A DRA Global Group Company

NI 43-101
PRELIMINARY ECONOMIC ASSESSMENT OF THE ADUMBI DEPOSIT IN THE DEMOCRATIC REPUBLIC OF THE CONGO

Prepared for
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CAUTIONARY NOTES

This report contains forward-looking information which addresses activities, events or developments that are believed, expected or anticipated will or may occur in the future (including, without limitation, statements regarding estimates and/or assumptions in respect of production, revenue, cash flow and costs, estimated project economics, upside potential at Adumbi, mineral resource estimates, potential underground mineral resources, potential mineralisation, potential gold discoveries, drill targets, potential mineral resource increases, estimated exploration costs, exploration results, and future exploration and development plans) are forward-looking information. This forward-looking information reflects current expectations or beliefs based on information currently available and is subject to a number of risks and uncertainties that may cause the actual results to differ materially from those discussed in the forward-looking information. Even if such actual results are realised or substantially realised, there can be no assurance that they will have the expected consequences to, or effects. Factors that could cause actual results or events to differ materially from current expectations include, among other things, uncertainty of estimates of capital and operating costs, production estimates and estimated economic return, the possibility that actual circumstances will differ from the estimates and assumptions used in the Adumbi Preliminary Economic Assessment set out herein, the possibility that future exploration (including drilling) or development results will not be consistent with the expectations of Loncor Gold Inc. (“Loncor” or the “Company”), the possibility that drilling or development programmes will be delayed, activities of the Company may be adversely impacted by the continued spread of the widespread outbreak of respiratory illness caused by a novel strain of the coronavirus (COVID-19), including the ability of the Company to secure additional financing, risks related to the exploration stage of the Company's properties, uncertainties relating to the availability and costs of financing needed in the future, failure to establish estimated mineral resources (the Company’s mineral resource figures are estimates and no assurances can be given that the indicated levels of gold will be produced), changes in world gold markets or equity markets, political developments in the Democratic Republic of the Congo, gold recoveries being less than those indicated by the metallurgical test work carried out to date (there can be no assurance that gold recoveries in small-scale laboratory tests will be duplicated in large tests under on-site conditions or during production), fluctuations in currency exchange rates, inflation, changes to regulations affecting the Company's activities, delays in obtaining or failure to obtain required project approvals, the uncertainties involved in interpreting drilling results and other geological data and the other risks disclosed under the heading “Risk Factors” and elsewhere in the Company’s annual report on Form 20-F dated March 31, 2021, filed on SEDAR at www.sedar.com and EDGAR at www.sec.gov. Forward-looking information speaks only as of the date on which it is provided and, except as may be required by applicable securities laws, any intent or obligation to update any forward-looking information, whether as a result of new information, future events or results or otherwise, is hereby disclaimed. Although the Company believes that the assumptions inherent in the forward-looking information are reasonable, forward-looking information is not a guarantee of future performance and accordingly undue reliance should not be put on such information due to the inherent uncertainty therein.

The mineral resource figures referred to in this report are estimates and no assurances can be given that the indicated levels of gold will be produced. Such estimates are expressions of judgment based on knowledge, mining experience, analysis of drilling results and industry practices. Valid estimates made at a given time may significantly change when new information becomes available. While it is believed that the mineral resource estimates included in this report are well established, by their nature mineral resource estimates are imprecise and depend, to a certain extent, upon statistical inferences which may ultimately prove unreliable. If such estimates are inaccurate or are reduced in the future, this could have a material adverse impact on the Company.

Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that mineral resources can be upgraded to mineral reserves through continued exploration.

Due to the uncertainty that may be attached to inferred mineral resources, it cannot be assumed that all or any part of an inferred mineral resource will be upgraded to an indicated or measured mineral resource as a result of continued exploration. Confidence in the estimate is insufficient to allow meaningful application of the technical and economic parameters to enable an evaluation of economic viability worthy of public disclosure (except in certain limited circumstances). Inferred mineral resources are excluded from estimates forming the basis of a feasibility study.

This report includes certain terms or performance measures commonly used in the mining industry that are not defined under International Financial Reporting Standards (IFRS), including cash costs and AISC per payable ounce of gold sold. Non-GAAP measures do not have any standardised meaning prescribed under IFRS and, therefore, they may not be comparable to similar measures employed by other companies. In addition to conventional measures prepared in accordance with IFRS, certain investors use this information to evaluate performance. The data presented is intended to provide additional information and should not be considered in isolation or as a substitute for measures of performance prepared in accordance with IFRS.

National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) is a rule of the Canadian Securities Administrators which establishes standards for all public disclosure an issuer makes of scientific and technical information concerning mineral projects. Unless otherwise indicated, all resource estimates contained in this report have been prepared in accordance with NI 43-101 and the Canadian Institute of Mining, Metallurgy and Petroleum Classification System. These standards differ from the requirements of the U.S. Securities and Exchange Commission, and resource information contained in this report may not be comparable to similar information disclosed by U.S. companies.

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Unit	Description
°	degree
'	minutes
%	percentage
% m/m	percentage mass by mass
% w/v	percentage weight by volume
% w/w	percentage weight by weight
µg	microgram
µm	micrometre (micron)
µS	microsiemens
°C	degree Celsius
a	annum
A	ampere
bbl	barrels
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
C$	Canadian dollar
d	day
dB	decibel
dia.	diameter
dmt	dry metric tonne
dwt	deadweight tonne
ekW	generator output rating in kilowatts
°F	degree Fahrenheit
g	gram
G	giga (billion)
g/cm^3	gram per cubic centimetre
g/L	gram per litre
g/t	gram per metric tonne
Ga	billion years (10^9 years)
h	hour
ha	hectare
Hz	hertz
J	joule

Unit	Description
k	kilo (thousand)
kbar	kilobar of pressure
kg	kilogram
km	kilometre
km^2	square kilometre
koz	thousand troy ounces
kPa	kilopascal
kt	thousand metric tonnes
kt/a	thousand metric tonnes per annum
kV	kilovolt
kVA	kilovolt ampere
kW	kilowatt
kWe	kilowatt-electric
kWh	kilowatt hour
L	litre
L/s	litre per second
lb	pound
m	metre
m^2	square metre
m^3	cubic metre
m/s	Metre per second
MASL	metre above sea level
min	minute
mm	millimetre
Mm^3	million cubic metres
Moz	million troy ounces
MPa	megapascal
Mt	million metric tonnes
MVA	megavolt amperes
MW	megawatt
MWh	Megawatt hour
N	newton
Nm	newton metre
NTU	Nephelometric Turbidity Unit
oz	troy ounce
Pa	pascal
Pa s	pascal second

Unit	Description
ppb	part per billion
ppm	part per million
RL	relative elevation
s	second
s^{-1}	reciprocal second
t	metric tonne
t/a	metric tonne per annum
t/m^3	metric tonne per cubic metre
US$	United States dollar
V	volt
W	watt
wmt	wet metric tonne
wt%	weight percentage

It is noted that, throughout the report, table columns might not add up due to rounding.

LIST OF ABBREVIATIONS

Abbreviation	Description
AA	Atomic Absorption
AARL	Anglo American Research Laboratory
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic absorption spectroscopy
ABA	Acid-base accounting
ACSA	Albite-Carbonate-Silica Alteration
Ai	Abrasion index
AISC	All-in sustaining cost
ALS	ALS Laboratories
AMTEC	AMTEC Laboratories
ANSUL	Fire Suppression Supply Company
ARD	Acid rock drainage
ASM	Artisanal and small-scale mining
AusIMM	Australasian Institute of Mining and Metallurgy
BBWi	Bond ball work index
BESS	Battery energy storage system
BFA	Bench face angle
BHID	Borehole identification
BIF	Banded Iron Formation
BLEG	Bulk leach extractable gold
BM	Block model
BMP	Biodiversity Management Plan
BOCO	Base of complete oxidation
BOQ	Bill of quantities
BRGM	French Geological Survey
BRWi	Bond rod work index
BRT	Bottle roll test
C&I	Control and instrumentation
CA	Confidentiality Agreement
CAGR	Compound annual growth rate
CAMI	Cadastre Minier
CAPEX	Capital cost
CBS	Carbonaceous schist
CCTV	Closed-circuit television
CDF	Co-disposal facility

Abbreviation	Description
CHK	Central Hospital Kibali
CIL	Carbon in leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon in pulp
CMT	Constant money terms
COS	Coarse ore stockpile
CP	Competent Person
CPE	Standing Committee of Evaluation
CRM	Certified reference material
CSR	Community Social Relations
CSS	Closed side setting
CTSF	Cyanide tailings storage facility
COV	Coefficient of variation
CWi	Crushability work index
DC	Direct Current
DCF	Discounted cash flow
DD/DDH	Diamond drillhole
DFS	Definitive Feasibility Study
DMR	South African Department of Mineral Resources
DPEM	Direction de Protection de l'Environnement Minier
DRC	Democratic Republic of the Congo
DTM	Digital terrain model
DTP	DTP Company, subsidiary of Bouygues
E&I	Electrical and instrumentation
E, C&I	Electrical, control and instrumentation
EAP	Environmental Adjustment Plan
EC	European Commission
ECSA	Engineering Council of South Africa
EDA	Estimation Data Analysis
EGRG	Extended gravity recoverable gold
EIA	Environmental Impact Assessment
EIS	Environmental Impact Study
EM	Electromagnetic
EMPP	Environmental Management Plan of the Project
EMS	Environmental Management System
EOH	End of hole

Abbreviation	Description
EOM	End of month
EOY	End of year
EP	Equator Principle
EPC	Engineering, procurement and construction
EPCM	Engineering, procurement and construction management
Epoch	Epoch Resources
ERM	Environmental Resource Management
ESG	Environmental, Social and Governance
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Plan
EU	European Union
EW	Electrowinning
FA	Fire assay
FCF	Free cash flow
FEL	Front-end loader
FGO	Full grade ore
FoS	Factor of safety
FOS	Fine ore stockpile
FR	Fresh rock
FRM	Société Internationale Forestière et Minière du Congo
FS	Feasibility Study
FTSF	Flotation tailings storage facility
G&A	General and administration
GA	General arrangement
GC	Grade control
GDP	Gross domestic product
GHG	Greenhouse Gas Emissions
GIIP	Good international industry practice
GIS	Geographic information system
GISTM	Global Industry Standard on Tailings Management
GPS	Global positioning system
GT	Grade tonnage
HAS	High arsenic
HAZOP	Hazard and operability
HDPE	High-density polyethylene
HEP	Hydroelectric power

Abbreviation	Description
HFO	Heavy fuel oil
HQ	Core size (63.3 mm)
HR	High recovery
HR	Human resources
HW	Hanging wall
HY	High yield
I/O	Input/output
IBC	Intermediate bulk container
ICMC	International Cyanide Management Code
ICP	Inductively coupled plasma
ICP-AES	Inductively coupled plasma – atomic emission spectroscopy
ICP-MS	Inductively coupled plasma – mass spectrometry
ID	Inverse distance
IEC	International Electrotechnical Commission
IFC	International Finance Corporation
ILR	Intensive leach reactor
IP	Induced polarisation
IRR	Internal rate of return
ISO	International Organization for Standardization
IT	Information technology
IUCN	International Union for Conservation of Nature
JORC	Joint Ore Reserves Committee (of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and the Minerals Council of Australia)
JV	Joint Venture
KCD	Karagba Chauffeur Durba Orebody
KE	Kriging Efficiency
Kilo	Kilo Goldmines Ltd
KMS	Kibali Mining Services
KZ	KZ Structure
LAN	Local area network
LAS	Low arsenic
LBMA	London Bullion Market Association
LCT	Locked cycle test
LiDAR	Light detection and ranging
LIMS	Laboratory Information Management System
LME	London Metal Exchange

Abbreviation	Description
LOM	Life of mine
Loncor	Loncor Gold Inc.
LR	Low recovery
LV	Low voltage
MAS	Medium arsenic
MBA	Master of Business Administration
MCC	Motor control centre
MCE	Maximum Considered Earthquake
MCF	Mine Call Factor
MCP	Meta-Conglomerate Package
MEL	Mechanical equipment list
MG	Medium grade
MIA	Mining industrial area
MIBC	Methyl isobutyl carbinol
MIMMM	Member of the Institute of Materials, Minerals and Mining
Minecon	Minecon Resources and Services Limited
MO	Marginal Ore
MPS(P)	Mineral Processing Separating (Pumping)
MRMM	Mining Rock Mass Model
MRP	Mitigation and Rehabilitation Plan
MSI 3D	Mine Surveying International Limited
MSO	Minable Stope Optimiser (Datamine-based software for underground stope design)
MSS	Metasediments
MTO	Material take-off
MV	Medium voltage
NAG	Net acid generating
NGL	Natural ground level
NGO	Non-Governmental Organisation
NI 43-101	Canadian Securities Administrators’ National Instrument (NI) 43-101
NPV	Net present value
NQ	Core size (47.6 mm)
OC	Open cast
ODBC	Open database connectivity
OEM	Original equipment manufacturer
OFS	Optimised Feasibility Study
OK	Ordinary Kriging

Abbreviation	Description
OKIMO	DRC Governmental Entity
OMC	Orway Mineral Consultants
OP	Open Pit
OPEX	Operating cost
OREAS	Ore Research and Exploration Pty Ltd (CRM Manufacturer)
ORP	Operational Readiness Plan
P&G	Preliminary and general
P&ID	Piping and instrumentation diagram
PAG	Potentially acid generating
PAS	Process automation system
PAX	Potassium amyl xanthate
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PGA	Peak ground acceleration
PLC	Programmable logic controller
PM	Particulate matter
PQ	Core size (85.0 mm)
PSA	Pressure swing adsorption
PSD	Particle size distribution
PV	Photovoltaic
Q-Q	Quantile-quantile
QA	Quality assurance
QC	Quality control
QCS	Quartz carbonate schist
QG	QG Australia Pty Ltd
QP	Qualified Person
R&R	Rest and relaxation
RAB	Rotary air blasted
RAP	Resettlement Action Plan
RC	Reverse circulation
RD	Relative density
RED	Reducing
RES	Resource domain
RFBP	Request for budget pricing
RMCA	Royal Museum for Central Africa
RMR/MRMR	Rock mass rating/mining rock mass rating

Abbreviation	Description
ROM	Run of mine
RP	Replaced rock
RPA	Roscoe Postle Associates Inc.
RQD	Rock quality designation
RWD	Return water dam
RWG	Resettlement working group
SAG	Semi-autogenous grinding
SAIMM	Southern African Institute of Mining and Metallurgy
SAMREC	South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves
SAP	Saprolite or German Company
SCADA	Supervisory control and data acquisition
SCH	Schist
SEDAR	System for Electronic Document Analysis and Retrieval
SENET	NEW SENET (Pty) Ltd
SG	Specific gravity
SGS	SGS laboratories
SHE	Safety Health Environmental
SLTO	Social Licence to Operate
SMBS	Sodium metabisulphite
SMC	SAG Mill Comminution
SMPP	Structural, mechanical, plate work and piping
SMU	Selective mining unit
SOKIMO	Société Minière de Kilo-Moto
SOP	Standard operating procedure
SOX	Sarbanes Oxley Act
SP	Stockpiles
SQL	Structured Query Language Database
SR	Slope of Regression
SRK	Steffen Roberts and Kirsten, Engineering Company
STD/StdDev	Standard deviation
SWATH	One-dimensional analysis graph in a specific direction of interest
TD	Tailings dam
TDS	Total dissolved solids
TOFR	Top of fresh rock
TSF	Tailings storage facility
TSS	Total suspended solids

Abbreviation	Description
TSX	Toronto Stock Exchange
UC	Uniform conditioning
UCS	Uniaxial compressive strength
UFG	Ultrafine grinding
UPS	Uninterruptible power supply
UTM	Universal Transverse Mercator
VG	Visible gold
VGF	Vibrating grizzly feeder
VSD	Variable-speed drive
WAD	Weak acid dissociable (cyanide)
WAIMM	West African Institute of Mining, Metallurgy and Petroleum
WHO	World Health Organisation
WMF	Waste management facility
XC	Crosscut
XRF	X-ray fluorescence

1 SUMMARY

1.1 INTRODUCTION

This National Instrument 43-101 (NI 43-101) Technical Report was compiled by Minecon Resources and Services Limited (Minecon) and NEW SENET (Pty) Ltd (SENET) for Loncor Gold Inc. (Loncor) to support Loncor's press release dated December 15, 2021, announcing the results of the Preliminary Economic Assessment (PEA) of Loncor's Adumbi gold deposit within its 84.68 % owned Imbo Project in the Democratic Republic of the Congo (DRC). This report summarises the results of the Adumbi PEA.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Adumbi deposit is located within Loncor's Imbo Project in the Mambasa District of the Ituri Province, in the northeastern region of the DRC, 260 km west of Bunia, the capital of the Ituri Province, and 225 km northwest of the city of Beni. The Adumbi base camp within the Imbo exploitation permit area is located at latitude 1º 43' 58.76" N and longitude 27º 52' 4.01" E or 596,522 m E and 191,570 m N (WGS 84 UTM Zone 35N) (see Figure 1.1).

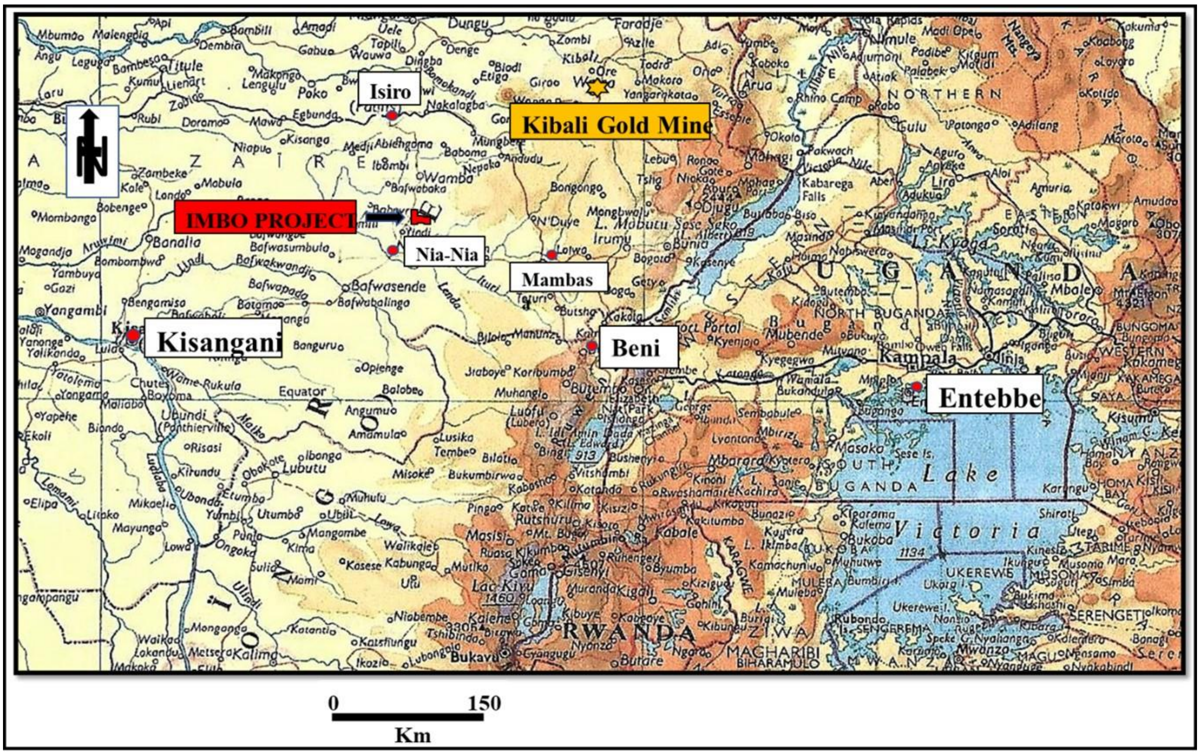


Figure 1.1: Location of the Imbo Project in East Africa

The Imbo Project covers Exploitation Permit Number 9691, has a total area of 122 km^2 and encompasses the known gold mineral deposits of Adumbi, Kitenge and Manzako and several prospects including Canal, Bagbaie, Adumbi West, Amuango, Monde Arabe, Vatican and Imbo East. Adumbi is located approximately 220 km by air southwest from the large operating gold mine of Kibali, operated by Barrick Gold, which in 2020 produced 808,134 oz.

1.3 MINERAL RIGHTS AND LAND OWNERSHIP

Loncor is a publicly listed Canadian gold exploration company and holds 84.68 % interest in the Imbo Project through its subsidiary Adumbi Mining S.A., with the minority shareholders holding 15.32 % (including the 10 % free-carried interest held by the Government of the DRC). The Imbo exploitation permit is valid until February 2039.

Minecon and SENET have relied on a letter on land tenure, licences, and permits dated June 8, 2020, from MBM-Conseil, one of the leading firms practising mining law in the DRC. The Imbo Project comprises a Permis d'Exploitation (PE 9691) or Exploitation Licence held by Adumbi Mining S.A., granted for the period February 23, 2009, to February 22, 2039 (and renewable for an additional 15 years), for gold and diamonds and covering a total of 122 km^2.

Under an agreement signed in April 2010 with the minority partners of Adumbi Mining S.A., Loncor agreed to finance all the activities of Adumbi Mining S.A., until the filing of a bankable feasibility study, by way of loans which bear interest at a rate of 5 % per annum. Within thirty days of the receipt of a bankable feasibility study, the minority partners may collectively elect to exchange their equity participation for either a 2 % net smelter royalty or a 1 % net smelter royalty plus an amount equal to €2/oz of Proven Ore Reserves.

The DRC 2018 Mining Code imposes a royalty tax payable to the State on the sale of minerals, at a rate of 3.5 % for precious metals.

1.4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, PHYSIOGRAPHY AND INFRASTRUCTURE

Located approximately 225 km by air southeast of the Adumbi deposit, Beni is the nearest major population centre to the Imbo Project and has a population of approximately 230,000. Loncor maintains an administrative office in Beni. The city has a lateritic airstrip with scheduled internal flights to other towns in DRC such as Goma, Bunia, Isiro, Kisangani and Kinshasa. The Isiro airstrip is approximately 200 km by lateritic road to the Imbo Project. From Beni, the Imbo Project is accessible via 322 km of lateritic road to Nia-Nia (where there is a lateritic airstrip), then to Village 47 (47 km north of Nia-Nia) and then via 7 km of lateritic roads to the Adumbi base camp.

The nearest international airport is located at Entebbe in western Uganda and linked by 440 km of paved road to the Kasindi Uganda-DRC border, followed by 80 km of unpaved lateritic roads to Beni. Entebbe has international scheduled flights to South Africa, Europe and Asia and is also linked to other African countries as well as the in-country towns of Kinshasa and Lubumbashi via Nairobi (Kenya).

The climate in the Imbo area is typically tropical and is characterised by a long wet season and short dry season of up to three months from mid-December to mid-March. The average annual rainfall is approximately 2,000 mm to 2,500 mm, with the highest rainfall generally occurring in October. Temperatures are uniformly high throughout the year, and there is little diurnal variability, varying between 19 °C and 23 °C, with daily lows and highs of 16 °C and 33 °C, respectively. Humidity is high throughout the year (75 % to 99 %).

The Imbo Project is located in the Ituri tropical rainforest within the upper reaches of the Congo River Basin. The project area topographically consists of an undulating terrain that varies from approximately 600 m above sea level to 800 m above sea level. Most of the surface area is covered with dense evergreen forests with a closed canopy; however, the hills tend to have relatively steep slopes, and the valley floors within the areas of the linear hills are relatively narrow.

The Imbo Project is drained by numerous creeks and streams, which flow into the Upper Ituri river and its main tributaries: the Epulu, Nepoko, Nduye, Lenda, Ebiena, and Ngayu rivers, which form part of the upper reaches of the Congo River Basin. The closest hydroelectric power station is situated near Kisangani together with the hydroelectric stations supplying power to Barrick Gold/AngloGold Ashanti's Kibali Gold Mine. The towns of Isiro and Beni are potential sources of skilled manpower, and there is sufficient local unskilled manpower in the surroundings of Adumbi.

Given its exploration stage of development, there is limited infrastructure currently available at Adumbi. Presently, infrastructure is composed of an exploration camp (the Adumbi base camp) with associated helicopter landing pad, administration building, accommodation buildings and facilities, field office, core logging and storage facilities, diesel generators and solar power generation, and a sample preparation laboratory.

1.5 EXPLORATION HISTORY

Belgian prospectors were the first to discover gold on the Imbo Project in the early 1900s, with gold production focusing on alluvial deposits until the late 1930s. Primary gold mineralisation was later discovered in the bedrock of the alluvial zones and was exploited in shallow pits and trenches. This was later followed by mining from deep trenches and underground galleries. From the mid-1970s to mid-1980s, the French Geological Survey (BRGM) undertook geological investigations of the Imbo Project area.

The mining rights for the mineral concessions in the Imbo Project area were initially held by Société Internationale Forestière et Minière du Congo (FORMINIÈRE or FRM) from the 1920s to the late 1950s. The Belgian colonial state was co-owner of a 50 % stake in FRM, with the remainder held by American interests. The Société Minière de la Tele (SMT), a subsidiary of FRM, oversaw development and exploitation. Following political independence in 1960, ownership has changed hands multiple times.

Highlights of the reported historical exploration include the following:

- 1980 to 1981: BRGM mapped and sampled the Adumbi and Bagbaie deposits on surface and in the historical underground openings. BRGM also drilled three holes at Adumbi and confirmed that (i) mineralisation extended at depth below the water table, (ii) other mineralised zones, parallel to the main one, also existed, and (iii) gold at depth was associated with sulphides.
- 1988: Bugeco International (Bugeco) produced a report on the property entitled "Gold Potential in the Ngayu Mining District Haut Zaire: the Adumbi and Yindi Old Mines".
- 2009: Kilo acquired the property and carried out extensive exploration activities including major drilling campaigns from 2010.

- By November 2013, Kilo had completed 167 diamond drillholes totalling 35,400 m on the Imbo Project.
- 2014: An independent engineering group Roscoe Postle Associates Inc. (RPA) completed technical studies, outlined an Inferred mineral resource, and made various technical recommendations to be executed by Kilo.
- 2014 to 2017: Kilo completed 63 drillholes totalling approximately 8,900 m to test gold-in-soil and magnetic anomalies at the Adumbi South, Adumbi West and Kitenge Extension targets.
- 2017: Four deeper core holes were drilled below the previously outlined RPA Inferred resource over a strike length of 400 m and to a maximum depth of 450 m below surface. All four holes intersected significant gold mineralisation in terms of widths and grades.
- 2018 to 2019: Negligible exploration groundwork was undertaken by Kilo due to financial constraints.
- In September 2019, Loncor initially acquired a 71.25 % interest in the Imbo Project, which was subsequently increased to 84.68 % in 2020.
- April 2020: An Inferred mineral resource of 2.19 Moz (28.97 Mt at 2.35 g/t Au) was determined, constrained within a US$1,500/oz pit shell at Adumbi.
- October 2020: Loncor commenced a core drilling programme at Adumbi to increase and upgrade mineral resources within a US$1,600/oz open-pit shell and at depth. A total of 24 core holes (10,071 m) were drilled during this programme as part of the PEA.

1.6 GEOLOGICAL SETTING AND GOLD MINERALISATION

The Adumbi gold deposit is found within the Ngayu Archean greenstone belt, one of a number of Archean-aged, granite-greenstone belts that extend from northern Tanzania into northeastern DRC and then into the Central African Republic. The greenstone belt terrain in northeastern DRC has a number of major gold belts including Moto (Kibali), Kilo, Mambasa, Ngayu and Isiro.

The majority of the gold occurrences within the Ngayu belt are located close to the contact of the Banded Ironstone Formation (BIF). Historically, only two deposits were exploited on any significant scale, namely Yindi and Adumbi. Styles of gold mineralisation within the Ngayu belt include shears within the BIF or on the BIF contacts, disseminated mineralisation, and shears within basalts and schists, resulting in discrete auriferous gold veins. Artisanal mining of weathered gold mineralisation preserved as elluvial or colluvial material is widespread throughout the belt.

Within the Imbo Project area, there is a strong association between gold mineralisation and the presence of the BIF, with the BIF constituting the host rock (e.g. Adumbi) or forming a significant part of the local stratigraphy in the Imbo Project area. The BIF forms both physical and chemical traps for mineralising hydrothermal fluids. The iron-rich BIF is a chemically reactive rock, the main interaction with hydrothermal fluids involving the reduction of magnetite to pyrite, resulting in the precipitation of gold. Mineralisation on the Imbo Project (PE9691) is known to occur at Bagbaie (referred to as Adumbi North), Adumbi, Kitenge, Manzako, Monde Arabe, Maiepunji and Vatican.

Adumbi is currently the most explored deposit within the Imbo Project. Adumbi forms a topographic high (Adumbi Hill) and incorporates the Canal prospect, which is the southeastern continuation of Adumbi. Based on examined drillholes, the rocks at Adumbi mainly comprise a subvertical sequence of metamorphosed clastic sediments (pelites, siltstones and greywacke) interbedded with units of BIF of varying width. The grade of metamorphism is probably lower greenschist facies, and the clastic units are petrographically classified as schists. Foliation is usually clearly defined in hand specimens although sedimentary features such as bedding are frequently preserved.

The Adumbi deposit displays five distinct geological domains with the BIF unit attaining a thickness of up to 130 m in the central part. There is a higher-grade zone of gold mineralisation termed the “replaced rock zone” (RP zone) associated with alteration and structural deformation that has completely destroyed the primary host lithological fabric. The RP zone occurs in the lower part of the Upper BIF package and in the Lower BIF package, and transgresses the Carbonaceous Marker, located between the Upper and Lower BIF packages, both along strike and down dip. The geological interpretation from the Loncor drill intersections demonstrates that the mineralised BIF increases in thickness with depth and thus confirms the existence of significant underground potential at Adumbi below the mineral resources within the open-pit shell.

The detailed logging of the mineralised cores indicated a direct relationship between gold values and the percentage of sulphide mineralisation and intensity of silicification. In general, pyrite is the dominant sulphide followed by pyrrhotite, then arsenopyrite. When pyrite and pyrrhotite are associated with arsenopyrite, the gold values are very significant, compared to when pyrite is associated with pyrrhotite only. Silica is associated with the highest degree of hydrothermal alteration within the zones and serves as a marker of mineralisation; however, without sulphides, the gold values are insignificant. Specks of visible gold are occasionally found, generally within fractures and are present in white to grey, glassy, weak to moderately brecciated quartz veins.

1.7 DEPOSIT TYPES

Gold deposits within the Imbo Project are associated with the globally important Neo-Archean orogenic gold deposits, examples of which are found in most Neo-Archean cratons around the world. Gold mineralisation is associated with the epigenetic mesothermal style of mineralisation. This style of mineralisation is typical of gold deposits in Neo-Archean greenstone terranes and is generally associated with regionally metamorphosed rocks that have experienced a long history of thermal and deformational events. These deposits are invariably structurally controlled.

Mineralisation in this environment is commonly of the fracture and vein type in brittle fracture to ductile dislocation zones. At the Adumbi deposit, the gold mineralisation is generally associated with quartz and quartz-carbonate-pyrite ± pyrrhotite ± arsenopyrite veins in a BIF horizon.

Examples of similar type BIF hosted gold deposits to Adumbi include Geita in Tanzania, Kibali in northeastern DRC, Tasiast in Mauritania, Homestake (U.S.A.), Lupin (Canada) and Moro Velho in Brazil.

1.8 EXPLORATION

The Imbo Project has been explored since the early 1900s by Belgian prospectors and more recently by Kilo and then Loncor. During the period 2010 to 2012, 44 trenches totalling 4,753 m were excavated over the Adumbi, Kitenge and Manzako targets. Accessible adits and underground workings were also geologically mapped and sampled at Adumbi; however, those at Kitenge and Manzako were not accessible. In all, a total of 907 m was sampled.

By November 2013, Kilo had completed 167 diamond drillholes totalling 35,400 m on the Imbo Project. Kilo outsourced sample preparation and analysis to independent assayers ALS Geochemistry (ALS). Drill core sample preparation was conducted at ALS Mwanza (Tanzania) from 2010 to August 2011, and then at an on-site purpose-built container facility supplied and managed by ALS Minerals. Analyses were undertaken by ALS Johannesburg (South Africa) and ALS Vancouver (Canada).

In February 2014, independent consultants RPA completed an independent NI 43-101 technical report on the Imbo Project and estimated 1.675 Moz (20.78 Mt grading 2.5 g/t Au) of Inferred Mineral Resources on the three separate deposits of Adumbi, Kitenge and Manzako.

RPA made several recommendations on Adumbi, which were addressed in subsequent exploration programmes. In September 2020, Loncor signed a management service agreement with Minecon to manage the infill and extension drilling programme on the Adumbi deposit.

1.9 DRILLING

The more recent drilling on the Imbo Project has been carried out by Kilo and then Loncor using contract drilling companies. The drilling programmes have been carried out in phases:

- 2010 to 2013 (Kilo)
- 2014 to 2017 (Kilo)
- 2020 to 2021 (Loncor)

As of November 15, 2013, Kilo had completed 167 diamond drillholes totalling 35,400 m on the Imbo Project. During the 2014 to 2017 drilling programme, 63 drillholes totalling 8,900 m were drilled.

The 2020 to 2021 drilling campaign was carried out by Orezone Drilling and a total of 24 holes totalling 10,071.44 m were drilled at Adumbi. The drill core was systematically logged and photographed before cutting and sampling. Reflex Act II orientation survey equipment was used for core orientation at every run of 3 m in competent material to aid in structural measurements. Structural measurements taken during the routine logging were from bedding, foliation, and quartz veins whereas structural measurements from lithological contacts, joints and shears were captured in detail under a separate geotechnical logging programme.

1.10 SAMPLE PREPARATION, ANALYSES AND SECURITY

During the 2014 to 2017 exploration activity, sample preparation and analyses were outsourced to the SGS laboratory in Mwanza, Tanzania (which is independent of Loncor). The

SGS laboratory operates a quality system that is accredited in accordance with ISO/IEC 17025:2017 and SANAS (South African National Accreditation System). The SGS laboratory acted as an umpire laboratory even while ALS Chemex was the principal laboratory; hence, correlational studies between the two laboratories have been undertaken.

As part of the 2020 to 2021 drilling programme, Loncor started using the on-site sample preparation laboratory. This has helped with the enforcement of stricter QA/QC policing on the analytical laboratory. Laboratory procedures have been documented and reviewed by Minecon senior management, and internal quality control measures have been taken. Based on the documentation and discussions with the laboratory management, Minecon's senior management does not have any concerns regarding the sample preparation for all Loncor samples.

Sample pulps are sent for analyses to SGS Mwanza, which serves as the primary laboratory. SGS is internationally accredited and utilises conventional sample preparation, sample analysis and associated quality control protocols. Once the samples are received at the SGS laboratory, the samples go through checking and reconciliation procedures, followed by the SGS sample preparation procedure (SGS Code PRP87).

Drill core, trench, adit, pit, rock chip and channel samples were analysed for gold at the SGS Mwanza laboratory using fire assay (FA) with flame atomic absorption spectrometry (AAS) to measure the gold (SGS Code FAA505), and the analyses were carried out on 50 g aliquots. The effective range for FAA505 is 0.01 ppm Au to 100 ppm Au. In addition, check assays were carried out by the screen fire assay method to verify higher-grade sample assays obtained by fire assay. Internationally recognised standards and blanks were inserted at the Adumbi sample preparation laboratory as part of internal QA/QC analytical procedures.

1.11 MINERAL PROCESSING AND METALLURGICAL TESTING

PEA-level metallurgical test work (comminution and gold recovery) was performed by Maelgwyn Mineral Services Laboratory in Johannesburg on the Adumbi mineralised samples to evaluate the process route required to obtain the highest gold recoveries that can be achieved.

Table 1.1 shows a summary of the PEA Adumbi metallurgical test work results.

Table 1.1: Adumbi Metallurgical Test Work Results

Parameters	Unit	Oxide	Transition	Fresh
Bond Rod Work Index	kWh/t	12.7	13.6	14.6
Bond Ball Work Index	kWh/t	11.8	13.7	14.2
Abrasion Index		0.19	0.25	0.34
Diagnostic Leach Carbon in Leach (CIL) Recovery	%	90.76	87.53	89.9

The average diagnostic leach recovery for the fresh (sulphide) material was the weighted mean of the RP and BIF lithologies relative to the volume of their occurrence (20 % RP:80 %

BIF) in the fresh material. Diagnostic leach recoveries of 80.10 % for RP and 92.37 % for BIF were realised for the fresh (sulphide) material.

Comminution results indicated that both the oxide and transition material are medium hard while the fresh material indicated that it is slightly hard. These results were taken into account in the design of the comminution flowsheet.

In order to optimise the gold recovery, further test work was conducted on the fresh and transition material whereby gravity was followed by flotation on the gravity tails. The results showed that most of the gold can be floated into float concentrates as summarised in Table 1.2.

Table 1.2: Flotation Results

Sample ID	Rougher Concentrate			
	Gold		Sulphur	
	Grade (g/t)	Recovery (%)	Grade (%)	Recovery (%)
Fresh – RP	9.57	95.06	25.07	93.03
Fresh – BIF	8.30	87.16	17.90	85.13
Transition	11.82	81.31	15.80	95.52

The concentrate samples that were generated were not sufficient to enable further processing routes such as the following:

- Fine milling followed by leaching with oxygen addition
- Fine milling followed by partial oxidation using high shear reactors and leaching
- Albion process
- Pressure oxidation
- Bio leaching
- Roasting

These recovery processes will be investigated during the next phase of the project to optimise the gold recovery in the transition and fresh ore types.

1.12 MINERAL RESOURCES

During Q3 of 2021, Loncor commissioned Minecon to re-evaluate and quantify the exploration work including drilling undertaken during the period 2020 to 2021. This has resulted in Minecon updating the Mineral Resource estimate of Adumbi according to the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014, as incorporated in NI 43-101. This follows a previous mineral resource estimate undertaken by Minecon in April 2021.

Compared to the Inferred Mineral Resource of 3.15 Moz of gold (41.316 Mt grading 2.37 g/t Au) outlined in April 2021 (see Loncor press release dated April 27, 2021), the additional drilling information and the increased gold price have contributed significantly to the increased mineral resources of the Adumbi deposit with improved confidence to 1.88 Moz

(28.185 Mt grading 2.08 g/t Au) of gold in the Indicated category and 1.78 Moz (20.828 Mt grading 2.65 g/t Au) of gold in the Inferred category.

Table 1.3 summarises the Adumbi Indicated and Inferred Mineral Resources based on an in-situ block cut-off grade at a 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material, and constrained within a US$1,600/oz optimised pit shell. A total of 84.68 % of the Adumbi mineral resources are attributable to Loncor via its 84.68 % interest in the Imbo Project.

Table 1.3: Adumbi Deposit Indicated and Inferred Mineral Resources (Effective Date: November 17, 2021)

Mineral Resource Category	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Indicated	28,185,000	2.08	1,883,000
Inferred	20,828,000	2.65	1,777,000

NOTES:
1. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
2. Numbers might not add up due to rounding.

Table 1.4 summarises the Indicated and Inferred category mineral resources in terms of material type.

Table 1.4: Adumbi Mineral Resources by Material Type (Effective Date: November 17, 2021)

Material Type	Indicated Mineral Resource			Inferred Mineral Resource		
	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Oxide	3,169,000	2.05	208,000	458,000	3.39	49,000
Transition	3,401,000	2.51	274,000	280,000	2.74	24,000
Fresh (Sulphide)	21,614,000	2.02	1,400,000	20,089,000	2.64	1,703,000
TOTAL	**28,185,000**	**2.08**	**1,883,000**	**20,828,000**	**2.65**	**1,777,000**

NOTES:
1. The CIM definitions were followed for Mineral Resources.
2. Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material constrained by a Whittle pit.
3. Mineral Resources for Adumbi were estimated using a long-term gold price of US$1,600/oz.
4. A minimum mining width of 32 m horizontal was used.
5. A maximum of 4 m internal waste was used.
6. Adumbi bulk densities of 2.45 for oxide, 2.82 for transition and 3.05 for fresh rock were used.
7. High gold assays were capped at 18 g/t Au for Adumbi, prior to compositing at 2 m intervals.
8. Numbers might not add up due to rounding.

1.13 MINERAL INVENTORY

The Mineral Inventory Statement uses the definitions and guidelines given in the CIM Definition Standards on Mineral Resources and Mineral Reserves and is reported in accordance with NI 43-101 requirements.

Table 1.4 shows a summary of the Adumbi Mineral Inventory for the various material types (oxide, transition and fresh) contained within the Adumbi practical pit designs.

The following summarises the pit optimisation assumptions and parameters used to constrain the depth extent of the geological model to generate the mineral inventory of the open pit for the Adumbi deposit:

- A gold price of US$1,600/oz
- A block size of 16 m × 16 m × 8 m
- A 32 m minimum mining width and a maximum of 4 m of internal waste was applied
- A mining dilution of 100 % of the tonnes at 95 % of the grade
- An ultimate slope angle of 45°
- An average mining cost of US$3.29/t mined
- Metallurgical recoveries of 91 % for oxide, 88 % for transition and 90 % for fresh
- An average general and administration (G&A) cost of US$4.20/t
- Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh materials, constrained by a US$1,600/oz optimised pit shell
- Transport of gold and refining costs equivalent to 4.5 % of the gold price

The results from the Adumbi Whittle pit optimisation for the gold price of US$1,600/oz allowed for the selection of the optimised final pit shell (Pit Shell 40) based on the maximum undiscounted cash flow for the practical pit design. The practical pit designs were prepared using the optimised pit shells as templates. The relevant Whittle pit shells were exported from the GEMS to Surpac software, where the practical pit designs were prepared. The practical pit design incorporates the ramps together with the appropriate inter-ramp slope angles. No practical pit design was prepared for the Final Pit; hence, the optimised pit shell (Pit 40) was used to define Cut 3 for the blocks to be scheduled.

The Qualified Person (QP) has performed an independent verification of the block model tonnage and grade, and in the QP's opinion, the process has been carried out to industry standards.

1.14 MINING METHODS

Over the life of mine (LOM) of Adumbi, a total of 49.77 Mt grading 2.17 g/t Au is expected to be mined from the open pit and delivered to the processing plant. In the QP's opinion, the parameters used in the Mineral Resource to Mineral Inventory conversion process are reasonable.

The scheduling process consists of developing a mine plan that is economically optimal using the inventory included in the practical pit designs and pit shell. The production schedule was based on a methodology of block selections from the block model inside the designed pits (see Figure 1.2).

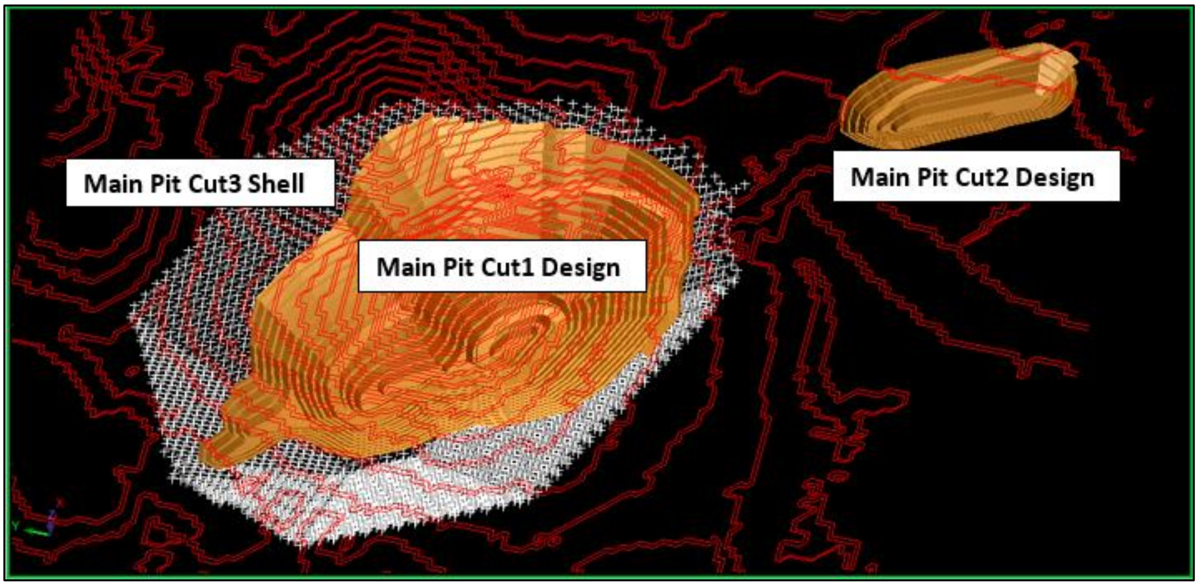


Figure 1.2: Adumbi Open-Pit Design

The mining production schedule sequence is planned to mine the Cut 1 and Cut 2 pit designs from the beginning of Year 1 to get rapid access to the ore and manage the strip ratio. Mining of the Cut 3 pit shell starts from the beginning of Year 2. The Adumbi deposit is planned to be to provide a throughput of 5.0 Mt/a of ore to the processing plant (see Table 1.5). Adumbi has an average stripping ratio of 11:1, which is a significant contributing factor to operating costs.

Table 1.5: Mining, Processing and Gold Production Schedule over Adumbi's LOM

Description	Unit	Year	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Period Number		**TOTAL**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
ROM Tonnes	t	**29,886,695**	3,080,333	1,980,361	2,633,447	2,887,295	4,215,615	3,884,952	2,523,768	2,291,908	4,274,748	2,114,268	-
ROM In-Situ Grade	g/t	**3.06**	3.05	3.45	2.77	2.68	2.76	2.68	2.95	3.14	3.42	4.14	-
ROM Diluted Grade	g/t	**2.9**	2.9	3.28	2.63	2.55	2.62	2.55	2.8	2.98	3.25	3.94	-
ROM Recoverable Grade	g/t	**2.61**	2.59	2.92	2.36	2.29	2.36	2.29	2.52	2.68	2.92	3.54	-
LG Tonnes	t	**19,884,165**	2,281,482	1,489,862	1,824,614	2,944,307	3,338,833	2,557,117	1,822,242	1,545,813	1,588,504	491,390	-
LG In-Situ Grade	g/t	**1.13**	1.07	1.02	1.09	1.18	1.13	1.17	1.14	1.16	1.21	1.11	-
LG Diluted Grade	g/t	**1.07**	1.01	0.96	1.04	1.12	1.07	1.11	1.08	1.1	1.14	1.05	-
LG Recoverable Grade	g/t	**0.96**	0.91	0.86	0.93	1	0.96	1	0.97	0.99	1.03	0.94	-
Waste Tonnes	t	**547,232,488**	67,178,178	82,619,771	85,738,796	74,468,394	62,670,499	49,166,557	41,165,882	39,962,277	37,936,746	6,325,388	-
Total Material (Ore + Waste)	t	**597,003,348**	72,539,994	86,089,994	90,196,857	80,299,996	70,224,947	55,608,626	45,511,892	43,799,998	43,799,998	8,931,047	-
Plant Feed Tonnes	**t**	**49,770,861**	**2,985,000**	**5,018,750**	**5,032,500**	**5,018,750**	**5,018,750**	**5,018,750**	**5,032,500**	**5,018,750**	**5,018,750**	**5,018,750**	**1,589,611**
Feed Grade (Undiluted)	g/t	**2.29**	3.04	2.04	1.96	2.04	2.5	2.34	2.05	2.06	3.09	2.42	1.16
Feed Grade (Diluted)	g/t	**2.17**	2.89	1.94	1.86	1.94	2.37	2.22	1.95	1.96	2.93	2.3	1.1
Feed Grade (Recoverable)	g/t	**1.95**	2.59	1.73	1.67	1.74	2.13	2	1.75	1.76	2.64	2.07	0.99
Recovery	%	**89.79**	**89.44**	**89.38**	**89.56**	**89.82**	**89.92**	**89.91**	**89.91**	**89.9**	**89.92**	**89.92**	**89.91**
Gold Produced	**oz**	**3,121,324**	**248,189**	**279,209**	**270,170**	**280,875**	**344,336**	**321,976**	**283,422**	**283,806**	**425,296**	**333,331**	**50,715**
Strip Ratio	t/t	**11**	12.53	23.81	19.23	12.77	8.3	7.63	9.47	10.41	6.47	2.43	-
Operational Waste	t	**443,675,980**	55,158,163	38,155,236	49,016,548	64,118,684	62,670,499	49,166,557	41,165,882	39,962,277	37,936,746	6,325,388	-
Capitalised Waste	t	**103,556,508**	12,020,015	44,464,534	36,722,248	10,349,710	-	-	-	-	-	-	-

The equipment units and costs have been selected from major global manufacturers. Although quotes have not been specifically requested for this project, the capital and transportation costs have been derived from similar projects from the relevant equipment suppliers. Outputs have been derived from generic calculations.

The mining personnel include all the salaried staff working in the mine operations and mining engineering departments, and the operational labour required to operate drilling, blasting, loading, hauling and mine support activities.

Minecon has generated manpower and staffing requirements from first principles and with knowledge of similar operations in remote conditions. These requirements have been based on the mining operations being carried out by contract mining as opposed to owner mining.

1.15 RECOVERY METHODS

The Adumbi process plant has been designed to process 5 Mt/a.

The process plant was developed from the interpretation of the results of the PEA test work.

The interpretation of these metallurgical test work results and simulations of the comminution circuit conducted by Orway Mineral Consultants (OMC) led to the development of the process flowsheet, which includes the following:

- Single-stage crushing facility, utilising a primary crusher to produce a crushed product size suitable for semi-autogenous grinding (SAG) milling
- A crushed ore surge bin which, when full, allows crushed ore to be diverted to an emergency stockpile from which ore can be reclaimed by front-end loader (FEL) to feed the mill during the periods when the crushing circuit is offline
- Two-stage SAG and ball mill combination, operating in closed circuit with the cyclone cluster to produce the required size
- Additional pebble crushing system to crush pebbles from the SAG mill when treating transition and sulphide ores, i.e. the SABC circuit
- Gravity concentration and removal of coarse gold from the milling circuit recirculating load and the treatment of gravity concentrate by intensive cyanidation and electrowinning to recover gold to bullion
- CIL circuit incorporating 1 pre-leach tank and 11 CIL tanks containing carbon for gold adsorption
- INCO air/SO_2 cyanide detoxification, arsenic precipitation and pumping of the detoxified tailings to the dam
- TSF and the return water pumping system
- 12 t acid wash, pressure Zadra elution circuit, electrowinning, carbon regeneration and gold smelting to recover gold from the loaded carbon to produce bullion
- Consumables and reagents
- Air and water services

Table 1.6 summarises the key process design criteria values which were utilised in the process design.

Table 1.6: Summary of Key Process Design Criteria

Item	Unit	Oxide	Transition	Fresh
Plant Throughput	Mt/a	5.0	5.0	5.0
Gold Head Grade	g/t Au	2.25	3.2	4.00
Design Gold Recovery	%	91.82	90.38	80.1 to 89.83
Crushing Plant Utilisation	%	65.0	65.0	65.0
Plant Availability	%	91.32	91.32	91.32
Comminution Circuit		Primary Crushing and SAB	Primary Crushing and SABC	Primary Crushing and SABC
Crush Size, P_{80}	mm	180	180	180
Grind Size, P_{80}	µm	75	75	75
Leach/CIL Residence Time	h	24	24	24
Leach Slurry Density	% w/w	40	40	50
Number of Pre-leach Tanks		1	1	1
Number of CIL Tanks/Stages		11	11	11
Cyanide Consumption	kg/t	0.99	1.32	1.31
Lime Consumption	kg/t	3.64	5.40	3.61
Elution Circuit Type		Pressure Zadra	Pressure Zadra	Pressure Zadra
Elution Circuit Size	t	12	12	12

A simplified flow diagram depicting the unit operations incorporated in the selected process flowsheet is shown in Figure 1.3.

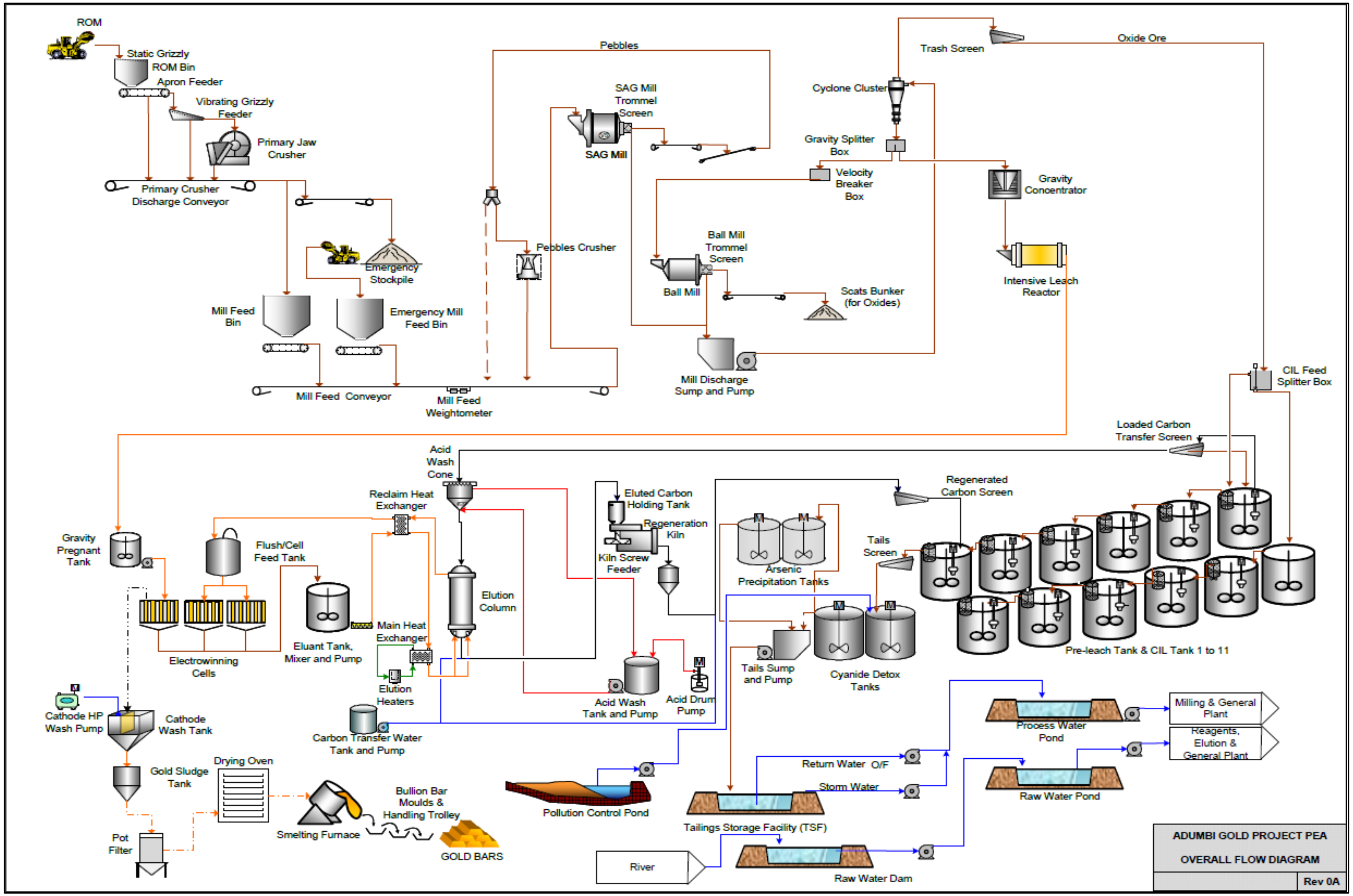


Figure 1.3: Proposed Process Plant Flowsheet

1.16 PROJECT INFRASTRUCTURE

The following existing and proposed project infrastructure will support the mining and plant operations:

- Existing infrastructure:
  - Exploration camp
  - Airstrip
- Proposed infrastructure:
  - In-plant infrastructure:
    - Plant buildings
    - Plant reagents and consumables stores
  - Other infrastructure:
    - Mining infrastructure
    - Access road to the project site
    - TSF
    - Water supply
    - Off-site power supply and distribution (hydroelectric power (HEP))
    - Diesel fuel storage and supply
    - Accommodation camps

A number of hydroelectric sites were identified close to Adumbi, and further studies are required to optimise the power generation concept for the operation. Loncor is already in discussion with potential power suppliers with experience in the DRC to project finance and build a hydroelectric facility at Adumbi and then have an offtake agreement with them.

Loncor has engaged with local contractors who have provided budget tenders for the plant buildings and accommodation camps. Further discussions will be held with these contractors to refine their quotes for the project. The local community will benefit from this initiative as it will provide them with employment and the development of skills.

Figure 1.4 indicates the layout of the existing and proposed infrastructure.

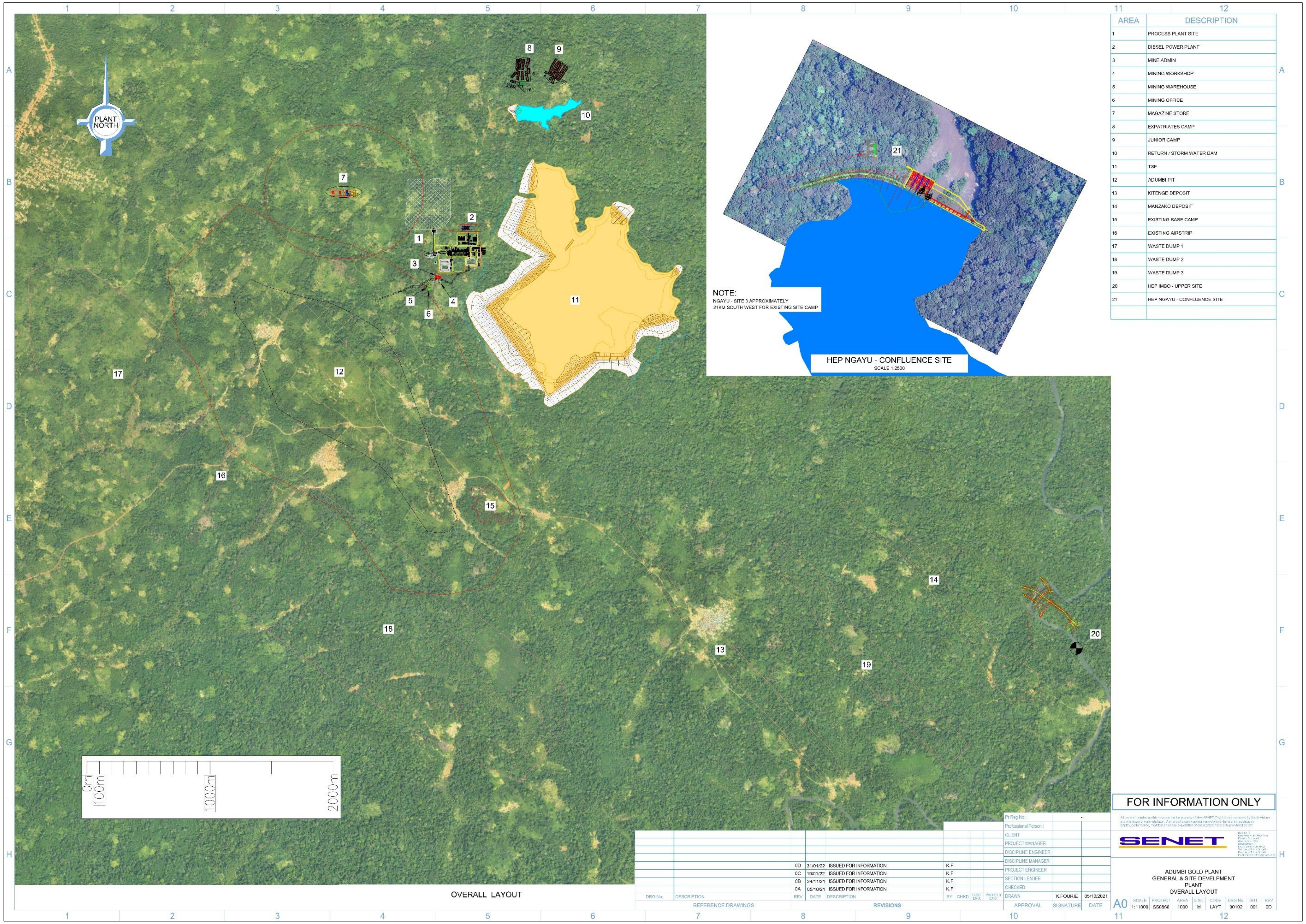


Figure 1.4: Layout of Existing and Proposed Infrastructure

1.17 MARKET STUDIES AND CONTRACTS

The financial evaluation and cut-off grade calculation for the Adumbi deposit and all the Mineral Inventory use a gold price of US$1,600/oz and an optimised pit design. This is in line with Loncor's corporate guidelines. A total royalty payable to the DRC government of 3.5 % of the gold revenue, inclusive of 4 % shipment fees, was used for the open-pit Mineral Inventory estimate. The study indicates that Loncor will pay a 30 % income tax to the DRC government. Depreciation for the economic model was set at 10 % on capital expenditure.

Adumbi is planned to produce 303,000 oz/a of gold contained in doré over a 10.3-year mine life. The combined gold and silver content is expected to be 98 % (the remaining 2 % is likely to consist of impurities such as copper, iron and zinc). The gold content is expected to account for approximately 95 % to 97 % of the precious metals content, with the remaining amount being silver content. No deleterious elements are indicated in the ore head grade assayed, and as such these are not expected to be in the doré.

Doré bars are planned to be cast in 804 oz (~25 kg) bricks with approximate dimensions of 190 mm length, 120 mm width and 80 mm height. Fifteen to twenty pours per month are planned, with the doré being transported weekly from the mine site to the export facilities. After weighing, sampling and assaying, the doré will be packed and secured in high-security tamper-evident carry boxes in the gold room in the presence of mine production and security personnel.

Industry-standard gold room and strong room facilities will be constructed on site to hold the doré until it is ready for export. Appropriate checks and balances, security cameras, alarms, insurance and security procedures will be implemented to cover the production and storage at the mine. Gold doré that will be produced on the mine will be shipped from site under secured conditions. Gold mines do not compete to sell their product given that the price is not controlled by the producers.

1.18 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The objectives of the environmental and social component of the Adumbi PEA were to evaluate the adequacy of the environmental and social studies undertaken to date and determine, at a screening level, the potential environmental and social issues associated with the proposed mine development. This included identifying key issues, fatal flaws, high-level environmental and social risks that may affect the project and require further investigation, and identifying costs for further environmental and social studies.

The material presented was to meet the requirements as per the DRC *Code Minier* or Mining Code, Law No. 007/2002 of July 11, 2002, as amended and completed by Law No. 18/001 of March 9, 2018. Information was obtained from the September 2021 site visit by Minecon's environmental consultant where an overview of the very limited environmental and social baseline data was assessed. Key legislation includes the 2006 Constitution of the Third Republic (The Constitution), the 1997 National Environmental Action Plan (NEAP) providing a framework for the management of the DRC's natural resources, as well as the Mining Code governing commercial and artisanal mining activities. The latter requires, inter alia, an

Environmental and Social Impact Study (ESIS), a Mitigation and Rehabilitation Plan (MRP), and an Environmental and Social Management Plan of the Project (ESMPP). Mining Regulations (Decree No. 038/2003 of March 26, 2003, as amended and completed by Decree No. 18/024 of June 8, 2018) give effect to the Code in terms of environmental social obligations, public consultation and requirements for ESIS and ESMPP reports. The Framework Law on the Environment (Act No.11/009 of July 9, 2011) aims to define the protection of the environment and the direct management of natural resources, and to reduce pollution and serve as a basis for sectors impacting the environment.

Minecon's environmental assessment of the Imbo Project included knowledge from other recent operating gold mines and other development projects in eastern DRC. The key environmental and social issues identified during this initial screening study that have possible impacts included surface water, air quality, biodiversity, climate, infrastructure, safety and security, community development, resettlement, land acquisition, land replacement and compensation, health and safety, labour and artisanal mining, project engagement and policies. On the basis of the available information, screening studies undertaken did not identify any fatal flaws from an environmental and socio-economic perspective. Key issues were identified that would need to be investigated further during the pre-feasibility study (PFS) and feasibility study (FS) phases of the project.

Six artisanal mining villages identified within the project footprint have been assessed for social repositioning. The total population of these villages is estimated to be approximately 5,800. The legal situation regarding compliance with the DRC resettlement plan process, assessment and reporting requirements is unclear on the repositioning of artisanal mining huts and in other mine developments in DRC, artisanal miners were not compensated or resettled. Future studies (ESIS, ESMPP and MRP) will be undertaken to clarify this situation. However, Minecon employed a conservative approach in the estimation of the social repositioning of the six artisanal settlements that will be impacted by the project development. Minecon believes that if the livelihood restoration programmes are accepted for the repositioning of artisanal miners within the Adumbi mine footprint, the capital budget could be significantly lower than estimated in this PEA study.

Minecon recommended a number of environmental and social studies that should commence during the next PFS stage, including the following:

- Preparation of design criteria for monitoring ambient and workplace environmental conditions
- Preparation of a preliminary environmental impact assessment and preliminary environmental management plan
- Baseline data collection and preparation of an International Finance Corporation (IFC) compliant ESIS and ESMPP

1.19 CAPITAL AND OPERATING COSTS

1.19.1 Capital Cost (CAPEX) Summary

The estimated initial capital costs are based on the two options that were considered for the project, with the determining factor being the power-supply solution:

- Option 1: Diesel generation power plant
- Option 2: Hydroelectric power (HEP) hybrid option, constituting a diesel generation power plant, an HEP plant located at the Ngayu Confluence, a solar photovoltaic (PV) plant and a battery energy storage system (BESS)

The total estimated cost of bringing the project into production is **US$392,010,000 for the diesel option and US$530,058,000 for the HEP hybrid option**, which is inclusive of contingency and EPCM and is summarised in Table 1.7. This cost excludes any costs for feasibility studies scheduled prior to the start of basic engineering.

Table 1.7: CAPEX Summary

Item	Diesel Option Value (US$ Thousand)	HEP Hybrid Option Value (US$ Thousand)
Mining Development Costs	59,986	59,986
Process Plant	171,369	171,369
Power Plant	14,405	152,452
TSF	65,880	65,880
Infrastructure	27,461	27,461
Access Road	7,800	7,800
Owner's Costs	45,110	45,110
INITIAL CAPITAL	**392,010**	**530,058**

1.19.2 Operating Cost (OPEX) Summary

The purpose of this OPEX estimate is to provide the operating costs, and the associated G&A costs, that will be utilised for the economic analysis of the Adumbi deposit. Two OPEX options for the process plant were developed, using diesel and HEP hybrid options as the source of power.

