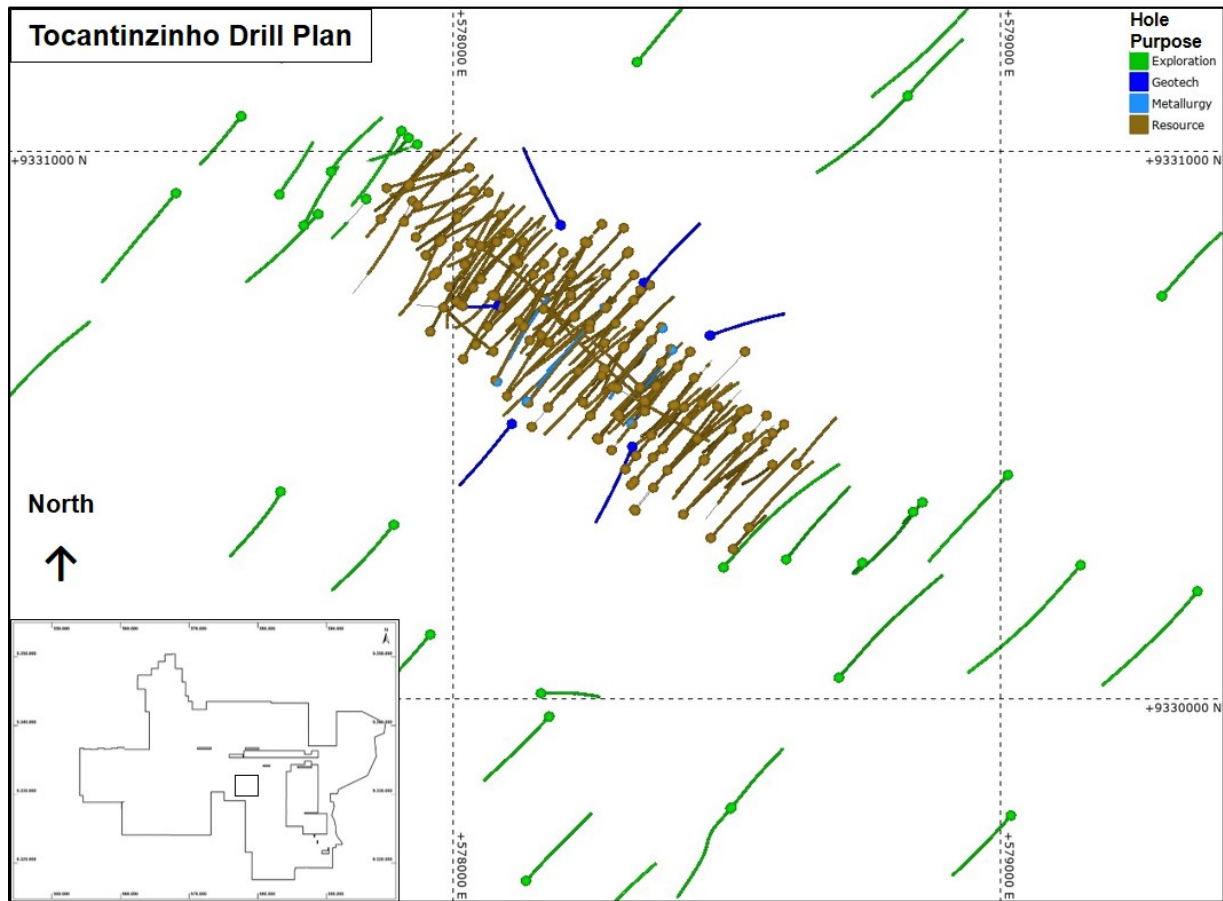


Figure 10-1: Tocantinzinho Deposit - Drill Plan

All diamond drilling by Brazauro and Eldorado were done by wire line method, conducted by Kluane International Drilling Inc. based in Vancouver, B.C, Canada. This drilling company provided light weight portable Hydrocore Gopher all-hydraulic drill rig capable of drilling about 350 m of BTW core during Brazauro's drilling phases and about 500 m during Eldorado's drilling phase.

The drilling phases executed by Brazauro were drilled with NTW-size (5.71 cm core diameter) and BTW-size (4.20 cm core diameter). In Eldorado's drilling phase more powerful drill rigs were available making it possible to drill deeper and with wider diameters (HQ-size 6.5 cm core diameter, NTW-size and BTW-size).

Drill holes collars were located using a total station instrument. Drill holes were drilled at inclinations ranging from 47° to 85° and azimuths generally due northeast or southwest. Inside the mineral resource estimate area, four of the holes were drilled parallel with the trend of the mineralization with the purpose of crosscutting the main sheeted veins trends at the optimal intermediate angle.

Down-hole survey deviations (azimuth and inclination) were taken approximately every 50 to 60 m using the Reflex EZ Shot instrument. The geotechnical drill cores were oriented using the ACT Reflex instrument. The infill holes followed the same procedure.

Standard conventions of logging and sampling were used to obtain information from the drill core. The core was photographed before being sampled and logged in detail onto paper logging sheets. The data were then reviewed and entered into the project database. Regular internal checks are conducted to assure the consistency of observations from hole to hole and between different loggers. In the mineralized units at the Tocantinzinho main deposits, core recovery was very good, averaging 95%.

10.1.1 Tenement-Wide Diamond Exploration Drilling

On July 2013, Eldorado presented the Economic Exploitation Plan to the DNPM to apply for the Mining Concession and Easement Concession at Tocantinzinho. According to Brazilian mining regulations, service or exploration drilling is restricted inside the Mining Concession Application while it is in process. As a result, drilling at the project shifted to tenement-wide exploration in the period between 2014 and 2015. The exploration drilling followed up on the spatially significant and coherent Cu-Mo soil anomaly at Santa Patricia and the Au anomaly at KRB. The drilling restriction at Tocantinzinho was lifted with the granting of the Mining Concession in May 2018.

10.1.1.1 Santa Patricia

Diamond drilling at Santa Patricia was conducted over two phases: An initial phase in 2012 comprised of four drill holes for a total of 1,331 metres and a second phase in 2014 consisting of 3,593.9 m in twelve drill holes. These two phases represent initial efforts in testing the 8-km long copper and molybdenum soil anomalies and are divided in four clusters (Figure 10-2). The northern edge of the soil anomaly remains untested by drilling.

Within individual clusters, drill hole section spacing is approximately 200 metres. Drill hole location and downhole surveys are presented in Table 10-2; significant intercepts are listed in Table 10-3. True mineralization thickness is unknown. In the southern cluster, drill holes TOC266, TOC267, TOC272 and TOC275 have ended in copper mineralization. In the two latter drill holes, the trend of increasing Cu grades downhole is noteworthy (Figure 10-2 insert 2). The southern cluster is approximately 3 km WNW of the main Tocantinzinho deposit.

Drilling results at Santa Patricia are encouraging and suggest the relationship between rock-types alteration and mineralization can be explained under a porphyry exploration model. Such a model offers potential upside to Santa Patricia as it is likely that the centre(s) of the potential porphyry system(s) have not been discovered. Evidence for this includes the lack of coeval porphyry related intrusions and the absence of abundant, early, irregular A-veins. Additionally, in the well mineralized zones Cu grades increase with depth and drilling to date has been limited to the upper 300 m.

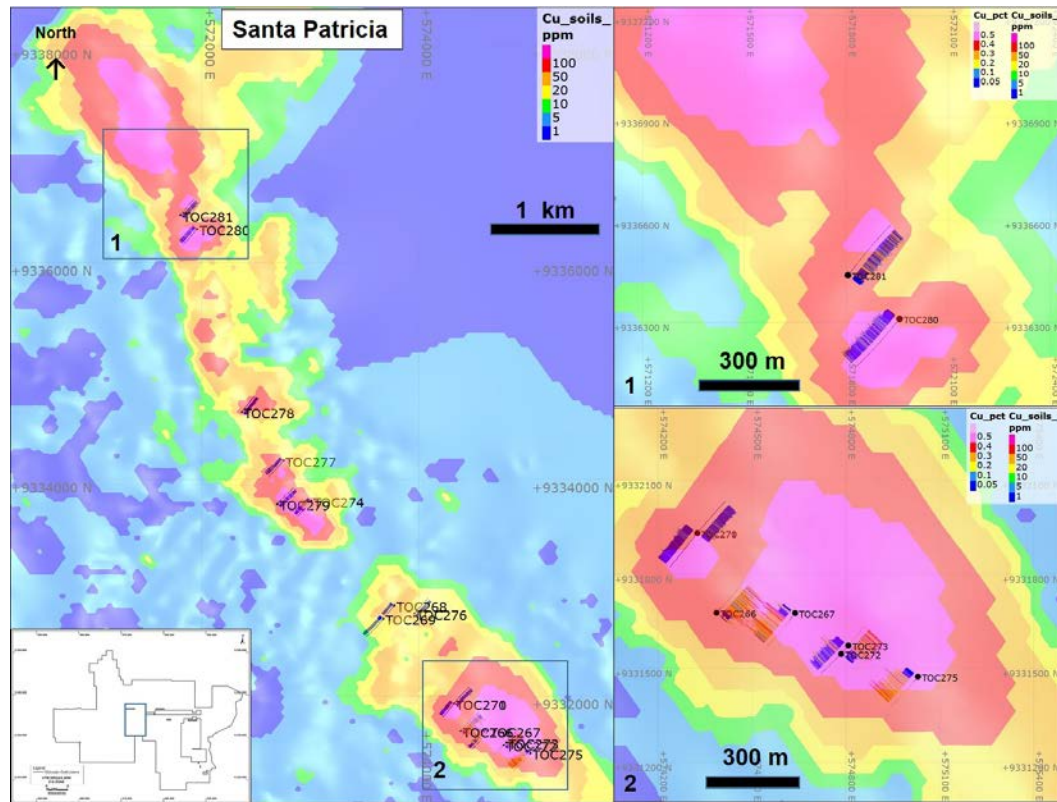


Figure 10-2: Santa Patricia Drill Plan with respect to the larger copper soil anomaly

Table 10-2: Santa Patricia Drill Location and Orientation

Hole ID	Hole Type	Length	Easting	Northing	Elevation	Dip	Azimuth
		m	m	m	m	°	°
TOC266	DDH	384.00	574387.30	9331675.40	179.10	-53.50	42.80
TOC267	DDH	354.00	574638.00	9331674.50	169.30	-51.50	218.30
TOC268	DDH	207.00	573774.70	9332834.60	163.90	-52.00	218.00
TOC269	DDH	366.00	573675.40	9332713.10	171.00	-50.90	220.20
TOC270	DDH	337.70	574325.60	9331929.30	170.00	-50.00	40.00
TOC271	DDH	250.90	574326.00	9331929.00	170.00	-50.00	220.00
TOC272	DDH	309.60	574785.00	9331544.00	148.00	-50.00	40.00
TOC273	DDH	231.80	574808.00	9331571.00	148.00	-50.00	220.00
TOC274	DDH	301.40	573010.00	9333800.00	140.00	-50.00	220.00
TOC275	DDH	344.70	575030.00	9331472.00	150.00	-50.00	220.00
TOC276	DDH	286.70	573960.00	9332760.00	160.00	-50.00	40.00
TOC277	DDH	306.50	572755.00	9334172.00	150.00	-50.00	220.00
TOC278	DDH	294.80	572370.00	9334620.00	180.00	-50.00	40.00
TOC279	DDH	330.40	572697.00	9333770.00	180.00	-50.00	40.00
TOC280	DDH	299.10	571956.00	9336309.00	157.00	-50.00	220.00
TOC281	DDH	300.40	571804.00	9336437.00	192.00	-50.00	40.00

Table 10-3: Santa Patricia Significant Intercepts

Hole ID	From	To	Interval	Cu	Mo	Au
	m	m	m	%	%	ppm
TOC266	41.00	335.80	294.80	0.190	0.0150	-
including	47.65	82.50	34.90	0.290	0.0210	-
and	114.00	131.20	17.20	0.250	0.0370	-
and	140.80	165.50	24.70	0.280	0.0180	-
TOC267	46.75	211.50	164.80	0.220	-	-
including	46.75	82.00	35.30	0.330	0.0060	-
TOC270	170.15	190.80	20.70	0.030	0.0100	0.14
TOC270	251.90	298.70	46.80	0.080	0.0130	-
including	266.10	267.20	1.10	0.740	0.0100	0.58
and	326.06	326.80	0.70	0.110	0.0010	0.61
TOC271	0.00	250.90	250.90	0.020	0.0050	-
TOC272	107.09	309.60	202.47	0.250	0.0160	-
including	228.00	279.90	51.85	0.320	0.0160	-
TOC273	42.86	69.60	26.74	0.130	-	-
TOC273	143.58	152.40	8.77	0.150	-	-
TOC274	54.50	145.20	90.70	0.110	0.0020	-
TOC274	227.86	242.20	14.29	0.140	0.0150	-
TOC275	71.43	344.70	273.22	0.220	0.0090	-
including	270.86	292.30	21.44	0.270	0.0040	-
and	310.25	320.50	10.25	0.490	0.0050	-
TOC276	0.00	14.60	14.56	0.000	-	0.12
TOC276	29.45	43.80	14.35	0.000	-	0.12
TOC276	106.58	137.60	30.97	0.010	-	0.18
TOC276	205.75	231.40	25.60	0.070	0.0100	-
TOC277	153.90	212.90	58.95	0.060	0.0020	-
TOC277	268.80	293.00	24.20	0.090	0.0040	-
TOC278	211.96	222.50	10.52	0.070	0.0150	-
TOC279	75.50	216.80	141.27	0.070	0.0040	-
TOC279	247.20	258.10	10.90	0.090	0.0040	-
TOC280	55.67	299.10	243.39	0.060	0.0040	-
TOC281	65.88	300.50	234.60	0.070	0.0030	-

True mineralization thickness is unknown.

10.1.1.2 KRB

Diamond drilling at KRB was carried out in 2015 and represents the most recent exploration campaign at the project. The 3,064.80 m campaign was designed to test the 2 km long core of a gold soil anomaly with values in excess of 80 ppb (Figure 10-3). This soil anomaly is located approximately 12 km SW of the main Tocantinzinho deposit, in a parallel NW-SE trend.

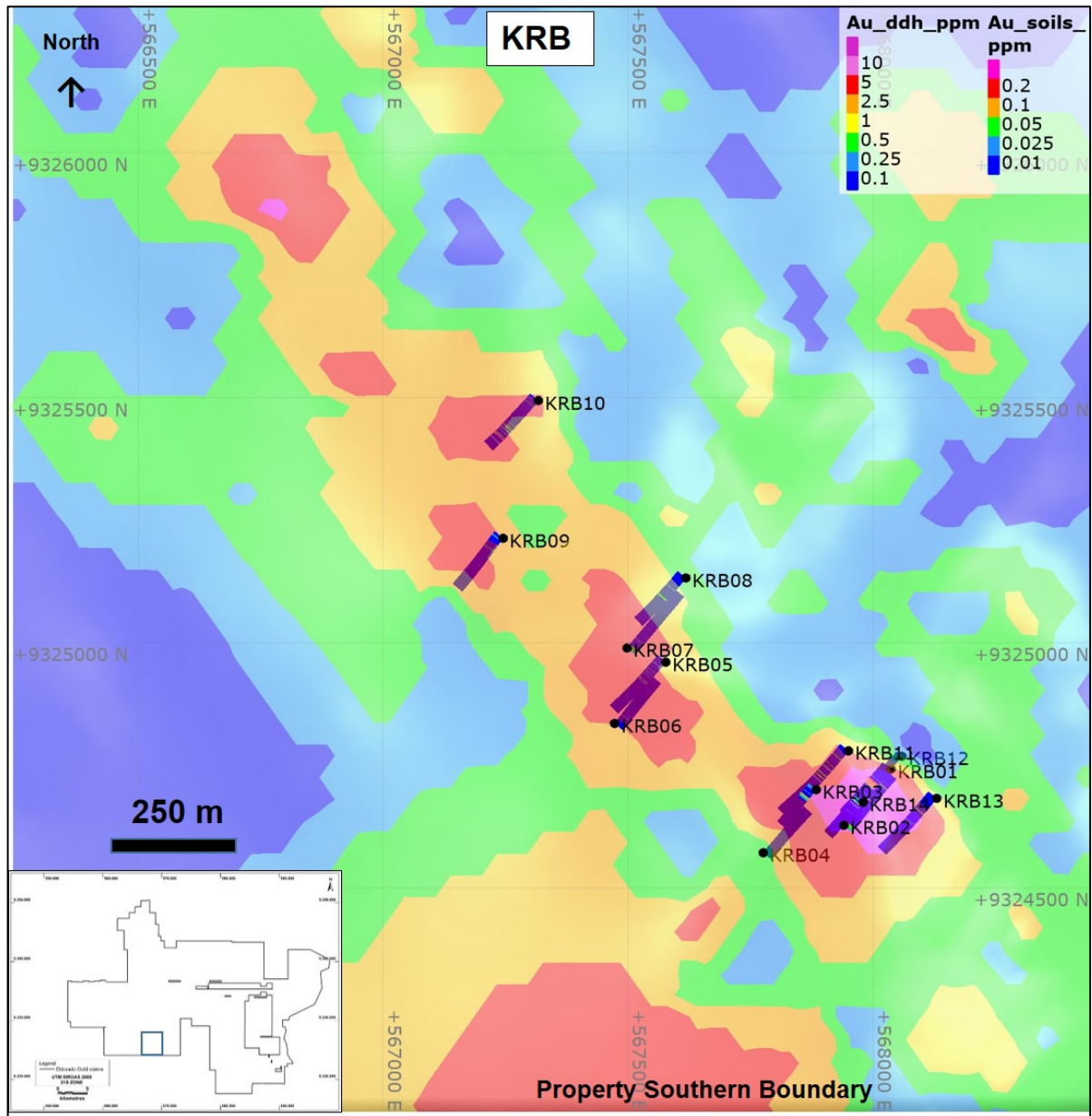


Figure 10-3: KRB Drill Plan with respect to the larger gold soil anomaly

KRB drill hole location and downhole surveys are presented in Table 10-4; significant intercepts are listed in Table 10-5. Drilling focused on the zones of higher gold in soils and employed scissor-style drill hole orientations. True mineralization thickness is unknown. Drilling intersected two separate NW-SE corridors but grade continuity is poor over the 100 m spacing between sections. The northern edge of the soil anomaly remains untested by drilling.

Table 10-4: KRB Drill Location and Orientation

Hole ID	Hole Type	Length	Easting	Northing	Elevation	Dip	Azimuth
		m	m	m	m	°	°
KRB01	DDH	202.80	568037.0	9324742.00	180.00	-50	220
KRB02	DDH	216.60	567940.60	9324627.10	222.00	-50	40
KRB03	DDH	173.50	567884.20	9324700.00	218.00	-55	220
KRB04	DDH	280.90	567776.00	9324571.00	148.00	-50	40
KRB05	DDH	218.10	567577.00	9324960.00	268.00	-50	220
KRB06	DDH	184.70	567472.40	9324835.00	230.00	-50	40
KRB07	DDH	241.00	567497.90	9324988.80	268.00	-50	40
KRB08	DDH	201.30	567618.00	9325132.00	262.00	-50	220
KRB09	DDH	200.50	567245.00	9325213.00	215.00	-50	220
KRB10	DDH	201.30	567317.00	9325494.00	215.00	-50	220
KRB11	DDH	271.40	567950.00	9324779.00	219.00	-50	220
KRB12	DDH	286.70	568059.00	9324768.00	177.00	-65	220
KRB13	DDH	235.50	568130.00	9324682.00	200.00	-50	220
KRB14	DDH	150.50	567980.00	9324674.00	221.00	-50	220

Table 10-5: KRB Significant Intercepts

Hole ID	From	To	Interval	Au
	m	m	m	ppm
KRB01	63.50	83.70	20.30	1.73
including	66.30	67.10	0.80	12.80
KRB03	40.50	45.20	4.70	2.53
including	42.80	43.58	0.80	6.98
KRB10	158.10	160.90	2.80	3.71
including	158.10	159.00	1.00	9.71

10.2 REVERSE CIRCULATION DRILLING

From October 2009 to February 2010 a reverse circulation drilling program was carried out with 19 holes totaling 8,452 m. These holes were drilled for exploration purposes in adjacent areas to the Tocantinzinho main deposit and were not used in the mineral resource estimate.

10.3 POWER AUGER DRILLING

Several power auger drilling campaigns have been carried out at the Tocantinzinho Project with the goal of obtaining samples of less weathered material deeper in the weathering profile than is possible with a conventional soil sample (Table 10-6). Auger samples were not used in the mineral resource estimate.

Power auger holes carried out by Eldorado were logged following the standard conventions established in Tocantinzinho Project and sampled at one metre intervals. Most of the holes were stopped at shallow depth because of reaching the water level. Average hole length was 11.2 m.

Table 10-6: Project Power Auger Drilling Summary

Purpose	Time Period	# of Holes	Meters	Company
Exploration	1998	87	1,318.00	Altoro
Exploration	1999	58	503.00	Altoro
Exploration	2008	106	934.00	Brazauro
Exploration	2009	112	416.00	Eldorado
Exploration	2011	237	3,065.00	Eldorado
Exploration	2012	108	1,491.00	Eldorado
Exploration	2013	61	874.00	Eldorado
Exploration	2014	47	582.00	Eldorado
Exploration	2015	16	63.00	Eldorado
Total		832	9,246.00	

The advantages of the power auger are that it is easily mobilized to the site and it is capable of being maintained and operated by local personnel. The limitations of the power auger sampling are that only vertical holes are possible and therefore samples are not obtained across geological features which are subvertical at Tocantinzinho. The depth limitation of the drill does not permit the sampling of fresh rock.

10.4 DIAMOND DRILLING LOGGING AND SAMPLING

The drill core was retrieved each shift from the drill site and brought to the camp site where the percent recovery and intervals are marked. The core was photographed on a tow box vertical stand and logged for geotechnical information including percent recovery, rock quality designation (RQD), joint frequency and condition, degree of breakage and weathering/alteration.

The geologist logged the full core and produced a Summary Log, recording lithology, alteration type and grade, texture, structure, observed sulphides as well as a brief description of important features. All the collected information was entered in the digital database (Datashed). While logging, the geologist measured and marked the intervals to be sampled, making sequential sample divisions and numbers in the core. An attempt was made to make two metre long sample intervals, diverging from these intervals for geological reasons, such as rock type contacts.

The core was then cut in half lengthwise, by means of a rock saw, flushed regularly with fresh water. To minimize sampling bias, the core was marked with a continuous linear cutting line before being split. Both halves of the core are placed back into the core-box. Once the entire hole core or a long section of the core has been cut, the geologist made a detailed description of each sample lithology, veining, alteration, mineralogy and record in the sampling form.

Two trained geotechnicians placed half of the core into new sample bags and clearly mark the interval, on the ribs of the core box, with the interval footages and sample number. The samples were consistently taken from the same side of the core. The bagged sample was marked, tagged and sealed for shipping to the laboratory. Groups of bagged samples were placed in larger sacks that are marked, showing the sample numbers.

Until hole TOC-09-123, samples were shipped to the SGS Geosol laboratory at Itaituba/PA for sample preparation. Samples were then shipped internally by SGS to their laboratory in Belo Horizonte/MG for analysis. From drill hole TOC-09-124 to TOC-269, samples were air shipped to ALS Chemex laboratory in Vespasiano/MG. From TOC-270 to TOC-281, samples were shipped to the ACME Laboratory at Itaituba/PA for sample preparation. Samples were then shipped internally by ACME to their analysis laboratory. From holes KRB-01 to KRB-14, the samples were air shipped to ALS Chemex laboratory in Vespasiano/MG.

SECTION • 11**SAMPLE PREPARATION, ANALYSES
AND SECURITY**

The mineral resource estimate is based on drill samples completed before December 2010, and sampling for this period is described in the QA/QC Program section. Exploration drilling beyond 2011 was focused on new targets which are not part of the mineral resource estimate with the corresponding sampling for later exploration programs discussed in the Samples Supporting the Exploration Results section.

**11.1 SAMPLES SUPPORTING THE MINERAL RESOURCE
ESTIMATE**

During the period of Eldorado operatorship, split core samples were prepared for analysis at the ALS Chemex Laboratories (ALS) facility in Vespasiano, Minas Gerais state in Brazil. The samples were prepared according to the following protocol:

- The entire sample is crushed to 70% passing 2 mm.
- A 1 kg subsample is riffle split from the crushed passing 2 mm sample and pulverized to 85% passing 75 µm (200 mesh).
- A subsample with up to 250 g is selected by riffle splitting from the pulverized 75 µm sample
- A 30 g sample is taken for fire assay analysis.

The sample batches were arranged to contain regularly inserted control samples. A standard reference material (SRM), a duplicate and a blank sample were inserted into the sample stream at every 10th to 15th sample. The duplicates are used to monitor precision, the blank sample can indicate sample contamination or sample mix-ups, and the SRM is used to monitor accuracy of the assay results.

All samples were assayed for gold by 30 g fire assay with an atomic absorption (AA) finish. A gravimetric finish was performed on fire assays returning more than 10 g/t gold. Samples with visible gold were submitted to a metallic screen analysis under ALS protocols (Au-SCR21 method).

11.1.1 QA/QC Program

Assay results were provided to Eldorado in electronic format and as paper certificates. Upon receipt of the assay results, values for standard reference materials (SRMs) and field blanks are tabulated and compared to the established SRM pass-fail criteria:

- Automatic batch failure if the SRM result is greater than the round-robin limit of three standard deviations.
- Automatic batch failure if two consecutive SRM results are greater than two standard deviations on the same side of the mean.
- Automatic batch failure if the field blank result is over 0.05 g/t Au.

If a batch fails, it is re-assayed until it passes. Override allowances are made for barren batches. Batch pass/failure data are tabulated on an ongoing basis, and charts of individual reference material values with respect to round-robin tolerance limits are maintained.

11.1.1.1 Blank Sample Performance

Assay performance of field blanks is presented in Figure 11-1 for gold. The analytical detection limit (ADL) for gold is 0.005 g/t. The rejection threshold was chosen to equal 0.05 g/t (Figure 11-1). The results show no evidence of contamination. Rare higher values were investigated and found to be caused by sample mix-ups.

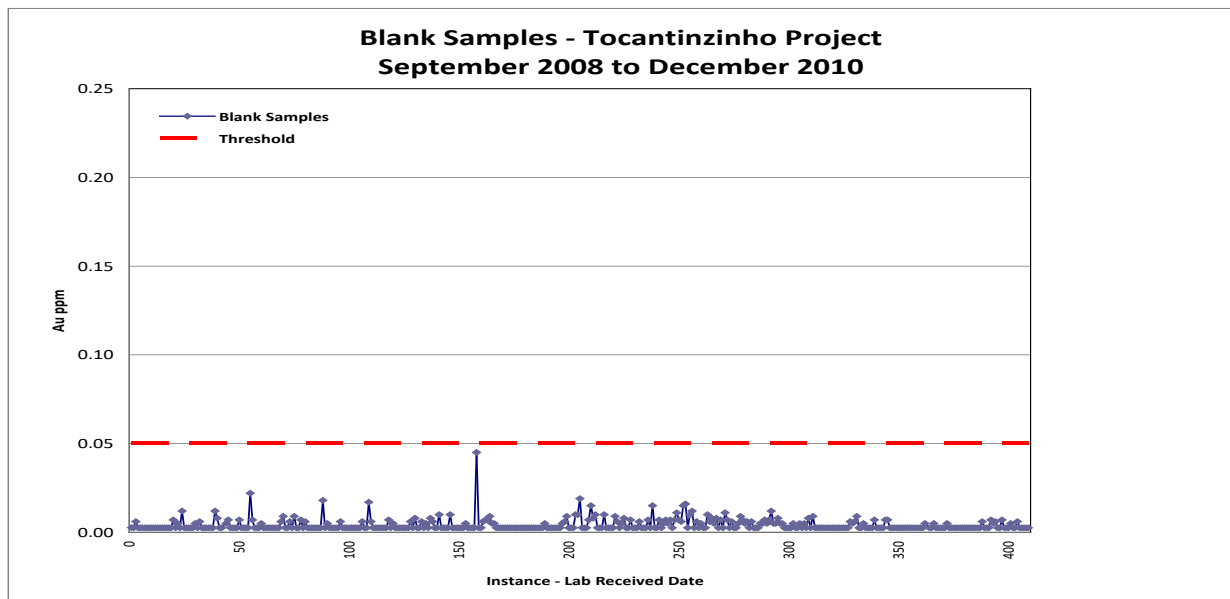


Figure 11-1: Tocantinzinho Blank Data – 2008 to 2010

11.1.1.2 Standards Performance

Eldorado strictly monitored the performance of the SRM samples as the assay results were reported. Seven SRM samples were used, covering a grade range between 0.8 g/t to 13.6 g/t. Examples of individual SRMs performance are shown as charted data in Figure 11-2 to Figure 11-5. All samples are given a fail flag as a default entry in the project database. Each sample is re-assigned a date-based pass flag when assays have passed acceptance criteria. All data used in the updated resource estimate had passed acceptance criteria.

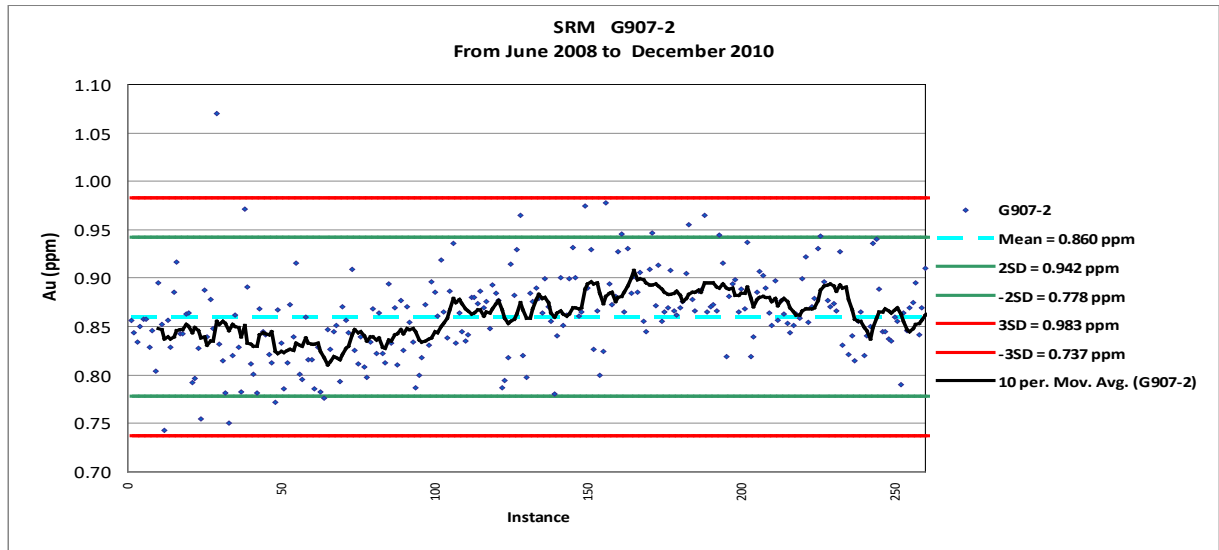


Figure 11-2: Standard Reference Material Chart, SRM G907-2

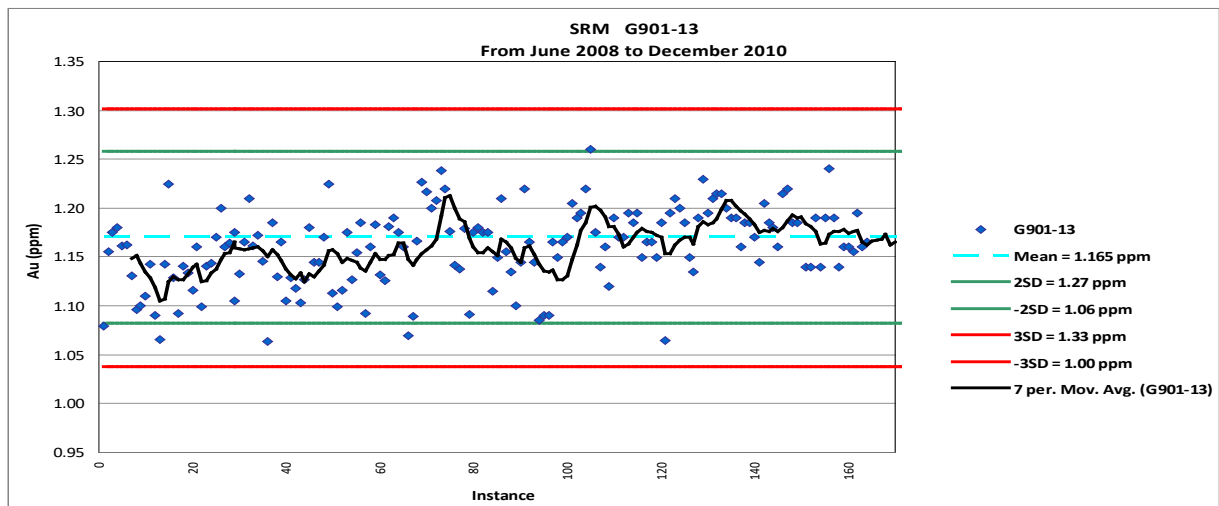


Figure 11-3: Standard Reference Material Chart, G901-13

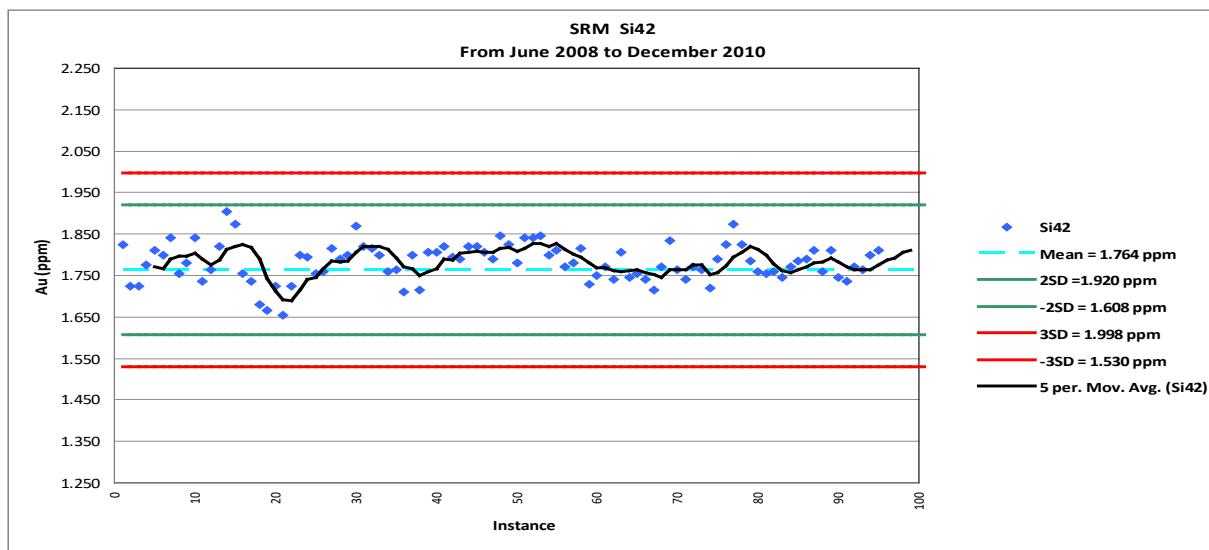


Figure 11-4: Standard Reference Material Chart, SRM Si42

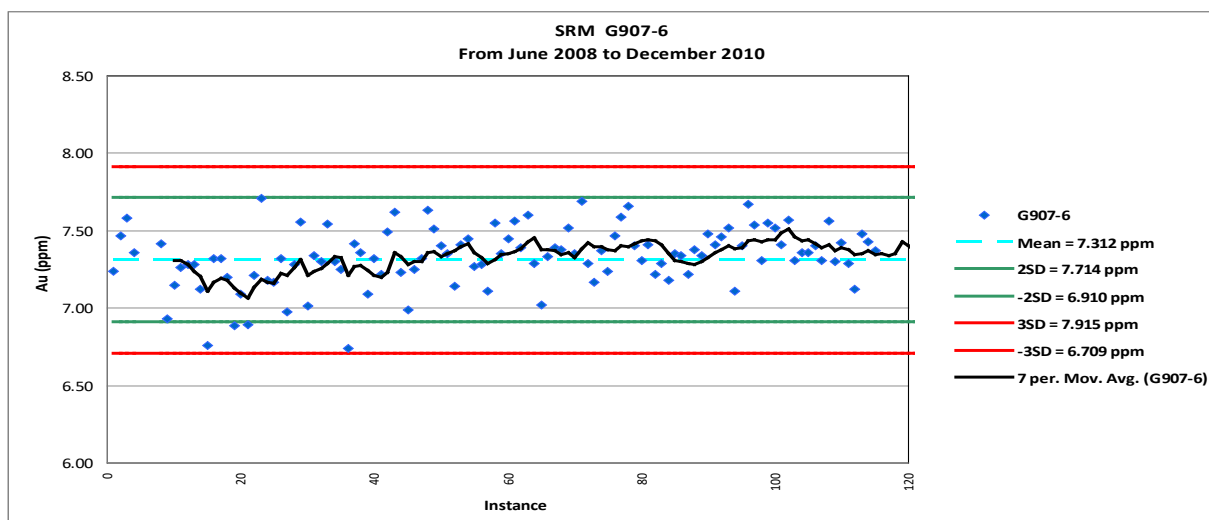


Figure 11-5: Standard Reference Material Charts, G907-6

11.1.1.3 Duplicate Performance

Eldorado implemented and monitored regularly submitted field duplicates, the other half of the core sample. These data reproduced only fairly well. The field duplicate data are shown in a relative difference chart in Figure 11-6. The large scatter around the suggested maximum difference of $\pm 30\%$ shows the effect of heterogeneous distribution of the gold mineralization at the core sample level. A better measure of the laboratory precision is given by the laboratory pulp duplicate data, shown in Figure 11-7. Results show the pulp data reproduced moderately well though still influenced by likely

liberated gold during the sample preparation process. Patterns in both charts symmetric about zero suggesting no bias in the assay process.

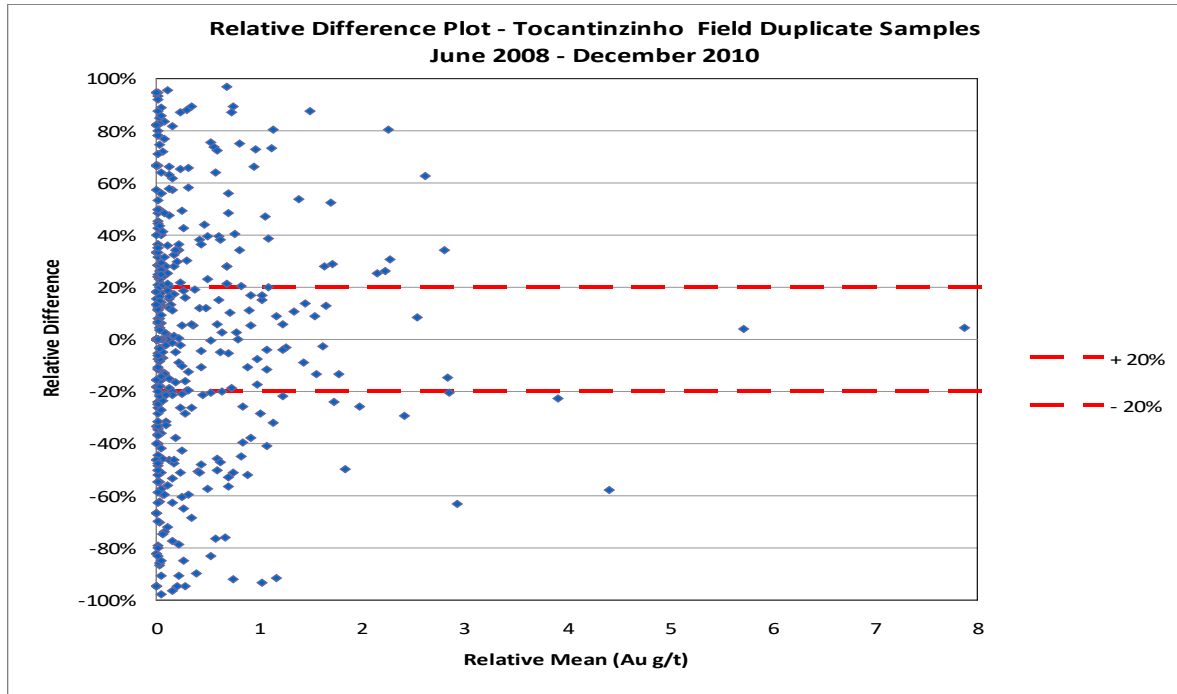


Figure 11-6: Relative Difference Plot of Tocantinzinho Field Duplicate Data, 2008 to 2010

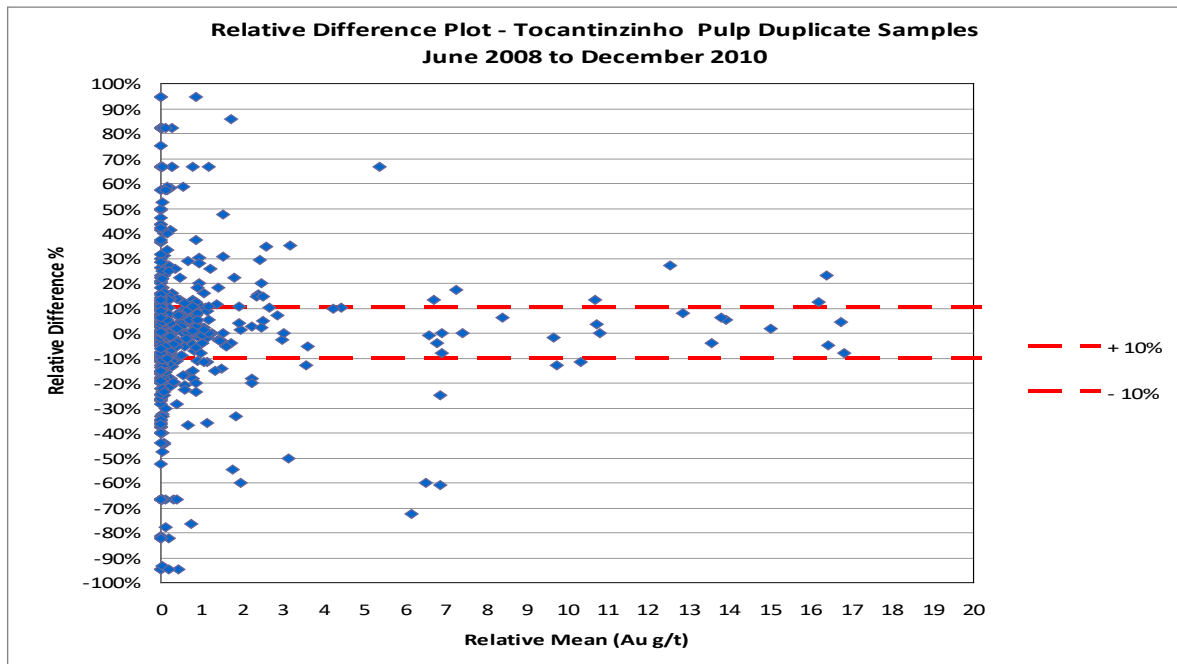


Figure 11-7: Relative Difference Plot of Tocantinzinho Pulp Duplicate Data, 2008 to 2010

11.1.1.4 Specific Gravity Program

A total of 872 samples taken from core holes were being measured for specific gravity and tabulated by rock type. Also 59 saprolite samples were selected and measured for specific gravity. The specific gravity for non-porous samples (the most common type) is calculated using the weights of representative samples in water (W2) and in air (W1). The bulk density is calculated by $W1 / (W1 - W2)$. Averages for key rock types are: 2.62 for the ore granites; 2.75 for the andesite; 2.68 for the quartz monzonite; 2.61 for the hematite granite; and 1.80 for the saprolite.

11.2 SAMPLES SUPPORTING THE EXPLORATION RESULTS

The vast majority of samples from the exploration drilling campaign carried out at the project in the period between 2011 and 2015 were assayed at the same ALS facility with substantially the same preparation, assaying and QA/QC procedures described above. The performance of blanks, standards and duplicates was monitored in a manner consistent with the discussion above.

Approximately 10% of exploration samples from this period were shipped to the ACME Laboratory at Itaituba/PA for sample preparation. Samples were then shipped internally by ACME to their analysis laboratory. Sampling preparation procedures were substantially the same as those employed by ALS except for the initial crush which was 80% passing 2 mm.

Multi-element geochemistry was performed systematically in the exploration campaigns between 2011 and 2015. At both ALS and ACME, multi-element analyses were done by inductively coupled plasma atomic emission spectroscopy (ICP-ES) using aqua regia digestion. At ALS, a suite of 35 elements were analysed while ACME employed a 33 element package. Approximately 50% of all diamond drill samples in the property were analysed for multi-elements.

11.3 ADEQUACY STATEMENT

In Eldorado's opinion, the QA/QC results demonstrate that the Tocantinzinho deposit assay database is sufficiently accurate and precise for resource estimation and the disclosure of exploration results.

SECTION • 12 DATA VERIFICATION

Checks to the entire drill hole database were undertaken. Comparisons of the digital database were made to original assay certificates and survey data. Any discrepancies found were corrected and incorporated into the current resource database. Eldorado concluded that the data supporting the Tocantinzinho Project resource work is sufficiently free of error to be adequate for estimation and disclosure of exploration results.

SECTION • 13

MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

This section provides a description of metallurgical test work, analysis of the results and comments on the test work program for the gold deposit, Tocantinzinho, carried out from 2001 to 2017.

The following metallurgical test facilities were involved with the test work program:

- Hazen Research, Golden, CO, USA
- SGS Mineral Services, Lakefield, ON, Canada
- Wardell Armstrong International, Cornwall, UK
- FLSmidth Knelson, Langley, BC, Canada
- FLSmidth UK, United Kingdom
- Bruce Geller, Advanced Geologic Services, Lakewood, ON, Canada
- CyPlus GmbH, Hanau-Wolfgang, Germany
- Ralph Meyertons Consulting, Nederland, CO, USA
- Federal University of Minas Gerais, Minas Gerais, Brazil

13.2 METALLURGICAL TEST WORK

The metallurgical test work completed for the Tocantinzinho Project included the following tests:

- Ore variability in terms of lithology, gold head grade, sulfur head grade, depth, and sample blending
- Metallurgical test work for primary sulfide ore, gold bearing soil, saprolite, transitional and artisanal mining (garimpeiros) tailings
- Detailed chemical analyses of ore feeds, flotation concentrates and flotation tailings
- Ore mineralogy and characteristics assessment
- Comminution testing including Bond crushing, rod milling, and ball milling indices; SMC index, and abrasion index
- Whole ore cyanide leach and cyanide leach of flotation concentrates
- Flotation including batch rougher and cleaner, locked cycle, and pilot plant
- Gravity recoverable gold
- Thickening testing of ore feed, flotation concentrate, leached residue and flotation tailing
- Cyanide detoxification (several methods) and aging test work on tailings and effluent
- Environmental and geotechnical testing of residue

The pilot plant flotation and cyanide test work were completed by Wardell Armstrong International in UK. The gravity test work completed by FLS Knelson in Canada and the tailings and cyanide destruction test work was latest carried out by SGS Mineral Services in Canada.

13.3 COMPOSITE AND SAMPLE SELECTION

The description of the samples as per the profile strata for individual and composite testing are summarized in Table 13-1.

Table 13-1: Description of Samples in Test Work Program

Sample Description	Feature
SMKG	Smoky Granite
SMIG	Salami Granite
TOP	Top half orebody
BOT	Bottom half orebody
ALL	Overall composite
S1 ¹	Top 100 m
S2	Middle 100 m
S3	Bottom 100 m
S4	Top 100 m
S5	Middle 100 m
S6	Bottom 100 m
S7	Low Gold Grade
S8	Average Gold Grade
S9	High Gold Grade
S10	Low Sulfur Grade
S11	Average Sulfur Grade
S12	High Sulfur Grade
S13	Life-Of-Mine Average
S1 ²	Soil
SP1	Saprolite
TST1	Transitional
T1	Garimpeiros Tailing
T2	Garimpeiros Tailing

¹ Sample S1 was sent to SGS for ore characteristic test work in 2007.

² Sample S1 was sent to Wardell Armstrong for gravity, whole ore cyanide leach and flotation test work in 2009.

Composites SMKG and SMIG were used to form two composites:

- Blend A: Consists of 75% SMKG and 25% SMIG
- Blend B: Consists of 25% SMKG and 75% SMIG

Samples S1 to S3 were used for hardness and abrasion testing by SGS Lakefield Research in Canada. Samples S4 to S13 were used for flotation and cyanide leach of flotation concentrates by Hazen Research in USA. Sample ALL was the overall composite that was sent to Wardell Armstrong for flotation batch tests and pilot plant. The drill holes that characterize the composites for ore

characterization and pilot plant test work are identified in Table 13-2 and Figure 13-1 and Figure 13-2.

Table 13-2: Drill Hole and Composite Description

Sample	Description	Drill Hole ID		Test Work	Contractor
SMKG	Smoky Granite	TOC-09-11TW	TOC-09-75TW	Ore Characteristics (SMC Ore Hardness, CWI, UCS, RWI, BWI, Abrasion)	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-35TW	TOC-09-102TW		
		TOC-09-48TW	-		
SMIG	Salami Granite	TOC-09-11TW	TOC-09-48TW	Ore Characteristics (SMC Ore Hardness, CWI, UCS, RWI, BWI, Abrasion)	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-35TW	TOC-09-75TW		
		TOC-09-36TW	TOC-09-102TW		
TOP	Top half orebody	TOC-09-11TW	TOC-09-102TW	Ore Characteristics (SMC Ore Hardness, CWI, RWI, BWI, Abrasion)	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-36TW	TOC-09-102TW		
BOT	Bottom half orebody	TOC-09-75TW	TOC-09-48TW	Ore Characteristics (SMC Ore Hardness, CWI, RWI, BWI, Abrasion)	SGS (2009) and Wardell Armstrong (2010)
		TOC-09-36TW	TOC-09-48TW		
ALL	Overall composite	TOC-09-11TW	TOC-09-48TW	Pilot Plant	Wardell Armstrong (2010)
		TOC-09-35TW	TOC-09-75TW		
		TOC-09-36TW	TOC-09-102TW		
S1	Top 100 m	TOC-04-03	TOC-05-32	Ore Characteristics (RWI, BWI, Abrasion)	SGS (2007)
		TOC-04-04	TOC-06-37		
		TOC-04-05	TOC-06-44		
		TOC-04-06	TOC-06-45		
		TOC-04-12	TOC-07-47		
		TOC-04-15	TOC-07-48		
		TOC-04-17	TOC-07-55		
		TOC-04-19	TOC-07-57		
		TOC-04-20	TOC-07-60		
		TOC-05-21	TOC-07-61		
		TOC-05-27	TOC-07-62		
		TOC-05-29	TOC-07-66		
S2	Middle 100 m	TOC-04-01	TOC-05-28	Ore Characteristics (RWI, BWI, Abrasion)	SGS (2007)
		TOC-04-04	TOC-05-30		
		TOC-04-06	TOC-05-31		
		TOC-04-10	TOC-05-32		
		TOC-04-11	TOC-05-34		
		TOC-04-16	TOC-06-36		
		TOC-04-17	TOC-06-39		
		TOC-04-18	TOC-06-43		
		TOC-05-21	TOC-06-45		
		TOC-05-22	TOC-07-47		
		TOC-05-24	TOC-07-57		
		TOC-05-25	TOC-07-62		
		TOC-05-27	-		

Sample	Description	Drill Hole ID		Test Work	Contractor
S3	Bottom 100 m	TOC-04-01	TOC-06-35	Ore Characteristics (RWI, BWI, Abrasion)	SGS (2007)
		TOC-04-11	TOC-06-36		
		TOC-04-16	TOC-06-36		
		TOC-04-17	TOC-07-47		
		TOC-04-19	TOC-07-48		
		TOC-05-21	TOC-07-52		
		TOC-05-22	TOC-07-56		
		TOC-05-24	TOC-07-60		
		TOC-05-30	TOC-07-61		
		TOC-05-32	TOC-07-62		
		TOC-05-34	-		

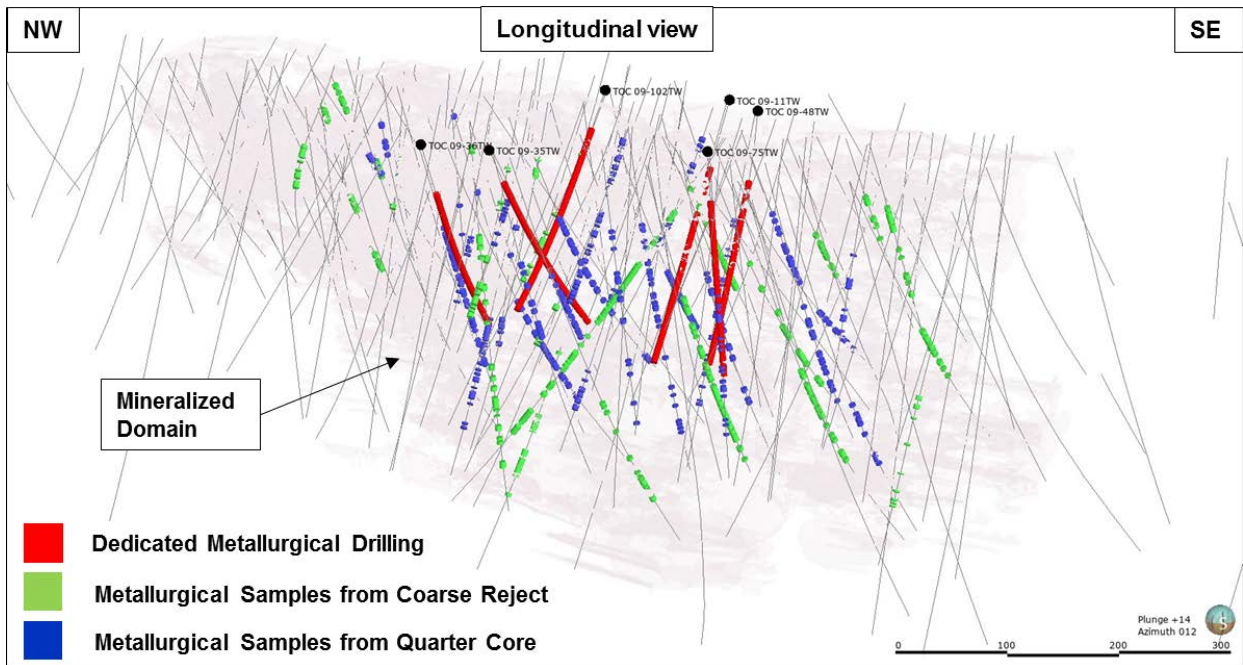


Figure 13-1: Drill Holes for Metallurgical Testing-Longitudinal View

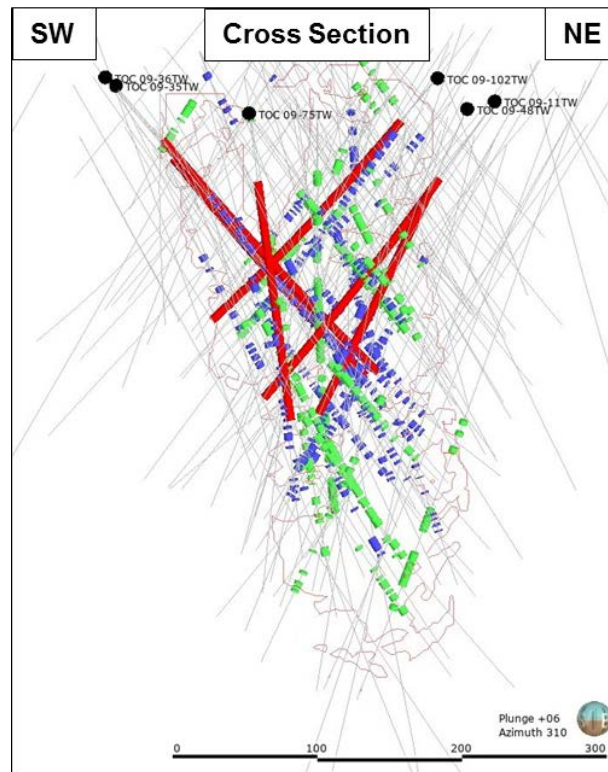


Figure 13-2: Drill Holes for Metallurgical Testing-Cross Section

13.4 ORE COMPOSITION

The Tocantinzinho mill feed during the LOM plan will come from three main sources. Granite representing 93% of the mine, saprolite representing 4% of the mine, and the garimpeiros tailings that were produced from previous artisanal mining, representing the remaining 3%. The distribution of the ore bodies is summarized in Table 13-3.

Table 13-3: Distribution of Ore

Ore Reserve	Tonnage tonnes	Distribution %
Granite	37,259,000	93.1
Saprolite	1,647,000	4.1
Garimpeiros Tailings	1,096,000	2.7
Total	40,002,000	100.0

The average annual plant head grade is 1.43 g/t Au for granitic ore, 1.21 g/t Au for saprolite, and 1.03 g/t Au for garimpeiros tailings. The combined average annual plant feed grade is 1.41 g/t Au with a maximum peak of 1.75 g/t Au in Year 4.

The head grade used for the design criteria is presented in Table 13-4.

Table 13-4: Ore Composition

Criterion	Units	Value		Notes
		Nominal	Design	
Plant head grade	g/t Au	1.41	1.76	Obtained from Mine Plan

Other elements including carbon, carbonate, arsenic, mercury, copper, zinc, etc. were assayed and described below.

- Silver in the orebody ranges from 0.03 to 1.11 ppm with an average of 0.66 ppm. Silver has not been considered in the economic evaluation of the Project
- Arsenic in the orebody ranges from 1.7 to 13 ppm with an average of 3.9 ppm
- Mercury is in the order of 0.01 ppm
- There are no other carbon elements other than carbonate
- Average sulphide content of 0.27%
- Carbonate (CO₃) level averages 0.65% and seems adequate in neutralizing any acid potentially generated from the oxidation of sulfide minerals within the ore body
- Average levels for cadmium and chromium are 2.2 ppm and 83 ppm, respectively
- Average levels for nickel, cobalt, copper, lead and zinc are 19 ppm, 3.6 ppm, 61 ppm, 115 ppm, and 104 ppm respectively

13.5 MINERALIZATION

There are two types of gold association with sulfide minerals; the first association occurs with pyrite (FeS₂), while the second association exists with pyrite (FeS₂), chalcopyrite (CuFeS₂), galena (PbS) and sphalerite (ZnS). Gold occurs as filling in the fractures and as inclusions (minor) in the following sulfide minerals:

- Disseminated pyrite
- Veinlets (in millimetre) of quartz, chlorite and pyrite (sheeted veins)
- Veins (in centimetre) of quartz, chlorite, carbonate, pyrite, chalcopyrite, galena and sphalerite

Pyrite occurs as liberated angular particles ranging in size from 30 to 400 µm with an average between 100 and 200 µm. Pyrite also occurs as intergrowth with galena, chalcopyrite, and rutile.

Chalcopyrite occurs as irregularly shaped particles ranging in size from 50 to 100 µm. The majority of chalcopyrite is liberated but may occur as inclusions in pyrite or intergrowth with galena and rutile.

The majority of gold is free at grain size ranging from 5 to 100 µm with an average range between 30 and 50 µm. The shape of gold grains is rounded, irregular shape, or elongated.

Pyrite in a disseminated form and in the sheeted veinlets hosts a bulk of gold mineralization. High-grade mineralization is often intimately associated with base metal veins. Gold also occurs less

frequently as liberated particles of irregular shape with a size ranging from 5 to 250 µm with an average between 100 and 200 µm.

13.6 COMMINATION

Seven samples were submitted to SGS Mineral Services in Canada for Bond Crushing Work index (CWi), Bond Rod Mill work index (RWi), Bond Ball Mill work index (BWi), SMC index (SMC is a shortened version of the standard JKTech drop-weight testing), Bond abrasion index and unconfined compressive strength (UCS).