The project's annual OPEX estimate for the LOM consists of the following:

- Mining operating costs estimated by Minecon
- Process plant operating costs estimated by SENET
- TSF operating costs estimated by Epoch
- Site G&A costs estimated by SENET
- Bullion transport, insurance and refining costs estimated by SENET

The overall OPEX summaries for Adumbi for both the diesel and the HEP hybrid options are given in Table 1.8 and Table 1.9, respectively.

Table 1.8: Overall Plant OPEX Summary for Diesel Option

Description	LOM		
	US$/t processed	US$/oz	Distribution (%)
Mining	31.33	499.56	55.01
Processing	17.95	286.21	31.52
TSF	0.55	8.75	0.96
G&A	3.40	54.19	5.97
Refining and Transport	0.22	3.50	0.39
Royalties	3.51	55.96	6.16
Total	**56.95**	**908.17**	**100**

Table 1.9: Overall Plant OPEX Summary for HEP Hybrid Option

Description	LOM		
	US$/t processed	US$/oz	Distribution (%)
Mining	31.33	499.56	58.62
Processing	14.44	230.22	27.02
TSF	0.55	8.75	1.03
G&A	3.40	54.19	6.36
Refining and Transport	0.22	3.50	0.41
Royalties	3.51	55.96	6.57
Total	**53.44**	**852.18**	**100**

1.20 ECONOMIC ANALYSIS

The Adumbi deposit financial analysis was prepared using the discounted cash flow model. Two economic models were run: one using diesel generating sets as a source of power and the other using the HEP hybrid option.

In preparing this model, several assumptions and material factors have been employed, as presented in Table 1.10.

Table 1.10: Financial Analysis Assumptions

Description	Unit	Assumption
Evaluation Start Date		January 1, 2022
Revenue		
Gold Price	US$/oz	1,600
Refining Losses	%	0.08
Discount Rate	%	5.0
Fuel Prices		
Diesel Price	US$/L	0.9

Description	Unit	Assumption
Fiscal		
Government Royalty	%	3.5
Tax Holiday	Years	2
Tax Rate (after tax holiday)	% of profits	30
Tax Rate if there is a loss	% of annual turnover	1
Dividend Tax	%	10
Depreciation	%	10 % over 10 years
Kilograms to Ounces	kg/oz	32.1505
Diesel SG	t/m^3	0.85
Other Charges		
Bullion Transport and Refining Costs	US$/oz	3.50
Exchange Rates	ZAR/US$	15.11
	US$/£	1.16
	AU$/US$	1.39
	US$/C$	0.74
	CFA/€	2,260
	CFA/US$	2,004

The findings of the model are summarised in Table 1.11.

Table 1.11: Summary of Financial Findings

Description	Unit	HEP Hybrid Option		Diesel-Only Option	
		Pre-Tax	After Tax	Pre-Tax	After Tax
LOM Tonnage Ore Processed	kt	49,771	49,771	49,771	49,771
LOM Feed Grade Processed	g/t	2.17	2.17	2.17	2.17
Production Period	Years	10.3	10.3	10.3	10.3
LOM Gold Recovery	%	89.8	89.8	89.8	89.8
LOM Gold Production	koz	3,121	3,121	3,121	3,121
LOM Payable Gold After Refining Losses	koz	3,119	3,119	3,119	3,119
Gold Price	US$/oz	1,600	1,600	1,600	1,600
Revenue	US$ million	4,990	4,990	4,990	4,990
Site Operating Costs	US$/oz	793	793	849	849
Total Cash Costs	US$/oz	852	852	908	908
All-In Sustaining Costs (AISC)	US$/oz	950	950	1,040	1,040
NPV (5 % Discount)	US$ million	895	624	843	600
Internal Rate of Return (IRR)	%	25.2	20.7	30.3	25.2
Discount Rate	%	5	5	5	5
Payback Period (from start of production)	Years	4.16	4.98	3.16	4.06
Project Net Cash	US$ million	1,495	1,087	1,353	992

Table 1.12 and Table 1.13 detail the NPV and IRR sensitivities of the project to gold price

Table 1.12: Key Project Metric Sensitivity to Gold Price – Diesel Option

Description	Unit	Value				
Change in Gold Price	%	−15	−10	0	10	15
Gold Price	US$/oz	1,360	1,440	**1,600**	1,760	1,840
NPV at 5 % – Pre-Tax	US$ million	321	495	843	1,191	1,365
NPV at 5 % – After Tax	US$ million	211	345	600	855	983
IRR – Pre-Tax	%	15.58	20.83	30.34	38.99	43.07
IRR – After Tax	%	12.68	17.21	25.18	32.57	36.11
Total Cash Costs	US$/oz	900	903	908	914	917
AISC	US$/oz	1,031	1,034	1,040	1,045	1,048
Cash Flow Payback – Pre-Tax	Years	7.53	5.46	3.16	2.18	1.93
Cash Flow Payback – After Tax	Years	8.02	6.28	4.06	2.77	2.39
NOTE: The value in bold reflects the base case.						

Table 1.13: Key Project Metric Sensitivity to Gold Price – HEP Hybrid Option

Description	Unit	Value				
Change in Gold Price	%	−15	−10	0	10	15
Gold Price	US$/oz	1,360	1,440	**1,600**	1,760	1,840
NPV at 5 % – Pre-Tax	US$ million	373	547	895	1,243	1,417
NPV at 5 % – After Tax	US$ million	238	368	624	879	1,006
IRR – Pre-Tax	%	14.1	18.0	25.2	31.9	35.1
IRR – After Tax	%	11.4	14.7	20.7	26.4	29.1
Total Cash Costs	US$/oz	844	847	852	858	861
AISC	US$/oz	941	944	950	955	958
Cash Flow Payback – Pre-Tax	Years	7.88	6.26	4.16	3.00	2.64
Cash Flow Payback – After Tax	Years	8.28	7.10	4.98	3.76	3.30
NOTE: The value in bold reflects the base case.						

A sensitivity analysis was also undertaken on variable discount rates, gold feed grades, CAPEX and OPEX. The sensitivity analysis indicates that the project is most sensitive to gold price, followed by head grade, recovery, OPEX and then CAPEX.

1.21 ADJACENT PROPERTIES

In addition to the Imbo Project, there have been other mineral exploration activities in the Ngayu Greenstone Belt in recent times, and mineral resources have been defined within the belt. Since 2010, Loncor has been the largest permit holder in the Ngayu belt and has been exploring a number of prospects on its own since 2010 or in joint venture with Barrick Gold Congo SARL (formerly Randgold Resources Congo SARL) (Barrick Gold) from 2016 to 2021.

Loncor undertook exploration over priority target areas at Yindi, Makapela, Itali, Matete, Nagasa, Mondarabe, Anguluku and Adumbi West prospects with airborne magnetic and radiometric surveys, geological mapping, stream sediment sampling, soil and rock sampling, trenching, augering and ground geophysical surveys. During the period 2010 to 2013, Loncor undertook drilling programmes on a number of prospects in Ngayu and outlined mineral resources at Makapela in the west of the belt. At Makapela, a total of 56 core holes (18,091 m) were completed in the vicinity of the Main and North pits, and 15 holes (3,594 m) were drilled at nearby Sele Sele. In April 2013, Loncor announced mineral resource estimates for Makapela with an Indicated Mineral Resource of 0.61 Moz of gold (2.20 Mt grading at 8.66 g/t Au) and an Inferred Mineral Resource of 0.55 Moz of gold (3.22 Mt grading at 5.30 g/t Au). The deposit at Makapela is open down plunge and along strike.

Besides Makapela, Loncor drilled other prospects, and significant intersections were obtained at Yindi (21.3 m grading 3.3 g/t Au, 24.0 m grading 1.5 g/t Au and 10.3 m grading 4.1 g/t Au) and at Itali (38.82 m at 2.66 g/t Au, 14.70 m at 1.68 g/t Au and 3.95 m at 19.5 g/t Au). Further exploration including drilling is warranted on other prospects within the Ngayu belt including Yambenda, Mokepa and Mongaliema.

In terms of producing gold mines, the Kibali Gold Mine, approximately 220 km northeast by air from the Imbo Project, is located within the Archean-aged Moto greenstone belt and commenced gold production in September 2013. The mine is owned by Kibali Goldmines SA (Kibali), which is a joint venture company with 45 % owned by Barrick Gold, 45 % by AngloGold Ashanti, and 10 % by Société Minière de Kilo-Moto (SOKIMO). Barrick Gold is the operator and in 2020, Kibali produced 808,134 oz of gold at an AISC of US$778/oz of gold. Kibali had Measured and Indicated Mineral Resources of 15.5 Moz of gold, Inferred Mineral Resources of 1.5 Moz and Proven and Probable ore reserves at the end of 2020 of 9.33 Moz (from Barrick Gold 2020 Annual Report). Kibali is Africa’s largest producing gold mine.

1.22 INTERPRETATION AND CONCLUSIONS

1.22.1 Introduction

The Qualified Persons (QPs) note the following interpretations and conclusions in their respective areas of expertise, based on the review of the information available for this technical report.

1.22.2 Geology and Mineralisation

The Adumbi gold deposit is found within the Ngayu Archean greenstone belt, one of a number of Archean-aged, granite-greenstone belts that extend from northern Tanzania, into northeastern DRC and then into the Central African Republic. These gold belts contain a number of major gold mines including Kibali (DRC) and Geita, North Mara and Bulyanhulu (Tanzania). Gold deposits within these belts are associated with the globally important Neo-Archean orogenic gold deposits, examples of which are found in most Neo-Archean cratons around the world.

At the Adumbi deposit, the gold mineralisation is generally associated with quartz and quartz-carbonate-pyrite ± pyrrhotite ± arsenopyrite veins in a BIF unit. Examples of similar type BIF hosted gold deposits to Adumbi include the major Geita mine in Tanzania and Kibali mine in northeastern DRC.

1.22.3 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

Systematic exploration has been conducted on the Adumbi deposit and Imbo Project area, including airborne LiDAR (light detection and ranging) and geophysical surveys, gridding, geological mapping, soil, trench, adit and auger sampling together with a number of core drilling programmes. Sampling, sample storage, security, sample preparation and geochemical analyses and verification are considered appropriate for the resource estimate at Adumbi.

1.22.4 Mineral Resource Methodology and Estimation

The Mineral Inventory Statement undertaken on the Adumbi deposit uses the definitions and guidelines given in the CIM Definition Standards on Mineral Resources and Mineral Reserves and is reported in accordance with NI 43-101 requirements. The Adumbi Mineral Inventory for the various material types (oxide, transition and fresh) contained within the Adumbi practical pit designs consists of 1.883 Moz (28.185 Mt grading 2.08 g/t Au) of Indicated mineral resources and 1.777 Moz (20.828 Mt grading 2.65 g/t Au) of Inferred mineral resources. The data used for the resource estimate and methods employed are considered reasonable for the level of study for this PEA by the QP.

1.22.5 Open-Pit Optimisation and Mineral Inventory

Pit optimisation assumptions and parameters used to constrain the depth extent of the geological model to generate the mineral inventory of the open pit for the Adumbi deposit are considered appropriate for its location and infrastructural setting with appropriate metallurgical recoveries used from the test work and a gold price of US$1,600/oz, which is below current levels.

In the QP's opinion, the parameters used in the Mineral Resource to Mineral Inventory conversion process are reasonable.

1.22.6 Mining Plan and Methods

Over the LOM of Adumbi, a total of 49.77 Mt grading 2.17 g/t Au is expected to be delivered to the processing plant. The scheduling process consisted of developing a mine plan that is economically optimal using the inventory included in the practical pit designs and pit shell. The production schedule was based on a methodology of block selections from the block model inside the pit outlines.

The size and type of mining equipment is consistent with the size of the project, and equipment units and costs have been selected from major mining manufacturers. Although quotes have not been specifically requested for this project, the capital and transportation costs have been derived from similar projects from the relevant equipment suppliers. Outputs have been derived from generic calculations. Contract mining and associated costs were estimated for

Adumbi. The mine plan and estimated mine capital and operating costs are considered reasonable for this level of study.

1.22.7 Metallurgical Test Work and Process Recovery Plant

For the Adumbi deposit, representative core sample composites were selected for metallurgical test work from a range of depths and along strike and for the various mineralised host lithologies and styles of mineralisation. These representative samples were then submitted to an independent metallurgical laboratory for comminution and gold recovery test work to determine metallurgical recovery estimates using appropriate processing routes. The metallurgical test work undertaken is considered appropriate by the QP for the level of this study.

Results from the metallurgical test work programme were used by SENET to develop the process flowsheet and determine values to size and design the process plant. The processing plant is planned to process 49.77 Mt of material with an average head grade of 2.17 g/t Au and an average metallurgical recovery of 89.79 % to produce 3.12 Moz of gold. The processing plant has an annual throughput of 5.0 Mt. The process methods are conventional, with comminution and recovery processes widely used in the industry.

1.22.8 Infrastructure and Accessibility

The Adumbi deposit is situated in a remote part of the northeastern DRC, and infrastructure is limited. As part of the PEA study, a team of consulting engineers carried out site investigations to assess optimal positions for key infrastructure components of the mine site including potential tailings storage sites and HEP, diesel and PV power sources. Preliminary HEP sites were visited around Adumbi, indicating that a significant component for the estimated 32 MW requirement for the operation could be obtained from an HEP facility.

A number of potential access land routes were assessed from the major Port of Mombasa in Kenya via Uganda to Adumbi. The preferred route chosen is via Kenya and Uganda to Durba in DRC where Barrick Gold/AngloGold Ashanti’s Kibali Gold Mine is situated and then via Isiro and Wamba to Adumbi. Raw water for the project will be abstracted from the rivers in the area, which have significant flow throughout the year.

In the QP’s opinion, the estimated infrastructure CAPEX and OPEX are considered reasonable for this level of study.

1.22.9 Environmental and Social Considerations

The objectives of the environmental and social component of the Adumbi PEA were to evaluate the adequacy of the environmental and social studies undertaken to date and determine, at a screening level, the potential environmental and social issues associated with the proposed project development. This included identifying key issues, fatal flaws, high-level environmental and social risks that may affect the project and require further investigation, and identifying costs for further environmental and social studies.

The environmental assessment included knowledge from other recent operating gold mines and other development projects in eastern DRC. On the basis of the available information, screening studies undertaken did not identify any fatal flaws from an environmental and socio-economic perspective. Key issues were identified that would need to be investigated further during the PFS and FS phases of the project.

Six artisanal mining villages identified within the project footprint have been assessed for social repositioning. The total population of these villages is estimated to be approximately 5,800. The legal situation regarding compliance with the DRC resettlement plan process, assessment and reporting requirements is unclear on the repositioning of artisanal miners. Future studies (ESIS, ESMPP and MRP) will be undertaken to clarify this situation. However, a conservative approach has been taken in the estimation of costs for social repositioning of the artisanal settlements that will be impacted by the project development. It is believed that if the livelihood restoration programmes are accepted for the repositioning of artisanal miners within the Adumbi mine footprint, the capital included in the pre-production capital costs could be significantly lower than estimated in this PEA study.

1.22.10 Capital and Operating Cost Estimates

Pre-production capital costs are estimated for two power options: HEP hybrid and diesel only. For the HEP hybrid option, pre-production capital costs are estimated at US$530.05 million including US$74.42 million of contingencies while sustaining capital costs are estimated at US$304.96 million. Of the initial pre-production capital total, US$152.45 million is related to an HEP facility. However, Loncor is already in discussion with potential power suppliers with experience in the DRC to project finance and build a hydroelectric facility at Adumbi and then have an offtake agreement with Loncor to supply power for the mining operation. Such an agreement would improve the financial economics of the project by reducing the initial CAPEX for the HEP hybrid option.

For the diesel-only option, pre-production capital costs are estimated at US$392.01 million including US$62.96 million of contingencies. Sustaining capital for the diesel-only option is estimated at US$410.55 million.

The average LOM total cash costs for Adumbi are estimated at US$852/oz for the HEP hybrid option and US$908/oz for the diesel-only option, based on a diesel price of US$0.90/L. The AISC are estimated at US$950/oz for the HEP hybrid option and US$1,040/oz for the diesel-only option.

In the QP’s opinion, the CAPEX and OPEX for Adumbi are considered reasonable for this level of study.

1.22.11 Economic Analysis

Cash flow valuation models for the Adumbi HEP hybrid and diesel-only options were developed by the QPs by taking into account estimated annual processed tonnages, grades and recoveries, metal prices, site operating and total cash costs including royalties and refining charges, and initial pre-production and sustaining CAPEX estimates. The financial assessment of Adumbi has been carried out on a “100 % equity” basis for both pre-tax and post-tax considerations. The financial analysis assumed a base gold price of US$1,600/oz of gold, a two-year construction period, and an operating mine life of 10.3 years.

The pre-tax net present value discounted at 5 % (NPV at 5 %) is US$895 million for the HEP hybrid option and US$843 million for the diesel-only option. The post-tax net present value discounted at 5 % (NPV at 5 %) is US$624 million for the HEP hybrid option and US$600 million for the diesel-only option. The sensitivity analysis indicates that the project is most sensitive to gold price, followed by head grade, recovery, OPEX and then CAPEX.

The QP considers that the results from this economic valuation have outlined a robust project worthy of further follow-up feasibility studies.

1.22.12 Risks and Uncertainties

Considering Adumbi is at an early stage of development, with PFS and FS required before the project can advance to the development stage, a number of risks have been outlined below:

- Risks to the resource estimate resulting from future drilling
- Risks related to the interpretations of mineralisation geometry and continuity in the mineralised zones
- Changes to geotechnical and hydrogeological assumptions
- Changes to power assumptions
- Changes to historical mining assumptions
- Future metallurgical test work yielding results that vary from the current test work undertaken with lower metallurgical recoveries
- Estimated capital and operating costs varying from the current estimates
- Changes to the assumed gold price of US$1,600/oz used in this PEA model
- Delays in progressing the project due to security problems
- Inability of Loncor to obtain sufficient financing to advance Adumbi to the PFS and FS stages. Also, any future mine construction at Adumbi would require significant financing. No assurance can be provided that such financing would be available to Loncor.

The future PFS and FS stages will aim to reduce the risks and uncertainties associated with future development of Adumbi.

1.23 RECOMMENDATIONS

Based on the positive results of this PEA, further work is warranted at Adumbi to advance the project up the value curve by completing follow up feasibility studies on the project. A number of opportunities have been identified to increase the mineral resources and enhance and increase the economics and financial returns at Adumbi. It is recommended that Loncor follow up on these opportunities, which include the following:

- **Increasing and Upgrading Mineral Resources at Adumbi and within the Imbo Project**
  There is excellent exploration potential to further increase the mineral resources at Adumbi and within the Imbo Project. At Adumbi, the mineralised BIF host sequence increases in thickness below the open-pit shell, and wide-spaced drilling has already intersected grades and thicknesses amenable to underground mining. Further drilling is required to initially outline a significant underground Inferred Mineral Resource which

can then be combined with the open-pit mineral resource so that studies can be undertaken for a combined open-pit and underground mining scenario at Adumbi. It is also recommended that infill drilling be undertaken in the deeper part of the open-pit shell to upgrade the current Inferred resources into the Indicated category. Besides increasing the resource base, a combined open-pit/underground project could increase grade throughput and reduce strip ratios with the higher grade, deeper mineral resources being mined more economically by underground mining methods, which could increase annual gold production and drive down operating costs. Minecon also recommends that further studies should be undertaken to assist in estimating historical depletions and depletions by recent artisanal mining.

Additional deposits and prospects occur close to Adumbi and have the potential to add mineral resources and feed to the Adumbi operation. Along trend from Adumbi, the Manzako and Kitenge deposits have Inferred Mineral Resources of 313,000 oz of gold (1.68 Mt grading 5.80 g/t Au) and remain open along strike and at depth. Further drilling is warranted on these two deposits

- Along the structural trend, 8 km to 13 km to the southeast across the Imbo River and within the Imbo Project, four prospects (Esio Wapi, Paradis, Museveni and Mungo Iko) with similar host lithologies to Adumbi have been outlined with soil, rock and trench geochemical sampling. An initial shallow, scout drilling programme should be undertaken on these four prospects to determine their mineral resource potential.

- **Additional Mineral Resources within the Ngayu Greenstone Belt**
  Additional feed for the Adumbi processing plant could also come from Loncor's 100 % owned high-grade Makapela deposit, where Indicated Mineral Resources of 2.20 Mt grading 8.66 g/t Au (614,200 oz of gold) and Inferred Mineral Resources of 3.22 Mt grading 5.30 g/t Au (549,600 oz of gold) have been outlined to date with the high-grade material being able to be transported economically to Adumbi.

- **Additional geotechnical investigations**
  Additional geotechnical investigations including drilling are recommended to optimise and potentially steepen pit slopes especially for the competent fresh BIF host rock which could reduce the strip ratio and thereby lower mining costs at Adumbi.

- **Further metallurgical test work**
  Additional metallurgical test work, including additional flotation and petrographic studies, is recommended to confirm recoveries and reagent consumptions, and to optimise the flowsheet design.

- **Concluding Hydroelectric Hybrid Power Agreement with Third Parties**
  As described in this PEA study, hydroelectric sites have already been identified close to Adumbi, and further studies are required to optimise the power generation concept for the operation. Loncor is already in discussion with potential power suppliers with experience in the DRC to project finance and build a hydroelectric facility at Adumbi and then have an offtake agreement with them.

2 INTRODUCTION

Minecon Resources and Services Limited (Minecon) and New SENET (Pty) Ltd (SENET) were commissioned by Loncor Gold Inc. (Loncor) to prepare an independent National Instrument 43-101 (NI 43-101) Technical Report relating to the Preliminary Economic Assessment (PEA) of Loncor's Adumbi deposit within its 84.68 % owned Imbo Project, located in the northeastern part of the Democratic Republic of the Congo (DRC), as per the requirements of applicable Canadian securities laws.

Table 2.1 indicates the details of the Qualified Persons for this technical report and the sections that they have contributed to.

Table 2.1: Qualified Persons and Their Contributions

Qualified Person	Company	Contribution
Daniel Bansah	Minecon	Mineral resources, mining studies and Environmental Social Impact Assessment, compilation and review of report
Christian Bawah	Minecon	Management of drilling, geological and structural interpretation, compilation and review of report
Philemon Bundo	SENET	Metallurgical test work interpretation Processing plant and project infrastructure Economic evaluation Coordination and compilation of report

2.1 QUALIFICATIONS OF QUALIFIED PERSONS

The relevant sections of this NI 43-101 Technical Report were compiled by the Qualified Persons, as this term is defined in NI 43-101. The certificates of the Qualified Persons (QPs) are set out after the Date and Signature Page at the end of the report. A summary of their qualifications and responsible sections is given in Table 2.2.

Table 2.2: Summary of the Qualifications and Responsibilities of the QPs

QP	Qualification	Company	Site Visit	Responsibility (Section of Report)
Daniel Bansah	MSc (MinEx), MAusIMM (CP), FWAIMM, MGhIG	Minecon	Yes	Sections 2 to 5, 11 and 12, 14 to 16, 20 and 23, and part of Section 1 and Sections 24 to 27
Christian Bawah	BSc (Hons) Geology, MBA (Finance) MAusIMM (CP), MMCC, FWAIMM, MGhIG	Minecon	Yes	Sections 6 to 10 and part of Section 27
Philemon Bundo	BSc Eng Hons (Metallurgy), FAusIMM (Aus)	SENET	No	Sections 13, 17, 18, 19 21, 22 and part of Sections 1, 24, 25, 26 and 27

SENET provided all the process engineering services and also managed the other consulting companies:

- Epoch Resources: Tailings storage facility
- DRA Energy: Investigating suitable options for the power supply for the project, in conjunction with Knight Piésold

Minecon used the additional drilling data from the drilling completed since the previous resource estimates of April 2021 and completed the mining and environmental studies on the Adumbi deposit as part of the PEA.

The purpose of this report is to support the public disclosure of the PEA of the Adumbi deposit. This report is intended to comply with the requirements of NI 43-101, including Form 43-101F1.

Loncor is a Canadian gold exploration company with a substantial footprint in the DRC. Loncor's shares trade on the Toronto Stock Exchange. This report will be publicly filed by Loncor on SEDAR and EDGAR and may also be filed on Loncor's website.

2.2 TERMS OF REFERENCE AND PURPOSE

This technical report describes the Adumbi deposit (as well as other properties within the Imbo Project) in terms of its historical and recent exploration (infill and extension drilling), and summarises the results of the PEA completed on the Adumbi deposit. The resource modelling and estimations were restricted to the Adumbi deposit due to the significant implications of the drilling work carried out on the mineral resources of Adumbi. The economic assessment was performed by SENET and Minecon during the preparation of this report. The effective date of this report is December 15, 2021.

Loncor is a Canadian gold exploration company focused on the Ngayu Greenstone Belt in the DRC. The Loncor team has over two decades of experience of operating in the DRC. Ngayu has numerous positive indicators based on the geology, artisanal activity, encouraging drill results and an existing gold resource base. The area is 220 km southwest of the Kibali Gold Mine, which is operated by Barrick Gold (Congo) SARL (Barrick). In 2020, Kibali produced 808,134 oz of gold at all-in sustaining costs of US$778/oz.

Resolute Mining Limited (ASX/LSE: RSG) owns 23 % of the outstanding shares of Loncor and holds a pre-emptive right to maintain its pro rata equity ownership interest in Loncor following the completion by Loncor of any proposed equity offering.

The Imbo Project, in which the Adumbi and the two neighbouring deposits of Kitenge and Manzako are situated, is located within the Mambasa District of the Ituri Province in the northeastern region of the DRC, 250 km west of Bunia, the capital of the Ituri Province, and 225 km northwest of the city of Beni. The Adumbi base camp is located at latitude 1° 43' 58.76" N and longitude 27° 52' 4.01" E or 596,522 m E and 191,570 m N (WGS 84 UTM Zone 35N). Loncor holds an 84.68 % interest in the Imbo Project, and the balance is held by minority shareholders, including a 10 % free-carried interest owned by the DRC Government.

2.3 SOURCES OF INFORMATION

Minecon and SENET have relied upon various reports and information provided by Loncor and other experts. The document references are summarised in Section 27 and include internal documents compiled by Loncor and the previous owner of the Imbo Project, Kilo Goldmines Ltd (Kilo) . Minecon particularly relied on the Roscoe Postle Associates Inc. (RPA) NI 43-101 Technical report of February 28, 2014, including its recommendations as well as technical information provided by Loncor on all the work carried out between 2014 to date by Loncor and previously by Kilo. In particular, the results of Loncor's 2020 to 2021 drilling programme have been utilised in developing the new estimates. Additionally, digital maps and information available in the public domain, such as company websites and public library documents, have been utilised.

Loncor openly provided a hard drive containing all material information which, to the best of its knowledge and understanding, is complete, accurate and true, having made due enquiry. Neither Minecon nor SENET is aware of any current or pending litigation or liabilities attached to the Imbo Project.

2.4 SCOPE OF THE OPINION

This report has been compiled to incorporate all currently available and material information that will enable the reader to make a reasoned and balanced judgement regarding the PEA of the Adumbi deposit.

The Qualified Persons involved in the preparation of this report are members in good standing with their respective professional institutions.

This work has been based upon technical information which has been supplied by Loncor and its contractors, and Minecon and SENET carried out independent due diligence on the information, where possible.

Minecon and SENET confirm that, to the best of their knowledge and having taken all reasonable care to ensure that such is the case, the information contained in this report is in accordance with the facts and contains no omission likely to affect its import.

The Mineral Resource estimates on Kitenge and Manzako were prepared by RPA in 2014. These estimates have not been updated.

2.5 QUALIFIED PERSONS DECLARATION AND STATEMENT OF INDEPENDENCE

This report has been compiled by Minecon and SENET. Minecon and SENET have extensive experience in preparing technical, competent/qualified persons', technical and valuation reports for mining and exploration companies. The information in this report is based on information compiled by the Qualified Persons: Daniel Bansah, Christian Bawah and Philemon Bundo. The Qualified Persons' certificates are set out after the Date and Signature Page at the end of the report. The Qualified Persons have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Qualified Person, as defined in NI 43-101.

Neither Minecon nor SENET nor their respective staff or consulting engineers have, or have had, any interest in any of Loncor's projects capable of affecting their ability to give an unbiased opinion and have not received, and will not receive, any pecuniary or other benefits in connection with this assignment, other than normal geological, mining and environmental consulting fees. Neither Minecon nor SENET nor any of their respective personnel involved in the preparation of this report have any material interest in Loncor or in any of the properties described herein.

Minecon and SENET were remunerated a fixed fee amount for the preparation of this report, with no part of their fees contingent on the conclusions reached or the content of this report. Except for these fees, neither Minecon nor SENET has received, and nor will they receive, any pecuniary or other benefit whether direct or indirect for or in connection with the preparation of this report.

2.6 PERSONAL INSPECTION

A site visit was carried out by Daniel Bansah, Chairman and Managing Director of Minecon, in September 2021, together with Minecon's environmental and social scientists. During the visit, Daniel spent time reviewing all the field geological activities undertaken on the Adumbi deposit, the geological logging and the sampling procedures, including the sampling preparation protocols carried out in the sample preparation laboratory. Christian Bawah was also on site for a period of 8 weeks from October to November 2020. Christian was accompanied by Peter Kersi, a contributing engineer to this report. The following Minecon geologists and technical personnel: Bel Mapendo, chief geologist, Patient Zamakulu and Jean-Alain Chish, both senior geologists, and three of Minecon's laboratory technical and operational staff were on site for a period of 16 weeks, providing technical supervision and management of the 2020 to 2021 drilling programme and the management of the on-site sample preparation facility.

Tasks undertaken during the visit included a technical inspection of the site (proposed waste dump and other infrastructural sites including but not limited to six artisanal villages that could be impacted by the mine infrastructure), an inspection of the old drill core and a review of all the technical work carried out. In addition, the team reviewed the sampling and drill site protocols and security including QA/QC issues, and the on-site sample preparation facility.

Gordon France, Minecon's Database, GIS and IT Manager visited the Adumbi site for seven weeks from June to July 2021. The scope of work during the visit was to ensure that the Adumbi database was migrated onto a centralised data repository (the Century Database System).

The Minecon team worked in collaboration with Fabrice Matheys, Loncor's General Manager and geologist with +25 years of experience in the DRC and the African region.

3 RELIANCE ON OTHER EXPERTS

Minecon and SENET have prepared this technical report and, in so doing, have utilised information provided by Loncor and its contractors as to its operational methods, conclusions, opinions, estimates and forecasts. Where possible, this information has been reviewed by independent sources with due enquiry in terms of all material issues that are a prerequisite to comply with NI 43-101.

The authors of this report are not qualified to provide extensive commentary on legal matters associated with Loncor's right to the Imbo Project. The authors have therefore relied on the legal opinion of MBM-Conseil of Kinshasa Gombe, DRC, dated June 8, 2020, which has provided certain information in preparing this report which, to the best of Loncor's knowledge and understanding, is complete, accurate and true, and Loncor acknowledges that Minecon and SENET have relied on such information, in preparing this report. No warranty or guarantee, be it express or implied, is made by the authors with respect to the completeness or accuracy of the said legal matters.

Except as provided under applicable Canadian and US securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The 122 km^2 Imbo Project is located within the Mambasa Territory in the Ituri Province in the northeastern region of the DRC, 325 km northeast of the main cities of Kisangani and 225 km northwest of Beni (see Figure 4.1). The Imbo Project is found within Imbo Exploitation Permit PE 9691, which is valid until February 2039.

Bunia is the provincial capital of the Ituri Province and is situated approximately 260 km east by air from the Imbo Project. The village of Nia-Nia is approximately halfway between Beni and Kisangani and situated approximately 45 km south, by road, of the Adumbi base camp. The Adumbi base camp is located at latitude 1° 43’ 58.76” N and longitude 27° 52’ 4 01” E or 596,522 m E and 191,570 m N in WGS 84 UTM Zone 35N (see Figure 4.2 and Figure 4.3).

Figure 4.1: Locality Map of the Imbo Project in Africa

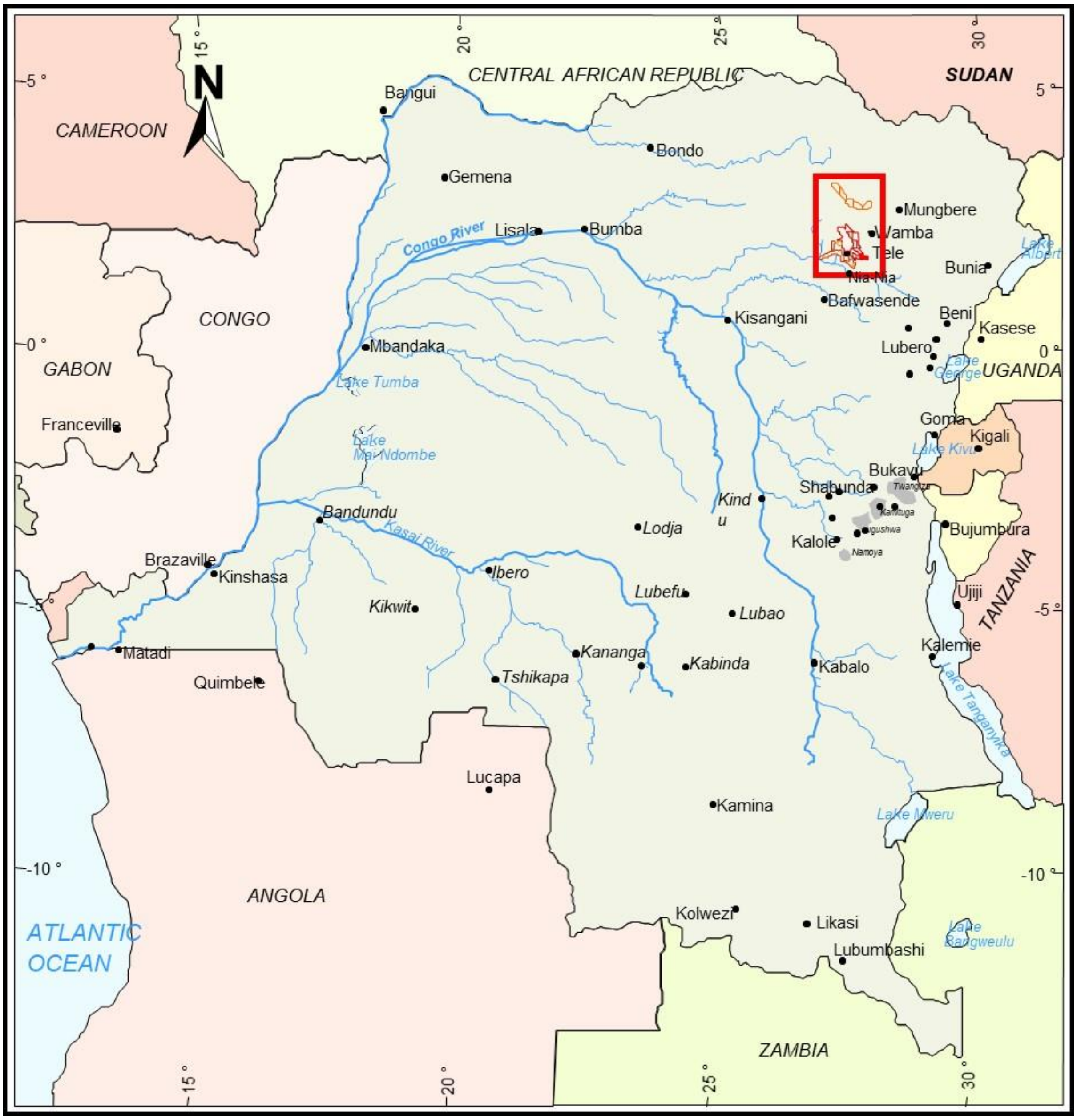


Figure 4.2: Location of Imbo Project within the DRC

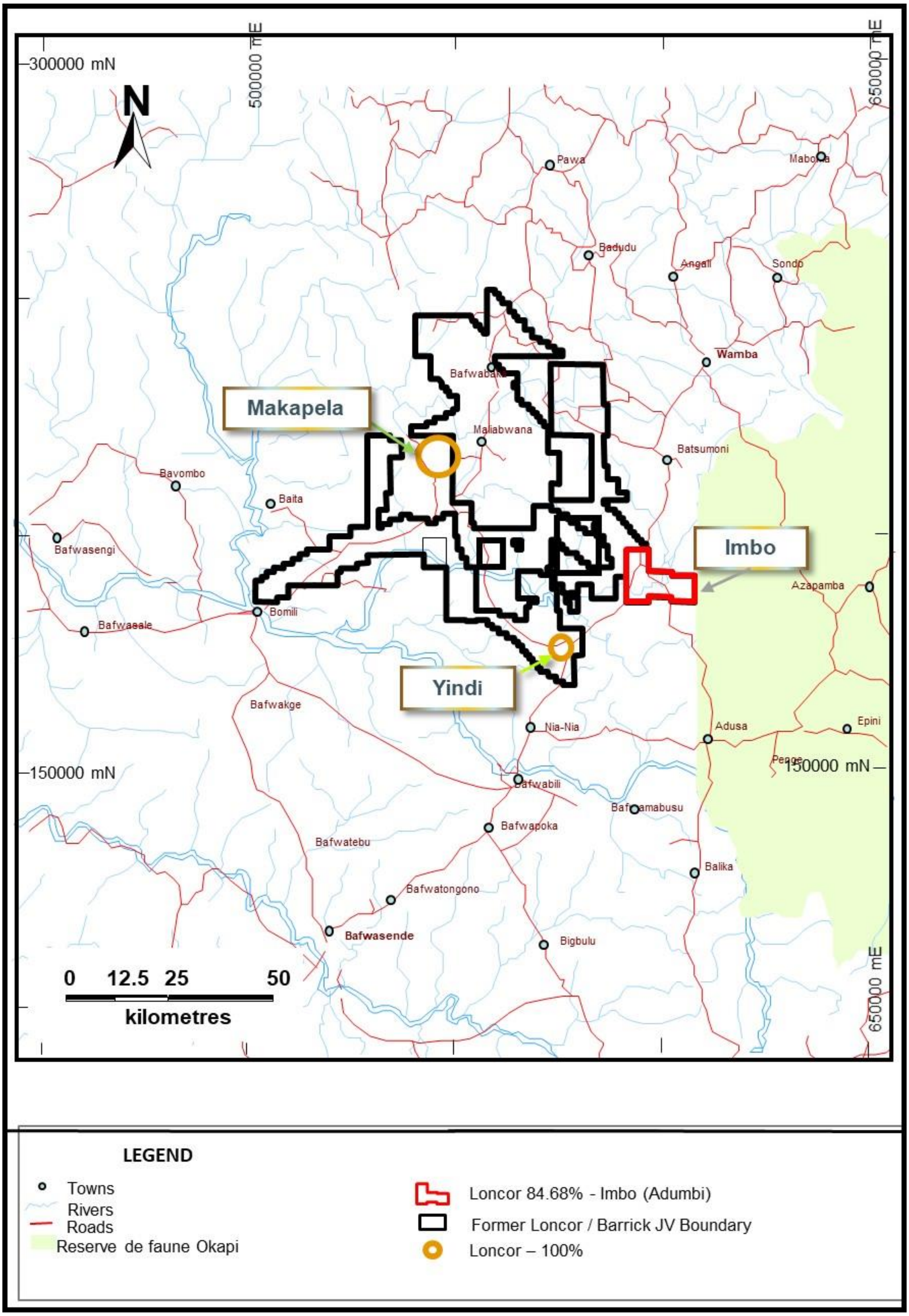


Figure 4.3: Locality Map of Imbo Project

4.2 PROPERTY OWNERSHIP

Loncor is a publicly listed Canadian company which owns 84.68 % of the Imbo Exploitation Permit through its subsidiary Adumbi Mining S.A. (Adumbi Holdco). The minority shareholders hold 15.32 % (including the 10 % free-carried interest owned by the DRC Government).

4.3 LAND TENURE

In accordance with the Mining Regulations of the DRC, the surface area of an exploitation permit is measured in a unit known as a “carré” (in English, a square), which is defined as an area that measures 30 s on each side. The sides must be oriented north-south and east-west. A square carré has an area of 84.955 ha or 0.84955 km^2. The word “quadrangle” is used as the unofficial English translation of the word carré.

4.4 IMBO EXPLOITATION PERMIT

Minecon has relied on a letter on land tenure, licences and permits dated June 8, 2020, from MBM-Conseil, one of the leading firms practising mining law in the DRC.

The Imbo Exploitation Licence (PE 9691) lies between X 594500 and 596000 and Y 191500 and 193100 (WGS 84 Zone 35N UTM coordinates). Table 4.1 lists the carré corners for the Imbo Exploitation Permit in longitude and latitude.

Table 4.1: Coordinates of the Imbo Exploitation Permit (PE9691)

Corner	Longitude	Latitude
1	27º 50' 00"	01º 41' 00"
2	27º 50' 00"	01º 47' 00"
3	27º 53' 00"	01º 47' 00"
4	27º 53' 00"	01º 44' 30"
5	27º 56' 00"	01º 44' 30"
6	27º 56' 00"	01º 44' 00"
7	27º 59' 00"	01º 44' 00"
8	27º 59' 00"	01º 41' 00"

The Imbo Licence covers a total area of 122 km2 (12,234 ha) and consists of 144 carrés.

The deposits and prospects on the Imbo Exploitation Permit, from northwest to southeast as shown in Figure 4.4, include the following:

- Adumbi Deposit, including Canal
- Bagbaie (previously known as Adumbi North) Prospect
- Adumbi West Prospect
- Amuango Prospect
- Monde Arabe Prospect

- Vatican Prospect
- Kitenge Deposit, including Senegal
- Manzako Deposit, including Lion
- Imbo East (previously termed Maiepunji) Prospects, including Paradis, Museveni, Esio Wapi and Mungo Iko

Adumbi is currently the most explored deposit within the Imbo Permit. The Kitenge deposit is located approximately 4 km southeast from Adumbi. The Senegal prospect has been incorporated into the Kitenge deposit as it is the probable fault-offset northwest continuation along strike of Kitenge.

Manzako is located 1.5 km northeast of Kitenge. The previously named Lion prospect is now considered to be the southeastern portion of Manzako which incorporates a series of sub parallel shear structures.

The Monde Arabe and Vatican prospects are located east of Adumbi. Amuango is situated west of Adumbi, and the Imbo East prospects are located approximately 5 km southeast of Manzako.

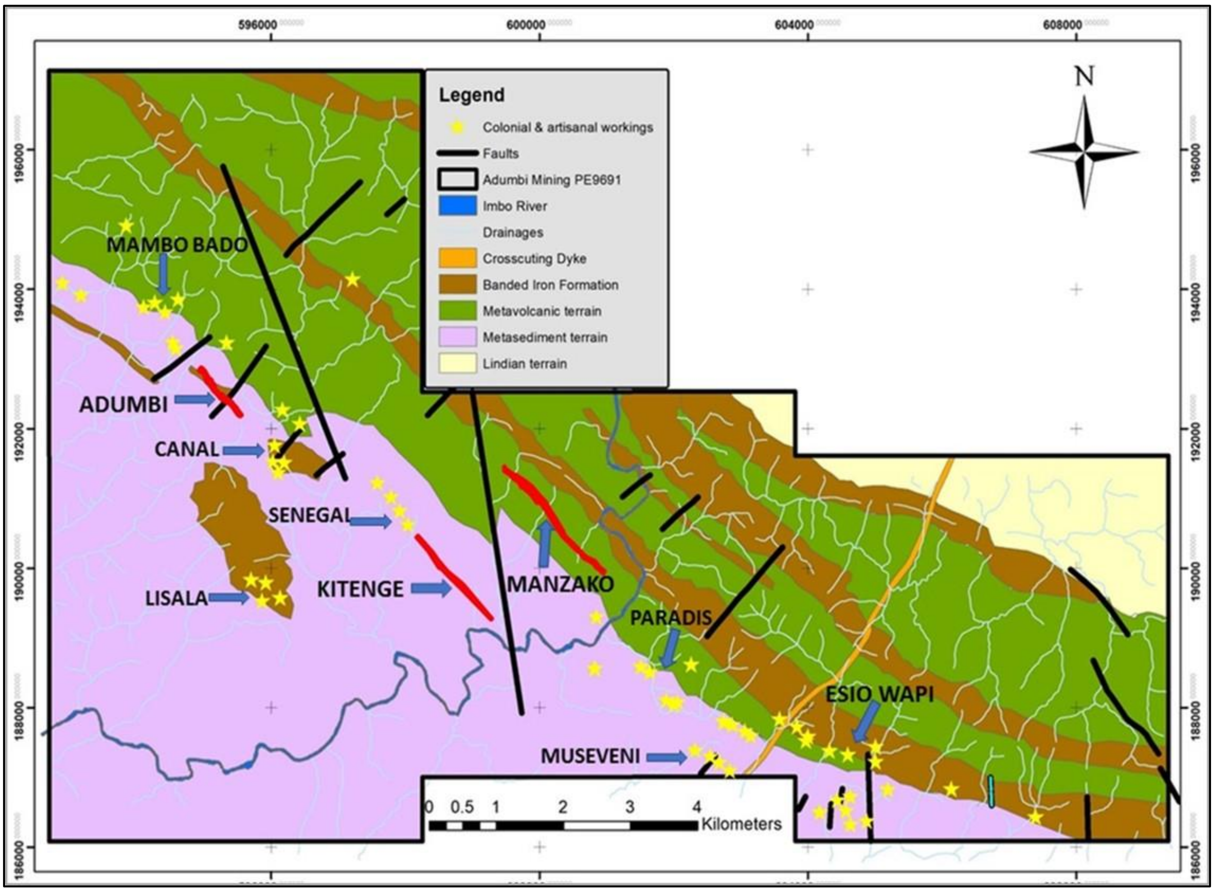


Figure 4.4: Imbo Project Simplified Geology

4.5 PERMITS

Adumbi Holdco does not have a work permit précis; however, they have provided Minecon with a copy of a DRC “attestation de travail”, which is a document confirming that the Imbo Exploitation Permit is in order.

4.6 ENVIRONMENTAL LIABILITIES AND PERMITTING

DRC law imposes environmental obligations on an exploitation permit holder which must be performed during the exploitation of the mine. Pursuant to its decision dated April 2, 2013, the Directorate of Environment has approved the Environmental Impact Study (EIS) and Environmental Management Plan of the Project (EMPP). Furthermore, the Mitigation and Rehabilitation Plan (MRP) was approved on April 2, 2013.

4.7 SURFACE USAGE/LAND LEASE

Article 64 of the DRC 2002 Mining Code provides that the exploitation permit entitles its holder to the exclusive right to carry out, within the perimeter over which it has been granted, and during its term of validity, exploration, development, construction and exploitation works in connection with the mineral substances for which the permit has been granted, and associated substances if the holder has applied for an extension. According to Article 280 of the Mining Code, the holder or lessee must compensate for the damages caused by the works it carries out in connection with its mining activities, even if they are authorised.

In order to maintain the validity of the permit, the holder must pay the annual surface fees per quadrangle for each subsequent year before the end of the first quarter of the calendar year. The surface annual fees for the Imbo Permit have been paid for the year 2021.

Minecon is not aware of any environmental liabilities on the property. Loncor has all the required licences and permits to conduct the proposed work on the property. Minecon is not aware of any other significant factors, other than potential political and related safety risks described in Section 24 that may affect access, title, or the right or ability to perform the proposed work programme on the property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES,INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Imbo Project is located within the Mambasa Territory in the Ituri Province of the DRC. Bunia is the provincial capital of the Ituri Province and is situated approximately 260 km east by air from the Imbo Project. Located approximately 225 km by air southeast of the property, Beni is the nearest major population centre to the Imbo Project and has a population of approximately 230,000. Loncor maintains an administrative office in Beni. The city is a United Nations MONUSCO base and has a lateritic airstrip with scheduled internal flights to other towns such as Goma, Bunia, Isiro, Kisangani and Kinshasa. The Isiro airstrip is approximately 200 km by lateritic road to the Imbo Project. From Beni, the Imbo Project is accessible via 322 km of lateritic road to Nia-Nia, then to Village 47 (47 km north of Nia-Nia) and then 7 km via lateritic roads to the Adumbi base camp. On the property, access is via trails using Mine Mule utility and four-wheel drive vehicles in addition to motorcycles. Away from areas of habitation and artisanal activity, access is on foot through the dense forest growth.

The nearest international airport is located at Entebbe in western Uganda and linked by 440 km of paved road to the Kasindi Uganda-DRC border, followed by 80 km of unpaved lateritic roads to Beni. Entebbe has international scheduled flights to South Africa, Europe and Asia and is also linked to other African countries as well as the in-country towns of Kinshasa and Lubumbashi via Nairobi (Kenya). Ethiopian airlines have direct flights between Addis-Ababa and Goma. In addition, Entebbe is linked to the DRC border points of Arua, Mahagi and Kasindi by paved highway from the deep seaport of Mombasa (Kenya). Due to security issues and the poorly maintained roads in the DRC, the preferred road from Kampala to access the property is via Arua/Aru to Doko (Kibali Mine) to Faradje to Dungu and Isiro. Rail links between Mombasa and Kasese (Uganda) are being upgraded to standard gauge.

At Nia-Nia, 52 km southwest of the Imbo Project, there is a 1,200 m long grass-covered, laterite base airstrip which can accommodate propeller-driven, charter aircraft including medium-sized cargo planes.

The large operating gold mine of Kibali is located approximately 220 km by air northeast of the Imbo Project (see Figure 5.1).

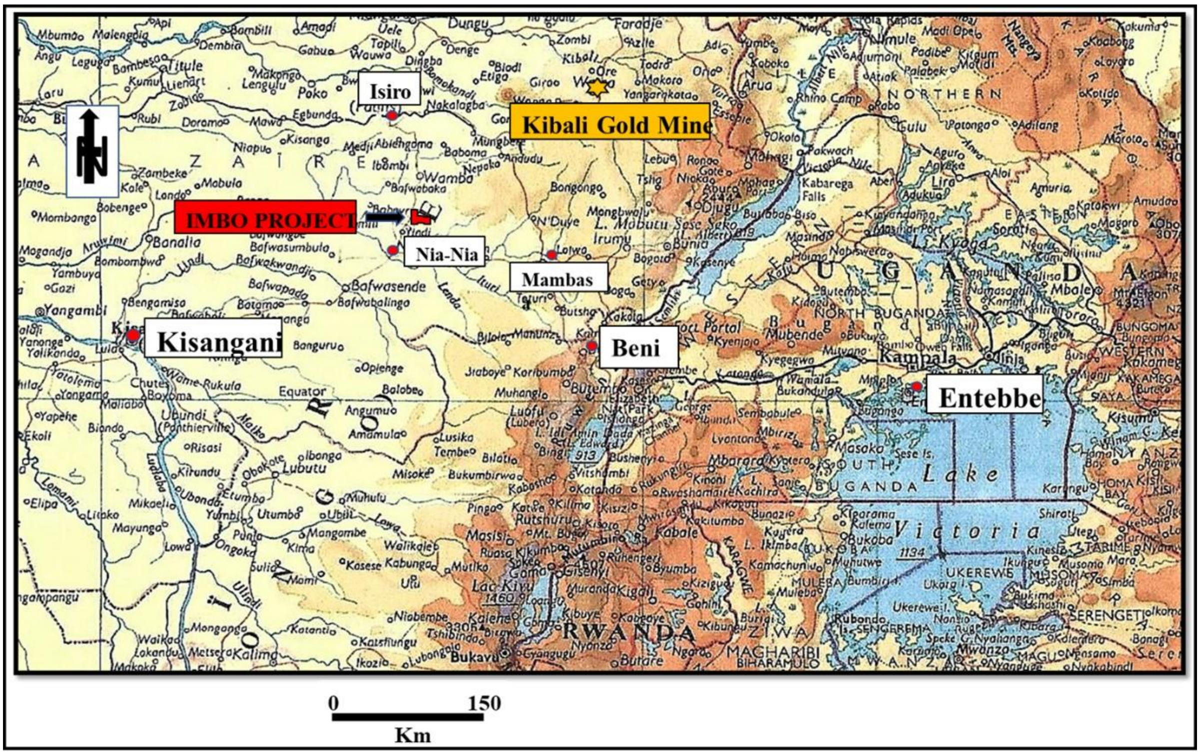


Figure 5.1: Accessibility and Locality Map

5.2 CLIMATE

The climate is typically tropical and is characterised by a long, wet season and a short dry season of up to three months from mid-December to mid-March. The average annual rainfall is approximately 2,000 mm to 2,500 mm, with the highest rainfall generally occurring in October. Even in the driest months, rainfall totals more than 50 mm. Temperatures are also uniformly high throughout the year, and there is little diurnal variability, varying between 19 °C and 23 °C, with daily lows and highs of 16 °C and 33 °C, respectively. Humidity is high throughout the year (75 % to 99 %).

The climate facilitates exploration and mining activities all year round although exploration is more challenging during the wettest months as roads can deteriorate as they are poorly maintained. Torrential downpours of rain are experienced; however, they are not generally long lasting. The prevailing wind direction is from the southeast, with the maximum wind velocity and average daily wind velocities being relatively low, approximately 12 m/s and 0.5 m/s, respectively. Notwithstanding, the area can be hit with severe storms. Climatic conditions have generally not affected exploration activities.

5.3 LOCAL RESOURCES

The land around the Imbo Project is mainly equatorial rainforest, with very tall trees and grass. A few small villages exist around the project area. Some wild animals exist in and around the project area, but most have been hunted out by the local population. Natural water sources are abundant. Recent water wells drilled in the community and inside the Adumbi base camp have produced high yields, confirming the groundwater potential in the project area. The closest hydroelectric power station is situated near Kisangani, together with hydroelectric stations supplying power to Barrick/AngloGold Ashanti's Kibali mine. Isiro and Beni are potential sources of skilled manpower, but there is sufficient local unskilled manpower in the surroundings of Adumbi.

Regional migration from the colonial period has resulted in an amalgam of people from different ethnic Bantu groups, along with indigenous populations of pygmies, residing in areas immediately adjacent to and along key transit routes to the Imbo Project.

Within the immediate area of the property, there are several small villages that generally consist of fewer than 300 residents. The estimated total population within 10 km of the surrounding area is approximately 8,500 who rely on subsistence farming, organised artisanal mining, and harvesting of wood. These villages are accessed by motorcycle, bicycle and on foot via unmaintained roads and trails. The nearest community to the Adumbi base camp is the Adumbi village. In general, the project enjoys the support of local communities.

Exploration supplies are generally sourced within the country or further afield in Uganda, Kenya, Tanzania or South Africa. Wherever possible, food and consumables are locally sourced. Manpower at the Adumbi base camp is sourced from the local area. Technical manpower consists of senior staff expatriates supplied by Minecon in addition to Congolese staff. Security is maintained by a private security agency as well as contracted posted DRC police officers.

There is a significant local labour pool available for training and recruitment for any envisioned mining operation. The local area would however not be capable of supplying sufficient materials other than timber to support any potential mine-site infrastructure. Although some main roads dissect the area, upgraded and additional access roads, including bridges, will be required for any potential large-scale mining operations.

There is no electrical distribution system within the local area, and diesel generators and solar power are relied upon. There are a number of potential locations for hydroelectric development located within the Imbo Project area. In September 2021, Knight Piésold carried out an orientation visit to the sites, and based on a positive outlook, they have recommended detailed site investigation studies.

At the time of Minecon's site visit in September 2021, infrastructure at the Adumbi base camp included the following:

- A fenced and gated compound that is patrolled by security and covers an area of approximately 8.5 ha
- A helicopter landing pad and privately operated weather station
- A brick-constructed administrative office building
- A wood-constructed first-aid post
- A brick-constructed kitchen and mess hall
- A brick-constructed washroom and shower facility
- Two private brick-constructed units for accommodations for nine people
- Private tented bedroom accommodations on concrete pads

- An outdoor recreational area with barbecue and satellite television
- Security office and camp support staff accommodation
- Gated and fenced core processing area consisting of the following:
    - Brick-constructed exploration office
    - Outdoor roofed open core logging areas
    - Outdoor roofed open core sawing area
    - Container storage for pulps and duplicates
    - Core storage racks
- Gated and fenced sample preparation facility
- Brick-constructed office and storerooms for drillers

The power supply at the site is provided by diesel generators with solar power also used for lighting. Water is taken from a natural spring located just outside the camp boundary. For any future development activities, it will be necessary to build all-weather access roads and bridges as well as infrastructure for sufficient power and water supplies. The Imbo Project surface rights allow sufficient areas for potential processing plant sites, tailings storage areas, and waste disposal areas.

5.4 PHYSIOGRAPHY

The Imbo Project is located in the Ituri tropical rainforest within the upper reaches of the Congo River Basin. The project area topographically consists of an undulating terrain that varies from approximately 600 m above sea level to approximately 800 m above sea level. Most of the landscape is covered with dense evergreen forests with a closed canopy; however, the hills tend to have relatively steep slopes, and the valley floors within the areas of the linear hills are relatively narrow. In most places, the overburden (in general less than 1 m to approximately 50 m in thickness) is oxidised sandy clay or sandy clay loam, ranging in colour from reddish brown through ochre to yellowish brown. The soils are acidic in nature, and the layer of humus is thin.

The property is drained by numerous creeks and streams. Almost all the landscape belongs to the Congo Basin and is covered with a dense network of permanent watercourses which flow into the Upper Ituri and its main tributaries: the Epulu, Nepoko, Nduye, Lenda, Ebiena, and Ngayu rivers.

The Adumbi deposit is well situated for potential mining development as it is located on a topographical high amenable to low strip ratios for initial mining access. The Kitenge and Manzako deposits are located in areas of less relief.

6 HISTORY

This section summarises the work completed on the Imbo licence area and in particular the drilling activities completed on the Adumbi deposit since the last update. The history of past exploration activity on the Imbo Project was originally summarised in the RPA NI 43-101 technical report entitled “Technical Report on the Somituri Project, Imbo Licence, Democratic Republic of the Congo” and dated February 28, 2014 (available from SEDAR at www.sedar.com).

Kilo contracted the Royal Museum for Central Africa (RMCA) in December 2006 to carry out a compilation of the RMCA archives on gold in the region of the Adumbi Project in the DRC. The historical exploration and historical gold production on the Imbo Project area outlined below is therefore based on the 2007 RMCA compiled report (RMCA, 2007). Most of the data available to the RMCA was from prior to the 1960 independence of the DRC.

6.1 PRIOR OWNERSHIP

The mining rights for the mineral concessions in the Imbo Project area were held by the Société Internationale Forestière et Minière du Congo (FORMINIÈRE or FRM) from the 1920s to the late 1950s. The colonial state was co-owner of a 50 % stake in FRM, with the remainder held by American interests. The Société Minière de la Tele (SMT), a subsidiary of FRM, oversaw development and exploitation. Following political independence in 1960, ownership changed hands multiple times. A Zairian company, Zafrimines, held the property licences from April 17, 1987. In 1997, Rhodes Mining NL of Australia entered a joint venture agreement with Busico of Uganda (20 %) and the DRC (20 %) and held the property licences from May 17, 1997, until August 2, 1998, when Kilo acquired the property.

6.2 EXPLORATION HISTORY

Belgian prospectors were the first to discover gold on the Imbo Project in the early 1900s, with gold production focusing on alluvial deposits until the late 1930s. Primary gold mineralisation was later discovered in the bedrock of the alluvial zones and was exploited in shallow pits and trenches. This was later followed by mining from deep trenches and underground galleries. From the mid-1970s to mid-1980s, the French Geological Survey (BRGM) undertook geological investigations of the Imbo Project area. Artisanal miners in organised groups in recent years have been exploiting alluvial and eluvial deposits, as well as oxidised mineralisation from deep trenches (up to 10 m), and the underground sill drifts and pillars at Adumbi.

Highlights of the reported historical exploration include the following:

- 1925: FRM completed the evaluation of interesting sites, and SMT was granted the rights for exploitation. It is reported that, during the Belgian exploitation, no geological maps were produced, and the operators mainly looked for mineralisation in quartz veins. Shallow exploration shafts or pits were systematically sited along the veins to facilitate delineation of the mineralisation.
- 1948: Manzako surface trenches and underground exploration discovered mineralised veins. It is reported that underground exploration drifts were driven at levels of −11 m,

−16 m, −30 m and −40 m below surface. Exploration on the −16 m level encountered generally low average grades with local high grades of 202.0 g/t Au (30 cm quartz) and 47.9 g/t Au (20 cm quartz and schists).

- 1940 to 1950: SMT conducted extensive surface and underground exploration in the Adumbi Hill area. BHP (1989) reports that trenching was undertaken on the surface and that adits, tunnels, and crosscuts were developed on three levels underground (the 721, 771, and 821 levels). Channel sampling was undertaken at 1 m intervals.
- 1973 to 1975: BRGM's Northern Zaire Project studied the geology of the area in detail. In 1975, BRGM conducted stream sediment and alluvial prospecting and produced a summary report. Arsenic anomalies were found to surround the historical mined areas, especially around Kitenge. BRGM noted that the quartz veins were irregular, erratically distributed, and returned low grades of mineralisation.
- 1980 to 1981: BRGM mapped and sampled the Adumbi and Bagbaie deposits on surface and in the historical underground openings. BRGM also drilled three holes at Adumbi and confirmed that (i) mineralisation extended at depth below the water table, (ii) other mineralised zones, parallel to the main one, also existed, and (iii) gold at depth was associated with sulphides.
- 1984: BRGM completed an assessment of the mineral potential at Adumbi.
- 1988: Bugeco International (Bugeco) produced a report on the property entitled "Gold Potential in the Ngayu Mining District Haut Zaire: the Adumbi and Yindi Old Mines".
- 1989: BHP Utah Minerals International carried out a property review of Kitenge and Adumbi.
- 1990: Genmin of South Africa carried out a property review of Kitenge and Adumbi.
- 2009: Kilo acquired the property and carried out extensive exploration activities including major drilling campaigns.
- By November 2013, Kilo had completed 167 diamond drill holes totalling 35,400 m on the Imbo Project.
- 2014: RPA completed technical studies and made various technical recommendations to be executed by Kilo.
- 2014 to 2017: Kilo completed 63 drillholes totalling approximately 8,900 m. A drilling programme was planned to test gold-in-soil and magnetic anomalies at the Adumbi South, Adumbi West and Kitenge Extension targets. This drilling programme was carried out by Orezone Drilling SARL based in Watsa in the DRC.
- 2017: RPA recommended additional drilling at Adumbi to test the down dip/plunge extent of the mineralisation. In 2017, four deeper core holes were drilled below the previously outlined RPA Inferred Resource over a strike length of 400 m and to a maximum depth of 450 m below surface. All four holes intersected significant gold mineralisation in terms of widths and grades.
- 2018 to 2019: Negligible exploration groundwork was undertaken by Kilo due to financial constraints.
- In September 2019, Loncor initially acquired a 71.25 % interest in the Imbo Project, which was subsequently increased to 84.68 % in 2020.
- In Q1 2020, Loncor commissioned independent consultants Minecon Resources and Services Limited (Minecon) to review, assess and quantify the 2017 exploration results.

- Q3 2020: Loncor engaged Minecon to manage its 12 deep hole 7,000 m drilling programme on the Adumbi deposit.

6.3 DEVELOPMENT AND PRODUCTION HISTORY

The first gold discoveries by Belgian prospectors on the Imbo Project occurred in the early 1900s, and early gold production was focused on alluvial deposits until the late 1930s. Gold was discovered in the bedrock of the alluvial zones, and these eluvial deposits were exploited in shallow pits and trenches. Primary gold deposits were later mined in deep trenches and underground galleries.

Kilo, via its agreement with Somituri SPRL, was granted the exploration licences for the project area in February 2009, and in September 2019, Loncor acquired Kilo.

Commercial alluvial gold production on the Imbo Project was undertaken from 1927 to 1951 on the Amuango River. The Amuango River covers the drainage basin from the west side of Adumbi to the area of Bagbaie, located north of Adumbi. Eluvial gold was also exploited over Adumbi Hill, and Kilo believes that this was also considered part of Amuango. The alluvial M'Boro-Adumbi and Amuango exploitations were made in the hydrographical system on the slopes of a ridge of which Adumbi Hill is the summit. A total of 83,000 oz (2.581 t) of gold were exploited during the period (see Table 6.1).

Table 6.1: Summary of Imbo Project Historical Alluvial Gold Production (1927 to 1951)

Deposit	Contained Gold (t)	Contained Gold (oz)
M'Boro-Adumbi	1.334	42,800
Amuango	0.846	27,200
Amuango	0.059	2,000
Maiepunji	0.342	11,000
Total	**2.581**	**83,000**

NOTES:

1. Sourced from the Royal Museum for Central Africa (RMCA, 2007).
2. This estimate is considered to be historical in nature and should not be relied upon; however, it does give an indication of the mineralisation on the property.
3. Numbers might not add up due to rounding.

From 1938 to 1955, surface and underground mining was also carried out on the Kitenge-Maiepunji and Adumbi mines. When underground mining began in 1943, a processing facility was built, "Usine de Kitenge", and commissioned in 1944. By the early 1950s, production had declined rapidly at Kitenge-Maiepunji due to the lack of defined mineral reserves. By 1955, production had declined at the Adumbi mine due to metallurgical challenges, the depth of the mine coupled with lack of energy for milling operations, and poor recovery in the amalgamation mills resulting in exorbitant processing costs. It is reported that a total of 86,400 oz (2.688 t) of gold was exploited at the Kitenge-Maiepunji mines between 1938 and 1955 (see Table 6.2). In addition, 177,500 oz (5.520 t) of gold was exploited from the surface and underground workings of the Adumbi mine between 1952 and 1959 (see Table 6.3). It is reported that Adumbi-Bagbaie closed in 1959, prior to the political independence. Recent exploitation has been carried out by artisanal mining operations, which have mined and recovered gold from most of the easily accessible processable gold.

Table 6.2: Summary of Kitenge-Maiepunji Mines Historical Gold Production (1938 to 1955)

Type	Mined (t)	Gold Grade (g/t)	Contained Gold (t)	Contained Gold (oz)
Surface and Underground	297	9.05	2.688	86,400
Total	**297**	**9.05**	**2.688**	**86,400**

NOTES:

1. Sourced from the Royal Museum for Central Africa (RMCA, 2007).
2. This estimate is considered to be historical in nature and should not be relied upon; however, it does give an indication of the mineralisation on the property.
3. Numbers might not add up due to rounding.

Table 6.3: Summary of Adumbi Mine Historical Gold Production (1952 to 1959)

Ore Type	Mined (t)	Gold Grade (g/t)	Contained Gold (t)	Contained Gold (oz)
Underground Quartz Veins	445	11.37	5.058	162,600
Surface Eluvial and Quarry	161	2.87	0.462	14,900
Total	**606**	**9.11**	**5.520**	**177,500**

NOTES:

1. Sourced from the Royal Museum for Central Africa (RMCA, 2007).
2. This estimate is considered to be historical in nature and should not be relied upon; however, it does give an indication of the mineralisation on the property.
3. Numbers might not add up due to rounding.