Original comminution tests were carried out on three composites (S1, S2 and S3) by SGS in 2007.

Additional comminution tests were carried out on four composites (SMKG, SMIG, Top and Bot) by SGS in 2009.

The sample were prepared according to Figure 13-3.

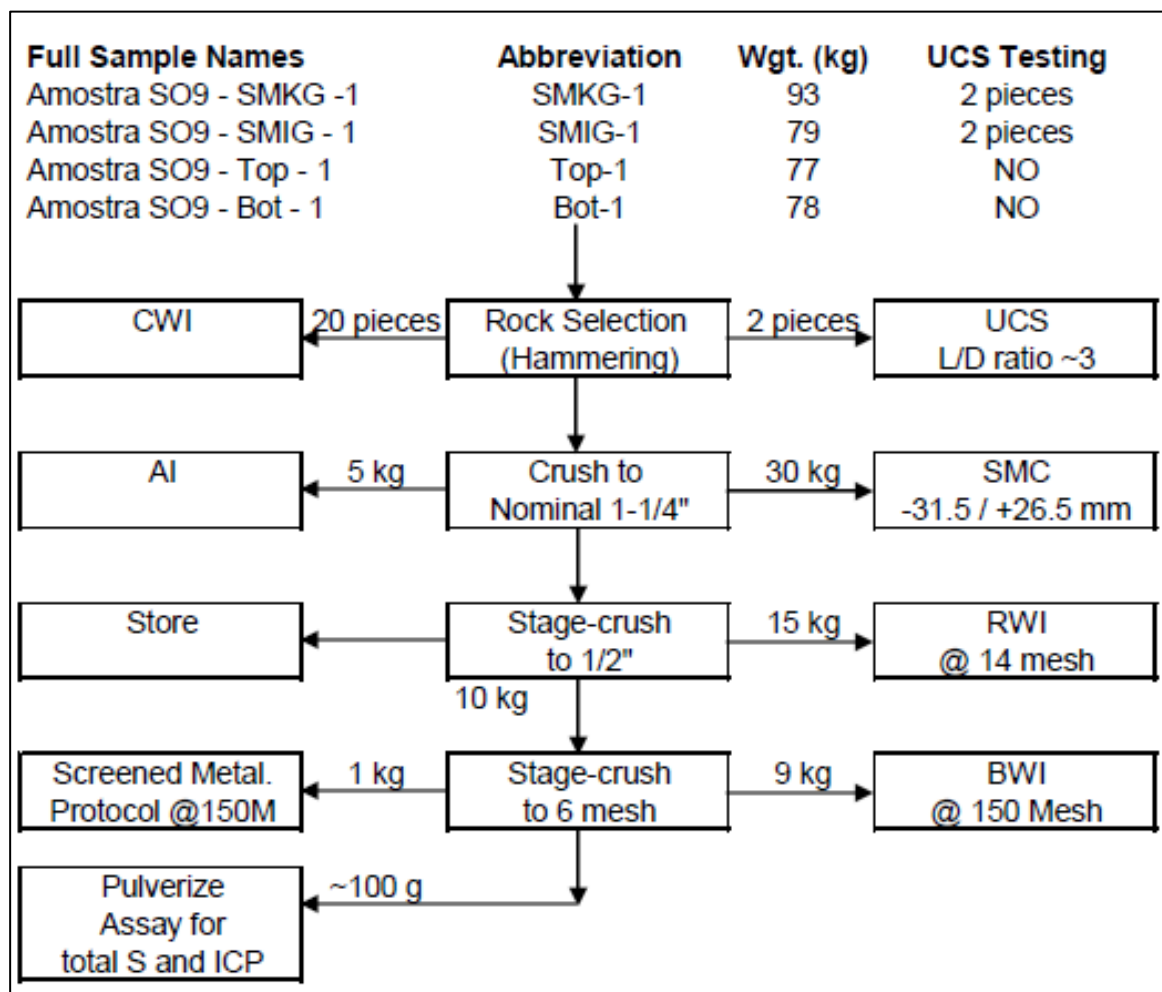


Figure 13-3: Sample Preparation Diagram for Comminution Test Work

From the test work the ore samples can be characterized:

- Uniform hardness and abrasiveness apart from Smoky Granite and Salami Granite being relatively easier to crush
- Medium to hard ore in terms of crushing, with work index varying from 10.1 to 15.5 kWh/t
- UCS test results ranged from 44.4 MPa to 114.7 MPa
- Moderately soft to medium hardness ore with respect to resistance to impact breakage (A x b) based on SMC test results. The (A x b) value varies from 51.5 to 59.3
- Medium ore in terms of rod milling, with work index varying from 13.2 to 14.7 kWh/t
- Hard ore with respect to ball milling, with work index varying from 16.8 to 18.5 kWh/t
- Strongly abrasive ore, with abrasion index varying from 0.418 to 0.717 gram

Eleven ore samples were selected for specific gravity test work. The value of specific gravity varies from 2.60 to 2.82 t/m³ with an average of 2.67 t/m³.

The results are summarized in Table 13-5.

Table 13-5: Comminution Test Work Summary

Sample	SMC					Bond Indices kWh/t			AI	UCS	SGS Report
	A x b	A	b	ta	DWI	CWI	RWI	BWI	g	MPa	Year
S1							14.1	17.6	0.695		2007
S2							14	17.6	0.717		2007
S3							14.7	18.2	0.612		2007
Amostra S09 - SMKG - 1	59.345	71.5	0.83	0.58	4.5	10.08	13.2	18.2	0.5009	44.4	2009
										114.7	
Amostra S09 - SMIG - 1	58.065	73.5	0.79	0.57	4.5	12.88	13.5	17.6	0.4501	104.4	2009
										75.4	
Amostra S09 - Top - 1	51.544	75.8	0.68	0.5	5.2	15.48	13.5	18.5	0.5578		2009
Amostra S09 - Bot - 1	53.436	73.2	0.73	0.52	5	15.31	14.1	16.8	0.4182		2009
Max	59.3	75.8	0.8	0.6	5.2	15.5	14.7	18.5	0.72	114.7	
Min	51.5	71.5	0.7	0.5	4.5	10.1	13.2	16.8	0.42	44.4	
Average	55.6	73.5	0.8	0.5	4.8	13.4	13.9	17.8	0.56	84.7	
85th Percentile	52.4			0.6	5.1	15.4	14.2	18.2	0.70	110.1	

The comminution values derived for the design criteria are summarized in Table 13-6.

- Unconfined Compressive Strength (UCS): 85th percentile out of the UCS test work used for the process design criteria (PDC).
- JK Breakage Parameter

- o (A x b), Drop Weight Index (DWi) and ta: Maximum hardness out of the SMC Test® work was used for the PDC to ensure the SAG mill can withstand variability of the ore.
- Bond Indices
 - o Crusher Work Index (CWI): 85th percentile out of the CWI test work was used for the design criteria.
 - o Rod Work Index (RWI): 85th percentile out of the RWI test work was used for the design criteria.
 - o Bond Work Index (BWI): 85th percentile out of the BWI test work was used for the design criteria.
- Abrasion Index: Maximum abrasion value obtained from the available test work was used for the PDC.
- Specific Gravity: Average value obtained from the available test work was used for the PDC.

Table 13-6: Design Criteria – Comminution Characteristics

Criterion	Units	Design	Notes
Unconfined Compressive Strength (UCS)	MPa	87.8	85th Percentile
JK Breakage Parameters			
A x b	-	51.54	Maximum Hardness from 4 Available Data
DWi	-	5.2	Maximum Hardness from 4 Available Data
ta	-	0.5	Maximum Hardness from 4 Available Data
Work Index			
Impact Crushing Work Index (CWI)	kWh/t	15.4	85th Percentile
Bond Rod Work Index (RWI)	kWh/t	14.2	85th Percentile
Bond Ball Mill Work Index (BWI)	kWh/t	18.2	85th Percentile
Abrasion Index	g	0.72	Maximum Value from All Available Data
Specific Gravity	-	2.67	Average Value from All Available Data

Calculations and modeling simulations (JKSimMet™) for the design and sizing of the comminution circuit was based on the values established in the design criteria. The target product grind size was 80% passing 125 microns based on pilot plant test work. A SAG Mill/Ball Mill (SAB) configuration was chosen and the size of the mills are summarized below:

- One 8.5 m diameter by 4.0 m long (Effective Grinding Length) SAG Mill with an installed power of 5,500 kW
- One 6.4 m diameter by 9.75 m long (Effective Grinding Length) Ball Mill with an installed power of 7,500 kW

13.7 GRAVITY CONCENTRATION

The gravity recovery test work was performed using a Knelson concentrator which obtained moderate high recoveries for gold and was therefore included in the process design.

The gravity concentration test work program was evaluated four times using the Knelson concentrator.

The first test work program was carried out by SGS Mineral Services in Canada on four composite ore samples (2005 report). Gold recovery increased with head grade, reaching 41.9% for a high-grade (12.3 g/t) ore sample. The concentrate was upgraded using a Mozley table and obtained a much higher concentrate grade.

The second test work program on gravity concentration was conducted by Wardell Armstrong International in UK on three composite ore samples (2009 report). Four stages of gravity concentration were applied consecutively to each ore sample as the grind size became finer. All three ore samples achieved high recoveries ranging from 73% to 90% with a mass pull of 4.2% and a concentrate grade ranging 26 - 36 g/t Au.

The third test work program on gravity concentration was conducted by Wardell Armstrong International in 2009. Seven composites were tested and obtained moderate-high gold recoveries, ranging from 53% to 70%, with an average mass yield of 4.4%. The average recovery of the seven composites was used to support gravity recovery values in the design criteria (see Table 13-7).

Table 13-7: Gravity Test Work on Composites by Wardell Armstrong (2009)

Sample			SMKG	SMIG	Blend A	Blend B	BOT	TOP	ALL	Avg.
Calculated Head Grade		g/t	0.49	1.81	1.05	1.71	1.63	1.89	1.48	1.44
Gravity Concentrate	Mass	%	5.03	4.48	4.14	4.46	4.07	4.50	4.28	4.42
	Grade	g/t	6.01	21.5	15.5	23.0	25.1	29.5	18.7	19.9
	Recovery	%	62.2	53.0	61.0	60.2	62.4	70.3	54.1	60.5

The fourth test program was a single test conducted by FLSmidth Knelson in Canada (2012 report). The test consisted of sequential liberation and recovery stages. It obtained gold recovery of 65% at a 1.1% mass yield. The final concentrate grade was 75.9 g/t Au. This test was used as the basis for gravity modeling and selection of the gravity concentrator.

A summary of the gravity test work is shown in Table 13-8.

The gravity recovery circuit will consist of a single centrifugal gravity concentrator and an intensive cyanidation unit (ICU). The gravity concentrate will be leached in batches and assumed to have a leach recovery of 94.5% for the design criteria. The leach recovery is based on the same cyanide leach recovery for flotation concentrate as per the pilot plant test work. There has been no cyanide leach test work performed on gravity concentrate and has been listed as a risk/opportunity in Section 13.15.

Table 13-8: Summary of Gravity Test Work Results

Gravity Test Program	Sample ID	Calc Head Grade	Grind Size 80% passing	Mass Pull	Concentrate Grade	Au Recovery	Concentrator Passes	Batch
		g/t Au	µm	%	g/t Au	%		
1st Gravity Test Program - SGS (2005)	Comp A	1.33	66	0.057	351	15	1 pass	1 kg
1st Gravity Test Program - SGS (2005)	Comp B	1.02	75	0.121	239	28	1 pass	1 kg
1st Gravity Test Program - SGS (2005)	Comp C	3.55	65	0.117	1178	39	1 pass	1 kg
1st Gravity Test Program - SGS (2005)	Comp D	9.89	45	0.08	5171	42	1 pass	1 kg
2nd Gravity Test Program - Wardell Armstrong (2009)	TOP	1.61	125	4.2	31.5	83	3 passes	11 kg
2nd Gravity Test Program - Wardell Armstrong (2009)	BOT	1.72	125	4.23	36.5	90	3 passes	11 kg
2nd Gravity Test Program - Wardell Armstrong (2009)	ALL	1.56	125	4.21	26.2	73	3 passes	11 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	SMKG	0.49	125	5.03	6.01	62	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	SMIG	1.81	125	4.48	21.5	53	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	Blend A	1.05	125	4.14	15.5	61	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	Blend B	1.71	125	4.46	23	60	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	BOT	1.63	125	4.07	25.1	62	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	TOP	1.89	125	4.5	29.5	70	1 pass	2 kg
3rd Gravity Test Program - Wardell Armstrong (2009)	ALL	1.48	125	4.28	18.7	54	1 pass	2 kg
4th Gravity Test Program - FLS Knelson (2012)	Tocantinzinho	1.3	70	1.1	75.9	65	3 passes	20 kg

13.8 FLOTATION

Flotation test work has been extensive since 2007. The test program includes batch rougher/scavenger, batch cleaner, locked cycle and continuous pilot plant to examine flotation kinetics, reagent selection, grind size and metallurgical performance of various ore types.

13.8.1 Preliminary Flotation Tests

Grind size was evaluated initially by Ralph Meyertons (2007). PAX and Aeroflot were used as collectors, and mineral oil and Aerofroth 65 were used as frother. A summary of the test work is shown in Table 13-9.

Overall gold recovery averaged 92.4% with a concentrate mass pull of 2.4%. Grind size correlation to gold recovery could not be established as the best two gold recovery results were achieved with the coarsest and the finest samples (Test 1E and 3A, respectively). The calculated head grade was also found to be quite variable.

Table 13-9: Flotation Test Work Results from Ralph Meyertons (2007)

Test ID			1D	1E	2A	2B	3A	3B	3C	Average
Calculated Head Grade		g/t	1.67	1.70	1.89	1.35	2.07	1.06	0.93	1.52
Grind Size	Passing 150 µm	%	86.0	86.0	~100	99.5	~100	95.4	99.5	-
	Passing 100 µm		-	-	98.8	94.5	98.8	74.1	93.5	-
	Passing 75 µm		-	-	88.9	75.5	88.9	54.0	75.5	-
Final Concentrate	Mass Pull	%	3.5	5.0	2.0	2.0	1.4	1.1	1.5	2.4
	Grade	g/t	44.7	32.8	87.8	62.5	137.0	88.1	55.9	72.7
	Recovery	%	90.5	96.1	93.4	92.9	95.9	88.5	89.6	92.4

Natural pH, 32% pulp density, 10 minutes conditioning time, 30 g/t PAX, 30 g/t Aeroflot 31, 30 g/t mineral oil, 20 ~ 40 g/t Aerofroth 65, 20 ~ 25 minutes flotation time, 1.46 g/t Au assay head grade

13.8.2 Initial Reagent Screening

Batch test work flotation tests were carried out by Hazen starting in 2007 prior to Eldorado's involvement. Initial tests were done on composite samples to examine reagent selection, operating conditions and upgradeability with scavenger and cleaning stages.

Additional variability and confirmatory test work were carried out by Hazen again in 2009 when Eldorado became involved. The reagents that were tested is listed below:

- Collectors/Promoters:
 - o ORFOM CO100 (Mercaptan)
 - o PAX (Potassium amyl xanthate)
 - o Aero 5688 (Monothiophosphate salt)
 - o Aerofloat 31
 - o S-8474
 - o Mineral Oil

- o CMC (Carboxymethyl cellulose)
- Frothers:
 - o Aerofroth 65
 - o DF-250
 - o MIBC (Methyl isobutyl carbinol)

A summary of the reagent variability test work is shown in Table 13-10 and Figure 13-4.

Table 13-10: Reagent Variability Flotation Test Work Summary

Test Number			SF1	SF2	SF3	SF4	SF5	SF6
Ore Material			Composite					
Calculated Feed Grade		Au g/t	1.3	1.5	1.5	1.4	1.4	1.1
		S %	0.40	0.44	0.40	0.40	0.37	0.35
Grind Size, 80% Passing		µm	~75 µm					
Concentrate Mass Pull		%	10.5	5.1	8.3	6.9	10.3	6.0
Concentrate Grade		Au g/t	11.1	28.1	17.0	19.0	12.4	16.9
Au Recovery		%	92.9	93.8	93.9	93.4	93.5	91.5
Collector/ Promoters	CO100 (Grind)	g/t	-	-	100	100	0	30
	PAX	g/t	45	45	30	30	45	45
	Aero 5688	g/t	-	-	60	60	90	90
	Aerofloat 31	g/t	45	-	-	-	-	-
	S-8474	g/t	-	45	-	-	-	-
	Mineral Oil	g/t	45	-	-	-	-	-
	CMC	g/t	-	100	-	-	-	-
Frothers	Aerofroth 65	g/t	28	-	-	-	-	-
	DF250	g/t	-	-	33	37	37	46
	MIBC	g/t	-	74	-	-	-	-

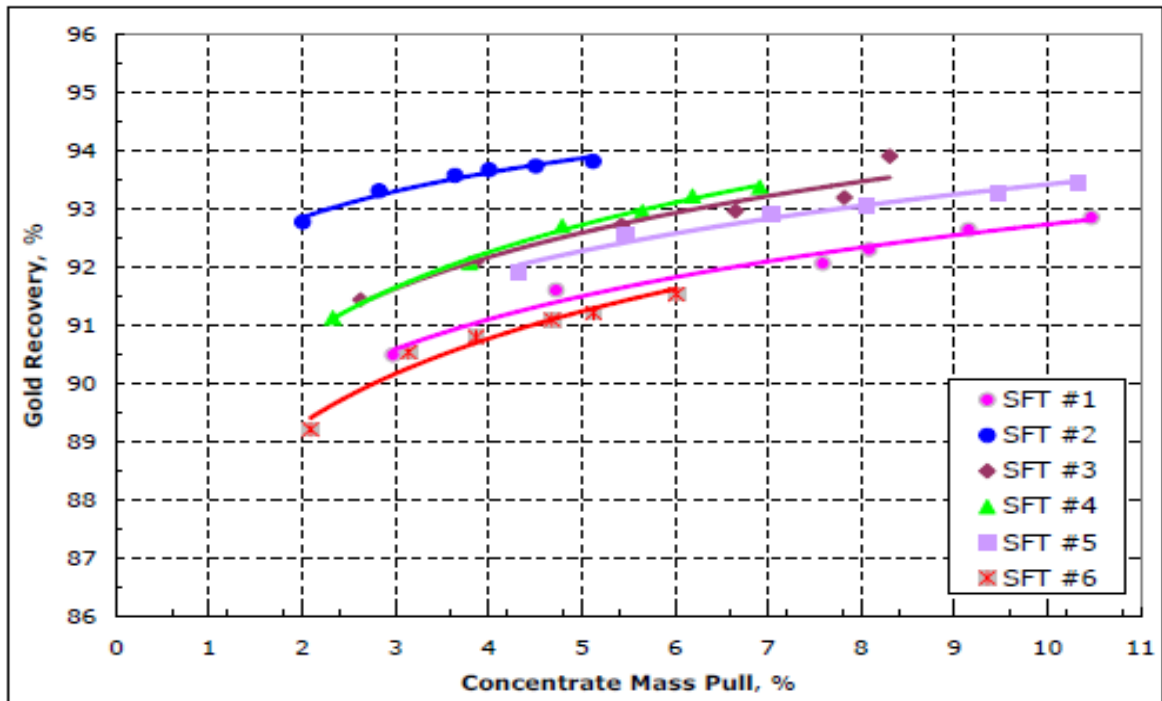


Figure 13-4: Gold Recovery vs Concentrate Mass Pull for Reagent Variability Test Work

Among these six tests, SFT #2 obtained the best performance, achieving nearly 94% recovery at 5.1% mass pull.

The operating conditions for SFT#2 are listed below:

- Grinding – 80% passing 75 μm , 15 g/t PAX, 15 g/t S-8474, 100 g/t CMC
- Flotation – natural pH, 30-35% pulp density, 48 minutes total flotation time, 30 g/t PAX, 30 g/t S-8474

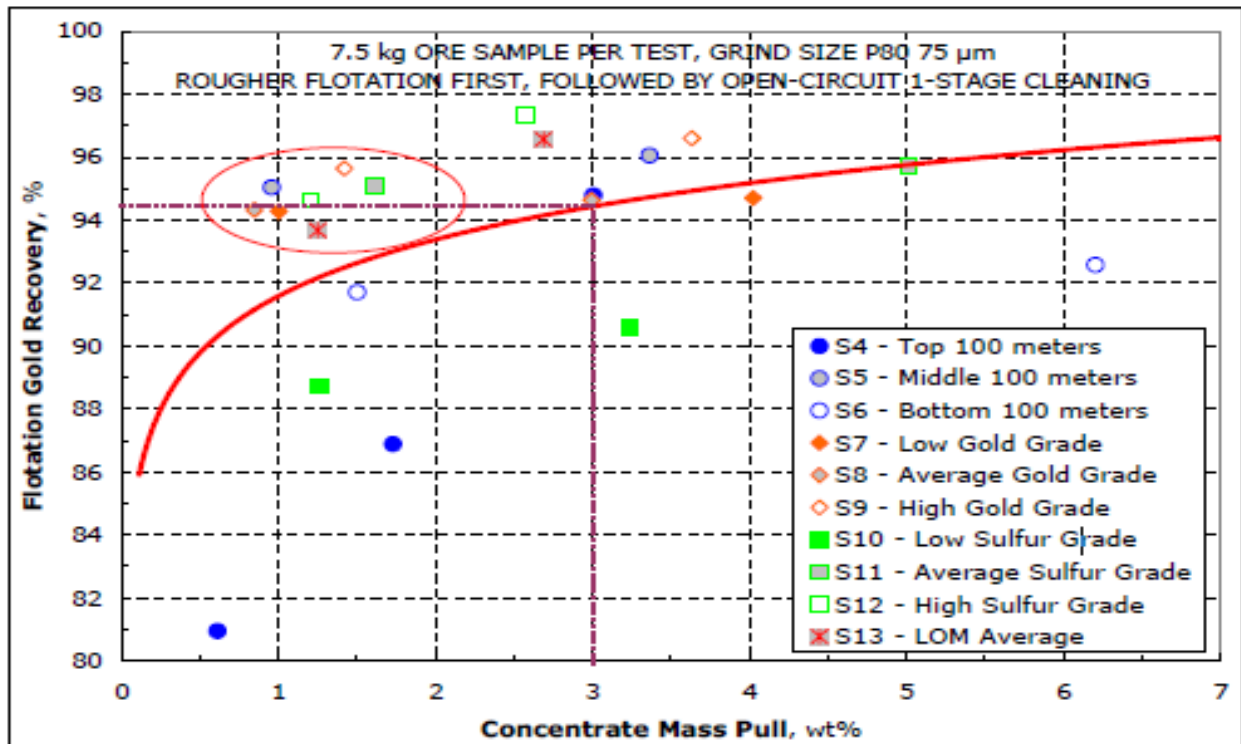
13.8.3 Initial Ore Variability Assessment

Following the operating conditions for SFT #2, Hazen completed testing of 10 different ore types in 2009.

These results are presented in Table 13-11 and Figure 13-5.

Table 13-11: Ore Variability Flotation Test Work Summary

Test Number		LF1	LF2	LF3	LF4	LF5	LF6	LF7	LF8	LF9	LF10
Ore Material		S4	S5	S6	S7	S8	S9	S10	S11	S12	S13
Calculated Feed Grade	Au g/t	0.24	0.93	0.92	0.56	1.17	1.85	0.96	0.89	0.89	1.35
Grind Size, 80% Passing	µm	~75 µm									
Concentrate Mass Pull	%	1.7	3.4	6.2	4.0	3.0	3.6	3.2	5.0	2.6	2.7
Concentrate Grade	Au g/t	50.1	26.4	13.7	13.1	37.1	49.2	27.1	17.1	66.1	48.6
Au Recovery	%	86.8	96.0	92.6	94.7	94.6	96.6	90.7	95.7	97.3	96.6
Collector/ Promoters	CO100 (Grind)	g/t	-	-	-	-	-	-	100	100	100
	PAX	g/t	60	60	60	60	60	60	60	40	40
	Aero 5688	g/t	-	-	-	-	-	-	80	80	80
	Aerofloat 31	g/t	-	-	-	-	-	-	-	-	-
	S-8474	g/t	60	60	60	60	60	60	-	-	-
	Mineral Oil	g/t	-	-	-	-	-	-	-	-	-
	CMC	g/t	100	100	100	100	100	100	-	-	-
Frothers	Aerofroth 65	g/t	-	1	1	1	1	1	-	-	-
	DF250	g/t	21	-	-	-	-	-	4	3	3
	MIBC	g/t	-	-	-	-	-	-	-	-	-

**Figure 13-5: Gold Recovery vs Concentrate Mass Pull for Ore Variability Test Work**

Flotation performance for these ten different composite samples demonstrate:

- Materials near the surface containing a lower sulfur level did not float as well
- Ore with higher gold grade or sulfur grade floated better
- Mass pull at 3%, gold recovery is expected to be ~94.5%
- Mass pull at 5%, gold recovery is expected to be ~96%

The results are in good agreement with the previous test work.

13.8.4 Additional Variability Test: Rougher Flotation

Additional variability test work, including rougher, cleaner and locked cycle tests, were completed in 2010 by Wardell Armstrong on 7 composites:

- SMKG, SMIG, BOT, TOP, ALL
- Blend A: Consists of 75% SMKG and 25% SMIG
- Blend B: Consists of 25% SMKG and 75% SMIG

The reagents that were tested includes:

- Collector: SIBX (Sodium isobutyl xanthate)
- Activator: Copper sulfate
- Frother: DF-250

Two grind sizes were tested for all 7 composites:

- 80% passing 75 μm
- 80% passing 125 μm

Additional testing at 80% passing 100 μm and 150 μm was done on ALL Composite.

A summary of the tests on the ore sample designated ALL can be found in Table 13-12. This ore sample represents a global proportion of ore types within the orebody.

Table 13-12: Rougher Flotation Test Work for the ALL (Global Composite) Ore Sample

Test	Conditions	Mass Pull %	Recovery %	
			Au	STot
FT1	Grind 75µm	3.5	96.3	87.7
FT2	Grind 100µm	3.8	96.0	88.2
FT3	Grind 125µm	3.8	96.7	80.4
FT4	Grind 150µm	3.8	95.2	87.3
FT5	As FT3 but with no CuSO ₄	7.2	94.2	82.8
FT6	As FT3 but with reduced CuSO ₄	3.1	94.8	88.4
FT7	As FT3 but with MIBC	3.4	92.1	84.8
FT8	As FT3 but with reduced SIBX	4.5	92.1	87.2
FT9	As FT3 but with increased SIBX	4.3	92.3	85.5
FT10	As FT3 but with SIBX+A5688	5.3	95.0	83.7
FT11	As FT10 but with increased SIBX+A5689	7.9	94.9	84.3
FT12	Hazen Conditions from Previous study	10.5	87.6	88.4
FT13	As FT3 but with PAX	2.4	92.2	78.7
FT14	As FT3 but with SIBX+S-7151	3.1	75.6	80.9
FT15	As FT3 but with SIBX+A8761	3.9	94.2	82.9
FT16	As FT3 but with 50% Increase in DF250	4.1	88.7	81.7
FT17	As FT3 but with 50% decrease in DF250	2.7	93.1	77.0

The results from the ALL rougher flotation test work demonstrate:

- Grind size at 125 µm brought similar to slightly better results when compared to finer grind sizes
- Flotation performance without copper sulfate is adversely affected (lower recoveries, higher mass pull)
- Varying dosage levels of SIBX and DF-250 were tested from Tests FT7 to FT17. Compared with the baseline reagent suite (Test FT3: 50 g/t CuSO₄·5H₂O, 80 g/t SIBX, 40 g/t DF-250), other conditions and alternative reagents did not generate better flotation performance

Test FT3 obtained the highest recovery and was chosen as the basis for further test work including the pilot plant.

The design criteria for retention time and reagent addition is based on the parameters for Test FT3 (Table 13-14).

Downstream test work such as cleaner, locked cycle and pilot plant test work used the parameters from Test FT3 for rougher flotation conditions.

13.8.5 Additional Variability Test: Cleaner Flotation

A more comprehensive cleaner flotation test was done on the global composite (ALL) and is summarized in Table 13-13 and Figure 13-6.

Rougher flotation was carried out based on the conditions in Test FT3 (See Section 13.8.4) and only one stage of cleaning was tested.

Table 13-13: Cleaner Flotation Test Work for ALL (Global Composite) Ore Sample

Test	Conditions	Mass Pull %	Recovery %	
			Au	S _{Tot}
FCT1	One clearer stage	3.0	94.1	82.4
FCT2	Scalp and clean rougher scavenger	0.9	83.0	69.7
FCT3	As FT1 but increase float time	2.4	92.7	78.7
FCT4	As FT1 but no DF250 or SIBX	2.5	95.2	82.6
FCT5	As FT1 but increase SIBX	2.2	91.7	77.3
FCT6	As FT1 but CuSO ₄ -added	2.4	92.3	85.3
FCT7	Airflow rate 1 l/min	2.3	92.3	90.4
FCT8	Airflow rate 0.5 l/min	2.4	90.3	85.4
FCT9	Airflow rate 1.5 l/min	2.6	90.8	86.6

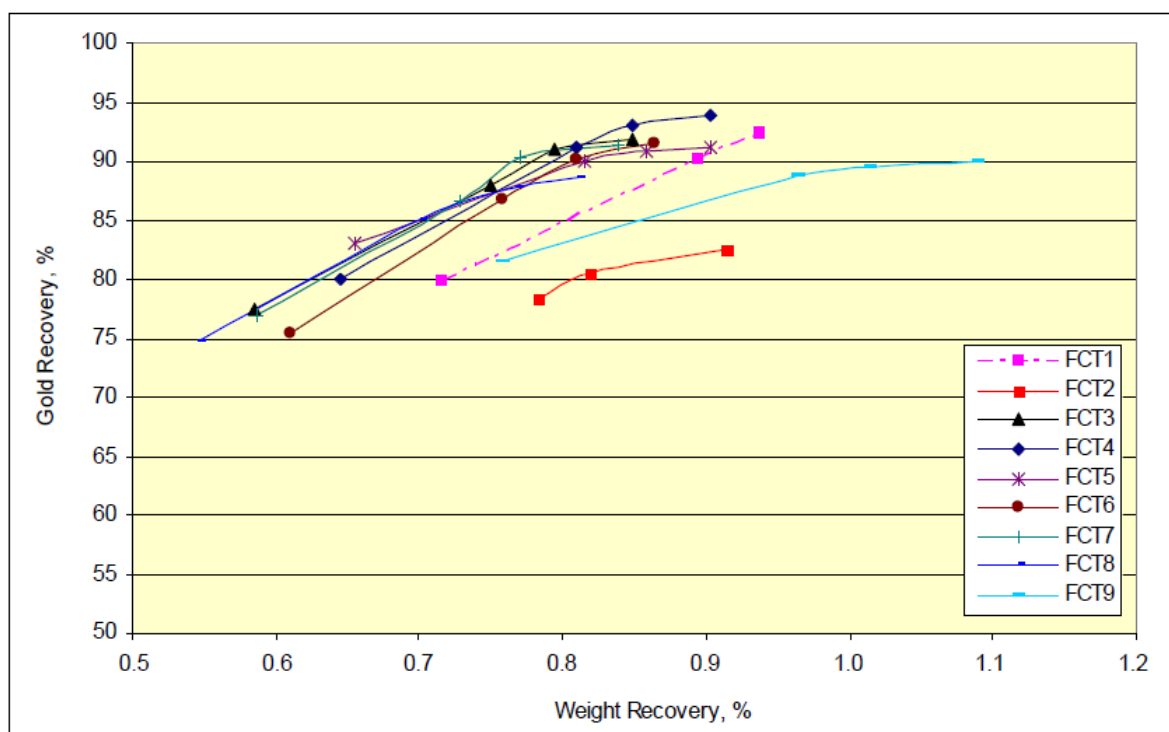


Figure 13-6: Gold Recovery vs Concentrate Mass Pull for Cleaner Flotation Test Work

Test FCT4 had a much higher feed grade which contributed to significant higher recovery and was therefore considered an outlier.

Test FCT3 obtained the best mass pull to recovery ratio and was used to develop the design criteria.

Combining the best conditions from rougher and cleaner test work (Test FT3 from Section 13.8.4 and Test FCT3 from Section 13.8.5), the flotation duration and the flotation reagent design criteria can be summarized in Table 13-14.

Table 13-14: Design Criteria - Flotation Retention Time and Reagent Addition

Flotation Duration Criterion	Units	Design	Notes
Conditioning	min	5.0	Client specified
Rougher Flotation - Lab Retention Time	min	6.5	Batch test work rougher and rougher scavenger total at 13 min (Based on Test FT3)
Rougher-Scavenger Flotation - Lab Retention Time	min	6.5	
Cleaner Flotation - Lab Retention Time	min	7.0	Cleaner batch test work at 7 min (Based on Test FCT3)
Flotation Reagent Addition Criterion	Units	Design	Notes
Collector: SIBX Addition Rate			Based on Wardell Armstrong Test Work FCT3 (same as Pilot Plant)
Rougher Flotation Conditioning Addition	g/t Mill Feed	60.0	
Rougher Flotation SIBX Addition	g/t Mill Feed	20.0	
Cleaner Flotation SIBX Addition	g/t Mill Feed	10.0	
Total SIBX Addition	g/t Mill Feed	90.0	
Frother: DF-250 Addition Rate			Based on Wardell Armstrong Test FCT3 (same as Pilot Plant)
Rougher Flotation Addition	g/t Mill Feed	30.0	
Rougher-Scavenger Flotation Addition	g/t Mill Feed	10.0	
Cleaner Flotation Addition	g/t Mill Feed	10.0	
Total Frother Addition	g/t Mill Feed	50.0	
Promoter: Copper Sulphate Addition Rate			Based on Wardell Armstrong Test FCT3 (same as Pilot Plant)
Grinding Addition	g/t Mill Feed	25.0	
Rougher Flotation Conditioning Addition	g/t Mill Feed	25.0	
Total Promoter Addition for Flotation	g/t Mill Feed	50.0	

13.8.6 Additional Variability Test: Locked Cycle Test Work

Wardell Armstrong completed locked cycle tests on seven composites in 2010. A summary of the results is presented in Table 13-15.

A simple reagent suite based on SIBX as collector, copper sulfate as activator and DF-250 as frother was used.

Table 13-15: Locked Cycle Flotation Test Work Summary

Ore Sample			SMKG	SMIG	Blend A	Blend B	TOP	BOT	All		
Calculated Head Grade	Gold	g/t	0.59	1.91	0.85	1.48	1.49	1.66	1.53	1.43	1.34
	Sulfur	%	0.34	0.35	0.28	0.26	0.44	0.48	0.33	0.32	0.29
Grind Size, 80% Passing		µm	125						125	100	125
Rougher Condition	SIBX	g/t	80						80	80	120
	DF250	g/t	40						40	40	60
Cleaner Condition	SIBX	g/t	20						10	10	15
	DF250	g/t	20						10	10	15
Concentrate Mass Pull		%	0.79	0.78	0.87	0.92	0.98	1.35	0.80	1.29	1.57
Concentrate Grade	Gold	g/t	69.6	221	83.8	146	144	118	175	101	79.7
	Sulfur	%	32.3	37.1	23.2	23.0	40.4	31.0	33.2	21.2	15.6
Recovery	Gold	%	92.5	89.9	86.0	91.3	94.3	95.5	91.6	91.7	93.4
	Sulfur	%	75.1	82.9	71.9	81.1	88.9	87.6	81.8	84.7	84.7

32% pulp density for rougher, natural pH (8.4 ~ 9.3) for rougher and cleaner, 50 g/t CuSO₄·5H₂O, rougher flotation time 13 to 15 minutes, cleaner time 6 to 7 minutes. Average results from 5th and 6th cycles.

Composite ore samples, namely TOP, BOT and ALL, outperformed slightly individual ore types, SMKG and SMIG. The average gold recovery was 93.3% with an average concentrate mass pull of 1.20%.

13.8.7 Pilot Plant Flotation

Pilot plant flotation test work was undertaken by Wardell Armstrong International in 2010-2011. The composite ore sample designated ALL, which represents a global proportion of ore types within the orebody, was used for this test work.

A total of 3,100 kg material was processed at a rate of 65 ~ 80 kg/h through crushing, grinding, rougher flotation and cleaner flotation. The purpose of the pilot plant campaign was to verify gold recovery and concentrate mass pull, and to generate tailing materials for other test work.

The reagent suite and dosage rates for the pilot plant test work were based on bench scale flotation test FT3 and FCT3 (See Section 13.8.4 and 13.8.5)

The pilot plant test work was tested at two grind sizes; 80% passing 125 µm and 85 µm, and the results are summarized in Table 13-16.

Table 13-16: Pilot Plant Test Work Summary

Day	Grind, µm	Feed Wt			Wt Pull, %	Recovery, %
		Kg	%	%wrt grind		
Day 1 - 6th	87	525.90	16.89	35.13	5.20	83.93
Day 2-1 - 7th	85	387.40	12.44	25.88	3.44	90.69
Day 2-2 - 7th	85	237.80	7.64	15.88	4.30	87.94
Day 3 - 8th	90	346.06	11.11	23.11	6.16	95.77
Fine Grind (Av. assay)		-	-	-	4.49	89.36
Fine Grind (Wt Av.)		1,497.16	48.08	100.00	4.82	89.05
Day 4 - 9th	115	528.37	16.97	32.68	4.74	91.51
Day 5 - 10th	120	537.48	17.26	33.24	6.88	95.77
Day 6 - 11th	125	551.04	17.70	34.08	1.98	93.10
Coarse Grind (Av. Assay)		-	-	-	3.62	93.52
Coarse Grind (Wt Av.)		1,616.89	51.92	100.00	4.51	93.46
Total		3,114.1	100.00	-	4.66	91.34

Copper sulphate solution was added to the first conditioner at a rate of 50 g/t CuSO₄·5H₂O and SIBX and DF-250 solution was added to the second conditioner at a rate of 90 g/t and 50 g/t respectively. (See Section 13.8.4 and 13.8.5)

At the coarser grind sizes (P80 from 115 to 125 µm), gold recovery achieved an average of 93.5% at a concentrate mass pull of 4.51%. The coarser grind was used as the flotation design criteria (Table 13-17) due to a higher recovery to mass yield ratio.

13.8.7.1 Grind Size

Pilot plant utilized two types of grind sizes. A finer target with P80 from 85 to 90 µm and a coarser target with P80 from 115 to 125 µm were chosen based on optimum results from batch variability tests (refer to Section 13.8.4). Based on the results from the batch test work and the pilot plant test program, a 125 µm P80 was selected for the design criteria.

Table 13-17: Design Criteria - Flotation Recovery

Criterion	Units	Design	Notes
Flotation feed size, P80	µm	125	Based on batch and pilot plant test work
Concentrate recovery, Au, relative to mill fresh feed	%	93.5	Flotation pilot plant test work
Gravity recovery (Design), relative to mill fresh feed	%	25	Based on gravity simulation and client data
Concentrate recovery, Au, relative to mill fresh feed	%	68.5	Flotation pilot plant recovery minus gravity recovery relative to overall plant circuit
Cleaner mass pull	%	4.51	Weighted average from pilot plant test work (Day 4, 5, 6)

13.8.8 Additional Flotation Test Work

Impact of process water on flotation

- Recycling process water marginally increased gold recovery but primarily impacted concentrate mass pull
- Concentrate mass pull increased from 2.4% (FT18-1) to 6.6% (FT18-3) after process water was recycled twice
- Reagent requirement during plant operation was not tested but is expected to drop significantly. Normally one third in comparison with what consumed in the laboratory where only fresh water is used.
- Impact of site make-up water on flotation
- There was no significant impact on gold recovery despite slight slower flotation kinetics. At a concentrate mass pull of 4.51%, similar gold recovery is expected in comparison with clean tap water.

Table 13-18: Water Test Conditions and Results

Test ID	Conditions	Concentrate Mass Pull	Gold Recovery	Sulfur Recovery	Calculated Head Grade	
		%	%	%	g/t Au	%S
FT18-1	Tap water	2.4	94.4	87.2	1.74	0.30
FT19	Water from the drill hole TOC-91 (WS-1)	4.2	93.5	79.2	1.77	0.37
FT20	Water from the Teodorao River (WS-2)	4.8	94.8	81.3	1.28	0.30

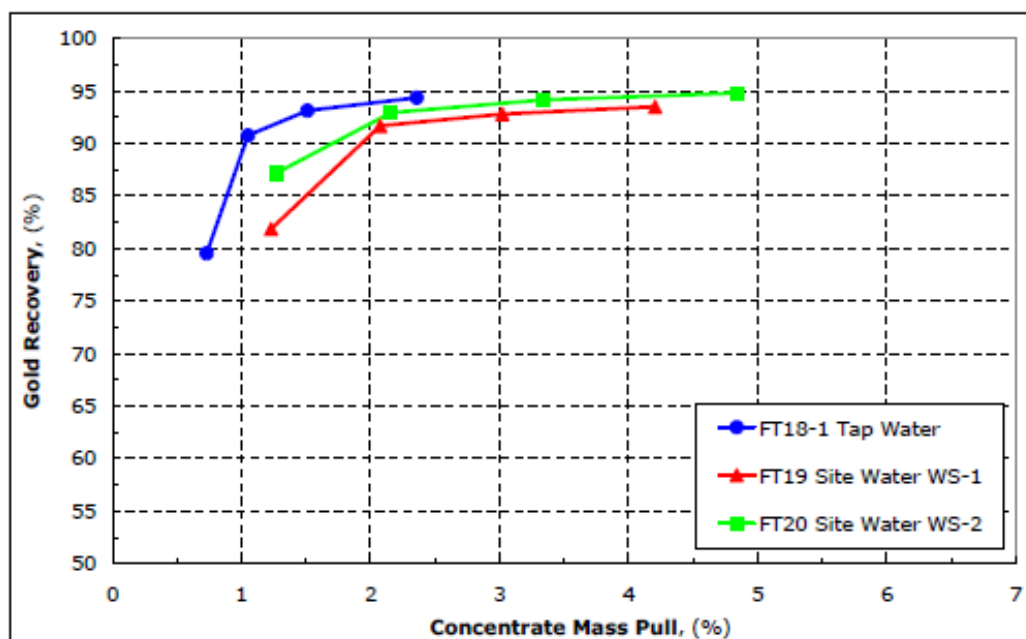


Figure 13-7: Gold Recovery and Water Test Conditions

13.9 COMPOSITION OF FLOTATION CONCENTRATE

Ten cleaner flotation concentrates were analyzed to obtain a general impression of the composition. Due to variations in ore composition and concentrate mass pull during flotation, concentration compositions vary.

The average flotation concentrate assay of notable elements below were primarily from the locked cycle tests in the Wardell Armstrong International pilot plant test program in 2010 (Mass Pull ~1%).

- Silver = 89 ppm
- Aluminum = 4.2%
- Arsenic = 176 ppm
- Carbonate carbon = 0.26%
- Gold = 113 ppm
- Calcium = 0.9%
- Copper = 3,628 ppm
- Iron = 23%
- Mercury = 0.8 ppm
- Potassium = 2.0%
- Lead = 6,396 ppm
- Magnesium = 0.8%
- Zinc = 3,423 ppm
- Sodium = 0.9%
- Total sulfur = 24.7%

The cleaner concentrate grade used for design was calculated from a 4.5% mass yield and flotation recoveries from pilot plant test work with gravity taken into consideration (Table 13-19).

Table 13-19: Design Criteria – Flotation Concentrate Grade

Criterion	Units	Nominal		Design
Cleaner Mass Pull	%	4.5		Flotation Pilot Plant Test Work
Gold Concentrate Grade	g/t	21.6	24.4	Design grade based on 5% mass yield and same recoveries

13.10 CYANIDE LEACHING

Initial cyanide leaching test work was carried out by Hazen in 2009 to determine the leach kinetics of different concentrate samples generated from previous flotation test work. Eleven concentrates, without additional grinding were leached for 90 hours. All samples, excluding one outlier, achieved excellent gold recovery with an average of 98%. The leach rate was reasonable as most of the gold recovery began to plateau within 48 hours.

In 2010, Wardell Armstrong International obtained seven different concentrate samples for cyanide leaching. The test program compared oxygen/air sparging, cyanide concentration and concentrate regrind. The tests were leached for 48 hours before activated carbon was added and continued for an additional 6 hours.

The results of the batch and pilot plant test work are summarized in Table 13-20 with the description of the test work conditions listed below:

- PP-LT2: Concentrate P80 = 125 μ m, maintain cyanide at 2 g/L
- PP-LT3: Concentrate P80 = 125 μ m; re-ground to P80 = 18 μ m, maintain cyanide at 1 g/L

- PP-LT4: Concentrate P80 = 125 μm , maintain cyanide at 1 g/L and use oxygen for aeration
- PP-LT5: Concentrate P80 = 125 μm , maintain cyanide at 5 g/L
- PP-LT6: Concentrate P80 = 85 μm , maintain cyanide at 1 g/L
- PP-LT6: Concentrate P80 = 85 μm , maintain cyanide at 2 g/L

Table 13-20: Cyanide Leach on Flotation Concentrate Test Work (Batch and Pilot Test)

Sample ID	Grind Size 80% Passing	Pulp Density	Cyanide Concentration		Concentrate Regrinding	Sparging	Gold Recovery	Lime Consumed	Cyanide Consumed	Calculated Head Grade
			Initial	Maintained						
	µm	%	g/L NaCN				%	kg/t	kg/t NaCN	g/t Au
SMKG	125	4.3	2.0	2.0	no	air	95.7	0.44	32.8	69
SMIG		3.6					91.1	0.54	92.3	281
Blend A		5.4					97.0	0.35	44.7	190
Blend B		7.2					97.4	0.26	21.3	120
TOP		4.5					96.8	0.42	34.1	101
BOT		4.5					97.1	0.42	42.1	142
Average	/	/	/	/	/	/	95.9	0.41	44.6	/
ALL - PPLT2	125	27	4.9	2.0	No	air	95.0	0.23	6.34	30.6
ALL - PPLT3		27	4.9	1.0	yes- 18 µm	air	97.8	0.33	4.82	29.7
ALL- PPLT4		27	4.9	1.0	no	oxygen	94.0	0.22	3.89	32.6
ALL- PPLT5		34	10.3	5.0	no	air	96.3	0.15	16.4	30.0
ALL - PPLT6	85	35	10.0	5.0	no	air	96.8	0.20	12.6	13.0
ALL - PPLT7		35	4.0	2.0	no	air	97.2	0.20	5.81	13.9
Average	/	/	/	/	/	/	96.2	0.22	8.3	25.0

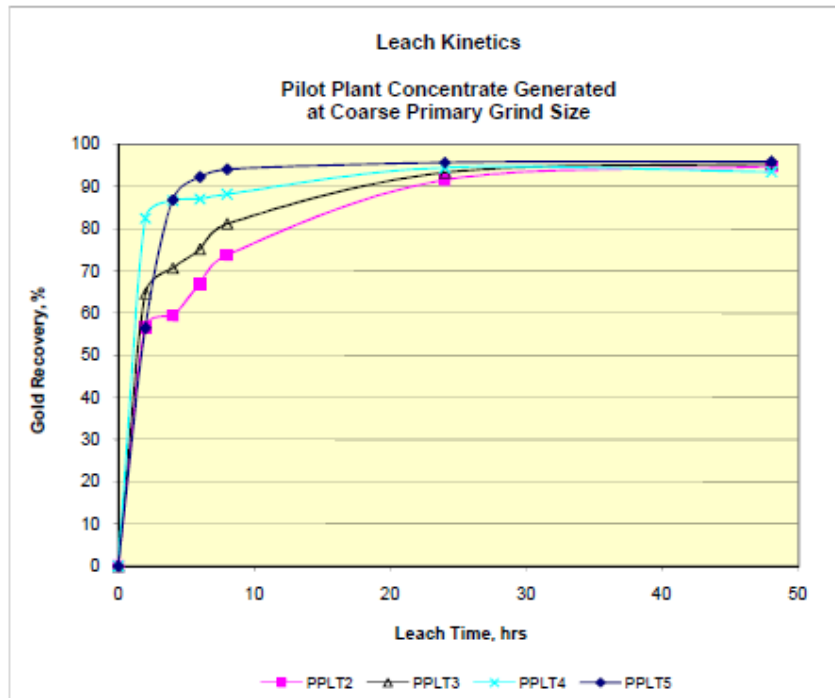


Figure 13-8: Leach Kinetics of Pilot Plant Concentrate Leach Test Work at Coarser Feed Size ($P_{80} = 125 \mu\text{m}$)

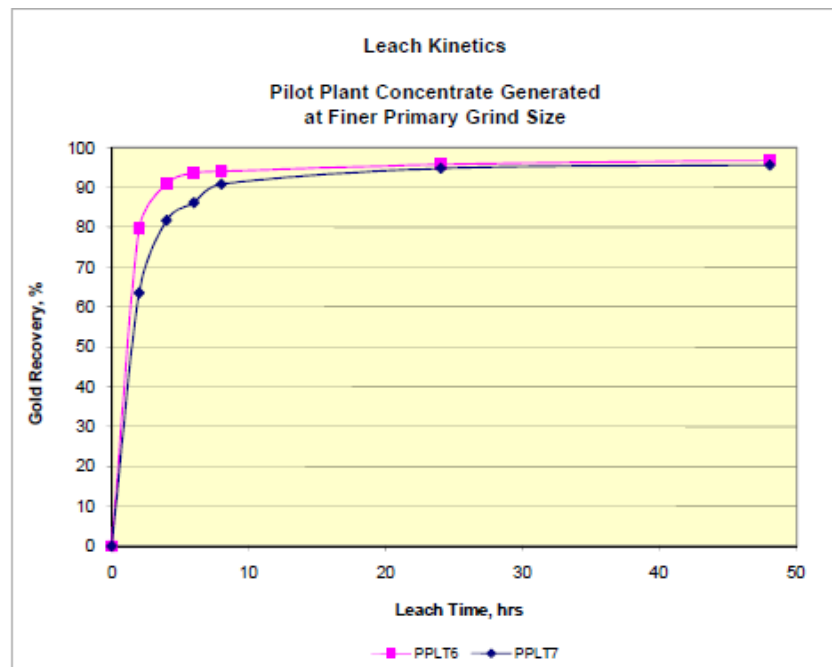


Figure 13-9: Leach Kinetics of Pilot Plant Concentrate Leach Test Work at Finer Grind Size ($P_{80} = 85 \mu\text{m}$)

The pilot plant test work results are summarized below:

- Leaching at concentrate (flotation feed size; $P_{80} = 125 \mu\text{m}$) achieved recoveries in excess of 94%
- Re-grinding or leaching at a finer particle size ($P_{80} = 85 \mu\text{m}$) improved recoveries (~97%)
- Leaching at increased cyanide concentrations (2 g/L vs 5 g/L) did not display improved gold recoveries but increased cyanide consumption
- Leaching kinetics began to plateau after 32 hours

In 2017, SGS completed additional leaching test work to test pre-aeration, dissolved oxygen, leach concentration and reagent dosage. The tests were leached over a 32-hour period with a finer concentrate ($P_{80} = 22 \mu\text{m} - 51 \mu\text{m}$). The test results are summarized in Table 13-21.

The SGS additional test work can be summarized below:

- Extraction of gold was greater than 94% in all tests
- Increasing the cyanide concentration from 1 g/L NaCN to 2 g/L NaCN resulted in a slight increase in gold extraction from 95.0% to 95.7% and improved leach kinetics
- Increasing the cyanide concentration to 3 g/L NaCN did not affect the gold recovery

The design criteria for leaching conditions and recovery are best represented in the pilot plant test work PP-LT2 and PP-T4 as pilot testing results are more reliable and since re-grind or a finer grind size was not chosen for this process. The average gold recovery of these two tests were used for the design criteria. Since the recovery of the pilot cyanide leach tests were calculated after carbon assays, the recovery used for the design criteria has already taken into account for gold loss in solution or carbon.

The design criteria for cyanide leaching is summarized in Table 13-22.

Table 13-21: SGS Leach Kinetics Test Work Summary

CN Test No.	Grind P ₈₀ μm	Actual DO mg/L		Preaer. Sol'n mg/L		Leach NaCN g/L	Reagents, kg/t of CN feed				Au Extr'n %	CN Residue g/t		CIP Recovery %		CIP Residue g/t		Head (calc) g/t	
		Preaer	CN	S ₂ O ₃	Au		Added		Consumed			Au	Ag	Au	Ag	Au	Ag	Au	Ag
							NaCN	CaO	NaCN	CaO									
CN-1	51	3-5	14-26	58	<0.01	1	2.53	1.66	1.70	1.66	95.6	1.86	13.4	95.0	76.4	1.70	12.4	35.2	52.5
CN-2	51	3-4	9-27	65	<0.01	2	3.99	1.60	2.09	1.60	95.9	1.70	<10	95.7	83.6	1.51	7.5	34.0	45.6
CN-4	51	4-5	11-13	55	<0.01	2*	3.56	1.08	2.32	1.06	95.8	1.62	9.3	95.8	86.2	1.45	6.7	35.5	48.5
CN-3	51	4-5	7-28	20	<0.01	3	5.52	1.53	2.54	1.53	96.2	1.57	<10	95.8	86.3	1.51	6.4	34.6	46.7
CN-5	33	4	7-13	120	<0.01	3	5.61	1.79	2.78	1.72	97.6	0.97	7.9	97.1	86.9	0.97	6.4	34.3	48.8
CN-6	22	3	9-14	250	0.05	3	6.16	1.71	3.44	1.66	98.3	0.68	6.7	98.3	88.9	0.58	5.7	34.9	51.2
CN-7	51	4	6-8	52	<0.01	3	6.36	1.33	4.01	1.21	97.9	-	7.2	94.7	85.4	1.89	7.6	36.6	52.2

* CN concentration maintained until 8h only

Table 13-22: Design Criteria – Cyanide Leaching

Criterion	Units	Design	Notes
Flotation Feed Grind Size, 80% passing	µm	125	Potential opportunity to reduce feed size
Cyanide Concentration	g/L	2.0	Based on pilot plant recoveries and reagent consumption
Leach Retention Time	h	40	Leaching appears to plateau after 32 hours
pH	-	11.0 - 11.6	Based on pilot plant test work
Solids Density	% wt	42	Based on pilot plant test work
Gold Extraction	%	94.5	Average of most representative pilot plant test work (PPLT2 and PPLT4)
Cyanide Consumption	kgt leach feed	5.0	Based on pilot plant test work and client recommendations
Lime Consumption	kgt leach feed	1.0	Based on pilot plant test work and client recommendations
Slurry Dissolved Oxygen Target	ppm	15.0	Hatch Recommended

13.10.1 Cycle Test

SGS conducted a single test in 2017 which recycled CIP barren solution for 3 cycles to determine the effects on recovery. The cycle test was based on the conditions of CN-3 (see Table 13-20). The leach was maintained at 3 g/L NaCN without any regrind. The results are presented in Table 13-23.

Key points from the cycle test work:

- Recoveries were unaffected by recycling
- Requirement for fresh cyanide was reduced by ~ 1.5 kg/t NaCN through recycling

Table 13-23: Cycle Test Summary

CN Test No.	Actual DO mg/L		Reagents, kg/t of CN feed					Au Extr'n %	CN Residue g/t		CIP %		CIP Residue g/t		Head (calc) g/t	
			Added			Consumed										
	Preaer	CN	NaCN	NaCN*	CaO	NaCN	CaO		Au	Ag	Au	Ag	Au	Ag	Au	Ag
CN-3 (baseline)	4-5	7-28	5.52		1.53	2.54	1.53	96.2	1.57	<10	95.8	86.3	1.51	6.4	34.6	46.7
CN-8A (cycle 1)	5	8-18	5.69		1.09	2.85	1.02	96.0	1.58	8.0	95.6	86.2	1.54	7.1	35.5	51.5
CN-8B (cycle 2)	5	7-12	5.41	4.12	1.29	2.31	1.16	96.2	1.50**	13.4	96.3	87.7	1.28	6.6	35.1	53.7
CN-8C (cycle 3)	3	10-17	5.40	4.18	1.42	2.60	1.35	96.3	1.53	7.6	95.3	86.9	1.64	6.7	36.3	51.2

*fresh cyanide only

**estimated value, actual assay was 5.28 g/t Au

13.10.2 Gold and Silver Loading

Gold and silver loading isotherm test were completed by SGS in 2017. The pregnant solution was obtained from following leach test CN-3 (see Table 13-21). The leach was maintained at 3 g/L NaCN without any regrind. Carbon was added for 72 hours for stabilization. The results for the isotherm tests are plotted in Figure 13-10 and Figure 13-11.

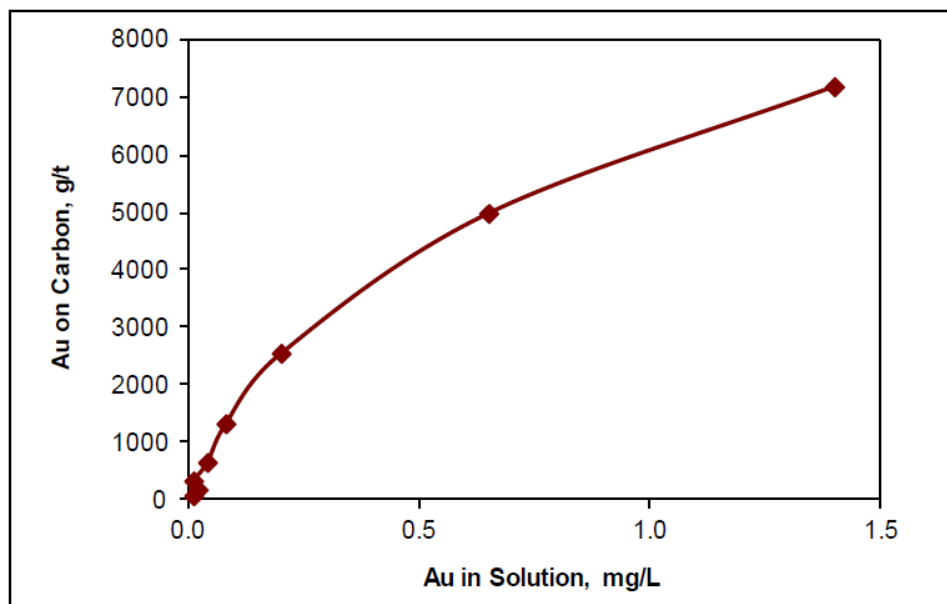


Figure 13-10: Equilibrium Loading of Gold on Carbon

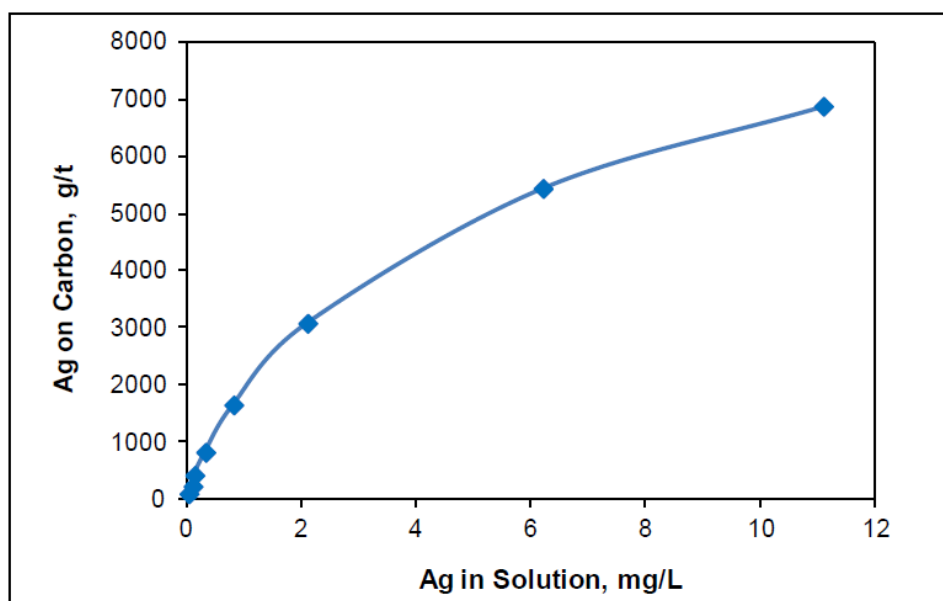


Figure 13-11: Equilibrium Loading of Silver on Carbon

13.11 CYANIDE DESTRUCTION

Various cyanide destruction treatment options including Cold Caro's Acid, CombinOx, a combination of Cold Caro's Acid/CombinOx method, sodium hypochlorite, SO₂/air process, activated carbon, and Caroate method were tested.