It is noted in historical documentation that there was a significant drop in production from 1955 as a result of processing only veins coupled with metallurgical challenges (non-amalgamable gold in less altered rocks). BRGM also reported that the refractory gold content in tailings increased with the mining depth, which corresponds with the reported increasing tailings grade (from 2.3 g/t Au in 1954 to 5.7 g/t Au in 1957). BRGM reported that Adumbi-Bagbaie closed in 1959, just prior to political independence, due to lack of energy for milling operations, exorbitant processing costs, and poor recovery in the amalgamation mills.

The old Belgian workings at Manzako were extended to 2.2 km following field activities. Thus, the northern continuation of the workings was extended by 600 m to the northwest of Drillhole SMDD0002. The old workings indicate the presence of multiple parallel mineralised zones, which were exploited by the Belgians and more recently by artisanal miners. In the southeast of the deposit, the mineralised zones are between 80 m and 150 m apart; however, in the northwest (based on the evidence of the old workings), they appear to be only 20 m apart.

The Kitenge old workings focused on shear zone hosted auriferous quartz vein(s) approximately 1 m to 2 m wide.

6.4 HISTORICAL RESOURCE ESTIMATES

In a 1984 study, BRGM estimated the Adumbi deposit potential to be 1.9 Mt at 19 g/t Au, equivalent to approximately 20 t or 643,000 oz of gold. This estimate was based on an extension of the main 5 m wide vein in a strike length of 900 m (700 m exploited on Adumbi Hill and 200 m to the north towards Bagbaie), in addition to a vertical extension of approximately 200 m below the water table. Minecon notes that this estimate pre-dates the 2014 NI 43-101, cannot be relied upon, and is quoted for historical purposes only.

In 1988, Bugeco concluded that the remaining mineral resources in the Adumbi “main zone”, after mine closure in 1959, were approximately 929,880 oz of gold. Bugeco further concluded that an additional 5 t of gold (approximately 160,750 oz) could be hosted outside the main zone within the remaining alluvium and other adjacent mineralised horizons at Adumbi. The total Bugeco mineral resource was estimated at 1,090,630 oz of gold as presented in Table 6.4. Minecon notes that this estimate pre-dates the 2014 NI 43-101, cannot be relied upon, and is quoted for historical purposes only.

Table 6.4: Adumbi Historical Mineral Resources (1988)

Zone	Type	Tonnage (t)	Grade (Au g/t)	Contained Gold (oz)
Main	Oxide	1,000,000	9.8	315,050
	Sulphide	2,225,000	8.5	614,830
Main Subtotal				**929,880**
Outside				160,750
Total				**1,090,630**

NOTES:

1. Sourced from the Royal Museum for Central Africa (RMCA, 2007) and the Bugeco Report 1988 Mission (Bugeco, 1988).
2. Minecon notes that this estimate pre-dates the 2014 NI 43-101, cannot be relied upon, and is quoted for historical purposes only.
3. A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves.
4. Minecon is not treating the historical estimate as current mineral resources or mineral reserves.
5. Numbers might not add up due to rounding.

It is assumed that recent artisanal mining operations have recovered most of the easily processable gold.

In April 2012, The Mineral Corporation (TMC) (which had been engaged by Kilo to carry out geological modelling and updated resource estimates of the Adumbi deposit) completed an independent NI 43-101 technical report on the Adumbi deposit. At a cut-off grade of 0.5 g/t Au, TMC outlined an Inferred Resource of 1.87 Moz (35.66 Mt grading 1.63 g/t Au) (see Table 6.5).

Table 6.5: Adumbi Historical Mineral Resources (April 2012)

Material Type	Tonnage (t)	Grade (g/t Au)	Contained Au (Moz)
Oxide	12,310,549	1.61	0.64
Transition	4,763,163	1.66	0.25
Sulphide	18,581,569	1.63	0.98
Total	**35,655,280**	**1.63**	**1.87**

In February 2014, independent consultants RPA completed for Kilo an independent NI 43-101 technical report on the Imbo Project and estimated 1.675 Moz (20.78 Mt grading 2.5 g/t Au) of Inferred Mineral Resources on the three separate deposits of Adumbi, Kitenge and Manzako (see Table 6.6).

Table 6.6: Mineral Resource Estimate of Adumbi, Kitenge and Manzako Deposits (Effective Date: December 31, 2013)

Deposit	Tonnage (Mt)	Gold Grade (g/t Au)	Contained Gold (Moz)
Adumbi	19.11	2.20	1.362
Kitenge	0.91	6.60	0.191
Manzako	0.77	5.00	0.122
Total	**20.78**	**2.50**	**1.675**

An assessment of the 2017 drilling and the results of various technical reviews by Minecon (which had been engaged by Loncor) resulted in Minecon outlining 2.19 Moz (28.97 Mt at 2.35 g/t Au) of Inferred Mineral Resources constrained within a US$1,500/oz pit shell at Adumbi (see Table 6.7). To allow Minecon to compare its estimates with those of the RPA 2014 model, a block cut-off of 0.9 g/t Au was applied to the model.

Table 6.7: Inferred Mineral Resource of the Adumbi Deposit (Effective Date: April 17, 2020)

Material Type	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Oxide	3,820,000	2.44	300,000
Transitional	3,320,000	2.69	290,000
Sulphide	21,820,000	2.28	1,600,000
TOTAL	**28,970,000**	**2.35**	**2,190,000**
NOTE: Numbers might not add up due to rounding.			

The Inferred Mineral Resources for the Adumbi, Manzako and Kitenge deposits (Imbo Project) as at April 17, 2020, totalled 2,503,000 oz of gold (30,650,000 t grading 2.54 g/t Au) and are summarised in Table 6.8. For the purposes of this report, no modelling work was carried out on the Kitenge and Manzako deposits by Minecon. Reference was therefore made to the RPA 2014 technical report on the estimates reported for Kitenge and Manzako.

Table 6.8: Inferred Mineral Resources for the Imbo Project (Effective Date: April 17, 2020)

Deposit	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Adumbi	28,970,000	2.35	2,190,000
Kitenge	910,000	6.60	191,000
Manzako	770,000	5.00	122,000
TOTAL	**30,650,000**	**2.54**	**2,503,000**
NOTE: Numbers might not add up due to rounding.			

By April 2021, six additional core holes totalling 2,557.25 m had been drilled, with the initial focus in areas within the pit shell where insufficient drilling had been undertaken. The significant intersections obtained from this drilling programme on the Adumbi deposit resulted in the open-pit Inferred Mineral Resources increasing by 44 % to 3.15 Moz of gold as of April 27, 2021.

Table 6.9 summarises this Adumbi Inferred Mineral Resource based on an in-situ block cut-off at 0.68 g/t Au for oxide and transition materials and 0.72 g/t Au for fresh material, and constrained within a US$1,500/oz optimised pit shell.

Table 6.9: Adumbi Deposit Inferred Mineral Resource by Material Type (Effective Date: April 27, 2021)

Material Type	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Oxide	4,623,000	2.24	333,000
Transition	3,674,000	2.53	299,000
Fresh	33,019,000	2.38	2,521,000
TOTAL	**41,316,000**	**2.37**	**3,153,000**
NOTE: Numbers might not add up due to rounding.			

This mineral resource assessment was undertaken by Loncor's independent geological consultants Minecon. The updated estimate for Adumbi was based on the additional drilling and a review of the Adumbi deposit including remodelling, grade estimation, and considering the CIM requirement for mineral resources to have "reasonable prospects for economic extraction".

As a result of the increased mineral resource at Adumbi, the total Inferred Mineral Resource of the Imbo Project as of April 27, 2021, stood at 3.466 Moz of gold (42.996 Mt grading 2.51 g/t Au) and is summarised in Table 6.10.

Table 6.10: Inferred Mineral Resource for the Imbo Project
(Effective Date: April 27, 2021)

Deposit	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Adumbi	41,316,000	2.37	3,153,000
Kitenge	910,000	6.60	191,000
Manzako	770,000	5.00	122,000
TOTAL	**42,996,000**	**2.51**	**3,466,000**
NOTE: Numbers might not add up due to rounding.			

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 REGIONAL GEOLOGY

Most of the northeastern corner of the DRC is underlain by an Archean Basement, called the Upper-Congo Granitoid Complex or Bomu Craton, formerly known as the Upper-Zaïre Granitoid Massif. This basement is covered by Lower and Upper Kibalian rocks, Neo-Archean in age, that consist of volcano-sedimentary formations with intercalations of quartzites and itabirites (banded iron formation (BIF)). The Kibalian rocks have been metamorphosed to greenschist facies and, in the project area, constitute the greenstone belt. The Neoproterozoic Lindian Supergroup occurs to the south of the area and consists of a sedimentary sequence with a thickness of more than 2,500 m. The rock types in the sequence are mainly arkoses, sandstones, quartzites, shales and conglomerates (see Figure 7.1).

The Upper Congo Granitoid Complex constitutes, together with associated metasediments and volcanics, the western part of the Nyanza-Kibali granite-greenstone terrain, which extends from northern Tanzania into the Central African Republic. The greenstone terrain is hosted within the Kibalian series, which outcrops in numerous zones surrounded by granitoids, the most important (i.e. Moto, Kilo, Mambasa, Ngayu and Isiro) are more than 100 km in strike length. They can be distinguished both by their shape and their lithological composition. Some of these zones constitute narrow belts (less than 10 km wide, 30 km to 60 km in length) made up of units which are isoclinally folded along subvertical axial planes and sub-horizontal fold axes. Others are more or less isometric and show a synclinorial tectonic style. The isoclinally folded unit possesses a metavolcanic to metasediment volumetric ratio (v:s) of approximately 1, that of the isometric exceeds three (up to 10).

An Upper Kibalian (v:s of approximately 1) overlies a Lower Kibalian (high v:s) in the belts of Moto and Ngayu. By extrapolating this relationship to other zones, it can be concluded that two generations of greenstones exist; the one forming narrow bands, rich in sedimentary rocks, belongs to the younger of the two generations. This distinction is also supported by geochronology. The Lower Kibalian of Ngayu and Moto is intruded by 2.8 Ga old tonalities and the Upper Kibalian by 2.45 Ga old granites. Most volcanics of the Lower Kibalian are akin to oceanic tholeiites while those from the upper division contain distinct andesitic members together with less typical tholeiites. Nowhere has the Lower Kibalian series been observed to be associated with high-grade gneissic rocks likely to represent their basement. The Upper Kibalian series, on the other hand, is typically associated both with the tonalite-Lower Kibalian association and with the gneissic series (i.e. the West-Nile Gneissic Complex), suggesting a different geodynamic setting for the two series.

The Ruwenzori tectonic episode (ca 2 Ga) strongly affected the southern flank of the Upper Congo Granitoid Complex, which resulted in the formation of shear belts cutting through the Kibalian zones, and in the cataclasis of the associated granitoids.

In the region bordering the Western Rift, NNE-SSW trending shear belts, ca 950 Ma, strongly reactivated parts of the West-Nile gneissic Complex. Parallel trending belts cutting through the Kibalian zone of the Kilo belt are probably linked to the same event. The tectonic episodes of ca 790 Ma and 700 Ma affected the northern flank of the Upper Congo Granitoid Complex and consequently the Kibalian zone of Moto. By reactivating the late-Archean suture between the West-Nile Complex and the Congo Granitoid Complex, these episodes contributed to the present shape of the Moto zone.

Gold is the only commodity to have been extracted commercially in the Ngayu belt. Several years ago, Rio Tinto assessed the BIF as a potential source of iron ore, but although haematite-rich zones of good grade were reportedly drilled, tonnage was below the economic requirement. Diamonds are recovered by artisanal miners from the Ngayu River; the source of the stones is unknown but is probably outside the area under discussion. No other mineral occurrences of potential significance are known.

The majority of the gold occurrences within the Ngayu belt are located close to the contact of the BIF. Historically, only two deposits were exploited on a large scale by previous owners, namely Yindi and Adumbi.

Several styles of gold mineralisation have been identified in the Ngayu belt and are summarised below:

- **Shear-zone hosted gold**
  - Mineralisation of shears within the BIF, or on the BIF contacts, leading to quartz veining and sulphidation of the BIF and immediate wall rock, e.g. Adumbi, and Makapela Reef 2.
  - Mineralisation of shears within basalts and schists (and to a much lesser extent intermediate intrusives), resulting in discrete auriferous quartz veins with limited wall rock mineralisation, e.g. Makapela Reef 1, and the Yindi vein field.
- **Disseminated mineralisation in the BIF**
  Sulphidation of the BIF by fluids utilising nearby cross-cutting and parallel structures, such as thrusts and shears e.g. Yindi BIF-hosted mineralisation and Nagasa Anomaly 1. This style of mineralisation has the potential to form deposits of very large size, e.g. Geita in Tanzania.
- **Sheeted veins**
  Shear zones resulting in auriferous sheeted quartz veins and veinlets developing mainly parallel to the foliation and forming packages over widths of up to 40 m, often with disseminated mineralisation between the veins, e.g. Itali, Mondarabe.
- **Elluvial/Colluvial deposits**
  Artisanal mining of weathered gold mineralisation preserved as elluvial or colluvial material is widespread throughout the belt, particularly in the Imva Fold area and Anguluku.
- **Alluvial deposits**
  - Palaeoalluvial deposits are locally exploited by artisanal miners by digging pits to the basal gravel layer of old river channels, e.g. Nagasa, Mondarabe, Matete.
  - Exploitation of modern alluvium is widespread throughout the Ngayu belt and is particularly common in the Imva Fold area.

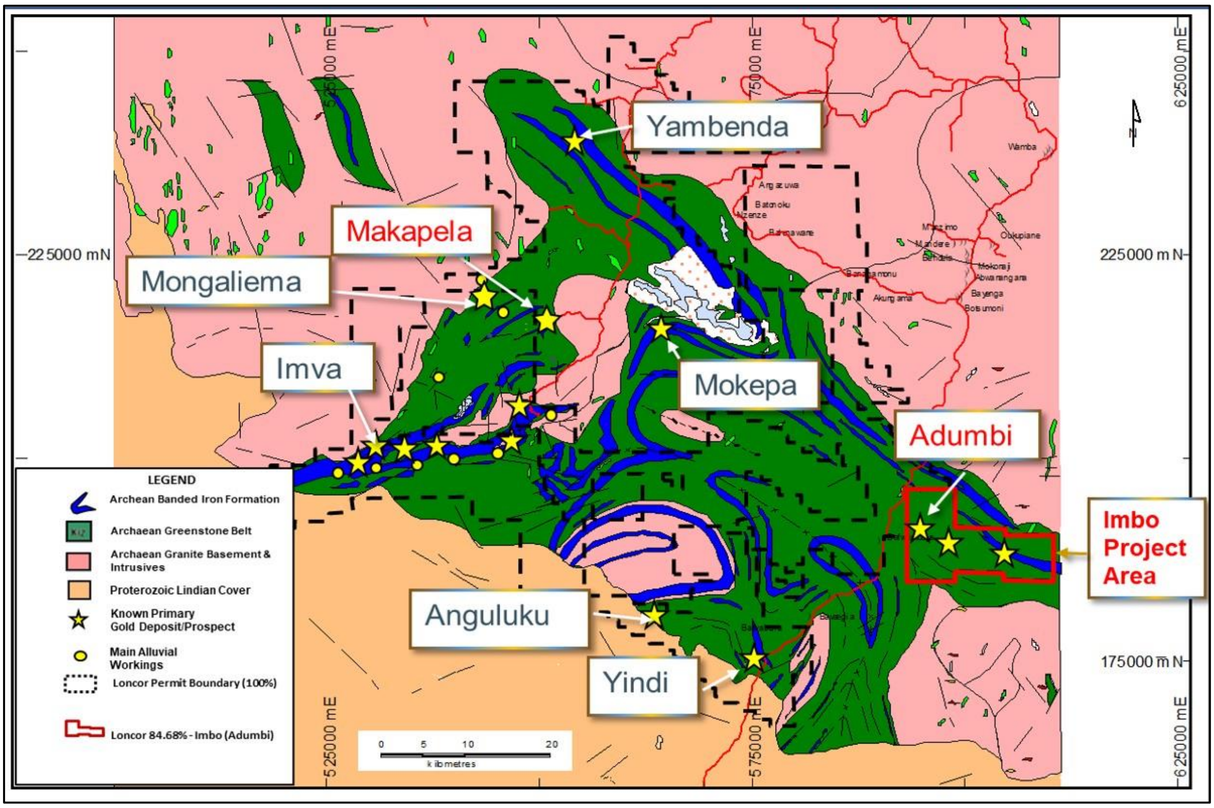


Figure 7.1: Main Gold Projects and Prospects within the Ngayu Greenstone Belt

7.2 LOCAL GEOLOGY

The Imbo Project is located within the Upper Kibalian represented by the greenstone belt made up of metasediments and metavolcanics of greenschist facies, including the prominent BIF, which forms prominent ridges throughout the Ngayu Greenstone Belt.

Intruding all the basement formations are intrusive rocks consisting of possibly Late Proterozoic dolerite/diabase and doleritic gabbro and diorite. Quartz veins are predominantly associated with the Upper Kibalian. The Proterozoic Lindian metasedimentary rocks unconformably overlie the Kibalian rocks. Palaeozoic, Cenozoic, and Quaternary metasediments and alluvial sediments are locally present within the project area. The Karoo Formation comprises black shales, eluvial and alluvial deposits. Post-Karoo rocks are essentially represented by lateritic cuirasse. The simplified geology of the Imbo area is illustrated in Figure 7.2.

Gold is associated with sulphide mineralisation within the Archean Kibalian Formation of the Ngayu Greenstone Belt. Gold generally occurs with quartz veins; host rocks to the quartz veins include BIF, metasedimentary, and tuffaceous rocks.

Within the Imbo Project area, there is a strong association between gold mineralisation and the presence of the BIF, the BIF either constituting the host rock (e.g. Adumbi) or forming a significant part of the local stratigraphy in the Imbo Project area.

The BIF forms both physical and chemical traps for mineralising hydrothermal fluids as follows:

- **Competency contrasts between the BIF and the interlayered rocks**
  When interlayered with incompetent lithologies such as the metasedimentary schists and volcaniclastics, the BIF constitutes relatively hard rock, more likely to develop brittle fracturing than the more ductile surrounding rocks. Also, shearing may preferentially take place in the schists, on the contact with the BIF. These fractures and shears can act as channel ways, focusing hydrothermal fluids into the chemically reactive BIF.
  When interlayered with competent rocks such as massive basalts, the BIF units (especially if relatively thin like those at Makapela) may act as zones of weakness, along which shear and faults may propagate. Again, the tectonic fabric within the BIF can facilitate the flow of hydrothermal fluids.
- **Sulphidation of magnetite**
  The iron-rich BIF is a chemically reactive rock, the main interaction with hydrothermal fluids involving the reduction of magnetite to pyrite, resulting in the precipitation of gold.

Mineralisation on the Imbo Project (PE9691) is known to occur at Bagbaie (referred to as Adumbi North), Adumbi, Kitenge, Manzako, Monde Arabe, Maipinji and Vatican (see Figure 7.2)

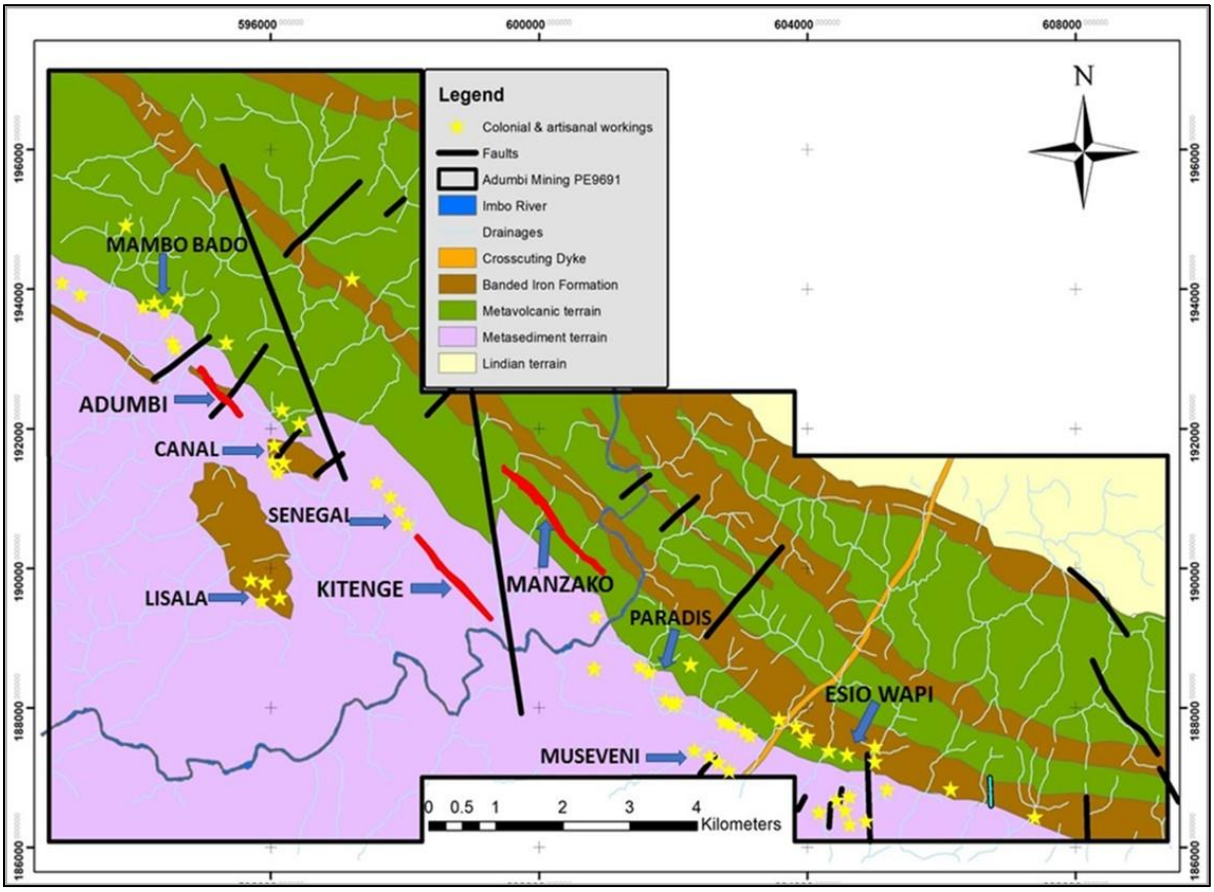


Figure 7.2: Imbo Project – Simplified Geology

7.3 PROPERTY GEOLOGY

Gold occurrences on the Imbo Project are hosted within quartz veins in the sheared Upper Kibalian Formation, which consist of chemical metasedimentary units including the BIF, clastic metasedimentary rocks assigned the field name "greywacke", and mafic volcanic flows. Adumbi, Kitenge and Manzako are the three main deposits within the Imbo Project with mineral resources and are separately discussed below.

7.3.1 Adumbi

Adumbi is currently the most explored deposit within the Imbo Project. Adumbi forms a topographic high (Adumbi Hill) and incorporates the Canal prospect, which is the southeastern continuation of Adumbi.

The published geological map and historical reports indicate that the Adumbi deposit is underlain by Upper Kibalian rocks with the dominant lithologies including a well bedded BIF unit, tuffaceous metasedimentary rocks (referred to as greywacke), black shale, and a mafic intrusion.

Based on examined drillholes, the rocks at Adumbi mainly comprise a subvertical sequence of metamorphosed clastic sediments (pelites, siltstones and greywacke) interbedded with units of BIF of varying width. The grade of metamorphism is probably lower greenschist facies, and the clastic units are petrographically classified as schists. Foliation is usually clearly defined in hand specimens although sedimentary features such as bedding are frequently preserved.

7.3.1.1 Lithological Units

Recent drilling and re-logging of the core at the Adumbi deposit display five distinct geological domains with the BIF unit attaining a thickness of up to 130 m in the central part (see Figure 7.3 and Figure 7.4). From northeast to southwest, these are as follows:

1. Hanging Wall Schists: dominantly quartz carbonate schist, with interbedded carbonaceous schist.
2. Upper BIF Sequence: an interbedded sequence of BIF and chlorite schist, 45 m to 130 m in thickness.
3. Carbonaceous Marker: a distinctive 3 m to 17 m thick unit of black carbonaceous schist with pale argillaceous bands.
4. Lower BIF Sequence: BIF interbedded with quartz carbonate, carbonaceous and/or chlorite schist in a zone 4 m to 30 m in thickness.
5. Footwall Schists: similar to the hanging wall schist sequence.

There is a higher-grade zone of gold mineralisation termed the "replaced rock zone" (RP zone) associated with alteration and structural deformation that has completely destroyed the primary host lithological fabric. The RP zone occurs in the lower part of the Upper BIF package and in the Lower BIF package, and transgresses the Carbonaceous Marker, located between the Upper and Lower BIF packages, both along strike and down dip (see Figure 7.4).

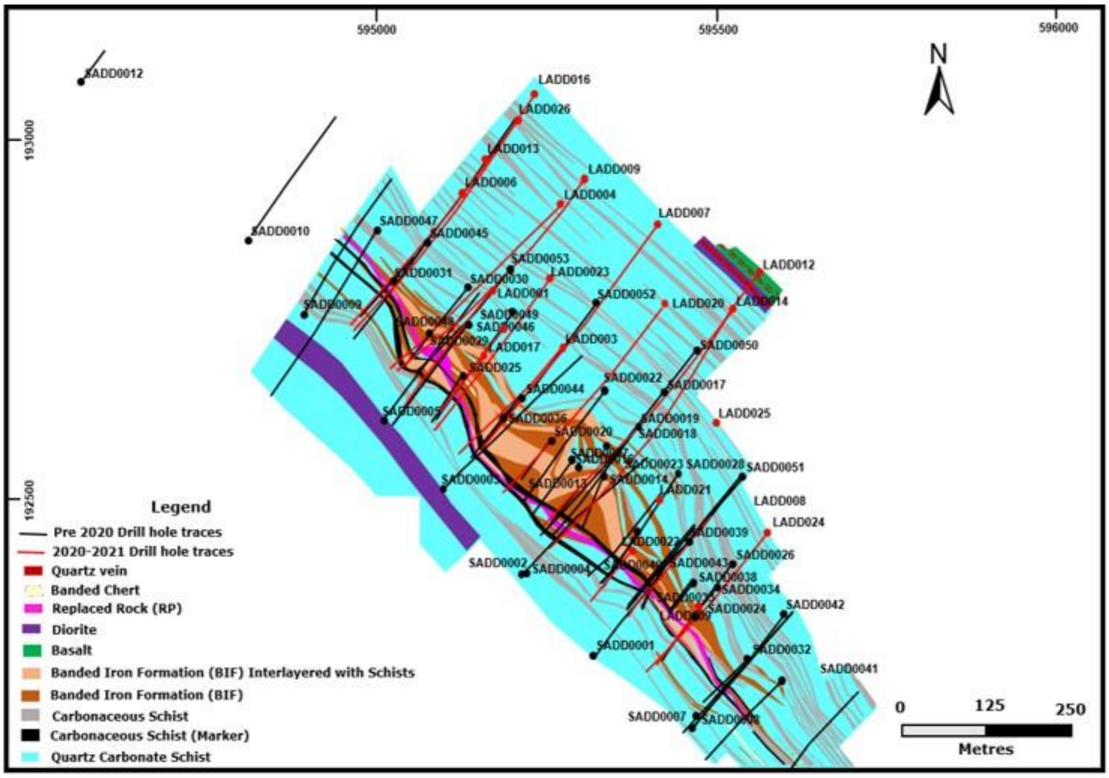


Figure 7.3: Adumbi Deposit – Geological Map

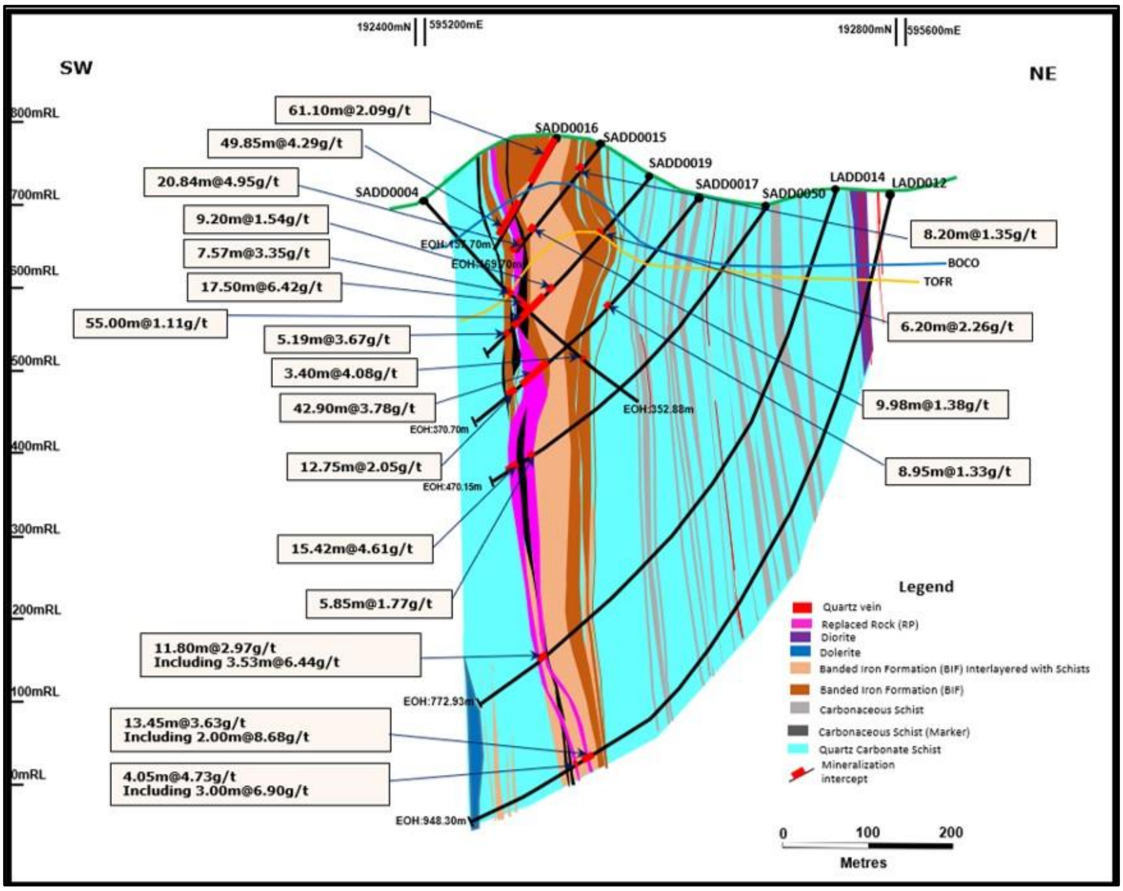


Figure 7.4: Adumbi Deposit – Geological Cross Section

Further details on the individual rock types are as follows:

7.3.1.1.1 Quartz Carbonate Schist

Fine- to medium-grained, pale grey to pale greenish grey schist, comprising subrounded, dark grey quartz grains up to 1.5 mm (probably remnant clastic grains) in a finer-grained matrix of quartz, white mica and carbonate (possibly ankerite). The carbonate forms irregular, elongated grains orientated parallel to the foliation. It is the most abundant rock in the Adumbi sequence.

Pyrite often occurs as irregularly distributed subhedral to anhedral crystals up to 10 mm across. In the core observed to date, the lack of associated hydrothermal alteration and the absence of pressure shadows and evidence of rotation indicate that the pyrite formed as porphyroblasts after the main deformation event. However, the technical report prepared by RPA refers to pressure shadows and rotated grains, so the possibility of earlier (possibly diagenetic) pyrite formation cannot be ruled out.

It is interpreted that the rock was probably originally a poorly sorted, calcareous, muddy, fine-grained arenite, possibly a greywacke.

7.3.1.1.2 Carbonaceous Schist

Very fine-grained, dark grey to black schist, consisting of carbonaceous material and (according to petrographic data) varying amounts of white mica. Quartz is rare. Banding due to variations in the proportion of white mica reflects the bedding in the original sediment. The nature of the carbonaceous material was not determined petrographically but based on samples of similar material from elsewhere in the Ngayu belt, it is probably amorphous carbon rather than graphite. The rock was probably originally a black shale formed in a deep marine environment. Pyrite porphyroblasts similar to those in the quartz carbonate schist, are irregularly distributed. Pyrite also occurs locally as very finely disseminated grains. The carbonaceous schist occurs as robust units up to several metres in width, but more frequently as thinner units interbanded with quartz-sericite schist. The carbonaceous schist however also occurs (a) with white to pale grey siliceous bands, which probably represent recrystallised chert, and (b) interbanded with whitish, soft, very fine-grained argillaceous material, which could possibly represent thin layers of volcanic ash.

7.3.1.1.3 Banded Iron Formation (BIF)

The BIF consists of black, fine-grained magnetite-rich bands alternating with white to pale buff chert. The width of the magnetite bands is variable, ranging from laminae only a few millimetres wide, to bands up to about 10 cm across.

The BIF at Adumbi is distinctly different to that seen elsewhere in the Ngayu belt, which comprises either (a) a thinly bedded rock, with magnetite laminae separated by quartz-rich bands of similar width, or (b) a more massive magnetite-rich rock with poorly defined banding.

7.3.1.1.4 Chlorite Schist

A fine-grained rock, superficially similar to the carbonaceous schist in hand specimens, but with a dark greenish tinge and a lack of bedding, that occurs interbanded with the BIF in the central part of the deposit, rarely forming units greater than 3 m in thickness. It forms more massive units up to 14 m in width, but is locally finely interbedded with quartz carbonate schist, indicating a sedimentary rather than volcanic origin. In places the chlorite schist is distinctly magnetic, probably due to the presence of finely disseminated magnetite.

7.3.1.1.5 Banded Chert

This rock type is not widespread, occurring in the Canal zone in the SE of the prospect, in units up to 4 m in width. It superficially resembles the BIF, but the dark bands comprise fine-grained clastic sedimentary material instead of chemically precipitated magnetite.

7.3.1.1.6 Dolerite

Mafic intrusive rock, massive (not deformed), dark greenish in colour, fine- to medium-grained with localised irregular veins and veinlets of quartz carbonate.

7.3.1.2 Interpretation of the Adumbi BIF Package

The gold mineralisation at Adumbi is directly related to the chemical and physical properties of the BIF package. The geological interpretation from the drill intersections demonstrates that the mineralised BIF increases in thickness with depth (see Figure 7.5). The above thus confirms the existence of significant underground potential at Adumbi. Further drilling is recommended to unearth this potential.

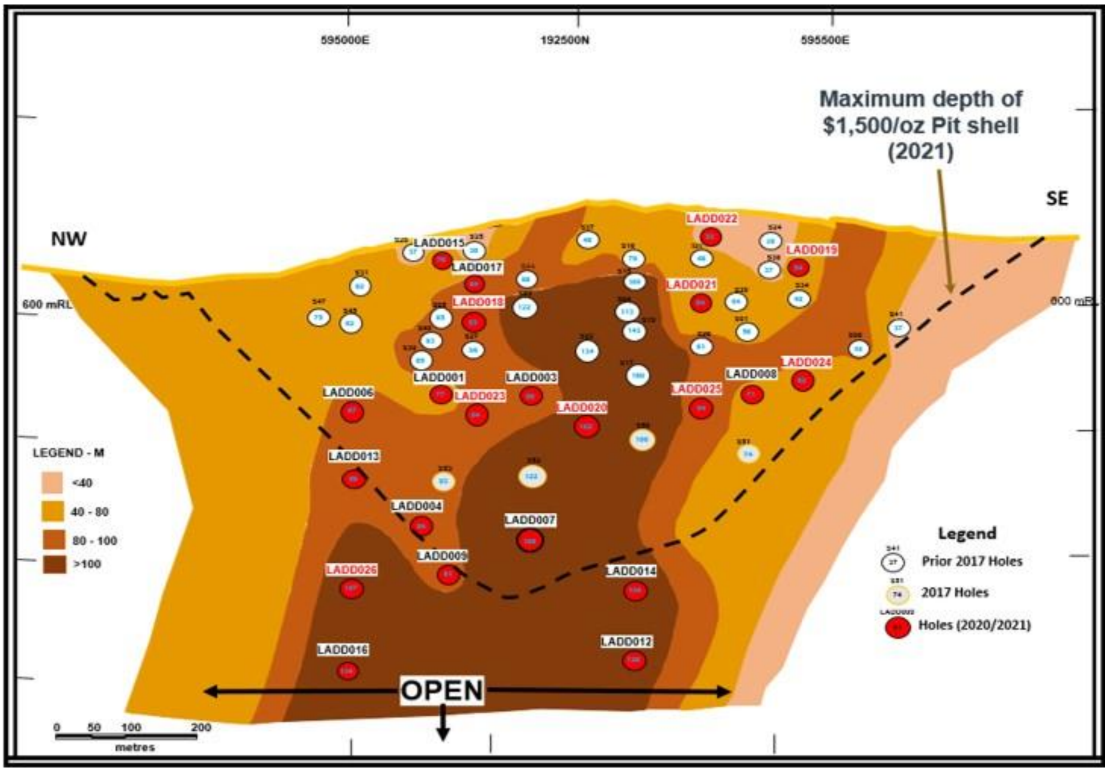


Figure 7.5: Adumbi Deposit – Interpretation of BIF Package

7.3.2 Kitenge

The Kitenge deposit is situated approximately 4 km southeast of the Adumbi deposit, and it may be a strike extension of the shear zone structure that hosts the Adumbi deposit but left laterally fault offset approximately 500 m to the northeast (see Figure 7.2). The Senegal prospect has been incorporated into the Kitenge deposit as it is the probable fault offset northwest continuation along strike of Kitenge.

7.3.2.1 Lithological Units

Lithological units within the Kitenge deposit area have been classified into three principal lithological packages (see Figure 7.6) as follows:

- Upper Schist Sequence: Characterised by quartz carbonate schist interbedded with subordinate carbonaceous schist. In this sequence, beddings are clearly displayed in quartz carbonate schist in places where it is not interbedded with carbonaceous schist. Typical carbonaceous schist also forms part of this sequence.
- Middle Schist Sequence: Dominant quartz carbonate schist, fine- to medium-grained, generally massive and weakly foliated. Most of the gold mineralised zone, characterised by quartz veining, shearing and sulphide mineralisation, occurs in this sequence.
- Lower Schist sequence: Very similar to the Upper Schist sequence with quartz carbonate schist dominating over carbonaceous schist.

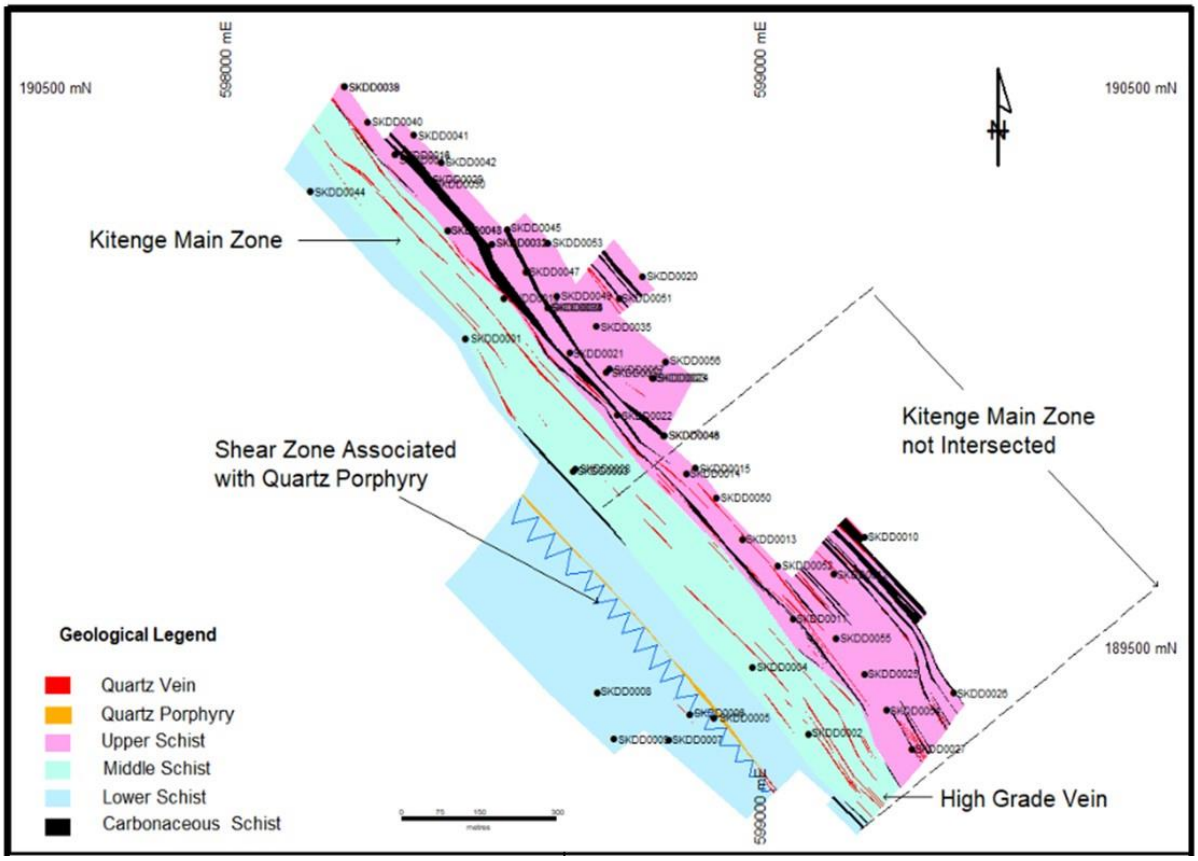


Figure 7.6: Kitenge Deposit – Surface Geological Map

The main rock type at the Kitenge deposit is quartz carbonate schist, identical to that at Adumbi. Bands of carbonaceous schist up to a few metres in width occur in places.

A summary of the rock types occurring in the re-logged Kitenge holes is as follows:

- Quartz carbonate schist
- Carbonaceous schist
- Quartz carbonate schist with interbanded carbonaceous schist
- Carbonaceous schist with interbanded quartz carbonate schist
- Quartz porphyry
- Quartz veins

Except for the quartz porphyry, the rest are as described under the Adumbi lithologies (see 7.3.1.1).

Quartz porphyry is a greenish grey, medium-grained intrusive igneous rock composed mainly of quartz phenocrysts embedded in a fine siliceous matrix. This unit is not widespread and was only intersected in one hole (SKDD0028) located in the SE of the central part of the drilled area in the Kitenge deposit. The quartz porphyry occurs as a narrow unit with an approximate width of 40 cm. A well-defined fine-grained chill margin is developed at the quartz porphyry contacts with the country rock and below it is extensive ankerite alteration, bleaching and quartz veining in association with strong shearing and isolated low-grade mineralisation. Although it has not been established to have associations with gold mineralisation at Kitenge, its presence in association with shearing and the aforementioned alteration might be of geological importance as elsewhere, intrusive rocks have been recorded to be a source of hydrothermal fluids associated with gold mineralisation.

7.3.2.2 Hydrothermal Alteration

Hydrothermal alteration at Kitenge is associated with the shear zones. The alteration comprises pervasive bleaching, with chlorite preferentially developed along the shear planes. Quartz veins are also present and are usually developed parallel to the shear fabric. They are typically white or grey, glassy, and vary from veinlets to robust veins up to 1.90 m in width. Disseminated euhedral crystals of dolomite are also present in the alteration zones, usually associated with quartz veins.

Sulphides are irregularly distributed as stringers and disseminated grains, and consist of pyrite, arsenopyrite and rare pyrrhotite. The sulphides occur in variable proportions and constitute up to 20 % of the rock.

The main styles of hydrothermal alteration at the Kitenge deposit are associated with clearly defined zones of shearing and comprise the following:

- Pervasive and disseminated ankerite
- Dolomite as disseminated crystals and patches associated with quartz veins
- Sulphides comprising pyrite, pyrrhotite, arsenopyrite and rare chalcopyrite
- Bleaching, which is in most cases associated with shearing
- Quartz as irregular and foliation parallel veins, locally with visible gold

7.3.3 Manzako

The Manzako deposit is located approximately 1.5 km northeast of Kitenge (see Figure 7.2). This includes the previously named Lion prospect, which is now considered to be the southeastern portion of Manzako and incorporates a series of subparallel shear structures.

7.3.3.1 Lithological Units

The main lithological unit within the Manzako deposit is basalt, with some dolerite intrusive (see Figure 7.7).

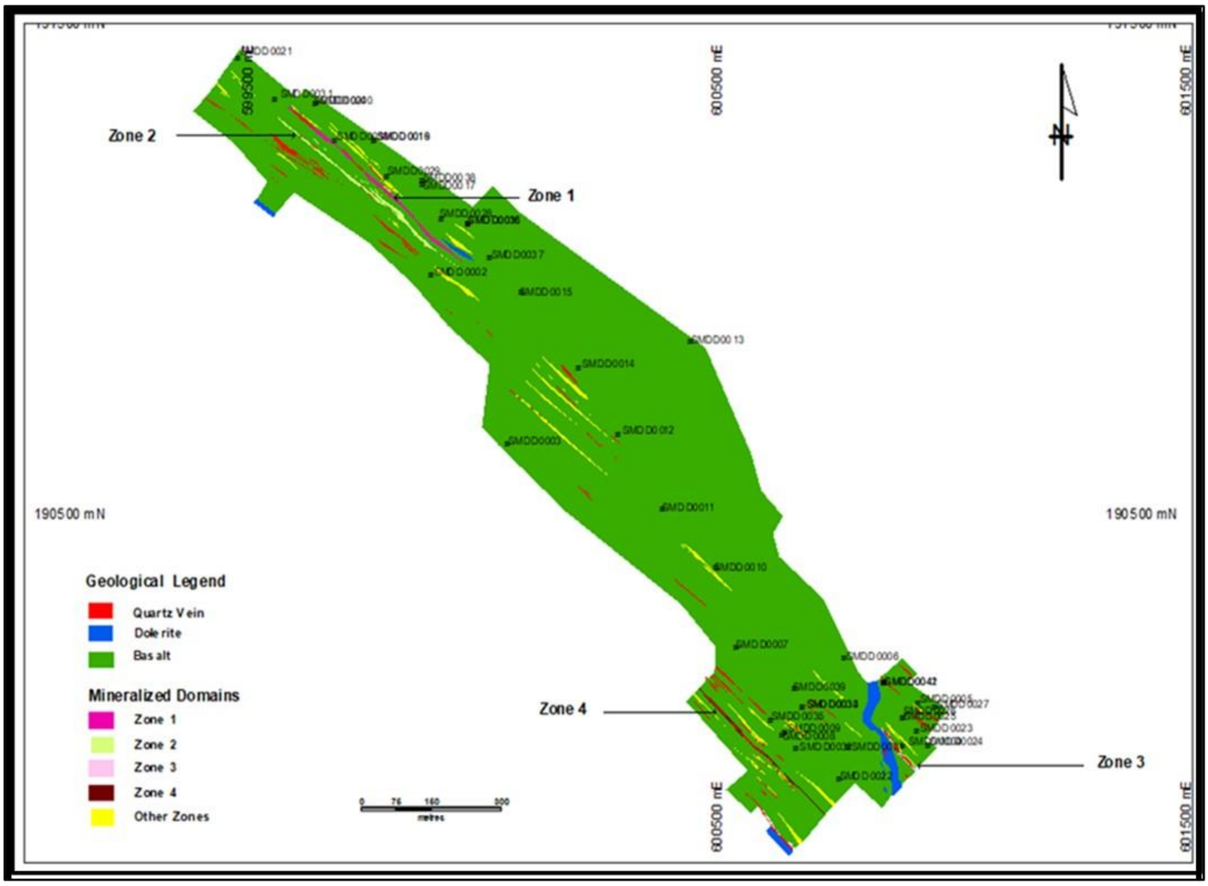


Figure 7.7: Manzako Deposit – Geological Map

7.3.3.1.1 Basalt

Two categories of the basalt unit identified are as follows:

- Unaltered, greenish, fine-grained, amygdaloidal basalt.
- Altered, grey, fine-grained, sheared, bleached and silicified basalt. The altered basalt in places has angular to subrounded secondary quartz crystals, interpreted to represent highly sheared and brecciated vein quartz, which locally may resemble clastic sedimentary rock.

7.3.3.1.2 Dolerite

The dolerite is dark green, fine to medium grained, and is locally weakly magnetic. In places, the dolerite has sharp contacts with the basalt, but elsewhere the contacts are gradational. Where the contacts are gradational, the dolerite probably represents the more slowly cooled, central parts of thicker basalt flows, rather than intrusive bodies. This is a common feature at the basalt-hosted Makapela deposit in the north of the Ngayu belt. The main occurrence of dolerite is in the SE of the deposit where it appears to be intrusive with a general N-S orientation and is traceable for approximately 200 m along strike (see Figure 7.6). The average width of the dolerite is approximately 25 m.

The Manzako mineralised structures appear to be fairly uniform in strike and dip and are subparallel to the controlling structures at Adumbi and Kitenge, i.e. approximately parallel to the lithological strike. The detailed work on the RP zone at Adumbi has, however, shown that the main structure does undulate and cross-cut strike at acute angles.

7.3.3.2 Hydrothermal Alteration

The main styles of hydrothermal alteration noted in the re-logged drillholes at Manzako are associated with clearly defined zones of shearing and comprise the following:

- Pervasive haematite
- Sulphides comprising pyrite, arsenopyrite and rare pyrrhotite
- Bleaching, which is in most cases associated with shearing
- Quartz as irregular and foliation parallel veins
- Tourmaline occurring as patches
- Epidote occurring as patches
- Sphalerite associated with haematite

7.4 MINERALISATION

Gold mineralisation at Adumbi is generally associated with quartz and quartz-carbonate pyrite ± pyrrhotite ± arsenopyrite veins in a BIF horizon.

In the central part of the Adumbi deposit, three main zones of gold mineralisation are present (see Figure 7.3 and Figure 7.4):

1. Within the Lower BIF Sequence
2. In the lower part of the Upper BIF Sequence (Zones 1 and 2 are separated by the Carbonaceous Marker, which is essentially unmineralised)
3. A weaker zone in the upper part of the Upper BIF Sequence

Gold mineralisation at Kitenge is associated with zones of shearing with strong quartz veining, higher grades being associated with relatively abundant sulphides and particularly the presence of arsenopyrite (see Figure 7.8).

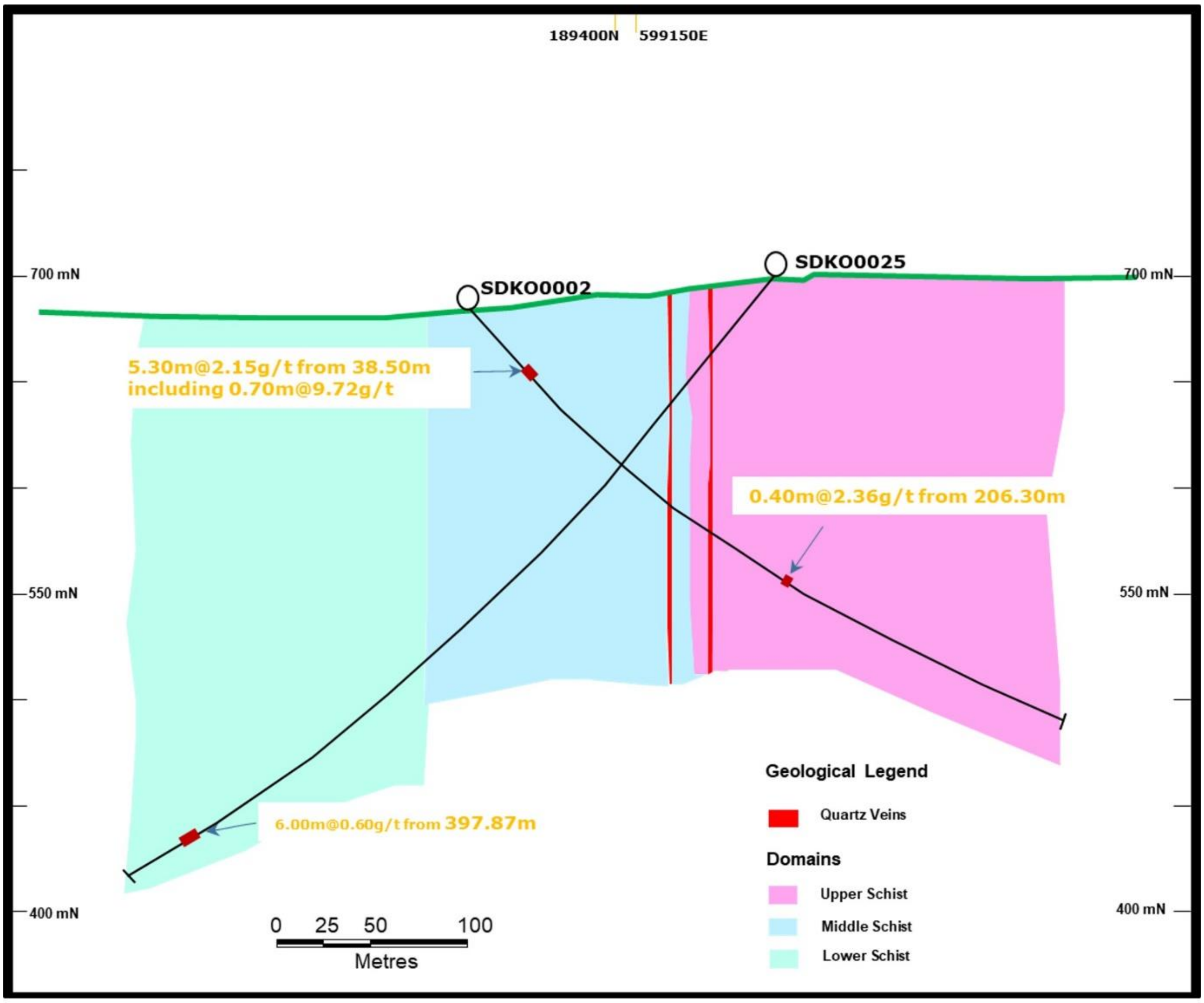


Figure 7.8: Kitenge Deposit – Cross Section through Drillholes SKDD0002 and SKDD0025

Gold mineralisation at Manzako is associated with quartz veining within shear zones, with associated sulphides especially arsenopyrite, and pervasive haematite. The continuity of mineralisation along strike and down dip is erratic; the best developed zones (see Figure 7.9) are the following:

- Zone 1: 450 m strike length, located in the NW of the deposit
- Zone 2: 450 m strike length, parallel to and 25 m south of Zone 1
- Zone 3: 100 m strike length, located in the SE of the deposit and proximal to dolerite intrusions
- Zone 4: 400 m strike length, located in the SE of the deposit

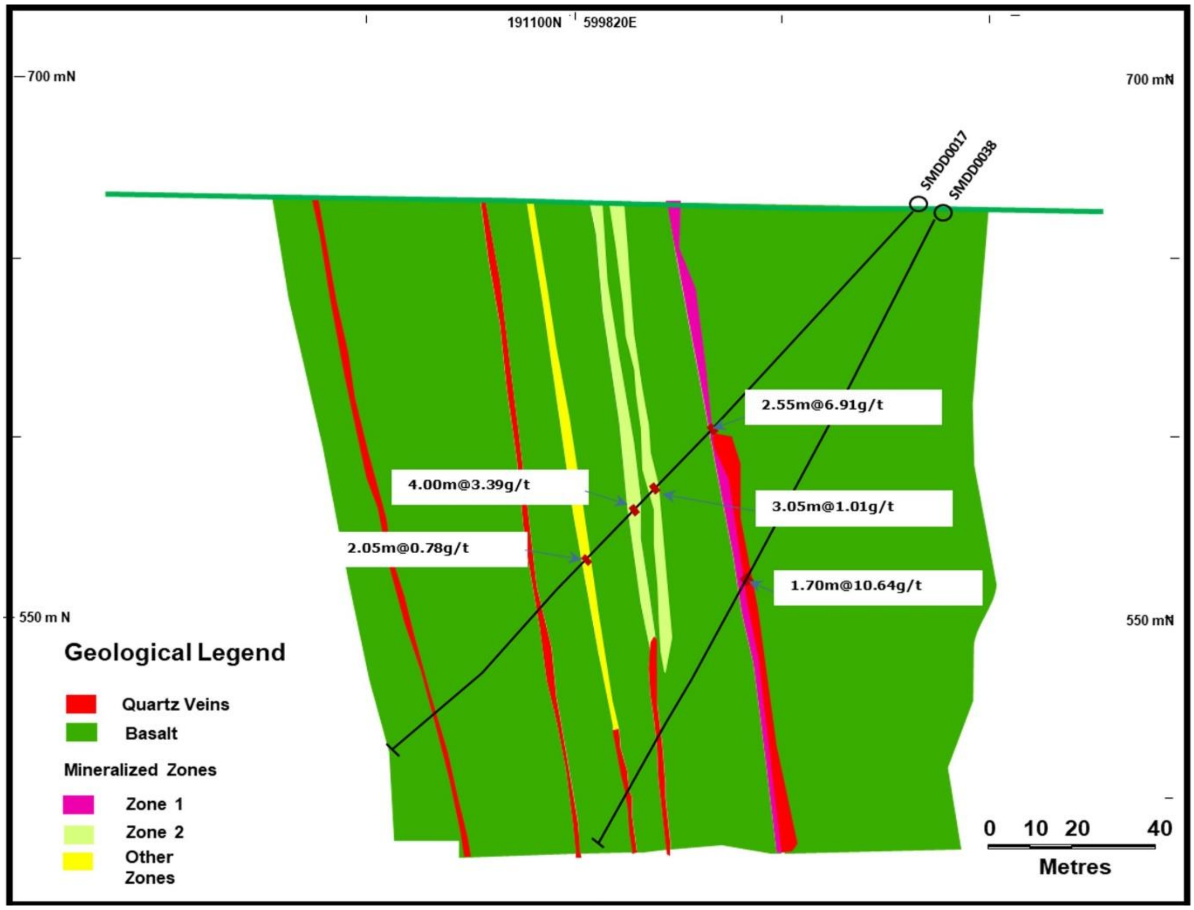


Figure 7.9: Manzako Deposit – Cross Section through Drillholes SMDD0017 and SNDD0038

7.4.1 2020 to 2021 Drill Assay Results

The significant mineralised intercepts from LADD001, LADD003, LADD004, LADD006, LADD007, LADD008, LADD009, LADD012, LADD013, LADD014, LADD015, LADD016, LADD017, LADD018, LADD019, LADD021, LADD022, LADD023, LADD024 and LADD025 are presented in Table 7.1.

Table 7.1: Significant Mineralised Intercepts from Completed Drillholes.

Borehole Identification (BHID)	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
LADD001	202.58	223.35	20.77	1.72
LADD001	231.27	237.17	5.9	1.89
LADD001	251.27	258.6	7.33	5.8
LADD001	295.25	298.7	3.45	2.1
LADD001	301.62	321.95	20.33	2.47
LADD001	Including 317.11	321.95	4.84	5.4
LADD003	224.55	235	10.45	3.88
LADD003	253.5	286.8	33.3	3.25

Borehole Identification (BHID)	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
LADD003	Including 253.50	259.2	5.7	7
LADD003	Including 277.73	286.8	9.07	5.11
LADD004	429	457	28	3.26
LADD004	Including 432.00	436.9	4.90	6.96
LADD004	Including 450.62	454.15	3.53	8.3
LADD004	473.8	478.4	4.60	2.07
LADD004	505.85	526.15	20.3	2.83
LADD004	Including 506.85	513.4	6.55	4.64
LADD004	Including 523.85	526.15	2.30	7.25
LADD006	299.37	302.25	2.88	2.64
LADD006	308	309	1	21.2
LADD006	322.1	337.3	15.2	1.67
LADD006	353.35	357.85	4.5	3.25
LADD007	99.95	107.8	7.85	1.45
LADD007	540.62	596.05	55.43	2.76
LADD007	Including 583.60	596.05	12.45	8.11
LADD007	607.9	611.27	3.37	4.61
LADD008	235.05	278.15	43.1	1.68
LADD008	291.8	298.9	7.1	1.34
LADD008	305.15	305.93	0.78	21.8
LADD008	323.8	338.78	14.98	3.62
LADD008	Including 335.75	338.78	3.09	13.28
LADD009	559.76	564.76	5	3.17
LADD009	581.9	614.05	32.15	6.17
LADD009	Including 599.05	600.51	1.46	94.77
LADD009	629.56	644.92	15.36	3.73
LADD009	Including 632	637.89	5.89	6.56
LADD009	650.5	657.95	7.45	1.48
LADD012	784.35	797.8	13.45	3.63
LADD012	Including 784.35	786.35	2	9.56
LADD012	806.3	810.35	4.05	4.73
LADD013	394.06	401.1	7.04	2.68
LADD013	418.65	438.65	20	4.21
LADD013	Including 419.75	430.75	11	6.91
LADD013	452.3	469.6	17.3	2.48
LADD013	Including 457.35	465.55	8.2	4.71
LADD014	670	681.8	11.8	2.97
LADD014	Including 670	673.53	3.53	6.44
LADD015	24.43	31.5	6.07	1.77
LADD016	672.85	680.94	8.09	1.9

Borehole Identification (BHID)	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
LADD016	731.51	757.1	25.59	2.39
LADD016	Including 737.18	743.27	6.09	4.78
LADD016	Including 749.67	752.56	2.89	4.98
LADD016	672.85	680.94	8.09	1.9
LADD017	45.55	62.7	17.15	1.9
LADD017	92.68	118.45	25.77	6.24
LADD017	Including 100.76	110.05	9.29	9.68
LADD017	Including 112.95	118.45	5.5	9.75
LADD018	93.34	113.7	20.36	0.93
LADD018	152.48	178.2	25.72	2.26
LADD019	4.57	11.6	7.03	2.13
LADD021	75.21	88.17	12.96	2.09
LADD021	99.74	106	6.26	1.09
LADD021	144.78	160.51	15.73	5.28
LADD021	Including 144.78	149.78	5	13.7
LADD022	20.5	42	21.5	2.23
LADD022	Including 25.5	34	8.5	4.23
LADD023	227.1	261.73	34.63	3.12
LADD023	Including 231.65	237.4	5.75	7.23
LADD023	Including 248.1	255.25	7.15	5.55
LADD023	270.43	300.25	29.82	1.77
LADD024	216.15	227.65	11.5	3.47
LADD024	Including 224.1	227.65	3.55	7.79
LADD024	235.97	253.75	17.78	3.2
LADD025	258.38	266	7.62	1.16
LADD025	279.5	286.35	6.85	3.44
LADD025	301.1	311.57	10.47	1.74
LADD025	321.6	336.2	14.6	2.11
LADD025	342.65	361.75	19.1	4.11
LADD025	Including 349	357.75	8.75	5.4

NOTES:

1. It is estimated that the true widths of the mineralised sections for the drillholes are as follows: LADD001 (82 %), LADD003 (80 %), LADD004 (81 %), LADD006 (95 %), LADD007 (89 %), LADD008 (62 %), LADD009 (82 %), LADD012 (86 %), LADD013 (85 %), LADD014 (78 %), LADD015 (65 %), LADD016 (69 %), LADD017 (71 %), LADD018 (75 %), LADD019 (65 %), LADD021 (73 %), LADD022 (58 %), LADD023 (76 %), LADD024 (77 %), and LADD025 (78 %) of the intercepted widths given in this table.
2. Drillholes LADD002, LADD005, LADD008, LADD010, and LADD011 were discontinued before intersecting the mineralised zone.

7.4.2 Relationship Between Sulphides ± Silicification and Gold Grades

An exercise was undertaken to establish the relationship between gold values and sulphides (pyrite, pyrrhotite and arsenopyrite)/silicification. This was done for Drillholes LADD001 (from 130.80 m to 360.30 m), LADD003 (from 107.00 m to 309.20 m), LADD004 (from 418.50 m to 566.30 m), LADD006 (from 252.00 m to 395.35 m), LADD007 (from 490.20 m to 647.75 m), LDD008 (from 224.00 m to 365.35 m), LADD009 (from 552.76 m to 689.30 m), LADD012 (740.00 m to 948.30 m) and LADD013 (360.75 m to 485.80 m) as presented below.

Table 7.2 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD001, and the corresponding gold grades.

Table 7.2: Relationship between Sulphides ± Silicification and Gold Grades in LADD001

Composition	Observation	Typical Gold Grades	Sample No.
Pyrite only:			
High % of pyrite + moderate silicification	Weak gold values	0.63 g/t	62861
Pyrite + Pyrrhotite:			
High % of pyrite + pyrrhotite + strong silicification	High gold values	10.40 g/t	62859
Low % of pyrite + pyrrhotite + strong silicification	Low gold values	0.19 g/t	62841
Pyrite + Arsenopyrite:			
High % of pyrite + arsenopyrite + strong silicification	High gold values	10.50 g/t	62808
Low % of pyrite + arsenopyrite + strong silicification	Low gold values	0.10 g/t	62761
Pyrite + Pyrrhotite + Arsenopyrite:			
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	14.70 g/t	62919
Medium % of pyrite + pyrrhotite + arsenopyrite + strong silicification	Medium gold values	3.52 g/t	62931
Low % of pyrite + pyrrhotite + arsenopyrite + strong silicification	Low gold values	0.43 g/t	62946

Figure 7.10 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD001.