The target of the pulp treatment program was to reduce the total cyanide concentration to less than 1 mg/L as a first stage and the supernatant treatment program to be less than 0.2 mg/L for total cyanide (CN_T) and for free cyanide (weak acid dissociable) (CN_{WAD}) concentrations.

At the time of this report, effluent release standards based on CONAMA resolution 430 (May 13, 2011), the Brazilian federal requirement, for cyanide concentration in effluents was:

- Total cyanide 1.0 mg/L CN
- Free cyanide (weak acid dissociable) 0.2 mg/L CN

SO₂/ Air process followed by aging was found to be the preferred cyanide detoxification method as the cyanide levels could be significantly reduced after the process (to 0.41 CN_T mg/l and 0.2 CN_{WAD} mg/l) and achieve < 0.2 mg/l for both CN_T and CN_{WAD} after only 10 days of aging (Table 13-26 and Table 13-27 provide more details). In addition, this process is a straight-forward and reputable method with low operating costs and utilized worldwide.

13.11.1 Initial Cyanide Destruction Test Work

Initial and preliminary cyanide destruction test work was conducted by Wardell Armstrong in 2010 but was not successful in reaching the target levels.

CyPlus performed various cyanide destruction tests in 2011 including the SO₂/ air process. Following the SO₂/Air process, CyPlus was able to reduce the pulp to total cyanide levels below 2 mg/L. However, the aging test on the supernatant was unable to achieve the final supernatant target of 0.2 mg/L total cyanide even after a twentyfold dilution.

CyPlus also conducted aging tests by treating both leach tailings and flotation tailings together. The cyanide concentration did not decrease after dilution but increased (attributed to the presence of ferrocyanide in the effluent). This prompted the decision to treat leach tailings and flotation tailings separately.

13.11.2 Batch and Continuous Cyanide Destruction Test Work

SGS performed batch and continuous cyanide destruction test work in 2013 focusing on the SO₂/Air process and aging tests. Preparation for cyanide destruction test work included pre-aeration and leaching the flotation concentrate at 5.0 g/L NaCN for 48 hours with a continuous supply of air. 10 g/L of pre-attrition activated carbon (CIP) was added to the pregnant leach pulp to recover gold.

Batch and continuous cyanide destruction tests (SO₂/Air process) were performed on the barren leached products. Batch tests were conducted to optimize reagent dosage and retention time. The results from the test work is summarized in Table 13-24.

Table 13-24: Cyanide Destruction Test Work Results by SGS (2013)

Test No	Mode	Retention Time, min	Feed/ Product	pH	FeCl ₃ addition in Feed, mg/L	FeCl ₃ addition in product, mg/L	Assays, mg/L				Cumulative Reagent Addition					
							CN _T	CN _{WAD}	Cu	Fe	g/g CN _{WAD}			g/L pulp		
											SO ₂ eq.	Lime	Cu	SO ₂ eq.	Lime	Cu
CND B1	Batch	120	Feed	10.7	-	-	2550	1520	361	274	6.61	2.88	0.22	8.51	3.71	0.28
			Product	8.6			1967	347	462	211						
CND B2	Batch	150	Feed	10.7	-	-	2550	1520	361	274	19.3	11.0	0.53	24.9	14.2	0.68
			Product	8.6			1.08	0.20	32.3	0.20						
CND B3	Batch	150	Feed	10.2	-	-	2550	1520	361	274	23.1	13.4	0.66	29.8	17.3	0.85
			Product	8.6			30.9	1.63	1.03	0.03						
CND C1	Continuous	84	Feed	10.2	-	15.0	2550	1520	361	274	20.8	12.0	0.20	26.8	15.5	0.26
			Product	8.6			45.5	1.14	103	85.8						
CND B4	Batch	180	Feed	10.2	15.0	-	2550	1520	361	274	8.50	6.15	0.66	10.9	7.93	0.85
			Product	8.5			1.75	0.84	-	-						
CND C2	Continuous	200	Feed	10.2	15.0	-	2550	1520	361	274	25.6	20.8	1.26	33.0	26.8	1.62
			Product	8.6			1.50	0.69	0.37	0.34						
CND B5	Batch	180	Feed	10.2	15.0	-	2550	1520	361	274	11.0	6.81	0.66	14.2	8.77	0.85
			Product	8.6			0.86	0.76	-	-						
CND C3	Continuous	186	Feed	10.2	15.0	-	2550	1520	361	274	23.0	15.6	0.88	29.6	20.1	1.13
			Product	8.6			0.34	0.46	1.31	0.2						

Aging tests were tested on cyanide destruction product itself (Aging-01) and a blend of flotation tailings and cyanide detoxed product (Aging-02). The tests were run for 55 days at room temperature (~25°C) to determine if environmental disposal can be applicable. The results from the aging test work is summarized in Table 13-25.

Table 13-25: Aging Test Work by SGS (2013)

Time	CN Species, mg/L	Aging -01	Aging -02
Day-01	CN _T	0.86	14.9
	CN _{WAD}	0.76	0.51
Day-07	CN _T	0.25	9.05
	CN _{WAD}	0.17	0.20
Day-15	CN _T	0.20	3.33
	CN _{WAD}	0.16	0.21
Day-33	CN _T	0.24	0.46
	CN _{WAD}	0.23	0.20
Day-45	CN _T	32.7	0.45
	CN _{WAD}	1.22	0.07
Day-55	CN _T	N/A	0.28
	CN _{WAD}	N/A	0.08

N/A=Not Applicable

The detoxified leached flotation concentrate contained less than 1.0 mg/L total cyanide by using the SO₂/Air process. The cyanide detox product was then aged for 55 days to lower CN_T below 0.3 ppm. It was unable to reach the target of 0.2 ppm required for environmental discharge.

Aging with flotation tailings show higher cyanide levels associated with insoluble precipitation with iron or copper. This agrees with previous CyPlus test work which suggests separating flotation tailings from cyanide tailings.

13.11.3 Confirmation Cyanide Destruction Test Work

Additional cyanide destruction test work using the SO₂/Air process was completed by SGS in 2017. The flotation concentrate received was much finer (P₈₀ = 51 µm) but was leached at cyanide concentration levels more representative to the design criteria. The concentrate was pre-aerated and leached for 32 hours with oxygen being sparged. 15 g/L of pre-attrition activated carbon (CIP) was added to the pregnant leach pulp to recover gold and silver.

The test conditions were based on previous cyanide destruction test work conducted by CyPlus in 2011 which included 3 stages. Copper sulfate and air were added at the start of the cyanide destruction test. For the 1st stage, copper sulfate and sodium metabisulfite was added continuously until a low residual cyanide level was reached (target < 1 mg/L CN_T). The overflow would enter the 2nd stage, where additional copper sulfate and air was added. The 3rd stage would include adding ferric chloride. Each stage had a retention time of approximately 1 hour. The results from the test work are summarized in Table 13-26.

Table 13-26: Confirmatory Aging Test Work by SGS (2017)

Aging Time days	Solution Analysis			
	CN _T mg/L	CN _{WAD} mg/L	Cu mg/L	Fe mg/L
1	< 0.1	< 0.1	3.11	< 0.05
3	0.16	< 0.1	2.23	< 0.05
7	0.29	0.2	1.84	< 0.05
10	0.12	< 0.1	1.28	< 0.05
14	0.16	0.15	1.25	< 0.05

The barren pulp from the most representative cyanide destruction test work (Test CND1-3) was used for an aging test. The results are shown in Table 13-27.

Tests CND1-1 and CND 1-3 were able to reduce the total cyanide concentration to less than 1 mg/L in two stages.

Test CND1-2 was terminated earlier since the CN_T levels did not reach this target. This may be due to the lack of copper addition in the first stage.

Test CND1-3 is identical to CND1-1 without the 3rd stage and confirms that the target cyanide concentration can be reached without ferric chloride. The solution from this test was used for aging test and was able to stabilize and reach the environmental discharge limit (< 0.2 mg/L CN_T and CN_{WAD}) after 14 days. Test CND1-3 was used as the basis for the cyanide destruction design criteria.

Table 13-27: Confirmatory Cyanide Destruction Test Work Results by SGS (2017)

Feed/Test	Total Test Time min.	Retention Time min per stage	Stage	Test Conditions								Test Results Solution Analysis					
				pH	emf mV	DO mg/L	Reagent Addition g/g CN _{WAD}			Reagent Addition							
							SO ₂ Equiv.	Lime	Cu	mg/L solution							
										Cu	FeCl ₃						
Feed:				9.6								1050	799		420	104	
CND1-1	300	54	1	8.5	174	3.2	5.50	3.63	0.02	19	19	<0.1	< 0.1	5.8	14.0	<0.2	
			2	8.4	153	8.0	0	0	0.03	21	21	<0.1	< 0.1	0.6	4.2	<0.2	
			3	8.2	165		0	0	0	0	5	<0.1	< 0.1	0.5	3.6	<0.2	
Feed:												1100	649				
CND1-2	150	64	1	8.7	100	3.3	7.08	4.1	0	0	0	0.56	0.1	9.2	18.5	0.1	
			2	8.2	166	8.0	0	0	0	23	0	0.25	0.1	2.2	2.9	0.1	
Feed:												998	674				
CND1-3	360	60	1	8.4	188	2.4	6.00	4.7	0.03	22	0	0.13	<0.1	0.5	6.39	<0.05	25.7
			2	8.2	174	8.8	0	0	0.03	21	0	0.41	0.2	0.8	4.28	<0.05	23.9

* CN_{WAD} analysis by picric acid method

13.11.4 Cyanide Destruction Design Criteria

A summary of the cyanide destruction test work is shown in Table 13-28 for comparison.

Table 13-28: Cyanide Destruction Test Work Summary

Test Work	Leach Cyanide Concentration	Feed CN _T	Feed CN _{WAD}	SO ₂	Product CN _{WAD}	After Aging CN _T	After Aging CN _{WAD}
	g/L	mg/L	mg/L	g/g CN _{WAD}	mg/L	mg/L	mg/L
4	3	1050	674	6.0	0.20	0.16	0.15
SGS 2013/ CNDC3	5	2550	1520	23	0.46	0.28	0.08
CyPlus 2011/ T25	-	988	302	5.3	0.60	>10	>10

Test CND1-3 from the SGS report in 2017 was able to successfully use the SO₂/Air process to reduce the total cyanide concentration to below target (< 2 mg/L CN_T) and less than 0.2 mg/L CN_T after aging.

The design conditions and reagent addition for cyanide destruction is summarized in Table 13-29.

Table 13-29: Design Criteria – Reagent Addition for Cyanide Destruction

Reagent	Units	Design	Notes
SMBS	kg/t Mill Feed	0.203	Assumed 50% recovery from thickener
Copper Sulfate	kg/t Mill Feed	0.001	Assumed 50% recovery from thickener
Lime	kg/t Mill Feed	0.107	Assumed 50% recovery from thickener

13.12 ENVIRONMENTAL TEST WORK

Environmental test work was carried out by SGS in 2017 to assess contaminant release potential associated with CIP tailings from the SO₂/air process. The test work carried out on the barren pulp generated from Test CND 1-3 (See Section 13.11.3) was able to reduce CN_T < 1 ppm and was used as the design basis for cyanide destruction.

The effluent from test CND 1-3 was analyzed for the following:

- Analysis 1: CND 1-3 solution prior to aging: Cyanide, Cu, Fe and ammonia (Table 13-26)
- Analysis 2: CND 1-3 solution after aging: Cyanide, Cu and Fe (Table 13-27)
- Analysis 3: CND 1-3 solution prior to aging: Full Chemical Suite (ICP, ammonia, cyanide, sulphate and thiocyanate)

The effluent discharge must comply with the Effluent Discharge for Para State, and Federal standards (CONAMA). Assumptions and considerations for the water treatment of effluent are summarized below:

- The total copper in the solution after fourteen (14) days of aging is reduced from 4.28 mg/L to 1.25 mg/L and is slightly above the Federal limit of 1.0 mg/L total copper in solution.
- A higher level of ammonia was identified in the post-detox effluent. The concentration of total unionized ammonia was 23.9 mg/L from Analysis 1, whereas the required minimum level is 20 mg/L. No aging test was performed to measure the ammonia degradation.
- It is expected that the metals concentration level will decline due to dilution from rainfall precipitation and natural degradation in the tailings pond.
- The decant solution contains very high levels of sulphate (Analysis 3: 5,500 mg/L). There are no sulphate limits; however, this will impact treatment selection. Currently, there is no discharge limit for thiocyanate; however, thiocyanate concentration in the residue decant is high (Analysis 3: 520 mg/L) and may require further treatment if thiocyanate limits and/or toxicity tests are imposed. There is currently no treatment allowed for thiocyanate.

A bench-scale test should be completed with aged effluent to confirm design basis for water treatment plant. Feasibility level design was not completed for the water treatment system, as the design basis is not fully developed at this stage.

13.13 THICKENER TEST WORK

Thickener and flocculant test work was completed by FLSmidth in 2010. Samples were generated from the Wardell Armstrong pilot plant and locked cycle test work.

Fine (P80 = 85 μ m), coarse (P80 = 125 μ m), TOP and BOT tailings were tested to determine flocculant dosage and settling rates. The results are shown in Figure 13-12 to Figure 13-15 and summarized in Table 13-30 to Table 13-31 and are based on sizing for a High Rate Thickener (HRT).

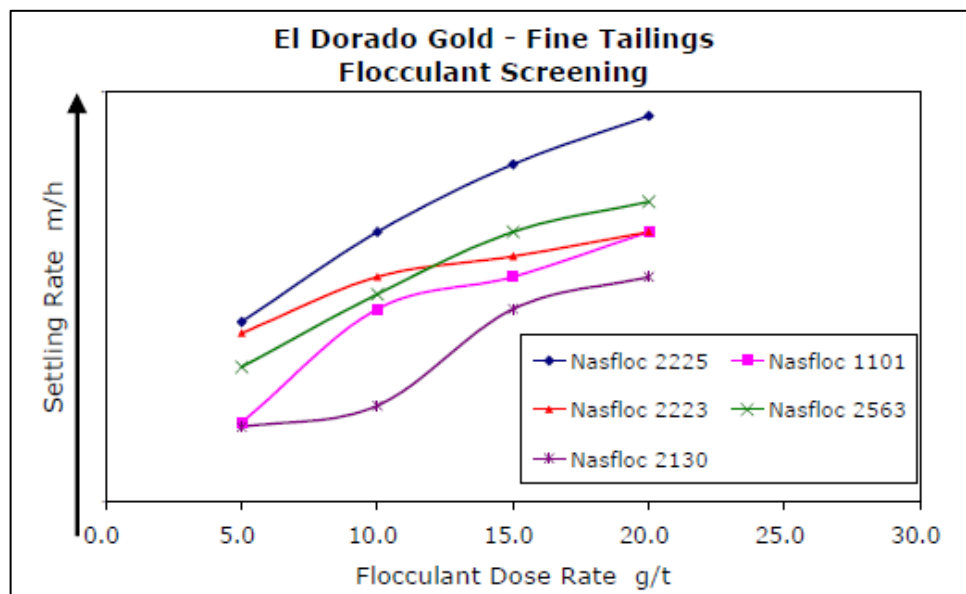


Figure 13-12: Flocculant Test Work for Fine Tailings

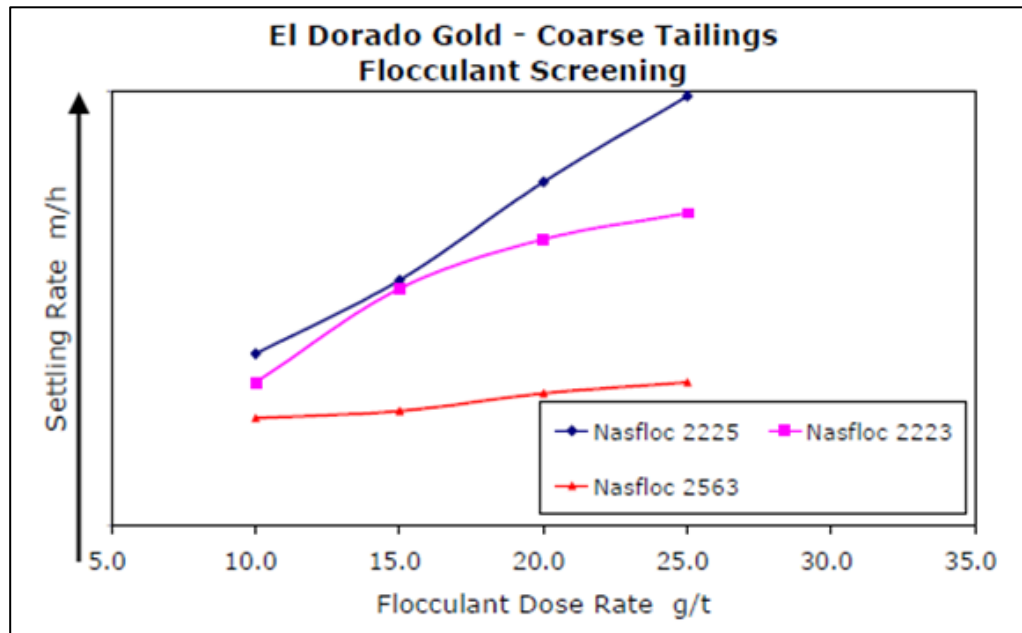


Figure 13-13: Flocculant Test Work for Coarse Tailings

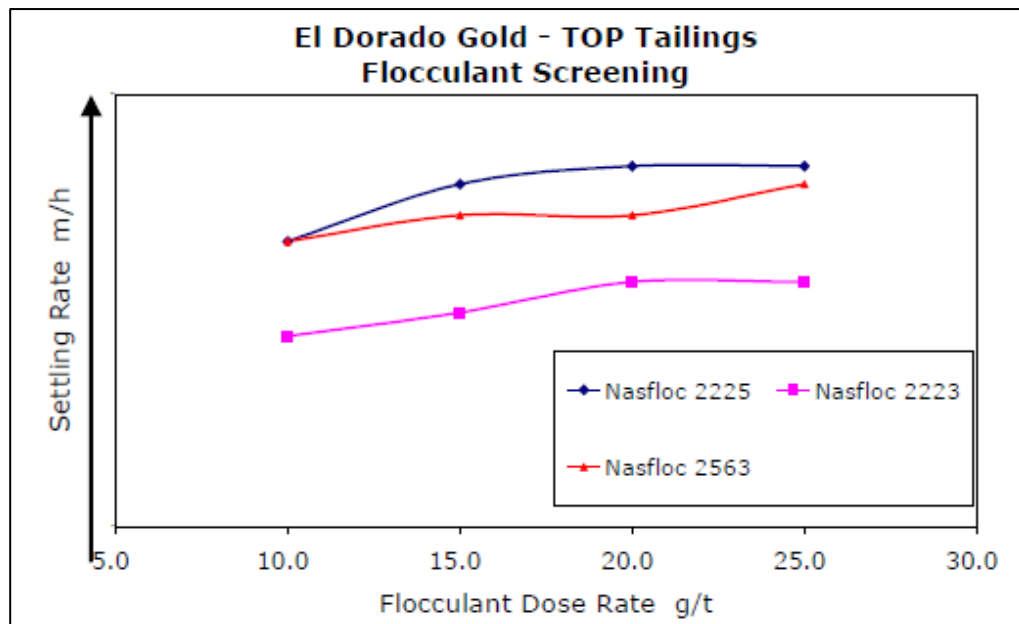


Figure 13-14: Flocculant Test Work for TOP Tailings

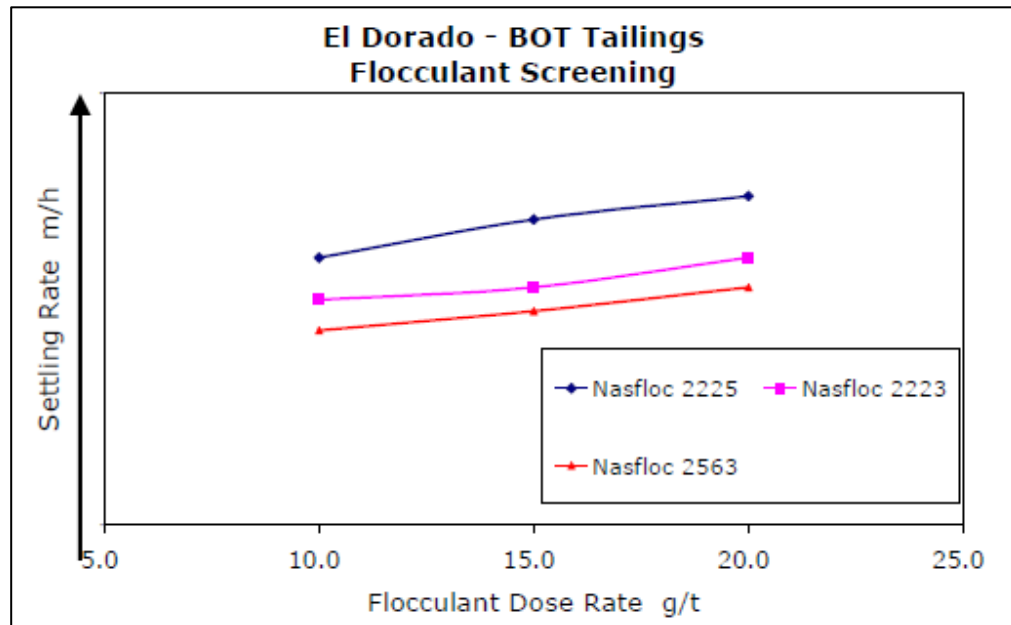


Figure 13-15: Flocculant Test Work for BOT Tailings

Table 13-30: Thickener Settling Test Summary on Flotation Concentrate

CONCENTRATE – THICKENER based on Coarse Concentrate	HRT
Feed Solids Concentration Required (wt%)	3
Recommended Flocculant Dose (g/mt)	35
Recommended Minimum Unit Area (m ² /mtpd)	0.348
Minimum Recommended Diameter (m)	16
Design Underflow Density (wt%)	35
Retention Time Required (hrs)	3.5
Yield Stress at Design Underflow (Pa)	<20

Table 13-31: Thickener Settling Test Summary on Leached Residue

CN RESIDUE – THICKENER based on Coarse CN Residue	HRT
Feed Solids Concentration Required (wt%)	3
Recommended Flocculant Dose (g/mt)	50
Recommended Minimum Unit Area (m ² /mtpd)	0.087
Minimum Recommended Diameter (m)	8
Design Underflow Density (wt%)	35
Retention Time Required (hrs)	1.25
Yield Stress at Design Underflow (Pa)	<20

Nasfloc 2225 provided the best flocculant settling rates. However, it was reported that the overflow clarity was found to be poor.

The settling rate for the leached residue was unexpectedly much faster than the flotation concentrate. Hatch recommends keeping the slower settling rate (higher value) for the determination of unit area since it is unusual to see such a big difference between similar materials.

The design criteria used for flocculant dosage and thickener sizing is summarized in Table 13-32.

Table 13-32: Design Criteria – Thickener

Flotation Concentrate Thickener	Units	Design	Notes
Solid Loading, Feed Rate	m ² /tpd	0.348	
Flocculant Dosage	g/t	35	
Leached Residue Thickener			
Type of Thickener	-	HRT	
Solid Loading, Feed Rate	m ² /tpd	0.348	Used same rate from flotation concentrate thickener
Flocculant Dosage	g/t	50	

13.14 OTHER TEST WORK COMPLETED

Other test work completed for the Eldorado gold deposit Tocantinzinho included:

- Heavy liquid separation – potential gold upgradeability and potential concentrate grade.
- Whole ore cyanide leach testing – 24-hour leach resulted in high recoveries and low cyanide consumption indicated this process could be a suitable alternative. A second phase of test work found that recoveries were not too sensitive to grind size up to 150 microns. Also, the entire Tocantinzinho ore is very amenable for cyanide leaching the entire body (both oxides and sulfides).
- Gravity concentration followed by flotation – Found that gravity concentration and flotation are equally effective at gold concentration. Combining both gravity and flotation does not improve gold recovery when compared with flotation alone. However, a combination of both would be beneficial treating partially or fully oxidized material.

13.15 RISKS AND OPPORTUNITIES

The risks and opportunities identified for mineral processing and metallurgical tests are summarized below.

13.15.1 Opportunities

- There is potential to decrease the size of the cyanide recovery thickener since the settling rate was conservatively assumed due to discrepancies with the test work
- There is potential to reduce or eliminate the oxygen plant since leaching can potentially use air instead of oxygen due to relatively small difference in gold recovery from the existing test results

- There is potential to reduce or eliminate the oxygen plant since cyanide destruction can use air instead of oxygen
- There is a potential that actual intensive cyanidation unit (ICU) recoveries may be higher than the assumed value (based on leaching tests) currently used in the design criteria if ICU test work is conducted.

13.15.2 Risks

- Even with extended metallurgical test programs, there is always a potential risk that the samples might not represent the general characteristics of the ore. The test program is considered quite adequate for most of the mineral processing stages investigated but this risk still exists.
- Variability in ore can cause high mass pull in flotation circuit and negatively impact leaching and CIP residence times and potentially reduce overall gold recovery.
- The comminution testing was conducted with a small number of samples (4 samples SAG milling and 7 for ball milling) and may compromise the comminution circuit design.

13.15.3 Recommendations

It is recommended to perform additional tests to confirm the design basis by doing:

- Comminution tests on additional samples
- Specific bench-scale leaching test work to assess ICU on gravity concentrate to develop a better basis for ICU recovery

It is recommended that a more detailed study on the CIP tailings and effluent detoxification is conducted to increase confidence in the design basis.

- This would include the following test work:
- Optimization of cyanide destruction test work on a larger scale
- Bench-scale tests with aged effluent after cyanide destruction
- Detailed water balance supported with detailed hydrology and hydrogeology studies
- Dilution/sensitivity analysis of the treated discharge
- Perform additional settling test for cyanide recovery thickener.

SECTION • 14 MINERAL RESOURCE ESTIMATES

The mineral resource estimates for the Tocantinzinho Project were carried out by the Technical Services team at Eldorado and reviewed by Rafael Gradim, P.Geo. The estimates were made from a 3D block model utilizing commercial mine planning software. Gold was estimated by ordinary kriging utilizing samples contained in a single estimation domain, constrained by a corridor of mineralized granites and an internal barren andesitic intrusion. Block model cell size is 10 m east x 10 m north x 10 m high.

14.1 GEOLOGIC AND MINERALIZATION MODELS

Eldorado built three main 3D geological models to build the estimation and block model coding (Figure 14-1):

- The main NW-SE structural corridor defining a volume of mineralized granites (Mineralization Corridor), and bound on the footwall and hanging wall by unmineralized hematite granite and quartz monzonite units.
- The main andesitic Intrusion in the central upper portion of the deposit. Other smaller andesitic dykes within the granitic body were not modeled and represent local areas of weaker mineralization or internal dilution in the resource model.
- Top of fresh rock surface, typically marking the contact between un-weathered granite and saprolite.

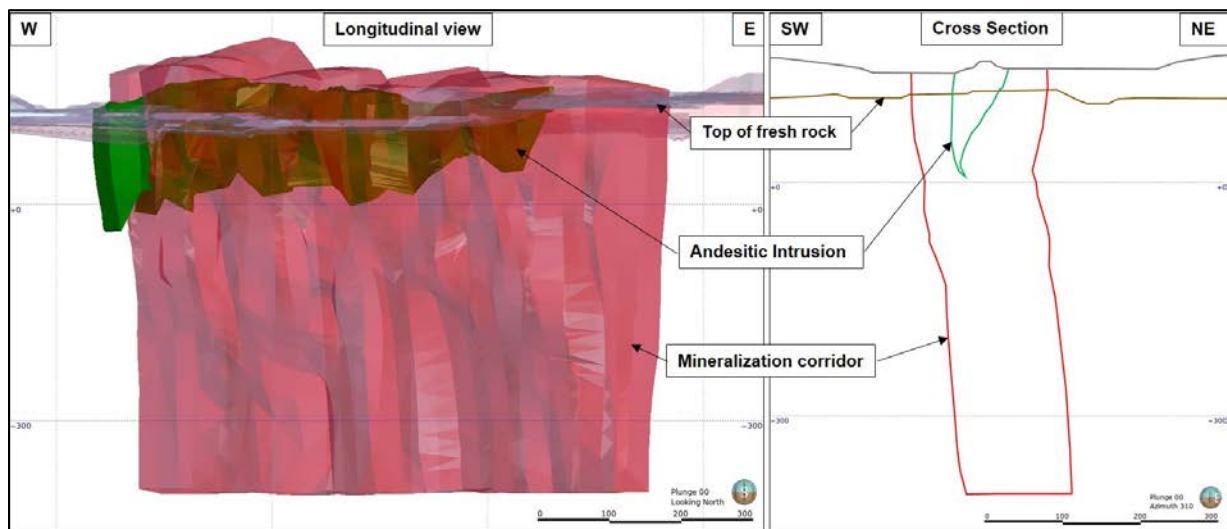


Figure 14-1: Main 3D Geological Models

To constrain gold grade interpolation for the Tocantinzinho deposit, Eldorado created 3D mineralized envelopes, or shells solely within the Mineralization Corridor (i.e. andesitic intrusion excluded). The shells were based on initial outlines derived by a method of probability assisted constrained kriging (PACK), involving the following steps:

- The threshold value of 0.30 g/t Au was determined by inspection of histograms and probability curves as well as indicator variography.
- Assay samples are converted into a binary indicator code equal to one if the composited assay is equal to or higher than 0.30 g/t Au or zero if the assay is less than 0.30 g/t.
- The indicators are estimated into a block model generating a range of probability values between zero and one.
- Shell outline selection was done by inspecting contoured probability values. The 42% probability value was determined to be the best fit for defining the boundaries of mineralization above threshold grade.
- The 42% shell was then clipped to be bound inside the mineralization corridor, and edited on plan and section views to remove artifacts and isolated portions without 3D continuity.
- Assays samples are selected inside the PACK shell and outside the modeled andesitic intrusion. Figure 14-2 shows the relationship between the PACK, or mineralized shell, and the lithology units.

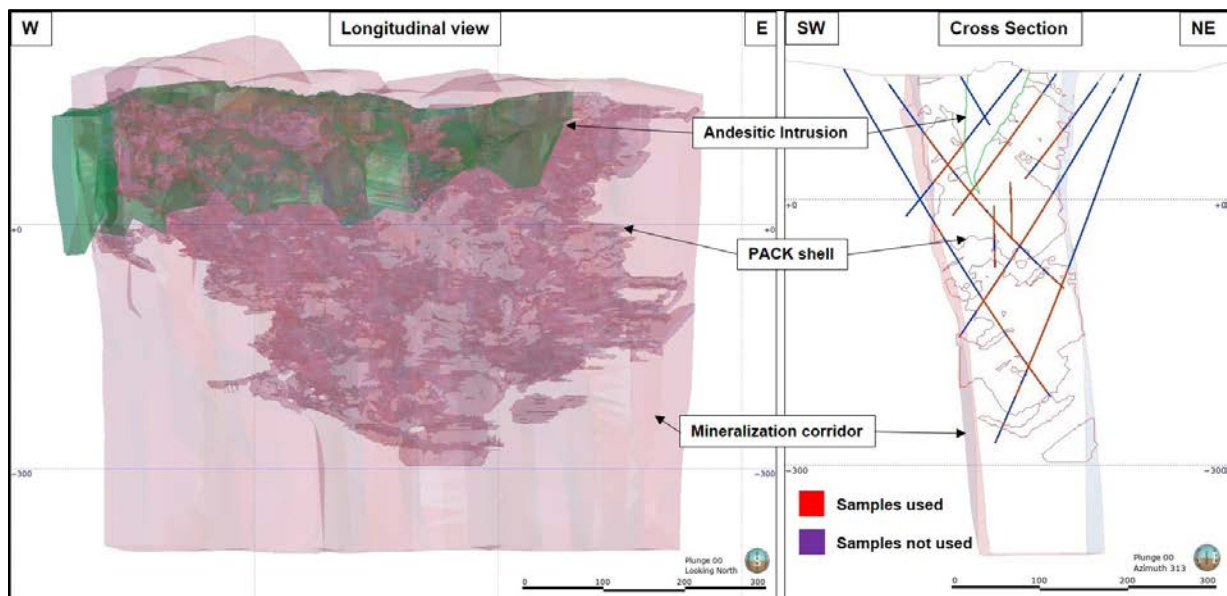


Figure 14-2: Relationship between PACK or Mineralized Shell and Main 3D Geological Models

14.2 DATA ANALYSIS AND ESTIMATION DOMAIN

The volume defined by the PACK shell and the exclusion of the andesitic intrusion define a single hard boundary estimation domain for the purposes of sample selection and block estimation. In this context, a hard boundary means that estimation inside the domain is not affected or informed by samples outside the domain (Figure 14-2, where drill holes samples in red are used in the estimation and those in blue are not used). Additionally, blocks outside the estimation domain are not estimated for grade.

The decision to employ a single estimation domain was made after investigating possible sub-domains controlled by the type of alteration (salami versus smoky granite) and weathering (weathered versus un-weathered zones). Contact profiles or plots generated to explore the relationship between grade and geological units within the mineralized shell showed transitional trends. Thus no internal hard grade boundary was used.

Assays samples inside the estimation domain were capped and then composited to 2 m interval lengths to be used as input to the estimation. Descriptive statistics for original samples, capped, and composited-capped samples inside the estimation domain are summarized in Table 14-1. The capped and composited samples show a global mean of 1.37 and adequate coefficient of variation (CV) of 1.45.

Table 14-1: Descriptive Statistics for Samples inside the Estimation Domain – Au g/t data

	No. of Samples	Minimum	Maximum	Mean*	CV
		g/t	g/t	g/t	
Original	9486	0.003	374.40	1.43	2.73
Capped	9486	0.003	25.00	1.37	1.67
Composited	8571	0.003	25.00	1.37	1.45

*Weighted by sample length

14.3 EVALUATION OF EXTREME GRADES

Extreme gold grades were examined using histograms and cumulative probability plots (Figure 14-3). The latter shows a slight inflection around the 99.9 percentile corresponding to 25 g/t Au. Capping at this value affects 35 samples, and reduces the global mean by 3.9%.

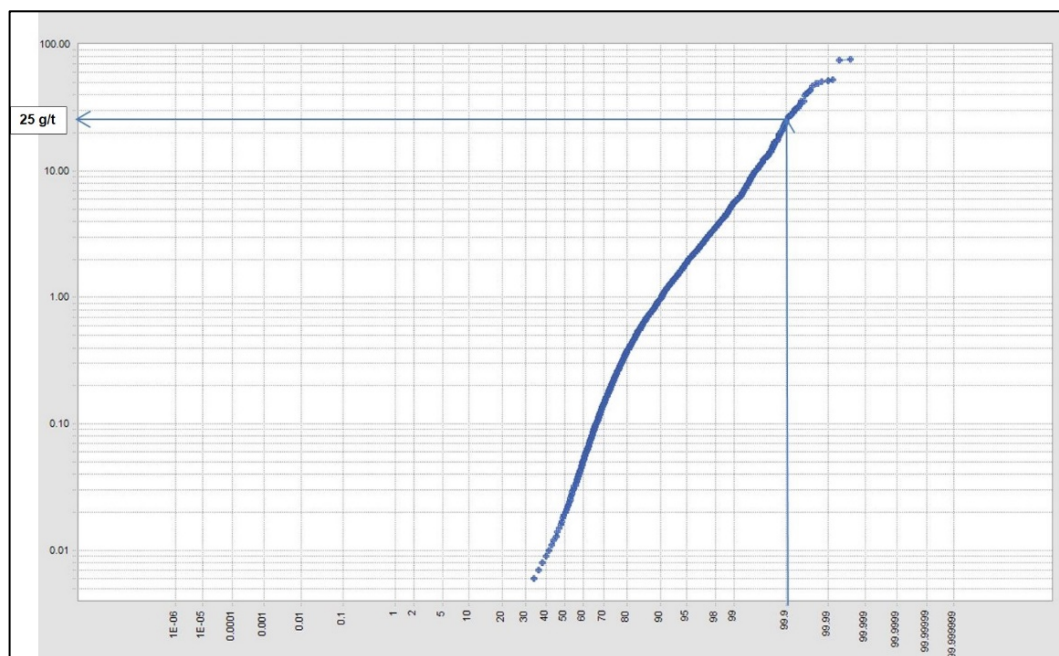


Figure 14-3: Cumulative Probability Plot for Gold Composed Samples inside the Estimation Domain

14.4 VARIOGRAPHY

Variography, a continuation of data analysis, is the study of the spatial variability of an attribute. Correlograms were calculated for gold inside the estimation domain. Variogram model parameters and orientation data of rotated variogram axes are shown in Table 14-2 and Table 14-3.

Gold was modeled using two spheroidal structures and a moderate nugget, both consistent with the grade continuity and known controls on mineralization.

Table 14-2: Au Variogram Parameters

	Model	Nugget Co	Sills		Rotation Angles						Ranges					
					°						m					
			C1	C2	Z1	X1'	Y1''	Z2	X2'	Y2''	Z1	X1'	Y1''	Z2	X2'	Y2''
Inside Mineralized Shell	SPH	0.200	0.779	0.021	-60	-9	97	-90	-5	-15	7.2	8.2	10.6	417	103	594

Notes: Models are spherical (SPH). The first rotation is about Z, left hand rule is positive; the second rotation is about X', right hand rule is positive; the third rotation is about Y'', left hand rule is positive.

Table 14-3: Azimuth and Dip Angles of Rotated Variogram Axes

	Axis Azimuth						Axis Dip					
	°						°					
	Z1	X1	Y1	Z2	X2	Y2	Z1	X1	Y1	Z2	X2	Y2
Inside Mineralized Shell	209	263	300	341	1	270	-7	79	-9	75	-14	-5

Notes: Azimuths are in degrees. Dips are positive up and negative down.

14.5 MODEL SETUP

The block size for the Tocantinzinho model was selected based on mining selectivity considerations (open pit mining). It was assumed the smallest block size that could be selectively mined as ore or waste, referred to the selective mining unit (SMU), was approximately 10 m x 10 m x 10 m. In this case, the SMU grade-tonnage curves predicted by the restricted estimation process adequately represented the likely actual grade-tonnage distribution. Projects limits, in UTM coordinates, are 577,565 to 578,935 E, 9,330,066 to 9,331,276 N, and -300 to +260 m elevation.

The assays were composited into 2.0 m fixed-length down-hole composites. The composite data were back-tagged by the mineralized shell and lithology units (on a majority code basis). The compositing process and subsequent back-tagging was reviewed and found to have performed as expected.

Various coding was done on the block model in preparation for grade interpolation. The block model was coded according to lithologic domain and mineralized shell (on a majority code basis). Percent below topography was also calculated into the model blocks as well as the weathered zone.

14.6 ESTIMATION

Modelling consisted of grade interpolation by ordinary kriging (OK) for the estimation domain inside the mineralized shell. Nearest-neighbour (NN) grades were also interpolated for validation purposes. Blocks and composites were matched on estimation domain.

The search ellipsoids were oriented preferentially to the orientation of the mineralized shell for within shell runs and structures defined in the spatial analysis for background runs. All searches had the longest ranges oriented NW – SE.

A three-pass approach was instituted for interpolation. The first pass employed a 65 m x 60 m x 35 m search ellipse and required a minimum of four (4) samples, allowed a total maximum of 9 samples, and maximum of three (3) samples per drill hole. The first pass was overwritten by a second pass with the same parameters but smaller estimation range (35 m x 35 m x 20 m). This approach was used to provide a better distribution of the local grades observed during visual checks. The third pass employed an 80 m x 75 m x 45 m search ellipse and required a minimum of two (2) samples, allowed a total maximum of six (6) samples, and maximum of two (2) samples per drill hole. This approach was used to enable most blocks to receive a grade estimate. On all three passes, an

outlier restriction was also used to control the effects of isolated high-grade composites. Composites with grade greater than 20 g/t Au could only be used if they are less than 25 m of a block center.

These parameters were based on the geological interpretation, data analyses, and variogram analyses. The number of composites used in estimating grade into a model block followed a strategy that matched composite values and model blocks sharing the same ore code or domain. The minimum and maximum number of composites was adjusted to incorporate an appropriate amount of grade smoothing. This was done by change-of-support analysis (Discrete Gaussian or Hermitian polynomial change-of-support method), as described in the validation section below.

Bulk density data were assigned by rock type. The measured density data were averaged by material type and the averages used for the assignment process. The ore hosting granite value equalled 2.62 in the primary region and 1.80 in the weathered portion.

14.7 VALIDATION

14.7.1 Visual Inspection

Eldorado completed a detailed visual validation of the Tocantinzinho resource model. According to Eldorado the model was checked for proper coding of drillhole intervals and block model cells, in both section and plan. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values.

14.7.2 Model Check for Change-of-Support

An independent check on the smoothing in the estimates was made using the Discrete Gaussian or Hermitian polynomial change-of-support method. This method uses the “declustered” distribution of composite grades from a nearest-neighbour or polygonal model to predict the distribution of grades in blocks. The histogram for the blocks is derived from two calculations:

- The block-to-block or between-block variance;
- The frequency distribution for the composite grades transformed by means of hermite polynomials (Herco) into a less skewed distribution with the same mean as the declustered grade distribution and with the block-to-block variance of the grades.

The distribution of hypothetical block grades derived by the Herco method is then compared to the estimated grade distribution to be validated by means of grade-tonnage curves. The grade-tonnage predictions produced for the model show that grade and tonnage estimates are validated by the change-of-support calculations over the range of mining grade cut-off values (0.3 g/t to 0.5 g/t Au).

14.7.3 Model Check for Bias

The block model estimates were checked for global bias by comparing the average metal grades (with no cut-off) from the model with means from nearest-neighbour estimates (NN). The nearest-neighbour estimator declusters the data and produces a theoretically unbiased estimate of the

average value when no cut-off grade is imposed and is a good basis for checking the performance of different estimation methods. For the mineralized shell domain, there is good agreement between the two estimates, as the kriged mean grade is 0.86% lower than the NN mean.

The model was also checked for local trends in the grade estimates by grade slice or swath checks. This was done by plotting the mean values from the nearest-neighbour estimate versus the kriged results for benches, northings and eastings (in 10 m swaths, see example in Figure 14-4). The kriged estimate should be smoother than the nearest-neighbour estimate, thus the nearest-neighbour estimate should fluctuate around the kriged estimate on the plots. The observed trends behave as predicted and show no significant trends of gold in the estimates in Tocantinzinho model.

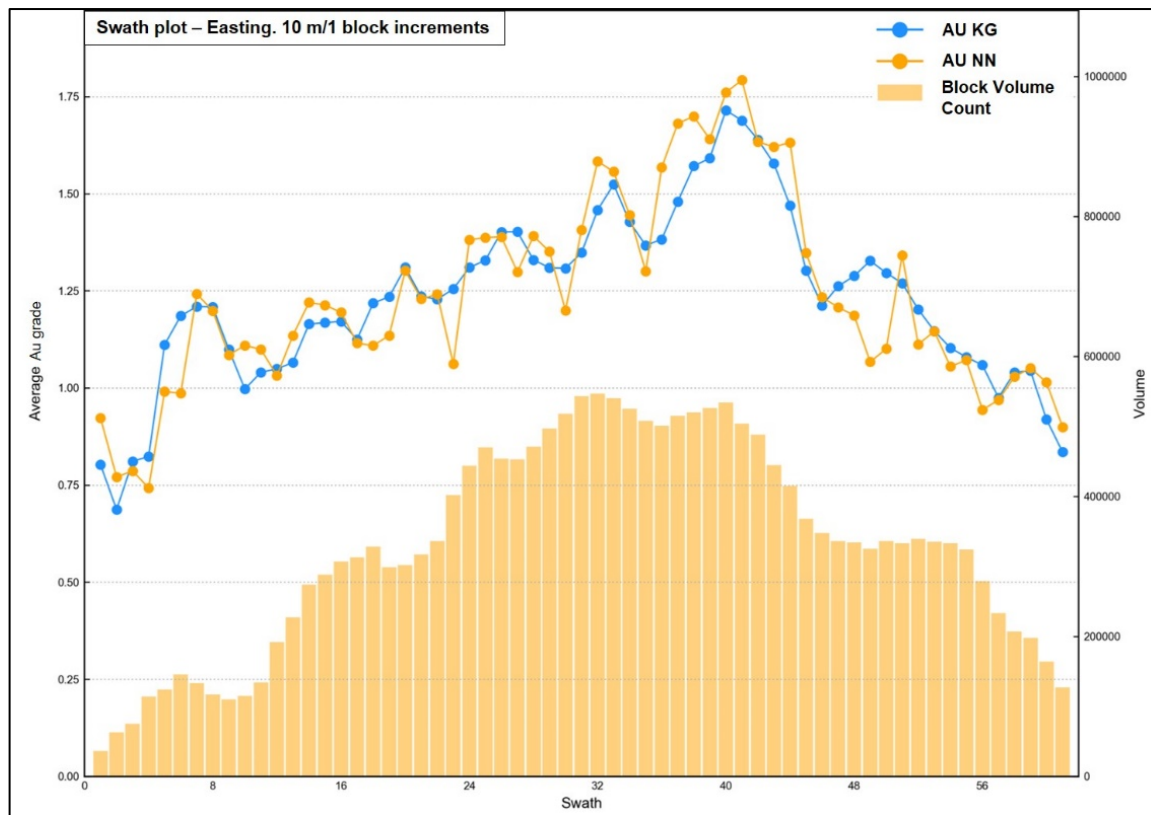


Figure 14-4: Example of Swath Plot (easting) comparing OK and NN Estimates

14.8 MINERAL RESOURCE CLASSIFICATION

The mineral resources of the Tocantinzinho deposit were classified using logic consistent with the Canadian Institute of Mining (CIM) definitions referred to in NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified into measured, indicated, and inferred mineral resource categories.

Inspection of the Tocantinzinho model and drill hole data on plans and sections, combined with spatial statistical work, contributed to the setup of various distance to nearest composite protocols to help guide the assignment of blocks into measured or indicated mineral resource categories.

Reasonable grade and geologic continuity is demonstrated over most of the Tocantinzinho deposit, which is drilled generally on 35 m spaced sections. A two-hole rule was used where blocks containing an estimate resulting from two or more samples, all within 45 m and from different holes, were classified as indicated mineral resources.

Where the sample density was high along adjacent sections, the confidence in the grade estimates were the highest thus permissible to be classified as measured mineral resources. A three-hole rule was used where blocks containing an estimate resulting from three or more samples, all within 45 m and from different holes, were classified as measured mineral resources. Results of these assignment protocols were used to create 3D shells for tagging the model as indicated or measured mineral resources. Constructing these tagging shells helped eliminate the presence of artefacts common in computer based numeric assignments in resource models.

All remaining model blocks containing a gold grade estimate were assigned as Inferred mineral resources.

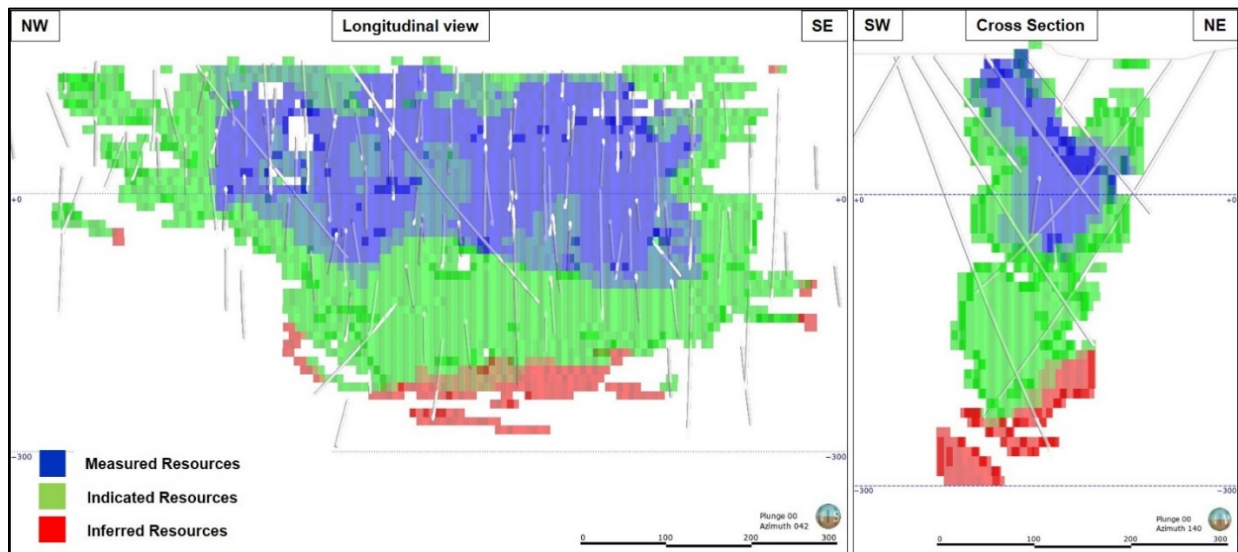


Figure 14-5: Mineral Resource Classification with respect to Supporting Drilling

A test of reasonableness for the expectation of economic extraction was made on the Tocantinzinho mineral resources by developing a series of open pit designs based on optimal operational parameters and gold price assumptions. Those pit designs enveloped most of the measured and indicated mineral resources thus demonstrating the economic reasonableness test for the new estimate and reporting cut-off grade of the Tocantinzinho mineral resources.

14.9 MINERAL RESOURCE SUMMARY

The Tocantinzinho mineral resources estimated by Eldorado as of September 30, 2018 are shown in Table 14-4. The mineral resource is reported at a 0.3 g/t Au cut-off grade.

Table 14-4: Tocantinzinho Mineral Resources as of September 30, 2018

Mineral Resource Category	Resource t x 1,000	Grade Au g/t	Contained Au oz x 1,000
Measured	17,530	1.51	851
Indicated	31,202	1.26	1,264
Measured & Indicated	48,732	1.35	2,115
Inferred	2,395	0.90	69

SECTION • 15 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVES

The mineral reserves at Tocantinzinho have been estimated for open pit mining. Key assumptions and economic considerations applied in the estimation and reporting of mineral reserves are described in this section.

15.2 PIT OPTIMIZATION

15.2.1 Introduction

The open pit optimization was carried out using Minesight® mine planning software. A series of 50 nested unsmoothed pit shells were created using a Lerchs Grossmann algorithm with revenue factors declining from unity to 0.20. The unsmoothed pit shells were then used as a guide for developing a detailed design and starter pit to be used in production scheduling and reserve reporting.

15.2.2 Economic Parameters

15.2.2.1 Metal Price

Mineral reserves were estimated using a gold price of US\$1,200/oz. The transportation and refining cost applied was US\$17.00/oz.

15.2.2.2 Royalty

An NSR royalty of 1.5% and a CFEM royalty of 1.0% was applied to the gold value net of transportation and refining.

15.2.2.3 Metallurgical Recovery

A metallurgical recovery estimate of 88.36% was provided by Hatch Ltd. for both saprolite and hard rock lithologies.

15.2.2.4 Operating Costs

General & administration costs were estimated to be US\$2.82/t and processing costs were estimated to be US\$9.15/t. Base mining costs were estimated to be US\$2.20/t including US\$0.20/t sustaining capital annuity.

Mine operating costs were increased by US\$0.032/t/bench for ore and US\$0.052/t/bench for waste from the entrance bench at 160 m elevation.

15.2.2.5 Cut-off Grade

Based upon the parameters outlined above the cut-off grade applied to recover processing and general & administration costs was calculated to be 0.365 g/t Au.

15.2.3 Block model

15.2.3.1 General

The resource model developed by Eldorado is described in Section 14. The geology and grade block model with surfaces for topography, saprolite/hardrock interface and geology wireframes were imported to a new Minesight® model for mine planning. Block extents and dimensions are summarized in Table 15-1.

Table 15-1: Block Model Extents

Units	Minimum	Maximum	Size	Number
Metres	577,565	578,935	10	137
Metres	9,330,066	9,331,276	10	121
Metres	300	260	10	56

Block model items transferred from the geology model for mine planning included estimated grades for gold, ore percent, bulk density, oxide code as well as resource classification. Additional items were populated in the Minesight® model for rock codes, slope codes for design purposes, recovery, net value and possible scheduling destinations.

15.2.3.2 Resource Classification

Resource Class: The resource model includes measured, indicated and inferred resources. Measured and indicated resources have been used to define the pit limits and for reporting of mineral reserves for scheduling. Inferred resources were not used in the mine plan.

Mining Recovery: No global mining losses were applied to the ore reserves for the following reasons:

- The deposit shows good lateral and vertical continuity at the cutoff grades applied for scheduling. Isolated blocks surrounded on four sides by material below cut-off grade were excluded from reported mineral reserves.
- There is a broad width to the ore zones on individual benches.
- A detailed grade control program will be implemented.

Mining Dilution: Internal dilution was incorporated in the resource model by virtue of the compositing and interpolation method used to obtain the block grades. No additional dilution was applied in pit optimization.

15.2.3.3 Topography

The topographic surface of the Tocantinzinho area was imported into Minesight®. This surface was derived from laser measurements. The area is covered in dense jungle with a canopy that can reach 25 m. Earlier checks on the topographic surface were performed to test the accuracy compared to known surveyed ground points. Points on the surface were compared with surveyed points used in

geophysical (IP) surveys and with the surveyed drillhole collars. Based on the comparison, there did not appear to be a bias in the topographic contours in the area of the open pit. The original topography was translated in plan to correspond with the SIRGAS 2000 grid. Topography in the mine area is shown in Figure 15-1 with block model limits.

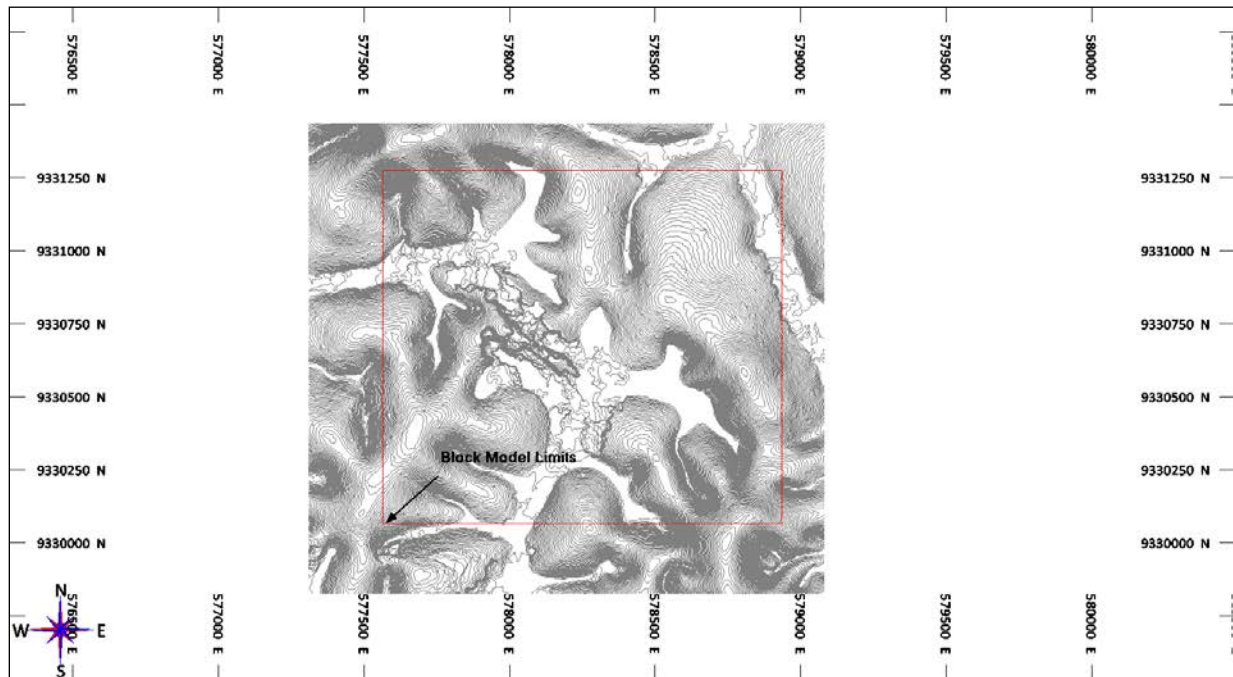


Figure 15-1: Laser Topography and Surface

The gridded surface used for pit optimization is shown in the perspective view of Figure 15-2.

The lithology codes are shown Table 15-2 and Figure 15-3.

Table 15-2: Rock Type and Density used in Block Model

Rock Type	Weathering	Block Model Code	Density
Tailings			1.60
Andesite	Saprolite	11	1.80
	Primary	12	2.74
Granite	Saprolite	31	1.80
	Primary	32	2.62
Quartz Monzonite	Saprolite	41	1.80
	Primary	42	2.68

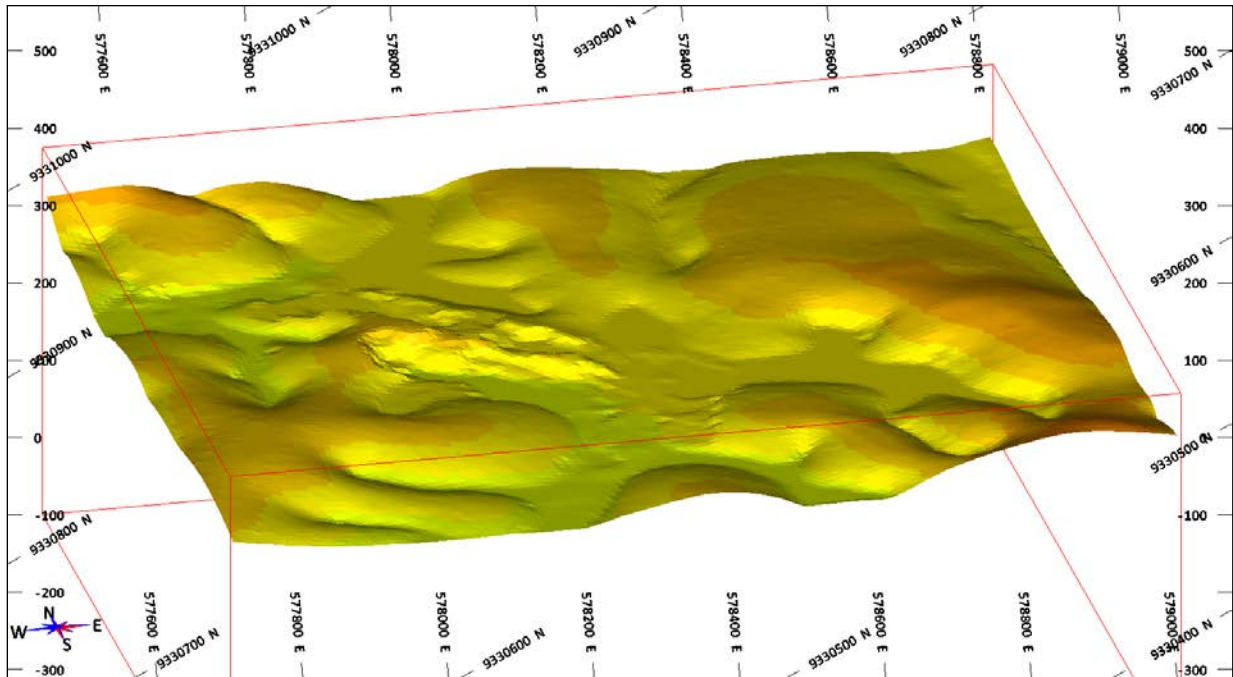


Figure 15-2: Topography Gridded Surface

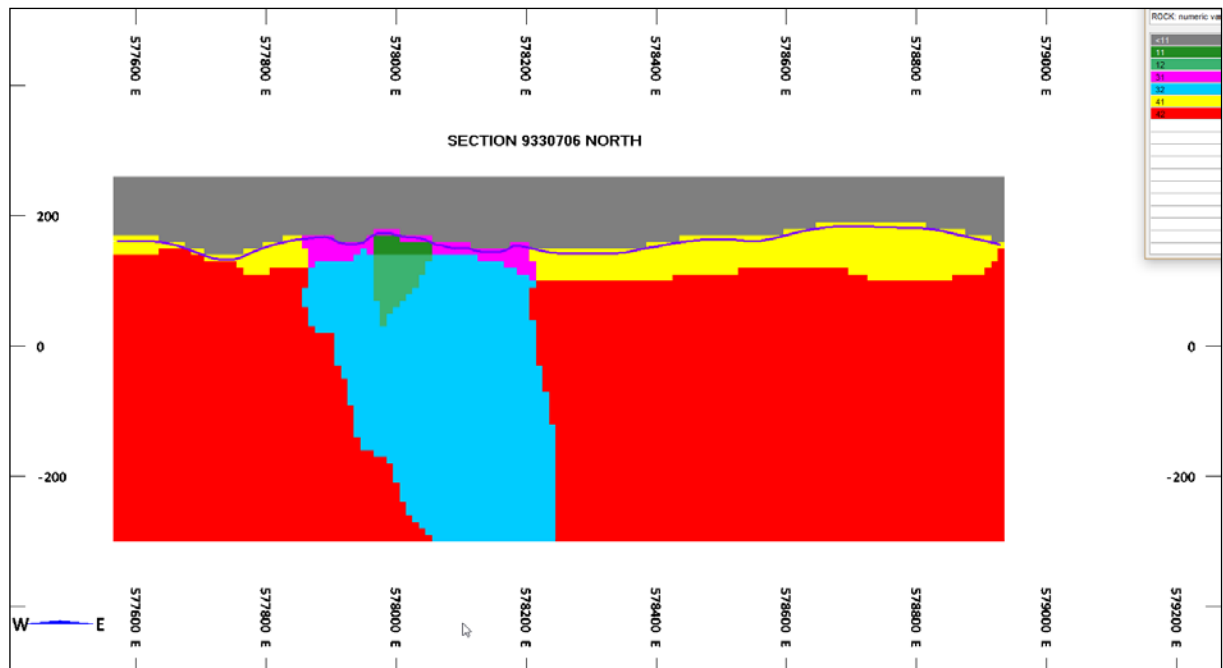


Figure 15-3: Rock Type Section

Gold distribution is shown below Figure 15-4 on the same section and in Figure 15-5 plan view.

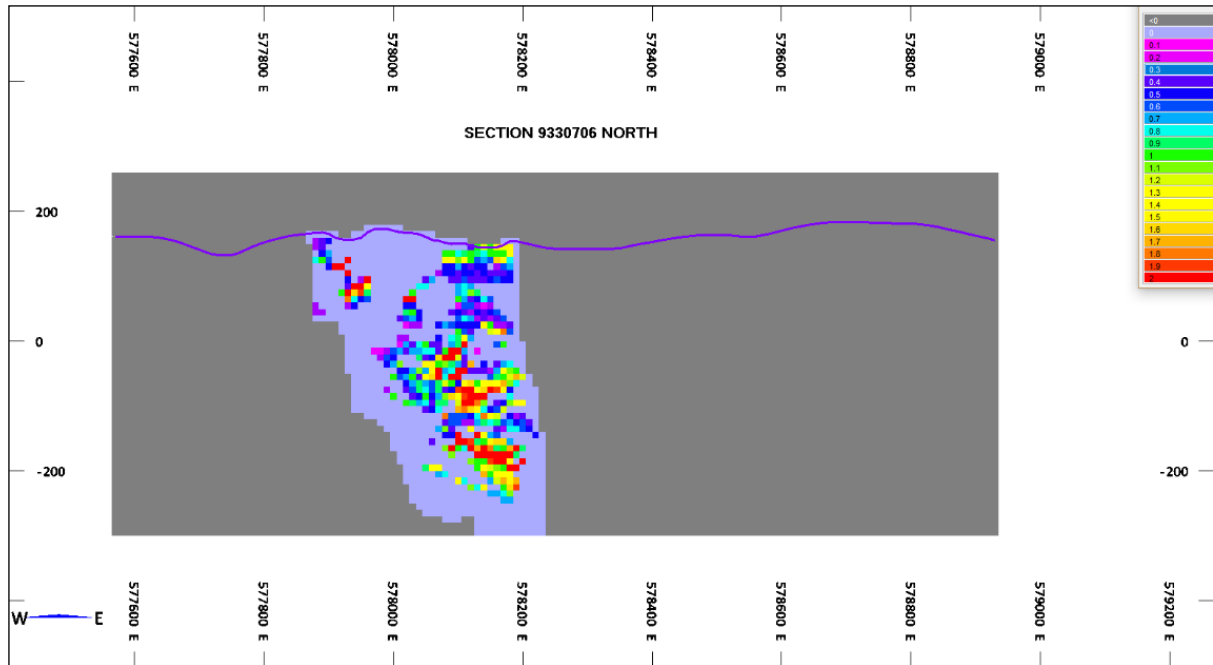


Figure 15-4: Section 9330706 - Gold Distribution

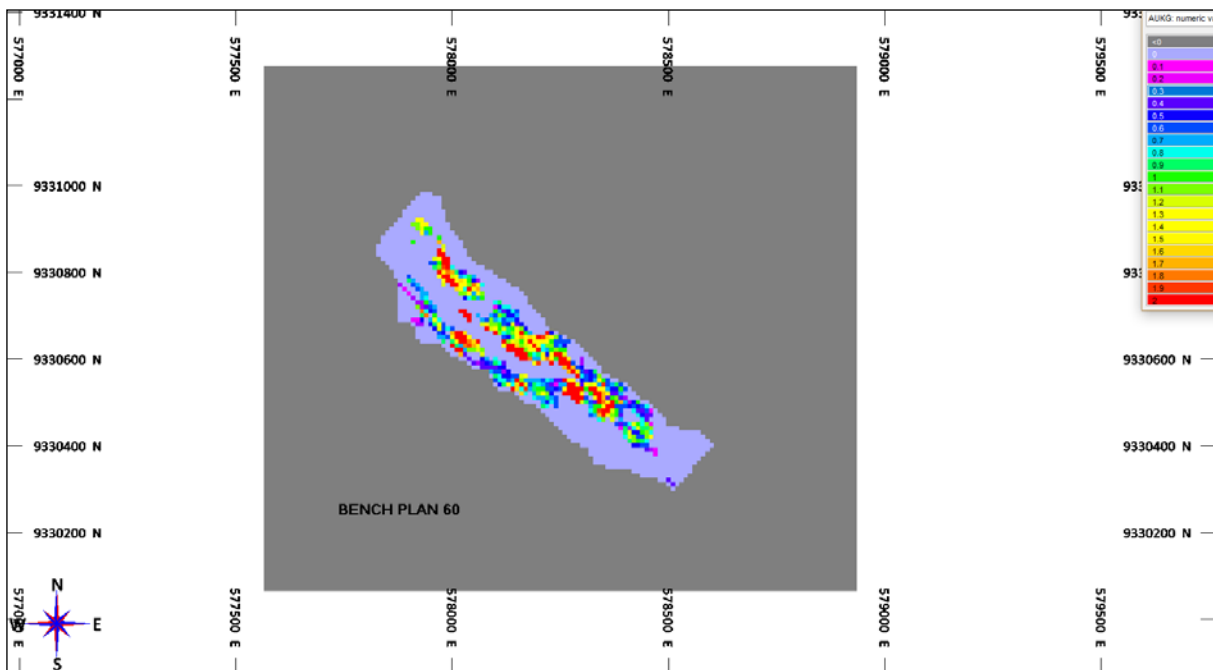


Figure 15-5: Bench Plan 60 - Gold Distribution

15.2.4 Wall Slope Design Parameters

In 2010, Golder Associates carried out a study of geomechanical modeling and slope design considering the estimated strength values of the materials that will make up the slopes of the final pit. Following this preliminary work additional laboratory tests were carried out to determine the actual strength parameters of the rock mass. After testwork completion in 2012 Golder undertook a review of slope design parameters, taking the lab results into consideration. Additional geotechnical reviews were provided by HTA Geotechnical Consulting and internal Eldorado geotechnical engineers. The current design criteria applied for pit optimization and design is summarized in the Table 15-3 below and the design sectors after Golder are shown in Figure 15-6. Sectors 1 through 6 are in rock and Sector 7 is saprolite above the weathered rock interface.

Table 15-3: Slope Design Criteria Applied for Pit Optimization

	Units	Sector 1	Sector 2	Sector 3	Sector 4	Sector 5	Sector 6	Sector 7
Bench Height	m	10.00	10.00	10.00	10.00	10.00	10.00	10.00
Berm Width	m	8.00	10.10	10.10	6.60	10.10	10.10	13.71
Bench Berm Spacing	#	2.00	2.00	2.00	2.00	2.00	2.00	1.00
Batter Angle	degrees	70.00	70.00	70.00	70.00	70.00	70.00	70.00
Wall Height	m	270.00	220.00	290.00	290.00	310.00	315.00	30.00
Inter Ramp Wall Angle	degrees	52.62	49.01	49.01	55.24	49.01	49.01	29.96
Road Width	m	30.00	30.00	30.00	36.80	30.00	36.00	30.00
Passes through Wall	#	1	1	2	2	2	3	0
Overall Wall Angle	degrees	48.81	44.85	42.91	46.54	43.26	39.53	29.96
Wall Flattening Angle	degrees	3.81	4.16	6.10	8.70	5.75	9.48	0.00
MSEP Input	degrees	48.80	44.90	42.90	46.50	43.00	39.50	30.00

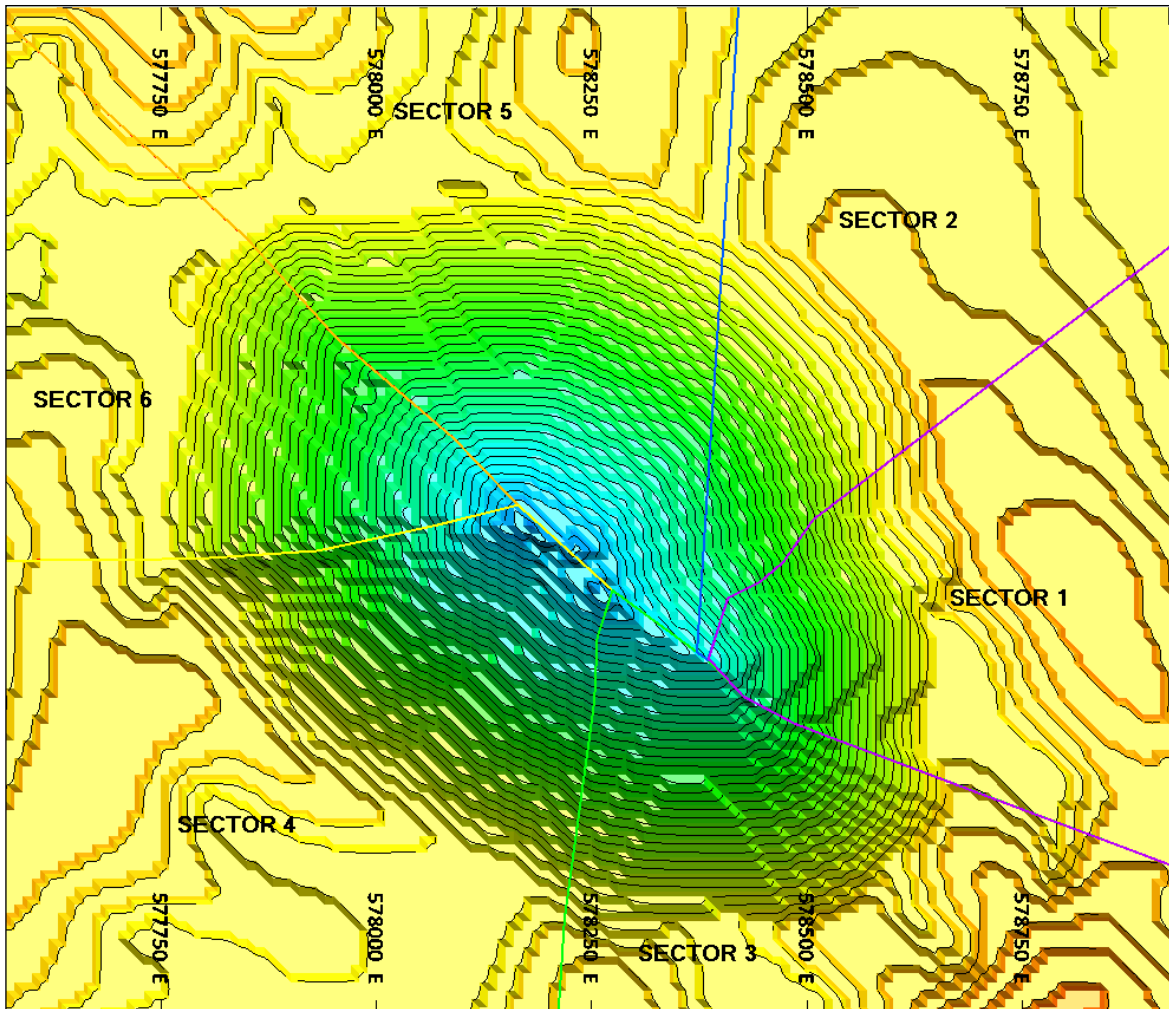


Figure 15-6: Slope Design Sectors

15.2.5 Pit optimization

Unsmoothed pit limits were developed using a Minesight® variable slope Lerchs Grossmann algorithm. The preliminary net minegate revenue and operating costs were used to estimate the value of each regular block in the model. A series of 50 nested pit limits were defined using revenue factors between 0.20 and 1.00. The nested pit limits used to develop the pit design are shown in Figure 15-7 and Figure 15-8. Mineral resources for the shells are summarized at 0.365 g/t Au cut-off grade in Table 15-4.

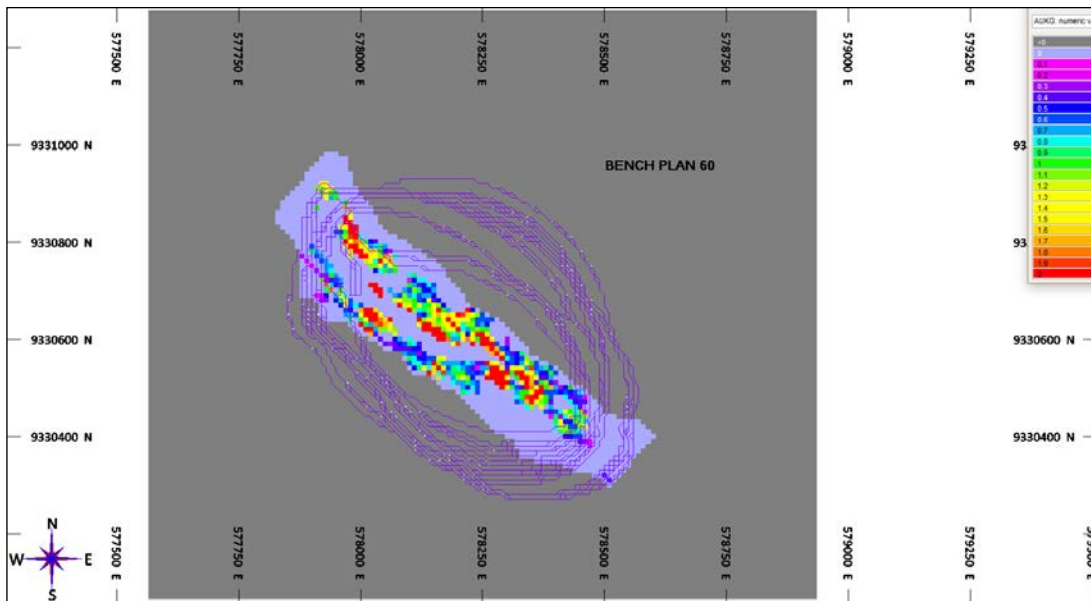


Figure 15-7: Bench Plan 60 Lerchs Grossmann Pit Limits

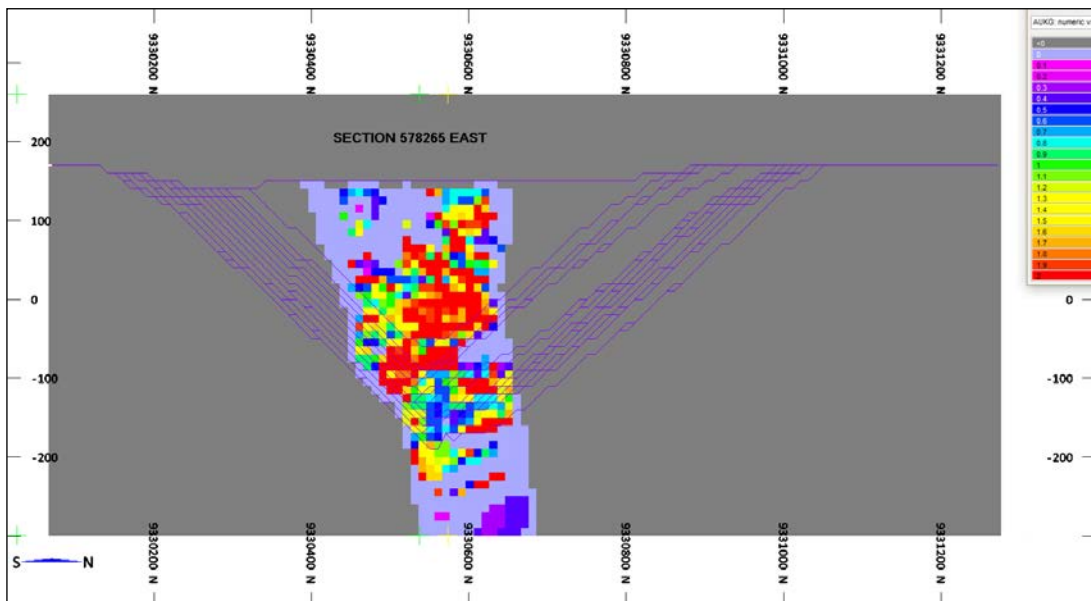


Figure 15-8: Section 578265 East Lerchs Grossmann Pit Limits

Table 15-4: Lerchs Grossmann Pit Shell Summary

Shell	Ore	Mine	Total	S/R	AUKG	NSR	Contained
	kBCMS	kt	kt				oz x 1,000
1	8,190	20,645	35,058	1.70	1.48	48.58	984.1
2	9,115	23,069	41,965	1.82	1.48	48.64	1,101.0
3	9,223	23,352	42,984	1.84	1.49	48.74	1,116.8
4	9,507	24,096	45,151	1.87	1.49	48.66	1,150.7
5	9,710	24,628	46,895	1.90	1.48	48.61	1,174.6
6	9,774	24,795	47,372	1.91	1.48	48.58	1,181.9
7	9,925	25,191	49,639	1.97	1.49	48.88	1,208.2
8	9,956	25,272	49,888	1.97	1.49	48.85	1,211.4
9	9,980	25,335	50,107	1.98	1.49	48.83	1,213.9
10	11,443	29,168	63,076	2.16	1.45	47.65	1,363.9
11	11,443	29,168	63,076	2.16	1.45	47.65	1,363.9
12	11,461	29,215	63,360	2.17	1.45	47.67	1,366.5
13	11,658	29,731	66,371	2.23	1.46	47.75	1,393.2
14	12,589	32,171	78,621	2.44	1.46	47.71	1,506.2
15	12,604	32,210	78,817	2.45	1.46	47.71	1,507.9
16	12,702	32,467	80,224	2.47	1.46	47.72	1,520.2
17	12,705	32,475	80,257	2.47	1.46	47.72	1,520.5
18	12,728	32,535	80,546	2.48	1.46	47.70	1,523.0
19	12,948	33,111	84,437	2.55	1.46	47.79	1,552.7
20	13,143	33,622	87,166	2.59	1.46	47.71	1,574.1
21	13,430	34,374	91,453	2.66	1.45	47.62	1,606.3
22	13,546	34,678	92,796	2.68	1.45	47.52	1,617.0
23	13,694	35,066	94,924	2.71	1.45	47.45	1,632.7
24	13,738	35,181	95,438	2.71	1.45	47.41	1,636.8
25	13,767	35,257	96,048	2.72	1.45	47.42	1,640.7
26	14,014	35,904	99,868	2.78	1.44	47.31	1,666.8
27	14,026	35,936	100,123	2.79	1.44	47.31	1,668.4
28	14,029	35,943	100,165	2.79	1.44	47.31	1,668.7
29	14,046	35,988	100,601	2.80	1.44	47.32	1,671.2
30	14,135	36,221	101,971	2.82	1.44	47.26	1,679.8
31	14,211	36,420	103,344	2.84	1.44	47.23	1,688.0
32	14,220	36,444	103,505	2.84	1.44	47.23	1,689.0
33	14,933	38,312	117,749	3.07	1.44	47.05	1,768.9
34	14,937	38,322	117,823	3.07	1.44	47.05	1,769.3
35	14,985	38,448	118,543	3.08	1.43	47.01	1,773.7
36	15,100	38,749	121,195	3.13	1.43	47.00	1,787.1
37	15,100	38,749	121,195	3.13	1.43	47.00	1,787.1

Shell	Ore	Mine	Total	S/R	AUKG	NSR	Contained
	kBCMS	kt	kt				oz x 1,000
38	15,100	38,749	121,195	3.13	1.43	47.00	1,787.1
39	15,105	38,763	121,279	3.13	1.43	47.00	1,787.6
40	15,205	39,025	123,178	3.16	1.43	46.95	1,797.8
41	15,205	39,025	123,178	3.16	1.43	46.95	1,797.8
42	15,244	39,127	123,530	3.16	1.43	46.89	1,800.4
43	15,346	39,394	125,653	3.19	1.43	46.85	1,810.9
44	15,395	39,522	126,702	3.21	1.43	46.82	1,815.9
45	15,401	39,538	126,871	3.21	1.43	46.82	1,816.6
46	16,118	41,417	142,714	3.45	1.42	46.52	1,890.6
47	16,127	41,440	142,853	3.45	1.42	46.51	1,891.4
48	16,192	41,610	144,370	3.47	1.42	46.48	1,897.8
49	16,193	41,613	144,374	3.47	1.42	46.48	1,897.9
50	16,213	41,665	144,827	3.48	1.42	46.47	1,899.8

15.2.6 Pit Design

The final pit design is shown in Figure 15-9. The bottom bench is at -170 m elevation and the ramp exit to the primary crusher on the north side of the pit is at 150 m elevation. The pit will be approximately 980 m across in the north-south direction and 960 m across in the east-west direction. Exits have been provided on the west side to the waste and slow grade stockpiles and to the south to access tailings impoundment and aggregate crusher location.

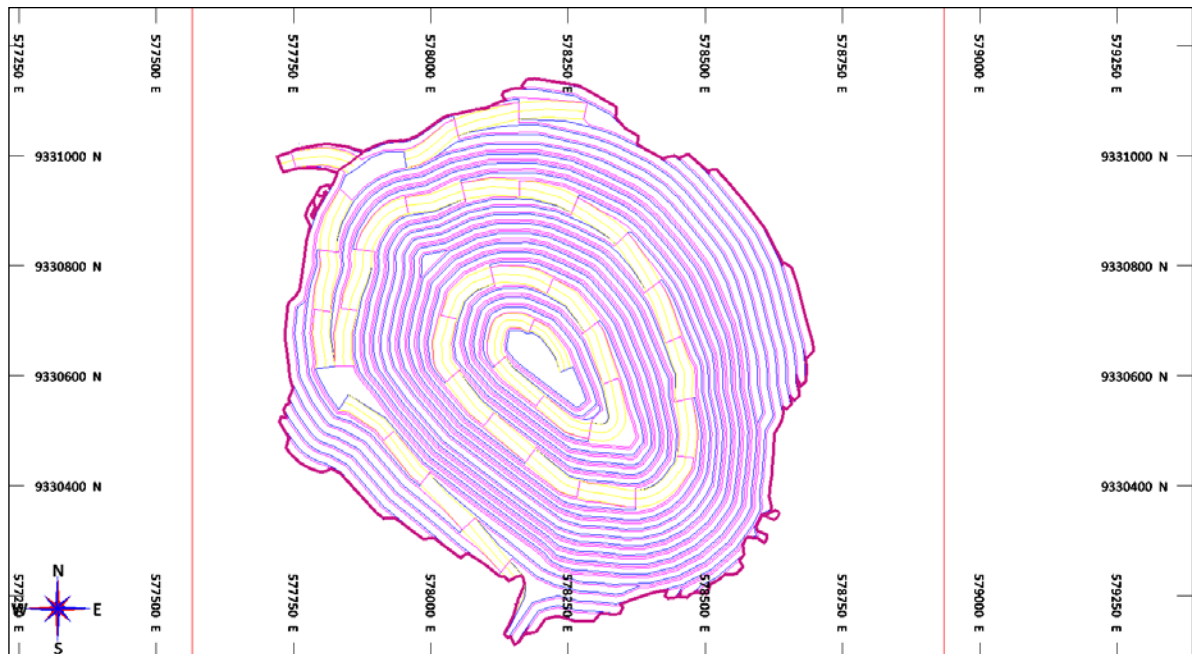


Figure 15-9: Pit Design

15.2.7 Mineral Reserve Statement

Mineral reserves were calculated in accordance with CIM standards using the mineral resource block model within an engineered pit design. The pit design was based on an optimized pit shell using a US\$1,200/oz gold price. Blocks above a 0.365 g/t Au cut-off grade are considered mineral reserves. Those mineral resource blocks with a measured resource class were converted into proven reserves, while the indicated resource blocks were converted into probable reserves. Mineral resource blocks classed as inferred were treated as waste. No additional modifying factors were used in the reserve estimate. Table 15-5 presents the mineral reserve estimate for the Tocantinzinho Project as of March 31, 2019.

Table 15-5: Tocantinzinho Mineral Reserve Estimate as of March 31, 2019

Category	Tonnes kt	Gold Grade g/t	Gold Contained oz x 1,000
Proven	17,007	1.52	834
Probable	21,898	1.35	949
Proven and Probable	38,906	1.42	1,779

SECTION • 16 MINING METHODS

16.1 INTRODUCTION

The Tocantinzinho mine will be developed as a conventional open pit using excavators and trucks with wheel loader support. Near surface saprolite material and old tailings will be mined using a fleet of articulated 40 t capacity trucks and 6 m³ excavators in pioneering areas. Some saprolite and fresh granite, quartz monzonite and andesite will be drilled, blasted and loaded by 17 m³ excavators and an 11.6 m³ capacity wheel loader into 90 t rigid frame off road haulage trucks.

The mine will be developed in two phases. Pre-stripping will be undertaken in Phase 1 over a two year period. Road and drainage development will be undertaken initially as well as stripping of a borrow pit which will provide fresh rock for construction and site preparation. Stockpile and waste dump preparation will take place west of the open pit area. A total of 22.7 Mt will be excavated in the mine during the pre-production period. The mine will deliver 4.37 Mt of ore annually to the processing facility. Peak production years will be Year 1 and Year 2 with a total mining rate of 26.7 Mt. The open pit will operate to Year 9 of the plan. Stockpile recovery and processing will continue into Year 10.

16.2 GEOTECHNICAL AND HYDROGEOLOGICAL ANALYSIS

The geotechnical and hydrogeological section below has been duplicated from the Technical Report for the Tocantinzinho Gold Project Brazil, May 2011.

16.2.1 Golder-Geotechnical Studies and Pit Slope Design

A program of geotechnical data collection and analysis was conducted to form the basis for slope design used in the open pit. This program was carried out by Golder and consisted of drilling and logging six oriented core holes and an additional logging of 11 exploration holes which were non-oriented. Golder also conducted kinematic and limit equilibrium analysis and prepared recommendations for slope designs.

16.2.1.1 Data collection by Golder

Six diamond core holes were drilled to test the rock mass in the planned pit wall, the locations of the holes are shown in Figure 16-1. The holes were oriented using the ACT REFLEX device and the punch mark method. The holes were then logged for lithology, fracture orientation, fracture roughness, RQD and rock mass classification. An additional 11 exploration holes were logged for lithology, RQD and rock mass classification.

A selected summary of the geotechnical data that was collected is presented in the sections that follow.

16.2.1.2 Fracture orientation

Fracture orientations were measured by Golder from the oriented drill cores. The strike and dip of all fractures were recorded and then summarized in stereo net plots. Figure 16-2 shows all fractures measured in the six oriented geotechnical drillholes.

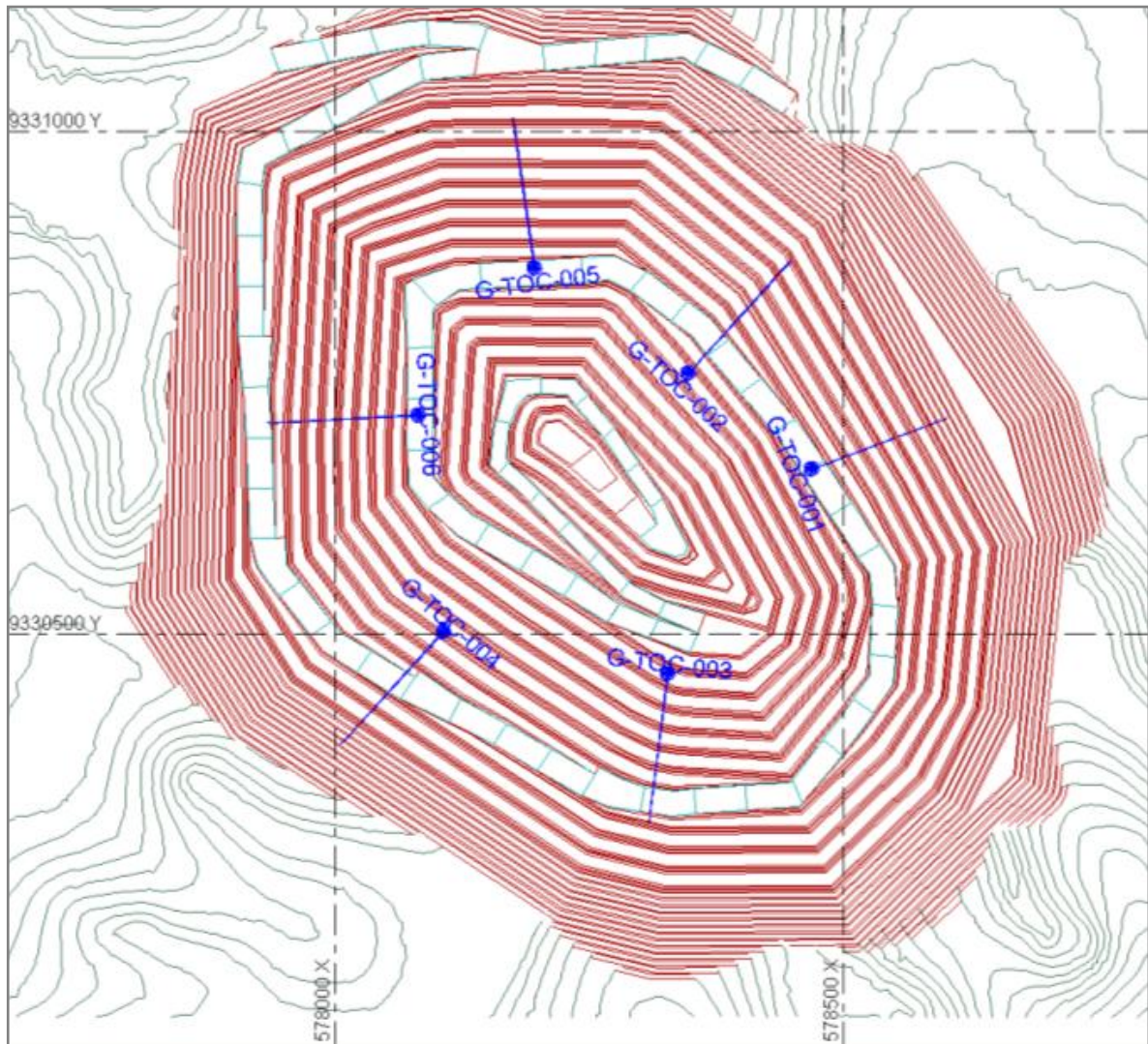


Figure 16-1: Locations of Oriented Geotechnical Drill Holes

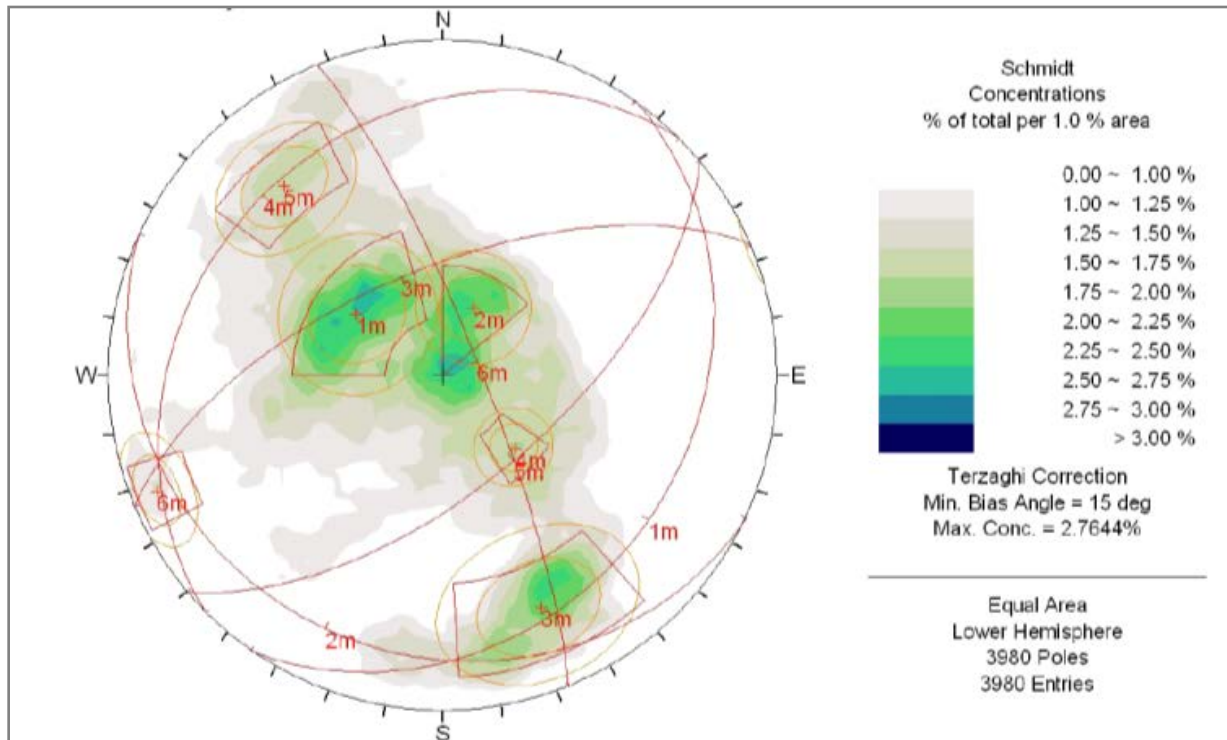


Figure 16-2: Fracture Orientations Measured in all Geotechnical Drill Holes (3980 poles)

16.2.1.3 Classification logs

In total 17 core holes were logged and intervals were assigned a geo-mechanical class designation of either Class II/I, Class III, Class IV and Class V. Generally, Class V corresponds with saprolite material, while Class II/I is fresh un-weathered rock. Classes III and IV represent a rock mass which is either slightly weathered or highly fractured. A series of geomechanical sections were created from the classification, an example section spanning G-TOC-004 and G-TOC-002 is shown in Figure 16-3.

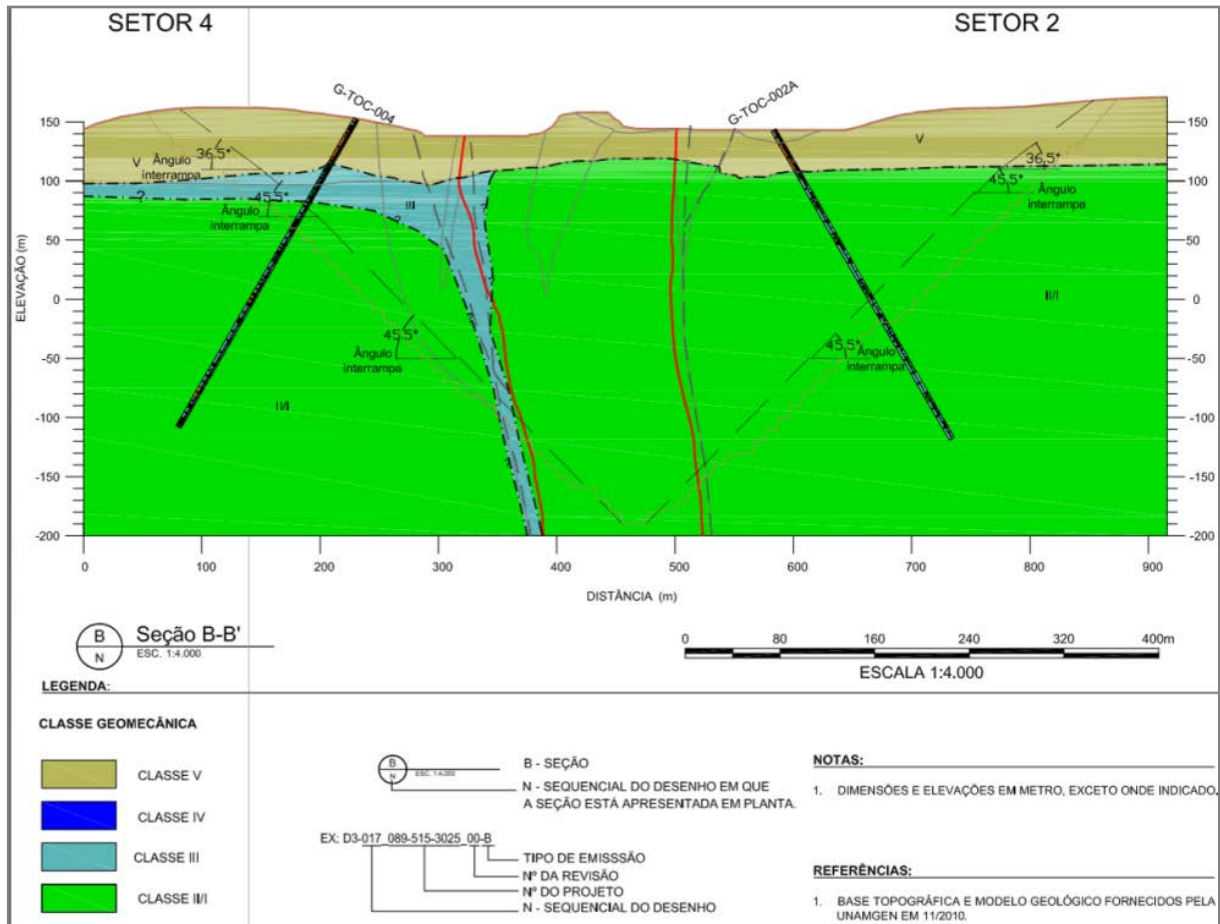


Figure 16-3: Geomechanical Model of Tocantinzinho

16.2.1.4 Data analysis by Golder

Golder also analyzed the geotechnical data. Two basic analyses were performed and these were kinematic failure analysis and overall slope failure analysis.

Kinematic analysis

Kinematic failure analysis was used to determine the likely mode of failure on a given bench orientation. An example of the analysis is shown in Figure 16-4 and Figure 16-5. For each geotechnical hole and corresponding stereonet, Golder took the orientation of the pit wall and conducted a graphical analysis to determine which fracture planes are able to slip in a planar mode of failure and which fracture plane combinations are able to slip in a wedge plane type of failure. This analysis was used in recommending the slope design.

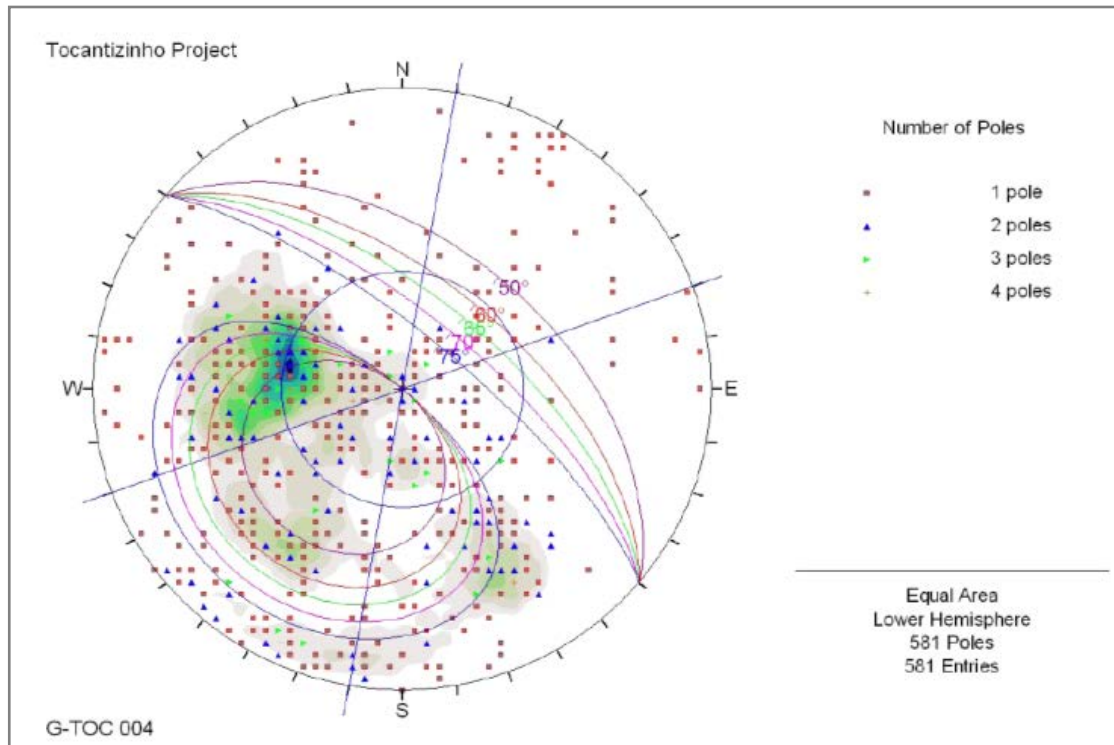


Figure 16-4: Example of Kinematic Failure Analysis (G-TOC 004 – Planar Failure Analysis)

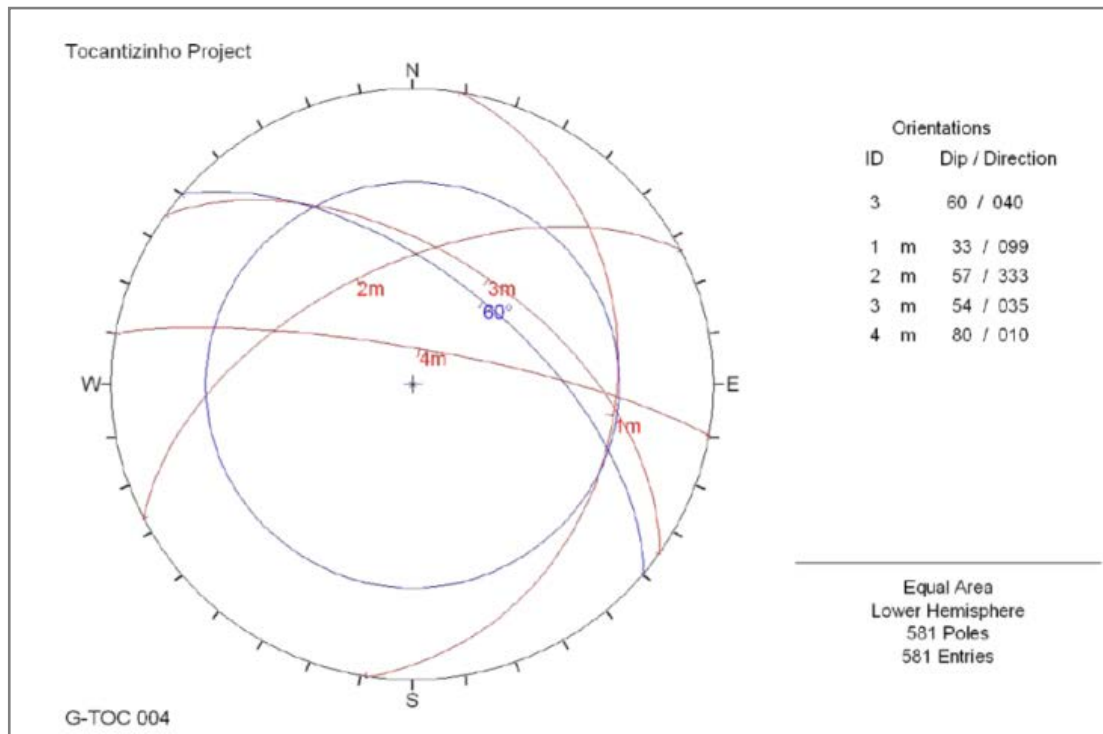


Figure 16-5: Example of Kinematic Failure Analysis (G-TOC 004 – Wedge Failure Analysis)

Slope failure analysis

An analysis of overall slope failure using an industry standard software product based on a limit equilibrium numerical method was conducted. The input parameters were estimated from Golder's experience with other projects. A recommendation was made to carry out additional strength testing including uniaxial compressive tests and direct shear tests to get site specific information. The input parameters used in the overall slope failure analysis are shown in Table 16-1.

Table 16-1: Input Parameters for Slope Failure Analysis

Type	Rock Class	Cohesion KPa	Friction Angle °	Unit weight kN/m ³
Granite	V/IV	25	32	18
	III	300	38	24
	II/I	700	42	27

The results of the overall slope failure analysis are shown in Figure 16-6 and Figure 16-7. The results indicate that a 46 degree overall slope with a 300.0 m height will have a factor of safety of 1.33 while a 42 degree slope of the same height will have a factor of safety of 1.41.

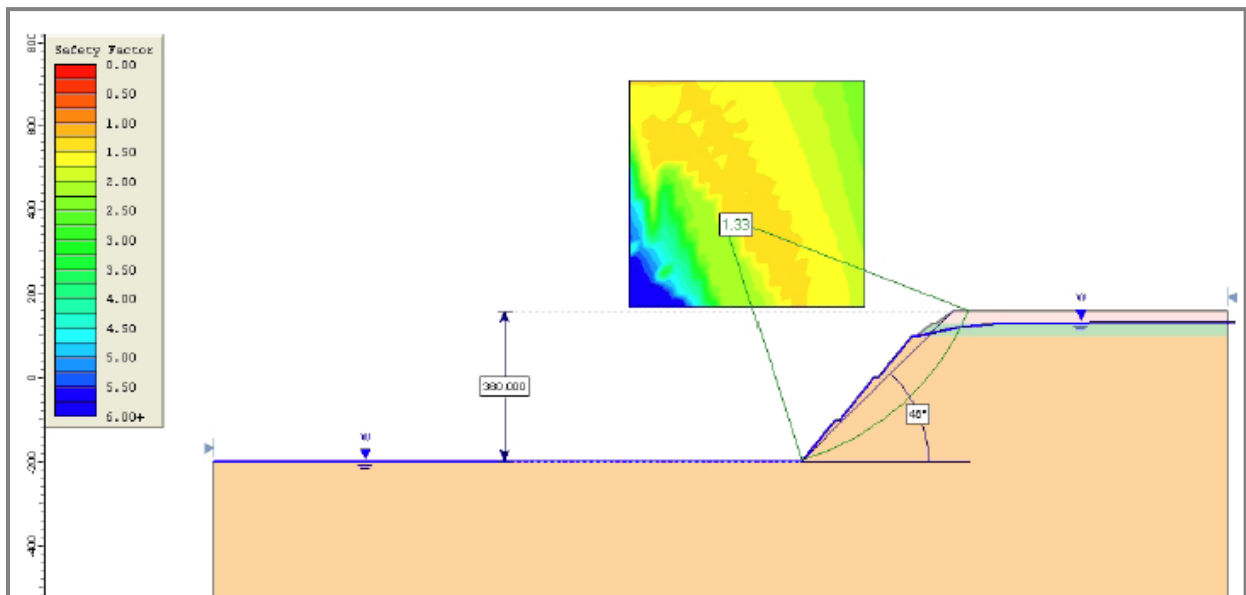


Figure 16-6: Results of Overall Slope Failure Analysis (FS=1.41)

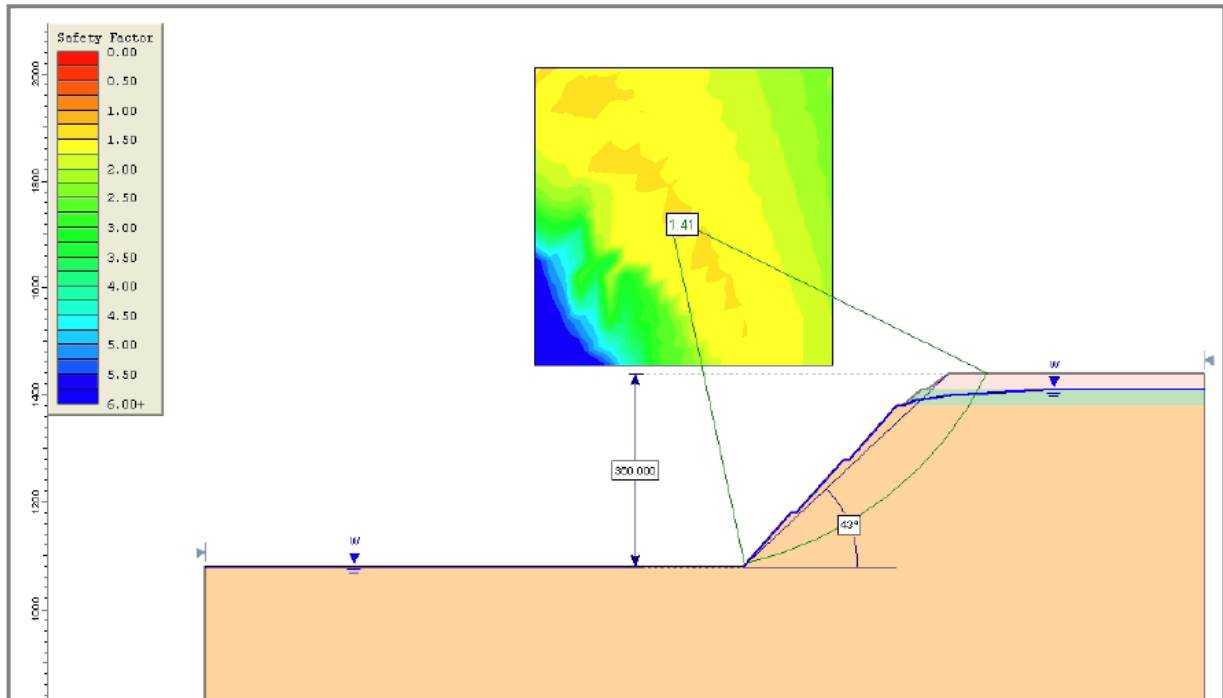


Figure 16-7: Results of Overall Slope Failure Analysis (FS=1.33)

Golder conclusions and slope design recommendations

Golder concluded that overall slope failure would not be the most critical type of failure in the open pit. They also concluded that 6 geotechnical sectors were relevant with wedge failures being most likely in sector 4 and with planar failures being the most likely in sectors 1 and 3.

Recommendations for slope design which are shown in Table 16-2, included 10 slope designs varying between 4 sectors and 3 rock mass classes.

Table 16-2: Golder Recommendations for Slope Design

Sector	Rock Mass *	Average Thickness m	Bench Geometry			IRA foot x foot	Overall Angle (Considering the inter Ramp Berms)
			Bench Face Angle	Height m	Berm Width m		
1	Class V	30	55°	10	6.5	36.5°	44.5°
	Class III with some class II	30	60°	10	6.2	40.0°	
	Class II/I	300	65°	20	8.0	49.0° **	
2-5	Class V	30	55°	10	6.5	36.5°	43.0°
	Class II/I	300	60°	20	8.1	45.5° **	
3-4	Class V	30	55°	10	6.5	36.5°	42.0°
	Class III with some class II	30	60°	10	6.2	40.0°	
	Class II/I	300	60°	20	8.1	45.5° **	
6	Class V	30	55°	10	6.5	36.5°	46.0°
	Class II/I	300	65°	20	8.0	49.0° **	

*10 m berm between each class of rock mass;

**Maximum IRA height of 140 m (7 benches) with a 10.5 m berm every 7 benches.

16.2.2 Hydrogeological Analysis

This section describes the hydrogeologic conditions of the Tocantinzinho Project area. Most of the information used in this section was based on the VOGBR Report titled “Preliminary Report on Hydrogeological Pre-Feasibility Studies for the Tocantinzinho Project – Pit” (2010).

16.2.2.1 Geologic conditions

The principal geologic units in the project area consist of intrusive igneous bedrock composed of granite and quartz monzonite, altered granite, andesite, and rhyolite associated with the Tocantinzinho shear zone, and overlying saprolite and residual soils derived from deep weathering of the bedrock. Locally, the Tocantinzinho shear zone is within two extensive sub-vertical faults that strike N-NW. In the project area, the shear zone is approximately 500 to 700 m in width.

The intrusive igneous bedrock complex in the Tocantinzinho Project area is extensive and shows no foliation or cleavage features. Below approximately 70 m depth, the rock mass is fresh, very competent, and exhibits few fractures. Within the mineralized zone of the shear zone, however, the rock mass is more altered and fractured.

The residual soil and saprolite layer varies in thickness from 10 to 35 m. The texture of the saprolite varies, depending on the original parent rock. Saprolites derived from granitic rocks typically have a sandy-clayey texture.

Below the residual soil/saprolitic layer is a horizon of highly-fractured fresh bedrock, with thickness ranging from 20 to 70 m. The average thickness of this horizon is approximately 50 m.

Groundwater recharge in the project area is not well known but was estimated to range from 220 to 660 mm per year, based on 10 to 30 percent of average annual rainfall. However, based on our past experience of working on projects in similar conditions (i.e. sub-tropical climates with low permeability saprolitic soil profiles), the average annual recharge rate is more likely to be in the range of 5 to 10 percent of annual rainfall, which results in a range of approximately 100 to 200 mm per year.

16.2.2.2 Hydrogeologic units

VOGBR identified three principal hydrogeologic units based on weathering profile and degree of fracturing. The three principal hydrogeologic units include:

Residual Soil/Saprolite (SR)

The residual soil and saprolite zone (SR) is generally uniformly distributed across the project site, but exhibits local variation in thickness, ranging from 8 to 80 m, with an average thickness of 35 m; the thicker sequences of this zone are found in topographically higher areas. Saprolite is formed from the deep weathering and oxidation of the underlying bedrock and is comprised primarily of fine-grained soil particles, such as a silty-clay. Residual soils and saprolite derived from granitic bedrock are typically coarser with a textural composition of a sandy-clay.

Highly-Fractured Rock Mass (RMF)

A horizon of highly-fractured, slightly altered bedrock underlying the saprolite is prevalent throughout the project area. This horizon ranges in thickness from 20 m to over 70 m, with an average thickness of 50 m. The primary conduit of fluid flow is through fissures and fractures in the rock. No hydraulic conductivity or permeability testing data were available to review for this report, but the hydraulic conductivity was qualitatively estimated to be “moderate”.

Fresh Rock Mass (RS)

Fresh bedrock occurs below 70 m depth on average across the Tocantinzinho project area. This unit is described as very competent rock with few fractures and faulting observed in the wall rock and is anticipated to have very low hydraulic conductivity. However, fractures and faulting are observed in the mineralized zone within the shear zone; therefore, the hydraulic conductivity in the mineralized zone is expected to be greater. Based on these observations, VOGBR separated the RS into two units based on degree of fracturing:

- **Fractured Fresh Rock Mass (RSF)** - The fractured fresh rock mass occurs within the mineralized zone of the shear zone and is more permeable than the surrounding, slightly fractured bedrock. The orientation of the fracturing is generally in the same direction as the shear zone (NW-SE) and is sub-vertical. The depth of this unit is not known, but fracturing and faulting were observed at depths of up to 300 m. Evidence of water from staining and alteration were observed in fractures during exploratory drilling.
- **Slightly Fractured Fresh Rock Mass (RSPF)** - This unit is primarily composed of quartz-monzonite and granite bedrock and occurs on the NE and SW sides of the shear zone. This

unit is described as having fewer fractures than the RSF unit and is considered very competent and of low permeability.

16.2.2.3 Groundwater occurrence

Springs

VOGBR identified 42 spring and seep locations during their field investigation in September 2010. Figure 16-8 is a summary of the springs and seeps in the project area, including lake water level elevations and water course elevations. Many of the springs were described as “diffuse” and were observed in mostly sandy-clayey (areno-argillaceous) soil types; higher seepage rates were typically observed from coarser-grained soil types. Measurable seepage from these springs ranged from 0.04 to 0.91 litres/second (L/s). The seepage from these springs is considered to be representative of baseflow, because they were measured at the end of the dry season.

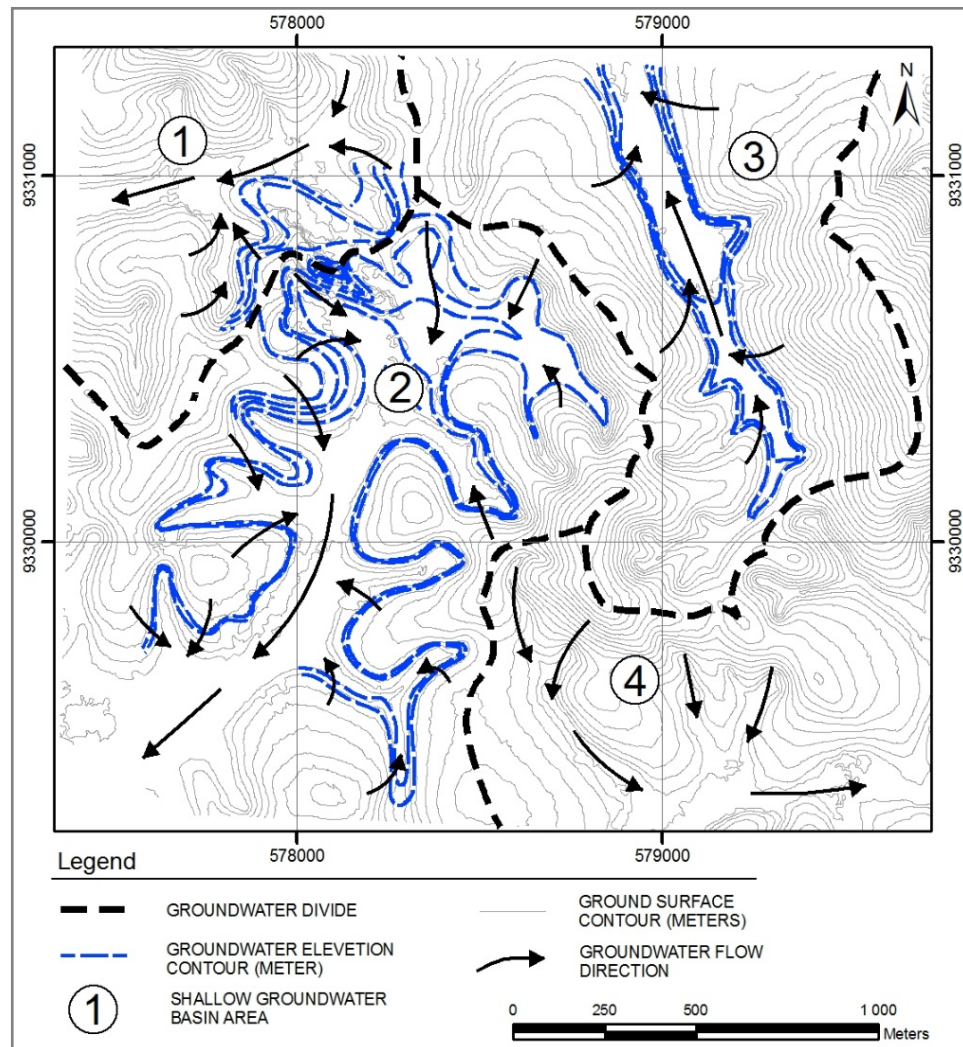


Figure 16-8: Groundwater Contour Map

Residual soils/saprolite (SR)

The VOGBR report, groundwater occurrence in the SR is primarily characterized by unconfined conditions and typically mimics surface topography. The occurrence of groundwater in the project area is based on depth to water measurements made in 37 exploratory drillholes. In the local highlands, groundwater was typically observed near the contact between the saprolite and the underlying bedrock. Depth to groundwater observed in the exploratory holes ranges from flowing artesian to over 18 m below collar elevation. Groundwater elevations ranged from 136 to 149 m. Flowing artesian conditions were observed in TOC-115 and TOC-145A, which are both deep boreholes located in topographic low areas.