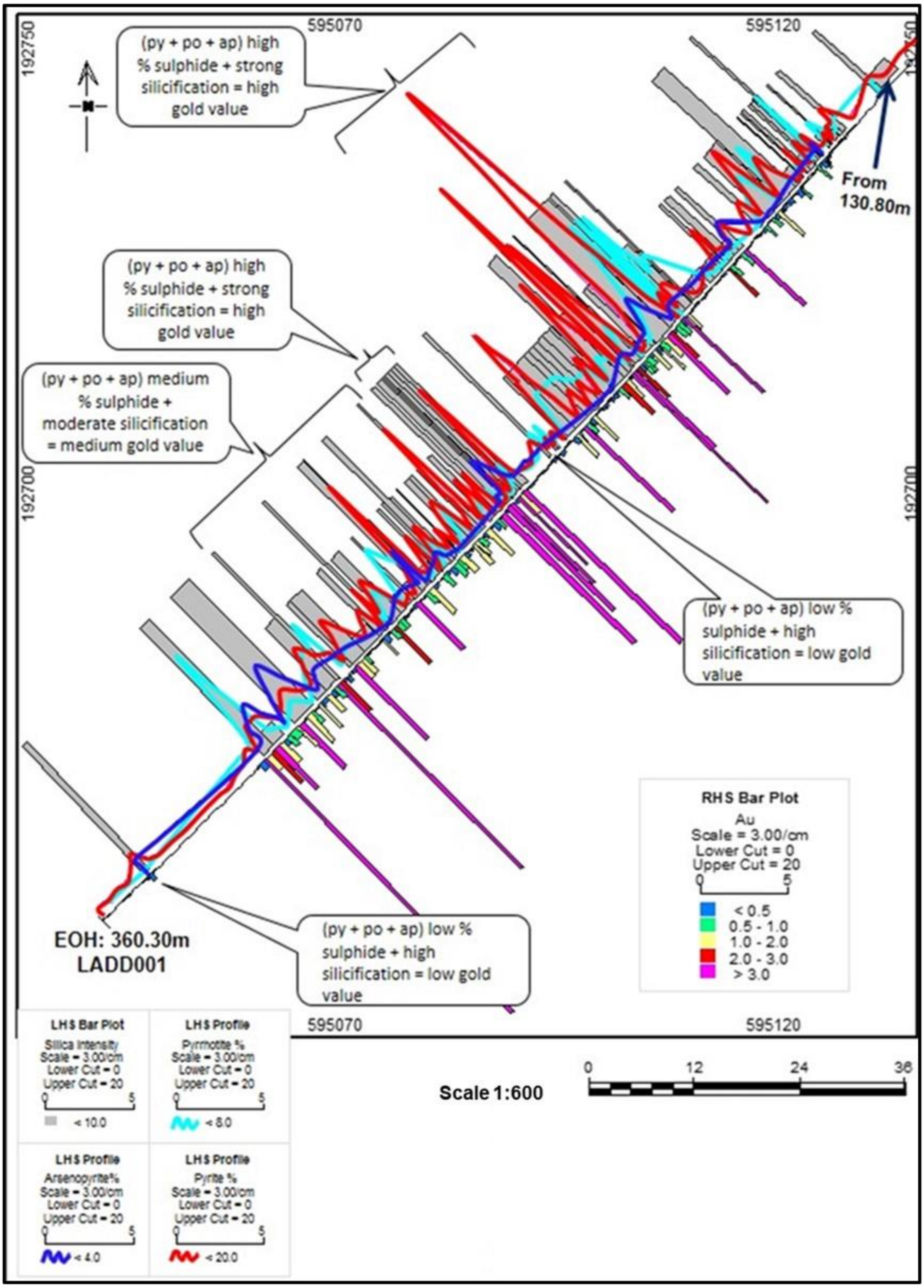


Figure 7.10: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD001

Table 7.3 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD003, and the corresponding gold grades.

Table 7.3: Relationship between Sulphides ± Silicification and Gold Grades in LADD003

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only			
Moderate silicification	Low gold values	0.04 g/t	63112
Pyrite only:			
Low % of pyrite + moderate silicification	Low gold values	0.09 g/t	63084
Medium % of pyrite + weak silicification	Low gold values	0.98 g/t	63096
Medium % of pyrite + moderate silicification	Medium gold values	1.27 g/t	63010
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + weak silicification	Low gold values	0.02 g/t	63136
Medium % of pyrite + pyrrhotite + weak silicification	High gold values	4.71 g/t	63194
Pyrite + Arsenopyrite:		8.07 g/t	63119
High % of pyrite + arsenopyrite + strong silicification	High gold values	8.07 g/t	63119
Pyrite + Pyrrhotite + Arsenopyrite			
Moderate % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	Medium gold values	2.72 g/t	63127
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	5.78 g/t	63121

Figure 7.11 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD003.

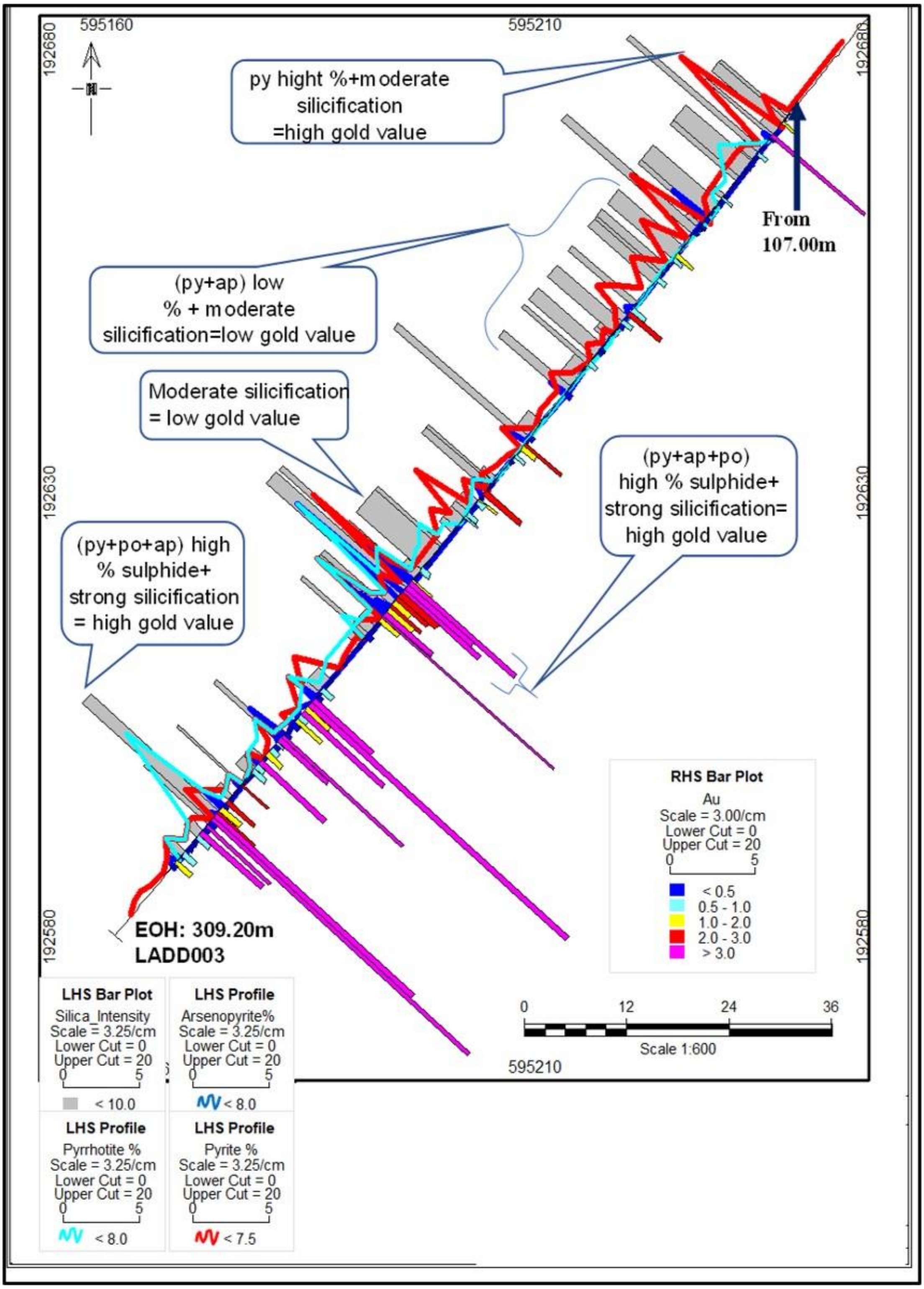


Figure 7.11: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD003

Table 7.4 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD004, and the corresponding gold grades.

Table 7.4: Relationship between Sulphides ± Silicification and Gold Grades in LADD004

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only			
Weak silicification	Low gold values	0.24 g/t	63315
Pyrite only:			
Low % of pyrite + weak silicification	Low gold values	0.03 g/t	63322
Low % of pyrite + moderate silicification	Low gold values	0.15 g/t	63303
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + moderate silicification	Low gold values	0.34 g/t	63271
Medium % of pyrite + pyrrhotite + weak silicification	Medium gold values	1.41 g/t	63247
Medium % of pyrite + pyrrhotite + moderate silicification	Medium gold values	1.37 g/t	63326
Medium % of pyrite + pyrrhotite + strong silicification	Medium gold values	2.14 g/t	63286
High % of pyrite + pyrrhotite + weak silicification	High gold values	6.79 g/t	63301
Pyrite + Pyrrhotite + Arsenopyrite			
Medium % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	High gold values	3.36 g/t	63335
High % of pyrite + pyrrhotite + arsenopyrite + weak silicification	High gold values	4.73 g/t	63332
High % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	High gold values	5.86 g/t	63354
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	8.52 g/t	63334

Figure 7.12 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD004.

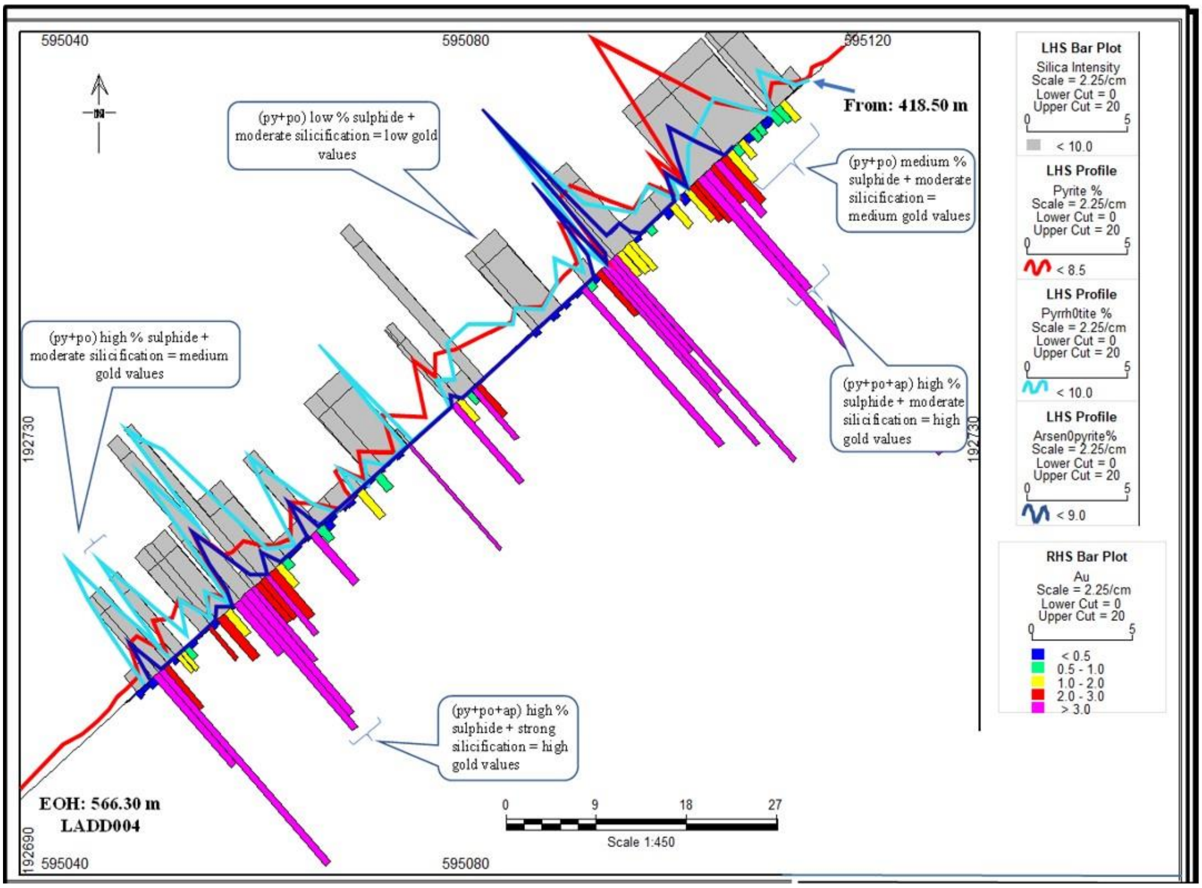


Figure 7.12: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD004

Table 7.5 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD006, and the corresponding gold grades.

Table 7.5: Relationship between Sulphides ± Silicification and Gold Grades in LADD006

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only:			
Weak silicification	Low gold values	0.07 g/t	63437
Moderate silicification	Low gold values	0.11 g/t	63431
Pyrite only:			
Low % of pyrite + weak silicification	Low gold values	0.18 g/t	63490
Low % of pyrite + moderate silicification	Low gold values	0.32 g/t	63485
Medium % of pyrite + moderate silicification	Low gold values	0.69 g/t	63420
Pyrrhotite only:			
Low % of pyrrhotite + weak silicification	Low gold values	0.27 g/t	63495

Composition	Observation	Typical Gold Grades	Sample No.
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + weak silicification	Low gold values	0.45 g/t	63422
Low % of pyrite + pyrrhotite + strong silicification	Low gold values	0.12 g/t	63498
Medium % of pyrite + pyrrhotite + moderate silicification	Medium gold values	1.34 g/t	63492
Medium % of pyrite + pyrrhotite + moderate silicification	High gold values	6.57 g/t	63489
Medium % of pyrite + pyrrhotite + strong silicification	Low gold values	0.79 g/t	63415
Pyrite + Arsenopyrite:			
Medium % of pyrite + arsenopyrite + strong silicification	Medium gold values	2.45 g/t	63556
High % of pyrite+ arsenopyrite + strong silicification	High gold values	4.73 g/t	63354
Pyrite + Pyrrhotite + Arsenopyrite:			
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	Medium gold values	2.12 g/t	63418
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	4.92 g/t	63519

Figure 7.13 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD006.

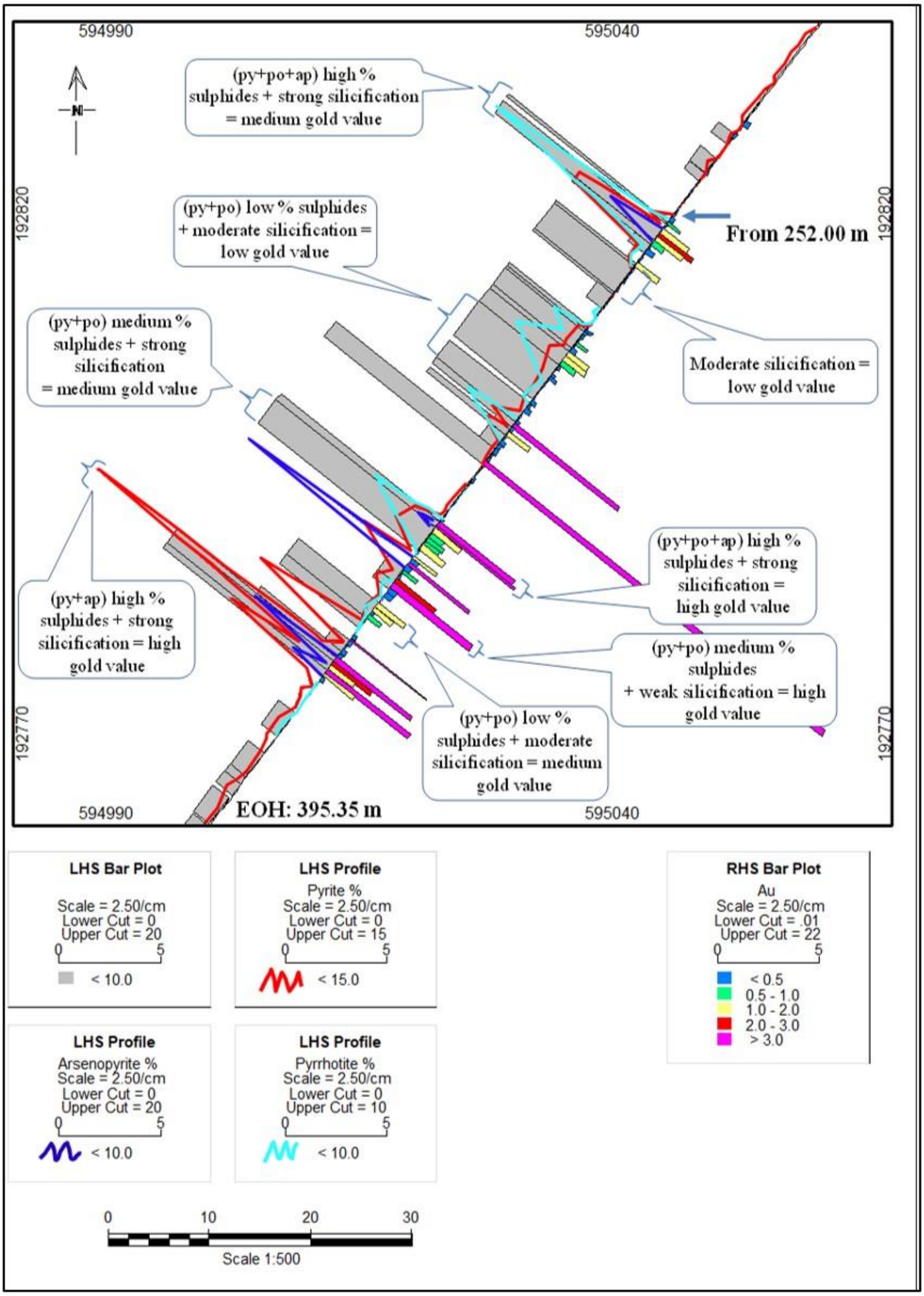


Figure 7.13: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD006

Table 7.6 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD007, and the corresponding gold grades.

Table 7.6: Relationship between Sulphides ± Silicification and Gold Grades in LADD007

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only:			
Weak silicification	Low gold values	0.02 g/t	63661
Moderate silicification	Low gold values	0.98 g/t	63704
Strong silicification	Low gold values	0.08 g/t	63709
Pyrite only:			
Low % of pyrite + weak silicification	Low gold values	0.01 g/t	63648
Low % of pyrite + moderate silicification	Low gold values	0.22 g/t	63703
Medium % of pyrite + moderate silicification	Medium gold values	2.80 g/t	63705
High % of pyrite + moderate silicification	Medium gold values	1.56 g/t	63726
Pyrrhotite only:			
Low % of pyrrhotite + weak silicification	Low gold values	0.14 g/t	63682
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + weak silicification	Low gold values	0.01 g/t	63648
Medium % of pyrite + pyrrhotite + moderate silicification	Medium gold values	2.20 g/t	63659
High % of pyrite + pyrrhotite + moderate silicification	Medium gold values	1.56 g/t	63726
Pyrite + Arsenopyrite:			
Low % of pyrite + arsenopyrite + weak silicification	Low gold values	0.54 g/t	63758
High % of pyrite + arsenopyrite + strong silicification	High gold values	6.17 g/t	63710
Pyrite + Pyrrhotite + Arsenopyrite:			
Medium % of pyrite + pyrrhotite + arsenopyrite + weak silicification	High gold values	3.96 g/t	63759
Medium % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	Medium gold values	2.04 g/t	63660
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	8.63 g/t	63767

Figure 7.14 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD007.

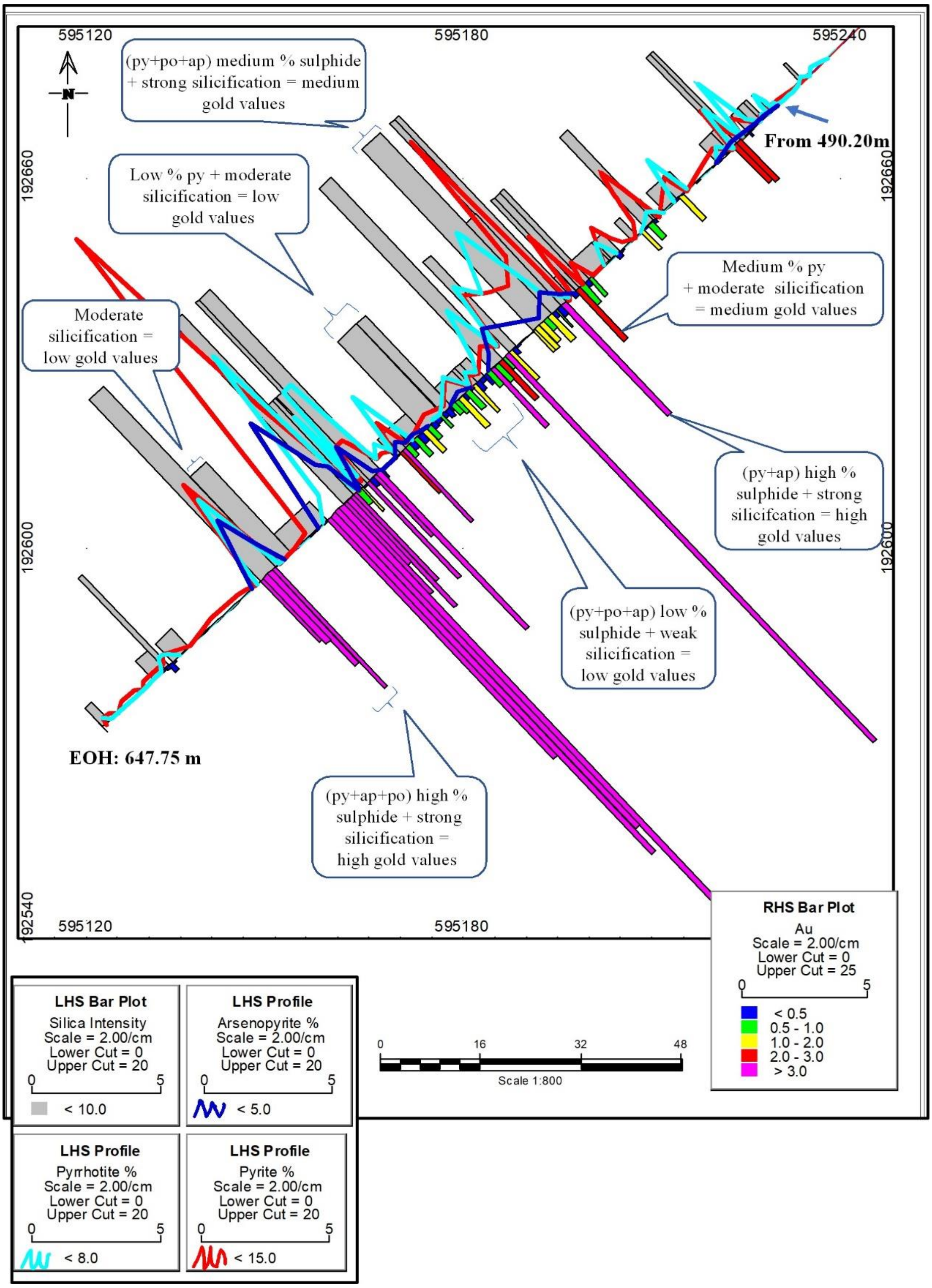


Figure 7.14: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD007

Table 7.7 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD008, and the corresponding gold grades.

Table 7.7: Relationship between Sulphides ± Silicification and Gold Grades in LADD008

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only:			
Weak silicification	Low gold values	< 0.01 g/t	63884
Pyrite only:			
Low % of pyrite + weak silicification	Low gold values	0.02 g/t	63953
Low % of pyrite + moderate silicification	Low gold values	0.02 g/t	63918
Medium % of pyrite + weak silicification	Low gold values	0.41 g/t	63894
Medium % of pyrite + moderate silicification	Low gold values	0.36 g/t	63898
Medium % of pyrite + strong silicification	Low gold values	0.96 g/t	63897
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + strong silicification	Medium gold values	1.19 g/t	63908
Medium % of pyrite + pyrrhotite + weak silicification	Low gold values	0.02 g/t	63893
Medium % of pyrite + pyrrhotite + moderate silicification	Medium gold values	1.06 g/t	63921
High % of pyrite + pyrrhotite + strong silicification	High gold values	3.33 g/t	63899
Pyrite + Arsenopyrite:			
Medium % of pyrite + arsenopyrite + moderate silicification	Medium gold values	1.04 g/t	63886
Pyrrhotite + Arsenopyrite:			
Low % of pyrite + arsenopyrite + strong silicification	Medium gold values	1.30 g/t	64012
Pyrite + Pyrrhotite + Arsenopyrite:			
Low % of pyrite + pyrrhotite + arsenopyrite + weak silicification	Low gold values	0.16 g/t	63906
Low % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	Low gold values	0.78 g/t	63997
Low % of pyrite + pyrrhotite + arsenopyrite + strong silicification	Low gold values	0.38 g/t	64024
Medium % of pyrite + pyrrhotite + arsenopyrite + weak silicification	Medium gold values	1.76 g/t	63944
High % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	Medium gold values	1.52 g/t	63925
Medium % of pyrite + pyrrhotite + arsenopyrite + strong silicification	Medium gold values	2.56 g/t	63976
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	6.28 g/t	63914

Figure 7.15 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD008.

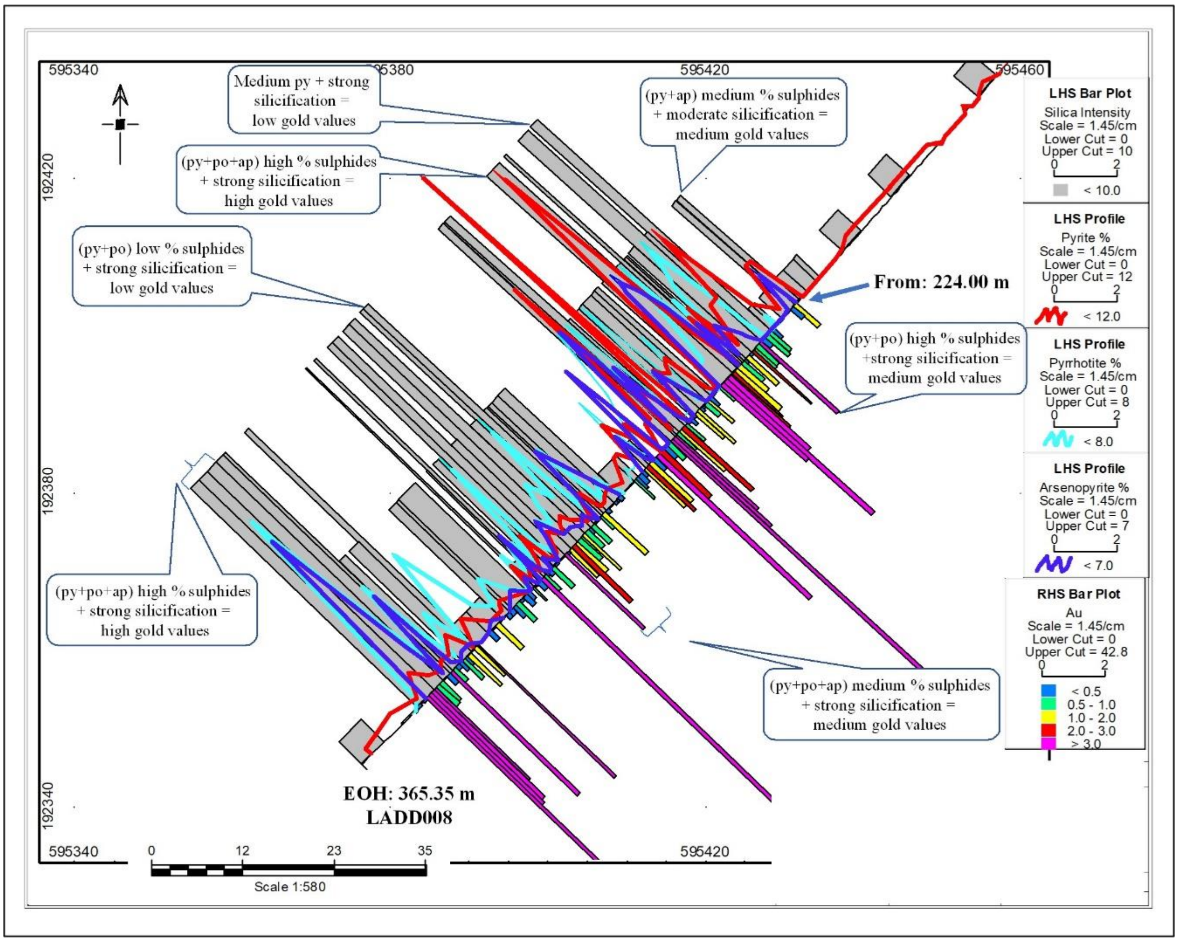


Figure 7.15: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD008

Table 7.8 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD009, and the corresponding gold grades.

Table 7.8: Relationship between Sulphides ± Silicification and Gold Grades in LADD009

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only:			
Weak silicification	Low gold values	0.04 g/t	64174
Moderate silicification	Low gold values	< 0.01 g/t	64048
Pyrite only:			
Low % of pyrite + weak silicification	Low gold values	0.08 g/t	64064
Medium % of pyrite + weak silicification	Medium gold values	1.50 g/t	64077
Medium % of pyrite + moderate silicification	Medium gold values	2.62 g/t	64113
Pyrrhotite only:			
Low % of pyrrhotite + weak silicification	Low gold values	0.06 g/t	64080

Composition	Observation	Typical Gold Grades	Sample No.
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + weak silicification	Low gold values	0.03 g/t	64051
Medium % of pyrite + pyrrhotite + weak silicification	Low gold values	0.48 g/t	64059
Medium % of pyrite + pyrrhotite + strong silicification	High gold values	5.06 g/t	64116
Pyrite + Pyrrhotite + Arsenopyrite:			
Medium % of pyrite + pyrrhotite + arsenopyrite + weak silicification	High gold values	3.12 g/t	64084
Medium % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	High gold values	9.13 g/t	64170
High % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	High gold values	4.18 g/t	64055
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	17.5 g/t	64153

Figure 7.16 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD009.

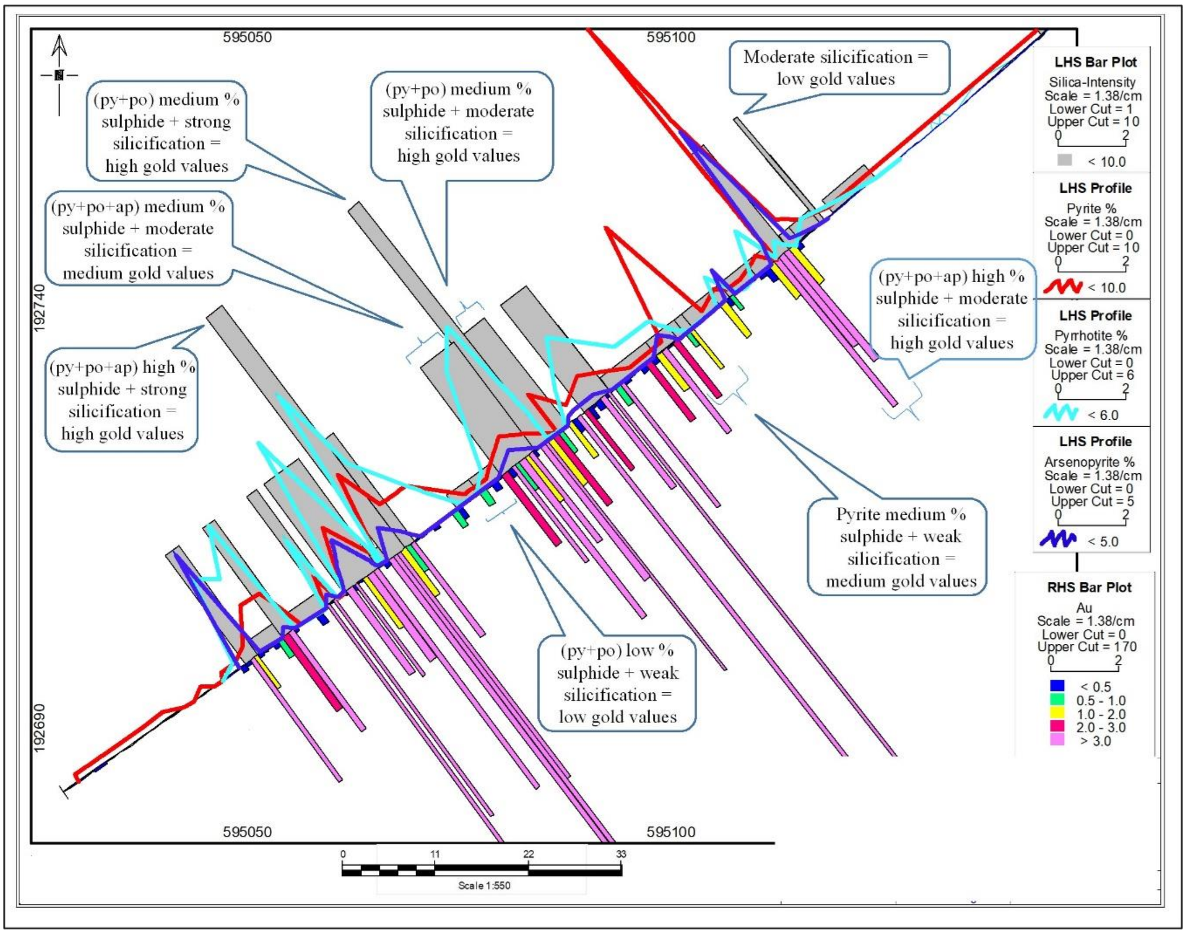


Figure 7.16: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD009

Table 7.9 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD012, and the corresponding gold grades.

Table 7.9: Relationship between Sulphides ± Silicification and Gold Grades in LADD012

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only:			
Weak silicification	Low gold values	0.07 g/t	64386
Moderate silicification	Low gold values	0.02 g/t	64603
Strong silicification	Low gold values	0.02 g/t	64576
Pyrite only:			
Low % of pyrite + weak silicification	Low gold values	< 0.01 g/t	64357
Low % of pyrite + moderate silicification	Low gold values	< 0.01 g/t	64352
Low % of pyrite + strong silicification	Low gold values	0.19 g/t	64594
Pyrrhotite only:			
Low % of pyrrhotite + moderate silicification	Low gold values	0.22 g/t	64672
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + weak silicification	Low gold values	0.04 g/t	64332
Low % of pyrite + pyrrhotite + moderate silicification	Low gold values	0.14 g/t	64573
Medium % of pyrite + pyrrhotite + weak silicification	Low gold values	0.04 g/t	64116
Medium % of pyrite + pyrrhotite + strong silicification	Low gold values	0.02 g/t	64350
High % of pyrite + pyrrhotite + moderate silicification	Low gold values	0.04 g/t	64617
Pyrite + Arsenopyrite:			
Low % of pyrite + arsenopyrite + weak silicification	Low gold values	0.05 g/t	64381
Medium % of pyrite + arsenopyrite + moderate silicification	Low gold values	0.69 g/t	64390
Pyrrhotite + Arsenopyrite:			
Low % of pyrrhotite + arsenopyrite + moderate silicification	Low gold values	0.28 g/t	64674
Pyrite + Pyrrhotite + Arsenopyrite:			
Medium % of pyrite + pyrrhotite + arsenopyrite + weak silicification	Low gold values	0.84 g/t	64355
Medium % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	Low gold values	0.03 g/t	64346
High % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	High gold values	16 g/t	64370
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	10 g/t	64397

Figure 7.17 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD012.

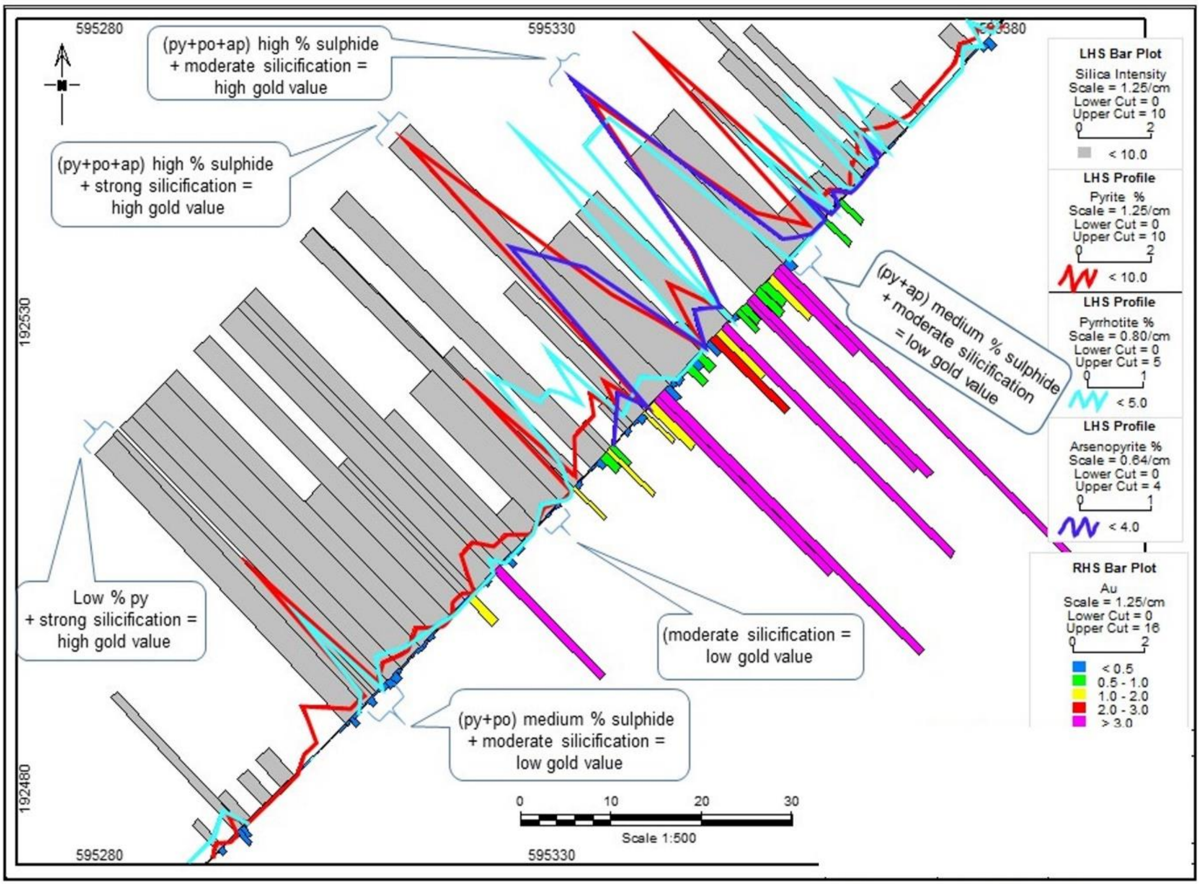


Figure 7.17: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD012

Table 7.10 presents varying intensities of sulphide types in combinations with various degrees of silicification within the mineralised zones in LADD013, and the corresponding gold grades.

Table 7.10: Relationship between Sulphides ± Silicification and Gold Grades in LADD013

Composition	Observation	Typical Gold Grades	Sample No.
Silicification only:			
Weak silicification	Low gold values	0.33 g/t	64508
Strong silicification	Low gold values	0.06 g/t	64472
Pyrite only:			
Low % of pyrite + weak silicification	Low gold values	< 0.01 g/t	64415
Low % of pyrite + moderate silicification	Low gold values	0.04 g/t	64417
Low % of pyrite + strong silicification	Low gold values	0.09 g/t	64466
Medium % of pyrite + moderate silicification	Low gold values	0.40 g/t	64533
Pyrite + Pyrrhotite:			
Low % of pyrite + pyrrhotite + weak silicification	Low gold values	0.03 g/t	64469

Composition	Observation	Typical Gold Grades	Sample No.
Low % of pyrite + pyrrhotite + moderate silicification	Low gold values	0.03 g/t	64427
Medium % of pyrite + pyrrhotite + moderate silicification	Medium gold values	1.22 g/t	64506
Medium % of pyrite + pyrrhotite + weak silicification	Low gold values	0.67 g/t	64522
High % of pyrite + pyrrhotite + strong silicification	Low gold values	0.8 g/t	64471
Pyrite + Arsenopyrite			
Low % of pyrite + arsenopyrite + weak silicification	Low gold values	0.12 g/t	64447
Medium % of pyrite + arsenopyrite + moderate silicification	Medium gold values	2.13 g/t	64487
Pyrite + Pyrrhotite + Arsenopyrite:			
Medium % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	Medium gold values	1.33 g/t	64457
High % of pyrite + pyrrhotite + arsenopyrite + moderate silicification	High gold values	4.46 g/t	64509
High % of pyrite + pyrrhotite + arsenopyrite + strong silicification	High gold values	6.09 g/t	64461

Figure 7.18 displays the relationship between sulphides ± silicification and gold grades in Drillhole LADD013.

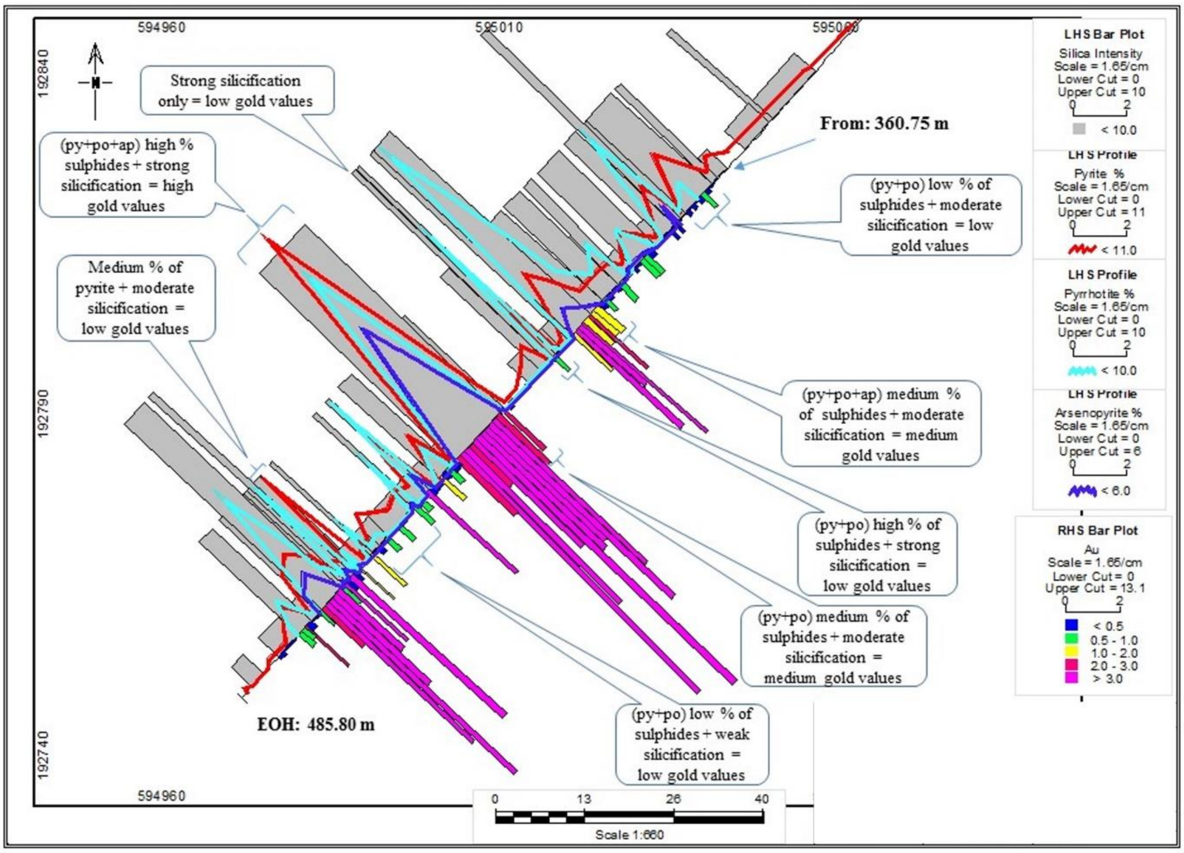


Figure 7.18: Relationship between Sulphides ± Silicification and Gold in Drillhole LADD013

7.4.3 Range of Classifications

The range of classifications used are shown in Table 7.11.

Table 7.11: Range of Classification of Sulphides, Silicification and Gold Grades

Sulphides	Silicification	Gold Values
≤ 1 %: low percentage	Weak	Au ≤ 1.0 g/t: low gold value
> 1 % to ≤ 5 %: medium percentage	Moderate	1.0 g/t < Au ≤ 3 g/t: medium gold value
> 5 %: high percentage	Strong	Au > 3.0 g/t: high gold value

As evidenced from the above, a direct relationship exists between gold values and the percentage of sulphide mineralisation and the intensity of silicification. Some of the composition assemblages are absent within the sampled zones, for instance a high percentage of pyrite only + strong silicification etc. Pyrite is associated with all the assemblages; hence, it is difficult to have only a pyrrhotite + arsenopyrite + silicification composition.

In general, pyrite is the dominant sulphide followed by pyrrhotite, then arsenopyrite. When pyrite and pyrrhotite are associated with arsenopyrite, the gold values are very significant, compared to when pyrite is associated with pyrrhotite only. Silica is associated with the highest degree of hydrothermal alteration within the zones and serves as a marker of mineralisation; however, without sulphides, the gold values are insignificant.

Specks of visible gold, generally within fractures, are present in white to grey, glassy, weak to moderately brecciated quartz veins (with variable widths from a few centimetres up to 1 m), with low percentages of sulphide, mainly localised within the RP zone in some drillholes. Thus, a low percentage of py + ap + str qv = high gold values (21.20 g/t gold in sample No. 63500 in LADD006).

7.4.4 Visible Gold (VG)

Visible gold was logged in Drillhole LADD001 in a 5.6 m RP zone (highly silicified with 1.5 % pyrite, 1 % arsenopyrite, and 0.5 % pyrrhotite) logged from 251.30 m. In the unsplit core, four spots of visible gold were identified at 255.60 m, and between 256.2 m and 256.3 m (see Figure 7.19 a and b). Upon splitting the core, three specks of visible gold were identified from 256.10 m to 256.13 m and at 256.54 m (see Figure 7.19 c and d).

a) VG at 255.60 m in unsplit core

b) VG between 256.2 m and 256.3 m in unsplit core

c) VG between 256.10 m and 256.13 m in split core

d) VG at 256.54 m in split core

Figure 7.19: Visible Gold in Unsplit Core and Split Core

Four specks of visible gold were also logged in Drillhole LADD026 within the interval 593.26 m to 594.84 m (see Figure 7.20 a and b).

a) VG in quartz vein within RP zone at 593.26 m

b) VG in quartz vein within RP zone at 594.71 m

Figure 7.20: Visible Gold in Drillhole LADD026

7.5 STRUCTURES

Gold mineralisation within the Adumbi deposit is related to the northwest trending shear zones, which dip steeply towards the northeast and which, in some parts of the area, seem to utilise the competency contrast between two lithologies, namely the BIF-chert and the tuffaceous-greywacke metasedimentary rocks.

This mineralisation occurs over a strike length of 2 km in a zone approximately 100 m wide to a depth of approximately 560 m. The continuity of the mineralisation appears to be oriented vertically close to the wall rocks of the BIF. The strike orientation of the BIF is northwest-southeast, which is parallel to the trend of the Upper Kibalian rocks. The BIF is interpreted to have a steep, near-vertical dip. A series of north-northwest striking faults appear to dislocate the BIF, and it is interpreted that these faults have a strike-slip component, resulting in an apparent thickening of the BIF in the central part of Adumbi.

Structural logging for the Kitenge holes is limited due to the lack of orientated core. However, some observed structural features include zones of strong shearing associated with extensive ankerite alteration, bleaching, quartz veining and isolated low-grade gold mineralisation in Drillhole SKDD0028. These zones are very important as gold mineralisation in Kitenge is mostly associated with these zones especially when there is a relative high content of sulphides and, in particular, the presence of arsenopyrite

Structural logging for the Manzako holes is limited due to the lack of orientated core. Quartz veining within the shear zones control the mineralisation.

7.5.1 Imbo Project Structural Data Analysis

Structural data compilation and interpretation for the Imbo Permit was undertaken to collate all the available data from recent and previous mapping programmes, domain the data sets, and plot and interpret the data using Dips software. The objectives were to

- Interpret the structural framework of the Imbo Permit on a regional and prospect scale, and to determine the regional and local structural controls on the distribution of gold mineralisation.
- Use this in conjunction with geophysical and geochemical data to
  - Prioritise new prospect areas for follow-up.
  - Investigate potential extensions in the vicinity of known mineralisation.

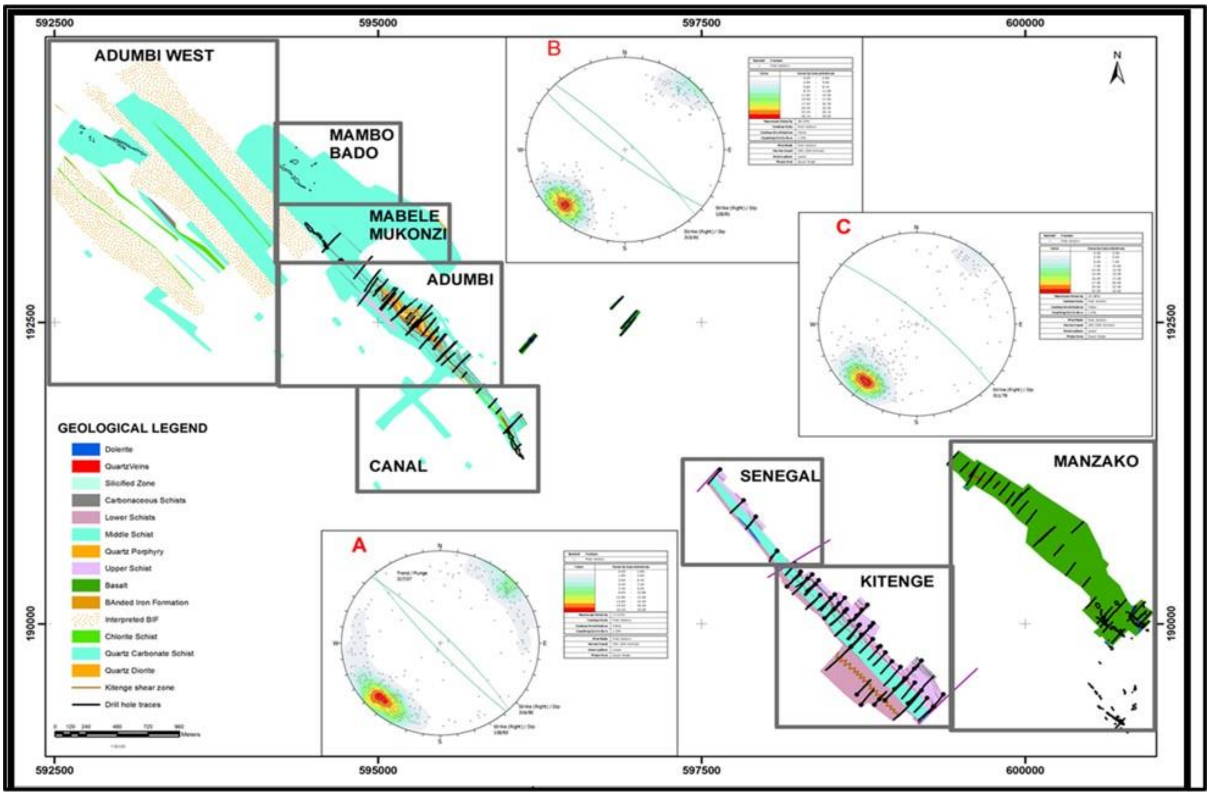
Data was collated from the following sources:

- Structural readings taken since March 2014, which are recorded in database format and plotted in plan
- Integration of the underground mapping data gathered by N. Hewson from Adumbi
- Earlier structural data extracted from maps

Once all the future drill cores are oriented, it would be possible for structural measurements to be taken and integrated with the structural data from other sources.

A total of 1,046 measurements (bedding, foliations and quartz veins) covering the Adumbi West, Adumbi, Canal, Senegal, Kitenge and Manzako deposits were compiled from the above-mentioned sources. These measurements were taken using a strike (right)/dip convention. Plans showing foliation, bedding and quartz veins with inserts of respective stereonet plots are presented in Figure 7.21.

In general, the stereonet plots for the available data on the Imbo Project show that quartz veins are generally subparallel to the foliation and bedding with average orientations of 311°/78°, 315°/81° and 316°/80°, respectively. This conforms well to the regional trend that is well defined in the geophysical data of the Imbo Project. It can also be noted that stereonet plots for bedding show two major planes which define a fold oriented 317°/07°, a possible regional fold representing an early folding event of the Imbo Project.



NOTE: Geology, stereonet plot for bedding (A), foliation (B), quartz veins (C), drillhole traces and location of targets/prospects.

Figure 7.21: Imbo Project – Bedding, Foliation and Quartz Veins with Stereonet Plots

A further analysis of the structural data involved domaining the data on the basis of deposits/targets and its association with the known mineralisation within the deposit/ target. This was done with the aim of assisting in a detailed structural interpretation on a deposit/target scale. Details for Adumbi, Kitenge-Senegal and Manzako are provided below.

7.5.1.1 Adumbi Deposit

Most of the existing structural data for Adumbi is from underground mapping with some additional data from regional mapping that commenced in March 2014. Figure 7.22 to Figure 7.24 show bedding, foliations and quartz veins plotted on plans with the inserts of the respective stereonet plots.

Stereonet plots for bedding show two major planes oriented 315°/81° and 137°/84° defining a shallow northwesterly plunging fold (316°/07°), see Figure 7.22. The geometry of this fold does not conform to the architectural behaviour of the Adumbi mineralisation described in this section due to the fact that this fold possibly represents an earlier folding event that has been mostly over-printed by the later shear-related folding. This is further emphasised by the fact that most of the bedding measurements were taken in the areas that are not in the strongly folded and deformed zones.

It is observed that foliations are generally parallel to bedding (see Figure 7.23), with average orientations of 314°/79° and 315°/81°, respectively, while the quartz veins have a general relatively less northerly orientation of 309°/79°.

Figure 7.24 also shows the stereonet plot for the Adumbi quartz veins, which have two major planes oriented 309°/79° and 125°/83° defining a linear structure that is shallowly plunging to the southeast. It is not known if the intersection of these quartz vein major planes is associated with the mineralising event, but it is doubtful as it is known from previous interpretation that mineralisation at Adumbi is characterised by steep plunging shoots.

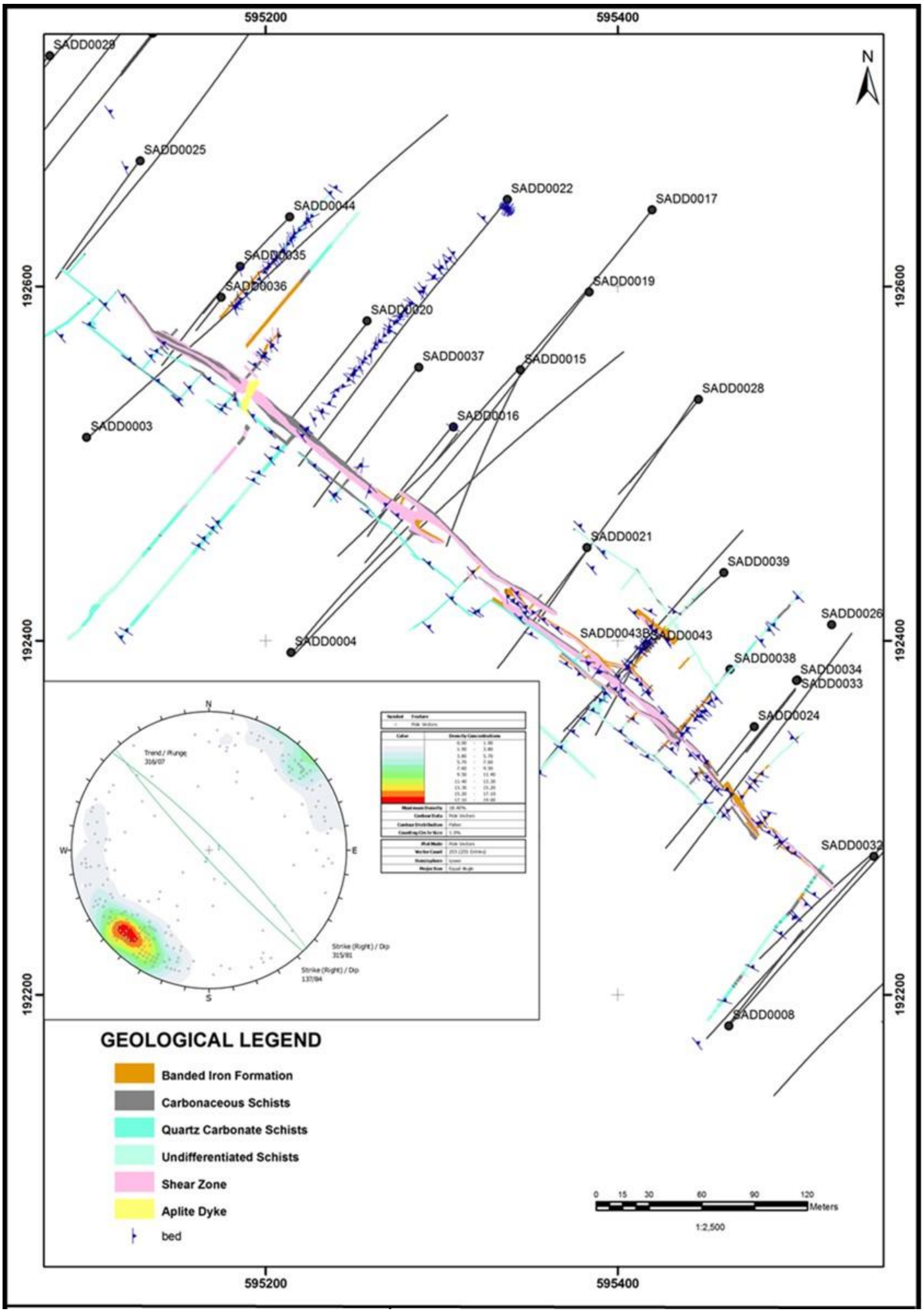


Figure 7.22: Adumbi Deposit – Geology from Underground Mapping Bedding Planes (Insert of Stereonet Plot for Bedding)

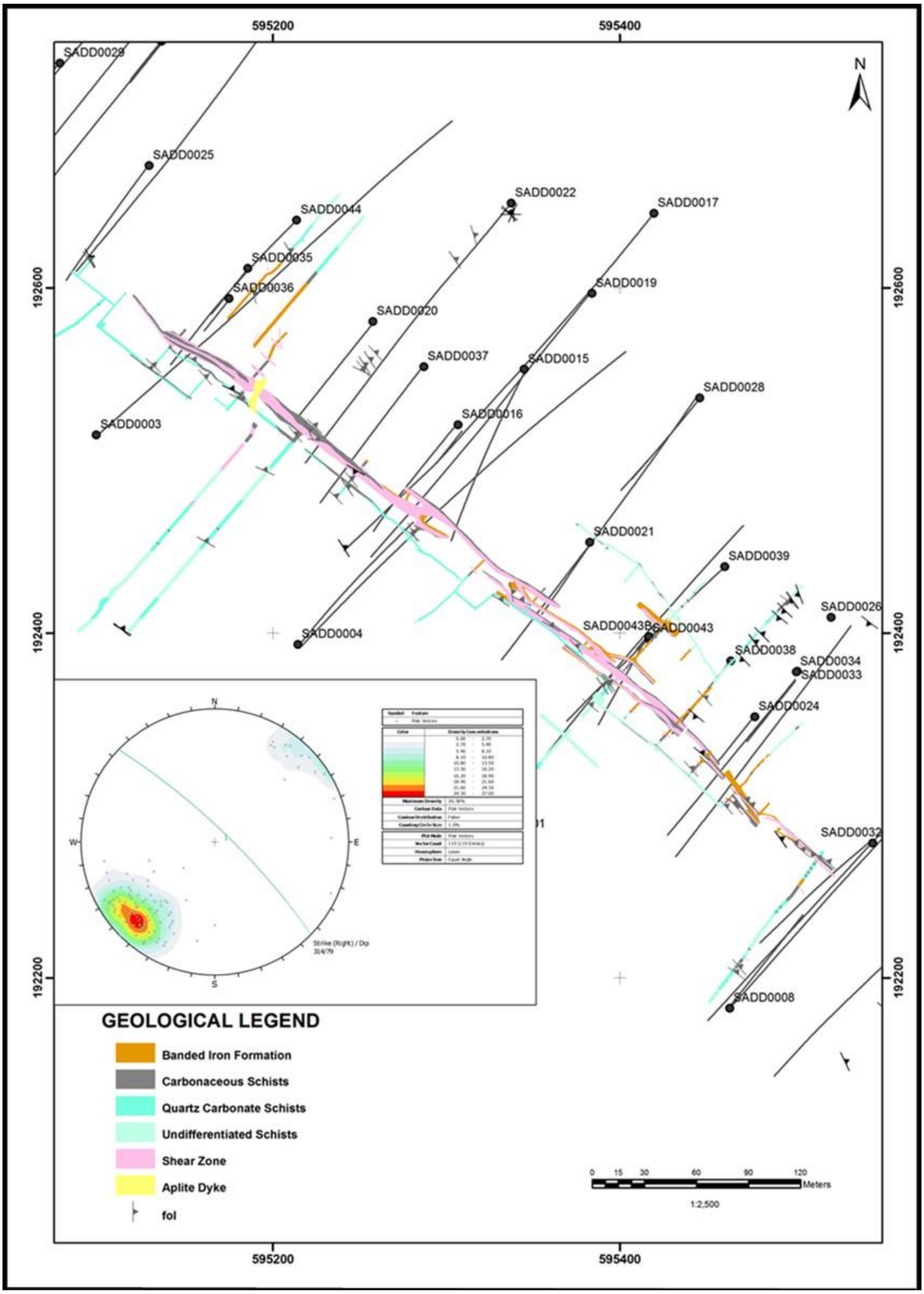


Figure 7.23: Adumbi Deposit – Geology from Underground Mapping Bedding Planes (Insert of Stereonet Plot for Foliation)

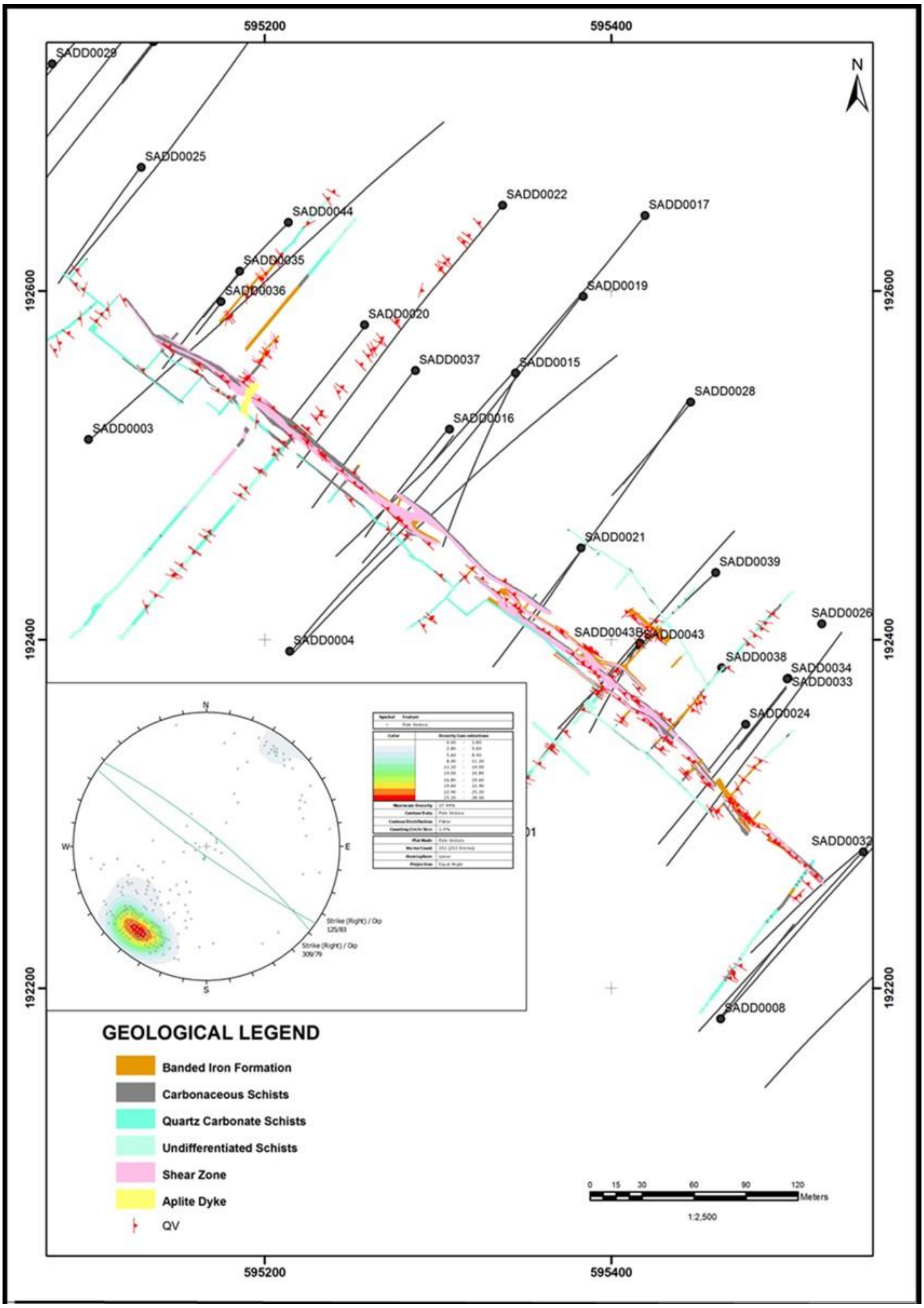


Figure 7.24: Adumbi Deposit – Geology from Underground Mapping, Quartz Veins (Insert of Stereonet Plot for Quartz Veins)

For a better understanding of the structural behaviour along strike and across the Adumbi mineralisation, the structural data was domained, and stereonet plots for bedding in selected domains (blocks labelled 1 to 4) were inserted as shown in Figure 7.25.

The stereonet plot for bedding in Domain 1 shows two major planes oriented 316°/82° and 137°/82° that define a northwesterly shallow plunging fold (316°/04°), possibly representing the earlier folding event of the Imbo Project.

Bedding in Domain 2 shows two major planes oriented 317°/79° and 351°/76°, defining a northeasterly, steeply plunging fold (087°/76°).

Bedding in Domain 3 shows three major planes oriented 318°/79°, 162°/81° and 013°/81°, defining folds that are trending 063°/78°, 330°/51° and 177°/58°.

Bedding in Domain 4, which covers the area of no shearing or deformation, shows a major plane oriented 319°/83°, representing a regional trend of the Imbo Project.

Folds defined in Domains 2 and 3 are possibly shear-related folds and are probably minor folds that represent a major fold which is partially exposed in Adumbi. Underground mapping suggests that the fold axes of these minor folds are parallel to the Adumbi shear zone and that the shear zone possibly represents the axial plane of a major fold.