Figure 16-8 also presents a groundwater contour map in the Tocantinzinho Project area in the vicinity of the proposed open pit location. The groundwater system is recharged via the infiltration of precipitation in topographically higher areas with discharge in the topographically lower areas (i.e. stream drainages and lakes). In general, localized groundwater flow is from hill tops to stream drainages or lakes. Groundwater flow in the SR unit is dissected into about four different areas due to the undulating topography. The general groundwater flow directions are summarized below:

- Area 1 (Northwest area) – groundwater flow is ultimately to the west;
- Area 2 (Central area) – groundwater flow is ultimately to the south-southwest, following the main stream channel;
- Area 3 (East-Northeast area) – groundwater flow is ultimately to the north-northwest; and
- Area 4 (Southeast area) – groundwater flow is ultimately to the east-southeast.

Highly-fractured rock mass (RMF)

Groundwater occurrence in the highly-fractured rock mass is not well understood, but it is assumed to be fully saturated, given the overlying saturated conditions in the saprolite. The primary conduit for groundwater flow in the RMF is likely through the high density fractures with minor contribution of water from the rock matrix. The direction of groundwater flow may follow the same pathways as the overlying SR.

Fresh rock mass (RS)

Groundwater occurrence in the RS unit is not well understood but based on the results of the drilling exploration program, groundwater will primarily occur in the fractures of the RSF unit. The RSPF unit is much more competent and less fractured than the RSF unit, and therefore may act as a barrier to groundwater flow. The groundwater occurrence in the RSF unit is under confined conditions and the general groundwater flow is likely horizontal due to the relatively low regional relief of the area. The direction of groundwater flow is presumed to be primarily in a NW-SE direction, parallel to the strike of the Tocantinzinho shear zone. Current available data show no evidence of a hydraulic connection between the deep groundwater in the pit area and the Tocantinzinho River. However, the presence of faults trending NW-SE have the potential to provide hydraulic communication between the open pit and the river and influence the quantities of groundwater inflow.

Open pit inflow

Mine inflows will be from groundwater, surface runoff, and precipitation. For the groundwater inflow component, the primary sources include the saprolite (SR), the highly-fractured rock mass horizon (RMF), and the fractured fresh rock mass within the mineralized zone (RSF). For dewatering purposes, VOGBR suggested that as the pit is excavated in the first phases of mining, the SR and RMF units will need to be dewatered and depressurized.

Dewatering or depressurization the SR using wells will probably not be effective, because of the low permeability and high storage capacity characteristics of this unit. If dewatering and depressurization of the ST unit is required for geotechnical stability purposes, horizontal drains would be required. Water from the drains would be collected on the pit benches and pumped from the pit.

Depending on the results of the planned hydraulic testing by VOGBR, the RMF and RSF units may be depressurized using dewatering wells located at the northwest and southeast ends of the pit. The RMF will likely have the best potential for dewatering from groundwater extraction wells, due to the degree of fracturing and absence of fine-grained materials.

During the later stages of mining, more of the fresh rock mass (RS) will be exposed in the pit walls. Groundwater contribution to the pit from the slightly fractured wall rock (RSPF) is expected to be minimal and can be controlled by sump pumps installed in the bottom of the pit. Groundwater inflow from the fracture rock mass (RSF) may be more of a concern, depending on the permeability of this unit and its hydraulic connection to the Tocantinzinho River.

Mine inflows from precipitation and surface water run-off from the pit slopes will occur primarily during the rainy season (i.e. November to June), with the greatest amount of inflow occurring during February and March (on average). In the dry season (i.e. July to October), evaporation exceeds monthly rainfall on average thus inflow from rainfall and surface run-off will be minimal.

The water balance was prepared for the pit to estimate the quantities of precipitation, surface water and groundwater inflow to the open pit based on the hydrogeological and hydrological investigations. The dewatering system was designed based on these estimates on an annualized basis.

Post closure conditions

The post-closure condition of the pit is expected to be a pit lake with discharge via groundwater and or surface water to the tributaries of the Tocantinzinho River. Upon completion of mining and cessation of dewatering, the pit will fill with water from groundwater, surface runoff, and precipitation. The amount of time it will take for the pit lake to reach steady-state cannot be estimated from the available data, but will depend on the ultimate pit depth and volume of excavation as well as the estimated total annual average combined inflow.

16.3 MINING

This section provides a description of the mine design, development plan, mining fleet requirements and operating costs for the Tocantinzinho Project. The mine was designed as a single open pit operation using two pit phases mined over 11 years, including two years of development and pre-

stripping, with a peak mining rate of 26.7 Mt of material per year. The open pit is designed to a depth of 350 m using variable wall slopes. The mine will operate on 10 m benches using truck and excavator fleets. The average cost of mining a tonne of material is estimated to be US\$2.47/t based operating with three crews of operators and quotes for major equipment maintenance components, diesel, explosives and tires.

16.3.1 Open Pit Design

The mine design takes into consideration the proposed 10 m bench height and the size of the equipment in the fleet. The mine will operate two fleets of equipment. Saprolite pioneering and mining of tailings will be undertaken by 6 m³ excavators and articulated 40 t trucks. Waste rock and ore will be moved by 17 m³ excavators, an 11.6 m³ wheel loader and 90 t trucks.

16.3.1.1 Slope Design

The slope design used in the open pit design was based on the geotechnical data collected, recommendations from Golder, experience, and best practice.

The designs for the hard rock slopes vary by sector, as recommended by Golder. Controlled blasting techniques will be used including buffer blasts and pre-splits. The pre-split is planned for a double bench (20 m) application and will be drilled once every 2 benches. The inter-ramp slope angles per sector are the same as the Golder recommendations, however the bench face in all sectors was set at 70 degrees as in the NI 43-101 Report of 2011.

16.3.1.2 Ramp Design

Haulage ramps were designed with consideration for the size of equipment. The larger haulage units are 90 t trucks which have an operating width of 6.2 m and a tire height of 2.6 m.

Ramp design considerations included the following:

- A berm height equal to at least half the tire height
- A passing width between trucks equal to ½ the operating width
- A ditch along the toe of the wall

Based on these factors two designs were created. Main haul roads will have 30 m wide road allowance with a maximum 10% gradient. A 25 m wide road allowance was used to access the final 40 m of the pit from bench -130 to -170.

16.3.1.3 Final Pit Design

The final pit design is shown in Section 15. The cross sections and plan shown in Figure 16-9, Figure 16-10 and Figure 16-11 depict the relationship between the unsmoothed Lerchs Grossmann shell for US\$ 1,200/oz Au and the Phase 1 and Phase 2 designs.

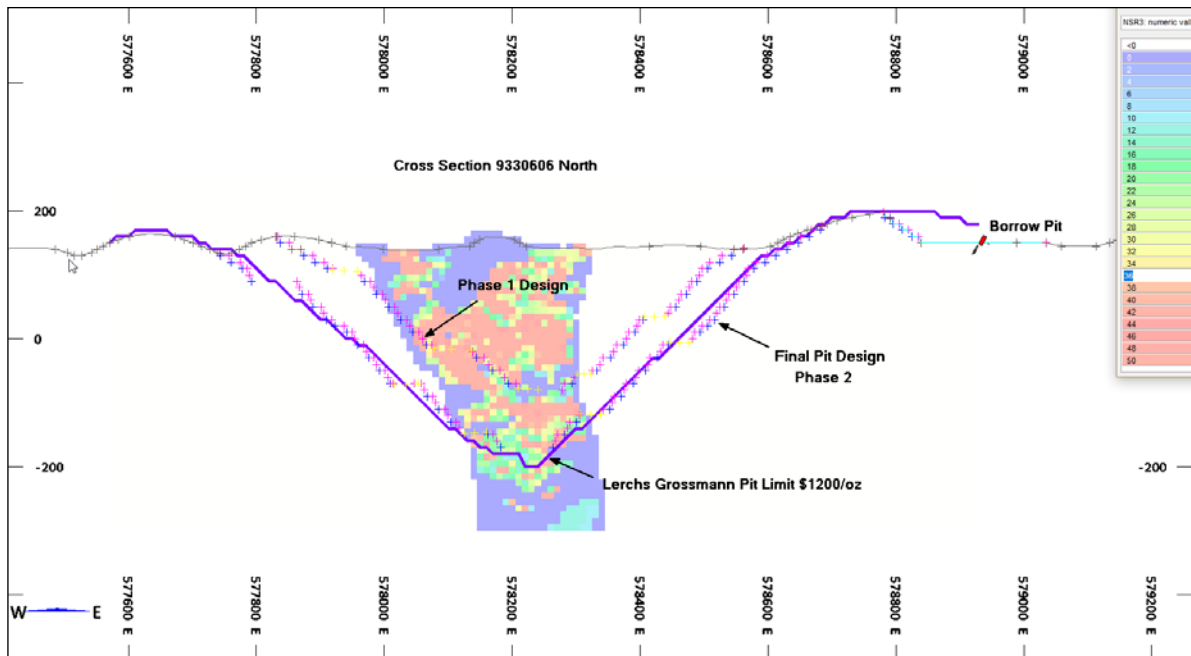


Figure 16-9: Section 9330606 North Pit Designs

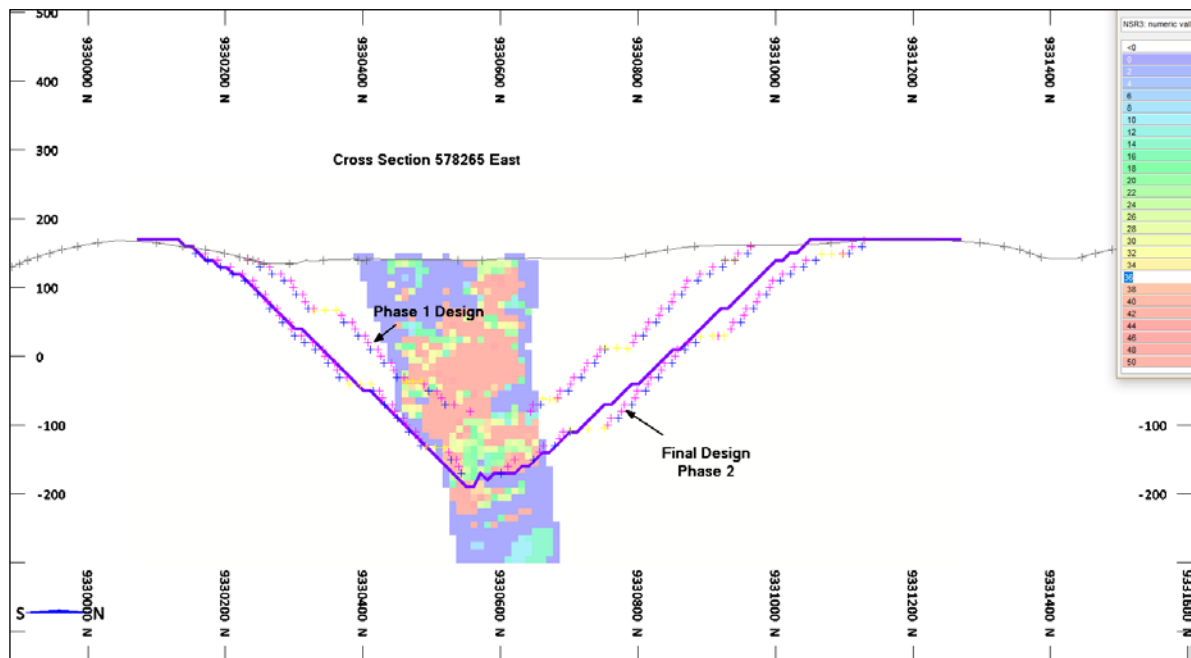


Figure 16-10: Section 578265 East Pit Designs

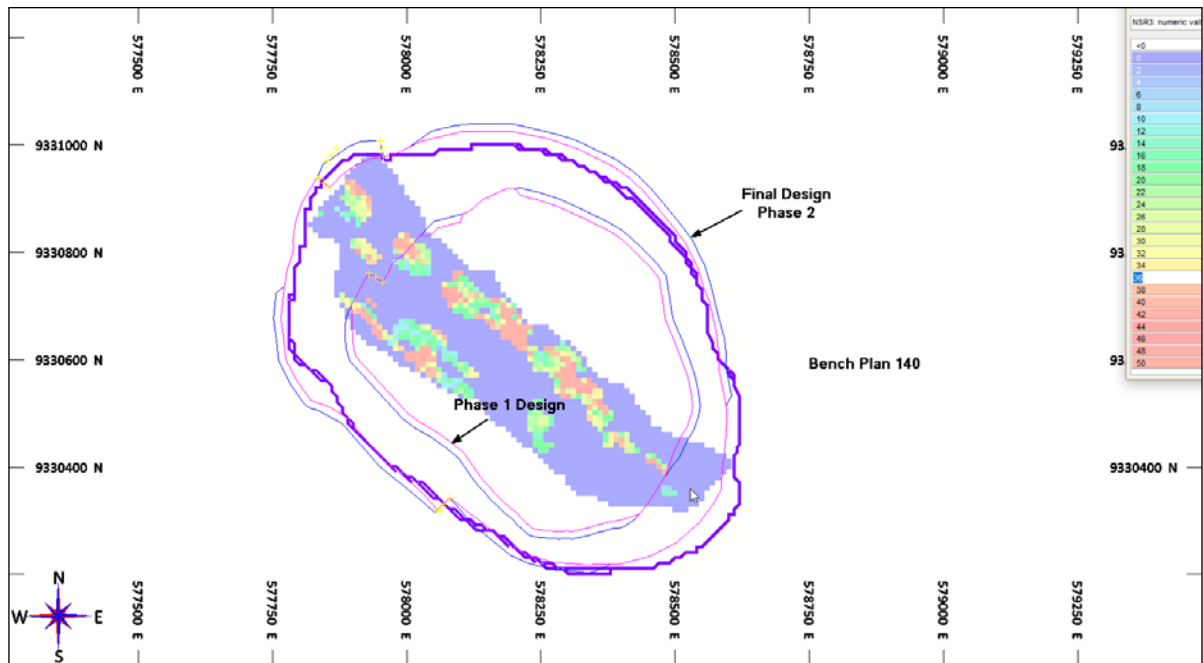


Figure 16-11: Bench Plan 120 Pit Designs

16.3.2 Production Schedule

The production schedule is shown in Table 16-3. A total of 40 Mt will be processed with an average grade of 1.41 g/t Au. The total waste moved will be 147.4 Mt including 29.5 Mt saprolite and 117.9 Mt waste rock.

The basis for the production schedule is as follows:

- Concentrator throughput 4.336 Mt per year
- Highest grade material mined is sent to the concentrator
- Low grade material stockpiled will be processed as required or at the end of the mine life
- Saprolite ore is processed at 176,000 t/a blended from stockpile
- Tailings material is processed at 120,000 t/a blended from stockpile
- Schedule by quarter to Year 6 then annual to end of mine life

Table 16-3: Production Schedule

Period	Units	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	TOTAL
Open Pit Production														
High Grade	t x 1000	31.9	717.9	3,247.2	4,237.5	4,456.8	4,397.4	2,718.0	3,893.8	3,904.5	3,252.0	987.0	-	31,844.0
	Au g/t	1.5	1.4	1.4	1.5	1.6	1.8	1.5	1.6	1.7	1.6	1.5	-	1.6
	NSR \$/t	50.4	46.9	46.5	50.8	52.6	58.9	48.1	50.9	56.8	51.7	48.9	-	52.2
Medium Grade	t x 1000	3.2	123.9	486.9	474.6	282.8	244.7	278.5	326.3	242.1	167.0	98.0	-	2,728.0
	Au g/t	0.5	0.6	0.5	0.5	0.5	0.5	0.5	0.5	0.6	0.5	0.5	-	0.5
	NSR \$/t	17.9	18.1	17.9	17.7	17.9	17.4	17.7	17.8	18.0	17.8	17.6	-	17.8
Low Grade	t x 1000	9.1	116.7	436.8	471.0	304.1	200.8	280.6	360.1	312.8	109.0	86.0	-	2,687.0
	Au g/t	0.8	0.5	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.5	-	0.4
	NSR \$/t	25.3	16.6	14.4	14.4	14.1	14.5	14.5	14.3	14.2	14.6	14.9	-	14.5
Saprolite	t x 1000	360.4	1,090.6	59.3	95.1	41.6	-	-	-	-	-	-	-	1,647.0
	Au g/t	1.3	1.2	1.1	0.9	0.8	-	-	-	-	-	-	-	1.2
	NSR \$/t	43.8	39.6	35.0	29.7	27.0	-	-	-	-	-	-	-	39.5
Tailings	t x 1000	208.9	539.8	15.9	325.4	6.0	-	-	-	-	-	-	-	1,096.0
	Au g/t	1.1	1.1	0.9	0.9	0.9	-	-	-	-	-	-	-	1.0
	NSR \$/t	34.2	34.2	-	0.5	2.7	-	-	-	-	-	-	-	31.6
Saprolite	t x 1000	6,519.2	8,875.2	4,694.6	6,904.2	2,459.7	-	-	-	-	-	-	-	29,453.0
Waste Rock	t x 1000	430.3	3,693.1	17,059.3	13,492.3	17,948.9	20,657.1	22,222.9	14,132.6	6,604.7	1,515.0	149.0	-	117,905.0
Waste	t x 1000	6,949.5	12,568.3	21,753.9	20,396.5	20,408.6	20,657.1	22,222.9	14,132.6	6,604.7	1,515.0	149.0	-	147,358.0
Total	t x 1000	7,563.0	15,157.1	26,000.0	26,000.0	25,500.0	25,500.0	25,500.0	18,712.8	11,064.0	5,043.0	1,320.0	-	187,360.0
Direct Mill Feed														
High Grade	t x 1000	-	-	3,247.2	4,025.0	4,040.0	4,040.0	2,718.0	3,752.0	3,904.5	3,252.0	987.0	-	29,965.7
	Au g/t	-	-	1.4	1.5	1.6	1.8	1.5	1.5	1.7	1.6	1.5	-	1.6
	NSR \$/t	-	-	46.5	50.8	52.5	58.7	48.1	50.7	56.8	51.7	48.9	-	52.2
Medium Grade	t x 1000	-	-	351.2	15.0	-	-	278.5	288.0	135.5	167.0	98.0	-	1,333.3
	Au g/t	-	-	0.5	0.5	-	-	0.5	0.5	0.6	0.5	0.5	-	0.5
	NSR \$/t	-	-	18.0	17.8	-	-	17.7	17.8	18.0	17.8	17.6	-	17.8
Low Grade	t x 1000	-	-	-	-	-	-	-	-	-	109.0	86.0	-	195.0
	Au g/t	-	-	-	-	-	-	-	-	-	0.4	0.5	-	0.4
	NSR \$/t	-	-	-	-	-	-	-	-	-	14.6	14.9	-	14.7
Saprolite	t x 1000	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au g/t	-	-	-	-	-	-	-	-	-	-	-	-	-
	NSR \$/t	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings	t x 1000	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au g/t	-	-	-	-	-	-	-	-	-	-	-	-	-
	NSR \$/t	-	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile Recovery														
High Grade	t x 1000	-	-	441.5	-	-	-	1,043.5	-	-	393.3	-	-	1,878.3
	Au g/t	-	-	1.4	-	-	-	1.6	-	-	1.6	-	-	1.6
	NSR \$/t	-	-	47.0	-	-	-	53.3	-	-	53.8	-	-	51.9
Medium Grade	t x 1000	-	-	-	-	-	-	-	-	-	118.7	1,000.0	276.0	1,394.7
	Au g/t	-	-	-	-	-	-	-	-	-	0.6	0.6	0.6	0.6
	NSR \$/t	-	-	-	-	-	-	-	-	-	20.4	20.4	20.4	20.4
Low Grade	t x 1000	-	-	-	-	-	-	-	-	-	-	1,869.0	623.0	2,492.0
	Au g/t	-	-	-	-	-	-	-	-	-	-	0.4	0.4	0.4
	NSR \$/t	-	-	-	-	-	-	-	-	-	-	14.5	14.5	14.5
Saprolite	t x 1000	-	-	176.0	176.0	176.0	176.0	176.0	176.0	176.0	176.0	176.0	63.0	1,647.0
	Au g/t	-	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
	NSR \$/t	-	-	40.5	40.2	39.4	39.2	39.2	39.2	39.2	39.2	39.2	39.2	39.5
Tailings	t x 1000	-	-	120.0	120.0	120.0	120.0	120.0	120.0	120.0	120.0	120.0	16.0	1,096.0
	Au g/t	-	-	1.1	1.1	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
	NSR \$/t	-	-	34.2	28.4	21.3	21.3	21.3	21.3	21.3	21.3	21.3	21.3	23.5
Milling														
	t x 1000	-	-	4,336.0	4,336.0	4,336.0	4,336.0	4,336.0	4,336.0	4,336.0	4,336.0	4,336.0	978.0	40,002.0
	Au g/t	-	-	1.3	1.5	1.6	1.7	1.4	1.4	1.7	1.5	0.8	0.6	1.4
	NSR \$/t	-	-	43.6	49.6	51.1	56.9	46.3	46.4	53.9	47.4	24.9	17.9	46.2
	t/d	-	-	11,879.5	11,879.6	11,879.5	11,879.5	11,879.5	11,879.5	11,879.5	11,879.5	11,879.5	2,679.4	
Total Mining														
	t x 1000	7,563.0	15,157.1	26,617.6	26,176.0	25,676.0	25,676.0	26,719.5	18,888.8	11,240.0	5,731.0	4,365.0	962.0	194,772.0

Material movement schedule quantities and head grades are shown in Figure 16-12.

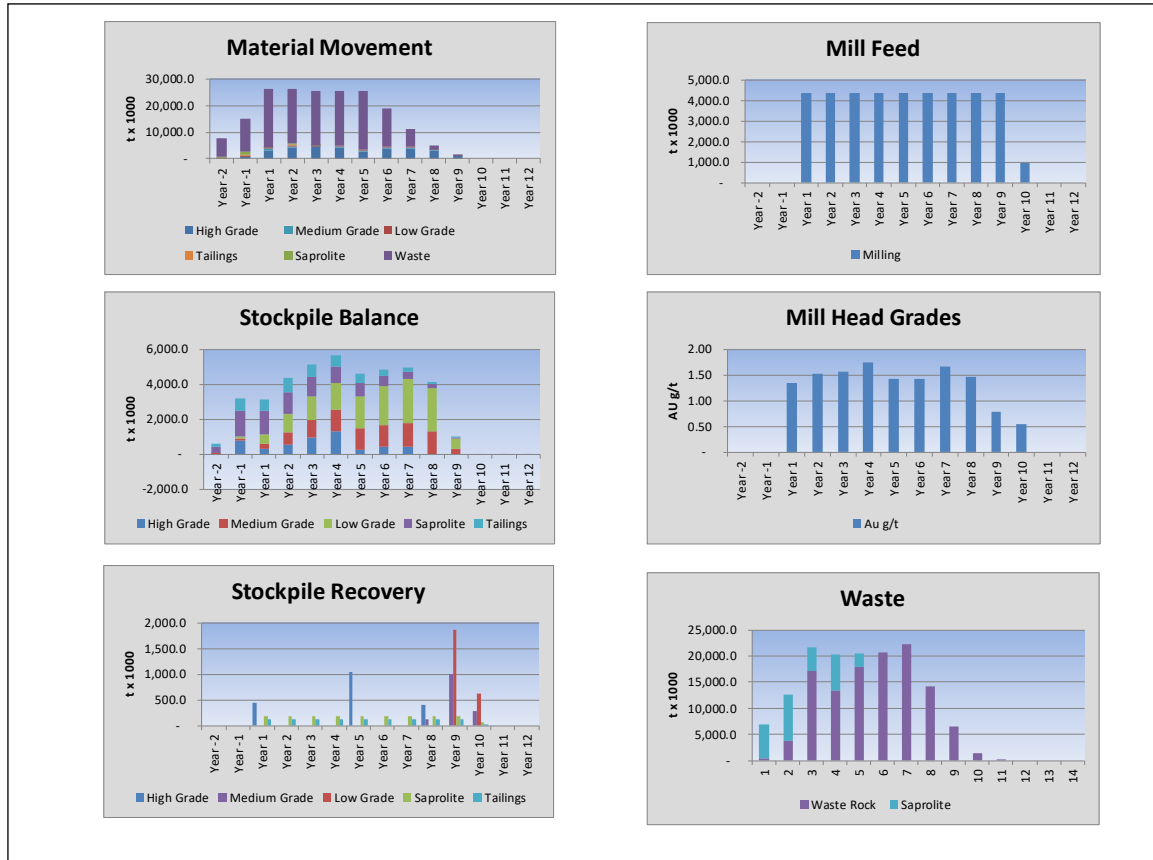


Figure 16-12: Material Movement Summary

Mine developments for selected periods are shown in Figure 16-13 through Figure 16-17.

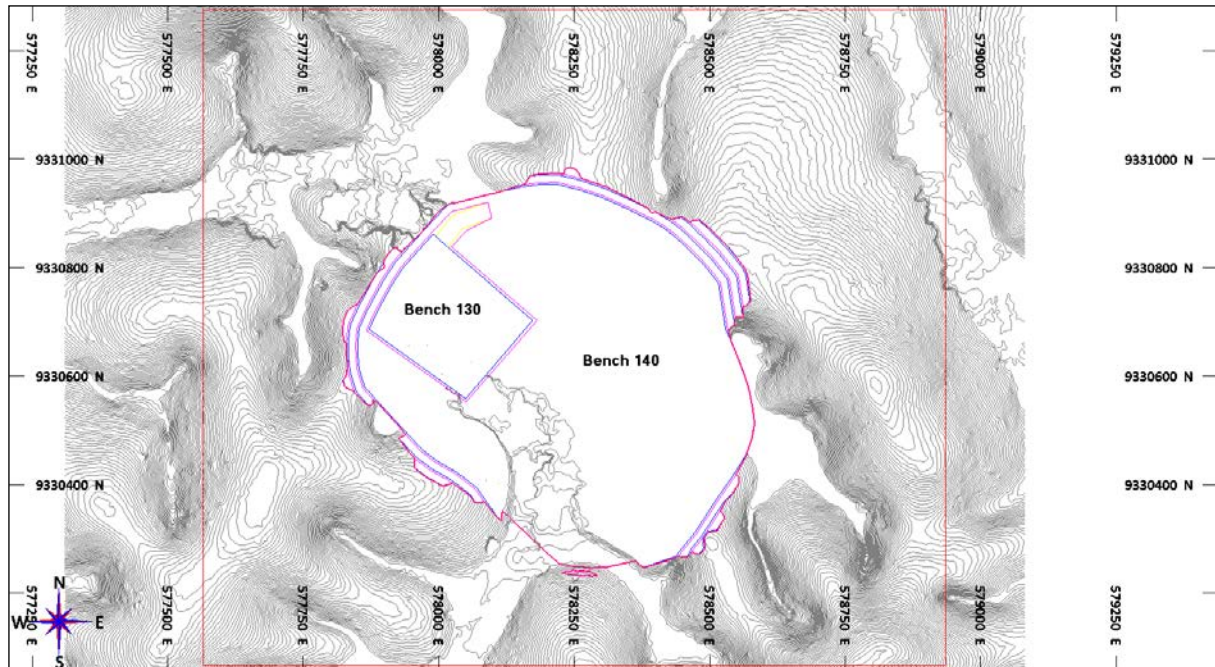


Figure 16-13: Mine Development Year -2

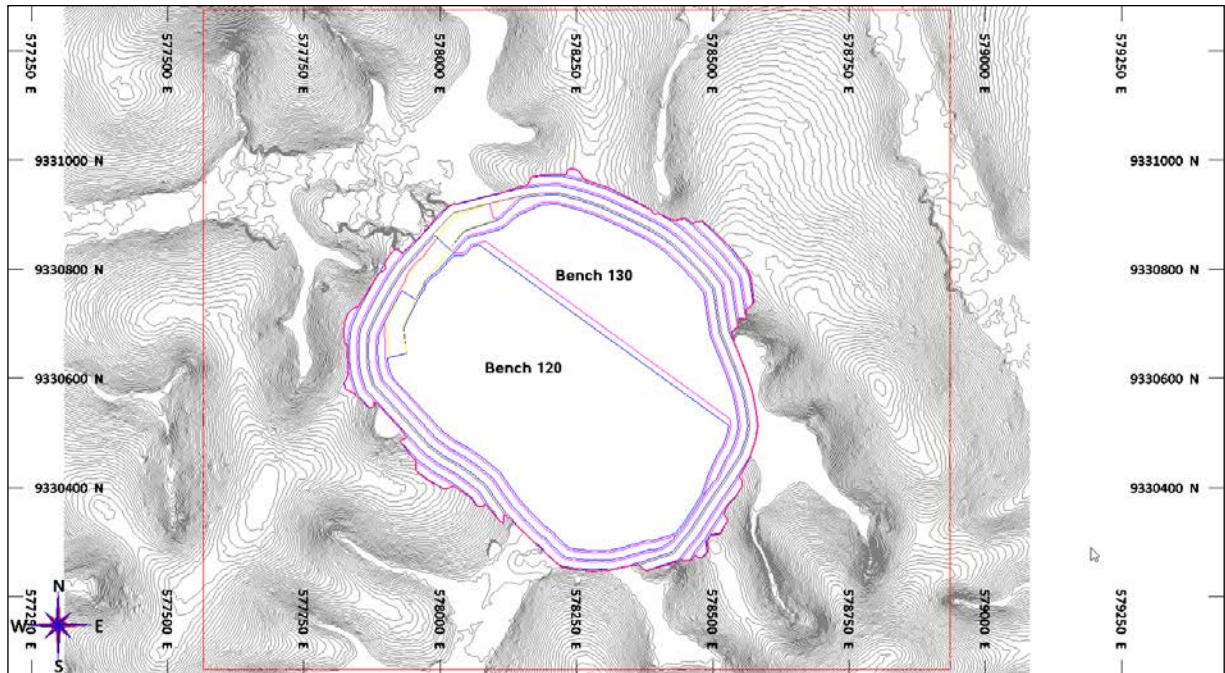


Figure 16-14: Mine Development Year -1

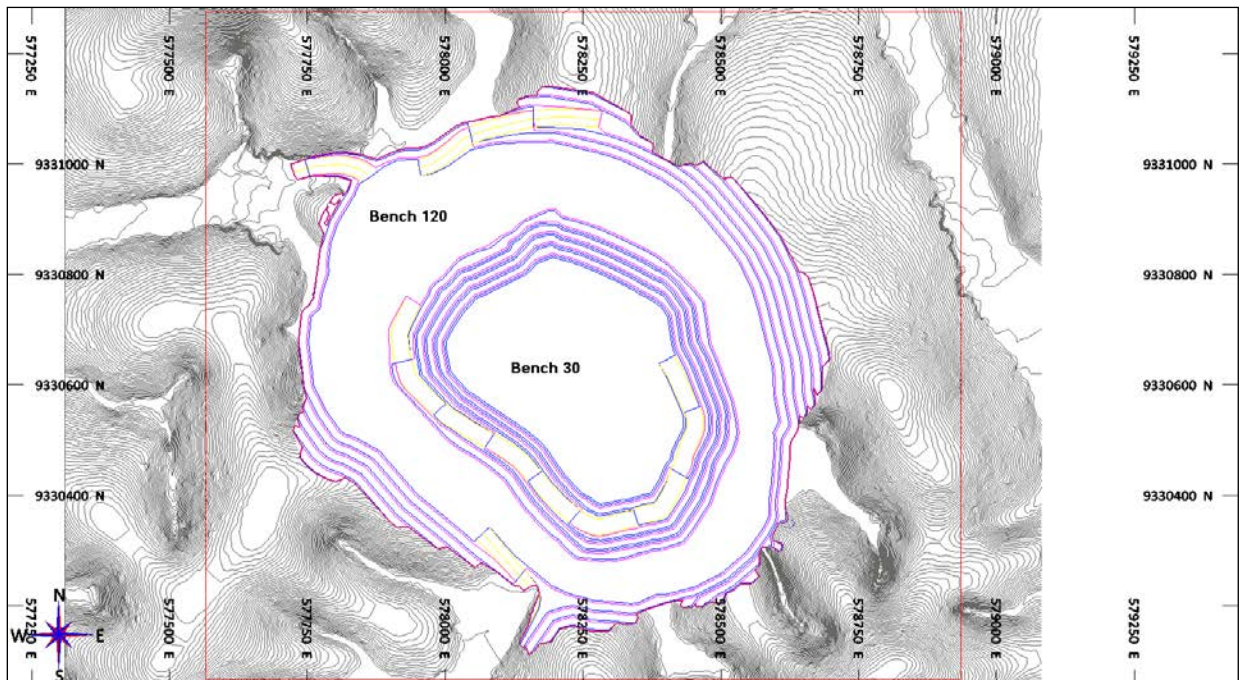


Figure 16-15: Mine Development Year 2

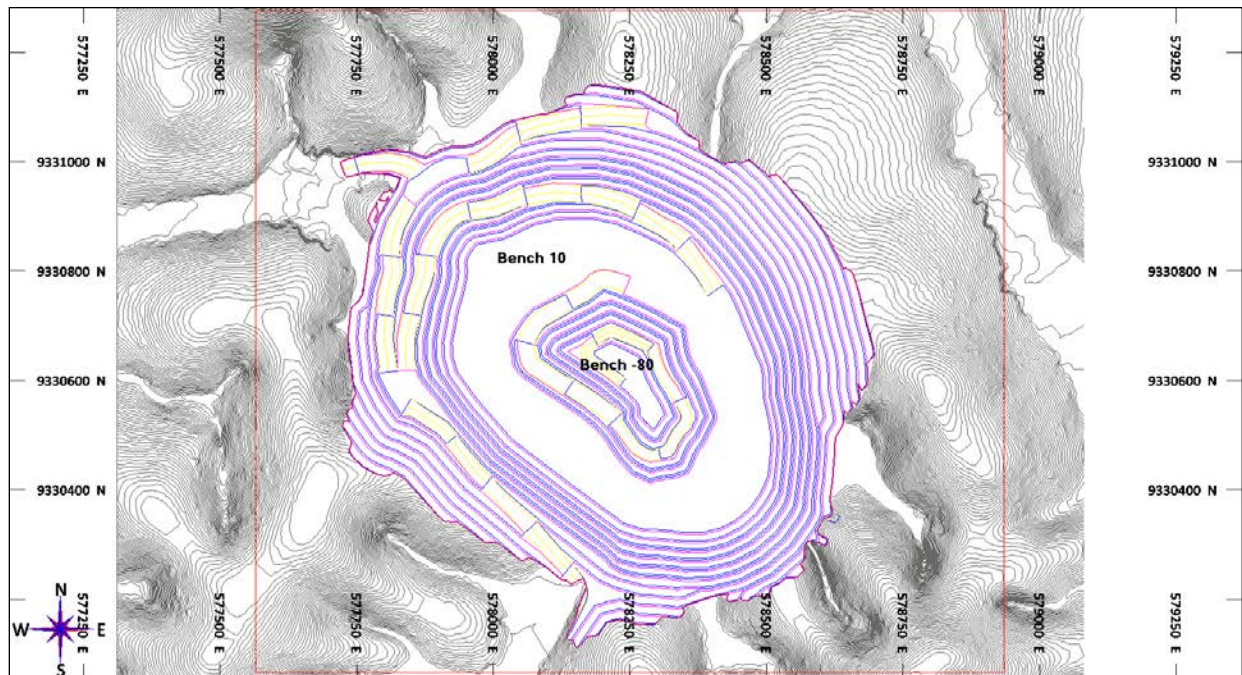


Figure 16-16: Mine Development Year 5

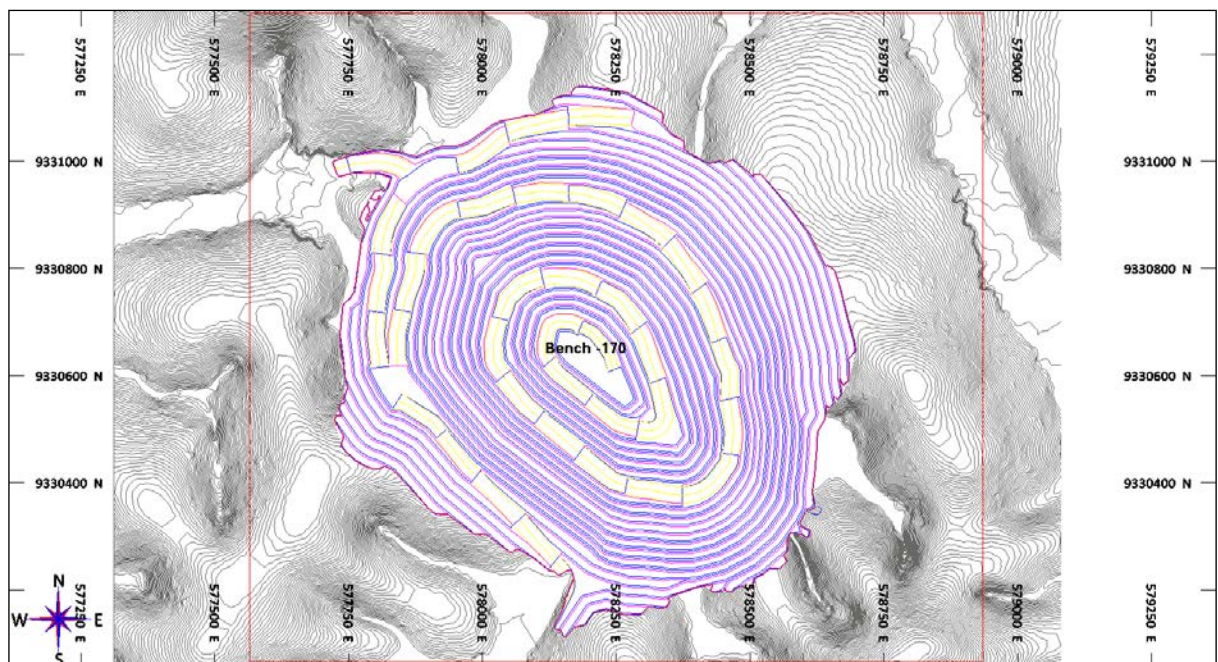


Figure 16-17: Mine Development Year 9

16.3.3 Waste Rock Disposal

A waste rock disposal area was designed to permanently store the waste rock that will be mined from the Tocantinzinho open pit. The following design criteria were used:

- Storage capacity of approximately 82.6 million m³ on a footprint of 126 ha
- Maximum height of 142 m – elevation 265 m
- Lift height of 20 m with angle of repose slopes of 37 degrees
- Berm width 10 m and overall slope of approximately 28 degrees
- A surface water drainage plan that allows for collection and settling of all water run-off
- A detailed system of drains to be constructed in the base of the storage facility;
- An offset distance of 300 m horizontally and 5 m vertically from the Tocantinzinho River

The planned waste rock storage area is shown in Figure 16-18. Progress development of the storage facility at the end of the pre-production period is shown Figure 16-19.



Figure 16-18: Waste Rock and Stockpile Disposal Layout

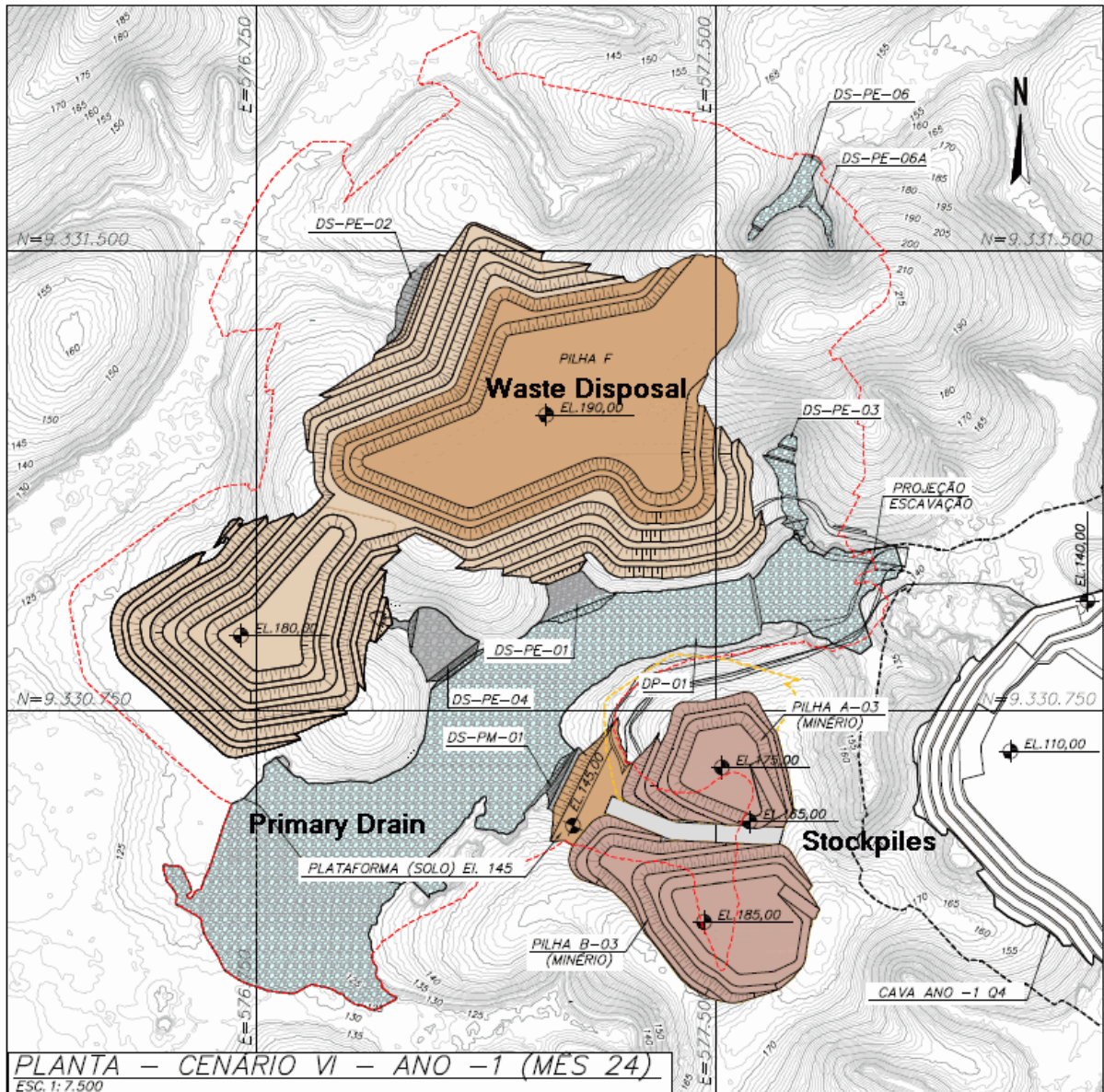


Figure 16-19: Waste Storage Facility Development Year -1

A section through the storage facility in Figure 16-20 shows a 3% slope on the berm surfaces to allow for drainage on the surface. This design was provided by Tec3 Geotechnia E Recursos Hidricos.

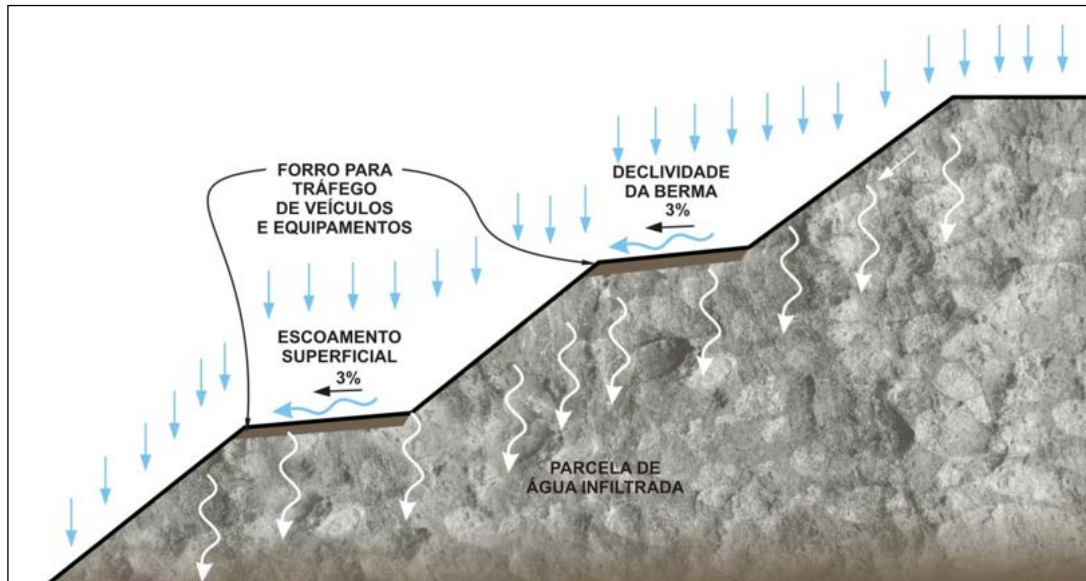


Figure 16-20: Inclined Berms

Detailed stability analyses of the storage facility have been undertaken for undrained and drained conditions. The locations of the analyses are shown in Figure 16-21.

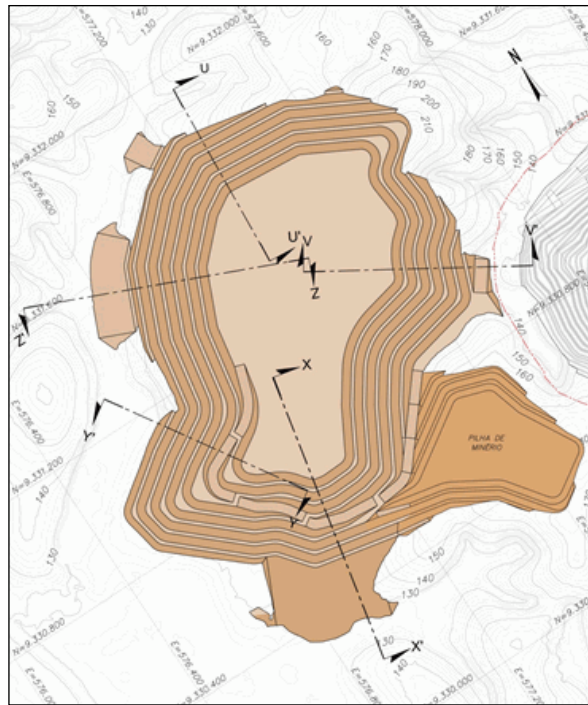


Figure 16-21: Section Line Locations - Stability Analyses

Results of the analyses are shown in Table 16-4. Calculated factors of safety exceeded minimum standards in all cases.

Table 16-4: Stability Analyses Results

Condition	Location	Saturation Setting	Minimum Factor of Safety	Factor of Safety Obtained
-	Between Berms	-	1.5	1.63
Non Drained	Section U-U'	Nominal Angle of Operation	1.3	1.53
	Section V-V'			1.50
	Section X-X'			1.83
	Section Y-Y'			1.50
	Section Z-Z'			1.66
Final	Section U-U'	Nominal Angle of Operation	1.5	1.53
	Section V-V'			1.50
	Section X-X'			1.97
	Section Y-Y'			1.50
	Section Z-Z'			1.72

16.3.4 Mine Equipment and Personnel

The type of mining equipment was selected to match the materials encountered during mining, the pit geometry and the production requirements. Unit operations were divided into drilling, blasting, loading, hauling and support activities. The size of the fleet was estimated to meet the production requirements allocated and included estimation of the haulage routes over the life of the project and the productivity assumptions for drills, loading and hauling units.

16.3.4.1 Drilling

The drills selected for production will be capable of single pass rotary and down the hole hammer drilling in a range from 152 mm to 270 mm. Production estimates have been made for the 216 mm configuration. Wall control drilling will be done with 114 mm hydraulic drills.

16.3.4.2 Blasting

The drill and blast patterns were designed by International Blasting Consultants to provide adequate explosive distribution for optimal fragmentation. The primary blast pattern design is based on the following:

- 216 mm (or 8.5") drill holes
- A sub-drill of 1.8 m in ore and 2.13 in waste
- Burden & Spacing for Ore 5.5 m x 6.1 m and Waste 5.8 m x 6.7 m

Typical blast patterns for ore, waste and wall control are shown in the Figure 16-22 to Figure 16-24.

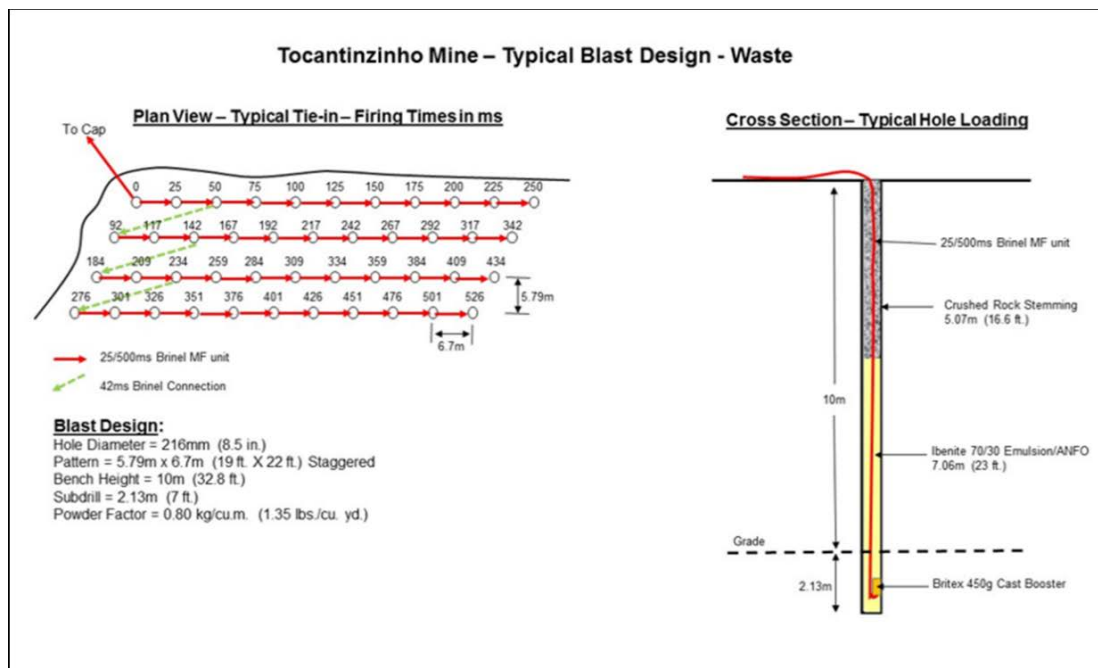


Figure 16-22: Blast Design Layout Ore

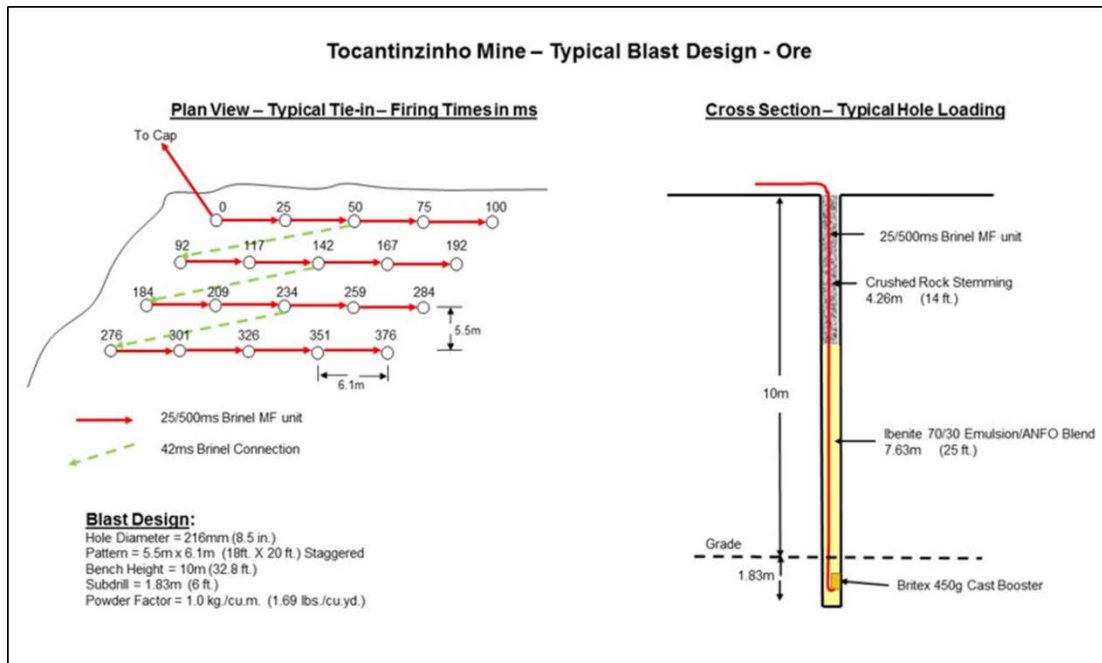


Figure 16-23: Blast Design Layout Waste

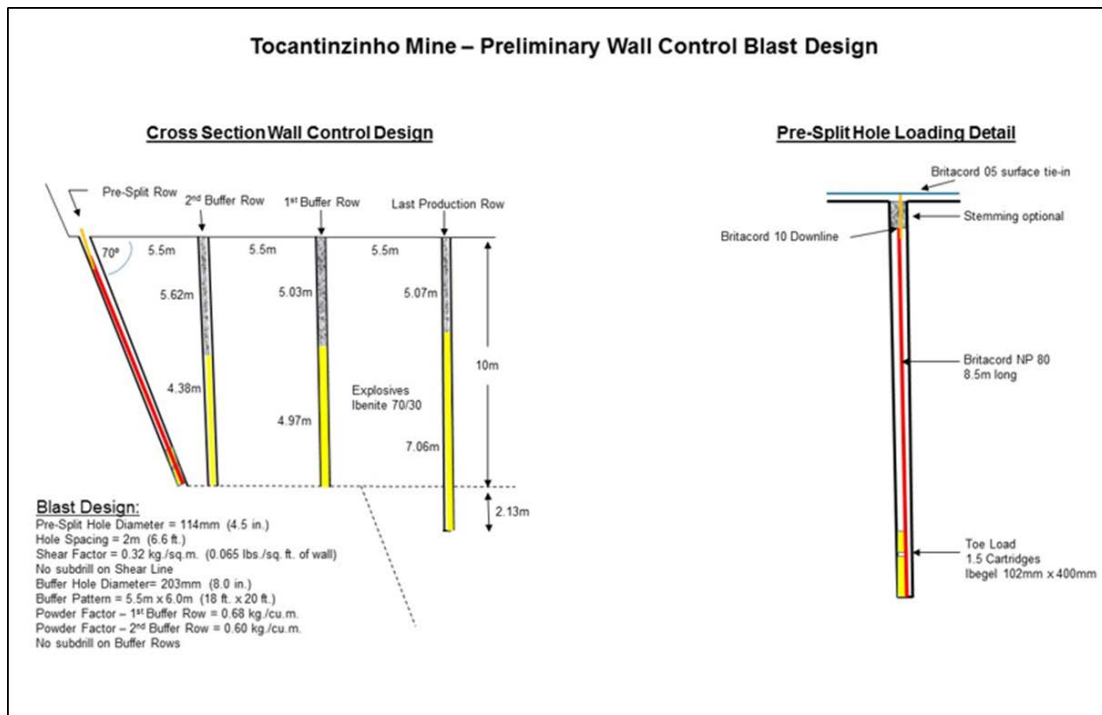


Figure 16-24: Blast Design Wall Control

16.3.4.3 Explosives

The explosive type and loading amounts required to develop adequate fragmentation will depend on numerous site conditions such as hardness of the rock and in-hole water conditions. These conditions will vary throughout the pit and will vary during wet and dry seasons of the year.

It is assumed that saprolite material will be primarily free digging and 90% of the saprolite will not require blasting. All hard rock material will require blasting. Ibenite 70/30 Emulsion/ANFO is the proposed blasting agent. The following explosive consumption parameters were assumed in this study:

- An average powder factor of 0.37 kg/t of ore and 0.29 kg/t waste
- All blast holes tied-in with non-electric surface delays and detonation cord
- One primer and cap per hole
- Crushed rock stemming

The loading & hauling was divided into two fleets for:

- Saprolite pioneering and tailings excavation
- Waste mining and ore mining. Mine personnel requirements were estimated based on the fleet size, the shift roster, and estimates for administration staff and maintenance staff.

A fleet of 6 m³ excavators matched with 38 t articulated haulage trucks will be used in the pioneering areas to move tailings and saprolite to various construction sites, road building locations and to the stockpiles and waste dumps.

A fleet of 17 m³ excavators matched with 90 t rigid frame haulage trucks will be used in mining both saprolite ore and waste rock.

Support equipment will include bulldozers, graders, smaller wheel loaders, rock breaker, water truck and service vehicles.

16.3.4.4 Haulage requirements

The haul truck requirements were estimated using destination based cycle times from each bench and material type. Loaded uphill truck speeds were set to 10 km/hr on 10% grades and rough bench speeds were limited to 25 km/hr. Loaded truck speed limits were set to 36 km/hr and empty truck speeds were set to 38 km/hr. Rimpull based speeds were otherwise applied for a rolling resistance of 3%. Schedule periods were by quarter to Year 6 and then then annual to the end of the life of mine. Cycle times by elevation and material type are shown in Figure 16-25 and Figure 16-26.