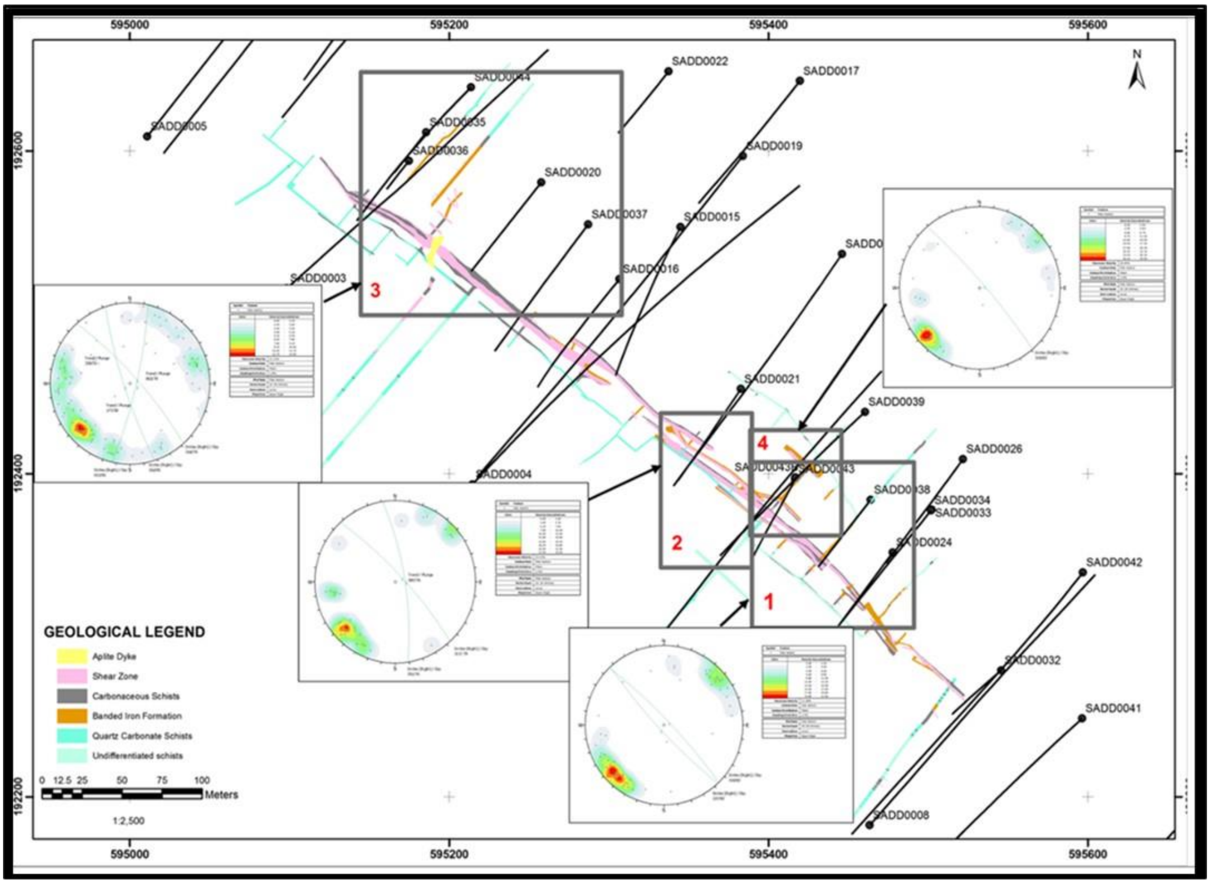


Figure 7.25: Adumbi Deposit – Geology from Underground Mapping (Inserts of Stereonet Plots for Selected Domains (Blocks 1 to 4))

7.5.1.2 Kitenge and Senegal

The Kitenge and Senegal deposit is located southeast of the Canal area. Gold mineralisation is hosted in quartz veins within sheared and altered metasediments, mainly quartz carbonate schist, and the structure is interpreted as a faulted structure of Adumbi.

The stereonet plot for foliation attitudes indicates an average orientation of 318°/79° (see Figure 7.26, Insert 3), which is generally similar to the regional trend. Due to a lack of quartz and bedding measurements, no stereonet plots were produced.

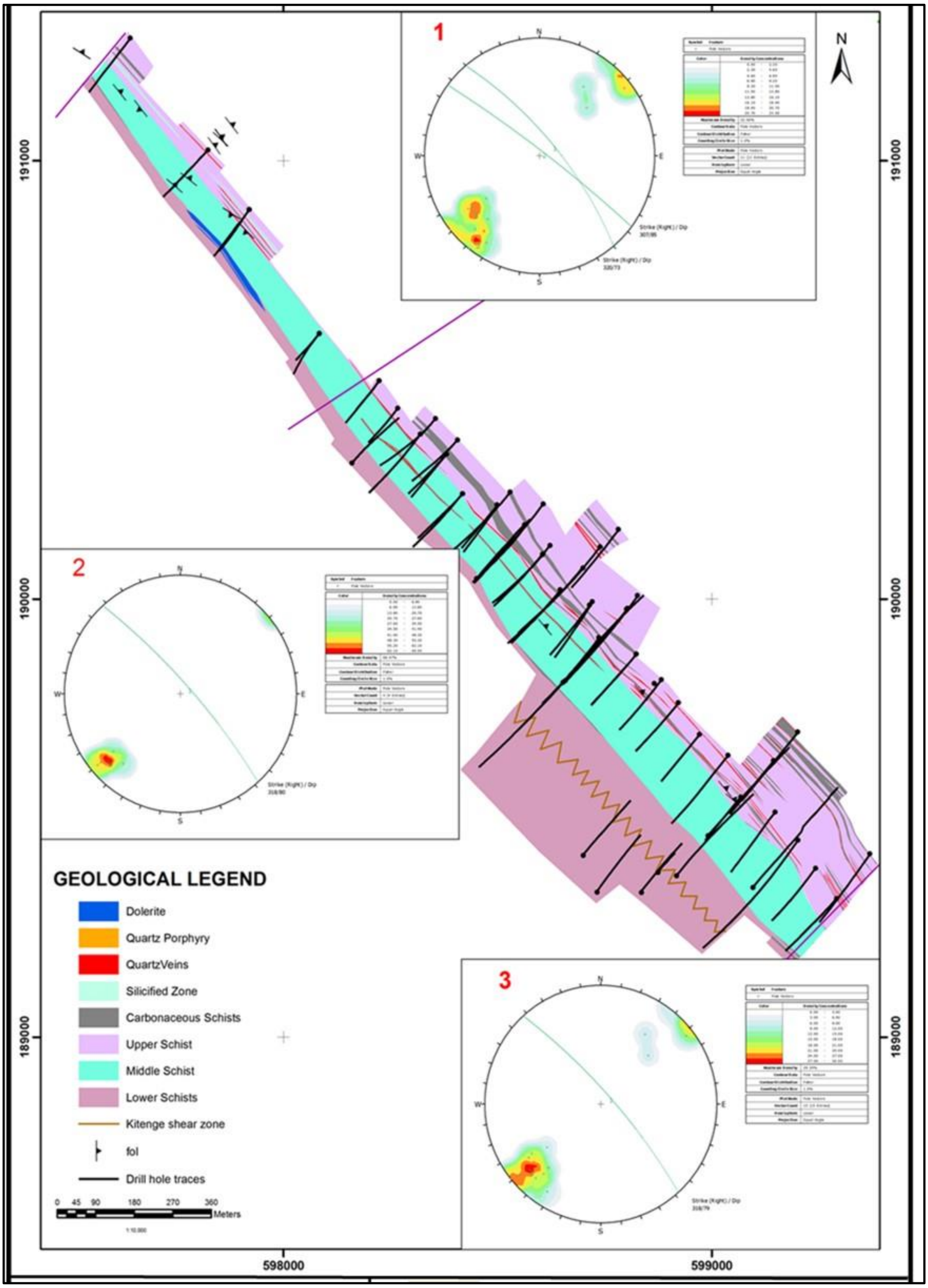


Figure 7.26: Adumbi Deposit – Geology Foliations (Inserts of Stereonet Plots for Foliations at Senegal, Kitenge and Senegal-Kitenge Area)

7.5.1.3 Manzako

The current interpretation shows that the mineralised structure in Manzako is different from the Adumbi-Canal-Senegal-Kitenge structure.

Gold mineralisation in the Manzako deposit is hosted in quartz veins emplaced within sheared basalt.

It is observed that Manzako has two distinct foliation trends orienting at 316°/78° and 148°/76°, respectively. The intersection lineation plunges shallowly to the northwest (see Figure 7.27, Insert 1). There are few quartz measurements; the available data suggests quartz veins cross cuts foliations at 302°/81° (see Figure 7.27, Insert 2).

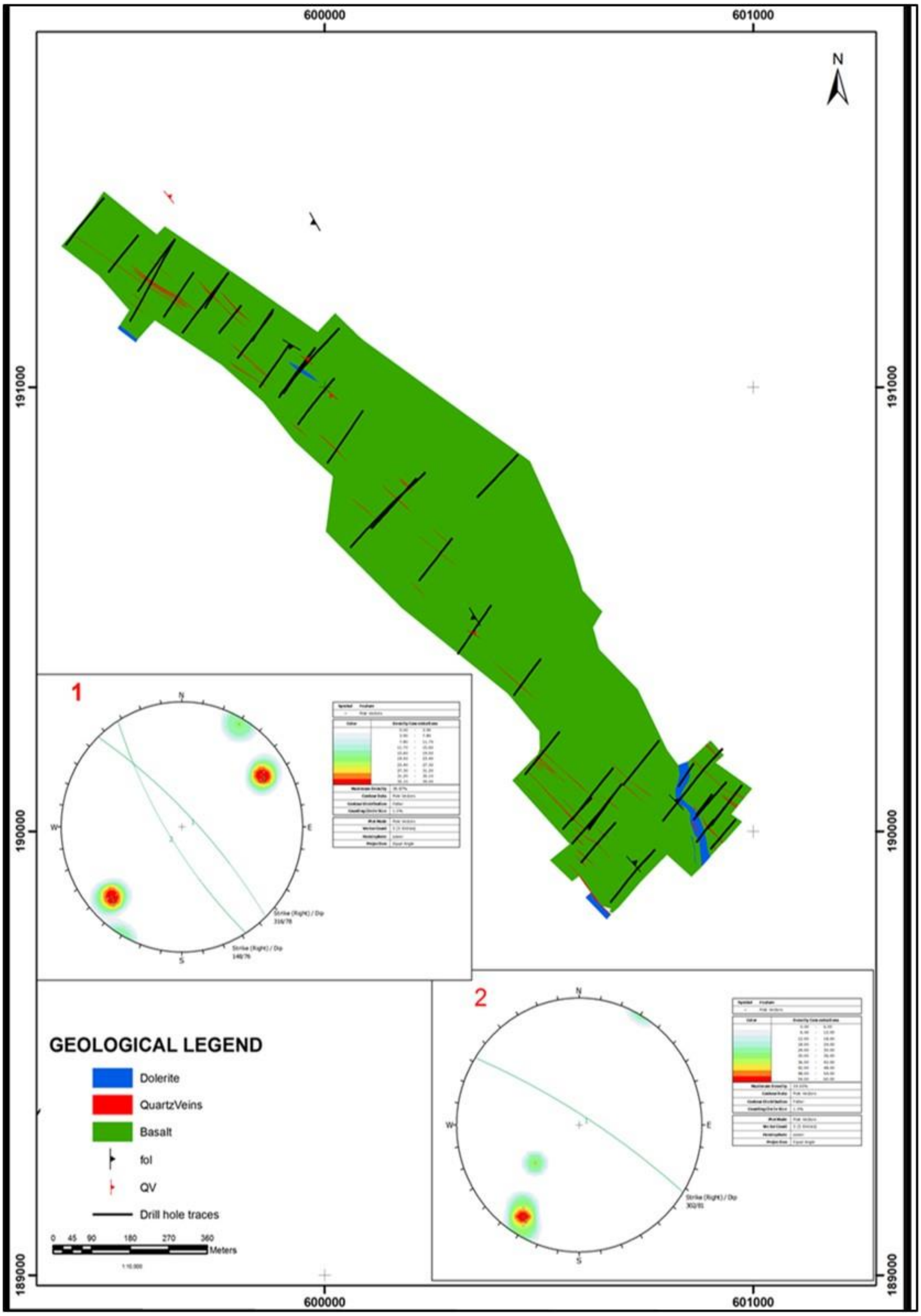


Figure 7.27: Manzako Deposit – Geology, Foliations, Quartz Veins (Inserts of Stereonet Plot for Foliations and Quartz Veins)

In summary, the following are observations derived from the structural analysis of the Imbo Project:

- Presence of regional fold (Imbo fold), which plunges shallowly to the northwest (07°/316°).
- Regionally, foliations are subparallel to beddings.
- Possible presence of shear-related tight fold at Adumbi area, indicated by steeply plunging folds adjacent to mineralised structure.
- Presence of two structures which intersect at Adumbi and split in the NW and SE of Adumbi in the Mabele Mokonzi-Mambo Bado and Canal areas, respectively.
- Foliations and mineralised quartz vein trends at Vatican have generally fewer northerly orientations in comparison to those at Adumbi.

7.5.2 Structural Interpretation from 2020 to 2021 Drilling Programme at Adumbi

The structural interpretation to date from the drillholes completed during the 2020 to 2021 drilling programme at Adumbi is presented below.

7.5.2.1 Bedding

The stereonet plot for bedding for 2,820 poles from LADD001, LADD003, LADD004, LADD006, LADD007, LADD008, LADD009, LADD012, LADD013, LADD014, LADD015, LADD016, LADD017, LCDD001, LADD018, LADD019, LADD020, LADD021, LADD022, LADD023, LADD024, LADD025 and LADD026 shows a plane oriented at 314°/88° dipping NE (see Figure 7.28), which confirms the general trend of the Adumbi formation which is NW-SE and dips to the NE.

Other poles, however, plot 134°/89°, dipping to the SW, thus representing folding.

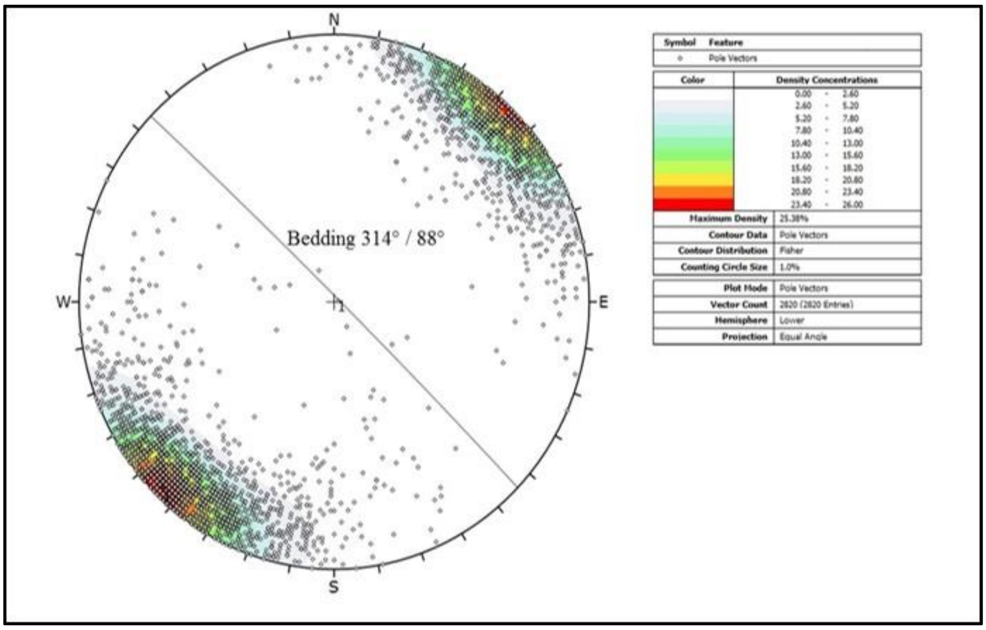


Figure 7.28: Stereonet Plot for Bedding Planes from Completed 2020 to 2021 Drillholes

7.5.2.2 Foliation

It is observed that foliations for 3,245 poles from LADD001, LADD003, LADD004, LADD006, LADD07, LADD008, LADD009, LADD012, LADD013, LADD014, LADD015, LADD016, LADD017, LCDD001, LADD018, LADD019, LADD020, LADD021, LADD022, LADD023, LADD024, LADD025 and LADD026 are generally parallel to bedding with an average orientation of 314°/87° (see Figure 7.29). Like bedding, other foliation poles plot 135°/89°, with a subvertical limb dipping to the SW, representing folding.

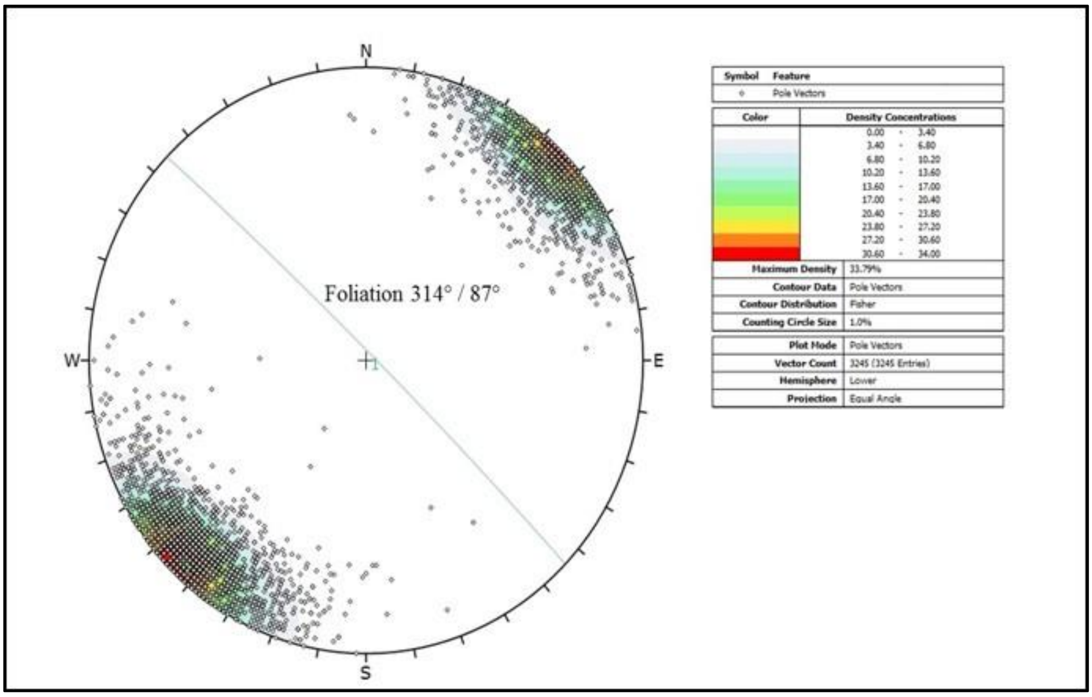


Figure 7.29: Stereonet Plot for Foliation Planes from Completed 2020 to 2021 Drillholes

7.5.2.3 Quartz Veins

The stereonet plot for 1,827 poles of quartz veins from LADD001, LADD003, LADD004, LADD006, LADD007, LADD008, LADD009, LADD012, LADD013, LADD014, LADD015, LADD016, LADD017, LCDD001, LADD018, LADD019, LADD020, LADD021, LADD022, LADD023, LADD024, LADD025 and LADD026 is generally oriented 313°/85°, which is almost parallel to the bedding/foliation (see Figure 7.30). However, the plot of quartz veins has a relatively less northerly orientation. A few post-mineralisation quartz veins are observed cutting across both bedding and foliation, and in some cases, they are suspected to have displaced the mineralisation.

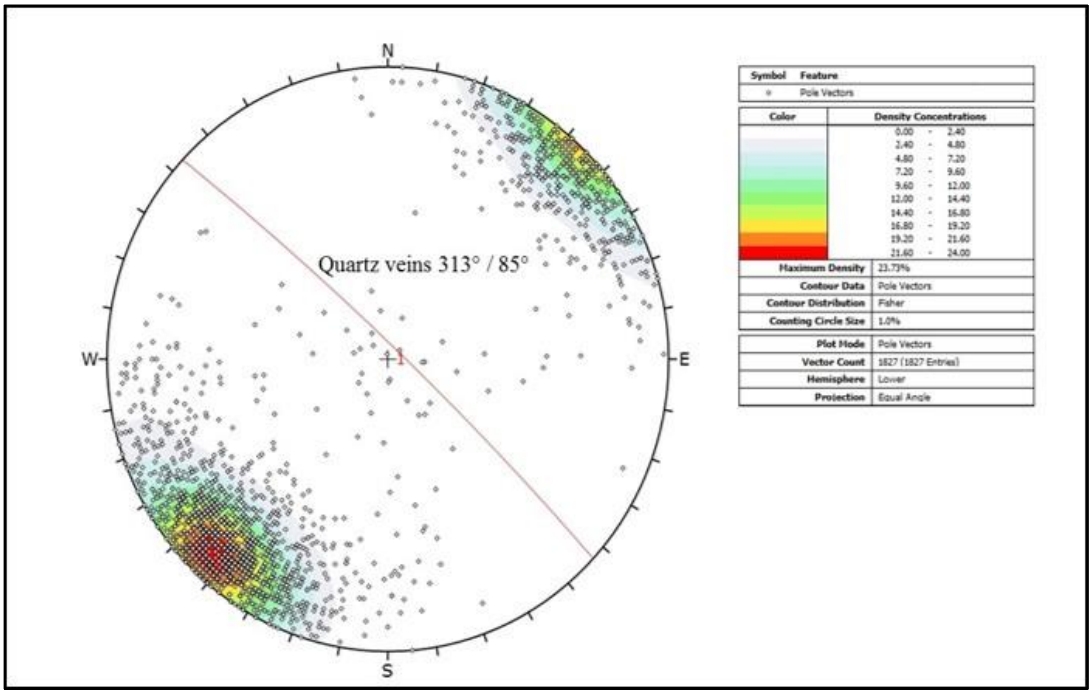


Figure 7.30: Stereonet Plot for Quartz Veins from Completed 2020 to 2021 Drillholes

7.5.2.4 Bedding/Foliation Intersection Lineation

The bedding/foliation intersection lineation is 00° at 133° (see Figure 7.31). If the foliation is axial planar, then this intersection lineation approximates a fold axis.

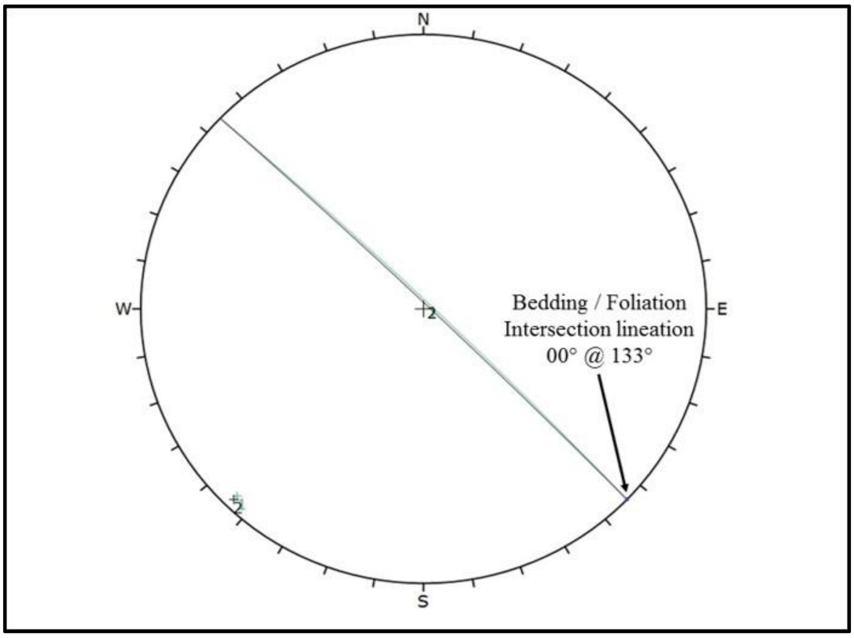


Figure 7.31: Stereonet Plot for Intersection Lineation of Bedding and Foliation from Completed 2020 to 2021 Drillholes

7.5.2.5 Foliation/Quartz Vein Intersection Lineation

The foliation/quartz vein intersection lineation is 26° at 315° (see Figure 7.32).

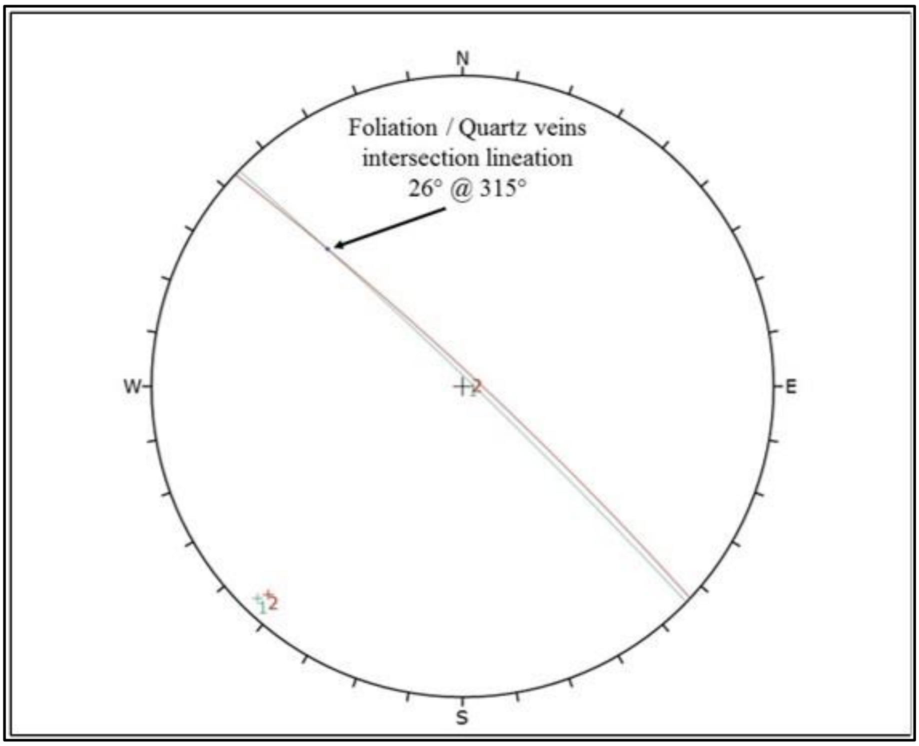


Figure 7.32: Stereonet Plot for Intersection Lineation of Foliation and Quartz Veins from Completed 2020 to 2021 Drillholes

The structural interpretation from the recent drilling programme thus supports the general Adumbi formation.

8 DEPOSIT TYPES

Gold deposits within the Imbo Project are associated with the globally important Neo-Archean orogenic gold deposits, examples of which are found in most Neo-Archean cratons around the world. Gold mineralisation is associated with the epigenetic mesothermal style of mineralisation. This style of mineralisation is typical of gold deposits in Neo-Archean greenstone terranes and is generally associated with regionally metamorphosed rocks that have experienced a long history of thermal and deformational events. These deposits are invariably structurally controlled.

Mineralisation in this environment is commonly of the fracture and vein type in brittle fracture to ductile dislocation zones. At the Adumbi deposit, the gold mineralisation is generally associated with quartz and quartz-carbonate-pyrite ± pyrrhotite ± arsenopyrite veins in a BIF horizon.

Examples of similar type gold deposits to Adumbi include Geita in Tanzania, Kibali in northeastern DRC, Tasiast in Mauritania, Homestake (U.S.A.), Lupin (Canada) and Moro Velho in Brazil.

9 EXPLORATION

This section includes a summary of the exploration work completed within the Imbo licence area during the 2020/21 exploration phase. The past exploration activity on the Imbo Project was originally summarised in the RPA NI 43-101 technical report entitled "Technical Report on the Somituri Project, Imbo Licence, Democratic Republic of the Congo" and dated February 28, 2014 (available from SEDAR at www.sedar.com).

9.1 SUMMARY OF PRE-2014 EXPLORATION

Kilo's main objectives for conducting exploration on the Imbo Project were to

- Enhance the understanding of the extent and style of the mineralisation in order to successfully conduct diamond drilling, leading to Mineral Resource estimates for Adumbi, Manzako, and Kitenge.
- Optimise the deposit models and exploration strategies to be applied in delineating other potential deposits within the Imbo Project.

Initial exploration in the Imbo Project in 2010 concentrated on the Adumbi deposit. The exploration techniques employed included soil sampling, geological mapping, and sampling of existing adits, trenching, and diamond drilling. Localities of historical and active artisanal mining operations provided guidance for the initial exploration activities.

9.1.1 Soil Sampling

A total of 9,246 soil samples (including quality assurance/quality control (QA/QC) samples) were collected over an area of 63 km², covering the Kitenge, Manzako, Canal, Vatican, Monde Arabe, and Adumbi deposits and prospects (see Table 9.1). Sample spacing over the Manzako deposit was 20 m × 80 m, and elsewhere it was 320 m × 20 m, with some infills at 160 m × 20 m. All soil samples were collected at a vertical depth of 1 m.

Table 9.1: Summary of Soil Sampling by Kilo on the Imbo Project

Year	No. of Soil Samples
2010	1,230
2011	3,282
2012	4,206
2013	528
Total	**9,246**

Analytical Solutions Ltd (ASL) compiled a report on the soil geochemistry of the Imbo Project in October 2013 and concluded as follows:

- Multi-element data mirrors the lithological interpretation based on the airborne magnetic and radiometric survey.
- There is limited mechanical or chemical dispersion of the medium sampled.

- Six gold anomalous areas were delineated underlain by metavolcanic rocks and void of historical or artisanal exploitation.
- Two gold anomalous areas were delineated underlain by metasedimentary rocks (and possibly some iron formation rocks) that warrant follow-up exploration.
- Elements usually considered “immobile” are reasonably well digested by aqua regia in deeply weathered soils allowing reliable lithological interpretation.

9.1.2 Geological Mapping

Geological mapping in 2010 was focused on areas of historical gold exploitation and active artisanal mining activities. Approximately 8.4 km² covering the Adumbi, Kitenge, Manzako, Adumbi North and the Vatican Prospects was mapped.

Lithological contacts and shear zones within the metasediments at Adumbi, as well as exposure of weathered or oxidised BIF and chert units on the top of Adumbi Hill, were mapped.

There was limited outcrop at Kitenge; nonetheless, multiple quartz veins within the Kitenge shear zone were mapped.

Mapping at Manzako identified a northwest-southeast trending shear zone (over 2 km strike length) hosting a number of existing adits and narrow open pits trending parallel to the strike direction of the shear zone.

Mapping at Bagbaie, Vatican and Monde Arabe identified a northwest-southeast trending quartz vein hosted shear zone with artisanal workings.

9.1.3 Trenching

Trenching was undertaken in order to evaluate near-surface gold mineralisation and to provide lithological information to determine the strike extent of the mineralisation and gold-bearing host rocks.

In all, 44 trenches totalling 4,753 m were excavated over the Adumbi, Kitenge and Manzako deposits from 2010 through to 2012. This comprised 23 trenches for 2,745 m at Adumbi, 6 trenches for 878 m at Kitenge and 15 trenches for 1,130 m at Manzako. Table 9.2 summarises some significant trench intercepts at Adumbi, Kitenge and Manzako.

Table 9.2: Summary of Significant Trench Intercepts at Adumbi, Kitenge and Manzako

Trench ID	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
SATR002	23.95	24.95	1	1.50
SATR004	0	13.5	13.5	1.18
	15	20.3	5.3	1.64
SATR005	73.3	79.2	5.9	2.06
SATR006	0	3	3	1.18
	4.9	15.8	10.9	0.96
	29.1	43.5	14.4	2.17

Trench ID	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
SATR007	3.3	8.8	5.5	5.15
SATR008	0	7.5	7.5	1.87
	59.5	63.5	4	1.38
SATR009	25.6	29	3.4	0.91
SATR010	7.7	12.7	5	1.03
	21.4	30.2	8.8	1.86
SATR013	26.1	38	11.9	1.64
SATR014	64.3	66.9	2.6	1.59
SATR015	21.8	25.8	4	1.48
SATR017	40.6	45	4.4	1.65
SATR018	10.4	13.1	2.7	4.02
	63.3	68.7	5.4	0.98

9.1.4 Underground Exploration

Accessible adits and underground workings were geologically mapped and sampled at Adumbi; however, those at Kitenge and Manzako were not readily accessible.

In 2010, Kilo geologists sampled four historical adits at Adumbi totalling 609 m and generated 549 horizontal channel samples (including QA/QC samples).

In 2012, a Kilo contract geologist mapped and sampled three additional adits and two crosscuts at Adumbi. He also mapped the four adits sampled in 2010 and other mine workings where accessible.

In all, a total of 907 m was sampled to generate 843 channel samples. Significant underground sample results at Adumbi are presented in Table 9.3. None of the other historical underground mine workings on the Imbo Project were geologically mapped or sampled by Kilo.

Table 9.3: Significant Underground Sample Results at Adumbi

Adit ID	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
SAAD001	101	109	8	2.63
	113	154	41	1.31
SAAD002	97.5	107.5	10	2.06
SAAD003	155.5	159.5	4	1.66
SAAD006	29	31	2	2.12
	111	114	3	2.37
	119	123	4	2.47

9.1.5 Airborne Geophysical Survey

Kilo contracted New Resolution Geophysics (NRG) from South Africa to complete a high-resolution, helicopter-mounted, XPlorer magnetic and radiometric survey for the Imbo Project. The survey was conducted from April 12 to 15, 2012, over 1,416 km at a line spacing of 100 m by 1,000 m orientated at 040° to 220°. NRG produced plots of the following:

- Total field gradient enhanced magnetics
- First vertical derivative magnetics
- Reduced to pole magnetics
- Analytic signal
- Four-channel normalised singular value deviation (NASVD) processed radiometric data (total count, potassium, uranium and thorium)
- Calculated digital terrain

The magnetic survey delineated a number of linear anomalies characterised by demagnetisation. In addition, a BIF was delineated over a strike length of 2 km from the demarcated northwestern limit of the Adumbi-Canal gold deposit. The total field and radiometric data was utilised by Kilo in the compilation of the structural and lithological interpretation for the Imbo Project.

9.2 POST-2014 TO 2020 EXPLORATION

Following the Inferred Mineral Resource of 1.675 Moz of gold outlined in February 2014 by RPA on three separate deposits, Adumbi, Kitenge and Manzako (see Figure 7.2), RPA made a number of recommendations on Adumbi, which were subsequently undertaken during the period 2014 to 2018. The following subsections outline the work carried out during the period.

9.2.1 Soil Sampling

In 2017, a soil sampling programme (area of 1.5 km × 5 km, on a 40 m × 160 m grid) was planned east of the Imbo River with the objective of further investigating the bulk leach extractable gold (BLEG) and rock chip anomalies identified in 2015. This however was not carried out as planned due to security concerns in the area.

In April 2020, soil sampling commenced in the Imbo East Prospect.

9.2.2 Regional BLEG Survey

A BLEG survey was carried out over the Imbo Project between March and June 2015. BLEG sampling is a regional geochemical technique involving the analysis of stream sediments with the objective of defining areas of gold anomalism for more detailed follow-up. It has the advantage of reliably assessing large tracts of ground relatively quickly and cost effectively.

The main objective of the programme by Kilo was to assess the parts of the Imbo Project not covered by grid mapping and soil geochemistry, in particular the area to the east of the Imbo River where no groundwork has been carried out. However, in order to compare results in these areas with zones of known mineralisation, the whole of the licence area was covered (see Figure 9.1).

The survey was conducted in two stages, Phases 1 and 2, covering the areas to the west and east of the Imbo River, respectively.

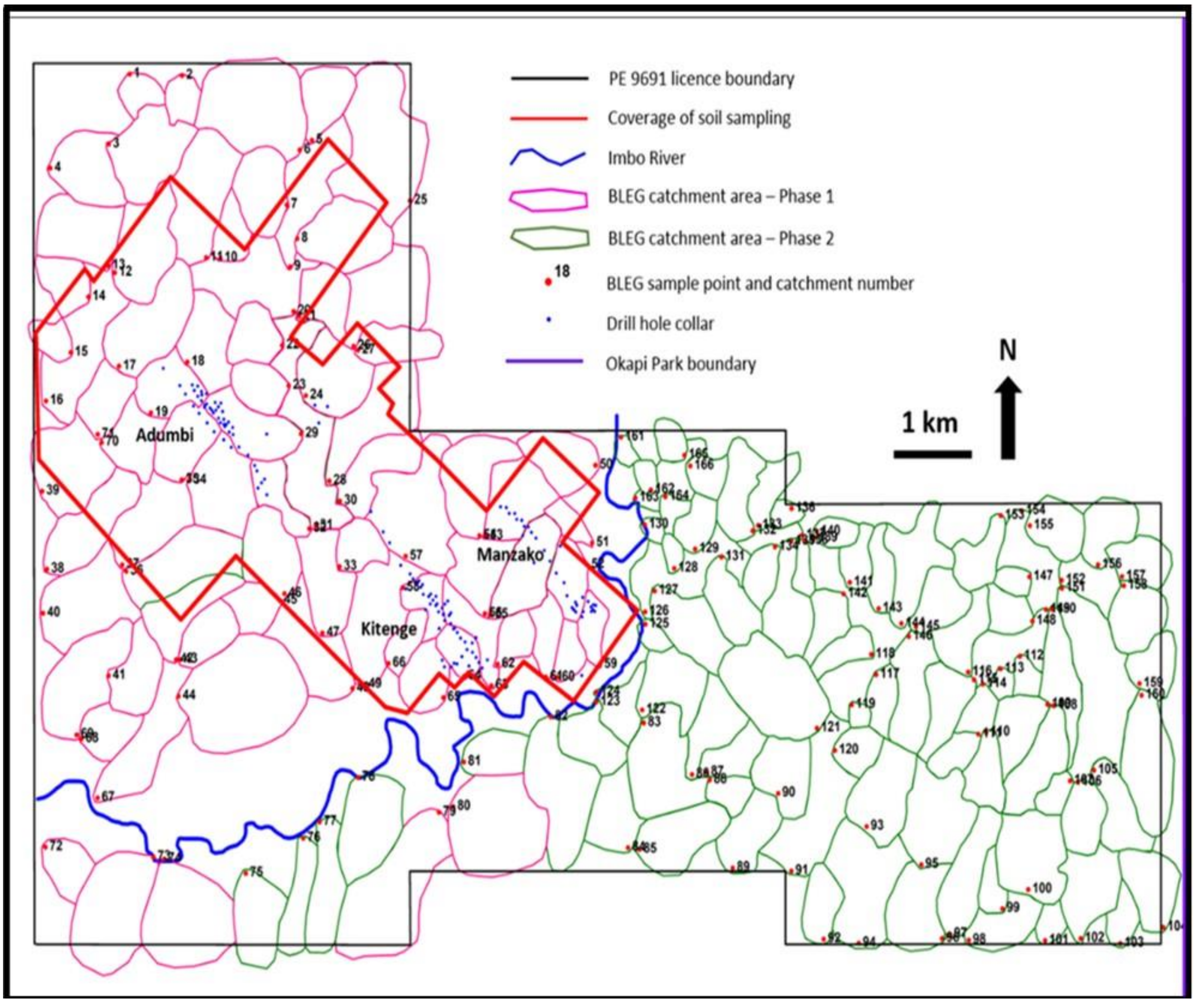
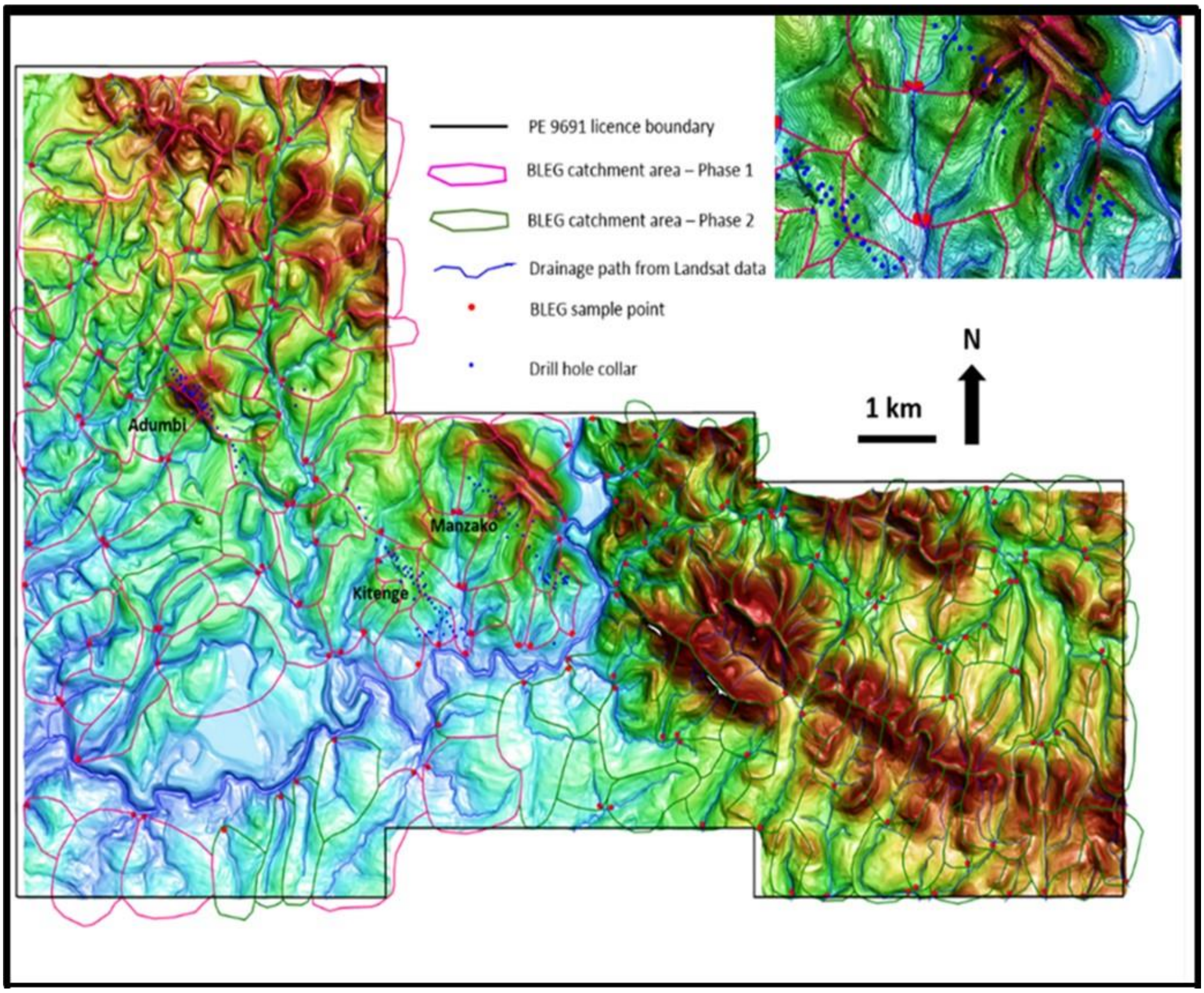


Figure 9.1: Location of BLEG Catchment Areas and Sampling Sites on the Imbo Project

9.2.2.1 Sample Selection

The drainages, catchment boundaries and sampling sites were delineated in Target® using a 5 m colour elevation image and hydrography vector map produced from Landsat data by Photosat in Toronto (see Figure 9.2). A 2 m topographic contour map, also generated by Photosat, was used where necessary (see Figure 9.2, insert).

A total of 166 drainage catchments were defined with a total area of 113 km², resulting in an average catchment size of 0.68 km². Universal Transverse Mercator (UTM) coordinates for the selected sample sites were derived from Target® and transferred to the hand-held GPS instruments used by the sampling teams.



NOTE: Insert shows detail and 2 m contours.

Figure 9.2: Imbo Project – Location of BLEG Catchment Areas, Sampling Points and Drainage Channels on the 5 m Image

9.2.2.2 Sampling Procedure and Sample Preparation

Phases 1 and 2 were carried out by two sampling teams, each consisting of a geologist accompanied by a field assistant and four labourers. The Phase 1 sampling sites were all accessed from the Adumbi base camp while four fly camps were established east of the Imbo River to facilitate Phase 2.

BLEG samples were collected according to the protocol detailed below:

- The sampling teams navigated to each site by handheld GPS.
- At the sampling site, the geologist recorded the characteristics of the stream and alluvial material, any sources of contamination such as artisanal workings and settlements, and mapped/sampled any outcrop in the vicinity. All the data was transferred to an electronic database in the base camp.

- Using plastic scoops, approximately 200 g of the finest sediment fraction (mud) was collected from the top of the stream bed, at about 15 places along the stream, within 20 m of the planned site.
- The material collected was transferred into a single plastic bucket, the bucket was filled with water, and the contents were swirled and allowed to stand for 15 s.
- The mud suspension was then passed through a 1 mm nylon mesh into a second plastic bucket to remove organic debris, leaving any sand and silt as a residue in the bottom of the first bucket.
- Pre-prepared Magnafloc® solution was then gradually added to the mud suspension until flocculation of the mud could be seen.
- After allowing the flocculated mud to stand for several minutes, excess water was decanted from it.
- The flocculated mud slurry was then poured into a pre-marked calico bag, allowing most of the remaining water to drain through the bag.
- As and when necessary, the calico bag was gently squeezed to further reduce the water content.
- The weight of the wet sample was recorded; a minimum of 3 kg is required to provide 1 kg of dry sample.
- Field duplicates were collected at every fifth sampling site. The 33 field duplicate samples were collected in exactly the same way as the original samples, from the same stretch of stream, and given independent sample numbers.
- Back at the camp, the samples were air dried for several days, with frequent agitation by hand to prevent caking.
- Final drying to remove any remaining moisture was done by placing samples in the laboratory oven for 12 h at 80 °C.
- Final disaggregation of the clay particles was carried out by gently rolling with a bottle.
- 1 kg of each sample was weighed and transferred into marked geochemical sample packets and sealed in plastic bags for despatch. Standards (1 per 50 samples) and blanks (2 per 50 samples), gaps for which had been left in the sampling sequence, were inserted at this stage.

At these localities, standard stream sediment samples were also taken, for comparison with the BLEG data. A total of 166 BLEG samples were collected for both Phases 1 and 2, in addition to 33 field duplicate samples. In addition, 33 stream sediments plus 33 field duplicates were collected during the exercise.

The original and duplicate BLEG samples were assayed as follows:

- No additional sample preparation was required.
- Au, Ag, Cu and Pd by cyanide leach bottle roll on 1 kg, with reporting limits for Au of 1 ppb to 10,000 ppb (Method Au-CN12).
- A suite of 53 elements by aqua regia digestion of the 0.5 g of sample, and analysis by ICP-MS and ICP-AES (Method ME-MS41L).

The original and duplicate stream sediment samples were dried and disaggregated at the camp, and were submitted to the laboratory for analysis as follows:

- Sieve to −180 micron (80 mesh).
- Conduct a fire assay of the −180 micron (80 mesh) fraction for Au, using a 50 g charge (Method Au-AA24).
- Conduct a test for a suite of 53 elements by aqua regia digestion of the 0.5 g of sample, and analysis by ICP-MS and ICP-AES (Method ME-MS41L).

A summary of the sample types, number of samples, and analytical methods is given in Table 9.4.

Table 9.4: Summary of Sample Types and Analytical Methods, Phases 1 and 2

Sample Type	No. of Samples			Analytical Methods
	Phase 1	Phase 2	Total	
BLEG	76	90	166	Bottle Roll (Au) Multi-Element ICP
BLEG – Field Duplicate	15	18	33	Bottle Roll (Au) Multi-Element ICP
Stream Sediment	15	18	33	Sieve to −80 mm mesh Fire Assay (Au) Multi-Element ICP
Stream Sediment – Field Duplicate	15	18	33	Sieve to −80 mesh Fire Assay (Au) Multi-Element ICP

9.2.2.3 BLEG Sampling Results

All the BLEG sample results (Au bottle roll) are as plotted in Figure 9.3. The map illustrates the spatial distribution of the individual gold values.

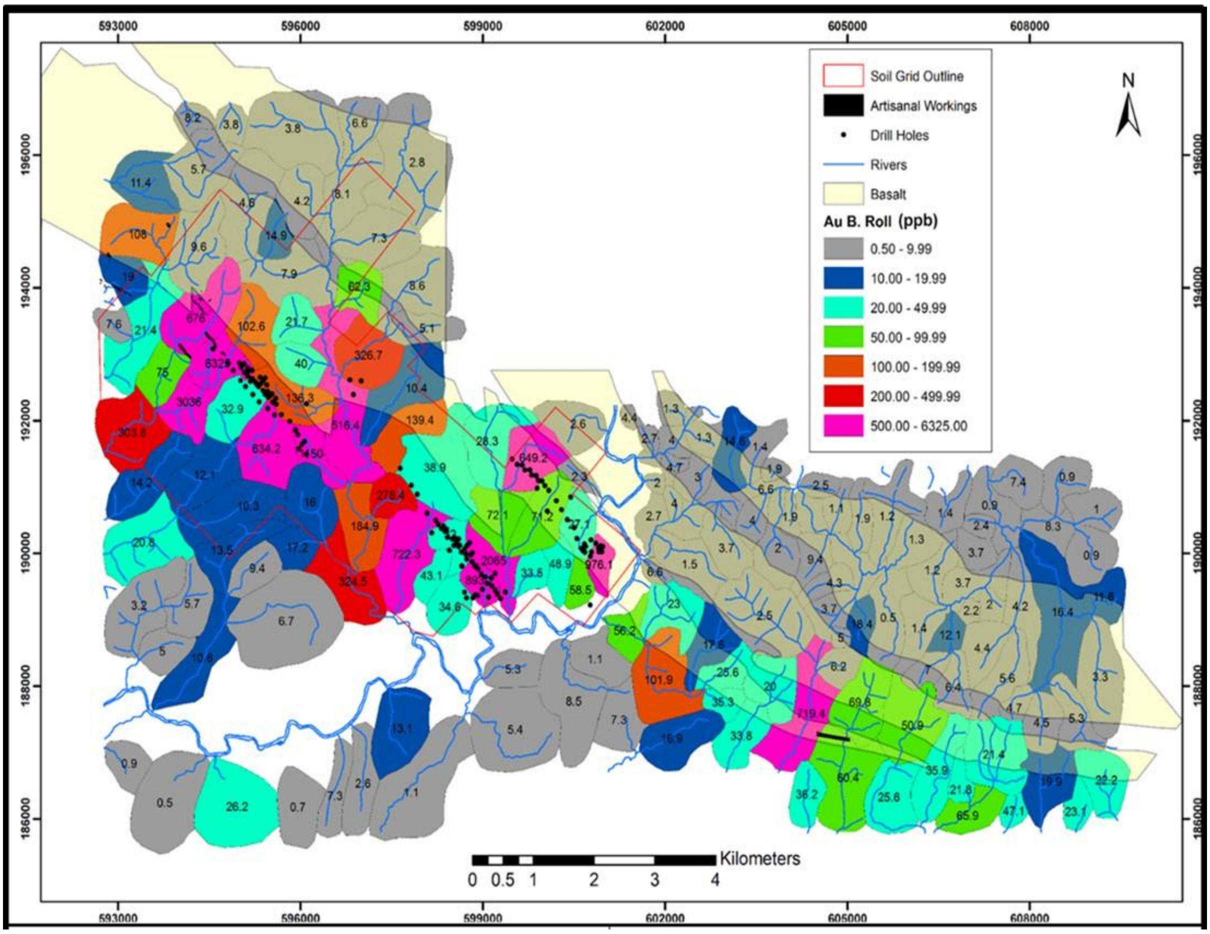


Figure 9.3: Phase 1 and 2 BLEG Results for Au showing Catchments Recommended for Follow-Up

9.2.2.3.1 Gold

The gold values for all the catchment areas are shown in Figure 9.3, which highlights the annotation of the anomalies. The following observations could be made:

- A close spatial relationship exists between the catchments with higher gold values and the known mineralisation at the Adumbi, Kitenge, Manzako and Monde Arabe prospects. It should be noted, however, that mining during colonial times, followed by intense artisanal activity over several decades, has probably increased the amount of gold released into the associated drainages. It should not be assumed, therefore, that lower-order anomalies elsewhere are not significant in terms of mineralisation potential.
- In the Phase 1 area, anomalous values of 62 ppb Au and 108 ppb Au were returned for Catchments 21 and 13, respectively. These catchments are not completely covered by the current soil sampling grid and are recommended for follow-up work.
- Catchment 48 returned a value of 324 ppb Au, significantly higher than the sample from the catchment upstream (185 ppb Au), which probably represents downstream

distribution of gold from the Canal and Vatican prospects. Additional work in Catchment 48 is recommended.

- Other catchments in the Phase 1 area, to the north and south of the current soil sampling grid, have a low gold mineralisation potential, and no further work is recommended in these areas.
- In the Phase 2 area, the gold data clearly indicates a southeastern extension of the Adumbi/Kitenge/Manzako mineralised zone, over a strike of at least 7 km. Anomalous values in this area range from 51 ppb Au to 719 ppb Au, the highest value occurring in a catchment in the Esio area immediately northwest of several colonial adits.
- Catchments in the northern part of the Phase 2 area generally returned background gold values, although weakly anomalous values of 12 ppb Au to 18 ppb Au occurred in some areas associated with alluvial diggings and a rock chip sample of BIF grading 1.69 g/t Au. Mineralisation in this area seems to be less well developed and more sporadic than in the zone to the south, and it is recommended that follow-up work should be concentrated in the southern zone at this stage.

A comparison of the BLEG and stream sediment samples indicates that, for samples with > 50 ppb Au, both methods provide similar results. However, for samples with < 50 ppb Au, the BLEG samples provide more consistent data, with less analytical scatter. The multi-element ICP data for original and field duplicates shows good correlations for both methods. However, correlation coefficients are slightly higher for the BLEG samples, indicating a lower nugget effect. It is therefore recommended that, for future regional surveys, BLEG sampling should be employed with gold analysis by bottle roll, rather than stream sediment sampling with gold analysis by fire assay.

The analytical results of the standards, blanks and field duplicates conclude that (a) the sampling method successfully produced representative samples with a low nugget effect and very good repeatability, and (b) the laboratory produced accurate and precise results, with no significant analytical error or bias.

9.2.2.3.2 Multi-Elements

In all, 52 elements were analysed in addition to gold and can be classified into the following groups: (a) elements associated with gold mineralisation, (b) elements preferentially associated with the metasedimentary terrain, (c) elements preferentially associated with the metavolcanic terrain, and (d) elements with no apparent association. This grouping is summarised in Table 9.5.

Table 9.5: Association of Elements in the Phase 1 and 2 BLEG Survey Areas

Association	Elements
Gold Mineralisation	Ag, As, Bi (weak), Hf (weak), Hg, Pb (weak), Th (weak), W (weak), Zr (weak)
Metasedimentary Terrain	Ce, Cs, K, La, Mo, Rb, Se, Sr, Ti, U
Metavolcanic Terrain	Al, Ca, Co, Cr, Cu, Fe, Ga, In, Li, Mg, Mn, Ni, P, Sb, Sc, Ti, V, Y, Zn
No Apparent Association	Ba, Be, Cd, Ge, Na, Nb, Pd, S, Sn, Te

9.2.2.4 Conclusion

To the west of the Imbo River, outside the known mineralisation in the Adumbi, Kitenge and Manzako areas, the most significant Au anomalies are as follows:

- Catchment 13 (108 ppb Au, 346 ppm As) located 3 km NW of Adumbi
- Catchment 21 (62 ppb Au, 790 ppm As) located 2 km NE of Adumbi
- Catchment 48 (324 ppb Au, 234 ppm As) located 3.5 km S of Adumbi

To the east of the Imbo River, anomalous Au values occur in a zone trending NW-SE over a strike of 7 km, which appears to be the strike extension of the Adumbi/Kitenge/Manzako mineralised trend. Maximum Au and As values for the BLEG samples are 719 ppb and 140 ppm, respectively. The anomalous zone covers an area of colonial and artisanal mining activity, with rock chip samples taken during the BLEG survey grading up to 15.1 g/t.

The current survey has enabled the Imbo Project to be geochemically sampled reliably, quickly, and cost effectively. It has been of particular importance in assessing the mineralisation potential of areas not previously explored on the ground, i.e. outside the soil grid to the west of the Imbo River, and the whole area east of the Imbo River.

The data quality and effectiveness of the BLEG technique are supported by the multi-element results, which correlate well with the distribution of metavolcanic and metasedimentary rocks, interpreted from the geophysical data.

9.2.2.5 Recommendation

It is recommended that

- Follow-up exploration should be prioritised in the zone of anomalous BLEG samples in the southern part of the Phase 2 block, commencing in the central part near the Esio workings, and extending along strike to the NW and SE.
- Second priority follow-up work should incorporate the three anomalous catchments to the west of the Imbo River, which lie outside the current soil grid. The As value for Catchment 21 is very high at 790 ppm and should be the initial focus.
- Work on the above anomalies should initially comprise soil sampling in areas of residual overburden (or auger drilling where the overburden is suspected to be transported) initially on 160 m spaced lines.
- Similar drainage sampling surveys should be carried out on Adumbi Holdco's other licences in the Ngayu belt. Sampling should be done by the BLEG method with Au analysis by bottle roll, rather than on non-flocculated samples by fire assay.

9.2.3 Geological Mapping

Mapping and channel sampling of workings in the Adumbi, Adumbi West and Adumbi Hill areas were undertaken, and a summary of the work completed is shown in Table 9.6 and Figure 9.4. Mapping was carried out on 50 m spaced lines, and in addition to lithological and structural data, various physical features such as old and active workings, tracks, streams and settlements were captured.

Part of the objective of mapping Adumbi Hill was to be able to correlate the surface geology, workings, adits and other surface information with that known from the drilling and other existing data including recently surveyed adits and workings.

Mapping in the area to the west of Adumbi Hill exposed several abandoned and active workings including trenches, artisanal pits, adits and some outcrops found along cross lines. These features are concentrated around the Mabele Mokonzi area located in the eastern part of the Mambo Bado artisanal camps, the western part of Adumbi Hill, Kananga located to the northeast of the grid, and a small part of Adumbi East Hill. A large riverine swamp being drained by the Adumbi River is the locus of moderate alluvial activity by artisanal miners.

Three zones of BIF were inferred based on a rare outcrop and float, and occur within a sequence of quartz carbonate, carbonaceous and chlorite schists. Quartz veins up to 45 cm wide occur within the schist and are being exploited by artisanal miners. In the vicinity of these veins, the host rocks contain weak to moderate foliation parallel quartz veinlets, patches of limonite, and may also display disseminated crystals of pyrite and boxworks.

Rock chip sampling was also carried out in tandem with the geological mapping exercise. A total of 267 samples were collected for assay.

Table 9.6: Summary of Mapping and Pitting Programmes in the Adumbi and Adumbi West Areas

Month	Activity							Number of Samples			
	Gridding	Trench		Pit		Other Channels					
		Number	Metres	Number	Metres	Number	Metres	Rocks	Trench	Regolith Pit	Other Channels
Mar 2014	0.00	0	0.00	0	0.00	0	0.00	0	0	0	0
Apr 2014	0.00	0	0.00	0	0.00	0	0.00	0	0	0	0
May 2014	0.00	0	0.00	0	0.00	0	0.00	0	0	0	0
Jun 2014	16.00	0	0.00	4	10.10	0	0.00	56	0	14	0
Jul 2014	39.64	0	0.00	0	0.00	0	0.00	55	0	0	0
Aug 2014	6.64	1	206.00	0	0.00	0	0.00	8	32	0	0
Sep 2014	21.20	0	70.60	0	0.00	0	0.00	44	34	0	0
Oct 2014	24.00	0	103.40	0	0.00	0	0.00	56	8	0	0
Nov 2014	24.00	0	0.00	0	12.90	0	0.00	5	0	0	0
Dec 2014	13.00	1	0.00	5	0.00	0	0.00	4	0	20	0
Total 2014	**144.48**	**0**	**380.00**	**9**	**23.00**	**0**	**0.00**	**228**	**74**	**34**	**0**
Jan 2015	7.00	0	0.00	0	0.00	19	143.10	8	0	0	140
Feb 2015	0.00	0	0.00	17	57.45	13	66.30	7	0	73	71
Mar 2015	0.00	0	0.00	26	67.60	4	19.55	14	0	91	26
Apr 2015	0.00	0	0.00	0	0.00	16	86.60	10	0	0	109
Total 2015	**7.00**	**0**	**0.00**	**43**	**125.05**	**52**	**315.55**	**39**	**0**	**164**	**346**
Total 2014 to 2015	**151.48**	**1**	**380**	**52**	**148.05**	**52**	**315.55**	**267**	**74**	**198**	**346**

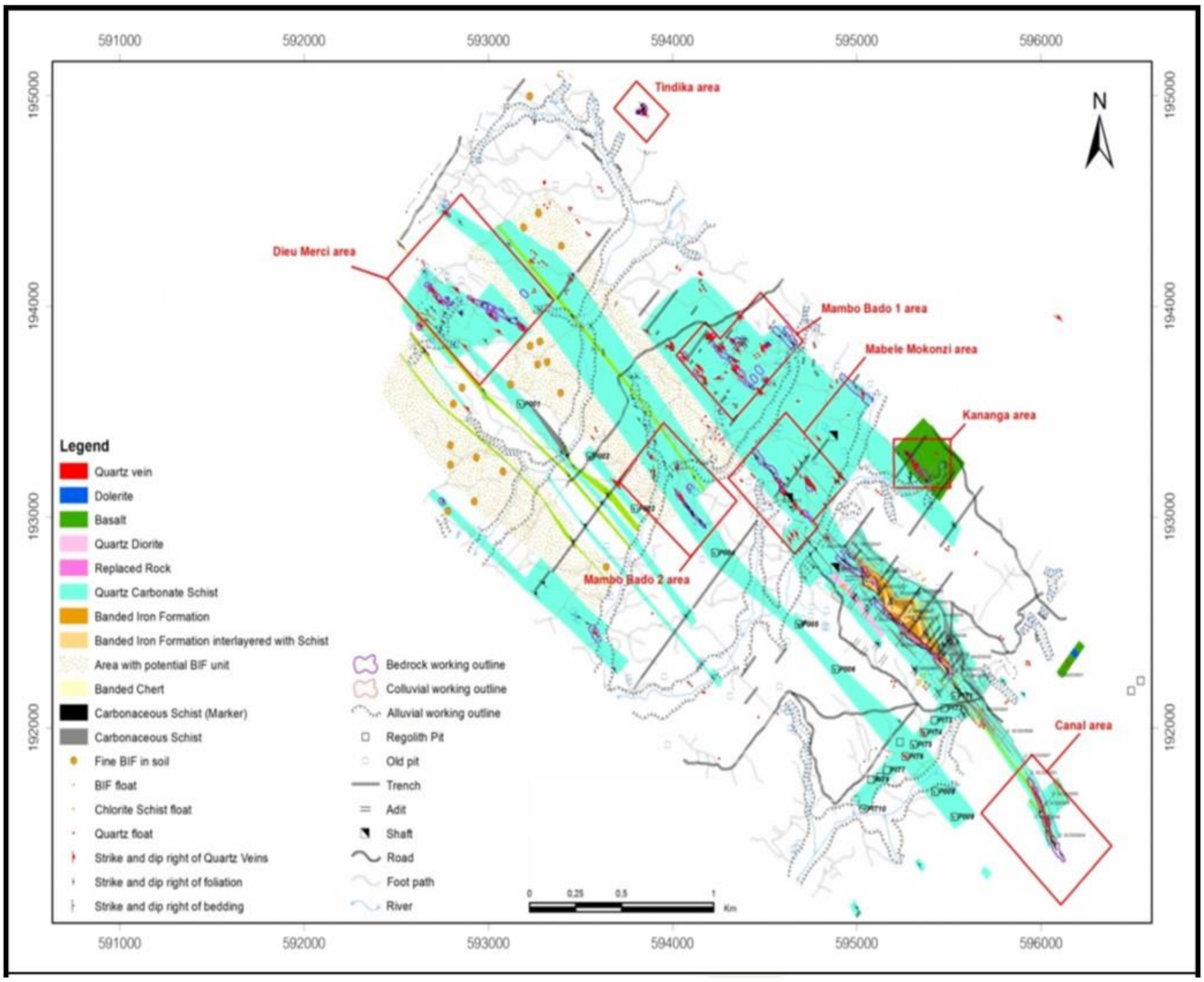


Figure 9.4: Geological Map of Adumbi and Adumbi West Areas showing Artisanal Activities

9.2.4 Trenching

Re-excavation of an 850 m long colonial trench was commenced in August 2014, aimed at exposing lithologies for lithostructural mapping purposes. Selective sampling was also carried out in places where a significant alteration was observed (see Figure 9.5). Trenching was however suspended in September 2014 due to continued sidewall collapse and repeated cleaning and clearing efforts required after heavy rainfalls.

A total 301 m was cleaned/reopened, and 74 samples were collected. Sampling was not carried out where no significant alteration was observed, or where the trench was deemed unsafe.

The main lithologies observed are quartz carbonate schist and chlorite schist, totally oxidised with weak foliation parallel veins of quartz ranging from 0.5 cm to 25 cm wide. The BIF unit targeted was not intersected, and no major altered or sheared zones were encountered prior to suspending the programme. The foliation and quartz veins have an average strike of 310° and dip mostly at approximately 70° to the NE.