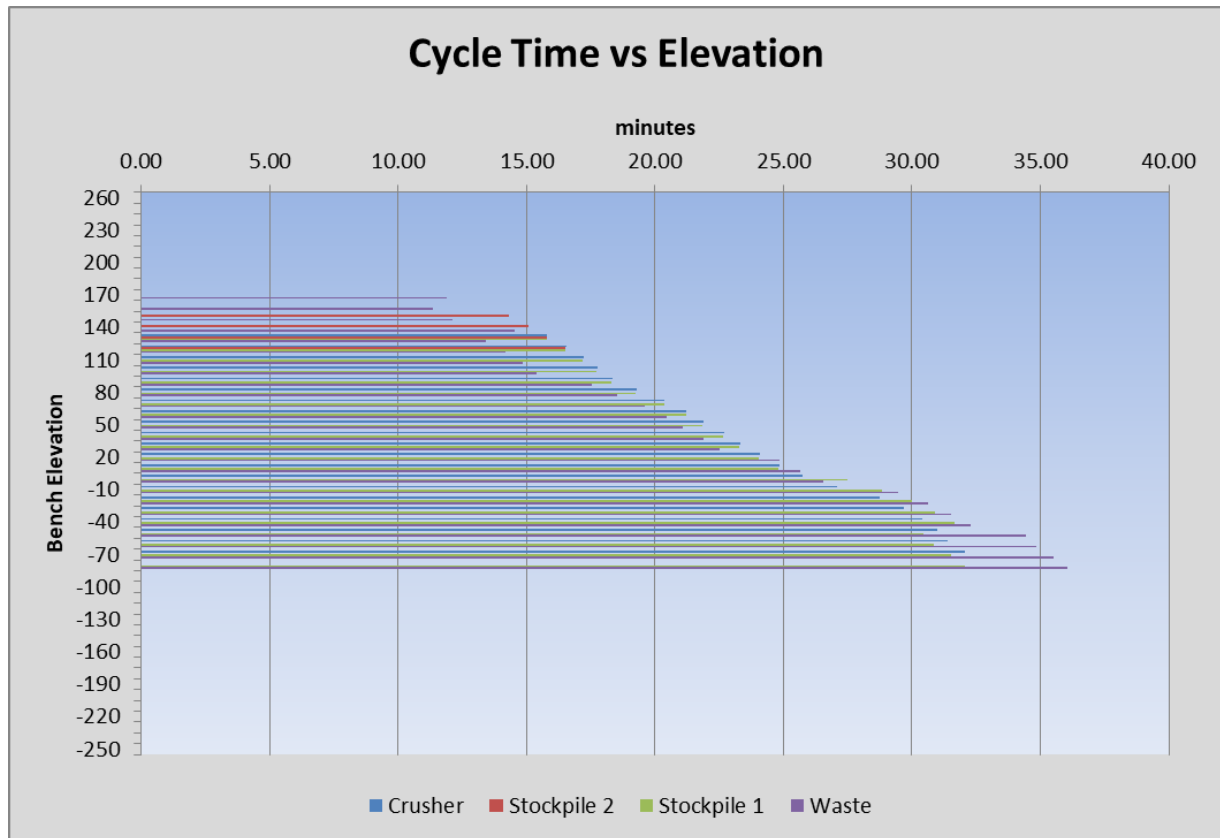


Figure 16-25: Phase 1 Cycle Times by Elevation

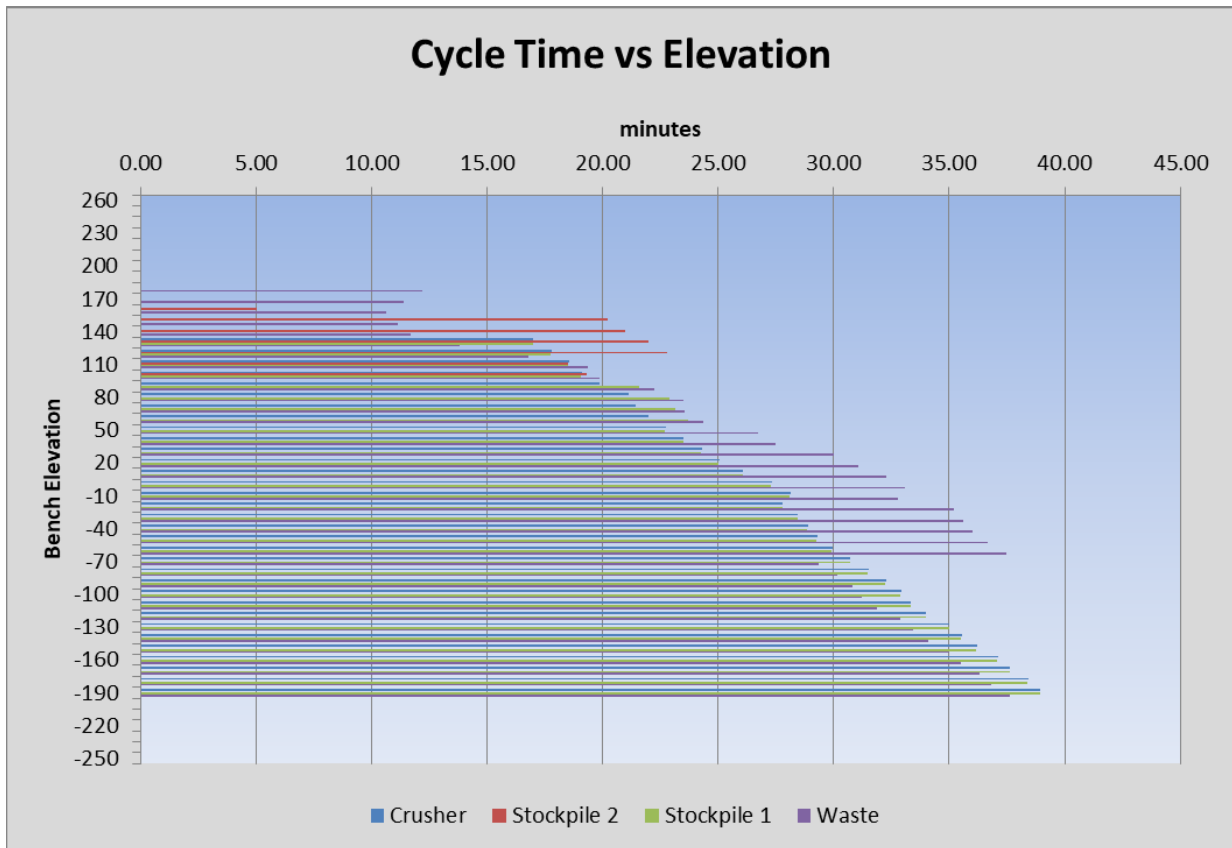


Figure 16-26: Phase 2 Cycle Times by Elevation

16.3.4.5 Shift roster, equipment availability, effectiveness and net utilization

The shift roster for the mining operation was based on the following criteria which were selected due to the remote setting of the Tocantinzinho Project:

- Two 12 hour working shifts per day, with one hour of break time for meals
- A 5 days 5 nights 5 days off rotation
- Three crews: one on day shift, one on night shift and one on break

Based on this rotation, the total use of available time was calculated to be 22 of 24 hours per day or 92%. It was assumed that blasting operations would take place during the break period for meals and thus would not reduce the total use of availability.

16.3.4.6 Mining fleet requirements

The fleet for major mining equipment per year are shown in Table 16-5. The major fleet was selected to match the expected mining conditions and to achieve the production schedule.

Table 16-5: Major Equipment Requirements by Year

	Make	Model	Size	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Drilling															
Blasthole Drill	Atlas Copco	PV-235	216 mm	1	2	3	3	3	3	3	3	3	3	2	-
Wall Control Drill	Atlas Copco	ROC L6	114 mm	-	1	1	2	2	2	2	2	2	2	2	-
Loading															
Hydraulic Shovel/Excavator	Caterpillar	6030	17 m ³	-	1	2	2	2	2	2	2	2	2	2	-
Wheel Loader	Caterpillar	992K	11.6 m ³	-	1	1	1	1	1	1	1	1	1	1	1
Wheel Loader	Caterpillar	980M	5.4 m ³	1	2	2	2	2	2	2	2	2	2	2	-
Excavator	Caterpillar	390F-ME	6 m ³	3	3	3	3	3	-	-	-	-	-	-	-
Hauling															
Haul Truck	Caterpillar	777G	90 t	-	4	13	14	16	20	24	19	11	11	4	4
Haul Truck	Caterpillar	740EJ	38 t	13	13	8	8	8	2	2	2	2	2	2	2
Roads & Dumps															
Track Dozer	Caterpillar	D9T	346 kW	2	2	2	2	2	2	2	2	2	2	2	1
Track Dozer	Caterpillar	D8T	264 kW	2	2	2	2	2	2	2	2	2	-	-	-
Wheel Dozer	Caterpillar	834K	370kW	-	1	1	1	1	1	1	1	1	1	1	1
Motor Grader	Caterpillar	16M	216 kW	1	1	1	1	1	1	1	1	1	1	1	1
Motor Grader	Caterpillar	14M	177 kW	2	2	2	2	2	2	2	2	2	1	1	1
Water Truck	M-Benz	2423K TK	15,000 l	2	2	2	2	2	2	2	2	2	2	2	1
Excavator	Volvo	EC700B	200 kW	-	-	2	2	2	2	2	2	2	1	1	1
Front End Loader	Caterpillar	980M	5.4 m ³	1	1	1	1	1	1	1	1	1	1	1	1
Rock Breaker	Volvo	EC360B	200 kW	1	1	1	1	1	1	1	1	1	1	1	1
Support Equipment															
Low Bed Transporter	M-Benz	3354	60 tonne	1	1	1	1	1	1	1	1	1	1	1	1
Truck Crane/Telehandler	TF/Caterpillar	ATS65/514	65t/5t	1	1	1	1	1	1	1	1	1	1	1	1
Tire Handler/Forklift	IMT	TH3565	25t	1	1	1	1	1	1	1	1	1	1	1	1
Fuel/Lube Truck	M-Benz	3340	20,000L	1	1	1	1	1	1	1	1	1	1	1	1
HD Mechanic's Field Truck	M-Benz	3340	-	1	2	2	2	2	2	2	2	2	2	2	1
Mechanic's Service Truck	M-Benz	3340	-	1	2	2	2	2	2	2	2	2	2	2	1
Welding Truck	Manufacturer	3340	-	1	1	1	1	1	1	1	1	1	1	1	1
Service Pickup	Manufacturer	4x4	3/4 tonne	10	12	12	12	12	12	12	12	12	10	10	2
Light Plant	Manufacturer	-	-	4	6	6	6	6	6	6	6	6	3	3	2
Crushing Plant	-	-	-	1	1	1	1	1	1	1	1	1	1	1	-
Mine Bus	-	-	-	-	-	1	1	1	1	1	1	1	1	1	-

16.3.4.7 Mine personnel requirements

Mine personnel were divided into hourly and staff positions and were divided between mine operations, mine maintenance, engineering and geology. Hourly positions were all associated with a shift roster of 2 weeks on and 1 off and as such each unit of equipment required 3 operators hired in hourly positions.

Staff positions were selected for administrative roles and generally were assumed to work a day shift only 5 days/week rotation. In some case where 24 hour support in the staff role was necessary, the staff position was planned to be on the same 2 week on one off rotation as the hourly staff.

16.3.5 Mine Infrastructure

Mine infrastructure was designed to support the open pit operation. Several key items were planned and outlined below.

16.3.5.1 Fuel Storage, Truck Shop, Truck Wash, Explosives

These items are covered in Section 18.

16.3.5.2 Aggregate Plant

An aggregate plant with a crusher, screen and associate conveyors is planned. This plant will have the capacity to produce a 7.62 cm (3") crushed rock material for use on haul roads in the pit and on the dumps and a fine screened sub ½" material for drillhole stemming.

16.4 MINE OPERATING COSTS

Mine operating costs have been estimated on an annual basis for all labour, equipment and consumables according to material types, destinations and quantities moved according to the production schedule.

SECTION • 17 RECOVERY METHODS

17.1 INTRODUCTION

The process selected for the Tocantinzinho Project is based on testwork described in Section 13 and is a flotation concentrate cyanide leach and carbon adsorption flowsheet comprising crushing, grinding, gravity concentration, flotation, cyanide leaching, carbon adsorption, cyanide detoxification, carbon elution and regeneration, gold refining, and tailings disposal.

The Tocantinzinho process plant will process run of mine (ROM) granite ore, along with minor amounts of saprolite and garimpeiros tailings, and produce gold doré bars and tailings.

The mill is designed with a nominal capacity of 4.3 Mtpa at a planned average feed grade of 1.41 g/t Au, producing 174,000 oz of Au annually with a life of mine of 9 years. The plant is designed to treat ore with a maximum head grade of 1.76 g/t Au.

The plant will consist of the following unit operations:

- **Primary crushing** – A vibrating grizzly and jaw crusher in open circuit producing a final product of 80% passing (P80) 148 mm
- **Coarse ore stockpile and reclaim** – A 12 hours live storage crushed ore stockpile with two reclaim apron feeders feeding the SAG Mill feed conveyor
- **Grinding** – A SAG mill equipped with a water-jet system for pebble recirculation producing a transfer product P80 of 1000µm to secondary grinding with a ball mill in closed circuit with hydrocyclones producing a final product P80 of 125µm
- **Gravity concentration** – Gravity concentration of hydrocyclone underflow from the secondary grinding circuit to produce a gold-rich concentrate for intensive leaching
- **Intensive cyanidation** – Gravity gold dissolution within the intensive cyanidation reactor for subsequent gold recovery in electrowinning
- **Sulphide flotation** - 2-stage flotation circuit to produce sulphide concentrate for cyanide leaching
- **Pre-aeration, cyanide leaching, and carbon adsorption** – Pre-aeration of feed followed by gold leaching by cyanidation, facilitated by oxygen, followed by adsorption of solution gold onto carbon particles via a Carbon in Pulp (CIP) carousel pump-cell configuration
- **Cyanide detoxification** – Detoxification of cyanide slurry via sodium metabisulphite for SO₂, oxygen and copper sulphate to achieve < 0.2 ppm for CN_{TOT} (total) and for CN_{WAD} (weak acid dissociable)
- **Carbon elution and regeneration** – Acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold rich solution, and thermal regeneration of carbon to remove organic foulants
- **Gold refining** – Gold electrowinning (sludge production), filtration, drying, and smelting to produce gold doré
- **Tailings** – Flotation tailings and concentrate cyanidation tailings (i.e. CIP tailings) are stored in separate tailings storage facilities

- **Water treatment (polishing) plant (WTP) *Future*** – Excess water from the CIP tailings is treated for the removal of metals in solution (i.e. Cu) prior to being released to the environment

Figure 17-1 presents the overall flowsheet for the Tocantinzinho Project. Figure 17-2 presents the overall layout for the Tocantinzinho processing plant.

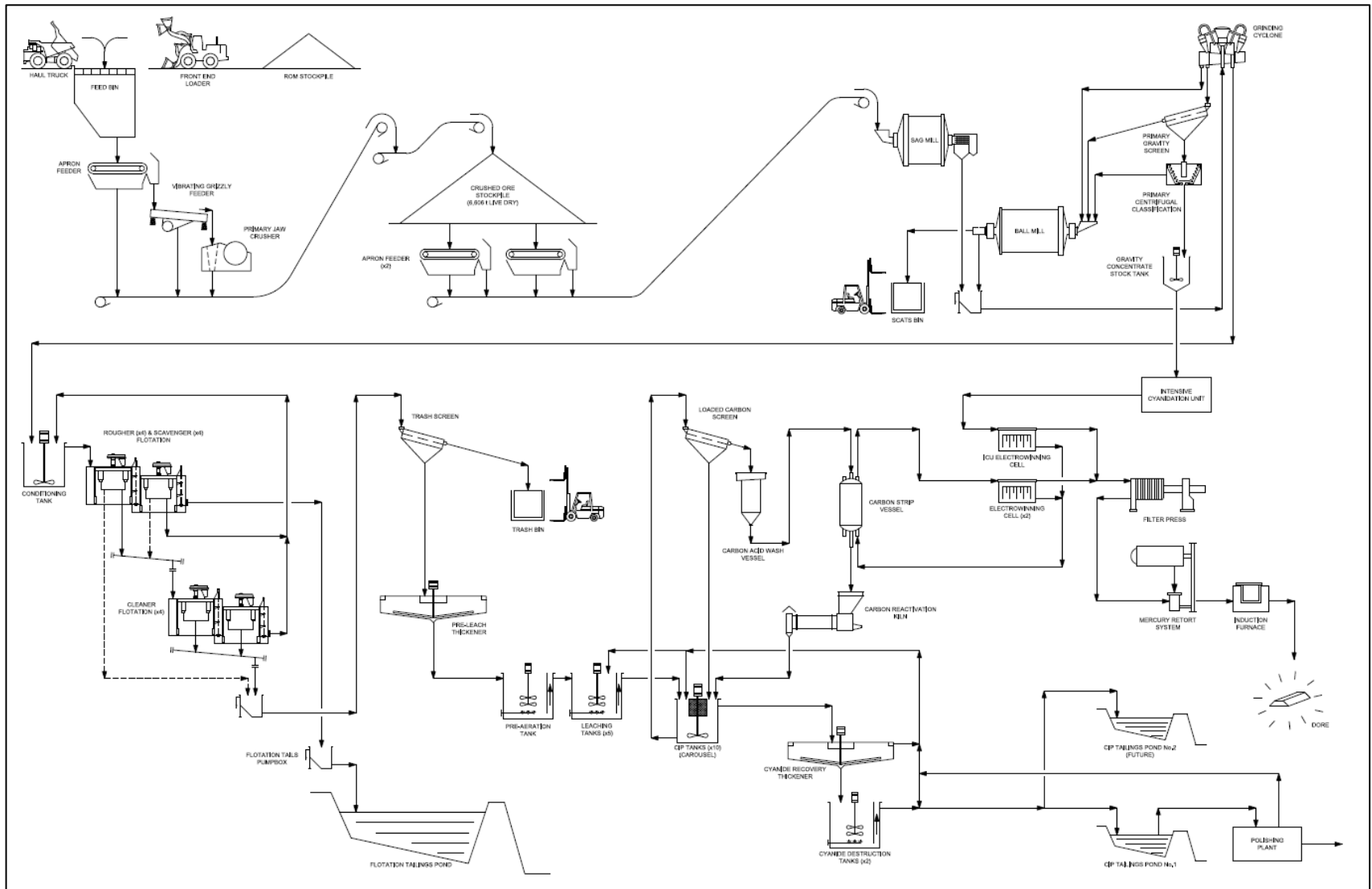


Figure 17-1: Overall Flowsheet

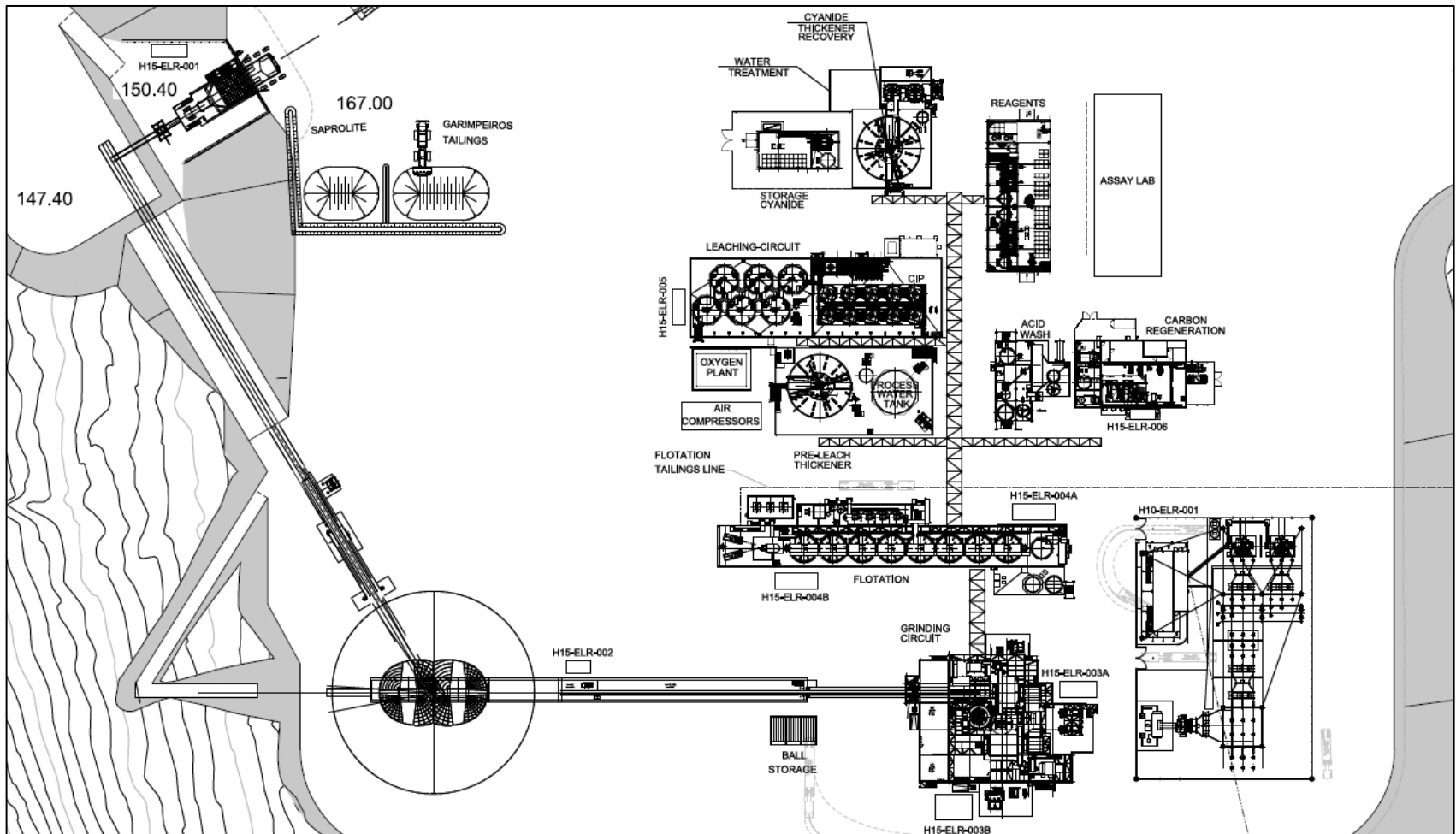


Figure 17-2: Overall Layout for Processing Facility

17.2 PROCESS DESIGN CRITERIA

The process design criteria detail the annual ore and product capabilities, major mass flows and capacities, and plant availability. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 21. Key process design criteria are given in Table 17-1 .

Table 17-1: Key Process Design Criteria

Area	Criteria	Unit	Nominal Value
General	Average gold head grade	g/t	1.41
	Daily Throughput	t/d	11,890
	Crusher Plant Availability	%	75
	Process Plant Availability	%	90
	Total Gold Recovery	%	88.4
	Average Annual Gold Production	oz/a	174,000
Crushing	Crusher Work Index	kWh/t	15.4
	Run of Mine (ROM), maximum size	mm	1,000
	Crusher Circuit Product Size (P80)	mm	148
	Stockpile Capacity (live)	h	12
Grinding	JK breakage parameter A x b	-	51.5
	Bond Ball Mill Work Index	kWh/t	18.2
	Ball Mill Circuit Product Size (P80)	µm	125
Gravity Concentration	Gravity Concentrate per Cycle	kg	60
	Gravity Concentrator Gold Recovery (Of Fresh Feed)	%	25
Intensive Cyanidation	Solids Capacity per Batch	kg	3,130
	Gold Recovery	%	94.5
Flotation	Rougher-Scavenger Flotation Design Retention Time	min	32.6
	Cleaner Flotation Design Retention Time	min	21
	Cleaner Flotation Mass Pull	%	4.5
	Rougher Flotation Slurry Feed Rate	t/h	2,270
	Cleaner Flotation Slurry Feed Rate	t/h	330
	Feed Density – Rougher Flotation	%w/w	28.9
	Au Flotation Recovery (Of Fresh Feed)	%	68.5
Pre-Leach Thickening	Thickener Underflow Density	%w/w	60
	Settling Rate	m ² /tpd	0.35
Leaching	Pre-Aeration Stages	-	1
	Pre-Aeration Residence Time	h	8
	Leaching Stages	-	5
	Leaching Residence Time	h	40
	Leaching Gold Extraction	%	94.5

Area	Criteria	Unit	Nominal Value
	Leaching Solid Density	%	42
Carbon in Pulp (CIP)	CIP Solids Density	%	35
	Stage Residence Time (per tank)	h	1.88
	CIP Carbon Concentration	g/L	40
	Loaded Carbon Grade, Au	g/t	2,761
ADR Plant	Number of Elution Vessels	-	1
	Elution Batch Size (Carbon)	t	4
Cyanide Detox	Cyanide Destruction System	-	Air/ SO ₂
	Number of Stages	-	2
	Total Retention Time	mins	180
	Solids Density	wt%	40
	Pulp Flowrate	m ³ /h	44.3

17.3 PROCESS PLANT DESCRIPTION

17.3.1 Primary Crushing

Ore from open pit mining operations will feed a primary jaw crusher system which produces a product size P80 of 148 mm. Saprolite ore and artisanal mining (garimpeiros) tailings will be stockpiled in designated areas near the ROM bin. A front end loader will transfer these ores from their designated stockpiles into the ROM bin with granite ore at specific blend ratios required at the processing facility.

Haul trucks will dump ROM ore onto a static grizzly with 700 mm openings overtop the ROM bin. The ROM bin will have a 300 t live storage capacity which corresponds to two dump truck loads. A mobile rock breaker will break any oversized material captured at the static grizzly. An apron feeder will draw material from the ROM bin and discharge onto a vibrating grizzly feeder with 102 mm spacing. The oversized material will discharge directly into the primary jaw crusher. The undersized material will bypass the jaw crusher and combine with primary crusher discharge onto the primary jaw crusher discharge conveyor. The primary jaw crusher discharge conveyor will feed the stockpile feed conveyor. The jaw crusher will have opening dimensions of 1,600 x 1,200 mm with installed motor power of 250 kW.

Fresh water will be used to mist the ROM bin, the vibrating grizzly and the jaw crusher to lower dust emissions generated at these locations.

17.3.2 Crushed Ore Stockpile

The stockpile feed conveyor will feed the crushed ore stockpile. The crushed ore stockpile will be uncovered. There will be two reclaim apron feeders located in the concrete reclaim tunnel under the stockpile. The apron feeders will transfer ore to the SAG mill feed conveyor with each feeder capable of providing total throughput to the plant when required.

The crushed ore stockpile will have 6,600 t live capacity that can support milling operations for 12 hours when the crushing plant is not operating.

17.3.3 Grinding

The SAG mill feed conveyor will transfer ore to the SAG mill from the crushed ore stockpile. An automated SAG mill media addition system will meter the required steel charge to the SAG mill feed conveyor. The transfer size produced from the SAG mill will be approximately 1000 μm .

The SAG mill will be 8.5 m in diameter by 4.0 m (effective grinding length) driven with installed 5,500 kW variable speed motor. The SAG mill discharge will pass through to a SAG mill trommel screen attached to the end of the SAG mill with trommel undersize going directly into the SAG/ball mill discharge pumpbox. Trommel oversize will be recirculated back into the SAG mill using a trommel water-jet system.

From the SAG/ball mill pumpbox the cyclone feed pump will pump a combined slurry into the ball mill cyclone cluster which will classify the feed slurry into coarse/ underflow and fine/ overflow fractions. The designed recirculation rate through this classification system is 300%. The cyclone overflow will have particle size P80 of 125 μm and will flow via gravity to the rougher flotation conditioning tank prior to sulphide flotation. The underflow will feed the ball mill for further grinding with a portion of the coarse fraction feeding the gravity separation circuit for coarse gold recovery. Tailings from the gravity separation will also report to the ball mill. The cyclone cluster will have a total of seven cyclones; five operating, two standby.

Copper sulphate will be added to the SAG/ball mill pumpbox. Milk of lime can be added to the SAG mill to adjust pH if required.

The ball mill will be 6.4 m in diameter by 9.75 m (effective grinding length) driven with installed 7,500 kW fixed speed motor. Slurry will overflow from the ball mill to a trommel screen, attached to the ball mill discharge end. Trommel undersize will discharge into the SAG/ ball mill discharge pumpbox.

17.3.4 Gravity Gold Recovery

The gravity recovery and intensive leach circuit will consist of a single centrifugal gravity concentrator unit equipped with a feed trash screen, concentrate storage tank, and an intensive cyanidation unit. The circuit will be designed to treat 25% of cyclone underflow. The gravity gold recovery unit will be located in a secured area within the mill structure.

Cyclone underflow will be screened using a linear vibrating screen removing 3 mm+ material. The oversized material will overflow directly into the ball mill. The undersize product will feed a 1.2 m gravity concentrator. Gravity concentrator tailings will discharge directly into the ball mill.

Periodically, the centrifugal concentrator will be bypassed and switched to flushing mode using fresh water to recover the collected concentrate. The collected concentrate will be stored in the gravity concentrate stock tank prior to being pumped to the intensive leach cyanidation unit (ICU).

The gravity concentrate will be batch processed in the intensive cyanidation unit in 24 hour intervals. The gravity concentrate will be leached to dissolve gold in a leach solution that includes sodium cyanide, caustic solution, and a leach accelerant. After the leach cycle is complete, the pregnant solution will be pumped to the electrowinning circuit while the intensive cyanidation unit residue will be pumped to the pre-leach thickener.

17.3.5 Flotation

The ball mill cyclone overflow will flow by gravity into the rougher flotation conditioning tank. The rougher flotation conditioning tank will provide 5 minutes conditioning time for flotation chemicals including copper sulphate (activator) and sodium iso-butyl xanthate (SIBX) (collector). Lime can be added into the tank to adjust pH if required.

The rougher and scavenger flotation circuit consists of a single bank of eight (8) 200 m³ mechanical tank-cells; four (4) cells for the rougher circuit and four (4) cells for the scavenger circuit. The conditioning tank overflow, scavenger concentrate and cleaner tailings will feed the rougher flotation circuit which will have an installed residence time of 22 minutes. The target design residence time based on 6.5 minutes laboratory testwork and a scale-up factor of 2.5 is 16.3 minutes. The rougher concentrate will flow by gravity to the cleaner flotation circuit for further cleaning.

The scavenger flotation circuit will process rougher tailings only and with an installed residence time of 24 minutes. The target design residence time based on 6.5 minutes laboratory testwork and a scale-up factor of 2.5 is 16.3 minutes. The combined rougher-scavenger flotation design retention time is 32.6 minutes. The scavenger concentrate will flow into the cleaner tailings tank and will be pumped back to the rougher feed box. The scavenger flotation tailings will be pumped to the flotation tailings storage facility.

The cleaner flotation will be a single bank of four (4) 30 m³ tank-cells providing an installed residence time of 22 minutes; the target design residence time from testwork is 21 minutes. Cleaner flotation concentrate will be pumped to the pre-leach thickener. Cleaner flotation tailings will be pumped back to the rougher flotation feedbox.

17.3.6 Pre-Leach Thickening

The cleaner flotation concentrate will be screened by a vibrating trash removal screen before entering the thickener. Flocculant will be added to the thickener feed to promote the settling of solids. The pre-leach thickener will have a diameter of 18 m and produce a thickened product of 60% solids which will be pumped to the leaching circuit. Thickener overflow water will be pumped to the process water tank.

17.3.7 Pre-aeration and Leaching

The pre-leach thickener underflow will be pumped to an 8.0 m diameter by 9.0 m height pre-aeration tank prior to being leached in five 8.0 m diameter by 9.0 m height leach tanks. The pre-aeration tank will oxidize some sulphide material to reduce cyanide consumption and improve gold recoveries using oxygen. The pre-aeration tank will be designed to provide 8 hours total retention time. Cyanide

will be added and maintained in the leach circuit for gold dissolution which will increase gold concentration in solution prior to contact and adsorption with activated carbon in the CIP circuit. The leach circuit will be designed to provide 40 hours total retention time.

The circuit will be operated as a single train. The first tank will be utilized as a pre-aeration tank. If the pre-aeration tank requires maintenance, the first leach tank will be used as the pre-aeration tank while bypassing the pre-aeration tank. Oxygen will be sparged from the bottom of the leach tanks at an aeration rate of 0.05-0.07 N m³/h/m³. Process air can be added to the pre-aeration and leach tanks if desired.

17.3.8 Carbon-in-Pulp (CIP) Circuit

The carbon-in-pulp (CIP) Carousel circuit will consist of 10 CIP tanks and will provide a total slurry retention time of 18 hours. The CIP circuit will be a carousel configuration using a distribution launder to distribute leached slurry to any of the CIP tanks. Each CIP tank will be 4.75 m diameter by 7.20 m height.

There will be a carbon inventory in each CIP tank. In CIP carousel configuration, carbon is not pumped counter current to slurry flow from tank to tank. In this configuration, as CIP tank 1 carbon is loaded, the carbon will be pumped to the ADR plant and newly regenerated carbon will be added to CIP tank 1. CIP tank 1 will then become the "tailings" (last in the circuit) tank and CIP tank 2 will become the new head tank.

This will occur on a daily basis and will continue through the tanks with CIP tank 1 eventually becoming the head tank once again. Each CIP tank will have a single inter-stage screen/ agitator which will retain carbon particles in the tank while still allowing slurry to overflow into the next tank. Carbon concentration will be 40 g/L in each of the CIP tanks. CIP tailings will feed a vibrating carbon safety screen which will capture any carbon that may have escaped the CIP tailings tank.

Solution gold values will decrease as slurry flows through the CIP circuit. Carbon will leave the first CIP tank once the carbon is loaded. A common loaded carbon pump will pump loaded carbon to the loaded carbon screen. The carbon will then be transferred to the acid wash tank on a daily basis.

17.3.9 Cyanide Recovery and Detoxification

The CIP tailings will be screened by a vibrating carbon safety screen before entering the thickener. Flocculant will be added to the thickener feed to promote the settling of solids. The cyanide recovery thickener will have a diameter of 18 m and produce a thickened product of 50% solids which will be pumped to the cyanide detoxification circuit. Thickener overflow water will be recycled back to the leach and the CIP circuit which will recycle cyanide into those circuits.

The cyanide destruction circuit will consist of two 4.7 m diameter by 6.0 m height mechanically agitated tanks, each with a live capacity of 87 m³ and 90 minutes retention time; 180 minutes retention time total. The SO₂/ Air process for cyanide destruction will be used for this application. The treated slurry will flow by gravity to the cyanide destruction tailings pumpbox for pumping to the CIP tailings pond.

The cyanide detoxification circuit will treat thickened slurry from the tailings thickener, process spills from various contained areas and process bleed streams: cold cyanide barren solution effluent, acid wash effluent, CIP tailings pond secondary treatment effluent and area sump pump discharge.

Oxygen will be sparged into the cyanide destruction tanks. Hydrated lime will be added to maintain the optimum pH 8.5 and copper sulphate (CuSO_4) will be added as a catalyst. Sodium metabisulphite (SMBS) will be dosed into the system as a solution as the source of SO_2 . The process will reduce total cyanide in solution from 998 mg/L to 0.46 mg/L and WAD cyanide in solution from 674 mg/L to 0.34 mg/L. The total cyanide and WAD cyanide level in solutions will eventually drop below 0.2 mg/L after the aging process in the CIP pond.

17.3.10 Carbon Acid Wash, Elution, and Regeneration

17.3.10.1 Acid Wash Column

Loaded carbon from the CIP circuit will be added into the loaded carbon tank for storage before discharging into the acid wash column where it will be treated with hydrochloric acid to remove inorganic foulants such as calcium, magnesium, sodium salts, and silica.

The carbon will first be rinsed with fresh water. Acid will then be pumped from the acid wash circulation tank to the acid wash vessel. Acid will be pumped upward through the acid wash vessel and overflow back to the acid wash circulation tank. The carbon will then be rinsed with fresh water to remove the acid and any mineral impurities.

Fresh acid will be pumped from drums into the acid wash tank when required. When mineral impurities build up, the acid wash tank will pump acid to the acid neutralization tank where caustic can be added to neutralize the solution before being pumped to the cyanide destruction tank.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into the elution vessel using recycled carbon transfer water. Carbon slurry will discharge directly into the top of the elution vessel.

17.3.10.2 Elution Column

The carbon stripping (elution) cycle will utilize barren solution to strip gold rich carbon to create a pregnant solution.

The strip column will hold 4.0 t of carbon and strip once a day. During the strip cycle, solution containing approximately 0.3 % sodium hydroxide and 0.2 % sodium cyanide, at a temperature of 135°C and 650 kPa will be circulated through the strip vessel. Solution exiting the top of the elution vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold solution, prior to the cold solution passing through the solution heater. The heated barren solution will then be heated again through the primary heat exchanger using heated water to bring the solution to its final temperature.

The hot barren solution will then be pumped into the elution column through the carbon bed and recirculated multiple times creating a pregnant solution. A barren solution tank will store barren solution and a pregnant solution tank will store pregnant solution.

The elution column can also be used as a cold strip circuit to remove copper from carbon if copper levels are too high.

17.3.10.3 Carbon Regeneration Kiln

Once stripped of gold, a recessed impeller pump will transfer the carbon from the elution vessel to the kiln feed dewatering screen. The kiln feed screen acts as both a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. A 167 kg/h diesel fired kiln will be utilized to treat 4.0 t of carbon per day, equivalent to 100% regeneration of carbon. The regeneration kiln discharge will be transferred to the carbon quench tank by gravity, cooled by fresh water and/or carbon fines water, and stored in the regenerated sized carbon tank prior to being pumped back into the CIP circuit.

A kiln scrubbing system will be installed on the carbon regeneration kiln to treat off-gas. A mister and carbon bed will capture mercury prior to discharging; resulting in cleaner off-gas. The mercury will be captured in the mister system and stored into glass vials prior to proper disposal.

To compensate for carbon losses by attrition, virgin carbon is added to the carbon attrition tank along with fresh water to mix and activate the carbon. The fresh carbon will then drain into the regenerated carbon tank.

17.3.11 Electrowinning

The pregnant solution generated from the elution column will be pumped to two electrowinning cells from the pregnant solution tank. These cells will operate on a single-pass basis to produce a gold sludge. The barren solution will be collected in the barren solution pumpbox where it will be pumped to the barren solution tank.

The primary flow from the barren solution pumpbox returns the solution to the elution circuit where it will be reused as barren stripping solution for the elution column.

The pregnant solution generated by the intensive cyanidation unit will be pumped to a separate ICU pregnant solution tank. Pregnant solution will then be pumped into a dedicated electrowinning cell for ICU solution. Pregnant solution will be recirculated through the dedicated electrowinning cell until all gold is deposited onto the electrowinning cathodes.

The electrowinning cathodes will be manually transferred from the electrowinning cells to the cathode washing tank where a high pressure washer will be used to dislodge gold sludge from the cathode surface. The sludge will be filtered by a filter press. The resulting filter cake will be dried in a drying oven and the resulting filtrate will be pumped back to the barren solution pumpbox within the refinery.

17.3.12 Doré Production

The dried filter cake will then be transferred manually into the electric smelting furnace with flux materials where it will be batch smelted into gold doré bars and stored in a secure vault.

A mercury retort system will be installed to capture any mercury from the off-gas generated in the drying oven, the furnace oven, and the electrowinning cells through the use of a mister system and carbon bed. The mercury will be captured in the mister system and stored into large glass vials. The resulting clean off-gas will join with the carbon regeneration kiln clean off-gas.

The mercury retort system will be installed to handle any possible mercury remaining in the garimpeiro tailings.

17.3.13 Tailings Ponds

For the total life of mine (LOM), there will be three tailings ponds: the flotation tailings pond, CIP tailings Pond #1 (for the first half of the LOM), and the future CIP tailings Pond #2 (for the second half of the LOM).

The flotation tailings pond will receive tailings from the flotation circuit as well as the mine dewatering flow. Tailings pond supernatant (reclaim water) will be pumped back to the processing water tank using vertical pumps on a barge.

The CIP tailings pond will receive tailings from the cyanide destruction circuit. The CIP tailings pond will be lined and monitored to prevent any effluent from entering the environment. The CIP tailings will provide natural cyanide degradation to further decrease cyanide to <0.2 ppm prior to the release of any effluent into the environment.

Each CIP tailings pond will be designed to hold settled CIP tailings for each half of LOM by reclaim supernatant for recycled process water and by releasing excess detoxed water flow into the environment (with further treatment). The first CIP tailings pond will have in excess of 440 days storage capacity when accounting for both CIP tailings deposition and precipitation/ evaporation prior to any requirements for effluent discharge treatment. A second CIP tailings pond will be used in the future when the first pond is filled.

A water treatment plant will be built and in operation after one year of processing plant operations to treat any excess effluent water prior to discharge.

17.3.14 Water Treatment Plant

Excess water from the CIP tailings pond will be pumped to the water treatment plant which will be located at the processing facility. The water treatment plant will decrease aqueous copper and sulphate levels prior to discharging clean water into the environment. The water treatment plant will be designed for up to 35 m³/hr flow. The nominal flow is 28 m³/hr.

Water will be pumped into a series of agitated mix tanks. Milk of lime slurry and recycled sludge will be added to lime sludge/ mix tank to maintain pH at approximately 9.5. The lime sludge slurry will overflow into the two lime reactor tanks (in series) to allow for additional retention time for reaction.

This will initiate the reaction for aqueous copper and sulphate to precipitate into solid copper hydroxide and calcium sulphate (gypsum), respectively. The slurry from the lime reactor will overflow into a clarifier for solids/liquid separation.

Flocculant will be added to the clarifier feed (in line) to promote settling of precipitated hydroxide solids and gypsum in a 7.5 m diameter clarifier. The clarifier will settle sludge and a part of the sludge will be recycled to the lime sludge/ mix tank. Periodically, sludge will be removed and pumped to the CIP tailings pond. Sulphuric acid will be added to the clarifier overflow in an agitated tank to decrease and maintain the pH at 6-7 to reduce un-ionized ammonia levels to discharge limits. The clarifier overflow will be discharged into the environment.

A flocculant system will be installed to service the water treatment plant. Lime slurry from the processing facility will be pumped to the water treatment plant. Sulphuric acid will be dosed from totes held at the water treatment plant.

17.3.15 Sampling

Several samplers will be provided throughout the plant to generate composite shift samples from key process streams. Two types of sampling will be performed; metallurgical and process control sampling.

Metallurgical samplers will be used to generate shift composite samples that will be assayed for plant metallurgical accounting. The following process streams will be equipped with metallurgical samplers:

- Primary cyclone overflow
- Flotation tailings
- CIP Tailings

The metallurgical samplers will sample feed and tailings product which will allow an accurate metal balance of the plant to be completed.

Process control sampler will generate samples used to monitor unit processes in the plant. The process control samplers will be used to generate shift composite samples on process streams that will provide plant operation performance data.

The following process streams will be equipped with process control samplers:

- Leach feed
- Leach tailings
- Final tailings
- Pregnant solution to electrowinning
- Barren solution after electrowinning

All samplers will produce 5-10 L of slurry that can be transported to the assay laboratory for further analysis.

17.4 REAGENTS

17.4.1 Hydrated Lime

Hydrated lime will be used as a pH modifier and will be supplied in dry form by road in bulk tankers and off-loaded into the storage silo using a blower. The storage silo will hold up to 7 days consumption. Hydrated lime will be added into a mix tank to prepare a milk of lime slurry before addition into the processing facility.

17.4.2 Sodium Cyanide

Sodium cyanide will be used as a gold lixiviant. The cyanide will be shipped in briquette form by road to site in one tonne bulk bags inside wooden boxes and stored in the cyanide mixing facility; separate from the reagent storage and mixing facility. The sodium cyanide will be mixed with fresh water to form a cyanide solution for use in the leaching circuit.

17.4.3 Copper Sulphate

Copper sulphate (CuSO_4) will be an activator for sulphide flotation and will also be used as a catalyst for cyanide destruction. The copper sulphate will be supplied as a dry flake in one tonne bulk bags and stored in the reagents storage area adjacent to the reagents mixing facility. The copper sulphate will be mixed with fresh water to form a copper sulphate solution ready for use in the processing facility.

17.4.4 Sodium Metabisulphite

Sodium metabisulphite ($\text{Na}_2\text{S}_2\text{O}_5$), also known as SMBS, will be the source for SO_2 in the Air/ SO_2 cyanide destruction process and will be supplied as dry briquettes in 1 tonne bulk bags. SMBS will be stored in the reagent storage area where it will be transferred to the mixing facility to produce a SMBS solution prior to use in the cyanide destruction process.

17.4.5 Sodium Iso-Butyl Xanthate

Sodium iso-butyl xanthate (SIBX) will be used as a sulphide mineral collector in the flotation circuit and will be supplied in 850 kg bulk bags as a dry reagent. SIBX will be shipped by road to site, offloaded by forklift and stored in the reagents storage area adjacent to the SIBX & frother mixing facility. The SIBX will be mixed with fresh water to form a SIBX solution prior to addition to the processing facility.

17.4.6 Sodium Hydroxide

Sodium hydroxide (NaOH), also known as caustic soda, will be used as a pH modifier and will be supplied as briquettes in one tonne bulk bags. Caustic soda will be mixed with fresh water prior to being used in the gold elution circuit.

17.4.7 Hydrochloric Acid

Hydrochloric acid (HCl) will be used to remove inorganic carbonates from carbon in the acid wash process within the ADR plant. They will be supplied in drums and stored in reagent storage area adjacent to the reagent mixing facility.

17.4.8 Flocculant

Flocculant is a liquid polymer used in both thickeners to settle solids. It will be supplied in 750 kg bulk bags as a dry reagent. Flocculant will be shipped by road to site, offloaded by forklift, and stored in the reagents storage area adjacent to the Reagents Mixing facility. Flocculant will be diluted using fresh water and further diluted using an inline mixer with process water prior to being added into the processing facility.

A separate flocculant system will be installed at the water treatment plant for use in the clarifier.

17.4.9 Frother

DF250 will be used as a frother to mechanically sustain bubbles in the flotation cells. The frother will be supplied in drums, offloaded by forklift, and stored in the reagents storage area adjacent to the SIBX & Frother mixing facility. Drums of frother will be unloaded into the frother storage tank by drum pump prior to being added into the flotation circuit.

17.4.10 Sulphuric Acid

Sulphuric acid (H_2SO_4) will be used as a pH modifier in the CN destruction circuit and the water treatment plant. It will be supplied by road in drums and offloaded by forklift. The drums will be stored in the reagents storage area of the warehouse facility and delivered by forklift to the detoxification and WTP areas where it is dosed to the circuit by drum pump.

17.5 PLANT SERVICES

17.5.1 Blower Air

The flotation blowers will supply low pressure process air to the flotation cells. There will be three (3) blowers (2 duty, 1 standby) installed to meet flotation air requirements.

17.5.2 Plant & Instrumentation Air

Three (3) plant air compressors will produce air for the processing facility. Plant air receivers will act as a buffer storing air to account for variations in demand prior to being distributed throughout the processing plant including the oxygen generation plant. Instrument air will be dried before being stored in the instrument air receivers and distributed throughout the plant.

The air compressors can bypass the oxygen generation circuit and feed the leaching circuit directly, if required.

17.5.3 Oxygen Generation

An oxygen generation plant will be used to provide industrial grade oxygen for the pre-aeration, leaching, and cyanide destruction circuit. The plant air compressors will supply air to the oxygen generation circuit. The oxygen generation plant will include an oxygen plant air drier, a pressure swing adsorption (PSA) oxygen generator, and an oxygen plant receiver. The oxygen generation system will produce up to 5 tonnes of 90% purity oxygen per day.

17.5.4 Fresh and Fire Water

Fresh water will be sourced under permit from Veados Creek. Fresh water from the creek will be pumped to the plant fresh/fire water tank by vertical turbine pumps. The tank will be located on a hill 60 m above the plant site and will use gravity to supply fresh water to all required users.

The plant fresh/fire water tank will serve as a combined storage for both fresh and fire water supply. Fresh water will draw from part way up the tank while the lower section of the tank is held in reserve for a dedicated fire water supply.

The fire water portion of the tank will have minimum capacity of 108 m³ and will feed the plant and permanent camp fire suppression systems fire hydrants and hose reels via a fire water ring main. Fresh water in the tank will be used to supply the following services:

- Primary crushing circuit dust suppression water
- Reagent preparation water
- Slurry pumps gland seal water
- Cooling water systems; i.e., elution circuit, mill motor cooling
- High pressure wash water in the refinery
- Make-up water for the process water system

Fresh water will be pumped to the fresh/fire water tank through multimedia filter to remove particulate matter that can be detrimental to users.

17.5.5 Potable Water

Raw water to feed the potable water system is supplied from wells using vertical well pumps. The raw water will be treated in a vendor-supplied potable water plant to produce potable water for the process plant and camp facilities distribution. The potable water will be used in the process plant for safety showers and washrooms.

17.5.6 Gland Water

Water for the gland water system will be supplied by fresh water from the fresh/fire water tank and cooling water returning from the elution circuit cooling heat exchanger. The gland seal water tank will store and distribute gland water to the plant with gland seal water pumps in a duty-standby configuration.

To prevent particulate matter from causing damaged gland seals throughout the plant, the water feeding the gland water tank will pass through 25 micron particulate filters.

17.5.7 Process Water

Process water will be comprised of pre-leach thickener overflow, flotation tailings pond reclaim water, and fresh water top-up when required. Process water will be stored in the process water storage tank and distributed by the process water pumps, in a duty – standby configuration, to non-cyanide consumption points in grinding, flotation, and CN detoxification.

Process water with cyanide from the cyanide recovery thickener will be recycled back to the leach and CIP circuits for density control and to recycle cyanide.

17.6 RISKS AND OPPORTUNITIES

Opportunities and risks for the recovery methods were identified and categorized:

Opportunities:

- There is a potential to reduce overall costs by optimizing the current leach and carbon adsorption configuration; either changing to CIL or leach/ traditional CIP. Carbon loadings and subsequent changes to the ADR plant can also be optimized.
- There is an opportunity to reduce current rougher and cleaner flotation circuit retention time which will reduce overall equipment costs and footprint.
- There is a potential opportunity to eliminate or reduce the water treatment plant for CIP supernatant once additional water treatment testwork is completed.

Risks:

- Hydrojet system for SAG recycle could be difficult to operate properly.
- The SAG mill design was based from four (4) samples. The ball mill circuit design was based from seven (7) samples. Although conservatism was applied to the comminution circuit design, there is a risk of not meeting design throughput if the ore body is significantly harder than the samples tested.
- Water treatment (of CIP pond reclaim water) is based on limited test work results, i.e.: single detoxification residue sample.

Recommendations:

- Complete a high level trade-off between the different leaching and carbon adsorption configurations and design criteria to determine the most economically feasible option for future investigation. Currently, there is a combined 58.8 hours leaching and CIP retention time which is significantly longer when compared to testwork results.
- Reduce the rougher and scavenger flotation circuit retention times to reduce overall costs. The rougher/ scavenger flotation requires a combined 32.6 minutes retention time determined from laboratory testwork and a scale-up factor of 2.5. The current design has an

installed retention time of 45.3 minutes which exceeds the required retention time by 12.7 minutes. Decreasing from the current 200 m³ cells to 160 m³ cells will reduce overall costs and footprint while still achieving >36 minute installed retention time.

- Consider replacing the hydrojet system with a convention pebble conveyor recycle system for the SAG mill.
- Perform more grindability tests on additional samples.
- Refer to Section 13 for recommendations related to the CIP tails water treatment plant.

SECTION • 18 PROJECT INFRASTRUCTURE

The layout of the Project complete with the onsite infrastructure is presented on Figure 18-1.



Figure 18-1: Site Layout

18.1 ROADS AND DRAINAGE

Currently there are existing roads and trail network that are navigable by either light vehicle or quadricycle.

A network of gravel surfaced roads for light and heavy vehicles will be built. The heavy haul roads around the mine and construction quarry will provide large truck access to the truck shops, crusher, aggregate plant and tailing dam areas. All other areas of the site will be accessible on light vehicle roads.

Ditching and culverts will divert water away from plant site infrastructure, process plant and roads. Plant site pads are sloped away from buildings towards diversion ditching. Surface water from areas

around the truck facilities will be collected and diverted to the oil/water separator located beside the sewage plant.

18.2 MINE

18.2.1.1 Open Pit Mine

The open pit mine and waste rock dumps are outlined in Section 16. There will be two stockpiles for high grade saprolite and garimpeiro tailings. A pit dewatering system will be installed to pump pit water to the flotation tailings pond.

18.2.1.2 Explosives Facility

The explosives facilities will be located on an abandoned air strip which will be widened and repurposed for placement of the explosive facilities. The area will be secured with fencing and access will be controlled by security. It will include storage containers for nitrate, emulsion and accessories. The emulsion plant will be in a fabric building.

18.3 PROCESS PLANT

The process plant details are outlined in Section 17.

18.3.1.1 Reagent Storage

Two fabric buildings will be in the process plant area to house and prepare reagents. One will be for cyanide while the other will house flocculant, caustic, SMBS, SIBX and copper sulphate.

18.3.1.2 Process Control

The process plant will be controlled with a DCS system centered around the single control room located adjacent to the grinding area.

18.4 TAILINGS DISPOSAL SYSTEM

The Tocantinzinho Project tailings disposal system will consist of a flotation tailings dam and two leaching effluent ponds (also referred to as CIP ponds). The dam will have two purposes, tailings disposal and water catchment. The eastern CIP pond will be built midway through the mine life.

These designs were elaborated according to the criteria established by Brazilian Standards NBR 13.028: 2006 and NBR 10.157: 1987, whenever applicable). The design also complies with the most recent version of the Brazilian Standards for tailings dam design (NBR 13.028 - ABNT, 2017). The tailings classification followed the criteria specified by Brazilian Standard NBR 10.004: 2004.

18.4.1 Flotation Tailings Dam

18.4.1.1 Design Description

The starter dam design (crest El. 151 m) is an earth dam formed by compacted clayey soil, with internal drainage composed of a vertical filter and a horizontal sand/gravel blanket. The final dam will have a mixed section, composed of an upstream clayey core and a downstream face of compacted rockfill (mine waste), with crest at El. 165.3 m.

The reservoir capacity for the starter dam is 7.68 Mm³, which will meet the first three years of mine production. The final dam can hold 29.8 Mm³, comprising eleven years of useful life, meeting the demand for the entire mine predicted operation.

It will be also necessary to build up a saddle dyke for the final dam, composed of a compacted earth fill, with crest elevation at El. 165.3 m, length of 175 m and maximum height of 11 m.

The starter dam will have a 20 m crest width and upstream and downstream slopes of 1 V: 2 H and 1 V: 2.5 H, respectively. The final dam, largely composed of compacted rockfill, will have a downstream slope of 1 V: 2 H, the same one of upstream slope. The downstream slope in both phases will have berms 5 m wide at every 10 m in height.

Table 18-1 shows the main features for the starter and final dam design.

Table 18-1: Flotation Tailings Dam features

Features	Unit	Starter Dam	Final Dam
Crest elevation	m	151.00	165.30
Crest length	m	578.00	884.50
Crest width	m	20.00	15.00
Dam height	m	29.50	44.05
Reservoir elevation (maximum normal water level)	m	148.00	163.00
Reservoir elevation (<i>maximum maximorum</i> water level)	m	149.44	164.17
Dam volume – compacted soil	m ³	621,474.00	102,417.00
Dam volume – rockfill and internal drainage	m ³	48,879.00	715,774.00
Reservation volume	Mm ³	7.68	29.80
Catchment area	km ²	2,84	2,84
Concentration time	min	52.00 (third year)	77.00
Return period	years	10,000.00	10,000.00
Maximum inflow	m ³ /s	7.60	10.90
Maximum outflow	m ³ /s	6.50	8.00

Figure 18-2 shows a typical section for the final dam.

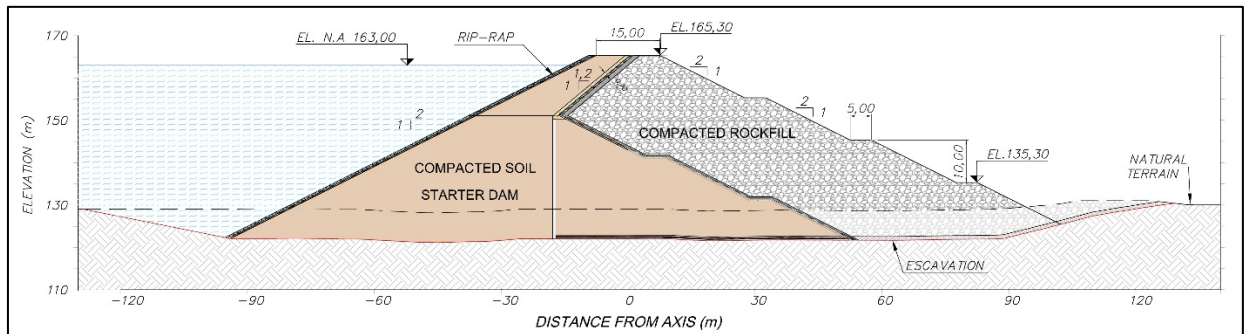


Figure 18-2: Typical section for the final Flotation Tailings Dam

The dam foundation is basically composed of residual soils from granite alteration. The existing “curimã” (“garimpo” sediments) will be totally removed. Foundation treatment for the final dam will take place during the construction of the starter dam.

Positioned on the left shoulder, the spillway system is of free sill. For the starter phase it will be composed by a rockfill inlet channel, followed by a low declivity concrete filled geocell channel that drains into a stepped channel, made with reinforced concrete and into a stilling basin. For the final dam, a new emissary channel will be deployed in shotcrete and both the stepped waterfall and the stilling basin will be kept.

18.4.1.2 Design Criteria and Assumptions

The following shows the criteria and assumptions used in the design:

- Design according to Brazilian Standard NBR 13.028 (ABNT, 2006). It also complies with the most recent version (NBR 13.028 - ABNT, 2017).
- Topographic base provided by Brazauro (file no. A00-10-1001-D.dwg and F00-26-001_A.dwg).
- Tailing Class II B - non-hazardous and inert.

18.4.1.3 Geological-Geotechnical Studies

The following outlines the studies done to understand the existing conditions:

- Geological-geotechnical mapping with identification and characterization of the main existing lithotypes at the foundation, the borrow areas for the compacted soil and for the internal and superficial drainage systems (aggregate).
- 22 SPT borings, 8 diamond cores drillings with SPT, 5 diamond core drilling and 5 sampling wells, where undisturbed samples were collected, being 4 at the main dam foundation and 1 at the saddle dyke area.
- 34 auger borings for borrow areas characterization purpose.

- Electrical resistivity Geophysical investigations (3,591.20 m in 8 sections) and electromagnetic (ground penetrating radar - 3,591.20 m in 8 sections) to determine the unconsolidated material thicknesses, at the thalweg area.
- 60 soil infiltration tests and 11 rock water loss tests.
- Laboratory tests for physical and special characterization to determine the foundation and fill material properties. These lab tests consisted of grain size determination, Atterberg Limits, moisture content, one-dimensional consolidation, variable load permeability and triaxial compression tests. Besides those tests, some geotechnical tests were carried out for flotation tailings characterization, such as grain size determination, specific weight, one-dimensional consolidation and hydraulic consolidation test (HCT).
- Stability analyzes were performed to verify the geometry for the starter Dam (according to Brazilian standard NBR 13.028), by using the Slide 6.0 software (ROSCIENCE, 2010).
- Evaluation of circular failures using the Spencer Limit Equilibrium method.
- The material strength parameters were obtained either from lab test results or estimated according to TEC3 expertise. The adopted parameters are shown in Table 18-2.

Table 18-2: Geotechnical Parameters considered for the Starter Dam Stability Analysis

Material		Specific weight kN/m ³	Effective Cohesion kPa	Internal friction Angle °	Source
Dam	Compacted Soil	18.5	17	28	Laboratory
	Filters and Transition	20	0	32	Estimated
Foundation	Mature residual soil	15	14	21	Laboratory
	Fresh Residual Soil	17	19	27	Estimated
	Rock Foundation	20	500	35	Estimated
	Rockfill	20	0	43	Estimated
Alluvium		18	0	30	Estimated

- The stability analyzes results showed Table 18-3 that all evaluated sections have safety factors above 1.5 for normal operation and above 1.3 for critical water level, what meets Brazilian Standard (NBR 13.028: 2006) .
- A monitoring system to record water level and pore pressure development is planned to verify dam performance. Visual inspections of the dam, at least twice a month, is also suggested.
- Tailings geotechnical properties were obtained from tests performed in two flotation tailings samples, obtained from the project pilot plant test. These tests were complete granulometry (sieving and sedimentation); specific weight; one-dimensional consolidation and HCT (hydraulic consolidation test).
- The GSD tests showed that the tailing has a fine sand predominance with a more than 70 % passing through the # 200 sieve.
- At the consolidation tailings tests, differences were observed from the results of the initial void ratios. It is highly recommended to perform new tests during operation, with samples

collected in the reservoir thus enabling to accurately estimate the consolidation behavior over the operations years.