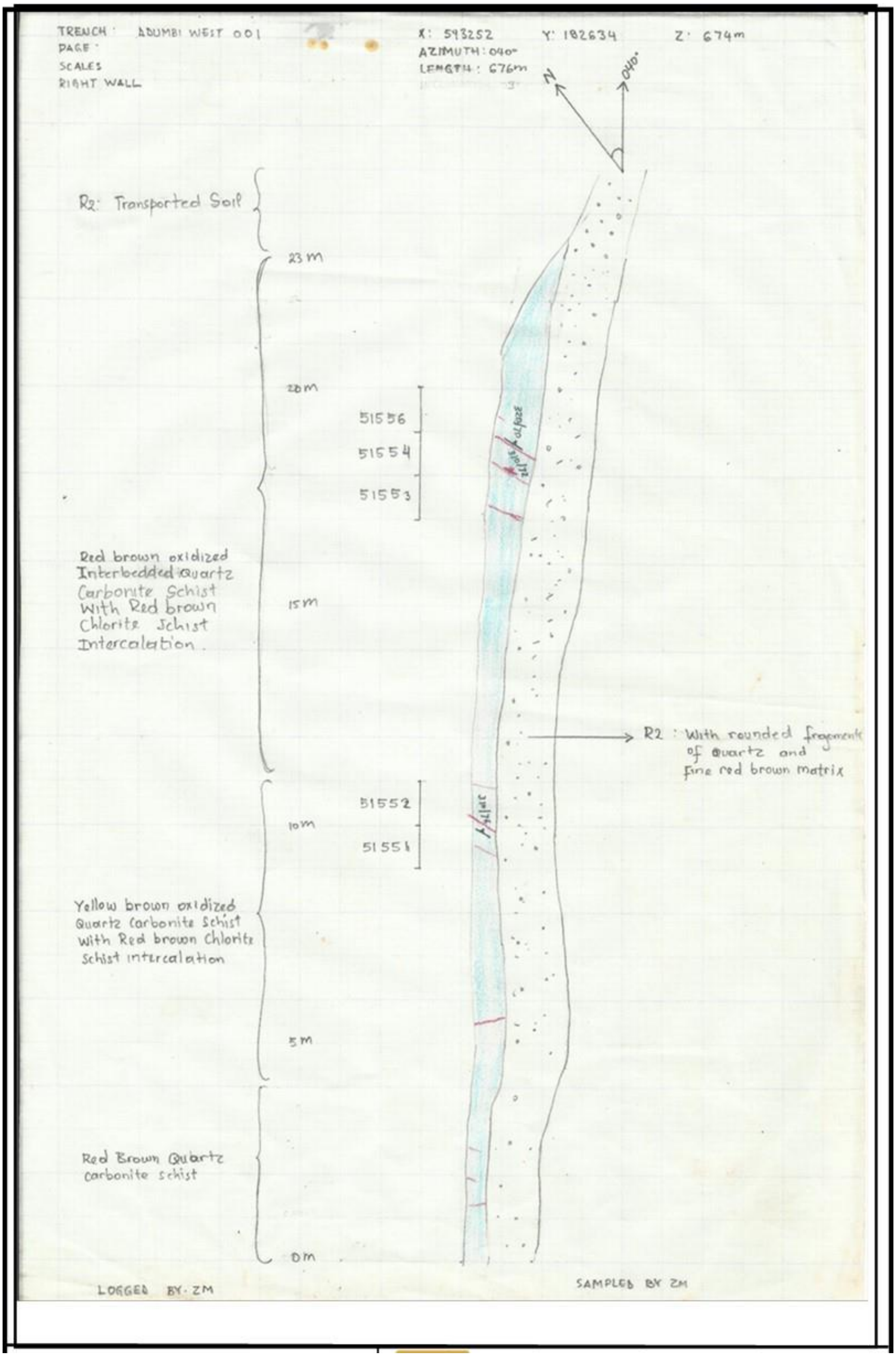


Figure 9.5: Adumbi West Prospect – Trench Mapping and Sampling

9.2.5 Pitting

A total of 52 pits on selected induced polarisation (IP) lines at 80 m intervals were dug in the Adumbi West, Adumbi South, Vatican and Senegal areas. The pits were designed to assist with the interpretation of responses from the underlying soil geochemistry and IP signatures, and to further the understanding of regolith patterns and distribution in these areas and the wider Imbo Project area. All the pits were vertically channelled, with the different regolith horizons and saprolite sampled separately.

The pit logging showed that many of the previous soil samples would have been taken within the transported horizon, despite being sampled at a depth of 1 m. Although the current programme suggests that some of the transported material may be proximal, this is not always the case. The possibility therefore exists that the soil results are locally (a) giving false anomalies, or (b) not detecting the underlying mineralisation.

The pitting programme demonstrated the complexity of the regolith in the Adumbi area and supports the conclusion from the radiometric and ICP data that a large proportion of the area is overlain by transported soil.

9.2.6 Topographical Survey

All the Adumbi drillhole collars, trenches and accessible adits and adit portals were accurately surveyed, and the data appropriately georeferenced. In addition, all the accessible underground excavations and workings were accurately surveyed.

Survey work commenced in late July 2014. Coordinates were based on the existing reference control points, which were corrected and re-fixed by a consulting surveyor from Map Africa, RSA. The three control beacons are located inside the Adumbi base camp and have the final adopted coordinate system as shown in Table 9.7, UTM (Zone 35 North) based on WGS 84.

Table 9.7: Adumbi Prospect Survey Control Points

PID	East-UTM	North-UTM	Elevation	Code
14MRSCM	596523.35	191570.88	649.6	10IPIC
14SCM1	596620.47	191457.32	644.39	10IPIC
14SCM2	596669.84	191500.62	646.41	10IPIC

9.2.6.1 Drill Collar Survey

The drillholes were surveyed by measuring the collar position on the concreted surface as shown in Figure 9.6. All the Adumbi and Canal drillholes (with the exception of abandoned holes) were surveyed, and all the data was saved in Loncor's survey computer.

Figure 9.6: Adumbi Deposit – Survey of Drillhole Collars

The old and new drill collar positions are shown in plan view in Figure 9.7. The following maximum differences are seen between the data sets: X = 11.20 m, Y = 10.90 m, and Z = 52.55 m.

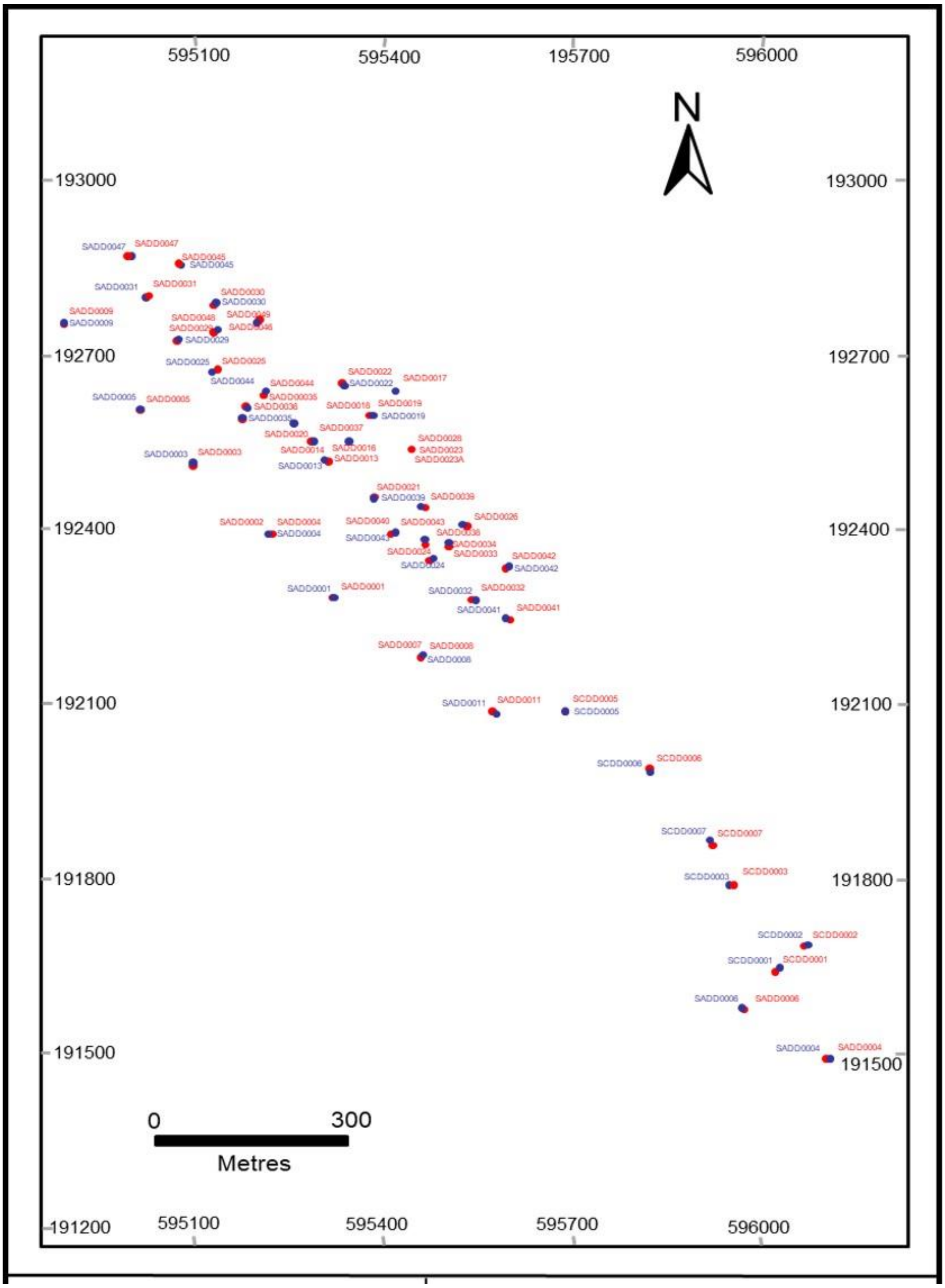


Figure 9.7: Comparison of Drillhole Collar Locations using Old and New Survey

9.2.6.2 Adit Survey

All the known adits in the Adumbi deposit were surveyed by DGPS R10 and total station S3 DR. These included the seven adits which were sampled and used for resource calculations (see Figure 9.8).

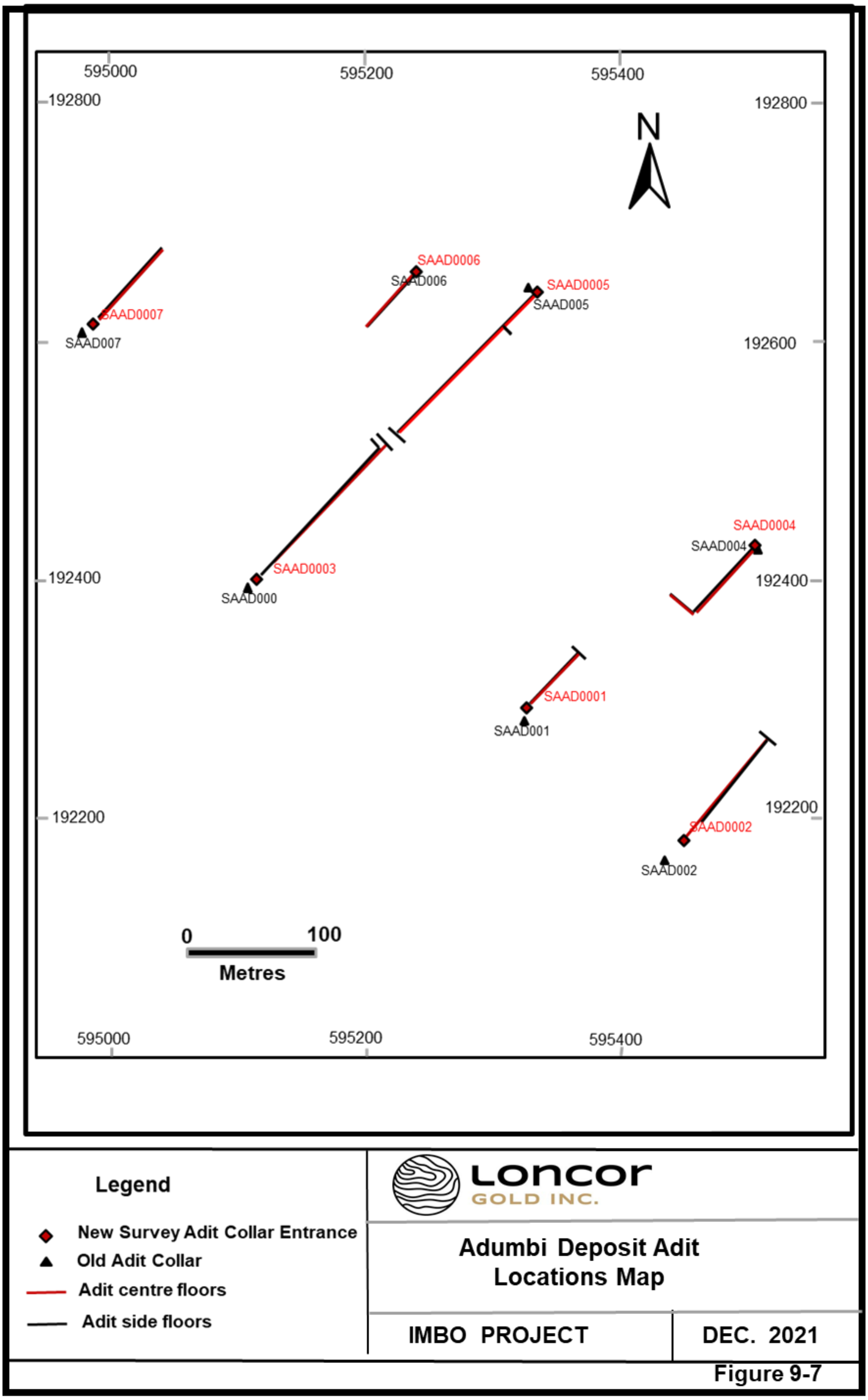


Figure 9.8: Adumbi Deposit – Adit Locations Map

The survey measurements were taken by fixing the entrance (portal) of the adits, followed by surveys inside the adits of the floor, roof, and side walls wherever possible (see Figure 9.9). Intersection points in the adits of crosscuts, reef drives, etc. were also surveyed, in order to aid the georeferencing of the existing underground geological maps.

Figure 9.9: Adumbi Deposit – Adit Surveying

All the final survey coordinates for the Adumbi adits were saved in Loncor's survey computer.

Following the accurate surveying of the 10 historical adits and appropriate georeferencing, the 796 adit samples (1,121 m in total), when applied, should have positive implications on the data spacing and classification of any future mineral resources.

9.2.6.3 Trench Survey

Surveys were carried out by locating the outlines and elevation in order to determine the shape and the original ground surface along the excavated trench. Some trenches however were damaged, either by backfilling or artisanal activities, and therefore made it difficult to accurately determine the original positions.

With the Adumbi drillhole collars, trenches, and accessible adits/portals as well as accessible underground excavations and workings now accurately surveyed and the data appropriately georeferenced, the new and improved quality of the exploration data will have positive implications on potential future classifications of the mineral resources.

9.2.7 Underground Exploration

The only underground exploration activity undertaken during the post-2014 exploration campaign was the surveying and georeferencing of the adits.

9.2.8 Airborne Geophysics Survey

IP and LiDAR (light detection and ranging) were the only geophysical surveys conducted during the post-2014 exploration campaign.

9.2.9 Induced Polarisation (IP) Surveys

An initial pole-dipole (PDP) orientation survey was undertaken over the known mineralisation, the results of which warranted a systematic PDP survey of sections in other prospective areas in order to generate drilling targets, in particular the Adumbi West prospect.

The IP equipment and operators, who were on loan from another company for three months, arrived on site on October 17, 2015, to commence the programme, which was completed on June 16, 2016.

9.2.9.1 Pole-Dipole Methodology

Unlike gradient array surveys, which measure near-surface resistivity and chargeability responses, the PDP method delivers greater depth penetration and cross-sectional data.

The PDP array is conceptually straight forward and works by applying an electric current to the earth using two electrode pots: the moving electrode pot, located 50 m from the starting point, and the infinity pot, which remains stationary and is located 2 km south of the starting point (transmitter). The moving electrode pot moves along the survey line, keeping a distance of 50 m from the infinity pot, and readings at the receiver are taken at 50 m intervals.

The receiver is connected to a series of eleven electrode pots via a multi-conductor electrical cable along the survey line. The transmitter and generator are fixed permanently at a convenient location in the centre of the survey lines. The electrical wires are connected to the transmitter and transmit current to the ground when connected to the electrode pot. The receiver simultaneously records the primary voltage, resistivity, and chargeability of the underlying rock formations.

9.2.9.2 Pole-Dipole Survey

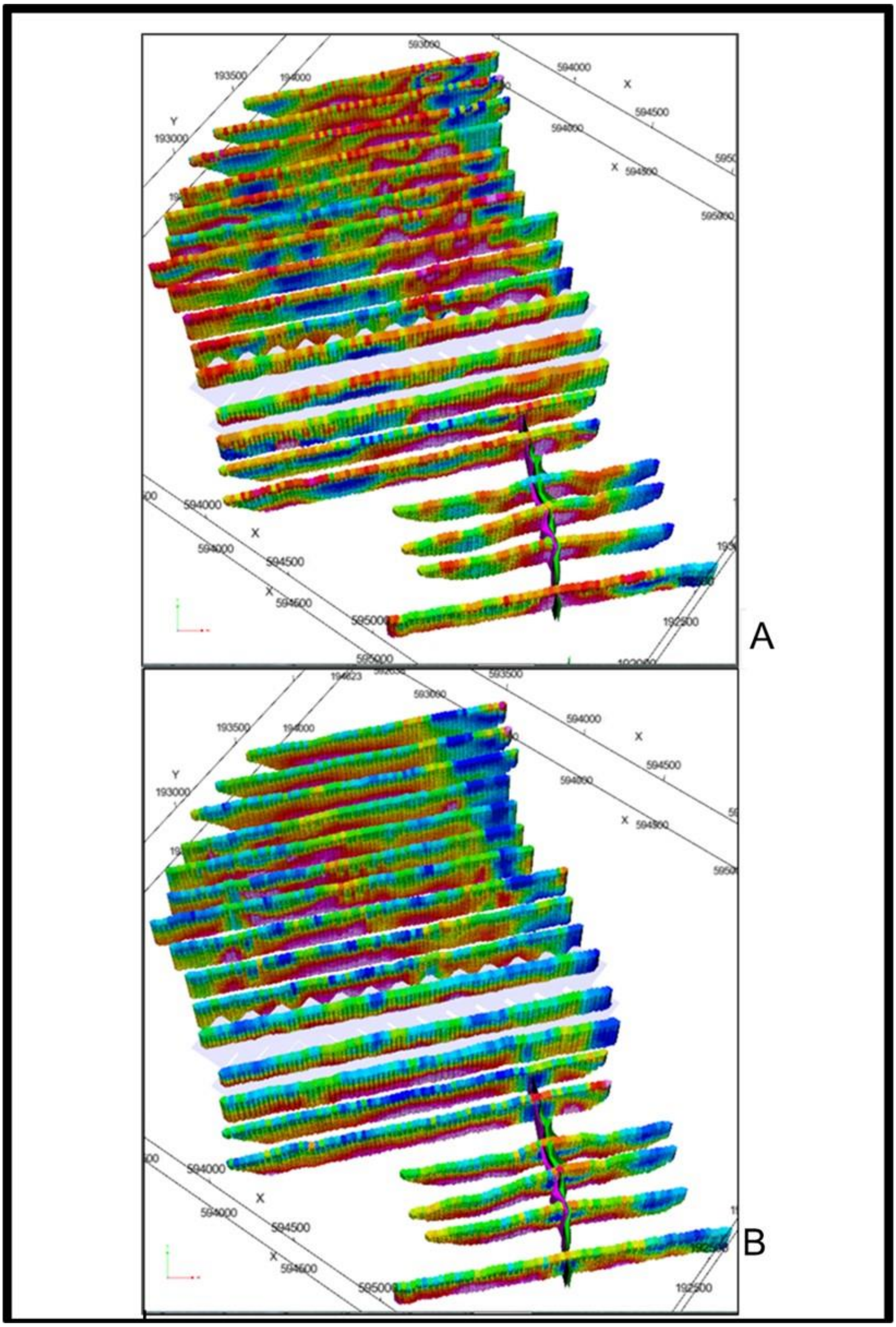
Three lines were selected for survey at Adumbi (AWL02, AEL02, and AEL06). This array covered the central part of the main Adumbi deposit and is considered to be the most representative of the Adumbi styles of mineralisation. In each case, lines were extended to the southwest beyond the known subsurface geology, to cover a broad untested geochemical anomaly striking parallel to the regional trend of approximately 310° to 315°.

9.2.9.3 Pole-Dipole Results

The chargeability and resistivity data is presented in 3D in Figure 9.10). A high-chargeability structure is present in the Adumbi area, and is coincident with the mineralised zone. However, in the Canal and Mabele Mokonzi areas, the mineralisation appears to follow a different structure which is situated in the footwall and hanging wall of the high-chargeability structure, respectively.

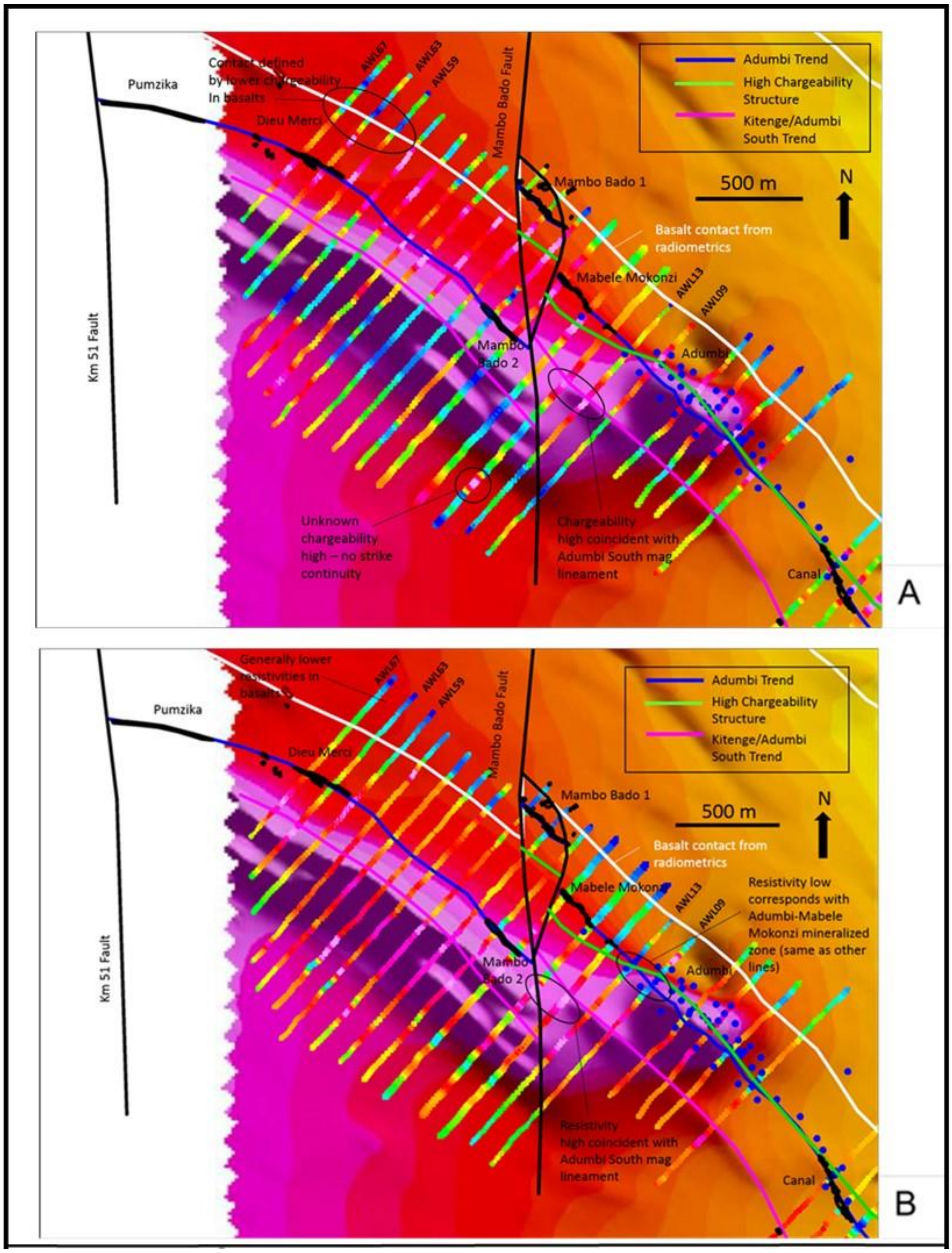
Significant observations from the new data are summarised in Figure 9.11, and include the following:

- The Adumbi mineralised structure is again associated with a resistivity low, a feature noted on all the other lines to the southeast, to the end of the Canal zone.
- There are elevated chargeability values in the interpreted position of the high-chargeability structure, similar in tenor to the other lines from Adumbi to Canal.
- A high-chargeability anomaly is present on Lines AWL13 and AWL 17, which is coincident with the Adumbi South magnetic lineament, interpreted to be a continuation of the Kitenge structure. There is a coincident resistivity high on Line AWL17.
- Extremely high chargeability values occur towards the SW end of Line AWL17. However, there is no trace of elevated chargeability values on strike to the SE on Line AWL13, and the cause of the anomaly is unknown.
- Lines AWL59 to AWL67 confirm the earlier observations that the metabasalt terrain is characterised by lower chargeability and resistivity values than the metasediments.



A Chargeability
B Resistivity
Green Mineralised Zone 2
Purple Mineralised Zone 3
Black Carbonaceous Marker

Figure 9.10: Pole-Dipole Voxels for Adumbi and Adumbi West



A Chargeability
B Resistivity

Figure 9.11: Pole-Dipole Results and the Adumbi, Mabele Mokonzi and Adumbi West Areas, Overlain on the Magnetics (Reduced-to-Pole)

9.2.10 Gradient Array Data

Given the fact that the sectional PDP IP data proved to be very useful in the structural interpretation of the Adumbi area, a gradient array IP was planned in order to provide chargeability and resistivity data in plan view. The gradient array surveys were carried out on 1 km × 1 km blocks, with a 50 m line spacing and a station spacing of 25 m along the lines. The layout of the gradient array grid, transmitter, injection points, receiver and electrodes (pots) is shown in Figure 9.12.

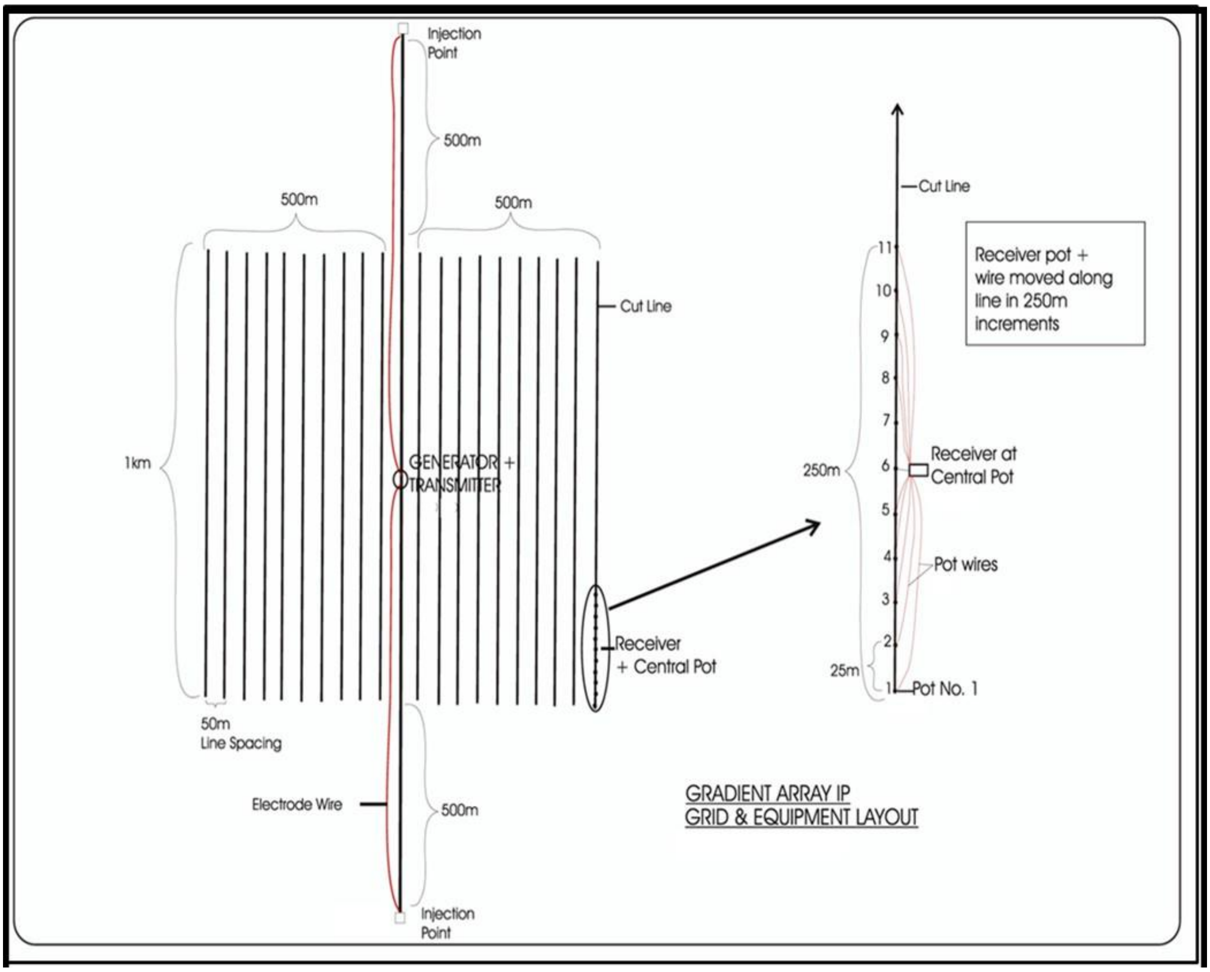
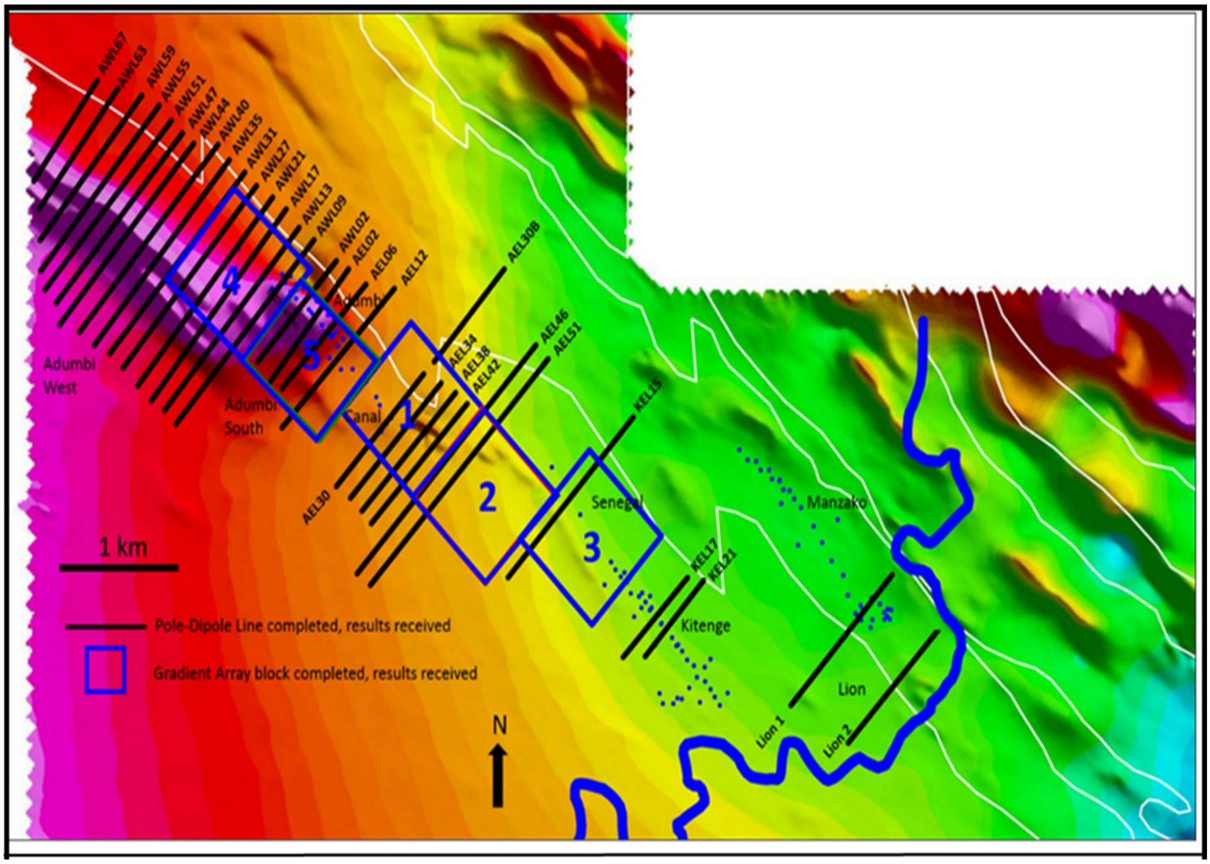


Figure 9.12: Gradient Array IP Layout

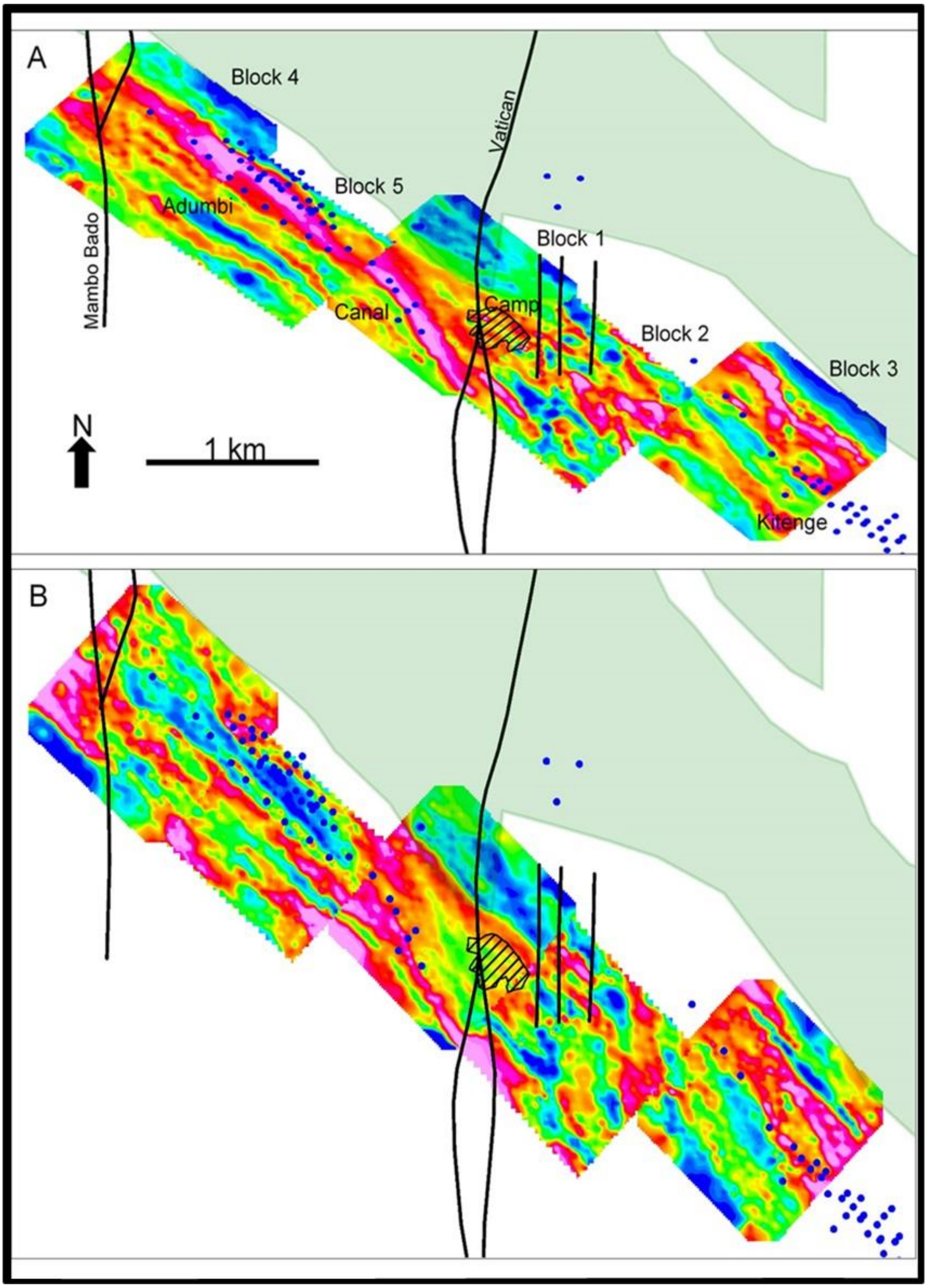
The gradient array (GA) survey was completed, and processed data was received from Spectral Geophysics (see Figure 9.13).



NOTE: Drillhole collars in blue

Figure 9.13: IP Coverage on the Imbo Project

Chargeability and resistivity maps of the GA data are shown in Figure 9.14A and Figure 9.14B, respectively. The chargeability map shows a prominent high associated with the Adumbi and Canal mineralisation, stretching from the Mambo Bado fault in Block 4 to the Vatican fault in Block 1. The continuity of the chargeability high into Block 2 is disrupted by the Vatican fault and its associated splays but is clearly defined in Block 3 in the hanging wall of the Kitenge mineralisation.



A Chargeability
B Resistivity

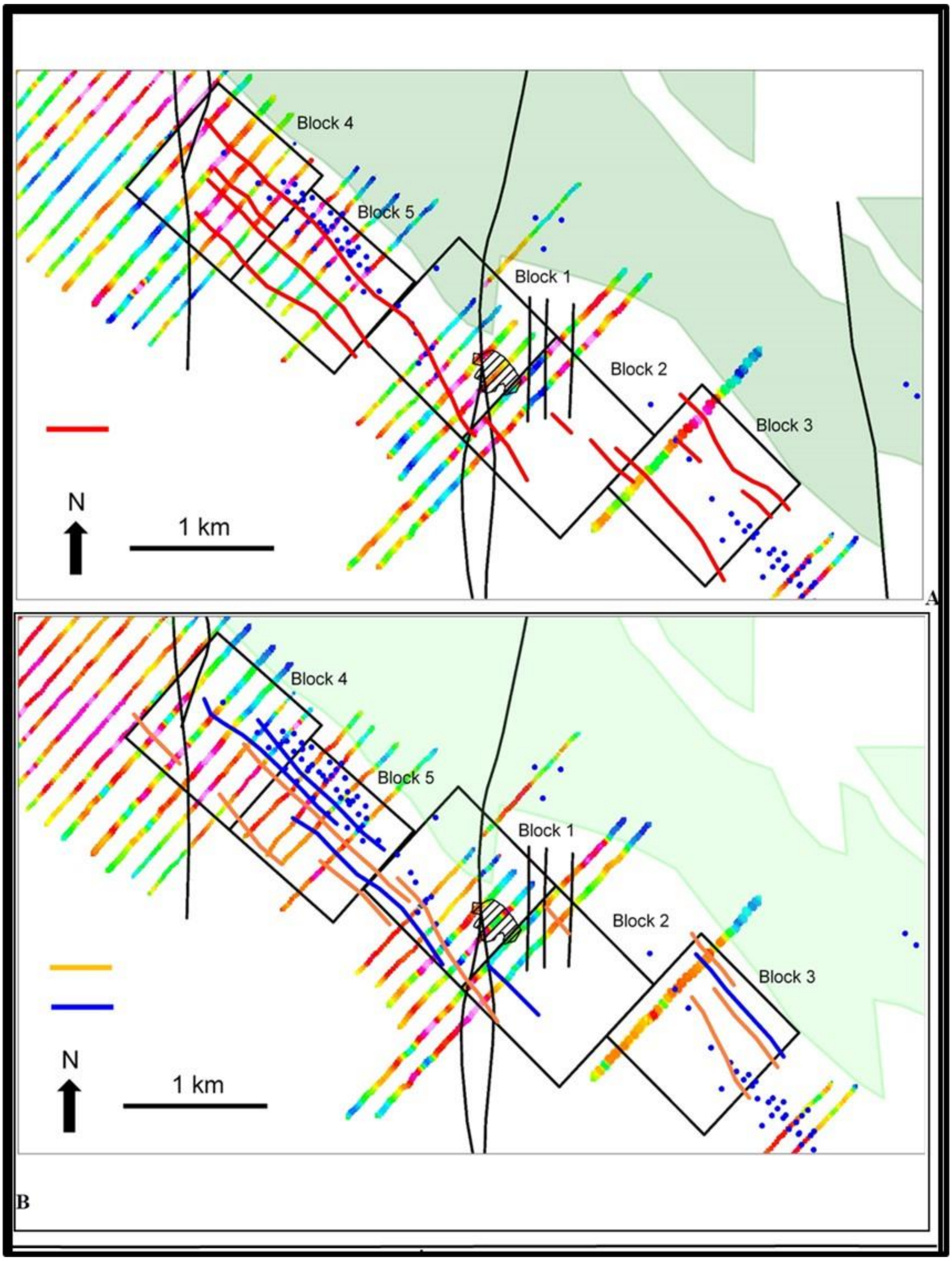
Figure 9.14: Gradient Array IP Maps for the Adumbi-Kitenge Area

The resistivity map shows a low associated with the best-developed section of the Adumbi mineralisation, but unlike the PDP resistivity data, this does not continue southeastwards into Canal. The other patchy resistivity lows (see Figure 9.15B) are not associated with known mineralisation and are probably lithological in origin. A linear resistivity high is present immediately southwest of the Adumbi low, which appears to extend to Canal and continue up to the Vatican fault. If this represents the same continuous zone, it supports the hypothesis that Canal does not represent the direct strike extension of Adumbi. The Kitenge prospect is associated with a GA resistivity high and is possibly the faulted equivalent of the Canal zone.

The main GA chargeability and resistivity features are overlain on the PDP data in Figure 9.15A and Figure 9.15B, respectively. Although there is a broad correlation between the two data sets, there are clear discrepancies. For example, in the Canal area, the chargeability high from the GA diverges from the high defined by the PDP sections, and in the northwest of Block 4, there is a clear displacement between the GA and PDP chargeability highs. For the resistivity data, the most obvious discrepancy is in the Canal area, where the mineralisation is represented by a well-defined low in the PDP sections, but as a relative high on the GA map.

The differences between the two data sets are principally due to the fact that the GA layout measures the IP properties of the rocks at relatively shallow depths below surface (approximately 40 m to 70 m) whereas the PDP array provides a profile of the IP response to a depth of approximately 200 m. In areas of relatively deep weathering, the GA will respond to the shallower saprolite, compared to the deeper parts of the PDP profile where minerals such as sulphides are unoxidised. It is therefore concluded that in moderately to deeply weathered areas with poor exposure, the GA is a useful tool for generating a basic map to assist with the early stages of exploration. The PDP array is more suitable for locating chargeability and resistivity anomalies for drill testing and assisting with the more detailed structural interpretation of the area.

It is recommended that, in future, all IP data be assessed by a geophysical consultant to confirm and expand upon the current in-house interpretations.

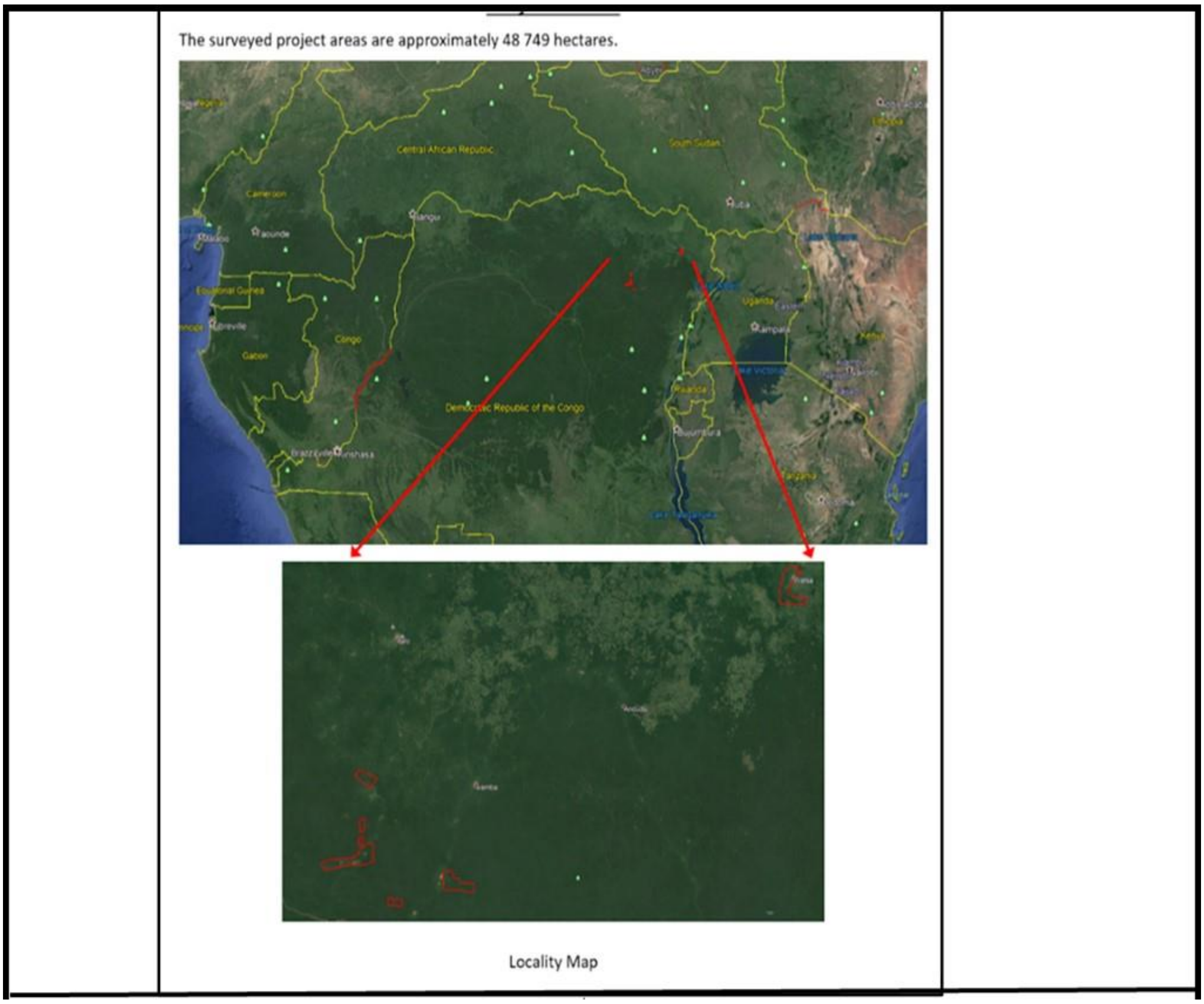


A Chargeability
B Resistivity

Figure 9.15: Gradient Array Anomalies Superimposed on the PDP at the 500 m RL

9.2.11 LiDAR Survey

Per RPA's recommendation, a LiDAR survey was completed over Adumbi by Southern Mapping of South Africa. The survey was carried out from January 17 to January 24, 2020, as part of a large programme covering the Ngayu Kibali areas encompassing the Imbo Project area (see Figure 9.16).



NOTE: The surveyed project areas are approximately 48.749 ha.

Figure 9.16: Imbo Project – Locality Map

The topographical survey was undertaken to produce rectified colour images and a digital terrain model (DTM) of the surveyed project area. The survey was carried out using an aircraft-mounted LiDAR system that scanned the ground below with a 125 kHz laser frequency rate, resulting in a dense DTM of the ground surface and objects above the ground. Digital colour images were also taken from the aircraft and rectified to produce colour orthophotos of the surveyed project area. The survey was flown at a height of approximately 750 m and ortho images with a 7 cm pixel resolution were produced.

The following equipment was used:

- Aircraft: Cessna F406 (ZS-SSY)
- LiDAR Scanner: Optech Orion M300 (12SEN306)
- Camera: iXU RS-1000 Phase One

Ground control points were placed and surveyed by Loncor, and their coordinate values were used for the vertical and horizontal checks on the full aerial LiDAR survey. The coordinate system is in WGS 84 UTM 35N.

The following information was supplied to Loncor following completion of the survey:

- CAD design files in Microstation DGN, DWG and DXF format showing the following:
  - Orthophoto tiles (1,400 m × 1,400 m) and LiDAR point block (1,500 m × 1,500 m) layout
  - Contours at 0.5 m, 1 m and 2 m intervals
  - The surveyed project area with boundaries
  - The contours have been smoothed and are merely an aesthetic representation of the ground shape.

- Ortho-rectified aerial images in ECW (enhanced compression wavelet) format with a 7 cm pixel resolution
- Full LiDAR points in LAS1.4 format with the feature classes shown in Table 9.8
- The LiDAR survey report by Southern Mapping of South Africa

Table 9.8: LiDAR Classification Values

Classification Value	Meaning
1	Unclassified
2	Ground
3	Low Vegetation (0.5 m to 2 m)
4	Medium Vegetation (2 m to 5 m)
5	High Vegetation (> 5 m)

All the above data is in the WGS 84 UTM35N coordinate system, with orthometric heights as calculated in TerraScan using the EGM1996 and EGM2008 geoidal models.

The LiDAR data will be interpreted to aid in structural and regolith mapping.

9.2.12 Relative Density (RD) Measurements

RD measurements on the Adumbi drill core were previously determined by ALS Chemex in Johannesburg and by an analytical laboratory in Vancouver as shown in Figure 9.17; however, major discrepancies existed between the two data sets, and in many cases the reported RD values were very different from what was expected from the drilled lithologies. RPA questioned the reliability of one or both data sets and advised a comprehensive review of the results.

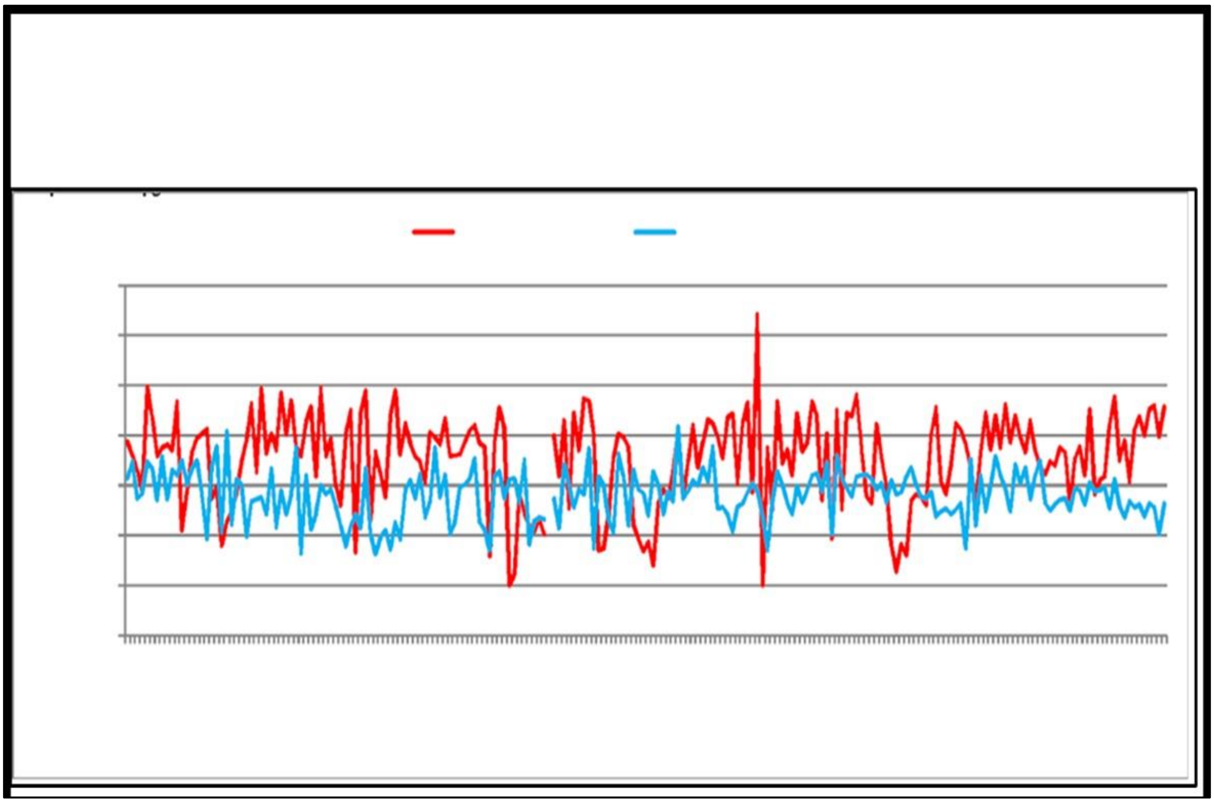


Figure 9.17: Comparison of Relative Densities from Laboratories for Drillhole SADD0019

Given the critical role reliable RD values play in resource estimation and mine planning, it was deemed necessary to carry out systematic measurements on all the Adumbi drill core. All RD measurements were undertaken on site following the summarised procedure below:

- Measure the RD at 1 m intervals in the mineralised zones.
- Measure the RD at 2 m intervals outside the mineralised zones.
- To avoid sampling bias, take the first piece of core after the metre mark weighing > 200 g.
- Dry all the samples completely in an oven, before coating with varnish to prevent water absorption during weighing.
- Take the measurements using an Archimedes balance, using the sample weights in air and water.
- QC procedures involve re-weighing after water immersion to ensure that the varnish coating has been effective, and that no significant absorption of water has taken place. Disregard any measurements where > 1 % water has been absorbed, and repeat the procedure using the next piece of core in the core tray.

A total of 5,385 samples were collected, 25 of which failed the QC criteria due to the fact that they were highly friable and could not be properly sealed with varnish.

The RD programme was thus completed, with a total of 5,360 measurements taken. The average RDs for all the oxide, transition and sulphide zone samples are shown in Table 9.9, and the measurements for mineralised (≥ 0.5 g/t Au) and unmineralised (< 0.5 g/t Au) rock are compared in Table 9.10. The average oxide, transition and sulphide zone RDs for mineralised rock are 2.45, 2.82 and 3.05, respectively.

Table 9.9: Summary of all RD Measurements on Adumbi Core

Type	Total	Pass	Fail	% Fail	RD All	RD Pass	RD Fail
Oxide	1,406	1,384	22	1.56	2.26	2.26	2.38
Transition	829	826	3	0.36	2.59	2.59	2.34
Sulphide	3,150	3,150	0	0	2.91	2.91	–
Total	**5,385**	**5,360**	**25**	**0.46**	**2.69**	**2.69**	**2.38**

Table 9.10: Summary of RD Measurements in Mineralised and Unmineralised Rock

Type	Mineralised		Unmineralised	
	No. of Samples*	RD	No. of Samples*	RD
Oxide	297	2.45	882	2.26
Transition	178	2.82	601	2.54
Sulphide	796	3.05	1,953	2.83
* Excludes samples which were not assayed				

The RD figures used by RPA for their 2014 NI 43-101 report were 1.8, 2.2, and 3.0 for the oxide, transition and sulphide zones, respectively. These were based on readings taken by Kilo staff using a water immersion method (no details provided), but only seven readings were taken in the oxide zone. It is also apparent from the re-logging exercise that the previous determinations of the oxide, transition and sulphide zone boundaries were very inaccurate. As a result, the base of complete oxidation (BOCO) used by RPA is up to 75 m too shallow (see Figure 9.18), which has resulted in an insignificant oxide resource in RPA's estimate for Adumbi (29,000 oz Au).

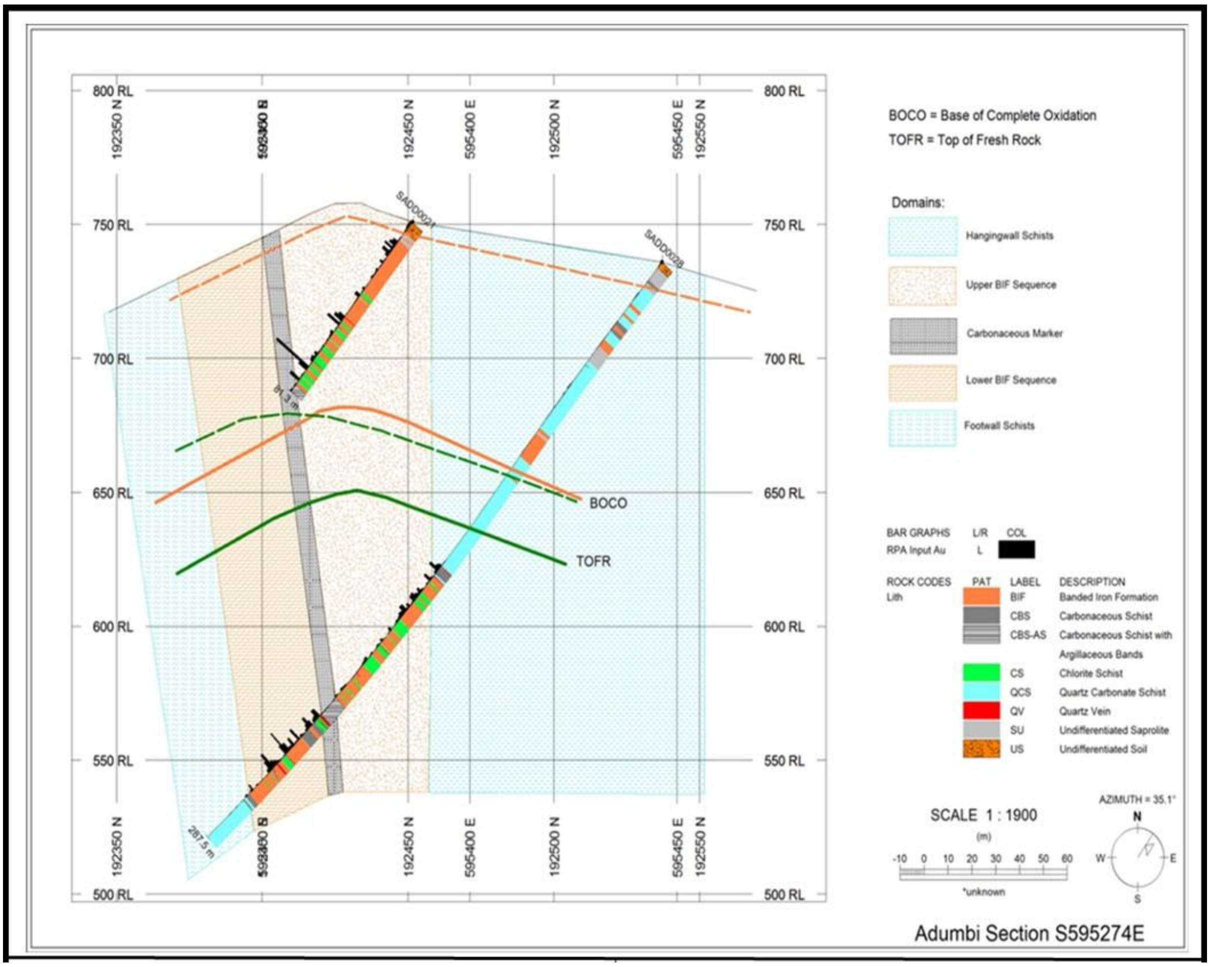


Figure 9.18: Comparison of the RPA Oxidation Levels with the Current Study

The average RD values for mineralised rock are 2.45, 2.82, and 3.05 for the oxide, transition and fresh material, respectively. The large differences between these figures and those used by RPA (i.e. 1.8, 2.2, and 3.0) are mainly due to the fact that (a) only seven oxide samples were previously used to derive the average RD for the oxide zone, and (b) the previous logging of the oxide and fresh rock boundaries were very inaccurate.

The values of 2.45 and 2.90 are relatively high compared to saprolite and saprock in general (see Table 9.11). However, the mineralisation at Adumbi is mostly in the BIF, which when oxidised consists of iron oxides interbanded with unweathered chert, rather than the leached, clay-rich assemblage of typical saprolite.

Table 9.11: Average RDs for the Different Lithologies at Adumbi

Lithology	Logging Code	Oxide RD			Transition RD			Sulphide RD		
		No.	RD Mineralised	RD Unmineralised	No.	RD Mineralised	RD Unmineralised	No.	RD Mineralised	RD Unmineralised
Banded Chert	BCH	2	2.35	2.40	28	2.86	2.93	27	3.11	3.04
Banded Iron Formation	BIF	508	2.45	2.54	226	2.88	2.83	775	3.12	3.10
Carbonaceous Schist	CBS	76	2.32	2.20	51	2.47	2.52	261	2.94	2.89
Carbonaceous Marker	CBS-AS	7	2.52	2.48	20	2.81	2.53	70	3.03	2.89
Chlorite Schist	CS	28	2.22	2.62	65	2.88	2.91	231	3.08	3.01
Interbedded Carbonate Schist and Quartz Carbonate Schist	IQCS and ICQS	131	2.34	2.11	97	2.53	2.40	445	2.94	2.78
Quartz Carbonate Schist	QCS	549	2.49	2.04	278	2.48	2.31	1,078	2.92	2.77
Quartz Vein	QV	55	2.55	2.54	40	2.66	2.58	137	2.84	2.79
Replaced Rock	RP	49	2.38	-	25	2.89	3.00	95	3.08	3.02

The increase in the sample population, coupled with the application of a more rigid RD determination procedure based on recommendations from the RPA 2014 NI 43-101, indicates that the new RD measurements from both mineralised and unmineralised material and from the various material types and lithological units have improved the confidence in the RD determination to be applied to any resource estimates (see Table 9.12). Table 9.13 indicates a positive variance between the previous model RD and the reviewed work for the oxide and transition materials.

Table 9.12: Average RD Measurements for Mineralised Zones 1, 2, 3 and 4 (RP Zone not yet separated)

Type	Average RD			
	Zone 1	Zone 2	Zone 3	Zone 4
Oxide	2.48	2.41	2.57	2.48
Transition	3.01	2.90	2.80	2.71
Sulphide	3.08	3.09	3.00	3.04

Table 9.13: Summary of Previous and Reviewed Mineralised Average RD Measurements

Material Type	RD used in Previous RPA Model	Additional RD Determinations	RD Variance (%)
Oxide	1.80	2.45	36.1
Transition	2.20	2.82	28.2
Sulphide	3.00	3.05	1.7

9.3 2020 TO 2021 EXPLORATION

During the period 2020 to 2021, exploration activities planned for the Imbo Project covered Imbo East.

The programme focused on soil sampling in tandem with geological mapping and sampling of rock chips from outcrops and floats, trenching and channel sampling. A total of 245 rock chip samples, 2,157 soil samples (including field duplicates) from 77.50 km, 126 trench samples from 421.90 m, and 134 channel samples from 175.10 m were collected (see Table 9.14).

Table 9.14: Summary of Imbo East Exploration Statistics (2020 to 2021)

Month	Activity							Number of Samples				
	Gridding (km)	Trench		Adit		Other Channels		Soil	Rocks	Trench	Adit	Other Channels
		Number	Metres	Number	Metres	Number	Metres					
Jan 2020	0	0	0	0	0	0	0	0	13	0	0	0
Feb 2020	0	0	0	0	0	9	71	0	39	0	0	73
Mar 2020	31.64	0	0	0	0	4	12.6	634	39	0	0	13
Apr 2020	39.44	0	0	0	0	0	0	1,269	122	0	0	0
May 2020	0	2	225	0	0	6	44.80	0	2	56	0	24
Jun 2020	0.96	5	196.90	0	0	8	46.70	26	19	70	0	24
July 2020	0	0	0	0	0	0	0	0	0	0	0	0
Aug 2020	0	0	0	0	0	0	0	0	0	0	0	0
Sept 2020	0	0	0	0	0	0	0	0	0	0	0	0
Oct 2020	0	0	0	0	0	0	0	0	0	0	0	0
Nov 2020	2.60	0	0	0	0	0	0	71	8	0	0	0
Dec 2020	0	0	0	0	0	0	0	0	0	0	0	0
Total 2020	74.64	7	421.90	0	0	27	175.10	2,000	242	126	0	134
2021	2.86	0	0	0	0	0	0	157	3	0	0	0
Total	**77.50**	**7**	**421.90**	**0**	**0**	**27**	**175.10**	**2,157**	**245**	**126**	**0**	**134**

Analytical results have been received for all soil samples from the completed 5.4 km × 2.3 km grid, east of the Imbo River where soil samples were collected every 40 m on lines 160 m apart. Geological mapping, soil geochemical, rock chips and channel sampling of old colonial trenches and artisanal workings have outlined four significant mineralised trends – Esio Wapi, Museveni, Mungu Iko and Paradis – approximately 8 km to 10 km southeast of the Adumbi deposit (see Figure 7.2 and Figure 9.19).

At Esio Wapi, the soil geochemical results have outlined a number of +130 ppb Au in soil anomalies with a maximum value of 2,230 ppb Au over a 1.9 km long mineralised trend (see Figure 9.19). Channel sample results from old colonial workings included 19.80 m grading 1.58 g/t Au (open to the northeast), 8 m grading 1.11 g/t Au, and 5.0 m grading 1.65 g/t Au in brecciated BIF and metasediment. Individual rock sample values included 15.10 g/t and 7.88 g/t Au in the quartz veins, 6.39 g/t and 3.08 g/t Au in the BIF, and 9.06 g/t, 7.91 g/t and 3.24 g/t Au in the metasediments.

On the Paradis trend, the soil sample results have outlined a broad 1.0 km trend (+130 ppb Au) with a maximum value of 1,070 ppb Au. Significant channel samples along the Paradis trend include 6.8 m grading 5.44 g/t Au (open to the southwest) in metasediments with quartz veins. Individual rock sample values included 22.40 g/t, 5.84 g/t, and 2.31 g/t Au in the quartz veins.

On the Museveni mineralised trend, anomalous soil samples and artisanal workings occur over a strike of 3.2 km with a maximum value of 5,850 ppb Au in the soils. Channel samples from the artisanal workings include 6.0 m grading 4.37 g/t Au and 1.40 m grading 62.10 g/t Au and represent high-grade quartz veins in the metasediment. Individual rock sample values included 53.90 g/t, 32.80 g/t, and 32.60 g/t Au in the quartz veins and 18.10 g/t Au in the metasediment.

On the Mungu Iko trend, soil samples have outlined a 3.1 km long mineralised trend (+130 ppb Au) with a maximum value of 1,540 ppb Au. Individual rock sample values include 12.30 g/t and 3.50 g/t Au in the brecciated BIF, 14.20 g/t, 4.81 g/t, and 3.68 g/t Au in the metasediments, and 1.97 g/t Au in the quartz veins. Further mapping is required to determine whether the eastern part of the Mungu Iko trend represents a faulted extension of the Esio Wapi trend.

Situated approximately 9 km from the key deposit of Adumbi on the eastern part of the Imbo Project, additional infill soil sampling, augering and channel sampling will be undertaken at Esio Wapi, Paradis, Museveni and Mungu Iko to better define these mineralised trends prior to outlining drill targets.