Table 18-3: Results of the Tailings Dam Stability Analyses

Section	Load condition	Slope	Groundwater	Minimum Safety Factor required	Safety Factor Obtained
Stake 12– stretch 1 (right shoulder)	Final construction	Downstream	Dry (ru=0.15)	1.3	1.92
		Upstream			1.55
	Normal operation regime	Downstream	Normal	1.5	1.79
	Critical operation regime		Critical	1.3	1.33
Stake 19+15.6 - stretch1 (central)	Final construction	Downstream	Dry (ru=0.15)	1.3	1.98
		Upstream			1.37
	Normal operation regime	Downstream	Normal	1.5	1.97
	Critical operation regime		Critical	1.3	1.54
Stake 24 – stretch 1 (left shoulder)	Final construction	Downstream	Dry (ru=0.15)	1.3	1.75
		Upstream			2.55
	Normal operation regime	Downstream	Normal	1.5	1.56
	Critical operation regime		Critical	1.3	1.31
Stake 25 – stretch 2 (right shoulder)	Final construction	Downstream	Dry (ru=0.15)	1.3	1.98
		Upstream			2.74
	Normal operation regime	Downstream	Normal	1.5	1.70
	Critical operation regime		Critical	1.3	1.38
Stake 28+2.8 - stretch 2 (central)	Final construction	Downstream	Dry (ru=0.15)	1.3	1.89
		Upstream			2.16
	Normal operation regime	Downstream	Normal	1.5	1.76
	Critical operation regime		Critical	1.3	1.39
Stake 30+14.4 - stretch 2 (left shoulder)	Final construction	Downstream	Dry (ru=0.15)	1.3	2.42
		Upstream			2.17
	Normal operation regime	Downstream	Normal	1.5	2.21
	Critical operation regime		Critical	1.3	1.75

18.4.1.4 Hydrological Studies

These studies were performed to dimension the hydraulic structures and to provide support for the dam reservoir water balance; some assumptions are listed below:

- The considered contribution basin has a drainage area of 2,842 km², a perimeter of 7.1 km and a main river average slope of 0.032 m / m. It has a rounded shape, being characterized by a dense vegetation at most parts.
- The climatological data was obtained from the climatological normals of the Itaituba station, defined by the National Department of Meteorology, with data from the 1961 to 1990. It

- shows that the rainy season occurs between November and April with an average annual precipitation of approximately 2,000 mm.
- Evaporation is more significant between the months of July and January with an average annual evaporation value of 900 mm.
 - Characterization data for the area intense rainfall regime was obtained from HIDROWEB of the National Water Agency (ANA). The rainfall station km 1326 BR-163 (555000) was adopted due to its proximity to the project area.
 - A construction of three river diversion structures is also predicted during the dam building phase. Design of these structures considered a 25-year return period (including the wet season period due to work delay possibilities).
 - The spillway system was designed for floods with a return period of 10,000 years.
 - The results of the hydrological simulations to determine the spillway design floods and cofferdams indicate that the design peak flood flows (and its associated critical periods) are equal to 6.59 m³/s, 6.50 m³/s and 8.00 m³/s for the first year of the diversion channel and the spillway and the end of the third year, respectively.
 - The design predicts the residual (sanitary) flows will be maintained by dedicated pumps (main and reserve), monitored by an hour-meter and a hydrometer.
 - This residual flow is calculated to be around 6 L/s (22 m³/h), determined according to the State of Pará state legislation, corresponding to the 95% residence time on flow duration curves.
 - It is also expected that the internal drainage system will provide an additional contribution of approximately 5.6 m³/h (25.5% of the residual flow).

18.4.1.5 Hydraulic Dimensioning

- The spillway was designed as an emissary channel with a trapezoidal section lined with a concrete filled geocell, having a 2 m short base, a 1.3 m minimum height and slopes 1 H: 1 V, located at El. 148 m (starter dam); longitudinal slope equal to 1.0%, which results in a flow depth equal to 0.79 m and average speed of 3.6 m/s.
- This channel flows to a stepped channel, having a rectangular section of 3 m base and a minimum height of 1 m at the step apex, which are 0.5 m high.
- At the stepped channel, the flow will be turbulent, thus providing energy dissipation. At the end of this section, the flow will reach a water depth of 0.54 m at the step apex, corresponding to an average speed of 4.9 m/s, resulting a 0.46 m freeboard.
- After the stepped channel, the flow energy will be dissipated in a stilling basin composed of a flat section in a rectangular section, with 3 m base and 1.75 m high, having a 6 m length. This basin will be enough to dissipate 88% of the flow kinetic energy, resulting in a water depth equal to 1.39 m high with outlet speed equal to 1.9 m/s. The basin counts with a downstream rip rap protected section to provide a reduction of this flow speed prior to its restitution to the main thalweg.

18.4.1.6 Tailings Disposal Plan

The tailings disposal plan was elaborated during the Basic Design phase when 22 disposal scenarios were performed considering the disposal beginning (Year 1) until the Tocantinzinho Project flotation tailings dam (final geometry) is totally completed in the Year 10.

Tailing beaches were determined from the initially proposed geometry (emerged slope equal to 1% and the submerged slope equal to 3%) and dry apparent density constant and equal to 1.325 t/m³. It is important to evaluate the density during operation, to verify the impacts of lower densities in the disposal plan.

Four disposal points were foreseen to ensure both a better use of the reservoir and a geotechnical safety. The forecast for a starter dam rising will be at the third year of disposal.

18.4.1.7 Hypothetical DAM Failure Evaluation (Dam Break)

Dam break evaluation for both the flotation tailings dam and CIP ponds considered these structures in their final configurations, with reservoirs at full capacity. The simulation has also considered a simultaneous failure of the dam and the two ponds, to evaluate the most critical scenario and the maximum hazard. Failure modes considered overtopping at the dam and piping (internal erosion) at the leaching ponds. The likelihood of the occurrence of such failures is low. Four scenarios were evaluated:

- A (base case): no failure, to simulate the impacts of a natural flood equivalent to 2 years return period at the downstream water courses.
- B: considers the impacts of a failure in the downstream water courses, during the dry season.
- C: no failure, to simulate the impacts of a natural flood equivalent to 10 years return period at the downstream water courses.
- D: rainy day, considering a failure during an extreme event on TSF and CIP, equivalent to 10,000 years return period.

The results of the simulations show the maximum flood levels downstream of the structures, the hydrodynamical risks and the maximum depth of the water level in several sections along the flooded area, in an event of failure.

As there is no permanent human population downstream and the area is composed by native forest, the main impacts are those associated with the environment. However, as there is eventual presence of artisanal miners ("garimpeiros"), they can also be impacted. In scenario D, the backwater effects could also reach the pit and the waste dump.

Figure 18-3 shows the flood plume and water depths for scenario D.

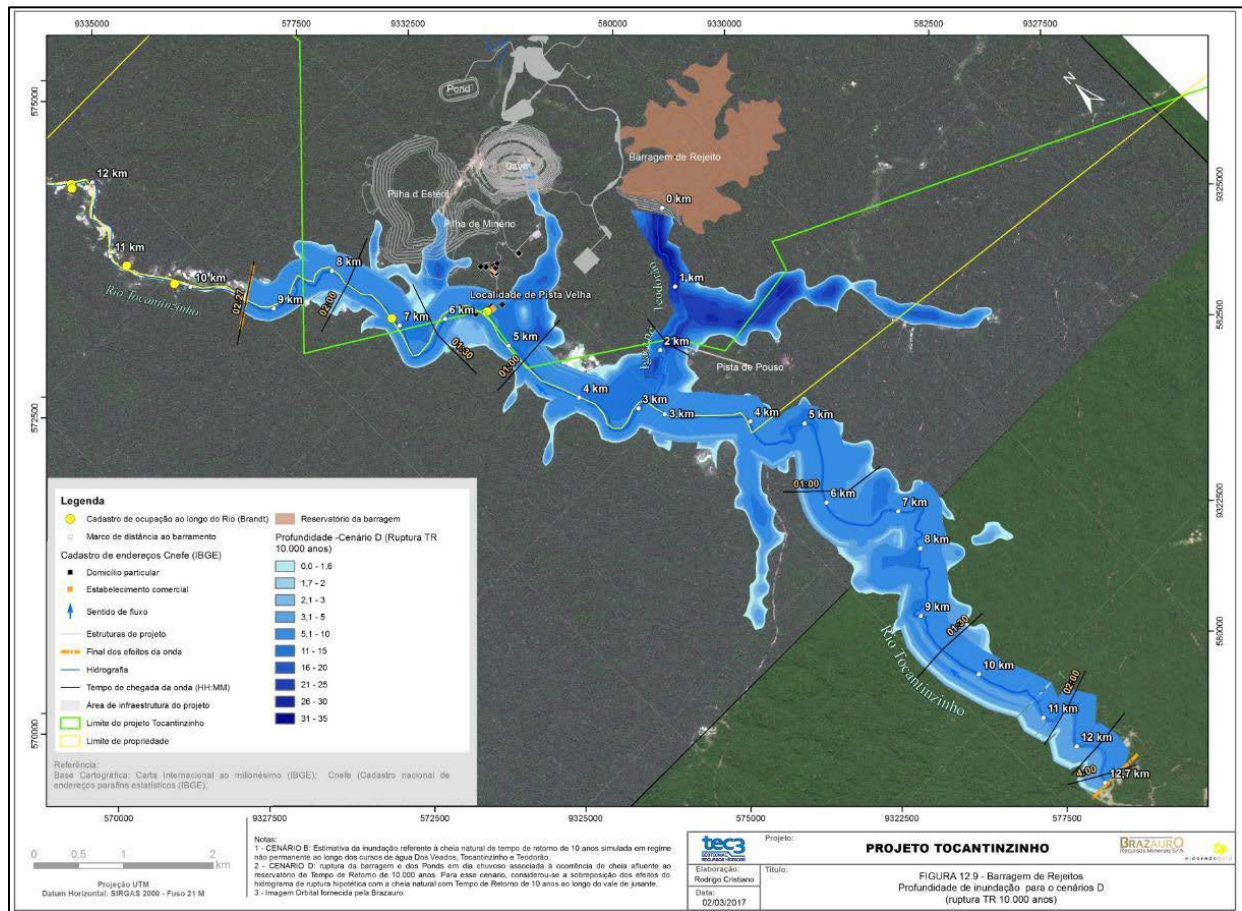


Figure 18-3: Scenario D Impacts

The emergency preparedness plan (PAEBM) will consider the results of the dam break evaluation and all the remedial and preventive measures.

18.4.2 Leaching Effluent Ponds

18.4.2.1 Design Description

The concept considers the ponds excavated in soil down to its design defined levels, and the excavated material used for the compacted perimeter dikes, to minimize both the disposal volume and borrow materials. Each pond is expected to address 5 years of mine operation in sequence, starting with Pond # 1. When Pond # 1 is near capacity, Pond # 2 will be commissioned.

For Pond # 1, a storage volume of 599,104 m³ is foreseen and for Pond #2, 571,328 m³ is predicted. Table 18-4 presents the ponds' main features.

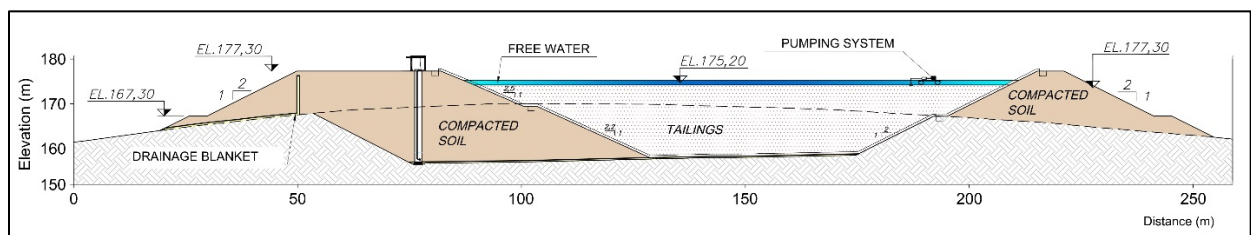
Table 18-4: Leaching Effluent Ponds features

Features	Units	Pond #1	Pond # 2
Crest elevation	m	177.30	170,80
Crest Perimeter	m	1,030.15	1,230.06
Occupation area – reservoir and dam	m ³	95,491.00	86,248
Maximum height – internal part	m	20.30	21.30
Reservoir level (normal maximum water level)	m	175.20	168.40
Reservoir level (<i>maximum maximum</i> water level)	m	175.77	168,60
Total dam volume – compacted soil	m ³	326,777.00	261,599
Storage capacity	m ³	599,104.00 (up to El. 175,20 m)	526,437.00
Catchment area	km ²	0.065	0.061
Design flow return period	years	1,000.00	1,000.00
Design flow	m ³ /h	200 (pumping capacity)	200 (pumping capacity)

The design foresees, to avoid contamination, that the whole internal face and the bottom of the pond will be lined with a layer of HDPE geomembrane (conductive high-density polyethylene), to guarantee impervious ponds.

Underneath the geomembrane, Pond # 1 will have a leak detection system composed of a sand layer connected to a pipeline and a pumping system for percolation exhaustion. This layer will be confined by a double layer of HDPE geomembrane, the top layer being conductive and the bottom layer being textured on both sides. In the contact between the natural terrain and the HDPE geomembrane will be placed a geotextile layer to protect against tears/rips. A similar configuration should be adopted for Pond #2.

Figure 18-4 shows a typical pond section.

**Figure 18-4: Typical Section of Pond # 1**

The leak detection system will be composed, at the bottom of the pond, by a layer of sand with a minimum hydraulic conductivity of 10^{-2} cm/s implanted below the conductive geomembrane to conduct the percolate resulting from eventual leaks to a collecting well, from which the fluids will be pumped. This well will also be used for frequent checking of the efficiency of the waterproofing system efficiency.

Associated with the above-mentioned detection system, each pond will have an internal drainage system installed in the compacted landfill that will also serve as a leak indicator in the upper section (compacted landfill). This system consists of a vertical sand filter connected to gravel drainage trenches and perforated piping directing to the collector boxes arranged at the toe of the pond embankment. This whole system (waterproofing and detection) shall be build up to ensure that the disposed waste remains confined within the ponds.

No overflow system is foreseen in the pond; however, a supernatant water collection system is planned, which has a purpose to remove a portion of this water for treatment destination and subsequent disposal.

18.4.2.2 Design Criteria and assumptions

- Project elaborated according to Brazilian Standards NBR 13.028 (ABNT, 2006) and NBR 10.157 (ABNT, 1987).
- Topographic base provided by Brazauro (file no. A00-10-1001-D.dwg and F00-26-001_A.dwg).
- Minimum 5 years useful life;
- Design elaborated considering that there will be no overflow.
- Waste considered Class I - Hazardous according to NBR 10.004 (ABNT, 2004).

18.4.2.3 Geological-Geotechnical Studies

- For the Pond # 1 design, 20 auger borings, 4 investigation wells, 6 SPT borings and 24 soil infiltration tests were performed.
- For the Pond # 2, 2 SPT borings and 8 soil infiltration tests were performed.
- Physical characterization and special laboratory tests were performed to determine Pond #1 foundation and landfill material properties. The tests performed were complete granulometry (sieving and sedimentation), Atterberg limits, moisture content, one-dimensional densification, triaxial compression and variable load permeability test.
- Stability analyzes were performed to verify the geometry (according to Brazilian Standard NBR 13.028), through Slide 6.0 software (ROCSCIENCE, 2010).
- Circular failures evaluation by the Spencer limit equilibrium method.
- The material strength parameters were defined either from laboratory test results or estimation based on TEC3 expertise. These used parameters are shown in Table 18-5.

Table 18-5: Geotechnical Parameters considered for Pond #1 Stability Analyses

Material		Specific Weight kN/m ³	Effective Cohesion kPa	Internal Friction Angle °	Source
Dam	Compacted Soil	18.5	17	28	Laboratory
	Filters and Transition	20	0	32	Estimated
Foundation	Mature residual soil	15	14	21	Laboratory
	Young Residual Soil	17	19	27	Estimated
	Rock Foundation	20	500	35	Estimated
	Rockfill	20	0	43	Estimated
Alluvium		18	0	30	Estimated

- The stability analyzes results indicated that all evaluated sections had safety factors above 1.5 for normal operation and 1.3 for critical and final water level operation, what complies with Brazilian Standard NBR 13.028: 2006 and 2017.
- The ponds visual and instrument monitoring is planned after construction, aiming for control and recording of the structure performance and it includes waterproofing and leak detection systems operation.

18.4.2.4 Hydrological Studies

The hydrological studies were performed both for designing the hydraulic structures and to provide support for the pond water balance; assumptions are:

- Events associated with a 1,000-year return period to verify the capacity of the recirculated water pumping system with a 1 m minimum freeboard were considered as design events.
- The generation of 1.6 million tons of leaching tailings in 10 years of operation was considered in the water balance, which corresponds to the generation of approximately 18.3 t / h of tailings, pulp form disposal with 34.34% bulk solid content, i.e., 37.5 m³ / h of water in the pond reservoir.
- The resulting load densification imposed by the continuous tailings' disposal on the material previously deposited was not considered in the studies. The material considered dry bulk density was 1.54 t/m³.
- To evaluate the safety of the reservoir against the overtopping, since the pond will not have an emergency spillway, the reservoir hydric balance was elaborated in a simulation model with daily interval, where daily rainfall is obtained from stochastic hydrological modeling, simulating the structures life in 1,000 operating scenarios.
- The results obtained at the end of those simulations considered a pumping system with a capacity of 100 m³/h (operational pump and another reserve, each one with a 100 m³/h capacity) to assure the minimum freeboard; in the most critical simulation (which led to the highest value of water level in the reservoir among 1,000 simulations), resulted in a freeboard of 1.53 m for 1,000 years of recurrence.

18.4.2.5 Hydraulic Dimensioning

The surface drainage system is composed of berm outlets which flows to slope channels that in turn, drain at stilling basins, directing the flows to the natural terrain.

To design the berms drainage, a 100-year recurrence time was used and for slope channels, a 500-year recurrence. Since it is an excavated pond, these reservoirs do not require any diversion structures during construction.

18.5 CAMP ACCOMMODATIONS

18.5.1 Geology camp

There is an existing camp on site that will be upgraded to support the early works construction activities. The current camp consists of individual Weather Haven soft shell buildings that can comfortably accommodate 100 people.

The existing camp will initially be expanded to 144 beds and then ultimately to 592 beds with additional infrastructure including new kitchen, dining hall, laundry and dorms.

18.5.2 Permanent camp

A 1,000-person permanent camp will be located within walking distance of the process plant. The occupancy is based on four people per room during construction. For operations, the hourly rooms will accommodate two people per room reducing the camp capacity to 500 people.

The operations camp complex will include a canteen, kitchen, administration, first aid/ambulance, recreation buildings, three soccer fields, general store, chapel/library/training and potable water plant.

18.6 SITE INFRASTRUCTURE

18.6.1 Electrical Power Distribution

The substation will step the power from the new 138 kV power line down to 13.8 kV through two 20/25 MVA, 138-13.8 kV transformers. Each transformer will power a 13.8 kV switchgear lineup and have capacity to power the entire site. A compensator, as required by the utility provider, will also be located in the substation.

From the 13.8 kV switchgear lineups, power will be distributed throughout the site. The voltage will be further stepped down to either 4.16 kV or 3.0 kV for larger loads while the remaining will be transformed to 480 V. Single phase loads such as lighting and other low voltage loads will be powered at 220/380 V.

Two overhead lines will feed facilities outside of the main site. The north overhead line will provide power to the sewage plant, fresh water intake pumps, CIP Pond #1 and the future CIP Pond # 2.

The south overhead line will deliver power to the flotation tailings pond, geology camp, explosives area, main gatehouse and landfill.

Emergency diesel generators located throughout the various site areas will provide backup power to strategic loads as required in the event of a loss of utility power. The pit dewatering pump system will be diesel powered.

18.6.2 Fresh/Fire Water

A vertical pumping station will be installed in Veados Creek which will supply fresh water to the site from a water tank on top of a hill south east of the main site. The lower portion of the tank will be dedicated to fire water. The fire water system will be a gravity fed ring main which supplies the building sprinkler systems and hydrants at the process plant and main camp complex.

18.6.3 Potable Water

Potable water will be supplied via wells located adjacent to the main site to a packaged potable water treatment plant located next to the permanent camp. The potable water system will supply the permanent camp, wash facilities in the process plant, emergency showers and ancillary buildings all located at the main site. Potable water for facilities outside the main site and the existing geology camp, will be provided by tanker truck or bottles.

18.6.4 Sewage Treatment and Oil Water Separation

A sewage treatment plant will be located north west of the process plant area to treat sewage from the plant and camp.

An oil water separation plant will be located adjacent to the sewage treatment plant. Water will be collected from the truck shop, truck wash and diesel fueling areas and directed to this for processing.

18.6.5 Landfill

A landfill waste facility will be located off the main access road south of the gate house. All waste streams to be taken off site will be sorted into hazardous and recyclables and will be stored in this area.

18.6.6 Airstrip

There is an existing 775 m long air strip located south of the existing geology camp. Minor civil upgrades will be made to the air strip during construction.

18.6.7 Communications

The site currently has two radio towers and one telecom tower which provide communications and internet. The systems will be upgraded to accommodate construction and operations. Currently the area does not have cellular coverage.

18.6.8 Plant Mobile Fleet

A number of mobile equipment has been transferred to the Tocantinzinho site from the Vila Nova operation. This includes an ambulance, light trucks, heavy duty trucks and quadricycles. Additional equipment has been refurbished and in storage in preparation for shipment to the site. This Vila Nova equipment is allocated to plant mobile equipment and mine support equipment.

18.7 ANCILLARY BUILDINGS

18.7.1 Maintenance Facility

The maintenance facility will be comprised of two buildings, one a fabric building that will include four truck bays sized for Cat 777 trucks and a conventional masonry two story building that will house a warehouse, dry, light shops and a second floor office area. The offices will have spaces for 75 employees complete with meeting rooms, storage and washroom facilities.

18.7.2 Truck Wash

A truck wash sized to service Cat 777 trucks will be located adjacent to the main truck shop and will be housed in a fabric structure. Wash water will be captured, settled and recycled.

18.7.3 Diesel Fuel Storage and Dispensing

The main diesel fuel storage and dispensing facility will be located east of the truck shop and will service both light and heavy vehicles. The fuel supplier will build-own-operate this facility.

18.7.4 Assay Laboratory

An assay laboratory will be located adjacent to the process plant. The lab will process 350 samples per day. The facility will be outfitted with lab equipment and includes offices, compressor room, washroom and a storage area.

18.7.5 Main Security Gate / Site Security

A security gatehouse will be on the south end of the property and positioned on the site access road. All vehicular traffic will have to pass through this security gate.

Areas requiring security such as the main site and leach tailings pond will be fenced.

18.8 OFF-SITE INFRASTRUCTURE

18.8.1 Site Access Road

The site is accessed via 103 km of all-weather roads, starting from the National highway, BR 163 at Morais Almeida. The first 31 km is on the Transgarimpeiro state road. Approximately 20 km from Morais Almeida on this state road, is the town of Jardim do Ouro, where there is a barge crossing over the Jamanxim River.

The balance of the 71 km to site is on an access road constructed in 2015 by Eldorado. The 71 km site access road is a municipal road. Maintenance of the road is the responsibility of the owner and it is accessible by the public.

18.8.2 Power Line

The site will be fed by a new 200 km 138 kV power transmission line which connects to the National Grid at Novo Progresso and terminates at the site substation near the process plant.

The power line includes 476 towers and upgrades to the exit bay at Novo Progresso. The new line will be parallel to the state highway 163 to Morais Almeida, then will turn west eventually connecting to the site substation at the plant site.

18.9 ENVIRONMENTAL

18.9.1 Greenhouse and Nursery

A greenhouse and nursery will be located at the geology camp. Flora will be grown here for future reclamation.

18.9.2 Deforestation

Deforestation pads will be located around the site to stockpile non merchantable deforestation materials and stripped top soil for future reclamation activities. Merchantable timber will be placed at the south deforestation pad for on site use.

SECTION • 19**MARKET STUDIES AND CONTRACTS**

19.1 MARKET**19.1.1 Market Study**

There has been no formal market study completed for the Project. Gold will be produced as doré and sold to final refiners.

19.1.2 Price

The price of gold is the largest single factor in determining profitability and cash flow from operations. Therefore, the financial performance of the project has been, and is expected to continue to be, closely linked to the price of gold. Reserves and resources have been modelled at a gold price of US\$1,200 per troy ounce.

19.1.3 Refining Charges

Refining charges are estimated at US\$6.23/oz Au based on recent quotations.

19.1.4 Transportation Charges

Refining charges are estimated at US\$5.57/oz Au based on recent quotations.

19.2 CONTRACTS

No contracts for gold sales or hedging are in place, gold will be sold at spot price. There are no contracts or purchase agreements in place regarding the construction or operation of the project relevant to this technical report.

SECTION • 20**ENVIRONMENTAL STUDIES,
PERMITTING AND SOCIAL OR
COMMUNITY IMPACT**

20.1 ENVIRONMENTAL STUDIES

Environmental baseline studies have been completed on the Project site and preparations are underway to complete the studies on the powerline corridor. The fieldwork including forest and fauna studies, hydrology and hydrogeology monitoring, geochemistry analysis and geotechnical analysis are complete. The archeological surveys have also been conducted.

20.1.1 Flora and Fauna Studies

The Project is located in the Amazonian biome, which is comprised of extensive areas covered by forest formations that over the years has been undergoing an uncontrolled human occupation, with a high rate of deforestation caused mainly by agriculture, logging and artisanal mining activities. In general, the local forest typology is known as ombrophilous forest, prevalent throughout Amazon Region. Qualitative and quantitative studies of flora within the Project site registered 196 botanical species, some of them are classified as threatened.

In the fauna studies, aquatic and terrestrial ecosystems were considered. Within the aquatic ecosystem, limnological organisms (algae and small aquatic organisms) and fish were investigated. The terrestrial ecosystem included groups of insects, amphibians, reptiles, birds, mammals and bats.

In the study areas various limnological organisms and 91 species of fish were identified. Of this total, 20 species are important for human consumption and 25 for ornamental fish culture practice use. Also identified were 126 species of insects (mainly mosquitoes and ants), 21 amphibian species, 33 reptile species, 222 bird species, 45 mammal species (10 threatened and 1 endemic) and 48 species of bats (1 vulnerable).

The biotic environment study was completed to predict impacts caused by the installation of the Project and establish mitigation measures to reduce those impacts on the ecosystems and maintain biodiversity in the area of influence.

20.1.2 Hydrology and Hydrogeology Monitoring

Studies to evaluate water resources were completed in the Tapajós River basin, the sub-basin of the Jamanxim River, with emphasis on the micro-basin of the Tocantinzinho River, including Teodorão and Veados creeks, its tributaries. Qualitative and quantitative aspects of water resources were considered, taking into account the geographic, hydrological, physicochemical and bacteriological parameters of the study area.

The Tocantinzinho River has an extension of 197 km and its basin measures 5,566 km². Its navigability is compromised because its bed is rocky and presents many rapids and sandbanks in the dry season, being navigable only for small boats. The micro-basins are characterized by having

embedded valleys in sedimentary rocks, of average altitude of 160 m, with dendritic pattern, high angularity, medium density and medium to high asymmetry. In these regions, processes of deposition predominate, developing extensive fluvial plains composed of recent alluvial material, subject to flooding during the most recent period.

Considering that the hydrogeological conditions belong to the Crystalline Basement and sediments of the Amazon Basin, the exploration of underground water in the region is done through shallow or tube wells. The underground flow direction is preferably from NE to SW and is directed to the thalweg of the Tocantinzinho River.

Systematic monitoring of hydrological, hydrometrical and hydrogeological variables was executed within the Project site and the monitored data provided rich background information on several variables of interest for the Project. The quantity and quality of the data obtained made it possible to check several correlations between rainfall, runoff flows and water levels (whether surface or underground).

The hydrological regime of the region is defined by the Tapajós River. The flood season extends from January to May, always peaking in the month of March. However, the analysis of the data collected showed mainly that the annual rainfall in the Project area is not uniform throughout the years, showing great variability in the monthly totals. The Tocantinzinho River and Teodorão Creek have a flow regime characterized by floods followed by a long-lasting recession period. However, the minimum flow is considered more than sufficient to meet the demand of new water for the Project.

20.1.3 Air Quality Monitoring

Air quality is dependent on atmospheric emissions from each Project stages and localized climatic and topographic conditions of the area. Currently, compliance with the legal standards is favorable since the area surrounding the Project is predominantly forestry and there are not industrial activities that have fixed sources of atmospheric emission. Thermal inversion is not a problem due to the low wind velocity and high rainfall. The dry season is long but there is still rainfall, allowing that the relative humidity of the air to always be high.

The analysis of the results obtained from the total suspended particles (TSP) monitoring concludes that, in general, the air quality in the Project site of the Project can be considered good when compared with standards established by the local regulatory bodies. To ensure the maintenance of emissions within permitted values monitoring of potential sources of atmospheric emissions, which may be stationary (for example gas scrubbers) or mobile (for example trucks) will be monitored during all stages of the Project and operation.

20.1.4 Geochemistry and Geotechnical Analysis

The chemical species formed by the weathering of rocky materials are largely stored in sediments and soils. Due to the wide artisanal mining activity in the past, extensive sampling of soils and sediments were completed to determine the concentrations of chemical parameters and establish geochemical background for the site. The information will also provide data for the evaluation of possible environmental interferences resulting from future mining activity in the area. The analysis

of the soil and sediment samples presented higher concentrations of aluminum in certain areas. Mercury also requires attention, since the analysis results have shown that due to the history of artisanal mining activity, this element is found in several areas. Although the magnitude of the mercury found is not of concern, it is important that this element be investigated in greater detail in the next phases of the Project as more can be encountered during the pit pre-stripping.

The geotechnical studies were done to provide technical information to design earthworks for access, earthworks in cuts and fills and the buildings and structures required for the Project. These studies were developed in the site with the main objective to characterize the quality of the soils of foundation (subgrade) and the quality/volume of the materials deposits for construction of the structural layers such as rock and sand.

In a geotechnical-construction context a geotechnical instrumentation program will be installed to monitor the predicted estimates for soils behavior and structures, natural deformation, specifically in embankments and in the slopes and foundations.

20.1.5 Archaeological Survey

An archaeological survey was done in the areas of the site and transmission line. Analysis of past data and search field data through non-intrusive exploration was conducted and areas were identified that have potential for the existence of archaeological sites.

During the second phase of the study, the potential areas of the Tocantinzinho Project were reviewed through soil subsurface verification and heritage education activities were carried out with local communities. During the survey, six archaeological sites were identified in the area of the Tocantinzinho site and seven in the area of the transmission line. The archaeological sites identified have been through the rescue stage, curator activities and analysis of the material collected in the laboratory with the final report approved by the Institute of Historic and Artistic (IPHAN). These areas can now be used for the Project.

20.1.6 Closure and Reclamation

The closure plan was established to identify environmental, social and economical risks after production ceases and to determine measures to be implemented during construction, operation and closure. It will be continuously updated and implemented prior to the shutdown of the operation.

During the Project construction, deforestation materials and topsoil will be stored in various locations on the site to be used for reclamation. A progressive rehabilitation approach will be used to reduce the long-term closure liability. Actively rehabilitating areas during the operational stage will provide the opportunity to develop and test the most effective methodologies. Area drainage will be modified as required prior to the reclamation process.

A seedling nursery will be built on site to grow native plants that will eventually be planted in reclamation areas.

20.1.6.1 Open Pit and Waste Rock Pile

With the cessation of mining, equipment and infrastructure will be removed from the pit which will fill with water. All mine-influenced water that is not suitable for discharge to the environment will be treated. Testing and studies will be carried out to predict the future water quality in the pit.

The waste rock pile was designed with gentle slope angles so no further sloping will be required to accommodate topsoil placement. Tests for acid rock drainage prediction (ARD) were completed and most of the tests of waste have shown low reactivity and is considered non-acid rock generating, only a few points had pH below the value minimum and in spite of the typical gold sulphide mineralization in the Project site, overall there is no evidence of pH alteration in the surface waters due to acid drainage generation processes.

20.1.6.2 Tailings Storage Facility

Tailings geochemical characterization must be fully understood before the tailings dam closure plans, including cover design, are finalized. Should the tailings present ARD potential, a wet cover (permanent flooding) may be recommended since this is an effective method to avoid ARD. Soil covers can also be used to control ARD, where conditions allow.

20.1.6.3 Process Plant, Camp, Onsite Infrastructure, Onsite Roads, Onsite Power Line

Depending upon future land use, it may be possible to maintain some structures such as the camp buildings, office buildings and workshops and transfer them to the future land owner. An evaluation of potential reuse for those structures should be carried out prior to the time of closure. The current closure plan has assumed that all structures and infrastructure will have to be removed.

At the same time, equipment will be evaluated for potential reuse. Non-reusable equipment and metallic structures will be segregated from other materials to be sold as scrap. Hazardous waste generated during demolition will be segregated and disposed of properly.

The areas will be reclaimed by revegetation of native species. For hard packed areas, ripping prior to revegetation will be required to aid revegetation.

20.1.6.4 Monitoring and Maintenance

Monitoring and maintenance will be necessary in the post-closure period (approximately 5 years) to ensure proper revegetation and repair erosion that may occur.

20.2 ENVIRONMENTAL SYSTEMS

20.2.1 Sustainable Development Policies and Guidelines

The Project will adapt the corporate environmental and sustainability practices in line with global practices.

20.2.2 Environmental Management System

The Environmental Management System (SGA) has as main objective to create procedures for the environmental management of the Tocantinzinho Project. In the adopted concept, the environmental management will be practiced by all the operational and administrative areas of the Project. In the future, this system will be further detailed to manage the construction and operation of Tocantinzinho and will have a dedicated department to manage it.

20.2.3 Waste Management

Waste management was designed so that during all stages of the Project, measures will be adopted to manage waste generation. In addition, measures of non-generation, reduction, reuse, recycling and treatment of solid waste will be used. Waste management will always consider the effects on the closure of the operation.

20.3 PERMITTING

20.3.1 Existing Licenses

The environmental licensing process for Tocantinzinho Project is complete. The Environmental Impact Assessment (EIA) was submitted to the Environment Department of Para State (SEMA-PA) in January 2012 and was approved in September 2012, with the granting of the Preliminary Environmental License, which includes two main structures: the site, including activities related to mining and ore processing and the access road to the Project.

In January 2016, an installation license for the mining project was requested, which was granted in April 2017 and modifications granted in August 2017. Table 20-1 summarizes the permits.

Table 20-1: Permits with Expiration Dates

Permit	Related License	Number	Expiration Date	Comments
Tocantinzinho Site	Installation License	2771/2017	2020-04-18	
	Permit for deforestation	3383/2017	2020-04-18	
	Permit to capture, collection, rescue, transportation and release of Wildlife	3384/2017	2018-04-18	A new permit was requested to environmental agency
	Permit to Wildlife Monitoring	3381/2017	2020-04-19	
Tailings Dam and CIP Pond	Installation License	2796/2017	2020-11-20	
	Preliminary Water Permit	710/2016	2018-10-07	
	Final Water Permit	3103/2018	2023-02-01	
Fuel Station	Installation License	2816/2018	2021-01-09	
Concrete Batch Plant	Installation License	2830/2018	2021-04-11	

Permit	Related License	Number	Expiration Date	Comments
Effluent Release	Final Water Permit	Process 40870/2017	-	
	Preliminary Water Permit	740/2017	2019-01-09	An extension in validity was requested to environmental agency
Supply wells	Preliminary Water Permit	655/2016	2018-04-29	
	Preliminary Water Permit	877/2018	2020-01-29	
	Final Water Permit	2575/2016	2020-07-25	
Crossings (drainage)	Final Water Permit	2772/2017	2022-02-03	
Supply Industrial water	Preliminary Water Permit	693/2016	2018-09-08	A new permit was requested to environmental agency
Pit Mine Dewatering	Final Water Permit	2481/2016	2020-05-03	
Transmission Line	Preliminary License	1692/2017	2018-12-28	
	Installation License	2797/2017	2020-12-27	
	Permit for deforestation	3642/2017	2018-12-28	A new permit was requested to environmental agency.
	Permit to capture, collection, rescue, transportation and release of Wildlife	3643/2017	2018-04-18	A new permit was requested to environmental agency
	Permit to Wildlife Monitoring	3644/2017	2018-04-18	A new permit was requested to environmental agency
Access Road	Permit to capture, collection, rescue, transportation and release of Wildlife	9181/2017		Pending
	Permit for Deforestation	39318/2016		Pending
	Installation License	2862/2018	2020-08-19	
	Final Water Permit	2764/2017	2022-03-02	

20.3.2 Pending and Maintenance of Existing Licenses

The pending licenses include:

- Deforestation permit for the new works on the access road - Process n° 2016/39318.
- Authorization for capture, collection, rescue, transportation and release of wildlife for access road - Process n° 2017/9181.

20.4 SOCIAL AND COMMUNITY IMPACT

20.4.1 Communities of Interest

Considering that the site is remote from any residence other than the garimpeiros in the vicinity of the site, the operation will have minimal impact on an existing community. The nearest indigenous population is the Munduruku located more than 100 km of the Project site. The nearest village is Jardim do Ouro which is approximately 85 km away from the site. With its small population of approximately 1,000 inhabitants and minimal infrastructure, impacts, other than employment, are considered to be minimal. Activity in Jardim do Ouro is expected to increase as this will become the hub for transportation of materials to the site especially during construction.

The operation will employ people from Para State with travel hubs expected to be set up in Moraes Almeida and Itaituba. In addition to employment hubs, small businesses that will support the mine are expected to be centered in these cities.

Brazauro has developed a series of social investment actions designed to improve the living conditions of individuals in the neighbouring communities. As part of these actions the company provided the following:

- Construction material to build a police station in Moraes Almeida and Jardim do Ouro village.
- Educational materials to equip a library and a multidisciplinary laboratory in a Municipal School in Moraes Almeida.
- Contributed to provide logistic support to the teams of the Ministry of Health and Municipal Health Department of Itaituba for tropical diseases control services in the areas surrounding the Project.

The Project will maintain a close relationship with governmental entities in the federal, state and municipal levels that regulate the mining activity.

The transmission line of the Project will go through 148 properties in which 117 have already been negotiated. Agreements for the remaining are currently being negotiated.

20.4.2 Engagement Plan

20.4.2.1 Current Engagement

A public hearing for the Tocantinzinho Project was held June 14, 2012 in Itaituba as part of the environmental permitting process. The meeting was attended by federal, state and municipal

representatives along with trade unions and the general public. Brazauro outlined the Project and presented the employment policy regarding the local population. Discussions were focused on the opportunities for local businesses in Itaituba and Moraes Almeida to service the Project demands during the construction and operation. The meeting was viewed as very positive but public concerns will need to be continually monitored and addressed.

20.4.2.2 Future Engagement

Several social programs related to community development were created and all of them are scheduled to be implemented during the Project installation. They include:

- Social communication program and relationship with stakeholders in Tocantinzinho Project that includes a plan for development and implementation of the Tocantinzinho Project website. The main objective is to contribute to the strengthening of the social dialogue between the community and the company, to give greater support to all activities that involve execution.
- Local development promotion program, which includes a rural economy promotion program
- Training, qualification and improvement of the workforce program, which includes an action plan for labor demobilization
- Occupational health and safety program
- Public management support program
- Environmental education program

SECTION • 21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

21.1.1 Basic Engineering 2017

During the second quarter of 2017, basic engineering was completed by Eldorado and Ausenco which produced a level 3 capital cost estimate as defined by the American Association of Cost Engineers (AACE) estimate classification.

21.1.2 Optimization 2018

In the second quarter of 2018, a Project optimization exercise was undertaken to investigate cost savings without sacrificing the process integrity.

The focus of the optimization program was to streamline the process flow sheet, consolidate the site infrastructure into one central location, optimize the facilities for a nine year mine life and update the costs.

During the optimization process, the estimate was updated with final design quantities and contractor proposals for the earthworks, deforestation and the high voltage overhead power line.

The tax calculations were revised to reflect the current tax laws and capture tax credits in the operating costs.

A detailed execution plan was prepared which included an early works program; a detailed resource loaded level 3 schedule; a contracting and procurement package plan; temporary construction infrastructure requirements and construction equipment lists and a Project staffing plan.

The delivery strategy for the Project expanded the owner self-perform scope to include project and construction management, mine development, all major earthworks and deforestation. The work rotation schedule was modified to reflect recent changes to the labour laws. The mine development plan and the major earthworks program were synchronized to ensure mine waste rock was available for aggregate production and mass fill requirements.

Construction indirect costs were estimated, a detailed logistics plan was developed with current contractor pricing and construction productivity rates were estimated and compared with contractor rates.

21.1.2.1 Technical Report Update 2019

In 2019, the estimate was updated with new cost information for selected items, particularly the SAG mill, mine equipment and mine preproduction costs; escalation and exchange rates. The 2019 review also added additional costs for electrical supply compensation and changes to selected material takeoff quantities. Costs are as of Q1 2019.

A summary of the capital cost in US dollars is shown in Table 21-1.

Table 21-1: Capital Cost Summary by Area

WBS	Description	Total US\$ M	% of Total
A	Overall Site	15.13	4
B	Mine	117.20	31
C	Crushing	9.04	2
D	Process Plant	59.19	16
E	Tailings	9.77	3
F	CIP Ponds	6.31	2
G	Camp	3.05	1
H	Infrastructure	24.35	6
J	Ancillary Facilities	7.87	2
K	Off Site Infrastructure	35.55	9
N	Geology	1.63	0
P	Environmental	0.24	0
Total Direct Cost		289.31	76
Q	Indirects	55.26	15
R	Capital Spares	7.30	2
S	EPCM	17.62	5
T	Owner's Cost	9.80	3
Total Indirect Cost		89.97	24
Total Project Cost		379.28	100
U	Contingency	62.50	16
Total Project /w Contingency		441.78	116

The level of accuracy for an AACE Class 3 estimate is -10% to -20% on the low side and +10% to +30% on the high side with a contingency level for a 50% probability of an overrun or underrun in the range of 5% to 15%.

The current capital cost estimate has an estimated accuracy of – 11% to +21% of the installed cost (before taxes and contingency). Contingency is 16.5% of Project cost before contingency or 14% of total Project cost including contingency. Therefore, the capital cost estimate is within the accuracy level of a Class 3 estimate.

The US\$5.5 M royalty buy down payout prior to production is not included in Table 21-1 but is included in the economic analysis in Section 22.

21.1.2.2 Sustaining Capital Costs

The life of mine sustaining costs are estimated at US\$151.2 M. The major sustaining costs include the following:

- Site preparations including deforestation for the tailings pond and waste rock dump expansions and aggregate production for pond and road construction and maintenance
- Mining equipment which includes new purchases, replacements, and rebuilds
- Process plant LOM sustaining improvements and a water treatment plant in Year 1

- Tailings pond expansion and dam raise in Year 2
- Construction of the second CIP pond constructed in Year 2 and 3
- LOM maintenance on ancillary facilities
- Access and site road LOM upgrades and maintenance
- Construction indirects including management, fuel and freight for the sustaining projects
- Completion of spares purchasing in Year 1
- Owners' costs allowance for LOM administrative equipment upgrades
- No recoverable consumption taxes on the sustaining costs

Table 21-2: Sustaining Capital Cost Summary by Area

WBS	Sustaining Capital	Total US\$ M	% of Total
A	Land Preparations and Aggregate	7.9	5%
B	Mining	50.8	34%
D	Process Plant	2.4	2%
E	Tailings Pond	25.3	17%
F	CIP Pond	4.6	3%
J	Ancillary Facilities	1.6	1%
K	Access and Site Roads	15.7	10%
Q	Construction Indirects	14.7	10%
R	Spares	4.2	3%
T	Owners Costs	1.4	1%
	Non Recoverable Taxes	22.7	15%
	Total Sustaining Costs	151.2	100%

In addition to the sustaining capital shown in Table 21-2 there is a mine closure cost of US\$29.4 M for the rehabilitation of the site partially offset by a US\$17.4 M credit for the sale of the used process and mining equipment and salvage value of steel and ancillaries. Closure cost and salvage credit both occur in Year 10 and are included in the economic analysis in Section 22.

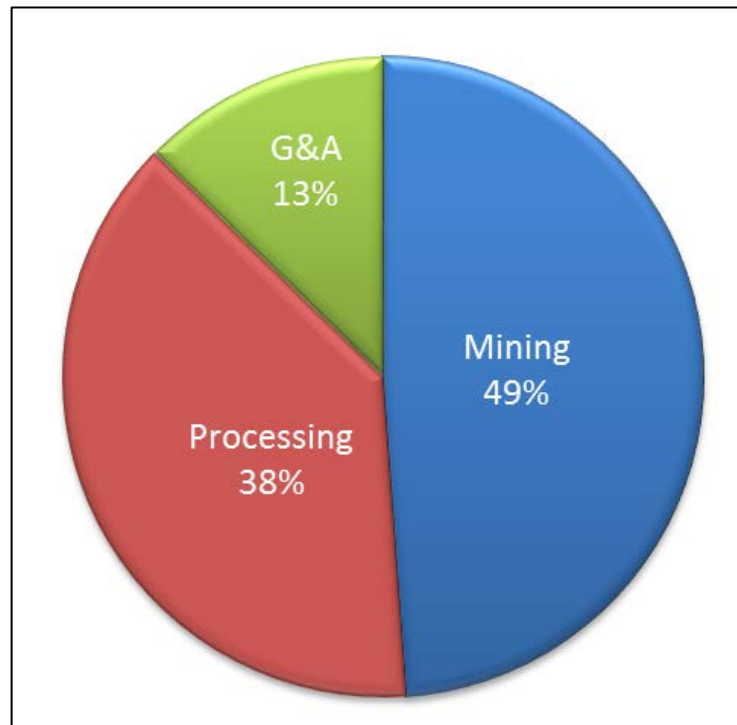
21.2 OPERATING COSTS

21.2.1 Summary and Manpower

The life of mine overall operation cost for the project is US\$23.41 per tonne of ore processed. As shown in Table 21-3, the operating cost includes the mine, process plant, and general and administration (G&A). Royalty payments are included in the financial analysis but not included in the G&A costs. All costs are as of Q1 2019. The Figure 21-1 shows the operating costs.

Table 21-3: Life of Mine Operating Costs

Area	Type of Cost	Unit Cost US\$/t Processed	Total Cost US\$ M/y
Mining	Manpower	2.14	9.28
	Diesel	2.61	11.31
	Consumables	6.63	28.75
	Other	0.05	0.23
	Total Mining costs	11.43	49.57
Processing	Manpower	1.17	5.06
	Power	2.14	9.26
	Consumables	5.21	22.59
	Maintenance	0.50	2.17
	Total Processing Costs	9.02	39.09
G&A	Manpower	1.20	5.19
	Camp	0.79	3.42
	Other	0.98	4.23
	Total G&A costs	2.96	12.83
Total	Total costs	23.41	101.49

**Figure 21-1: Operating Cost Summary**

Annual manpower for the operations is summarized below in Table 21-4.

Table 21-4: Manpower Operating Costs

Description	Number	Total Annual Compensation US\$ M	Ore US\$/t
Mine Staff	26	1.62	0.37
Mine Operations	149	4.53	1.04
Mine Maintenance	75	3.16	0.72
Total Mine	250	9.28	2.14
Process Staff	21	1.41	0.33
Process Operations	67	1.96	0.45
Process Maintenance	40	1.70	0.39
Total Process	128	5.06	1.17
G&A Staff	41	3.33	0.77
G&A Operations	65	1.86	0.43
G&A Contract	66	Included in camp costs	
Total General & Administrative	172	5.19	1.20
Total Operations	550	19.53	4.50

The organization chart and manpower list were derived from first principles. Salary and wage rates were based on labour surveys completed in 2017 and escalated with the annual inflation rates. The rates are based on local requirements for social charges, labour law and current practices in the state.

Positions were based on travel primarily from Itaituba and the remainder from Morais Almeida.

21.2.2 Mining

The mine operating costs were developed from first principles based on the mining methods as outlined in Section 16.

The mining will be performed by mine employees and will not be contracted.

The first two years of mining contributes to the capital expenditure and is focused on developing the mine through overburden removal as well as providing construction rock from the mine and the adjacent quarry. These operations prior to production are not included in the operating costs however the majority of the equipment and related staff will be in onsite and trained prior to Year 1.

The mining operations starts at 270 people in Year 1, peaks at 294 in Year 5 with significant reductions in Year 8 in anticipation of the completion of mining in Year 9. Numbers of manpower vary with the mining plan and maintenance requirements. Annual mining operations manpower is summarized in Table 21-4.

The mining production equipment will be a combination of new, refurbished and transferred equipment from Eldorado's Vila Nova mine that is no longer in operation. Refurbished production

equipment was selected for equipment that is commonly available on the market in country. The refurbished equipment will be purchased directly from the supplier, who will guarantee the availability. Equipment from Villa Nova has been assigned to mine support roles and will not be used for mine production. Maintenance costs were based on new or refurbished units. Minimal equipment will be leased.

Details of the unit and LOM costs can be found in Figure 21-2 and Table 21-5.

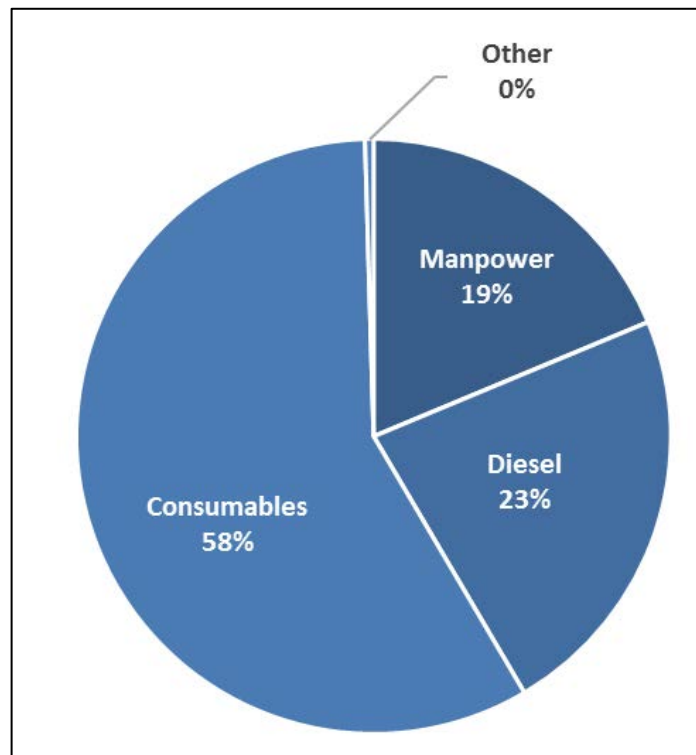


Figure 21-2: Mining Summary

Table 21-5: Mine Operating Unit Costs

Type of Cost	Ore US\$/t	Mined US\$/t	Total Cost LOM US\$ M
Tonnes	40,002	172,052	172,052
Salaries and Wages	2.14	0.50	85.61
Energy	2.61	0.61	104.32
Consumables, R&M	6.63	1.54	265.27
Equipment Leasing	0.05	0.01	2.11
Total	11.43	2.66	457.32
General Mine Expense	0.60	0.14	23.99
Drilling	1.36	0.32	54.47
Blasting	1.84	0.43	73.53
Loading	1.29	0.30	51.46
Hauling	4.83	1.12	193.01
Roads & Dumps	1.52	0.35	60.85
Total	11.43	2.66	457.32

21.2.3 Process Plant and Related Infrastructure

The operating cost for the process plant and related infrastructure at the Tocantinzinho Project site is based on the estimated direct costs for processing at nominal annual throughput of 4.34 Mt of run-of-mine ore. The concentrator availability has been assumed at 90% (7,884 h/y), which incorporates both scheduled and unscheduled shutdowns.

The process plant operating costs comprise the plant manpower, reagents & consumables, power required for the process, and operational maintenance. The Tocantinzinho process plant and related infrastructure annual operating cost is US\$39.1 M, equivalent to US\$9.02 per tonne milled. The details from LOM operating costs are summarized below in Table 21-6.

Table 21-6: Process Plant Operating Cost

Type of Cost	Ore US\$/t	Total Cost US\$ M/y
Manpower	1.17	5.06
Power	2.14	9.26
Consumables	5.21	22.59
Maintenance	0.50	2.17
Total Processing Costs	9.02	39.09

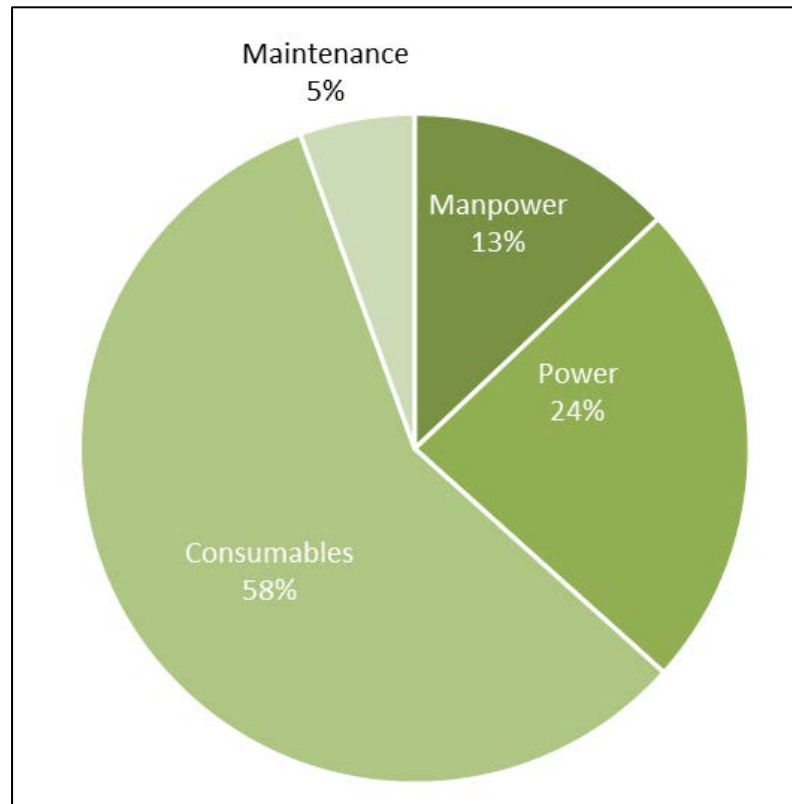


Figure 21-3: Process Summary

Annual process plant manpower is summarized in Table 21-4.

21.2.3.1 Power

Power consumption estimates have been adopted from the electrical load analysis. The total estimated power consumption per annum is 140.9 MWh. Power supply costs were based on the ANEEL (National Agency of Electric Energy) and the power market for the free consumer plus 25% ICMS of US\$0.066/ kWh.

The estimated process plant power cost is US\$9.3 M, equivalent to US\$2.1 per tonne milled. The power cost for the process plant is summarized in Table 21-7 below.

Table 21-7: Process Plant Power Costs

Category	Unit	Value
Total per Annum	MWh	140.90
Power Cost	US\$/ kWh	0.07
Annual Power Cost	US\$ M	9.26
Power Cost	US\$/t	2.14

21.2.3.2 Reagents and Consumables

The estimated annual reagent and consumables cost including the freight cost is US\$22.6 M, equivalent to US\$5.2 per tonne milled. Reagent consumptions are estimated from laboratory test results and mass balances calculation. The unit prices for supplies and consumables used in the operating cost estimate were provided by various potential suppliers completed during the pre-feasibility study and escalated with the annual inflation rates. The overall plant consumables are divided into four major areas including the wear components, process plant & gold room consumables, and future water treatment plant reagents.

The reagents and consumables costs for the process plant are summarized in Table 21-8 below.

Table 21-8: Process Plant Reagents and Consumables Costs

Category	Description	Total Cost US\$ M/annum	Unit Cost US\$/ t
Wear Components	Crusher Wear Parts	0.05	0.01
	Mill Liners	2.94	0.68
	Grinding Media	10.10	2.33
Process Consumables	Cyanide	3.58	0.83
	SIBX	0.78	0.18
	Frother DF250	0.87	0.20
	Flocculant	0.08	0.02
	Copper Sulphate	0.54	0.13
	Sodium Metabisulphite	0.78	0.18
	Lime	1.68	0.39
	Sodium Hydroxide	0.05	0.01
	Hydrochloric Acid	0.44	0.10
	Activated Carbon	0.49	0.11
Gold Room	Gold Room Reagents	0.02	0.00
Water Treatment Plant	Lime	0.18	0.04
	Flocculant	0.01	0.00
	Sulphuric Acid	0.00	0.00
Total	Total Costs	22.59	5.21

21.2.3.3 Maintenance

The estimated annual maintenance cost is US\$2.2 M equivalent to US\$0.51 per tonne milled. Process maintenance was factored assuming 5% of the direct mechanical equipment cost. Total direct cost is US\$43.5 M including taxes.

Table 21-9: Process Plant Maintenance

Category	Unit	Value
Total Direct Mechanical Equipment Cost	US\$ M	43.46
Annual Maintenance Cost	US\$ M	2.17
Maintenance Cost	US\$/t	0.51

21.2.4 General and Administration

Operating costs for General and Administration (G&A) include items that are not captured in the mine or the process costs. These costs include items such as the management and administration personnel (manpower), safety, medical, catering and travel expenses, support tools and shared equipment, emergency response, site-wide maintenance, insurance, legal fees and property taxes, as well as other miscellaneous fees.

The annual G&A costs are estimated at US\$12.8 M equivalent to US\$2.96/ t ore milled. Table 21-10 summarizes the total cost. Royalties are not included in Table 21-10 but are included in the economic analysis in Section 22. Royalties total US\$63.1 M or US\$1.58 per tonne milled.

Table 21-10: General and Administration

Description	Total Cost US\$ M/ annum	Unit Cost US\$/ t
Administration	1.13	0.26
Information Systems	0.19	0.04
Materials Management	0.52	0.12
Human Resources	0.53	0.12
Safety	0.59	0.14
Environmental	0.33	0.08
Community	1.91	0.44
Subtotal	5.19	1.20
G&A Overhead	4.23	0.98
Camp & Transportation	3.42	0.79
Total Cost	12.83	2.96

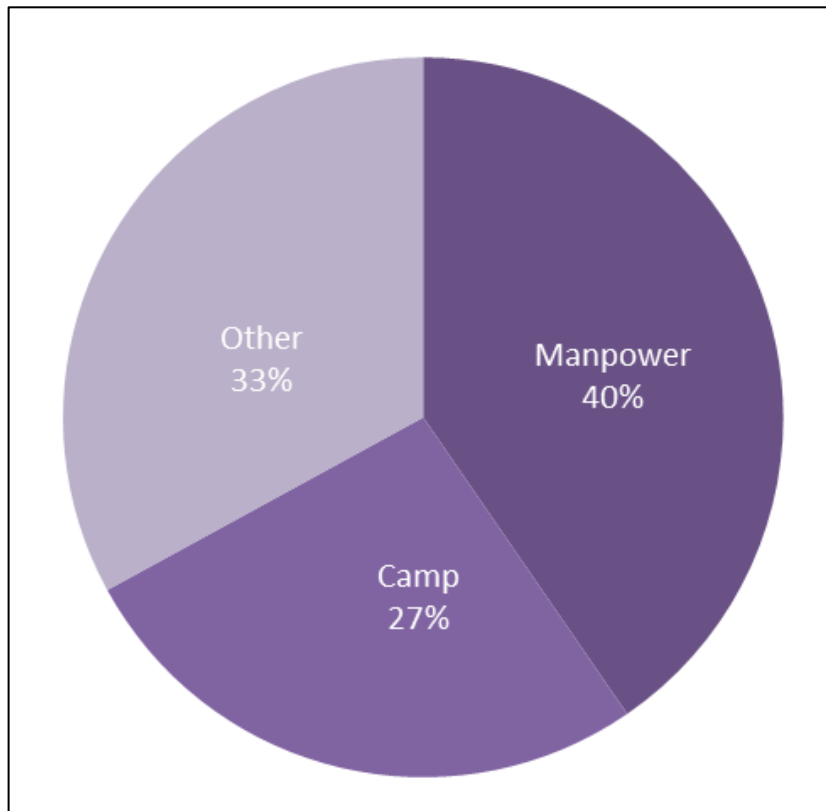


Figure 21-4: General and Administration Summary

SECTION • 22 ECONOMIC ANALYSIS

The financial analysis for the Tocantinzinho Project was completed using an Excel based discounted cash flow model developed by L&M. The model is set up to assess the project metrics and is customized for the project and related tax credits during operation.