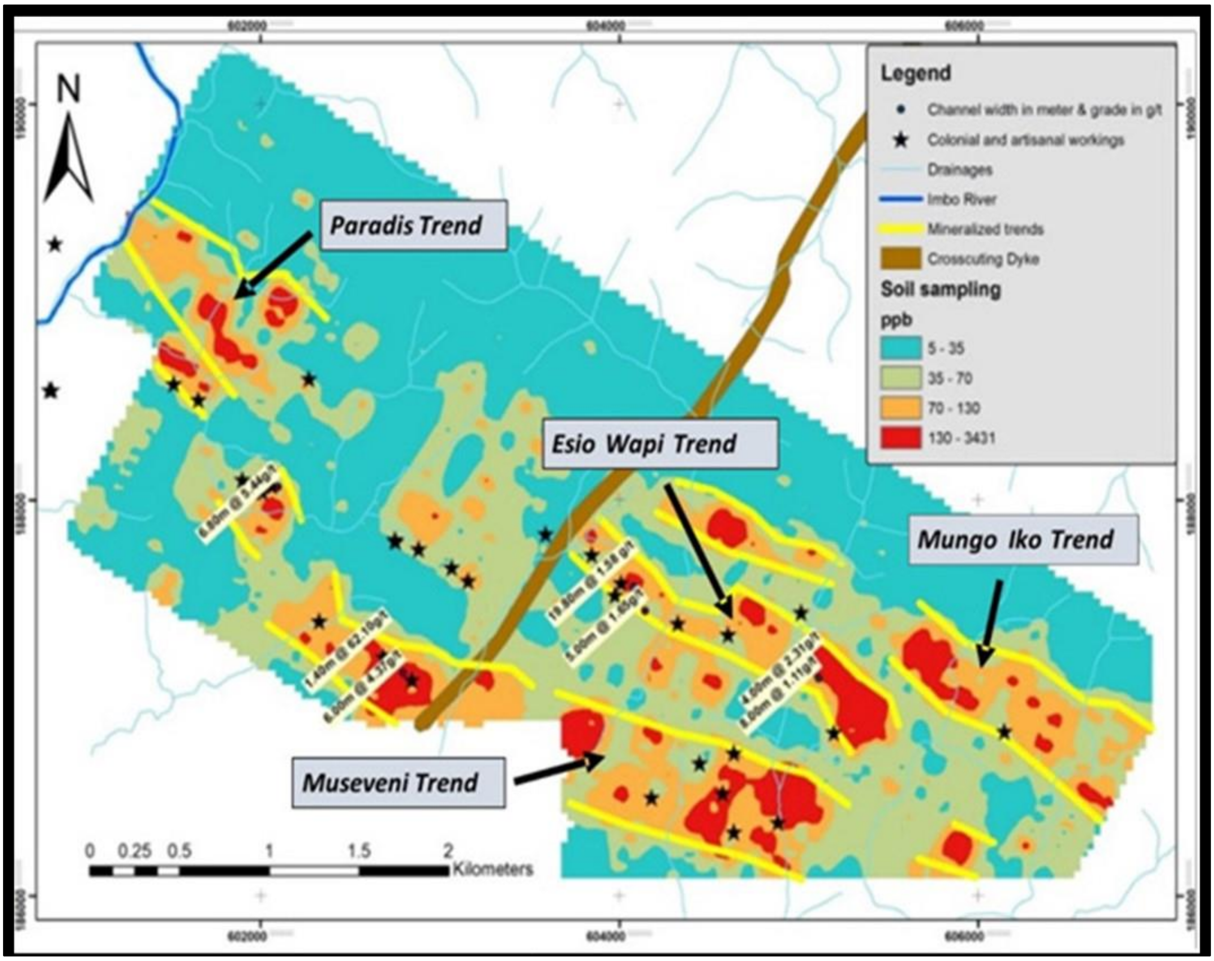


Figure 9.19: Soil Geochemical Trends with Colonial/Artisanal Workings and Channel Samples

10 DRILLING

10.1 PRE-2014 DRILLING

Historical work on the Imbo Project included three diamond drillholes completed by BRGM in 1980. Neither this drilling nor any historical trenching or underground sampling by Belgian explorers and operators has been included in the Kilo drillhole databases.

As of November 15, 2013, Kilo had completed 167 diamond drillholes totalling 35,400 m on the Imbo Project (see Table 10.1).

Table 10.1: 2010 to 2013 Drill Programme Summary of Imbo Project

Year	Prospect or Deposit	No. of Holes Drilled	Metres Drilled
2010	Adumbi	31	6,301
2010	Canal	1	304
2010	Kitenge	5	1,716
2010	Manzako	3	1,016
2010	Monde Arabe	1	302
2010 Subtotal		**41**	**9,639**
2011	Adumbi	18	2,859
2011	Canal	4	470
2011	Kitenge	4	789
2011	Vatican	3	843
2011	Manzako	2	276
2011 Subtotal		**31**	**5,237**
2012	Canal	3	387
2012	Kitenge	28	6,101
2012	Senegal	2	420
2012	Manzako	18	3,641
2012	Lion	1	204
2012 Subtotal		**52**	**10,753**
2013	Kitenge	20	5,581
2013	Senegal	4	772
2013	Manzako	19	3,420
2013 Subtotal		**33**	**9773**
Prospect or Deposit Subtotal:	Adumbi (including Canal)	57	10,321
	Kitenge (including Senegal)	63	15,379
	Manzako (including Lion)	43	8,555
	Monde Arabe	1	302
	Vatican	3	843
TOTAL		**167**	**35,400**
NOTES: 1. Excludes 63.4 m in SADD0023A as deflection to SADD0023 2. Numbers might not add up due to rounding.			

The 2010 and 2011 drilling campaigns were carried out under contract with Senex SPRL, a DRC subsidiary of the drilling company Geosearch, utilising two helicopter-portable Longyear 38 diamond drill rigs. Drilling commenced with PQ-sized drill rods (to produce an 85 mm diameter core). Once the upper weathered zone and fractured formations had been drilled, the drill rod was reduced to HQ-sized core (63 mm diameter core) through the transition zone from highly weathered and/or oxidised units to fresh unweathered competent rocks. The fresh rock was drilled with NQ-sized drill rods, producing a 48 mm diameter core. The drill site preparation was generally completed manually, although a bulldozer was used on accessible sites. Rehabilitation of the sites was carried out by Senex SPRL. Concrete markers were erected on all the drillhole collars.

From 2012, drilling was performed by Congo Core ETS, a DRC based drilling company, utilising two Zinex A-5 drill rigs (Kilo's bulldozer was used for all rig movements). Drilling commenced with HQ-sized drill rods and was reduced to NQ-sized drill rods in the fresh rock. The drill site preparation was generally completed by bulldozer. Rehabilitation of the drill sites was carried out by Kilo and Congo Core ETS. Concrete markers were erected on all the drillhole collars.

Core recovery was generally exceptionally good (> 95 %) in the mineralised sections and unweathered rock while recovery in the saprolite dropped to approximately 50 %.

Table 10.2 summarises the significant drill intercepts for the Adumbi deposit.

Table 10.2: Significant Drill Intercepts from the Adumbi Deposit

BHID	From (m)	To (m)	Intercept Width (m)	True Width (m)	Grade (g/t Au)
SADD0001	151.6	155.6	4	3.9	2.34
	166.6	173.5	6.9	5.27	3.67
	200	227.6	27.6	20.37	2.56
SADD0003	124.75	159.55	34.8	22.23	3.05
	169.75	176.75	7	5.03	2.78
	245.75	259.25	13.5	10.11	2.89
SADD0004	145.2	152.77	7.57	4.89	3.35
	162.6	180.1	17.5	13.65	6.42
	267.75	271.15	3.4	2.67	4.08
SADD0005	116.3	126.8	10.5	6.62	2.99
	130.5	162.5	32	29.11	2.45
	177.55	193.88	16.33	11.49	1.44
SADD0008	178.8	183.1	4.3	3.24	3.07
SADD0011	18.6	25.8	7.2	4.54	2.33
SADD0015	30.3	38.5	8.2	7.07	1.35
	125.75	135.73	9.98	7.48	1.38
	148.86	169.7	20.84	16.4	4.95
SADD0016	0.5	61.6	61.1	21.19	2.09
	86.95	136.8	49.85	25.51	4.29

BHID	From (m)	To (m)	Intercept Width (m)	True Width (m)	Grade (g/t Au)
SADD0017	165.2	174.15	8.95	6.3	1.33
	266.7	309.6	42.9	34.2	3.78
	316.44	329.19	12.75	10.07	2.05
SADD0019	87.4	93.6	6.2	4.94	2.26
	174.58	183.78	9.2	8.46	1.54
	189.1	244.1	55	34.7	1.11
	251.94	257.13	5.19	3.91	3.67
SADD0021	9.5	16.7	7.2	5.61	2.58
	45.33	55.46	10.13	7.1	1.76
	62.9	76.4	13.5	9.81	2.52
SADD0022	140.72	145.55	4.83	2.7	1.42
	157.1	163.04	5.94	3.97	1.28
	198.46	220.7	22.24	14.85	1.31
	242.8	272.5	29.7	21.62	3.50
SADD0024	8.2	15.3	7.1	4.51	2.37
SADD0025	30	48.85	18.85	13.34	2.59
SADD0026	155.34	186	30.66	20.7	5.52
	203.5	208.1	4.6	3.61	5.87
SADD0027	113.9	120.8	6.9	4.99	1.12
	140.5	151.35	10.85	7.68	1.31
	161.3	181.7	20.4	14.9	1.26
	191	224.09	33.09	24.64	1.81
	236.9	239.7	2.8	2.06	2.86
SADD0028	146.35	174.7	28.35	19.39	1.49
	222	251.22	29.22	21.84	2.22
SADD0029	22.23	42.3	20.07	16	1.40
SADD0030	101.7	112.83	11.13	7.66	1.81
	123.17	142.3	19.13	14.16	2.12
	205.06	214.01	8.95	6.3	11.55
SADD0031	19.4	39.5	20.1	13.41	1.42
	57.3	60.3	3	2.28	31.40
SADD0034	107.7	126	18.3	12.29	6.26
	138.4	142	3.6	2.12	4.23
SADD0035	24.4	41.9	17.5	11.75	2.16
SADD0037	89	99.15	10.15	6.39	1.34
SADD0038	71.8	80.4	8.6	**5.52**	**5.70**
SADD0039	104.9	116.9	12	8.65	3.93
	147.9	152.8	4.9	3.88	3.87
SADD0041	21.9	25	3.1	2.21	3.34
	159.88	165.68	5.8	3.61	4.23
SADD0042	247.9	249.9	2	1.87	2.48
SADD0043	33.3	53.45	20.15	12.67	1.66

BHID	From (m)	To (m)	Intercept Width (m)	True Width (m)	Grade (g/t Au)
SADD0044	93.16	103.9	10.74	7.4	7.23
	108	132.64	24.64	17.31	1.83
SADD0045	124.45	127.1	2.65	1.92	2.55
SADD0047	56.46	59.7	3.24	2.36	3.05
	102.95	105.5	2.55	1.68	11.81
SADD0049	64.2	84.21	20.01	8.35	4.22
	89.83	94.37	4.54	1.89	3.78
	109.11	139.53	30.42	13.82	1.29
	227.37	231	3.63	2.06	3.47
SCDD0001	33.4	46	12.6	7.97	7.71
SCDD0002	120.9	123	2.1	1.52	2.54
SCDD0003	51.75	54.75	3	1.79	3.71
	61.6	63.8	2.2	1.48	3.05
SCDD0004	59	65.35	6.35	4.61	4.08
SCDD0006	78.1	83.6	5.5	3.83	2.47
	86.25	97.7	11.45	6.93	3.26

Table 10.3 and Table 10.4 show the significant intercepts for the Kitenge and Manzako deposits, respectively.

Table 10.3: Significant Drill Intercepts from the Kitenge Deposit

Hole ID	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
SKDD0001	30.00	36.00	6.00	2.46
SKDD0003	133.50	136.80	3.30	6.71
SKDD0004	116.95	119.05	2.10	3.94
SKDD0017	100.15	105.84	5.69	1.62
SKDD0018	70.85	72.72	1.87	28.08
SKDD0019	46.19	48.65	2.46	3.42
SKDD0021	78.20	84.00	5.80	42.24
SKDD0022	71.35	74.30	2.95	9.19
SKDD0024	189.92	192.00	2.08	1.97
SKDD0027	149.65	150.95	1.30	3.31
SKDD0029	112.24	116.88	4.64	1.09
SKDD0030	152.70	160.50	7.80	11.47
SKDD0031	114.07	116.55	2.48	4.23
SKDD0035	167.70	168.55	0.85	118.09
SKDD0045	219.20	221.60	2.40	2.75
SKDD0051	245.62	247.60	1.98	10.00

Hole ID	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
SKDD0053	258.81	261.00	2.19	17.24
SKDD0054	103.54	109.36	5.82	2.21
SKDD0057	178.10	179.25	1.15	31.48
SSDD0001	14.50	17.80	3.30	2.49
SSDD0005	92.45	93.15	0.70	48.75
NOTE: Interval thickness can be taken as indicative of the true thickness as the deposit is vertical to subvertical.				

Table 10.4: Significant Drill Intercepts from the Manzako Deposit

Hole ID	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
SMDD0002	25.15	25.9	0.75	7.93
	94.6	99	4.4	10.08
SMDD0003	147.5	149.8	2.3	2.71
	217.43	218.8	1.37	5.49
	236.8	243.36	6.56	6.25
	282.43	284.05	1.62	5.84
SMDD0004	19.3	30.5	11.2	4.96
SMDD0005	114.68	115.8	1.12	1.26
	118.3	128.34	10.04	1.24
SMDD0008	74.85	77.85	3.8	168.2
SMDD0009	83.55	87.85	4.3	43.04
SMDD0014	54.25	57.95	3.7	2.29
	100.2	102.1	1.9	7.34
	179.15	180.3	1.15	12.46
SMDD0016	180.8	182.3	1.5	5.12
SMDD0017	81	83.7	2.7	6.68
	103.75	112.9	9.55	2.72
SMDD0018	126.83	129.4	4.07	17.25
	142.35	143	0.65	6.72
SMDD0019	183.3	184.45	1.15	8.54
SMDD0020	53.81	56.7	2.89	2.69
	100.15	102.15	2	23.46
SMDD0021	45.55	47.35	1.8	2.1
SMDD0022	109.55	111.7	2.15	3.34
SMDD0023	65.3	72	6.7	3.99
	38.45	41.5	3.05	3.39
SMDD0024	43	43.8	0.8	3.42
	73.05	73.45	0.4	11.4

Hole ID	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
SMDD0025	68.1	70.9	2.8	3.77
SMDD0026	83.5	86.25	2.75	6.52
SMDD0027	164.12	166.2	2.08	7.15
SMDD0028	65.05	66.55	1.5	1.54
	91.16	94.75	3.59	5.99
SMDD0029	16.34	19.3	2.96	3.54
	25.95	26.9	0.95	2.38
	34.95	37.95	3	3.19
	58.11	58.61	0.5	6.1
	67.55	68.1	0.55	67.5
SMDD0034	67.05	67.75	0.7	3.98
	149.6	150.15	0.55	9.98
	172.6	173.23	0.63	355.24
SMDD0035	87.17	88.2	1.03	15.24
	58.2	64.34	6.14	2.56
SMDD0039	189.8	193.1	3.3	6.54

10.1.1 Collar Surveys

Drillhole collar locations were determined in the field with a handheld Garmin 60CSx GPS (WGS 84 UTM Zone 35N coordinates) by Kilo geologists. A compass was used to establish a line oriented with respect to magnetic north to indicate the drillhole azimuth. Once the drill rig was moved onto the drill pad and set up, the orientation of the drillhole was verified with a clinometer and compass by a geologist.

Drillhole, trench, and adit portal elevations at Adumbi were derived from a satellite DTM as handheld GPS elevations were notoriously inaccurate due to the thick jungle canopy.

In the summer of 2013, Young, Stuart & Associates (YSA) of South Africa was appointed by Kilo to establish survey control points at the Adumbi base camp and conduct a tachometric survey of drillholes, section lines, baselines and trenches in the Imbo Project area.

10.1.2 Drillhole Downhole Survey

During the 2010 and 2011 drilling campaigns, downhole survey data was collected at 15 m intervals using a FlexIT survey tool with a digital readout. Since 2012, downhole survey data has been collected at 15 m intervals using a Reflex EZ TRAC survey tool with a digital readout. The data was digitally stored and downloaded by Kilo geologists to a Kilo computer.

10.1.3 Drillhole Database

RPA received and conducted an extensive review and validation of the drillhole database, which was in an MS Excel format, and concluded the following:

- **Adumbi Drillhole Database:**
  - Contains 87 records consisting of 57 diamond drillholes, 20 surface trenches, and 10 underground channel sample lines (represented as drillholes in the database), totalling 12,616 m. Drilling accounts for 82 % of the total length.
  - Contains 9,672 samples encompassing 12,495 m for 7,812 assays above the detection limit.
- **Kitenge Drillhole Database:**
  - Contains 69 records consisting of 63 diamond drillholes, 5 surface trenches, and 1 road cutting (represented as drillholes in the database), for a total of 16,268 m. Kilo drilling accounts for 95 % of the total length.
  - Contains 12,140 samples encompassing 14,557 m for 9,356 assays above the detection limit.
- **Manzako Drillhole Database:**
  - Contains 58 records consisting of 43 diamond drillholes, and 15 surface trenches, for a total of 9,698 m. Drilling accounts for 88 % of the total length.
  - Contains 7,154 samples encompassing 9,000.84 m for 4,143 assays above the detection limit.

10.2 2014 TO 2017 DRILLING

10.2.1 Planning

A drilling programme was planned to test the gold-in-soil and magnetic anomalies at the Adumbi South, Adumbi West and Kitenge Extension targets, the locations of which are shown in Figure 10.1. The planned programme comprised 63 drillholes totalling approximately 8,900 m and was carried out by Orezone Drilling SARL based in Watsa in the DRC.

The programme employed one track-mounted rig (commencing at Adumbi South) and one man-portable rig (commencing at Kitenge Extension). Drilling was initially on a single-shift basis for approximately one week, and then changed to double shifts.

Drillhole collar coordinates were determined using Target software, and the sites were pegged in the field using a handheld GPS (± 5 m accuracy).

The holes were planned to be drilled to an average downhole depth of 140 m (maximum of 167 m) and are inclined at −50°. All cores were orientated to facilitate structural interpretation, and half-core sampling was done based on geological features with a maximum sample length of 1 m. Samples were submitted for fire assay to SGS Mwanza, with whom a new contract was negotiated in August 2016.

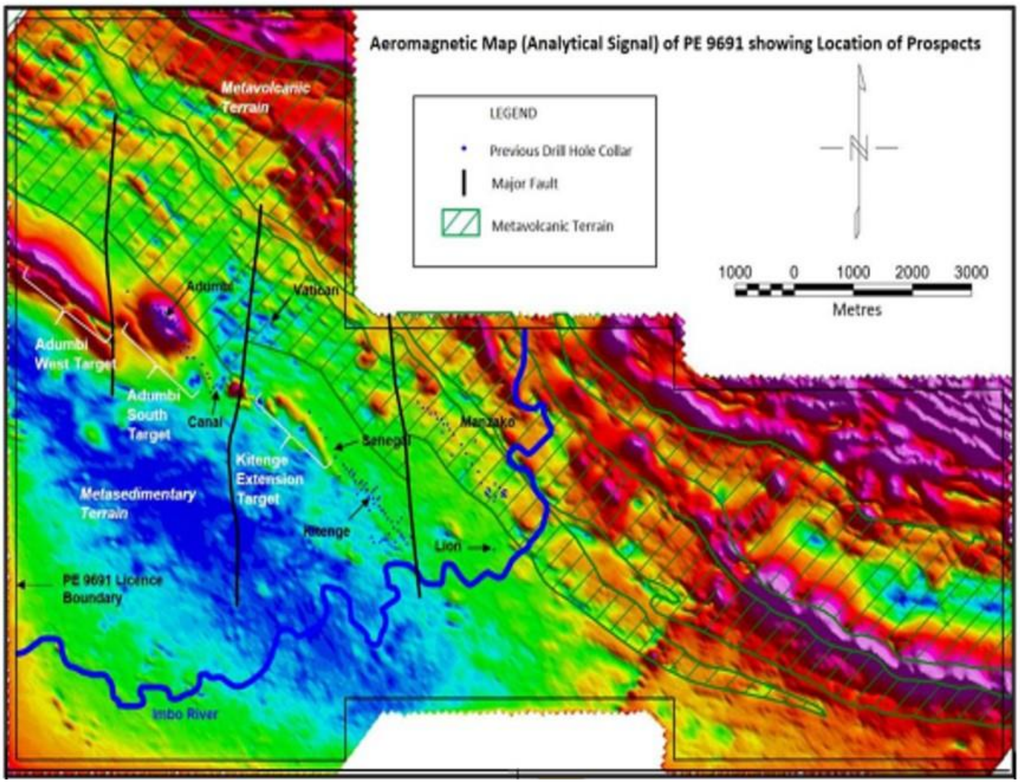


Figure 10.1: Location of Drill Targets on the Imbo Project (Adumbi South, Adumbi West and Kitenge Extension)

10.2.1.1 Adumbi South Target

The planned programme comprised 20 drillholes totalling 3,085 m, on 7 traverses at a spacing of 160 m along strike. The target lies 480 m to the south the Adumbi deposit and is defined by a 1.4 km long magnetic anomaly that appears to be demagnetised in places, and a > 200 ppb gold-in-soil anomaly. This target had similar geomorphological features to those of Adumbi West in that it occurs in a topographical low and is variably covered by transported soil with little to no lithological exposure. The nature of the gold-in-soil anomaly and the associated magnetic feature at Adumbi South was very similar to that associated with the Canal zone, which is thought to be the southeastern extension of the Adumbi mineralisation.

10.2.1.2 Adumbi West Target

The programme comprised 26 drillholes totalling 3,367 m, on 10 traverses at a spacing of 160 m along strike. This target lies to the west of the Adumbi deposit and is believed to be the faulted extension of Adumbi. It occurs in a topographical low, and for the most part is covered by transported material varying in depth from 30 cm to > 3 m. The target is defined by a 1.7 km long linear magnetic anomaly and a coincident and linear gold-in-soil anomaly with values of 50 ppb to 1,000 ppb. This magnetic feature is like that which defines the BIF at the Adumbi deposit.

10.2.1.3 Kitenge Extension Target

The programme comprised 17 drillholes totalling 2,435 m, on 7 traverses at a spacing of 320 m along strike. The target lies to the northwest of the Kitenge deposit and is defined by an approximately 2 km long magnetic feature with a coincident gold-in-soil anomaly with values from 50 ppb to 450 ppb. The magnetic feature has similar characteristics to that at the Canal and Adumbi South targets.

10.2.2 Drilling

The standard procedure required drill rig personnel to place the recovered drill core into metal core trays labelled at the drill site with the drillhole number. End-of-run markers were placed in the core tray between the end and start of each recovered drill run. Information on core recovery, depth of the run, stickup length, and ground conditions were recorded for each run and inspected by the rig geologists. The core was transported from the drill site by vehicle or helicopter to the core yard facility at the Adumbi base camp.

Prior to logging and sampling, the drill core was digitally photographed in order to maintain a permanent record. All the drill core photographs were downloaded into the project database, retained in company computers on site and at the corporate office in Toronto, Canada.

A total of 5,132 m from 34 holes were drilled.

10.2.3 Core Logging

Logging procedures included an initial visual assessment of the core with zones of good and poor mineralisation noted. This was then followed by geological logging of the lithology, alteration, structure, oxidation, mineralisation, general rock description, and magnetic susceptibility. The rock description recorded colour and approximate mineral assemblage. The drill data was summarised in cross section and also displayed in Strater log software.

10.2.4 Sampling and Assaying

One-metre sample lengths (adjusted for lithology) were marked on the core in the mineralised horizons during logging. The sample depths for each sample were entered into a sample ticket book, which contained removable duplicate sample ticket tags. The core sample numbers and sample intervals were written onto pre-printed diamond drill log forms. Each marked sample was split along its length by trained staff using a dedicated drill core diamond saw. The core was broken at the sample position marks using a geological pick. The sampling lengths were reduced, when necessary (e.g. where lithological contacts or core size changes were encountered), with the bottom/top end of the sample being approximately 2 cm from the contact. One half of the core was replaced in the core tray, and the remaining half was placed in a plastic sample bag, in which the sample number was folded in along the open end of the bag, which was then closed using a stapler. Sample tags were placed in the core tray at the position of the bottom end where samples had been obtained. A brick was sawn (“brick cleaning”) after each sample had been split to ensure that no cross-contamination took place between samples.

All the core samples were sent to the SGS Laboratory in Mwanza for assaying. The core samples were then crushed down to −2 mm and split with one half of the sample pulverised down to 90 % passing 75 µm. Gold analyses were carried out on 50 g aliquots by fire assay. In addition, checks assays were also carried out by the screen fire assay method to verify high-grade sample assays obtained by fire assay. Internationally recognised standards and blanks were inserted as part of Loncor's internal QA/QC analytical procedures.

10.2.5 Core Re-Logging of All Core

Per RPA's recommendations in 2014, re-logging of all the core by Loncor in Q1 2020 identified major differences between the depths of the base of complete oxidation (BOCO) and the top of fresh rock (TOFR) and the depths used by RPA in the 2014 model. In the RPA model, the BOCO was negligible and the TOFR corresponded approximately to the re-logged BOCO. The deeper levels of oxidation that were observed during the re-logging exercise has had positive implications for the Adumbi project with respect to ore type classification and associated metallurgical recoveries and mining and processing cost estimates.

The re-logging exercise defined the presence of five distinct geological domains in the central part of the Adumbi deposit where the BIF unit attains a thickness of up to 130 m. From northeast to southwest, these are as follows:

1. Hanging Wall Schists: dominantly quartz carbonate schist, with interbedded carbonaceous schist.
2. Upper BIF Sequence: an interbedded sequence of BIF and chlorite schist, 45 m to 130 m in thickness.
3. Carbonaceous Marker: a distinctive 3 m to 17 m thick unit of black carbonaceous schist with pale argillaceous bands.
4. Lower BIF Sequence: BIF interbedded with quartz carbonate, carbonaceous and/or chlorite schist in a zone 4 m to 30 m wide.
5. Footwall Schists: similar to the hanging wall schist sequence.

In the central part of Adumbi, three main zones of gold mineralisation are present (see Figure 10.2):

- Within the Lower BIF Sequence
- In the lower part of the Upper BIF Sequence (Zones 1 and 2 are separated by the Carbonaceous Marker, which is essentially unmineralised)
- A weaker zone in the upper part of the Upper BIF Sequence

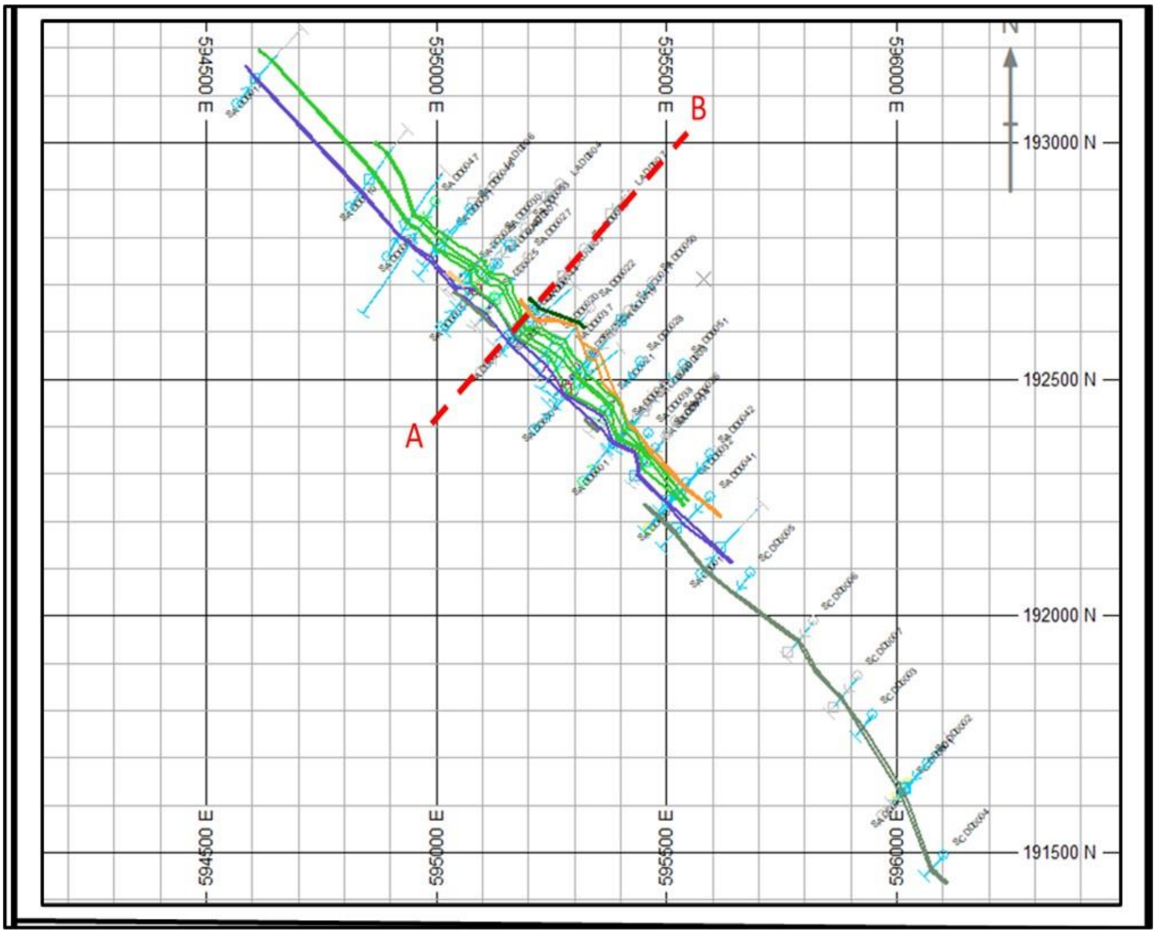


Figure 10.2: Plan of the Interpreted Mineralised Zones

10.2.6 Analytical Results

Sample results for the drilling at Adumbi South, Adumbi West and Kitenge Extension demonstrate that the gold mineralisation was confined to narrow and/or low-grade zones. The most significant intersections from the programme were as follows:

- **Adumbi South:**
    - 1.00 m at 3.85 g/t in ASDD003
- **Adumbi West:**
    - 1.45 m at 8.53 g/t in AWDD002
- **Kitenge Extension:**
    - 2.90 m at 1.05 g/t in SKDD0060
    - 1.60 m at 10.52 g/t in SKDD0063
    - 1.00 m at 3.08 g/t in SKDD0065
    - 7.36 m at 1.31 g/t in SKDD0070
    - 0.80 m at 23.90 g/t in SKDD0065

The results indicated little economic potential at Adumbi South, Adumbi West or Kitenge Extension and hence no further drilling was planned.

RPA recommended additional drilling at Adumbi to test the down dip/plunge extent of the mineralisation. In 2017, four deeper core holes were drilled below the previously outlined RPA Inferred Resource over a strike length of 400 m and to a maximum depth of 450 m below surface. All four holes intersected significant gold mineralisation in terms of widths and grade and are summarised Table 10.5.

Table 10.5: Summary of Significant Drill Intercepts from the 2017 Adumbi Deep Hole Drilling

Borehole	From (m)	To (m)	Intercept Width (m)	True Width (m)	Grade (g/t Au)
SADD0050	434.73	447.42	12.69	10.67	5.51
SADD0051	393.43	402.72	9.29	6.54	4.09
SADD0052	389.72	401.87	12.15	7.01	3.24
	419.15	428.75	9.60	5.54	5.04
SADD0053	346.36	355.63	9.27	5.70	3.71
	391.72	415.17	23.45	14.43	6.08

The above drilling results, which are shown on the longitudinal section (see Figure 10.3), indicate that the gold mineralisation is open along strike and at depth.

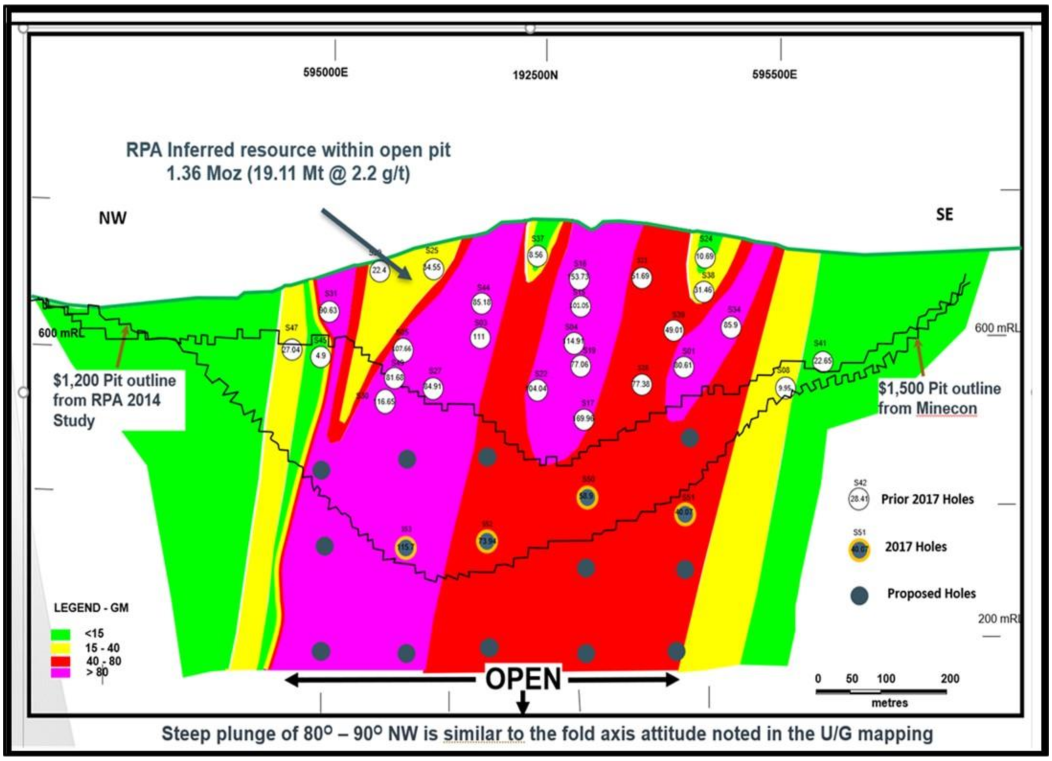


Figure 10.3: Longitudinal Section of Adumbi Showing the Down Dip/Plunge, Potential and Proposed Drillholes

10.3 2020 TO 2021 DRILLING

This section summarises the drilling activities completed on the Adumbi deposit during the Loncor 2020 to 2021 drilling programme.

Following Minecon's review of the Imbo Project, accompanied by an Independent NI 43-101 Technical Report dated April 17, 2020, a recommendation was made to drill 12 diamond core holes totalling 6,250 m at the Adumbi deposit (see Figure 10.3). This was aimed at outlining additional mineral resources to the reported 2.5 Moz at the Imbo Project (Inferred Mineral Resources of 30.65 Mt grading 2.54 g/t Au).

The drillholes were planned on the 220° azimuth with varying inclinations, and to a maximum depth of 710 m. These holes were subsequently reviewed and prioritised to establish a preferred sequence of drilling as shown in Table 10.6.

Table 10.6: Initial Planned Adumbi Diamond Drillholes with Sequence of Drilling

BHID	UTM-Easting	UTM-Northing	End of Hole (EOH) (m)	Planned PQ (m)	Planned HQ (m)	Planned NQ (m)	Sequence of Drilling
ADDP001	595128	192925	350	100	200	50	5
ADDP002	595165	192971	470	100	200	170	8
ADDP003	595206	193028	600	100	200	300	7
ADDP004	595173	192790	360	100	200	60	1
ADDP005	595270	192910	600	100	200	300	3
ADDP006	595275	192715	350	100	200	50	2
ADDP007	595413	192888	670	100	200	370	4
ADDP008	595522	192765	600	100	200	300	10
ADDP009	595566	192819	710	100	200	410	9
ADDP010	595500	192483	350	100	200	50	6
ADDP011	595581	192580	540	100	200	240	12
ADDP012	595620	192632	650	100	200	350	11
TOTAL			**6,250**	**1,200**	**2,400**	**2,650**	

The drilling contract was awarded to Orezone Drilling, following a tendering process. Orezone Drilling had previously drilled the four deep holes on the Adumbi deposit in 2017.

Drilling commenced in October 2020 with a Sandvik DE 710 rig (see Figure 10.4), initially drilling a 12 h day shift and later switched to a 12 h double shift. A second rig, an Atlas Copco CS14, was mobilised to site and commenced drilling on November 10, 2020, on the double shift (see Figure 10.5).

Figure 10.4: Sandvik DE 710 Rig, Drilling LADD001

Figure 10.5: Atlas Copco CS 14 Rig, Drilling LADD004

10.3.1 Drillhole Nomenclature

A new drillhole nomenclature (LADD00X) was adopted as part of this drilling campaign. Thus, the Borehole ID LADD00X refers to Loncor Adumbi Diamond Drillhole 00X, where 00X is the hole number.

10.3.2 Downhole Survey

The downhole survey was initially done with Reflex EZ Trac equipment at every 30 m, and reports were submitted to the rig geologist at the end of every survey. These were plotted progressively to determine any hole deflections and their possible impact on the objectives of the drilling programme. However, following a few discrepancies that were noticed in the survey readings, a recommendation was made to replace the Reflex EZ Trac survey equipment with a Gyro unit to avoid any possible effects of the magnetic properties of the BIF unit on the readings. Subsequently, a Sprint-IQ gyro was used for the subsequent drillholes.

10.3.3 Core Orientation and Structural measurements

Reflex Act ll orientation survey equipment was used for core orientation at every run of 3 m in competent material to aid in structural measurements. The surveys were verified by the rig geologist at the end of each run and marked as either reliable or unreliable.

Structural measurements taken during the routine logging were from bedding, foliation, and quartz veins (see Figure 10.6, Figure 10.7 and Figure 10.8) whereas structural measurements from the lithological contacts, joints and shears have been captured in detail under a separate geotechnical logging programme.

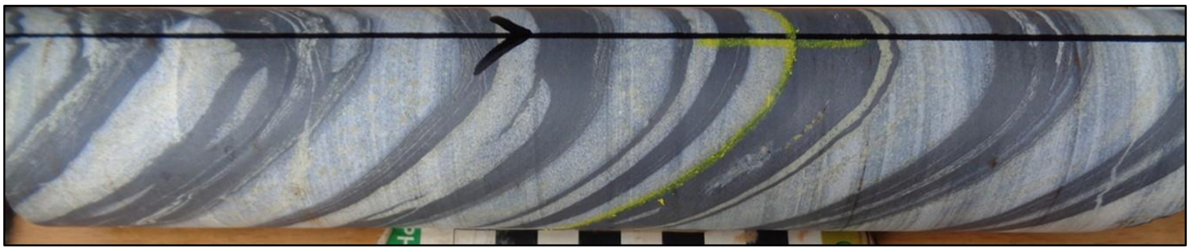


Figure 10.6: Bedding in BIF Unit of LADD001, from 153.20 m to 153.45 m

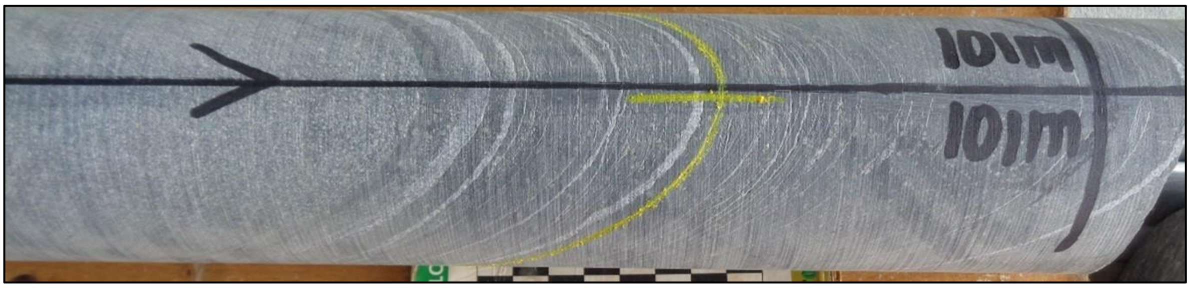


Figure 10.7: Foliation in QCS Unit of LADD003, from 100.80 m to 101.02 m

Figure 10.8: Quartz Veining in QCS Unit of LADD001, from 340.67 m to 340.87 m

All the structural measurements were taken using a kenometer, which measures alpha (α) and beta (β) angles using the bottom of hole line (BOHL as reference (see Figure 10.9).

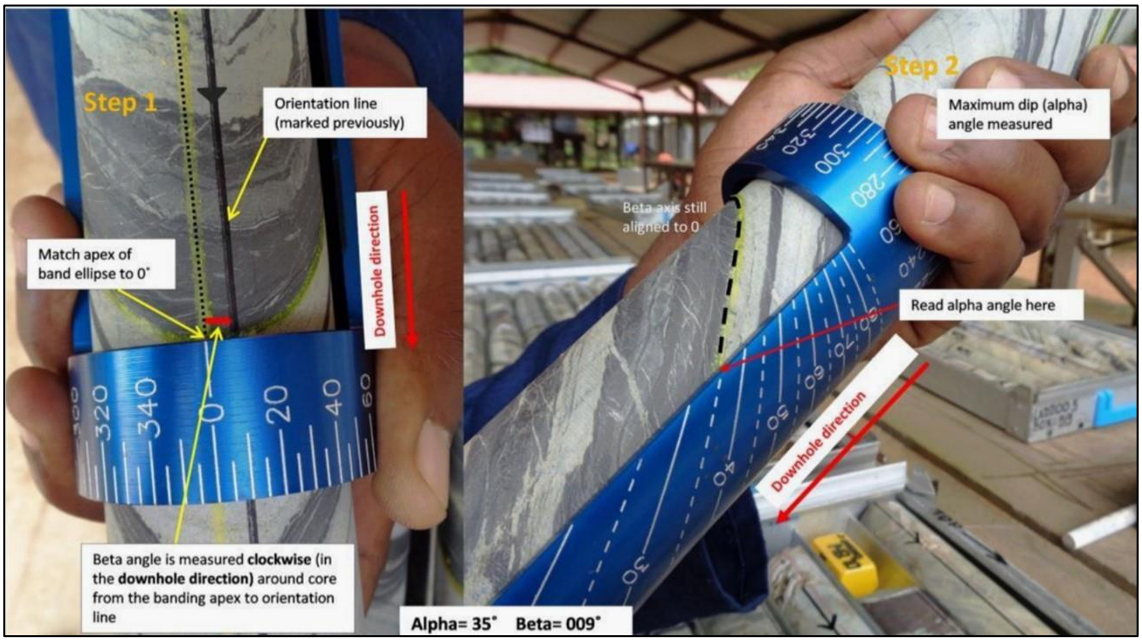


Figure 10.9: Use of a Kenometer to Measure Alpha (α) and Beta (β) Angles of an Oriented Core

These readings were then converted to the "strike/dip right" convention using the Dips Software. The converted combined structural measurements for LADD001, LADD003, LADD004, LADD006, LADD007, LADD008, LADD009, LADD012, LADD013, LADD014, LADD015, LADD016, LADD017, LCDD001, LADD018, LADD019, LADD020, LADD021, LADD022, LADD023, LADD024, LADD025 and LADD026 were plotted on a stereographic net to aid in interpretation. The structural interpretation for the above completed drillholes is presented in Section 7.5.

10.3.4 Rig Monitoring, Core Recovery and RQD Measurements

The rig geologist monitors the daily drilling activities to ensure that the quality of the core is not compromised. Once the core is placed on the angle iron by the driller, the geologist checks the orientation survey and marks the appropriate BOHL on the core, indicating whether the survey is reliable, unreliable or no survey at all. A solid black line is used to mark a reliable survey, a broken line to indicate an unreliable survey, and a dotted line to indicate where no survey was conducted. All these lines are marked with an arrow pointing to the downhole of the core. The BOHL is marked with a black permanent marker. After transferring the core from the angle iron into the appropriately labelled core tray (either PQ, HQ or NQ), the driller's metres are indicated with yellow labelled plastic blocks (see Figure 10.10). Any core losses are recorded, and the location marked with a labelled wooden block. The percentage core recovery is measured as well as the rock quality designation (RQD). Quick lithological logging is done at the rig site and any alterations/mineralisation recorded. All the core recovered during the shift, except for the last half-filled box, is transported to the camp core shed during the day.

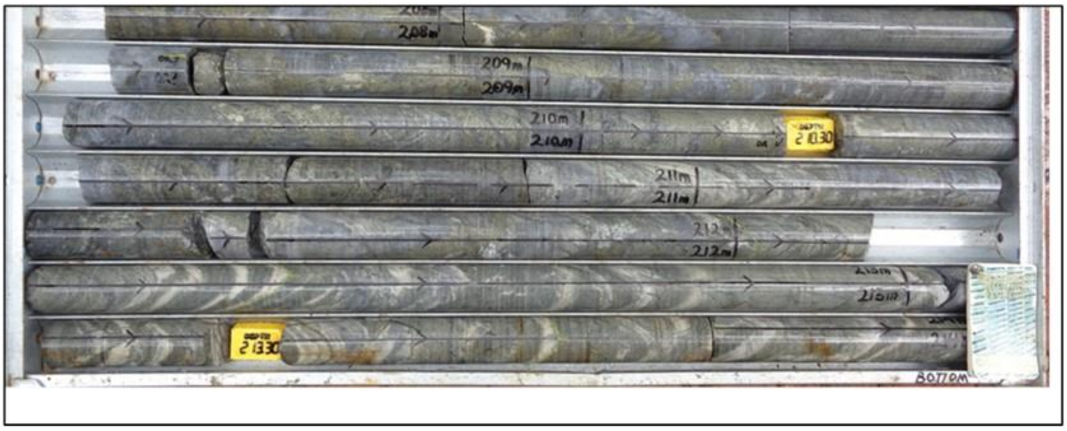


Figure 10.10: Core Tray showing BOHL, Metre Marks and Driller's Metre Blocks

10.3.5 Drillhole Collar Survey

All completed drillholes were surveyed using the global navigation satellite system (GNSS) Trimble R10. The survey data was derived from reference control points (see Table 10.7) located within the Adumbi base camp (established since 2014) as survey control points with UTM (Zone 35 North) based on the Datum WGS 84 coordinate system (see Figure 10.11).

Table 10.7: Adumbi Deposit Survey Control Coordinate Points in UTM

Point ID	Easting	Northing	Elevation	Code
14MRSCM	596523.35	191570.88	649.6	10IPIC
14SCM1	596620.47	191457.32	644.39	10IPIC
14SCM2	596669.84	191500.62	646.41	10IPIC

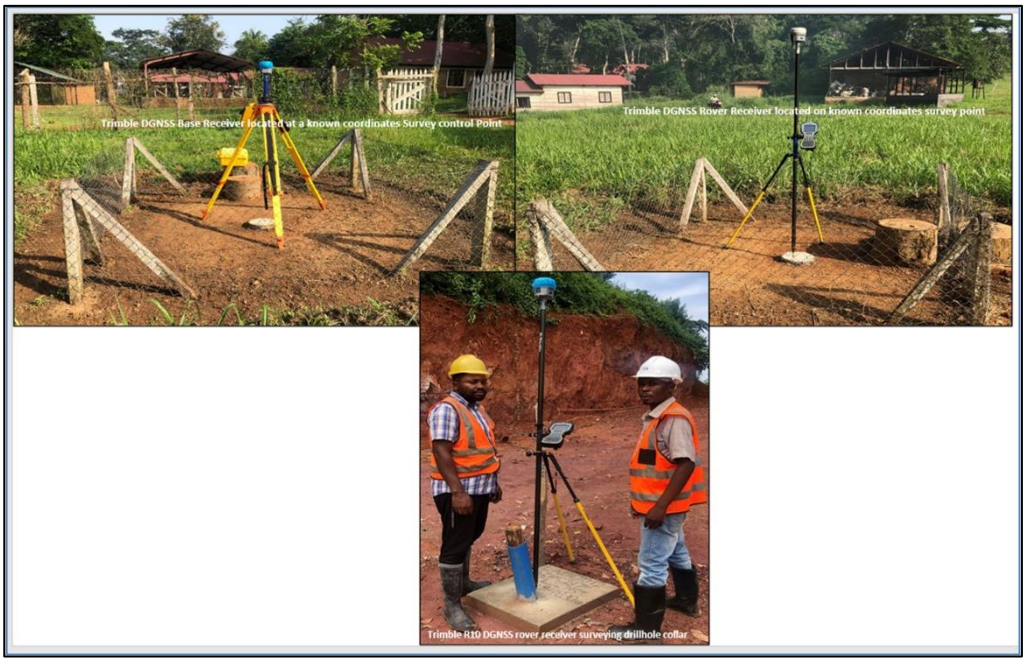


Figure 10.11: Trimble R10 GNSS Survey Control Points and Rover Receiver Surveying Drillhole Collar

Figure 10.12 presents the planned and completed drillholes and access roads for the 2020 to 2021 drilling programme.

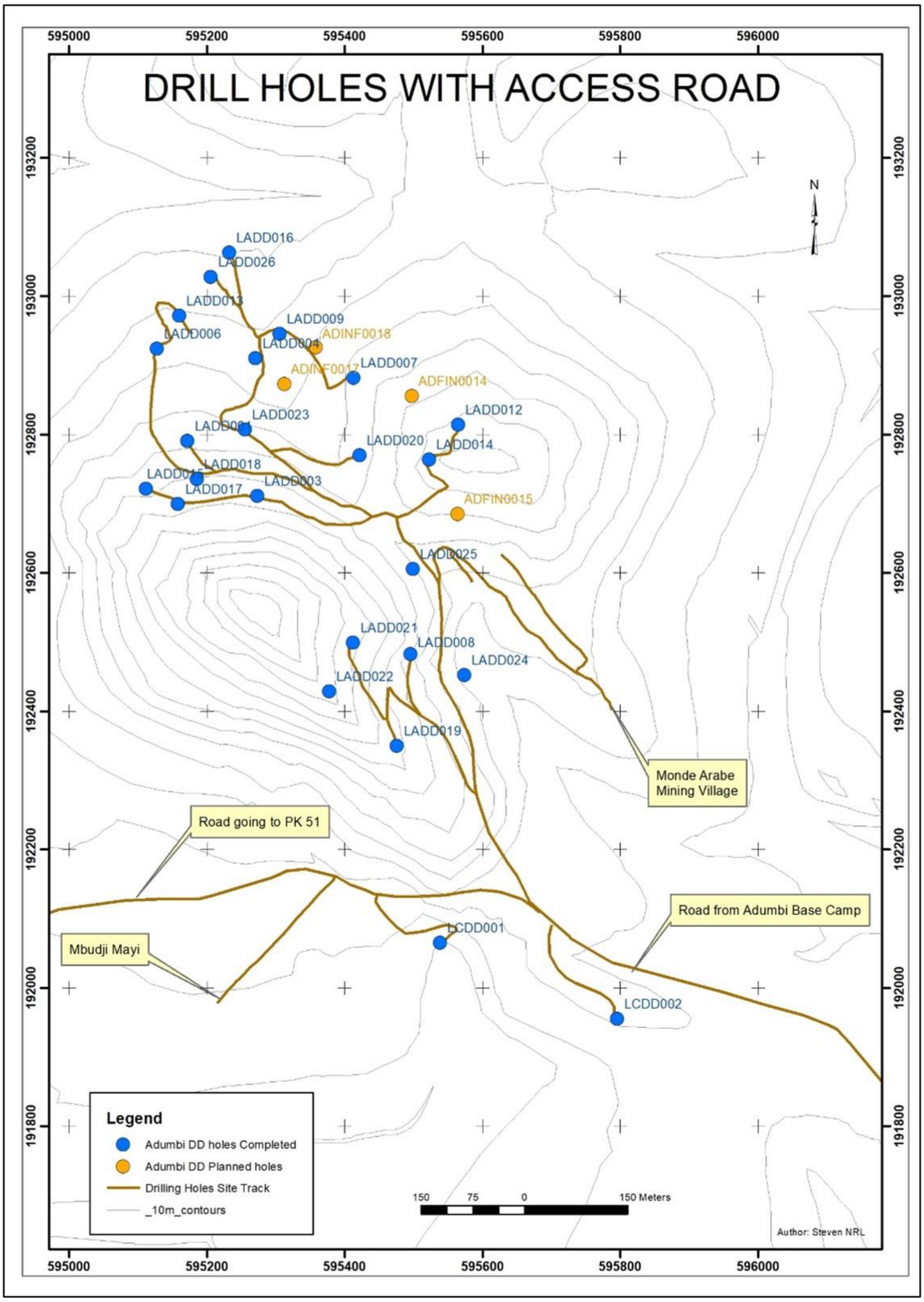


Figure 10.12: Adumbi Planned and Completed Drillholes with Access Roads

The 24 completed holes (totalling 10,071.44 m) drilled at Adumbi during 2020 to 2021 and covered by this report are detailed in Table 10.8.

Table 10.8: Drill Collars of Adumbi Completed Boreholes

BHID	Prospect	Easting (m)	Northing (m)	RL (m)	Azimuth (°)	Inclination (°)	End Depth (m)
LADD001	Adumbi	595172	192791	685.20	220	−65	360.30
LADD003	Adumbi	595273	192712	704.97	220	−57	309.20
LADD004	Adumbi	595271	192911	660.28	220	−70	566.30
LADD006	Adumbi	595127	192924	678.64	218	−58	395.35
LADD007	Adumbi	595413	192882	680.21	218	−68	647.75
LADD008	Adumbi	595496	192483	689.75	218	−65	365.35
LADD009	Adumbi	595306	192945	667.60	218	−75	689.30
LADD012	Adumbi	595565	192814	714.58	217	−75	948.30
LADD013	Adumbi	595160	192971	667.25	218	−72	485.80
LADD014	Adumbi	595523	192764	718.22	217	−72	772.93
LADD015	Adumbi	595112	192722	717.83	220	−49	168.50
LADD016	Adumbi	595232	193063	659.07	217	−75	867.95
LADD017	Adumbi	595158	192699	716.96	219	−46	147.40
LCDD001	Canal	595539	192065	640.44	40	−51	196.10
LCDD002	Canal	595796	191955	656.61	220	−50	163.50
LADD018	Adumbi	595186	192735	710.00	219	−45	204.66
LADD019	Adumbi	595476	192350	711.86	221.7	−50	151.40
LADD020	Adumbi	595422	192770	698.94	221	−45	433.80
LADD021	Adumbi	595413	192499	728.00	219.28	−49	219.10
LADD022	Adumbi	595378	192429	748.00	220	−56	149.50
LADD023	Adumbi	595256	192808	677.00	221	−50	377.55
LADD024	Adumbi	595574	192452	670.75	222	−49	352.80
LADD025	Adumbi	595499	192606	690.50	219	−48	393.60
LADD026	Adumbi	595206	193028	658.00	217	−72	705.00
Total							**10,071.44**

10.3.6 Core Logging

Upon receipt of the core at the camp core shed, the senior geologist proceeds with systematic core logging (see Figure 10.13). The logging procedures include an initial visual assessment of the core with zones of good and poor mineralisation noted. This is then followed by geological logging with separate log sheets capturing lithology, alteration, structure, geotechnical, oxidation, mineralisation, general rock description and magnetic susceptibility. The rock description records colour and approximate mineral assemblage. The drill data is then summarised in cross section and displayed in Strater log software. A typical Strater log for LADD023 is displayed in Figure 10.14 to Figure 10.20.

The BOCO and the TOFR for each drillhole are measured and recorded. Those for LADD001 were measured at 59.70 m (BOCO) and 71.03 m (TOFR).