22.1 PRINCIPLE ASSUMPTIONS

The exchange rate was based on 4.00 Brazilian reals (BRL) per 1.00 US dollar (US\$). This was based on an internal Eldorado assessment done in the 2nd half of 2018. The exchange rate is within the range of historical rates in the past year (Q2 2018). This was applied to the capital project costs as well as the long-term rate during operation of the mine.

Gold price is at US\$1,300 per ounce and is based on current market pricing. Mining reserves are based on a gold price of US\$1,200. Discount rate for NPV is 5%.

22.2 CASH FLOW FORECASTS

Capital costs for the project of US\$441.8 M plus an additional US\$5.5 M NSR royalty buy down payment would be spent in Years -2 and -1.

Sustaining capital over Years 1 to 9 would be US\$151.2 M not including mine closure. The sustaining capital includes the water treatment plant for the CIP pond effluent in Year 1. As well, during operation, CIP Pond # 2 with associated infrastructure will be built and the flotation dam will be raised. Replacement mining equipment will also be purchased during the operation.

Mine closure is estimated at US\$29.4 M and salvage of the assets on site would provide an income of US\$17.4 M

The operating expenditures are US\$23.41 per tonne of ore processed.

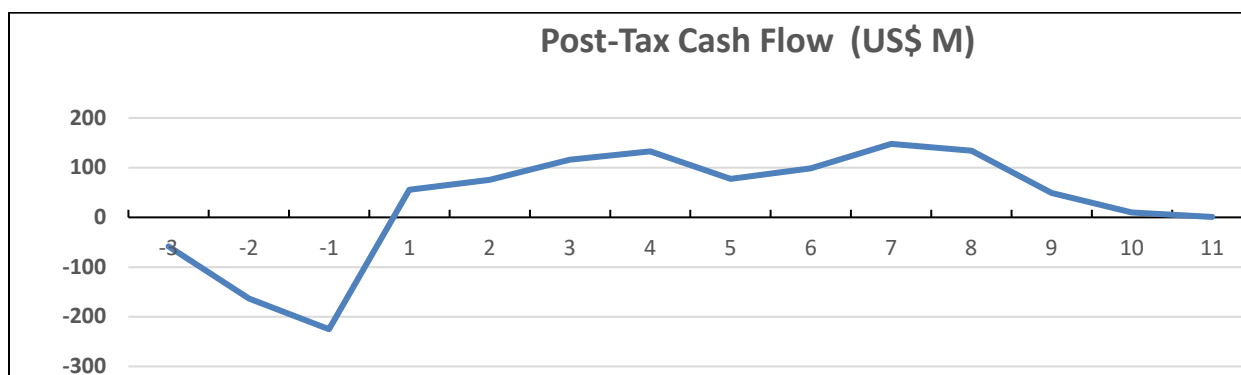
The capital and operating costs are summarized in Table 22-1.

Table 22-1: Capital and Operating Costs Summary

Item	Unit	Amount
Initial CAPEX	US\$ M	441.80
Sustaining Capital	US\$ M	151.20
Mine Closure	US\$ M	29.40
Salvage Value	US\$ M	-17.40
NSR Buy-Down Rights exercising	US\$ M	5.50
CAPEX (LOM Tax included)	US\$ M	610.40
Mining	US\$/ t ore	11.43
Processing	US\$/ t ore	9.02
G&A	US\$/ t ore	2.96
OPEX LOM Average (NRT included)	US\$/ t ore	23.41
<i>Mining unit cost (Operating phase)</i>	<i>US\$/ t mined</i>	<i>2.35</i>

NRT = Non Recoverable Taxes

Cash flow forecasts are summarized in Figure 22-1. Details of profit and loss are shown in Table 22-5 and cash flow in Table 22-6.

**Figure 22-1: Cash Flow Forecast**

22.3 FINANCIAL ANALYSIS

The financial analysis at a gold price of US\$1,300 per ounce yields an NPV of US\$216.3 M at a 5% discount and an IRR of 13.4% post tax, the results are summarized in Table 22-2.

Table 22-2: Financial Analysis

Financial Analysis	Unit	Post-Tax	Pre-Tax
Project NPV@5%	US\$ M	216.3 0	255.80
Total Cash Flow (NPV@0%)	US\$ M	451.70	511.60
Internal Rate of Return	%	13.4	14.5
EBITDA (annual average)	US\$ M	109.60	109.60
Payback	Years	4.4	4.3

22.4 TAXES AND ROYALTIES

22.4.1 Taxes and Fiscal Benefits

The applicable taxes are included in the economic analysis. A number of fiscal benefits are available and are also included in the analysis. The taxes and benefits include:

22.4.1.1 Federal Taxes

- II: Imposto de Importação
- IPI: Imposto sobre Produtos Industrializados
- IRPJ: Imposto de Renda da Pessoa Jurídica
- CSLL: Contribuição Social sobre o Lucro Líquido
- COFINS: Contribuição para o Financiamento da Seguridade Social
- PIS: Programa de Integração Social
- CFEM: Compensação Financeira pela Exploração de Recursos Minerais
- AFRMM: Adicional ao Frete para Renovação da Marinha Mercante

22.4.1.2 State Taxes

- ICMS: Imposto sobre Operações Relativas à Circulação de Mercadorias e sobre Prestação de Serviços de Transporte Interestadual e Intermunicipal e de Comunicação.
- DIFAL: Complemento relativo ao Diferencial de Alíquotas do ICMS

22.4.1.3 Municipal Taxes

- ISSQN: Imposto sobre Serviços de Qualquer Natureza

22.4.1.4 Fiscal Benefits at Federal Level

- RECAP - Suspension of PIS and COFINS on the acquisitions of machinery, instrumentation and equipment in the construction phase. The rules and the granting of the benefit are determined by the Secretaria da Receita Federal do Brasil (“SRF”). The legal basis of RECAP is in effect and provided for in Articles 12 to 16 of Law N° 11,196, of November 21, 2005 and the list of items considered as “BK” is contained in the Federal Decree N° 6581 of September 26, 2008.
- SUDAM - INCOME TAX - The Company is subject to corporate income tax in Brazil at a rate of 25% and to social contribution tax at a rate of 9%. The Company is entitled to a special Brazilian tax incentive granted by the Superintendence for the Development of the Amazon (“SUDAM”) that provides a 75% reduction to the corporate income taxes payable on eligible profits earned for the year in relation to the Tocantinzinho operations. The Company is entitled to the SUDAM tax incentive for a 10-year period commencing in the year of receipt of the Appraisal Certificate from SUDAM. To receive the full benefits of the exemption, the Company is required to make an application the SUDAM tax incentive for the implementation of the new operations. Such applications are subject to approval by SUDAM. Legal basis: Federal Law N° 13,799, of January 3rd, 2019.
- INCENTIVIZED ACCELERATED DEPRECIATION - SUDAM: This benefit allows for acceleration of the depreciation and amortization expenses for the purposes of income tax calculation. Legal basis: art. 31 of Law N° 11196 of November 21, 2005; Decree N° 5988, of October 19, 2006; Decree N° 4212, of April 26, 2002; and Decree N° 4213, of April 26, 2002.
- PIS and COFINS CREDITS ANTICIPATION - SUDAM: Granting period of 12 months from the purchase of credits of the contribution for the PIS and COFINS. Legal basis: art. 31 of Law N° 11196 of November 21, 2005; item III of §1 of art. 3 of Law N° 10637, of December 30, 2002; item III of §1 of art. 3 of Law N° 10833, of December 29, 2003; paragraph 4 of art.15 of Law N° 10865, of April 30, 2004; Decree N° 5988, of December 19, 2006; Decree N° 5789, of May 25, 2006; Decree N° 4212, of April 26, 2002; and Decree N° 4213, of April 26, 2002. This benefit ensures that the PIS and COFINS paid on purchases are credited.
- The annual income tax calculations (IRPJ and CSLL) and PIS and COFINS offsets are shown in Table 22-7.

22.4.1.5 Fiscal Benefits at State Level

- No benefit for the ICMS/ DIFAL has been considered in the Base Case.

22.4.1.6 Fiscal Benefits at Municipal Level

- ISSQN Reduction: While still depending on final negotiation and the signing of an agreement with the Municipality, a reduction of the ISSQN rate from 5% to 3% was assumed based on other large project incentives in the Northern Region.

22.4.2 Royalties

Two royalties were included in the analysis.

22.4.2.1 Royalty Payable to the Federal Government – CFEM: (Compensação Financeira pela Exploração de Recursos Minerais)

The Federal Constitution of Brazil has established that the states, municipalities, Federal districts and certain agencies of the federal administration are entitled to receive royalties for the exploitation of mineral resources by holders of mining concessions (including extraction permits). The royalty rate for gold is 1.5% of gross sales of the mineral product, less sales taxes on the mineral product, transportation and insurance costs.

22.4.2.2 Private Royalty (NSR)

A Net Smelter Royalty due to Sailfish Royalty Corp. covers all future mineral production from the Tocantinzinho Project and requires the payment of 3.5% of the gross revenue. The economic analysis included the exercise of a buydown right with Sailfish Royalty Corp. of US\$5.5 M at the beginning of the construction period, reducing the payment to 1.5%. The buydown right is not included in the costs presented in Section 21 however it is included in the economic analysis calculations.

22.5 SENSITIVITY ANALYSIS

The project financial performance is most sensitive to the gold price and exchange rate as well as the capital and operating costs. The results of the sensitivity analysis are shown in Table 22-3 and Figure 22-2 illustrates the sensitivity of the project in terms of NPV and IRR to the gold price.

Table 22-3: Gold Price Sensitivity

Gold Price Cash Flow Analysis		Gold Price (US\$/oz)				
		1,100	1,200	1,300	1,400	1,500
Post-tax NPV@5%	US\$ M	20.00	119.60	216.30	312.60	408.70
IRR	%	5.8	9.8	13.4	16.6	19.7
EBITDA	US\$ M	78.10	93.80	109.60	125.30	141.00
Payback	Years	6.4	5.4	4.4	3.8	3.4

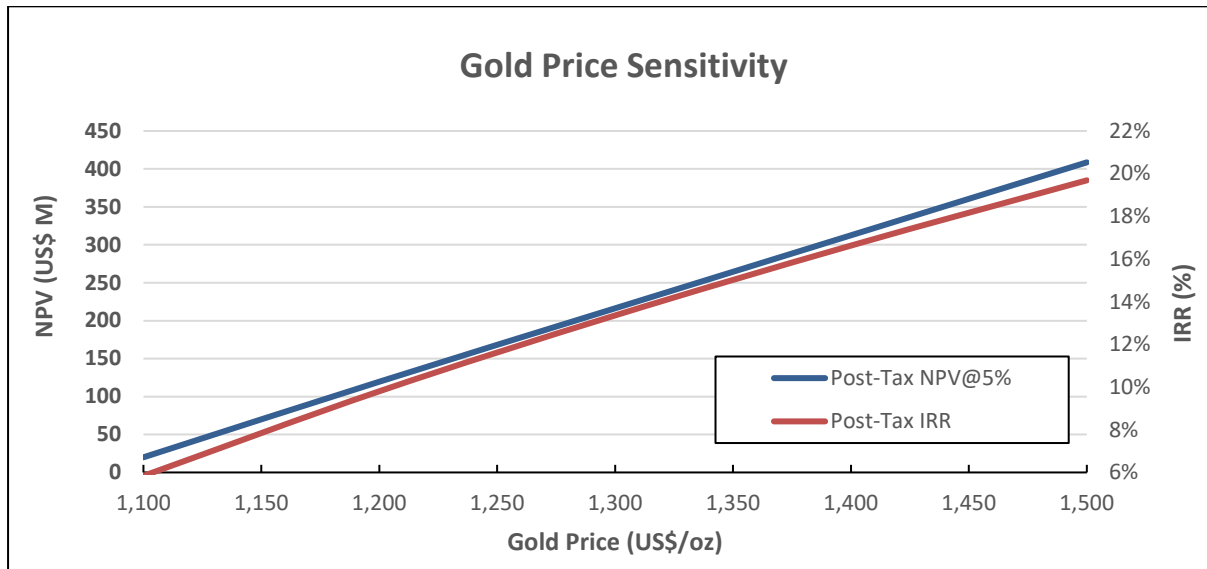


Figure 22-2: Gold Price Sensitivity

The project financial performance is also sensitive to fluctuations in the exchange rate of the Brazilian real. In the past year the real has ranged between 3.64 to 4.21 BRL, sensitivities were run between 3.5 BRL and 4.5 BRL to the US\$ and are shown in Table 22-4 and illustrated in Figure 22-3.

Table 22-4: Exchange Rate Sensitivity

Exchange Rate Cash Flow Analysis		Exchange rate (BRL/US\$)				
		3.50	3.75	4.00	4.25	4.50
Post-tax NPV@5%	US\$ M	124.40	173.50	216.30	254.00	287.50
IRR	%	9.6	11.6	13.4	15.0	16.5
EBITDA	US\$ M	101.70	105.90	109.60	112.80	115.60
Payback	Years	5.4	4.9	4.4	4.0	3.8

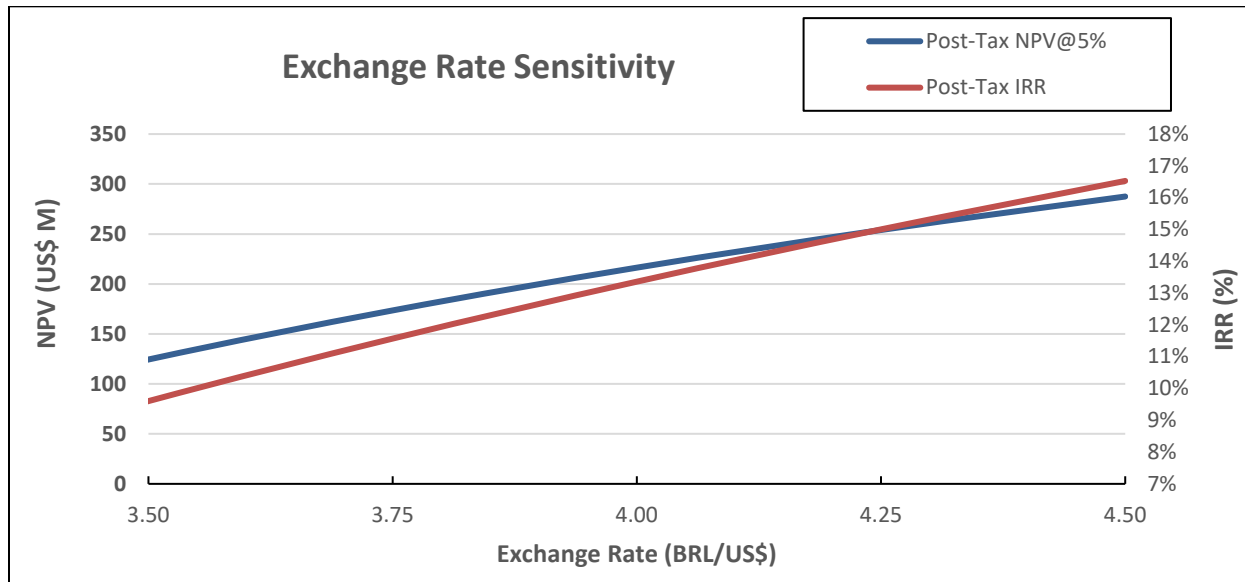


Figure 22-3: Exchange Rate Sensitivity

Figure 22-4 shows a comparison of the sensitivities of gold price, CAPEX, OPEX and the Brazilian real exchange rate to NPV.

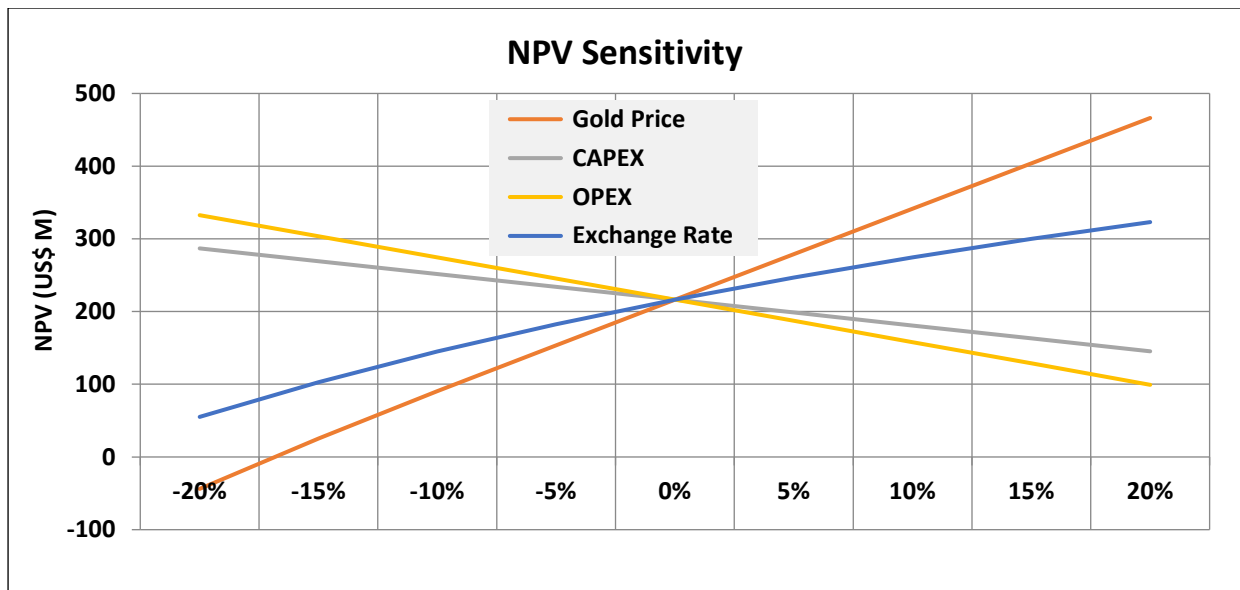


Figure 22-4: Sensitivity Analysis - Post-Tax NPV@5%

Figure 22-5 shows a comparison of the sensitivities of gold price, CAPEX, OPEX and the Brazilian real exchange rate to IRR.

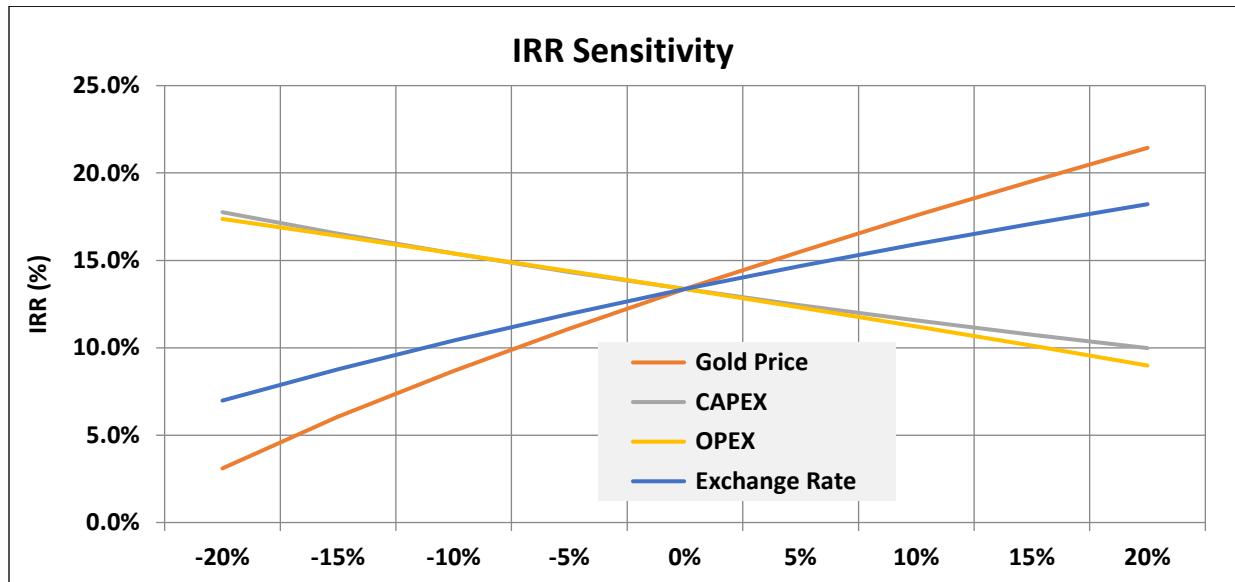


Figure 22-5: Sensitivity Analysis – Post-Tax IRR %

22.6 FINANCIAL PROJECTIONS

The annual cash flow forecast were built from a first principles financial model. The financial projections were based on the project economics described in Section 21. Financial models including the taxation and royalties, tax incentives described above, and depreciation rates were based on Brazilian accounting practices. The results are show in Table 22-5 Table 22-6 and Table 22-7.

Table 22-5: Profit & Loss

LOM CF Annual Projections		Project Year->	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11
PROFIT & LOSS																
GROSS REVENUE	(000USD)	2,112,946.8	0.0	0.0	0.0	197,419.7	246,525.2	254,947.7	283,675.8	230,893.5	231,403.1	268,731.2	236,545.1	124,411.8	38,393.7	0.0
Gold delivered	(koz)	1,625.3	0.0	0.0	0.0	151.9	189.6	196.1	218.2	177.6	178.0	206.7	182.0	95.7	29.5	0.0
Gold Price	(USD/oz)	1,300.0	0.0	0.0	0.0	1,300.0	1,300.0	1,300.0	1,300.0	1,300.0	1,300.0	1,300.0	1,300.0	1,300.0	1,300.0	0.0
Gold revenue	(000USD)	2,112,946.8	0.0	0.0	0.0	197,419.7	246,525.2	254,947.7	283,675.8	230,893.5	231,403.1	268,731.2	236,545.1	124,411.8	38,393.7	0.0
Cost Deductions	(000USD)	(80,032.0)	0.0	0.0	0.0	(7,477.7)	(9,337.6)	(9,656.6)	(10,744.8)	(8,745.5)	(8,764.8)	(10,178.7)	(8,959.6)	(4,712.3)	(1,454.2)	0.0
Refining & Transport	(000USD)	(16,897.0)	0.0	0.0	0.0	(1,578.7)	(1,971.4)	(2,038.8)	(2,268.5)	(1,846.4)	(1,850.5)	(2,149.0)	(1,891.6)	(994.9)	(307.0)	0.0
Net of tax/ CIF	"	(16,213.1)	0.0	0.0	0.0	(1,510.7)	(1,930.4)	(1,951.0)	(2,170.8)	(1,766.9)	(1,770.8)	(2,056.5)	(1,810.2)	(952.1)	(293.8)	0.0
Non recoverable taxes	"	(683.9)	0.0	0.0	0.0	(68.0)	(41.0)	(87.8)	(97.7)	(79.5)	(79.7)	(92.6)	(81.5)	(42.9)	(13.2)	0.0
Royalties	(000USD)	(63,134.9)	0.0	0.0	0.0	(5,898.9)	(7,366.2)	(7,617.8)	(8,476.2)	(6,899.1)	(6,914.3)	(8,029.7)	(7,068.0)	(3,717.4)	(1,147.2)	0.0
Government Royalty (CFEM)	"	(31,694.2)	0.0	0.0	0.0	(2,961.3)	(3,697.9)	(3,824.2)	(4,255.1)	(3,463.4)	(3,471.0)	(4,031.0)	(3,548.2)	(1,866.2)	(575.9)	0.0
Other Royalties (NSR)	"	(31,440.7)	0.0	0.0	0.0	(2,937.6)	(3,668.3)	(3,793.6)	(4,221.1)	(3,435.7)	(3,443.3)	(3,998.7)	(3,519.8)	(1,851.3)	(571.3)	0.0
NET REVENUE	(000USD)	2,032,914.8	0.0	0.0	0.0	189,942.0	237,187.6	245,291.0	272,931.1	222,148.0	222,638.2	258,552.5	227,585.5	119,699.5	36,939.4	0.0
OPERATING COSTS	(000USD)	(937,413.1)	0.0	0.0	0.0	(106,582.5)	(111,873.1)	(115,553.3)	(119,189.0)	(124,211.1)	(109,624.9)	(90,502.3)	(74,534.9)	(66,153.9)	(19,188.0)	0.0
On site costs before taxes	(000USD)	(900,633.2)	0.0	0.0	0.0	(90,943.1)	(95,101.2)	(98,086.7)	(100,933.3)	(104,826.4)	(93,057.0)	(77,815.8)	(64,900.7)	(58,027.3)	(16,941.8)	0.0
OPEX non recoverable taxes	(000USD)	(136,780.0)	0.0	0.0	0.0	(15,639.4)	(16,772.0)	(17,466.6)	(18,255.7)	(19,384.8)	(16,567.9)	(12,686.5)	(9,634.2)	(8,126.6)	(2,246.2)	0.0
EBITDA	(000USD)	1,095,501.7	0.0	0.0	0.0	83,359.5	125,314.5	129,737.7	153,742.0	97,936.9	113,013.3	168,050.2	153,050.5	53,545.6	17,751.4	0.0
	(%)	51.8%	0.0%	0.0%	0.0%	42.2%	50.8%	50.9%	54.2%	42.4%	48.8%	62.5%	64.7%	43.0%	46.2%	0.0%
Depreciation/ Amortization & Depletion	(000USD)	(618,965.4)	0.0	0.0	0.0	(100,423.0)	(116,346.8)	(89,113.6)	(64,466.2)	(62,478.6)	(45,233.6)	(45,401.2)	(39,037.8)	(30,546.9)	(25,917.6)	0.0
EBIT	(000USD)	476,536.3	0.0	0.0	0.0	(17,063.5)	8,967.6	40,624.1	89,275.9	35,458.3	67,779.7	122,649.0	114,012.8	22,998.7	(8,166.2)	0.0
Income Taxes (IRPJ/CSLL)	(000USD)	(157,786.1)	0.0	0.0	0.0	(355.5)	(2,244.1)	(9,668.5)	(22,132.5)	(12,055.8)	(23,045.1)	(41,700.7)	(38,764.3)	(7,819.6)	0.0	0.0
NET PROFIT AFTER TAX	(000USD)	318,750.2	0.0	0.0	0.0	(17,419.0)	6,723.5	30,955.6	67,143.3	23,402.5	44,734.6	80,948.3	75,248.4	15,179.2	(8,166.2)	0.0

Table 22-6: Cash Flow

LOM CF Annual Projections		Project Year->	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11
POST-TAX CASH FLOW																
NPAT		318,750.2	0.0	0.0	0.0	(17,419.0)	6,723.5	30,955.6	67,143.3	23,402.5	44,734.6	80,948.3	75,248.4	15,179.2	(8,166.2)	0.0
D&A add back	(000USD)	618,965.4	0.0	0.0	0.0	100,423.0	116,346.8	89,113.6	64,466.2	62,478.6	45,233.6	45,401.2	39,037.8	30,546.9	25,917.6	0.0
OPEX Recoverable taxes	(000USD)	(74,158.3)	0.0	0.0	0.0	(8,496.4)	(9,068.3)	(9,457.8)	(9,863.0)	(10,434.6)	(8,910.2)	(6,935.4)	(5,306.3)	(4,464.6)	(1,221.7)	0.0
PIS/COFINS	"	(74,158.3)	0.0	0.0	0.0	(8,496.4)	(9,068.3)	(9,457.8)	(9,863.0)	(10,434.6)	(8,910.2)	(6,935.4)	(5,306.3)	(4,464.6)	(1,221.7)	0.0
ICMS	"	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Income Tax Incentives & Compensations	(000USD)	198,556.5	0.0	0.0	0.0	33,492.7	15,438.0	15,779.4	23,316.7	18,804.8	21,982.7	30,058.4	26,805.5	8,898.6	3,979.8	0.0
IR SUDAM Incentive	"	87,014.4	0.0	0.0	0.0	196.0	1,237.5	5,331.9	12,205.4	6,648.4	12,708.7	22,996.7	21,377.4	4,312.3	0.0	0.0
Federal Taxes Compensation/ Refund	"	111,542.1	0.0	0.0	0.0	33,296.7	14,200.4	10,447.5	11,111.2	12,156.3	9,274.0	7,061.7	5,428.1	4,586.4	3,979.8	0.0
CAPEX	(000USD)	(610,387.6)	(58,526.9)	(163,603.1)	(225,151.9)	(49,146.6)	(53,819.8)	(10,043.9)	(12,288.0)	(17,230.8)	(4,353.7)	(1,261.7)	(1,212.0)	(1,212.0)	(12,537.1)	0.0
Development CAPEX	(000USD)	(441,781.9)	(53,026.9)	(163,603.1)	(225,151.9)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Net of tax/ CIF	"	(399,715.3)	(47,977.6)	(148,024.7)	(203,712.9)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Non Recoverable Taxes	"	(28,969.7)	(3,477.2)	(10,728.2)	(14,764.3)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Recoverable Taxes	"	(13,096.9)	(1,572.0)	(4,850.1)	(6,674.8)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Capital	(000USD)	(151,163.6)	0.0	0.0	0.0	(49,146.6)	(53,819.8)	(10,043.9)	(12,288.0)	(17,230.8)	(4,353.7)	(1,261.7)	(1,212.0)	(1,212.0)	(595.0)	0.0
Net of tax/ CIF	"	(128,621.8)	0.0	0.0	0.0	(39,558.7)	(47,513.9)	(8,765.9)	(10,492.7)	(14,895.4)	(3,525.9)	(1,126.6)	(1,089.9)	(1,089.9)	(562.9)	0.0
Non Recoverable Taxes	"	(10,224.6)	0.0	0.0	0.0	(3,795.7)	(3,381.8)	(573.3)	(754.4)	(1,003.2)	(520.9)	(65.7)	(57.3)	(57.3)	(14.9)	0.0
Recoverable Taxes	"	(12,317.3)	0.0	0.0	0.0	(5,792.2)	(2,924.1)	(704.7)	(1,040.9)	(1,332.2)	(306.9)	(69.4)	(64.8)	(64.8)	(17.2)	0.0
Mine Closure	"	(29,357.3)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	(29,357.3)	0.0
Salvage Value	"	17,415.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	17,415.2	0.0
Private Royalty NSR Buy-Down exercise	"	(5,500.0)	(5,500.0)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Working Capital Cash Adjustments	(000USD)	0.0	0.0	0.0	0.0	(3,359.4)	25.9	(21.9)	92.9	326.8	(60.8)	(296.0)	(86.9)	205.8	2,145.8	1,028.0
Post Tax Cash Flow	(000USD)	451,726.2	(58,526.9)	(163,603.1)	(225,151.9)	55,494.3	75,646.0	116,324.9	132,868.1	77,347.2	98,626.2	147,914.8	134,486.4	49,153.8	10,118.2	1,028.0
Project NPV@5% (*)	(000 USD)	216,326.3	(57,762.0)	(155,716.5)	(204,869.1)	47,765.7	62,171.1	91,036.8	99,148.2	54,969.2	66,754.1	95,347.2	82,563.0	28,739.2	5,634.2	545.2
IRR	(%)	13.4%														
Payback	(years)	4.4														
Exchange Rate	(BRL/USD)		4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00

Table 22-7: Income Taxes and Offsets

Income Taxes and Offsets		Project Year->	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11
IR/CSLL Calculation Basis		(000USD)	0.0	0.0	0.0	19,759.0	19,037.9	25,304.4	51,780.9	52,391.6	66,551.2	62,968.0	0.0	0.0	0.0	0.0
EBT		(000USD)	476,536.3	0.0	0.0	(17,063.5)	8,967.6	40,624.1	89,275.9	35,458.3	67,779.7	122,649.0	114,012.8	22,998.7	(8,166.2)	0.0
Accrued losses		"														
Opening Balance		"	20,625.8	20,625.8	20,625.8	39,183.0	36,873.2	29,159.8	24,180.2	0.0	0.0	0.0	0.0	0.0	0.0	8,166.2
Generated		"	0.0	0.0	0.0	18,557.2	461.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	8,166.2	0.0
Utilized		"	0.0	0.0	0.0	(448.1)	(2,828.7)	(12,187.2)	(24,180.2)	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Closing Balance		"	20,625.8	20,625.8	20,625.8	38,734.9	36,367.5	24,180.2	0.0	0.0	0.0	0.0	0.0	0.0	8,166.2	8,166.2
Payable Income Taxes (IR/CSLL)		(000USD)	(157,786.1)	0.0	0.0	(355.5)	(2,244.1)	(9,668.5)	(22,132.5)	(12,055.8)	(23,045.1)	(41,700.7)	(38,764.3)	(7,819.6)	0.0	0.0
IR to pay		"	(116,019.2)	0.0	0.0	(261.4)	(1,650.1)	(7,109.2)	(16,273.9)	(8,864.6)	(16,944.9)	(30,662.2)	(28,503.2)	(5,749.7)	0.0	0.0
CSLL to pay		"	(41,766.9)	0.0	0.0	(94.1)	(594.0)	(2,559.3)	(5,858.6)	(3,191.2)	(6,100.2)	(11,038.4)	(10,261.1)	(2,069.9)	0.0	0.0
IR/CSLL INCENTIVES & COMPENSATIONS		(000USD)	87,014.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
SUDAM INCENTIVE		"														
IR SUDAM Incentive		"	87,014.4	0.0	0.0	196.0	1,237.5	5,331.9	12,205.4	6,648.4	12,708.7	22,996.7	21,377.4	4,312.3	0.0	0.0
IR/CSLL to pay after SUDAM Incentive		"	(70,771.7)	0.0	0.0	(159.4)	(1,006.5)	(4,336.6)	(9,927.1)	(5,407.4)	(10,336.4)	(18,704.0)	(17,386.9)	(3,507.3)	0.0	0.0
FEDERAL TAXES TO RECOVER		"														
Credits available from purchases		(000USD)	111,542.1	0.0	0.0	33,296.7	14,200.4	10,447.5	11,111.2	12,156.3	9,274.0	7,061.7	5,428.1	4,586.4	3,979.8	0.0
From OPEX		"														
Generated		(000USD)	74,158.3	0.0	0.0	8,496.4	9,068.3	9,457.8	9,863.0	10,434.6	8,910.2	6,935.4	5,306.3	4,464.6	1,221.7	0.0
Available		"	74,158.3	0.0	0.0	8,496.4	9,068.3	9,457.8	9,863.0	10,434.6	8,910.2	6,935.4	5,306.3	4,464.6	1,221.7	0.0
From CAPEX		"														
Generated		(000USD)	18,424.0	2,211.4	6,822.9	9,389.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Accumulated credits		"	18,424.0	0.0	0.0	18,424.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Available		"	18,424.0	0.0	0.0	18,424.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
From Sustaining Capital		"														
Generated		(000USD)	18,959.7	0.0	0.0	6,376.3	5,132.1	989.7	1,248.2	1,721.8	363.8	126.3	121.8	121.8	2,758.1	0.0
Accumulated credits		"	18,959.7	0.0	0.0	6,376.3	5,132.1	989.7	1,248.2	1,721.8	363.8	126.3	121.8	121.8	2,758.1	0.0
Available		"	18,959.7	0.0	0.0	6,376.3	5,132.1	989.7	1,248.2	1,721.8	363.8	126.3	121.8	121.8	2,758.1	0.0
Federal Taxes Offset/ Refund		"														
Opening Balance		"	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
New credits available		"	111,542.1	0.0	0.0	33,296.7	14,200.4	10,447.5	11,111.2	12,156.3	9,274.0	7,061.7	5,428.1	4,586.4	3,979.8	0.0
Offset w / IR/CSLL		(000USD)	(46,108.1)	0.0	0.0	(159.4)	(1,006.5)	(4,336.6)	(9,927.1)	(5,407.4)	(9,274.0)	(7,061.7)	(5,428.1)	(3,507.3)	0.0	0.0
Tax refund		"	(65,434.0)	0.0	0.0	(33,137.3)	(13,193.9)	(6,110.9)	(1,184.1)	(6,749.0)	0.0	0.0	0.0	(1,079.1)	(3,979.8)	0.0
Closing Balance		"	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total credits offset/ refund		(000USD)	111,542.1	0.0	0.0	33,296.7	14,200.4	10,447.5	11,111.2	12,156.3	9,274.0	7,061.7	5,428.1	4,586.4	3,979.8	0.0
IR/CSLL payable after PIS/COFINS offset		(000USD)	(24,663.6)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

SECTION • 23 ADJACENT PROPERTIES

The Tocantinzinho deposit is located in the Tapajós Gold Province and there are a number of gold-focused international exploration and mining companies within a 100 km radius from the deposit. Additionally, the interest in the exploration potential for copper mineralization has recently increased in the region as evidenced by the staking of large exploration claims by base metal mining companies (Figure 23-1).

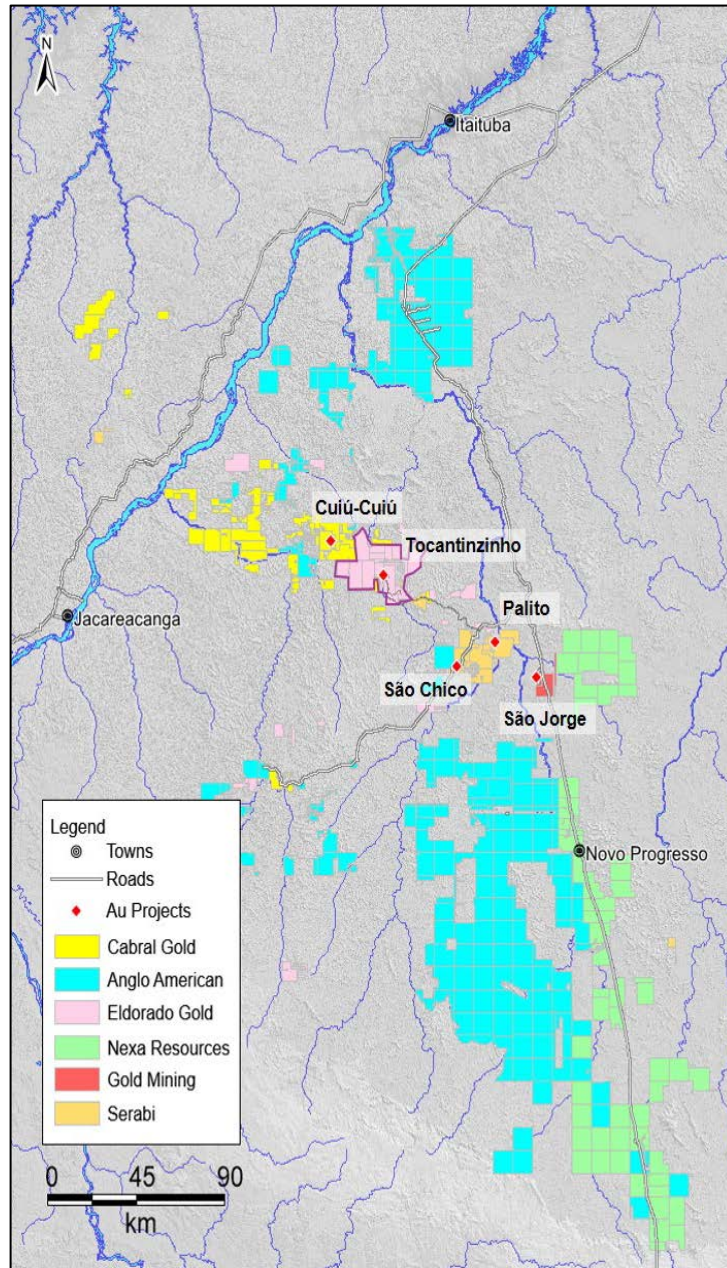


Figure 23-1: Adjacent Properties

Estimates for adjacent properties around the Tocantinzinho Project were obtained from publically available technical reports which have not been verified by Eldorado or the qualified persons for this report. The presence of significant mineralization on these properties is not necessarily indicative of similar mineralization on the Tocantinzinho Project.

23.1 CABRAL GOLD INC.

Cabral Gold Inc. ("Cabral") holds the Cuiú-Cuiú Project located approximately 32 km WNW in a straight line from Tocantinzinho. Cuiú-Cuiú is known regionally as a site of significant artisanal mining activity, and modern mineral exploration activity has been carried out by Cabral and predecessor company Magellan Minerals Ltd since 2005.

Cabral controls 71,793 ha in exploration permits and applications associated with the Cuiú-Cuiú Project, where 48,025 m of drilling were completed. The project holds a NI 43-101 compliant mineral resource estimate of 5,886 kt @ 0.90 g/t for 171 koz Au in the indicated category, as well as 19,520 kt @ 1.24 g/t for 776 koz Au in the inferred category, effective December 31, 2017. These figures combine open pit resources reported at 0.35 g/t Au cut-off and underground resources reported at 1.3 g/t Au cut-off (Micon, 2017).

23.2 SERABI GOLD PLC

Serabi Gold Plc ("Serabi") holds the Palito Mining Complex located approximately 63 km SE in a straight line from Tocantinzinho. The complex includes the Palito and the São Chico mines, both high grade, narrow vein underground currently in operation. The Mining Complex produced ~ 210 koz Au intermittently between 2005 and mid-2017 under Serabi's ownership (SRK, 2017), as well as 37.1 koz Au in 2018.

Serabi controls 56,631 ha in exploration permits and applications associated with the Palito Mining Complex. The São Chico mine holds NI 43-101 compliant proven and probable mineral reserves of 90 kt @ 8.43 g/t for 24 koz Au, and the Palito mine holds a NI 43-101 compliant proven and probable mineral reserves of 613 kt @ 7.99 g/t Au and 0.37% Cu, for 157 koz Au and 2.3 kt Cu, both effective June 30, 2017 (SRK, 2017).

23.3 GOLD MINING INC.

Gold Mining Inc. ("Gold Mining") holds the São Jorge Project located approximately 93 km SE in a straight line from Tocantinzinho. Some artisanal mining gold production is reported from São Jorge, but modern mineral exploration activity has been carried out by Gold Mining, Brazilian Gold Corporation and Talon Resources since 2005 (Coffey, 2013).

Gold Mining controls 47,646 ha in exploration permits and applications associated with the São Jorge Project, where 37,154 m of drilling were completed. The project holds a NI 43-101 compliant mineral resource estimate of 14,420 kt @ 1.54 g/t for 715 koz Au in the indicated category, as well as 28,190 kt @ 1.14 g/t for 1,035 koz Au in the inferred category, both reported at a 0.3 g/t Au cut-off grade and effective November 22, 2013 (Coffey, 2013).

23.4 ANGLO AMERICAN PLC

Anglo American PLC has amassed a very large package of approximately 1,722,000 ha in exploration permits and applications around Tocantinzinho and in Pará State (National Mining Agency public records). The vast majority of this package lists copper as mineral substance of interest (89% by area), and most of the ground was consolidated in the last two years (65%).

23.5 NEXA RESOURCES S.A.

Nexa Resources S.A. ("Nexa") has amassed a large package of approximately 690,000 ha in exploration permits and applications SE of Tocantinzinho and in Pará State (National Mining Agency public records). Nexa's exploration focus in the region is clearly on copper (96% by area), and most of the ground was consolidated in the last two years (78%).

SECTION • 24

OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data and information provided in this section.

SECTION • 25**INTERPRETATION AND
CONCLUSIONS**

25.1 SUMMARY

The Project has been investigated and optimized over several years and this technical report provides a summary of the results and findings. This report is based on Eldorado's familiarity with the site and its involvement in the Project since 2008 and with Brazauro since 2003.

The level of the study for the areas reviewed is considered to be consistent with feasibility study level for resource development projects. Based on the review of the technical aspects and its respective economics, this project is feasible and should advance to execution stage.

**25.1 GEOLOGY, DEPOSIT, EXPLORATION, DRILLING, SAMPLE
PREPARATION AND DATA VERIFICATION**

Geological controls on mineralization at Tocantinzinho are well understood and supported by drilling and sampling which were professionally managed and in line with modern industry standards.

The work completed to date is sufficient to characterize the mineral deposit boundaries and mineralization. Eldorado concluded that the data supporting the Tocantinzinho Project resource work is sufficiently free of error to be adequate for estimation and disclosure of exploration results.

Soil sampling has been an effective method to delineate zones of gold and copper anomalies and the primary tool to generate drill targets. However, only approximately 40% of the total exploration package has been covered by soil sampling. Initial exploration drilling performed in the soil anomaly target at Santa Patricia shows an encouraging trend of increasing copper grades at depth which warrants further drilling. Eldorado considers further exploration success at Santa Patricia and associated soil anomaly to be an opportunity to the Project.

25.2 MINERAL PROCESSING AND METALLURGICAL TESTING

The metallurgical testing was sufficient to design the process to recover the gold at Tocantinzinho. Additional testing could potentially reduce the cyanide recovery thickener since the settling rate used was conservative. Also, the oxygen plant may be reduced in capacity or eliminated since test results show a relatively small difference in gold recovery from the existing test results and also cyanide destruction can use air instead of oxygen. With additional testwork specific ICU recoveries may increase.

Even with extended metallurgical test programs, there is always a potential risk that the samples might not represent the general characteristics of the ore. The test program was considered quite adequate for most of the mineral processing stages investigated but this risk still exists. Variability in ore can cause high mass pull in flotation circuit and negatively impact leaching and CIP residence times and potentially reduce overall gold recovery. The comminution testing was conducted with a small number of samples and may compromise the comminution circuit design.

25.3 MINERAL RESOURCE ESTIMATES

The mineral resource estimates for the Tocantinzinho Project were made from a 3D block model constrained by geological domains and a grade shell. Gold was estimated in a single estimation domain using ordinary kriging and commercial software. The gold domain shows the desirable attributes of a low nugget (20%) and an adequate coefficient of variation (1.45), which is a reflection of the disseminated nature of mineralization observed at the drill sample scale.

Block model estimates were checked for smoothing using a change-of-support method as well as validated for bias using comparison with nearest-neighbour estimates, swath plots and visual validation. Estimates were found to be free of bias and excessive smoothing.

The mineral resources of the Tocantinzinho deposit were classified using logic consistent with the Canadian Institute of Mining (CIM) definitions referred to in NI 43-101. The mineralization of the project satisfies sufficient criteria to be classified into measured, indicated, and inferred mineral resource categories. A test of reasonableness for the expectation of economic extraction was made on the Tocantinzinho mineral resources by developing a series of open pit designs based on optimal operational parameters and gold price assumptions. Those pit designs enveloped most of the measured and indicated mineral resources thus demonstrating the economic reasonableness test for the estimate and reporting cut-off grade of the Tocantinzinho mineral resources.

25.4 MINERAL RESERVE ESTIMATES

The mineral reserves have been estimated using methods consistent with the CIM definitions referred to in NI 43-101. It is the opinion of the author that the information and analysis provided in this report is considered sufficient for reporting mineral reserves for this project.

25.5 MINING METHODS

A mine plan has been developed to extract the mineral reserve by conventional open pit mining methods. This plan addresses the various material types that will be encountered and the required production rates to provide consistent mill feed to the processing plant. Opportunities exist to optimize wall slopes in the Phase 2 pit once operating faces have been developed and exposed in the Phase 1 pit.

A suitable quarry location has been identified on site which is in close proximity to the pit. Its design was based on preliminary information and there is an opportunity to optimize the design to reduce overburden removal and related costs for construction rock.

25.6 RECOVERY METHODS

Silver has not been quantified in the resource however metallurgical test work has indicated that there will be a significant amount of silver extracted with the gold. The silver represents a substantial opportunity to increase the revenues from the Tocantinzinho mine.

Additional testwork will be required to see if a water treatment plant is required to treat the reclaim water from the CIP ponds. As this equipment is not required until the end of the first year of operation, there is time to do the additional testwork once the project starts to be executed. If future testwork shows better results, there will be an opportunity to remove US\$1.6 M from Year 1 of sustaining capital.

25.7 PROJECT INFRASTRUCTURE

The access road is used by others and depending on this use and weather, the road will require increased work to make it suitable for the Project.

A formal agreement with the electric utility is yet to be done however a different unit cost based on the volume purchased could be negotiated.

25.7.1 Tailings

The design of the flotation tailings impoundment and the two effluent ponds meet recent Brazilian code and are sufficient for the process and the life of mine. Extensive investigation on the foundation and construction materials was completed and the stability and seepage analyses show adequate factors of safety in the design.

25.8 MARKET STUDIES AND CONTRACTS

Market studies nor contracts have been concluded for the gold doré however gold is generally sold on the open market.

25.9 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Delays in the project will require additional work to renew the existing permits.

Legislation related to tailings impoundments is under review in Brazil. This could affect the dam design as well as infrastructure downstream of the dam.

25.10 CAPITAL AND OPERATING COSTS

Costs are based on values as of Q1 2019. Escalation and changes in the market could affect the costs. Exchange rate was based on BRL4.00 per US\$1.00 which could change.

25.11 ECONOMIC ANALYSIS

The economic analysis is sensitive to the exchange rate and the gold price.

SECTION • 26 RECOMMENDATIONS

It is recommended that the Project be executed in accordance to the plan developed to support this technical report. The following includes specific recommendations for future phases of the Project.

26.1 EXPLORATION

The soil sampling campaigns conducted at the Project site do not provide full coverage over the exploration package and it is recommended that they should be extended in prospective areas. Initially, soil samples should be collected at 50 m intervals, spaced 400 m apart in the same orientation as the existing lines to extend the coverage north and west of Sta Patricia, KRB and Tocantinzinho. The cost of such campaign is estimated at US\$300,000. Progressive infill between the soil lines would be contingent on positive results in this initial phase.

Encouraging drilling results at Sta Patricia warrant a follow-up drilling campaign. A total of 5,000 m of diamond drilling are recommended and estimated to cost US\$1.00 M (including direct drilling costs, sample assaying and associated labour). About a third of the drilling should be allocated to the testing of the deeper mineralization at the southern end of the copper soil anomaly. Second third should be allocated to the testing of the northern end of the copper soil anomaly, and the balance should be allocated to the testing of miscellaneous gold soil anomalies. Further drilling would be contingent on positive results from this initial drilling campaign.

26.2 MINERAL PROCESSING AND METALLURGICAL TESTING

It is recommended to perform additional metallurgical tests to confirm the design basis, specifically for the comminution and ICU leaching.

To determine if a water treatment plant is required in Year 1, some testwork should be done to further understand the quality of the CIP pond reclaim water.

26.3 MINERAL RESERVE ESTIMATES

Mineral reserves are limited by economic parameters. Measured and indicated resources have been used to report proven and probable reserves. The inferred resources within the pit design are insignificant and measured and indicated resources exist outside of the current pit designs. Future pit limits optimization should be undertaken if gold prices for reserve estimation can be elevated.

26.4 MINING METHODS

A preliminary design for a hard rock quarry has been made to provide construction materials for the site and open pit mine development. The design of this quarry may be further optimized with some additional drilling and geological modelling.

26.5 RECOVERY METHODS

A high-level trade-off study could be done between the different leaching and carbon adsorption configurations as the CIP retention time could be reduced. Also, retention times for the rougher and scavenger flotation may be reduced.

26.6 PROJECT INFRASTRUCTURE

The tailings disposal plan was based on the production schedule, assuming a constant dry apparent density of 1.325 t/m³ indicated by the testwork. As the initial densities may reach lower values, it is important to evaluate the density during operation. Consistent lower densities will result in the requirement to raise the dam earlier than planned so it would be worthwhile to monitor and adjust the sustaining capital timing as needed.

26.7 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Permits should be maintained and renewed when required. The interaction with the community should continue to facilitate a smooth transition to the execution of the Project.

SECTION • 27 REFERENCES

The following documents were referenced in the report:

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SECTION • 28**CERTIFICATES OF AUTHORS AND
DATE AND SIGNATURE PAGE**

DATE AND SIGNATURE PAGE

The effective date of this report entitled “Technical Report, Tocantinzinho Project, Brazil” is June 21, 2019. It has been prepared for Eldorado Gold Corporation by David Sutherland, P.Eng., Rafael Jaude Gradim, P.Geo., John Nilsson, P.Eng., Persio Pellegrini Rosario, P. Eng., William McKenzie, P. Eng. and Paulo Ricardo Behrens da Franca, AusIMM; each of whom are qualified persons as defined by NI 43-101.

Signed the 21st day of June 2019.

“Signed and Sealed”

David Sutherland

David Sutherland, P. Eng.

“Signed and Sealed”

Rafael Jaude Gradim

Rafael Jaude Gradim, P. Geo.

“Signed”

Paulo Ricardo Behrens da Franca

Paulo Ricardo Behrens da Franca, AusIMM

“Signed”

Persio Pellegrini Rosario

Persio Pellegrini Rosario, P. Eng.

“Signed and Sealed”

John Nilsson

John Nilsson, P. Eng.

“Signed and Sealed”

William McKenzie

William McKenzie, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

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I, David Sutherland, am a Professional Engineer, employed as Project Manager, of Eldorado Gold Corporation located at 1188 Bentall 5, 550 Burrard St., Vancouver in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019.

I am a member of the Engineers & Geoscientists of British Columbia. I graduated from the Lakehead University with a Bachelor of Science (Physics) in 2003 and a Bachelor of Engineering (Mechanical) in 2005.

I have practiced my profession continuously since 2005.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Tocantinzinho Project on numerous occasions with my most recent visit occurring on March 29-30, 2017.

I was responsible for coordinating the preparation of the technical report. I am responsible for the preparation or supervising the preparation of items 1, 2, 3, 4, 5, 6, 18 (excluding tailings), 19, 20, 22, 24, 25, 26, and 27 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1 and the items for which I am responsible in this report entitled, *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 21st day of June 2019.

“Signed and Sealed”

David Sutherland

David Sutherland, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

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I, Rafael Jaude Gradim, am a Professional Geologist, employed as Manager, Corporate Development – Technical Evaluations, of Eldorado Gold Corporation located at 1188 Bentall 5, 550 Burrard St., Vancouver in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019.

I am a member of the Engineers & Geoscientists of British Columbia. I graduated from the Federal University of Minas Gerais with a Geology Degree in 2004 and subsequently a Master of Science degree in Structural Geology through the Federal University of Ouro Preto in 2005.

I have practiced my profession continuously since 2005.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Tocantinzinho Project once occurring on February 21 to 23, 2019.

I was responsible for reviewing matters related to the geological data and directing the mineral resource estimation and classification work for the Tocantinzinho Project in Brazil. I am responsible for the preparation or supervising the preparation of items 7, 8, 9,10,11,12, 14 and 23 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am not independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1 and the items for which I am responsible in this report entitled, *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 21st day of June 2019.

“Signed and Sealed”

Rafael Jaude Gradim

Rafael Jaude Gradim, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

John Nilsson, P. Eng.
20263 Mountain Place
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Email: jnilsson@shaw.ca

I, John Nilsson, am a Professional Engineer, employed as President, of Nilsson Mine Services Ltd. and residing at 20263 Mountain Place in the city of Pitt Meadows in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated from Queen's University with a Bachelor of Science degree in geology in 1977 and subsequently a Master of Science degree through the Department of Mining Engineering in 1990.

I have practiced my profession in geology and mining continuously since 1977 and have worked on mining related precious and base metal projects in North America, Central America, South America, Africa, Europe and Asia.

As a result of my experience and qualifications, I am a qualified person as defined in National Instrument 43-101.

I have visited the Tocantinzinho Project site on February 21 to 23, 2019.

I was responsible for developing the mine plan for the Tocantinzinho Project in Brazil. I am responsible for the preparation or supervising the preparation of Sections 15, 16 and 21.2.2 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1 and the items for which I am responsible in this report entitled, *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 21st day of June 2019.

"Signed and Sealed"

John Nilsson

John Nilsson, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Persio Pellegrini Rosario, MSc, PhD, P.Eng.
Hatch Ltd.,
Suite 400, 1066 West Hastings St., Vancouver, BC V6E 3X2
Tel: 604 638 7019
Email: persio.rosario@hatch.com

I, Persio Pellegrini Rosario, P.Eng., am employed as Director-Comminution with Hatch Ltd.

This certificate applies to the technical report entitled *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019.

I am a Professional Mineral Processing Engineer. I graduated from the University Mackenzie (SP, Brazil) with a Bachelor in Applied Sciences degree in Mechanical Engineering in 1989, a Master in Applied Sciences degree in Mineral Processing Engineering in 2003, and a Doctor of Philosophy in Mineral Process Engineering in 2010.

I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration #32355).

I have practiced my profession for more than 25 years. I have been directly involved in the design and optimization of mineral processing plants for precious and base metals in Canada, Brazil, Mexico, Honduras, Peru, Argentina, Chile, Panama, Russia, and the United States. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43-101.

I have visited the Tocantinzinho Project on February 21 to 23, 2019.

I was responsible for reviewing matters related to the metallurgical data for the Tocantinzinho Project in Brazil. I am responsible for the preparation or supervising the preparation of items 13, 17, 21.2.3 in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1 and the items for which I am responsible in this report entitled, *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 21st day of June 2019.

“Signed and Sealed”

Persio Pellegrini Rosario

Persio Pellegrini Rosario, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

William McKenzie, P. Eng.
21160 – 18th Avenue,
Langley, BC

Email: wcmckenzie@hotmail.com

I, William McKenzie, am a Professional Engineer, employed as President, of Global Project Management Corporation and residing at 21160 – 18th Avenue, Langley in the Province of British Columbia.

This certificate applies to the technical report entitled *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019.

I am a member of the Engineers & Geoscientists British Columbia (formerly the Association of Professional Engineers and Geoscientists of British Columbia). I graduated from University of British Columbia with a Bachelor of Science degree in mechanical engineering in 1972.

I have practiced my profession in geology and mining continuously since 1972 and have worked on mining related precious and base metal projects in North America, Central America, South America, Africa, Europe, Australia and Asia.

As a result of my experience and qualifications, I am a qualified person as defined in National Instrument 43-101.

I have visited the Tocantinzinho Project site on March 22 to 23, 2018.

I was responsible for verifying the capital cost estimate and the operating costs for general and administration for the Tocantinzinho Project in Brazil. I am responsible for the preparation or supervising the preparation of Sections 21.1, 21.2.1 and 21.2.4 of the report.

I have not had prior involvement with the property that is the subject of this technical report.

I am independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1 and the items for which I am responsible in this report entitled, *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 21st day of June 2019.

“Signed and Sealed”

William McKenzie

William McKenzie, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Paulo Ricardo Behrens da Franca, Civil and Geological Engineer
Rua Desembargador Penna 131
Minas Gerais, 30.320-220, Brazil
Tel: (+55-31) 99941-4339
Email: pfranca@fzprojetos.com.br

I, Paulo Ricardo Behrens da Franca, am a Professional Civil and Geological Engineer, employed as President, of F&Z Consultoria e Projetos, Brazil and residing at Rua Desembargador Penna 131, Belo Horizonte, Minas Gerais, 30.320-220, Brazil.

This certificate applies to the technical report entitled *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st 2019.

I am a practicing geotechnical engineer and a member of the Australasian Institute of Mining and Metallurgy (AusIMM, No. 311775). I also hold an AusIMM Chartered Professional Accreditation under the Discipline of Geotechnics. I am a graduate of the Universidade Federal de Ouro Preto (Brazil) and hold a Geological Engineer title (1986), a graduate of the Escola de Engenharia Kennedy (Brazil) and hold a Civil Engineer title (1989). I also hold a Master of Science degree in Mining Engineering, from Queen's University (Canada), obtained in 1995.

I have practiced my profession continuously since 1986.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101.

I have visited the Tocantinzinho Project on February 21 to 23, 2019.

I was responsible for the tailings section of the technical report. I am responsible for the preparation or supervising the preparation of items 18.4 related to tailings in the technical report.

I have not had prior involvement with the property that is the subject of this technical report.

I am independent of Eldorado Gold Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.

I have read National Instrument 43-101 and Form 43-101F1 and the items for which I am responsible in this report entitled, *Technical Report, Tocantinzinho Project, Brazil*, with an effective date of June 21st, 2019, has been prepared in compliance with same.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the items of the technical report that I was responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading

Dated at Vancouver, British Columbia, this 21st day of June, 2019.

"Signed and Sealed"

Paulo Ricardo Behrens da Franca

Paulo Ricardo Behrens da Franca, P.Eng.