Figure 10.13: Senior Geologists Logging Core from LADD001

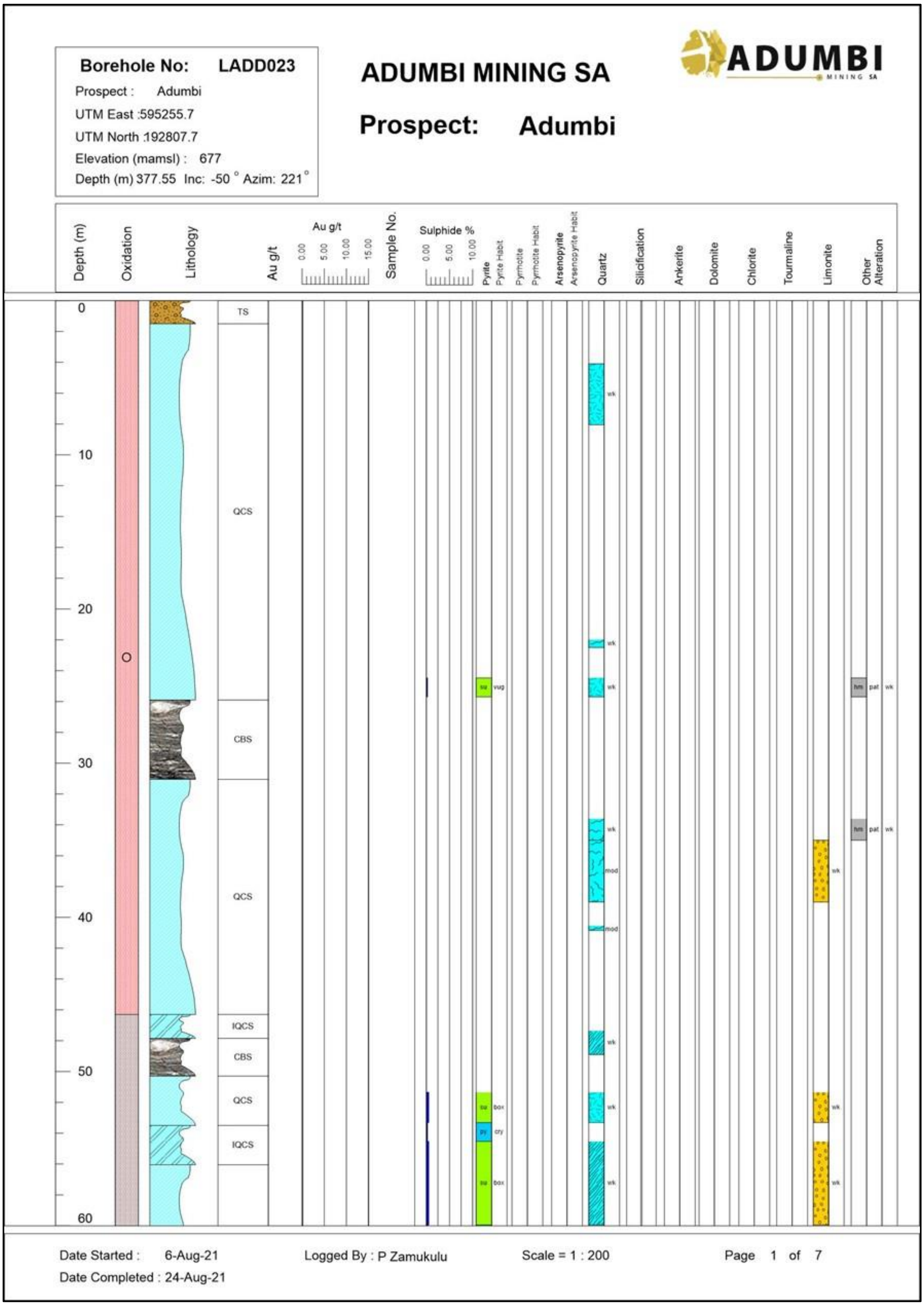


Figure 10.14: Strater Log for LADD023 – Page 1

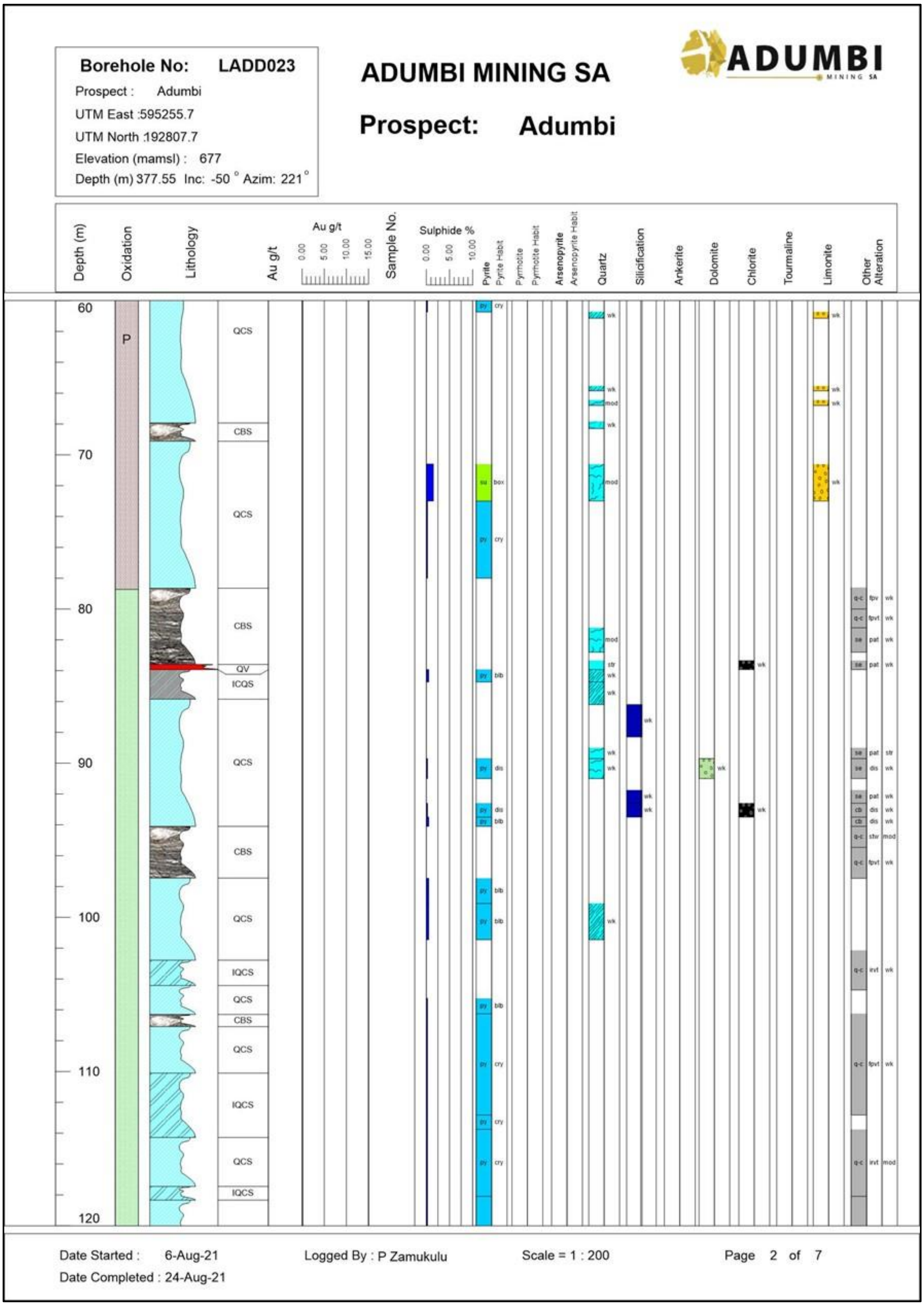


Figure 10.15: Strater Log for LADD023 – Page 2

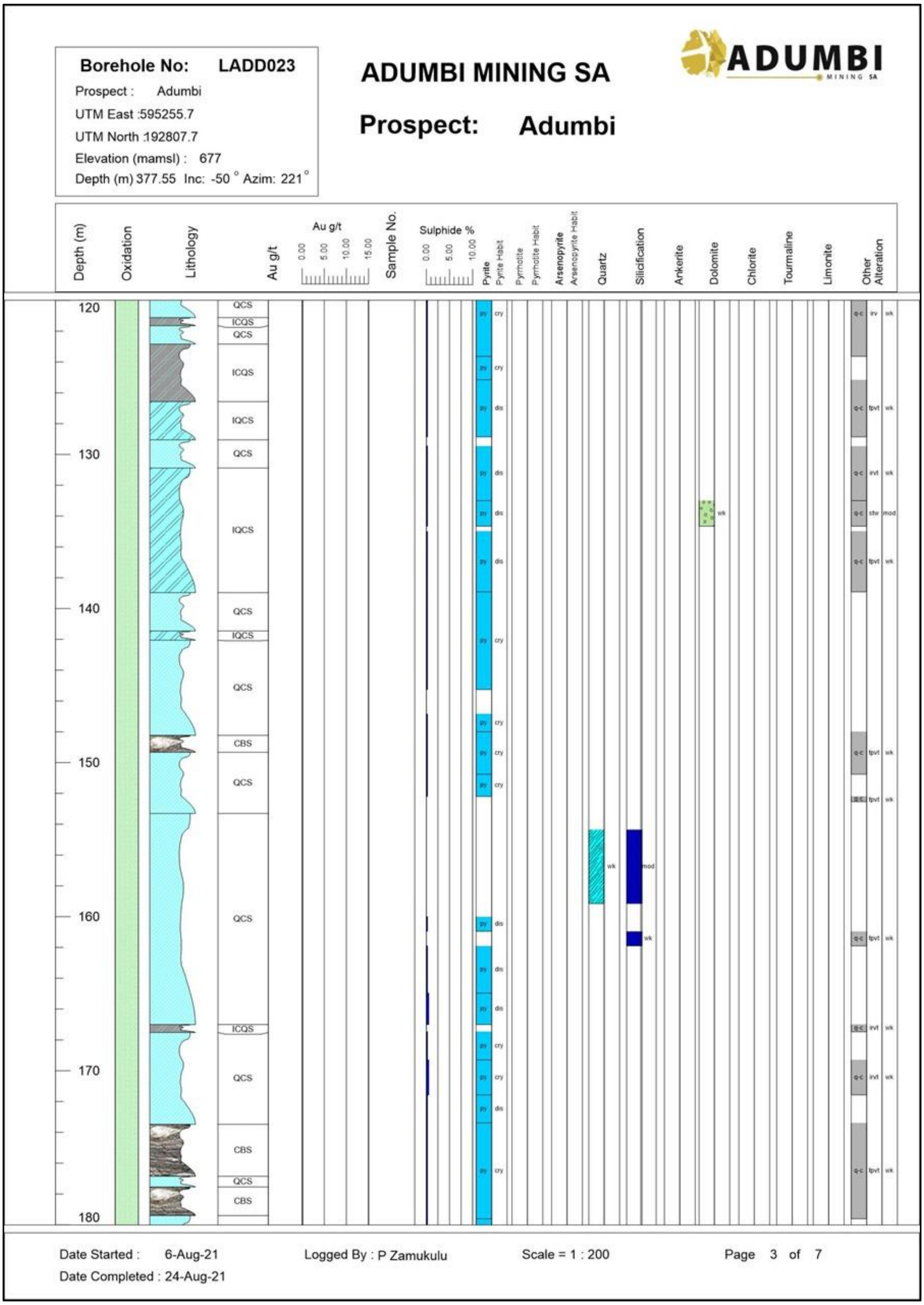


Figure 10.16: Strater Log for LADD023 – Page 3

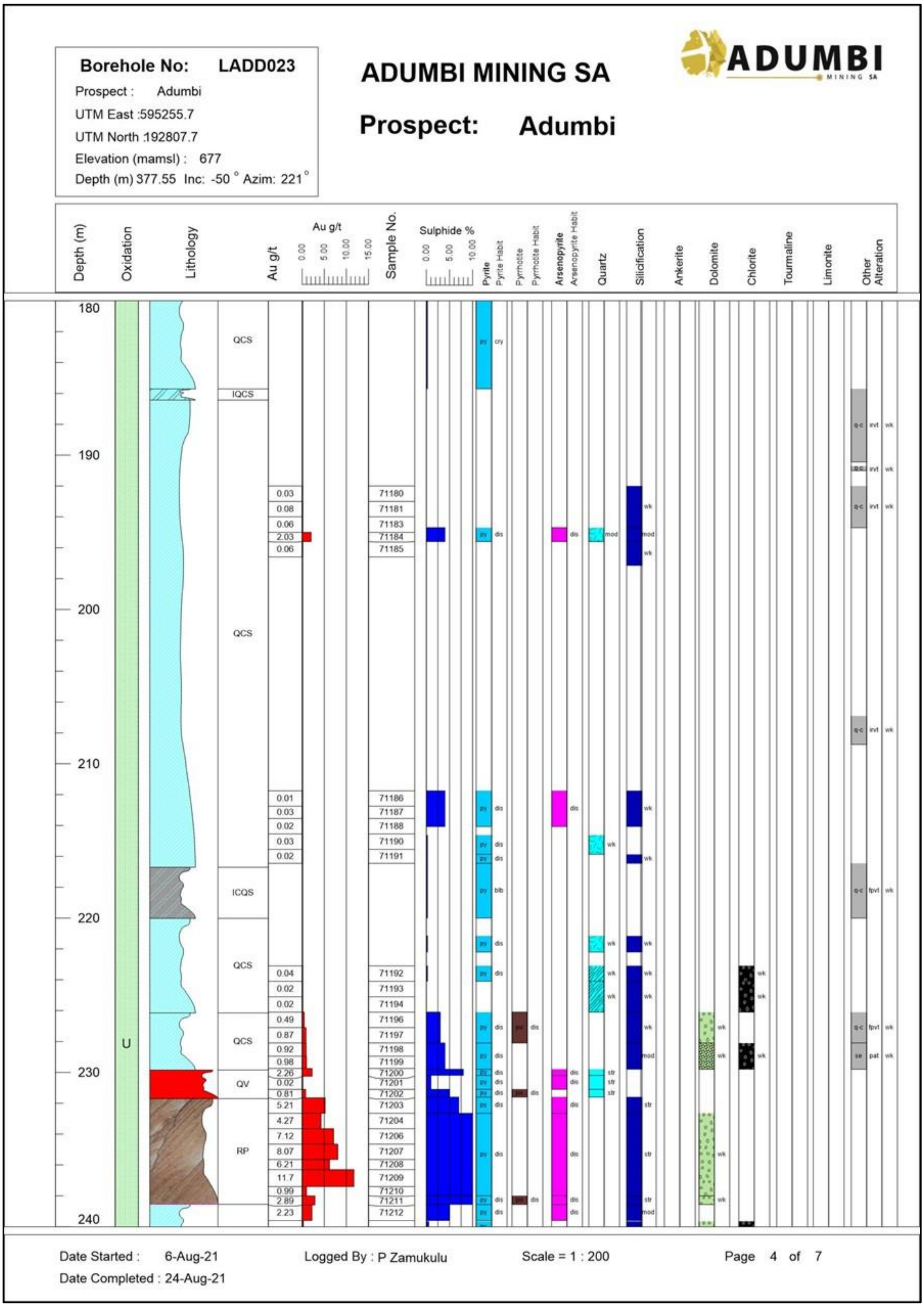


Figure 10.17: Strater Log for LADD023 – Page 4

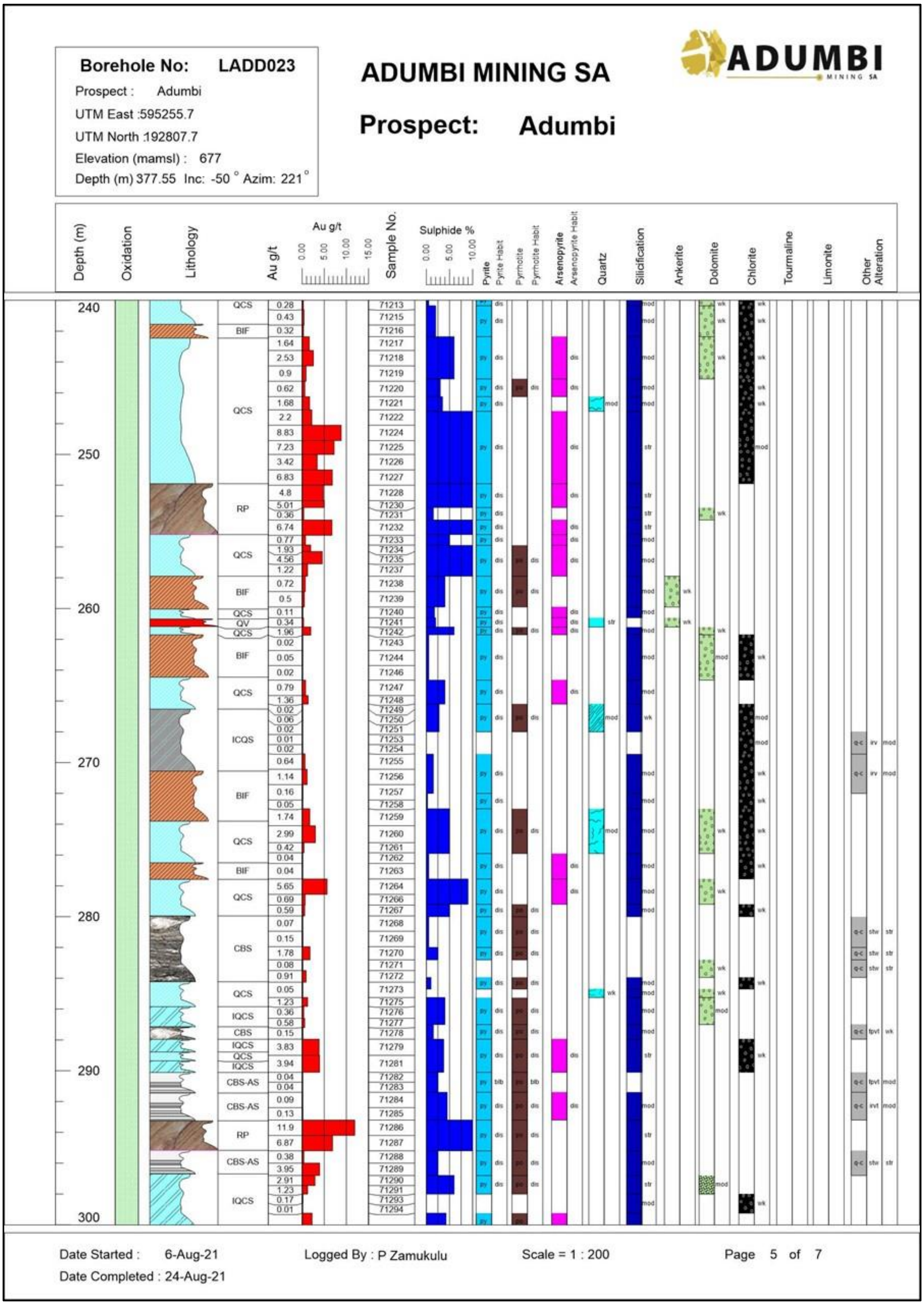


Figure 10.18: Strater Log for LADD023 – Page 5

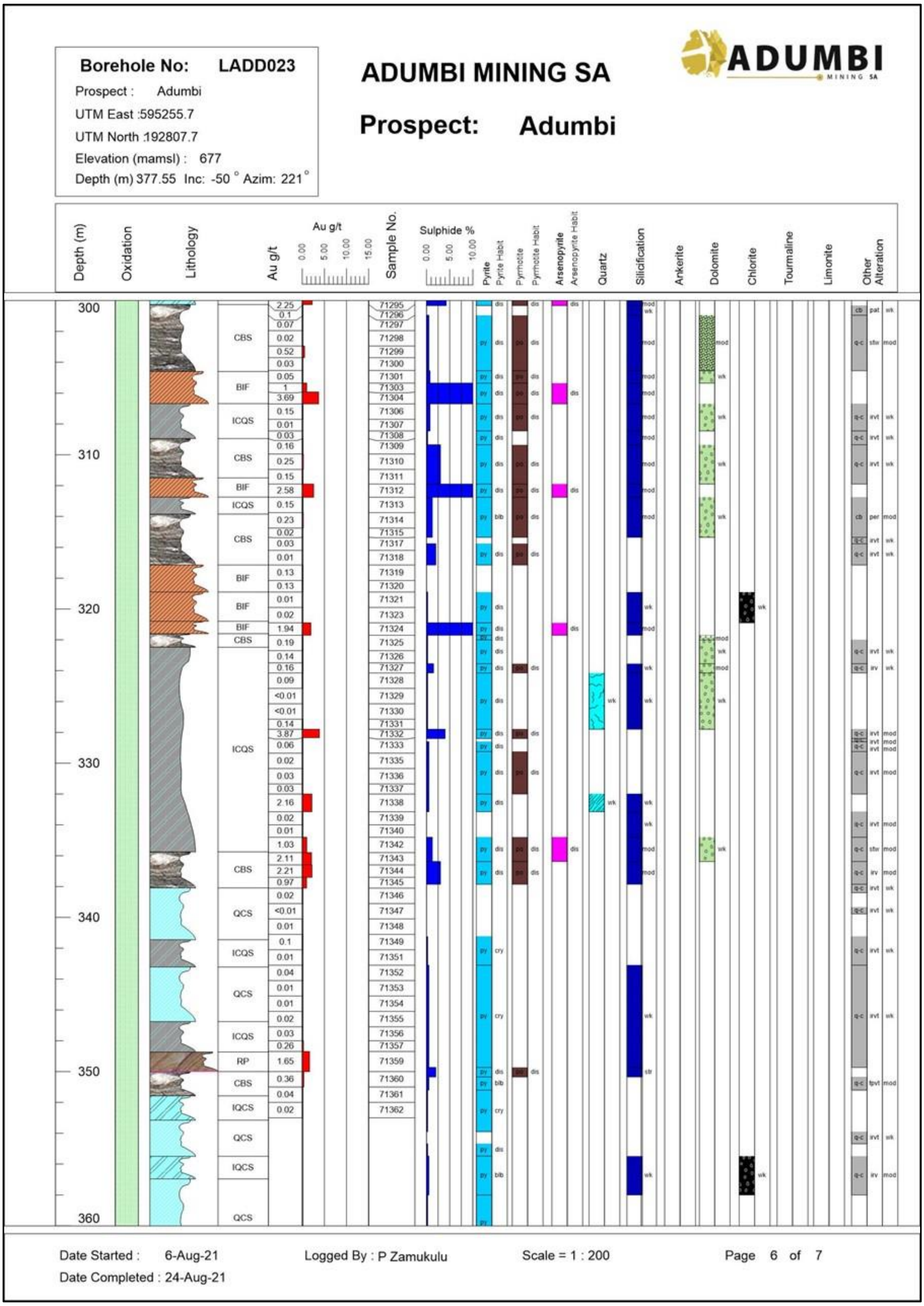


Figure 10.19: Strater Log for LADD023 – Page 6

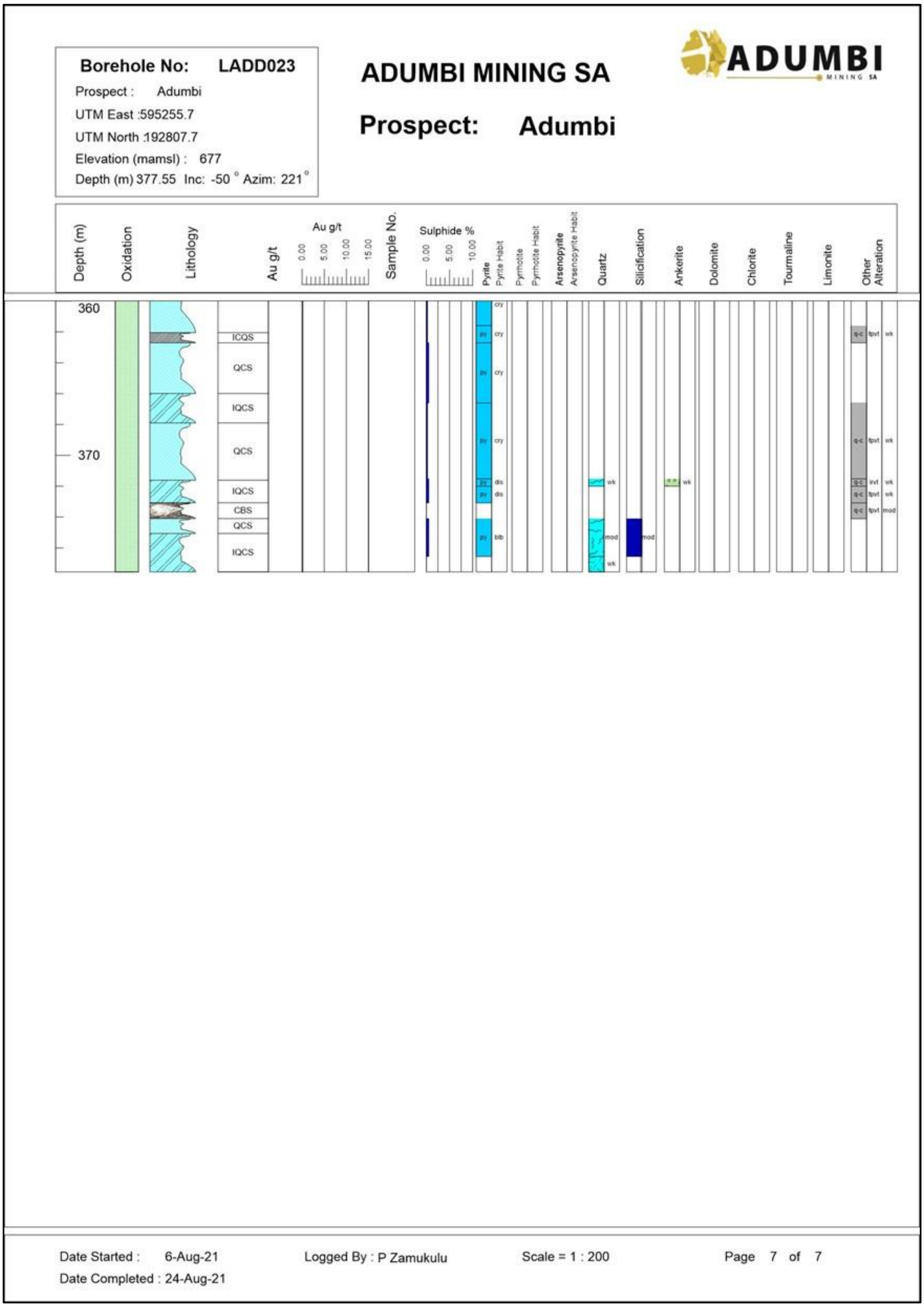


Figure 10.20: Strater Log for LADD023 – Page 7

A summary of the lithological units intercepted within the mineralised package and the composition of the alteration mineral assemblage of each drillhole is captured as presented in Table 10.9 for LADD025.

Table 10.9: Summary of the Lithological Units Intercepted within the Mineralised Package of LADD025 and Composition of the Alteration Mineral Assemblage

BHID	Lithology			Alteration and Mineralisation	Sampling		
	From (m)	To (m)	Code		From	To	No. of Samples
LADD025	146.35	158.65	CBS/QCS	Weakly pervasive silica, moderate irregular veins of quartz-carbonate, moderate patchy chlorite, weak patchy dolerite, 0.5 % disseminated pyrite, 0.25 % disseminated pyrrhotite	71364 71372	71370 71377	13
LADD025	158.65	171.00	QCS/CBS/BIF	Moderately pervasive silica, weak irregular veins of quartz-carbonate, weak patchy chlorite, 0.25 % disseminated pyrite, 0.25 % disseminated pyrrhotite	71379 71387 71394	71385 71392	14
LADD025	171.00	174.00	QCS	Moderately pervasive silica, weak patchy dolomite and chlorite	71395	71397	3
LADD025	174.00	177.9	QCS	Moderately pervasive silica, patchy chlorite, patchy ankerite, patchy dolomite, irregular veins of quartz, 0.5 % disseminated pyrite, 1 % disseminated pyrrhotite	71399 71520	71400 71521	4
LADD025	177.90	179.20	QCS	Moderately pervasive silica, strong irregular veins of quartz, weak patchy ankerite, 2.5 % disseminated pyrite	71523	71524	2
LADD025	179.20	180.80	QCS	Weakly pervasive silica, weak irregular veins of quartz-carbonate, weak patchy ankerite, 0.25 % disseminated pyrite	71525	71526	2
LADD025	180.80	190.52	QCS/CBS/BIF	Moderately pervasive silica, moderate irregular veins of quartz-carbonate, weak patchy chlorite, 1 % disseminated pyrite, 0.25 % disseminated pyrrhotite, traces of arsenopyrite	71527 71533	71531 71538	11
LADD025	201.55	205.17	QCS/CBS-AS	Moderate foliation parallel veins of quartz-carbonate, 1 % disseminated pyrite, 0.25 % disseminated pyrrhotite	71539 71541	71544	5

BHID	Lithology			Alteration and Mineralisation	Sampling		
	From (m)	To (m)	Code		From	To	No. of Samples
LADD025	211.52	215.75	QCS/BIF	Weakly pervasive silica, moderate irregular veins of quartz-carbonate, weak patchy ankerite and chlorite, 0.5 % disseminated pyrite, 0.25 % disseminated pyrrhotite	71545 71547	– 71550	5
LADD025	215.75	221.62	QCS/BIF	Weakly pervasive silica, weak irregular veins of quartz, moderate patchy chlorite, traces of pyrite and pyrrhotite	71551 71557	71555 –	6
LADD025	221.62	230.10	BIF/QCS	Moderately pervasive silica, moderate irregular veins of quartz-carbonate, weak patchy chlorite, 1 % disseminated pyrite, 1 % disseminated pyrrhotite, 0.25 % disseminated arsenopyrite	71558 71566	71564 71569	11
LADD025	230.10	236.42	IQCS/ BIF	Moderately pervasive silica, weak foliation parallel veins of quartz-carbonate, weak patchy chlorite	71570 71574	71572 71577	7
LADD025	236.42	252.30	BIF/QCS	Weakly pervasive silica, weak patchy dolomite, weak patchy chlorite, 0.75 % disseminated pyrite, 1 % disseminated pyrrhotite, traces of arsenopyrite	71578 71583 71593 71597	71581 71591 71595 71599	19
LADD025	252.30	254.05	IQCS	Weakly pervasive silica, moderate patchy dolomite, moderate patchy chlorite,	71600	71601	2
LADD025	254.05	263.50	QCS/BIF	Moderately pervasive silica, moderate irregular veins of quartz, weak patchy ankerite, dolomite and chlorite, 1.5 % disseminated pyrite, 2 % disseminated pyrrhotite, 0.5 % disseminated arsenopyrite	71603 71614	71612 71615	12
LADD025	263.50	266.00	RP	Strongly pervasive silica, weak patchy dolomite, weak patchy chlorite, 6 % disseminated pyrite, 4 % disseminated pyrrhotite, 3 % disseminated arsenopyrite	71617	71619	3
LADD025	266.00	277.57	QCS/CBS/BIF	Weakly pervasive silica, weak patchy dolomite, weak patchy chlorite, 0.25 % disseminated pyrite, 0.5 % disseminated pyrrhotite	71620 71630	71628 71634	14
LADD025	277.57	279.50	IQCS	Moderately pervasive silica, weak patchy dolomite, weak patchy chlorite, 0.5 % disseminated pyrite, 0.5 % disseminated pyrrhotite	71635	71636	2

BHID	Lithology			Alteration and Mineralisation	Sampling		
	From (m)	To (m)	Code		From	To	No. of Samples
LADD025	279.50	284.50	IQCS/CBS-AS/QCS	Moderately pervasive silica, weak patchy dolomite, weak patchy chlorite, weak irregular veins of quartz, 2 % disseminated pyrite, 1.25 % disseminated pyrrhotite,	71638	71643	6
LADD025	284.50	286.35	RP	Strongly pervasive silica, 3 % disseminated pyrite, 4 % disseminated pyrrhotite, 3 % disseminated arsenopyrite	71644 71646		2
LADD025	286.35	299.40	QCS/CBS/BIF	Moderately pervasive silica, weak patchy carbonate, weak patchy chlorite, weak irregular veins of quartz, 0.75 % disseminated pyrite, 1 % disseminated pyrrhotite	71647 71657 71665	71655 71663	17
LADD025	299.40	304.85	QCS/BIF/RP	Moderately pervasive silica, weak patchy carbonate, weak patchy chlorite, weak irregular veins of quartz, 2 % disseminated pyrite, 2 % disseminated pyrrhotite, 1.5 % disseminated arsenopyrite	71666 71672	71670 71673	7
LADD025	304.85	305.95	RP	Strongly pervasive silica, 5 % disseminated pyrite, 10 % disseminated pyrrhotite, 5 % disseminated arsenopyrite	71674		1
LADD025	305.95	310.95	QCS	Strongly pervasive silica, weak patchy carbonate, weak patchy chlorite, weak irregular veins of quartz, 1.5 % disseminated pyrite, 3 % disseminated pyrrhotite, 1 % disseminated arsenopyrite	71675 71680	71678 71681	6
LADD025	310.95	321.60	QCS/CBS-AS	Weak irregular veins of quartz-carbonate, weak patchy carbonate and dolomite, weak patchy chlorite, 1 % disseminated pyrite, 1 % disseminated pyrrhotite	71682 71690	71688 71693	11
LADD025	321.60	325.00	CBS-AS/RP	Strongly pervasive silica, 7 % disseminated pyrite, 4 % disseminated pyrrhotite, 5 % disseminated arsenopyrite	71695	71698	4
LADD025	325.00	333.95	QCS/CBS/BIF	Moderately pervasive silica, weak patchy dolomite, weak patchy chlorite, 1 % disseminated pyrite, 2.5 % disseminated pyrrhotite, 5 % disseminated arsenopyrite	71699 71703 71710	71701 71708	10

BHID	Lithology			Alteration and Mineralisation	Sampling		
	From (m)	To (m)	Code		From	To	No. of Samples
LADD025	333.95	336.20	RP	Strongly pervasive silica, weak patchy chlorite, 5 % disseminated pyrite, 10 % disseminated pyrrhotite, 2 % disseminated arsenopyrite	71711	71713	3
LADD025	336.20	342.65	CBS-AS/ICQS	Weakly pervasive silica, weak patchy dolomite, weak patchy chlorite, weak foliation parallel veins of quartz-carbonate, 1 % disseminated pyrite, 1.5 % disseminated pyrrhotite, 0.25 % disseminated arsenopyrite	71714 71720	71718 71722	8
LADD025	342.65	345.10	BIF/RP	Strongly pervasive silica, 2 % disseminated pyrite, 3.5 % disseminated pyrrhotite, 1.5 % disseminated arsenopyrite	71723	71725	3
LADD025	345.10	348.20	QCS/CBS	Weakly pervasive silica, weak patchy chlorite, weak foliation parallel veins of quartz, 0.25 % disseminated pyrite, 0.25 % disseminated pyrrhotite	71726	71729	4
LADD025	348.20	351.18	RP	Strongly pervasive silica, weak patchy dolomite, 7.5 % disseminated pyrite, 7.5 % disseminated pyrrhotite, 3.5 % disseminated arsenopyrite	71731 71734	71732	3
LADD025	351.18	355.20	BIF	Moderately pervasive silica, weak patchy chlorite, weak patchy dolomite, 3.5 % disseminated pyrite, 1.5 % disseminated pyrrhotite, 2 % disseminated arsenopyrite	71735	71739	5
LADD025	355.20	356.70	BIF	Weakly pervasive silica, 0.25 % disseminated pyrite, 0.5 % disseminated pyrrhotite	71740	71741	2
LADD025	356.70	359.90	QCS/BIF	Strongly pervasive silica, 3 % disseminated pyrite, 3 % disseminated pyrrhotite, 2 % disseminated arsenopyrite	71742	71745	4
LADD025	359.90	361.75	ICQS	Moderately pervasive silica, weak patchy chlorite, weak patchy dolomite, 0.25 % disseminated pyrite, 1 % disseminated pyrrhotite	71746 71748		2
LADD025	361.75	367.00	QCS/CBS	Weakly pervasive silica, weak irregular veins of quartz, weak patchy carbonate	71749	71754	6

Lithological units intersected in the completed 2020 to 2021 drillholes are mainly quartz carbonate schist (QCS) intercalated carbonaceous schist (CBS), banded iron formation (BIF) with or without QCS intercalations, and the RP zone. Sulphide mineralisation comprising pyrite, pyrrhotite and arsenopyrite in varying proportions within the mineralised zones is the main alteration mineral assemblage. Strong silicification and seldom weak chlorite are present. Massive dolerite was intersected in the footwall of the mineralisation in Drillhole LADD012.

Further descriptions of the individual rock types intersected in these drillholes are given below.

10.3.6.1 Quartz-Carbonate Schist (QCS)

Fine- to medium-grained, pale grey to pale greenish grey schist, comprising subrounded, dark grey quartz grains up to 1.5 mm (probably remnant clastic grains) in a finer-grained matrix of quartz, white mica, and carbonate (ankerite). The carbonate forms irregular, elongated grains orientated parallel to the foliation. It is the most abundant rock in the Adumbi sequence (see Figure 10.21).

It is interpreted that the rock was probably originally a poorly sorted, calcareous, muddy, fine-grained arenite, possibly a greywacke.

Figure 10.21: Quartz-Carbonate Schist, LADD014, 153.75 m to 154.00 m

10.3.6.2 Carbonaceous Schist (CBS)

Very fine-grained, dark grey to black schist, consisting of carbonaceous material and (according to petrographic data) varying amounts of white mica (see Figure 10.22). Quartz is rare. Banding due to variations in the proportion of white mica reflects the bedding in the original sediment. The nature of the carbonaceous material was not determined petrographically but based on samples of similar material from elsewhere in the Ngayu belt. It is probably amorphous carbon rather than graphite.

The rock was probably originally a black shale formed in a deep marine environment.

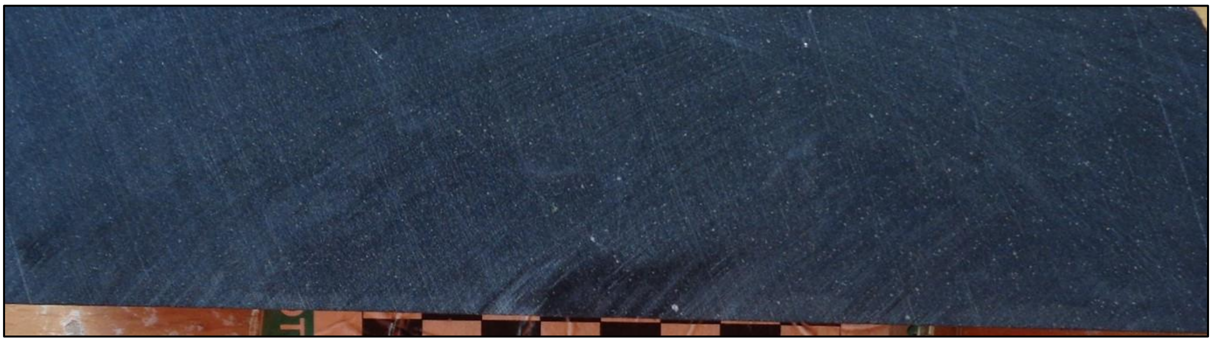


Figure 10.22: Carbonaceous Schist, LADD012, 767.07 m to 767.27 m

10.3.6.3 Banded Iron Formation (BIF)

The BIF consists of black, fine-grained magnetite-rich bands alternating with white to pale buff chert. The width of the magnetite bands is variable, ranging from laminae only a few millimetres wide, to bands up to approximately 10 cm across (see Figure 10.23).

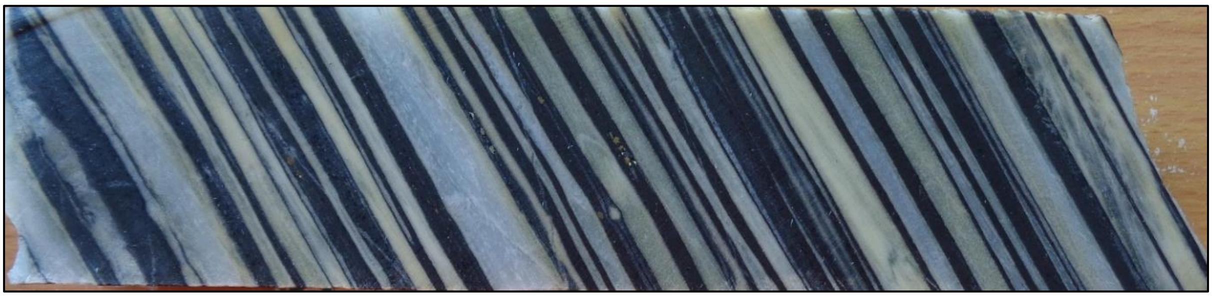


Figure 10.23: Banded Iron Formation, LADD013, 378.67 m to 378.87 m

10.3.6.4 Dolerite

Mafic intrusive rock, massive (not deformed), dark greenish in colour, medium-grained with localised irregular veins and veinlets of quartz-carbonate, intersected in the footwall of the mineralisation in Drillhole LADD012 (see Figure 10.24).

Figure 10.24: Dolerite, LADD012, from 939.20 m to 939.31 m

10.3.6.5 Replaced Rock (RP)

The RP zone is believed to have resulted from a highly intense hydrothermal alteration. As observed in other drillholes at Adumbi, the main hydrothermally altered zones are associated with the BIF. The alteration assemblages vary in style and intensity, progressing from the distal to the proximal. These have been intercepted in Drillhole LADD013 as shown in Figure 10.25, Figure 10.26 and Figure 10.27. The three stages of hydrothermal alteration in the BIF unit in increasing order of intensity are described below.

Stage 1

Carbonate (ankerite) replaces the magnetite bands in the BIF. The bands assume a pale brownish orange colour and become weakly to non-magnetic (see Figure 10.25).

Figure 10.25: Distal Alteration: Ankerite Replacement of Magnetite, LADD013, 430.97 m to 431.12 m

Stage 2

Pyrite ± pyrrhotite ± arsenopyrite aggregates largely replace the magnetite bands, together with quartz ± ankerite (see Figure 10.26). The proportions of the sulphides vary significantly, often over short distances of a few centimetres; pyrite is usually present but may be subordinate to any pyrrhotite and arsenopyrite present. The chert bands appear to have undergone some recrystallisation and show patchy ankerite alteration; sulphides may be present locally although in much smaller amounts than in the replaced magnetite bands. The original banding of the BIF can still be discerned.

Figure 10.26: Magnetite Bands Totally Replaced by Sulphides and Quartz, LADD013, 426.18 m to 426.33 m

Stage 3

Total recrystallisation and replacement of the BIF form an assemblage of sulphides and quartz (see Figure 10.27). Banding in the BIF is destroyed, and the original rock is unrecognisable. The proportion of sulphide varies, but averages approximately 26 %. The sulphide species are pyrite, pyrrhotite and arsenopyrite, although the ratios vary significantly. This proximal assemblage is logged separately as the RP Zone, and it is generally associated with higher gold grades.

Figure 10.27: Proximal Alteration resulting in Complete Recrystallisation and Replacement of the BIF, LADD013, 425.07 m to 425.23 m

Figure 10.28 is a surface geological map showing traces of the cross-section lines through Drillholes LADD009 (AıA), LADD012 (BıB) and LADD025 CıC).

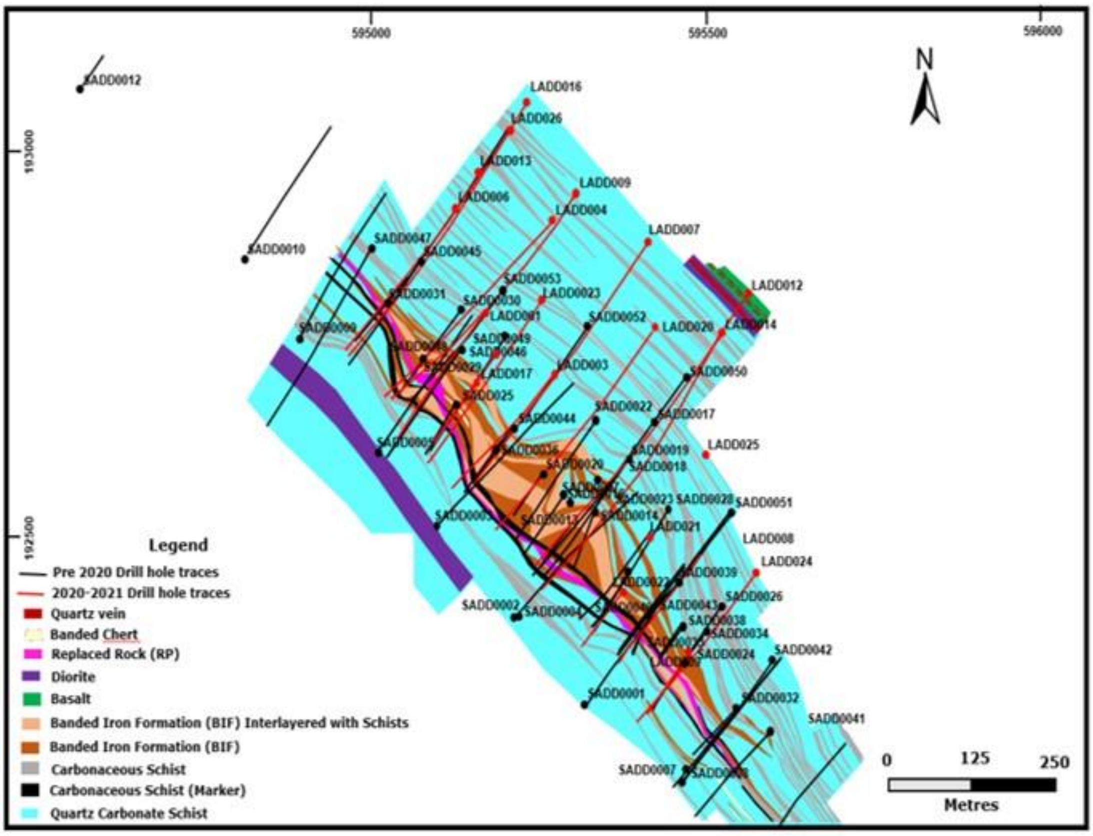


Figure 10.28: Adumbi Surface Geology Showing Section Lines through LADD009, LADD012 and LADD025

Typical drillhole cross sections through LAD009, LADD012 and LADD025 are shown in Figure 10.29, Figure 10.30 and Figure 10.31, respectively.

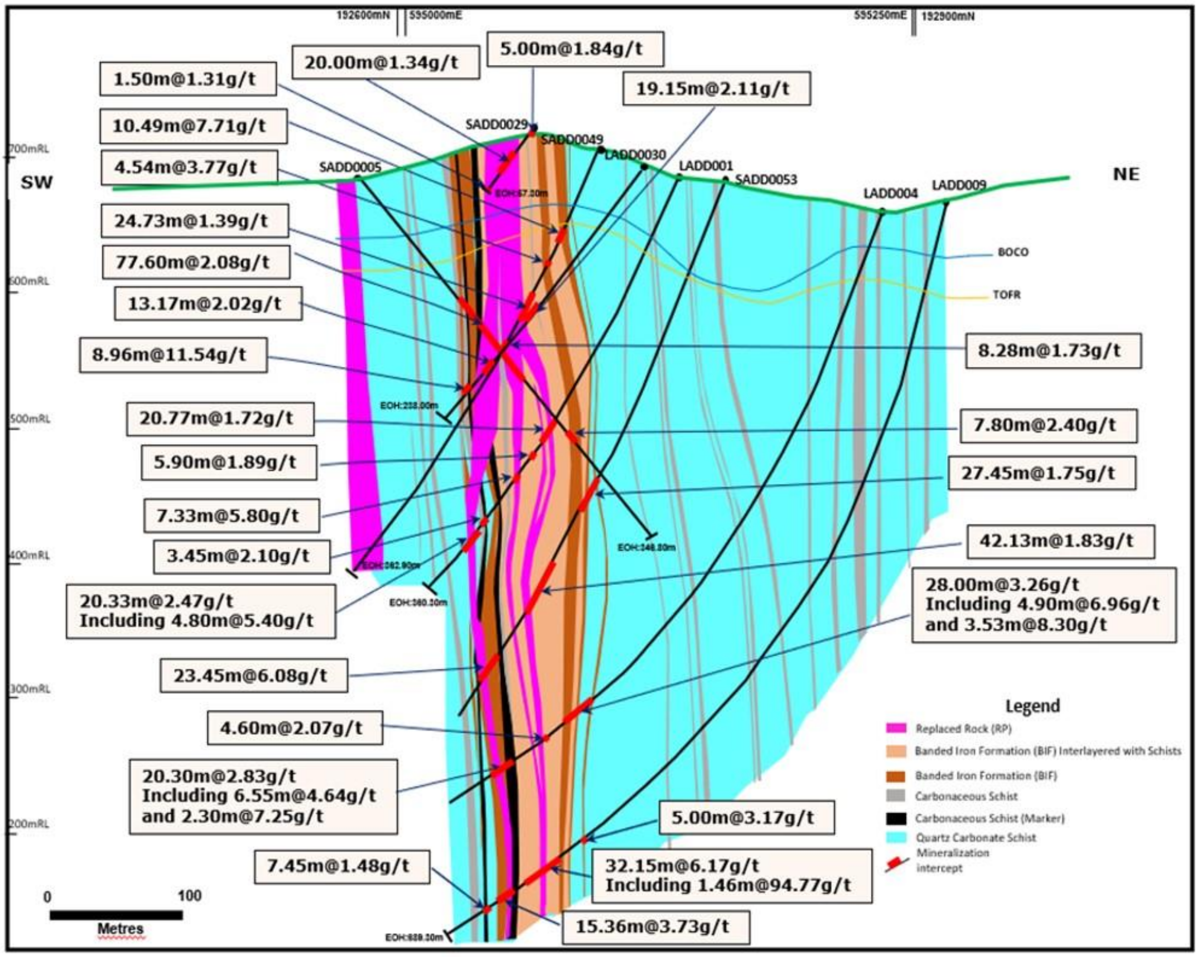


Figure 10.29: Cross Section through Drillhole LADD009

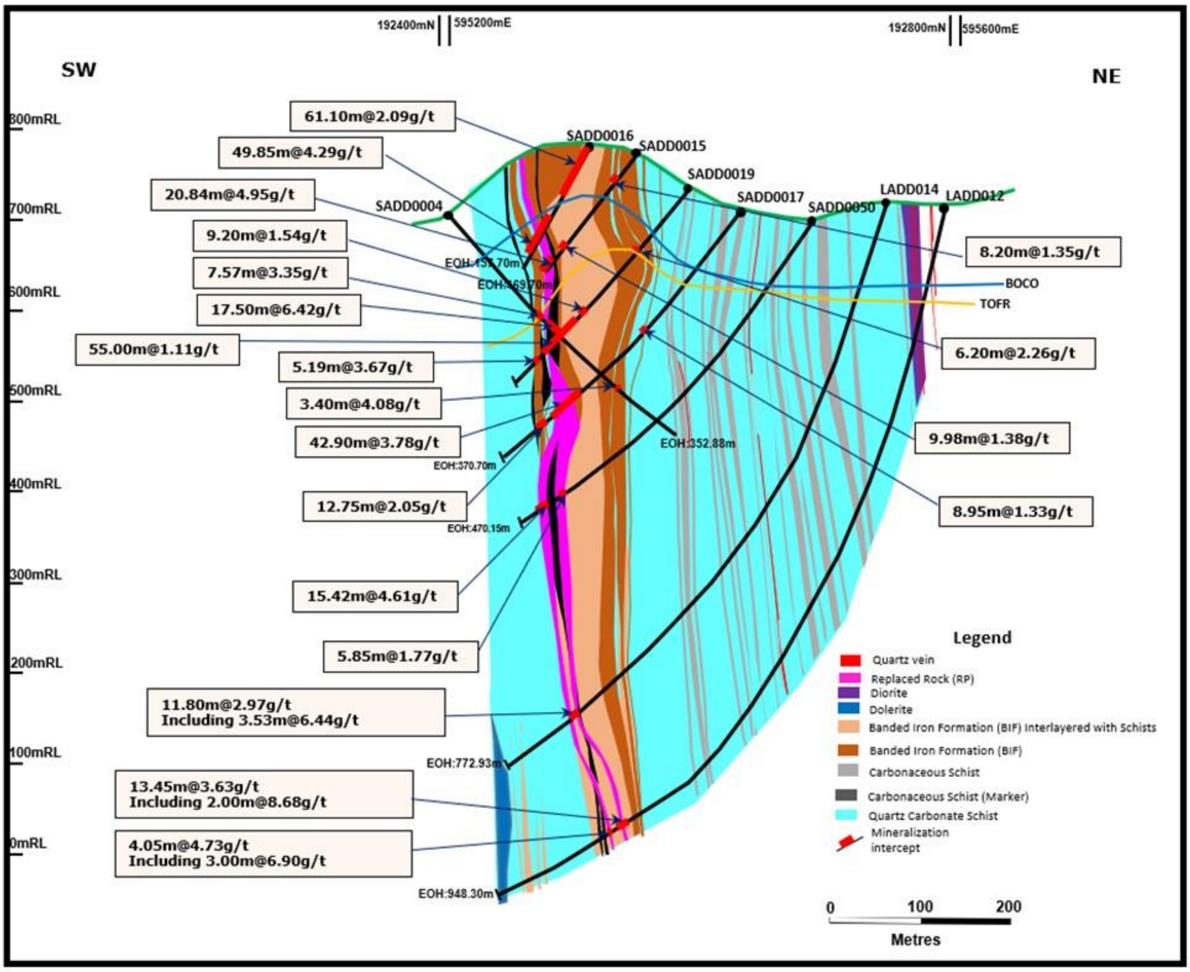


Figure 10.30: Cross Section through Drillhole LADD012

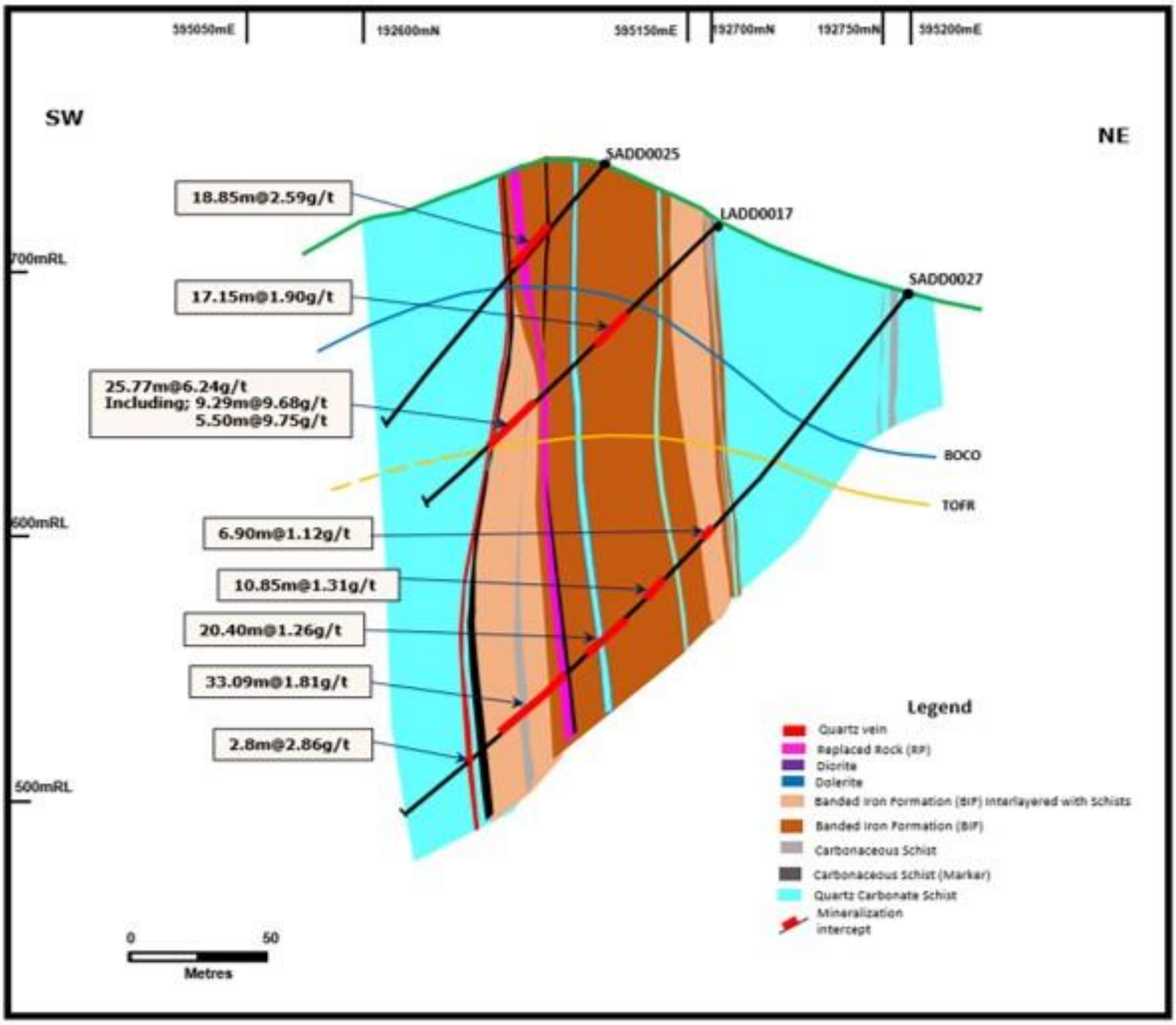


Figure 10.31: Cross Section through Drillhole LADD025

10.3.7 Core Photography

After logging, the core is photographed before cutting and sampling. An improvised fixed environment (see Figure 10.32) has been fabricated to facilitate core photography. This comprises a wooden box (1.2 m high) designed with the length and width being a few centimetres longer and wider than the size of the core tray, painted all white inside and fitted with fluorescent lights to the roof. A small hole (the size of the lens of a digital camera) is created at the top of the box and fitted with a digital camera at 1 m height from the base where the core tray slots in.

Prior to photographing, the entire core is made wet to enhance the picture quality. Once the core tray is slotted into the box, the door is closed and the light switched on. The pre-set digital camera is switched on from the top and the photograph taken. This is done so that all the core photographs are taken under the same conditions and from a fixed height to enhance standard quality and merging for future digital logging.

Figure 10.32: Improvised Fixed Environment for Core Photography

Each photograph covers one box of core, and the core is oriented in such a way that the metre marks and the BOHL are displayed.

Each photograph is saved on computer using the borehole number, tray, and from-to depths as the file name, e.g. LADD001-20-72-77 m.

10.3.8 Geotechnical Logging

Following Minecon's recommendation, geotechnical logging of the current drill core was undertaken by a dedicated senior geotechnical geologist. It is worth stating that all previous oriented core, including the four deep holes of 2017, was not logged geotechnically. However, it is appropriate to carry out geotechnical logging on the core before it is cut and sampled as such information is crucial for the PEA and future feasibility studies, in particular, when it comes to the determination of pit slopes and other engineering studies.

The geotechnical logging was carried out according to the system employed by Steffen Robertson and Kirsten (SRK) UK. The data is stored on an MS Access database designed by SRK, which enables calculations such as rock mass rating (RMR) to be easily made. All the completed 2020 to 2021 drillholes were geotechnically logged.

Table 10.10 and Table 10.11 summarise the geotechnical information captured from LADD001.

Table 10.10: Summary of Geotechnical Log of Drillhole LADD001 along Depth

BHID	From (m)	To (m)	Description by Crossing Depth
LADD001	0.00	1.00	Very moist, moderate brown soft intact, fine-grained hill wash material; transported
LADD001	1.00	25.90	Slightly moist, light grey, firm dense, fine- to medium-grained showing texture of bed rock, but highly weathered and completely oxidised with soil properties
LADD001	25.90	59.67	Light to dark grey, fine-grained units, moderately weathered, highly discoloured, medium-hard rock units with hardness of approximately 50 MPa and moderately fractured
LADD001	59.67	69.55	Dark grey units, fine-grained, slightly weathered and fractured, but showing difference from the fresh rock strength with hardness of approximately 100 MPa
LADD001	69.55	360.30	Unweathered intact rock units, very slightly fractured, no sign of staining, hard rock units generally above 150 MPa

Table 10.11: Hardness of Lithological Units

Code	Unit	Hardness Description of major Lithological Unit
QCS	Quartz Carbonate Schist	Strong and hard unit in original condition, estimated hardness approximately 200 MPa
BIF	Banded Iron Formation	Very strong when it is silicified and less fractured, hardness approximately 150 MPa
CBS	Carbonaceous Schist	Not very hard unless it has undergone strong silicification, which is usually observed in the area (hardness above 100 MPa) on fresh unit
QV	Quartz Vein	Strong small unit rarely banded with other units, hardness above 200 MPa when it is not fractured
CS	Chlorite Schist	The unit is strong when silicified, estimated hardness approximately 150 MPa
IQCS	Intercalation QCS and CBS	The unit is bedded with intercalation of two units (QCS and CBS), also hard when silicified, hardness approximately 200 MPa
RP	Replaced Rock	Hard unit and commonly highly silicified, estimated hardness above 200 MPa when not fractured

The rock units are slightly fractured (jointed), and most of the joints are cemented in the fresh zones, which makes the RMR value higher in unweathered zones.

The values of the hardness presented in Table 10.11 are based on field estimations by using a penknife, carbide scribe pen, and a geological hammer. Therefore, a uniaxial compressive strength (UCS) test will be useful for the standardisation of hardness of the rock units. As the geotechnical logging progresses, representative samples will be collected and recommended for the UCS test.

Table 10.12 is the automatically generated summary of the RMR report for LADD001. It is worth noting that the mining rock mass rating (MRMR) system and the mining adjustments highlighted in the table are the parameters applied in mining and depend on complex adjustments that cannot be defined based on only core logging.

Table 10.12: RMR Report for LADD001

Lithology	Development MRMR							Downhole	Mining Adjustments						
	Weathering	Min.	Average	Max.	Min.	Average	Max.	m	Downhole (m)	Weathering (%)	Orientation (%)	Stress (%)	Blasting (%)	Water (%)	Adjustment (%)
Banded Iron Formation	UW	52	**67**	100	0	**0**	0	58.67	16.0	0	0	0	0	0	0
Carbonaceous Schist	MW	23	**29**	37	0	**0**	0	22.25	6.1	0	0	0	0	0	0
Carbonaceous Schist	SW	30	**32**	39	0	**0**	0	3.23	0.9	0	0	0	0	0	0
Carbonaceous Schist	UW	39	**59**	100	0	**0**	0	25.14	6.9	0	0	0	0	0	0
Chlorite Schist	UW	60	**60**	60	0	**0**	0	1.71	0.5	0	0	0	0	0	0
Interbedded QCS and CBS	MW	30	**33**	43	0	**0**	0	4.90	1.3	0	0	0	0	0	0
Interbedded QCS and CBS	SW	30	**32**	33	0	**0**	0	4.50	1.2	0	0	0	0	0	0
Interbedded QCS and CBS	UW	37	**55**	70	0	**0**	0	41.80	11.4	0	0	0	0	0	0
Quartz Carbonate Schist	MW	27	**32**	37	0	**0**	0	7.93	2.2	0	0	0	0	0	0
Quartz Carbonate Schist	SW	27	**27**	27	0	**0**	0	2.15	0.6	0	0	0	0	0	0
Quartz Carbonate Schist	UW	40	**63**	100	0	**0**	0	149.99	40.9	0	0	0	0	0	0
Quartz Vein	UW	96	**99**	100	0	**0**	0	2.88	0.8	0	0	0	0	0	0
Replaced Rock	UW	56	**71**	100	0	**0**	0	17.02	4.6	0	0	0	0	0	0

MW Moderately Weathered
SW Strongly Weathered
UW Unweathered

10.3.9 Core Cutting and Sampling

After logging, the geologist marks a line with a chinagraph pencil approximately 3 mm to the left of the BOHL in the downhole direction along which core cutting is done (see Figure 10.33). One-metre sample lengths (adjusted for lithology) are marked on the core in the mineralised horizons during logging. In homogeneous rock, the maximum sample interval is 1 m. The minimum sample interval is 0.3 m. The sample depths for each sample are entered into a sample ticket book, which contains removable duplicate sample ticket tags. The core sample numbers and sample intervals are written onto pre-printed diamond drill log forms. Each marked sample is split along its length by trained staff using a dedicated drill core diamond saw (see Figure 10.34). The core is broken at the sample position marks using a geological pick. The sampling lengths are reduced, when necessary (e.g. where lithological contacts or core size changes are encountered), with the bottom/top end of the sample being approximately 2 cm from the contact.

Figure 10.33: Marked Line in Red along which Core cutting is Done

Figure 10.34: Adumbi Mining Staff cutting Core from LADD001

One half of the core is replaced in the core tray, and the remaining half is placed in a plastic sample bag, in which the sample number is folded in along the open end of the bag, which is then closed using a stapler (see Figure 10.35). Sample tags are placed in the core tray at the position of the bottom end where the sample is obtained. A brick is sawn ("brick cleaning") after each sample has been split to ensure that no cross-contamination takes place between samples.

Figure 10.35: Senior Geologist Sampling Core from LADD001

All the core samples collected are sent to the on-site sample preparation laboratory for pre-assay, after which 150 g of the pulverised material are placed in sample packets and shipped to the SGS Laboratory in Mwanza for wet chemistry assaying.

10.4 2020 TO 2021 DRILLING – MAMBO BADO

Mambo Bado 1 is located approximately at 1.5 km NW of the Adumbi deposit. Rock chip/channel samples collected from quartz veins and sheared metasediments returned very encouraging results as displayed in the surface map in Figure 10.36.

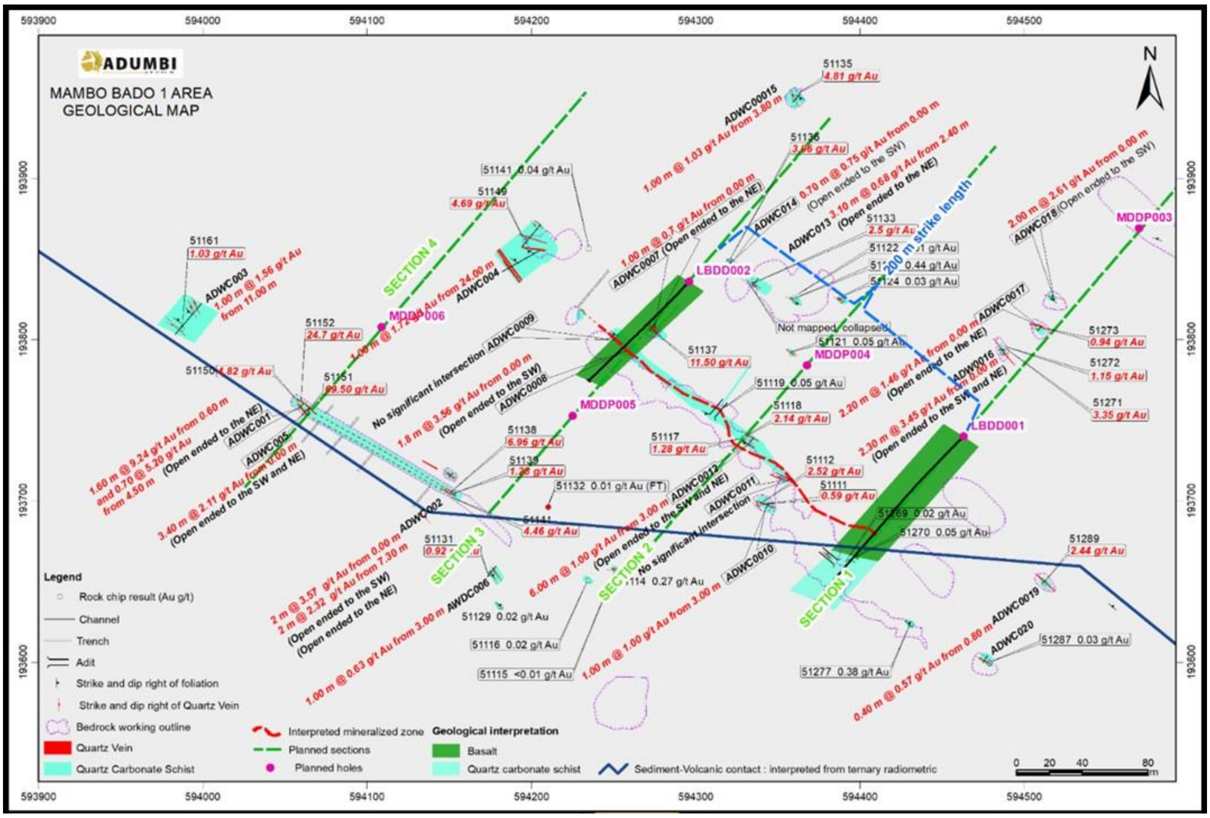


Figure 10.36: Mambo Bado Plan Map showing Location of Planned Drillholes, Channel and Bedrock Workings

A preliminary interpretation based on the surface information seemed to point to a series of parallel NW-SE mineralised zones with variable widths. It was envisaged that drilling to test these mineralised trends would aid in understanding the subsurface geology as well as ascertain the mineralisation potential of this prospect area.

Based on the above encouraging results and the surface structural interpretation, a shallow diamond drilling programme on four sections was proposed to test the subsurface mineralisation (see Table 10.13).

Table 10.13: Mambo Bado Planned Drillholes

Section	BHID	Easting	Northing	Azimuth (°)	Inclination (°)	Vertical Distance (m) to Mineralisation	EOH (m)	Targeted Mineralisation	Comments
Section 1	MDDP001	594463	193740	220	−55	55	130	8.40 m at 2.06 g/t in IWCH006	Open ended to both sides (NE and SW) along the main artisanal workings
								1.00 m at 1.00 g/t in ADWC010	Projected along the main artisanal workings
	MDDP003	594570	193869	220	−55	58	110	3.50 m at 0.96 g/t, including 2.20 m at 1.46 g/t in ADWC017	Open to the NE
								3.00 m at 1.78 g/t, including 2.00 m at 2.61 g/t in AWC018	Open to the SW
Section 2	MDDP004	594367	193785	220	−55	58	130	6.00 m at 3.62 g/t, including 1.00 m at 9.10 g/t in ADWC012	Open ended to both sides (NE and SW). Localised at 100 m NW of Section 1
Section 3	MDDP002	594296	193836	220	−55	60	95	1.80 m at 3.57 g/t in ADWC008	Open to the SW. Localised at 90 m NW of Section 2
								Soil anomaly up to 216 ppb	
	MDDP005	594215	193761	220	−55	79	115	6.00 m at 1.98 g/t, including 2.00 m at 3.5 g/t and 2.00 m at 2.32 g/t in ADWC002	Open to both sides (NE and SW)
								Rock chips grading 6.96 g/t and 4.46 g/t	
								Soil anomaly up to 103 ppb	
Section 4	MDDP006	594109	193805	220	−55	62	100	3.40 m at 2.11 g/t in ADWC005	Open to both sides (NE and SW)
								4.60 m at 4.05 g/t, including 1.60 m at 9.24 g/t and 0.70 m at 5.20 g/t in ADWC001	
								Rock chips grading 69.5 g/t, 24.70 g/t and 4.82 g/t	
								Soil anomaly up to 124 ppb	

Drilling was planned to initially test two sections (± 200 m apart on the NE-SW trend) by drilling holes MDDP001 and MDDP002. An orientation survey was planned to be conducted on competent cores to aid in the structural interpretation of the subsurface geology.

While the drilling contractor Orezone Drilling was awaiting additional HQ rods to continue drilling the deeper core holes at Adumbi, the Atlas Copco rig (CS14 Rig 2) was moved to drill the first two shallow holes at Mambo Bado. Table 10.14 presents the collars of the completed drillholes.

Two main lithological units were intersected: a greenish metavolcanic rock (possibly basalt), and metasedimentary rock (QCS). The holes started in a reddish, fine-grained, massive to weakly foliated undifferentiated saprolite (possibly after metavolcanic rock), containing weak irregular veinlets and veins of quartz, weakly patchy limonite-silica, and 0.25 % boxworks. Artisanal miners are busy exploiting the quartz veinlets within this saprolite.

Table 10.14: Drill Collars of Mambo Bado Completed Boreholes

BHID	Prospect	Easting (m)	Northing (m)	RL (m)	Azimuth (°)	Inclination (°)	End Depth (m)
LBDD001	Mambo Bado	594463	193740	671	220	−55	218.7 0
LBDD002	Mambo Bado	594296	193836	678	220	−55	143.50
Total							**362.20**

Assay results from drillholes LBDD001 and LBDD002 did not return encouraging results. The significant results from LBDD002 are presented in Table 10.15.

Table 10.15: Significant Mineralised Intercepts from Drillhole LBDD002

Borehole Number	From (m)	To (m)	Intersected Width (m)	Grade (g/t Au)
LBDD002	23.70	24.70	1.00	4.10
LBDD002	111.90	113.90	2.00	1.61

Based on these initial drilling results, Mambo Bado has been downgraded, and no further drilling is planned.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE PREPARATION AND ANALYSES

The sample preparation and analyses for samples from 2010 to 2013, which were undertaken by the ALS Chemex laboratory, have been outlined in the RPA 2014 NI 43-101 Technical Report.

During the 2014 to 2017 exploration activity, sample preparation and analyses were outsourced to the SGS laboratory in Mwanza, Tanzania (which is independent of Loncor). The SGS laboratory operates a quality system that is accredited in accordance with ISO/IEC 17025:2017 and SANAS (South African National Accreditation System). The SGS laboratory acted as an umpire laboratory even while ALS Chemex was the principal laboratory; hence, correlational studies between the two laboratories have been undertaken.

For the 2020 to 2021 drilling programme carried out by Loncor, the ALS Chemex sample preparation facility on site was recommissioned by Minecon's laboratory technical team and used for sample preparation. A full description of the laboratory has been outlined in the RPA 2014 NI 43-101 Technical Report.

Minecon's laboratory management personnel have been on site to render training to Loncor's laboratory staff and provide management services to the laboratory facility, and have continued to manage the facility from the recommission date in October 2020 to date. The laboratory is running efficiently and according to standard guidelines. Laboratory procedures have been documented and reviewed by Minecon senior management, and internal quality control measures have been taken. Based on the documentation and discussions with the laboratory management, Minecon's senior management does not have any concerns regarding the sample preparation for all Loncor samples.

Sample pulps are sent for analyses to SGS Mwanza, which serves as the primary laboratory. SGS is internationally accredited and utilises conventional sample preparation, sample analysis and associated quality control protocols.

11.1.1 Sample Preparation Procedure

Following from the Minecon April 17, 2020, NI 43-101 Technical Report, Minecon made some recommendations. One such recommendation was that "*The Company should consider re-using the on-site sample preparation laboratory, which has been lying idle for some years since it will help with enforcing stricter QA/QC policing on the analytical laboratory, standards and ordinary samples will be in the same matrix thus making it more difficult for an external laboratory to detect it. Issues of duplicates will be better handled with a sample preparation laboratory. Some concerns about shortage of samples for other important studies could as well be managed as both coarse and pulp rejects in addition to the half or quarter cores will be available for use.*"

In managing the 2020 to 2021 exploration programme, Loncor agreed to recommission the on-site sample preparation laboratory. A full description of the on-site sample preparation laboratory has been outlined in the RPA 2014 NI 43-101 Technical Report.

The key objective of the sample preparation laboratory is to ensure the prompt operation of a laboratory that processes samples to international standards using the best-known procedures and protocols and to ensure that adequate controls are in place for the effective and efficient operation of the facility.

The sample preparation laboratory is equipped with the necessary key sample preparation equipment, which together with the right procedures produce quality pulverised samples. Personnel with sample preparation experience have been recruited to operate the sample preparation laboratory. The pulps are dispatched to the SGS analytical laboratory in Mwanza for analysis. Producing pulps with a good turnaround time coupled with the reduction in cost of transporting whole samples, ensuring better QA/QC policing of the analytical laboratory, have all significantly impacted the efficiency of the exploration programme positively.

The SGS laboratory in Mwanza has sample preparation and analysis sections, which utilise the SGS standard procedures and control. SGS uses a laboratory information management system (LIMS) and has a QLAB system that is directly connected to the SGS laboratory network via the SGS laboratory information management system (SLIMS), which is used by SGS laboratories globally to generate client-specific reports and is the backbone of the SGS laboratory management and quality management systems. Typical samples sent to the SGS laboratory are accompanied by a sample submission form, which contains at least the following information:

- Company name and complete address
- Contact name
- Details for distribution of reports and invoices
- Method codes
- Instructions on sample preparation
- List or range of sample numbers

Once the samples are received at the SGS laboratory, the samples go through checking and reconciliation procedures, as listed below, followed by the SGS sample preparation procedure (SGS Code PRP87). The complete process includes the following:

- Checking samples
- Preparing sample reconciliation forms, which are sent to Loncor to confirm the quantities of samples received
- Weighing samples
- Drying field samples
- Crushing samples to 75 % passing 2 mm
- Splitting 1.5 kg by riffle splitter
- Pulverising 1.5 kg of 2 mm material to 90 % passing 75 µm in a ring and puck pulveriser
- Returning coarse and pulp rejects to Loncor upon request

Half of the drill core from Adumbi was sent to the SGS Mwanza laboratory while the other half core was stored at Loncor's core storage facility on site.

11.1.2 Sample Analysis

Drill core, trench, adit, pit, rock chip and channel samples were analysed for gold at the SGS Mwanza laboratory using fire assay (FA) with flame atomic absorption spectrometry (AAS) to measure the gold (SGS Code FAA505), and the analyses were carried out on 50 g aliquots. The effective range for FAA505 is 0.01 ppm Au to 100 ppm Au. In addition, check assays were carried out by the screen fire assay method to verify higher-grade sample assays obtained by fire assay. Internationally recognised standards and blanks were inserted at the Adumbi sample preparation laboratory as part of internal QA/QC analytical procedures.

The pre-2014 sample analysis procedure by ALS Chemex Laboratory was described in the RPA 2014 NI 43-101 Technical Report.

11.1.3 BLEG Samples

All BLEG samples were sent to ALS Minerals in Ireland for analysis. The original and duplicate BLEG samples were assayed as follows:

- No additional sample preparation was required.
- Au, Ag, Cu and Pd were assayed by conducting a cyanide leach bottle roll test on up to 1 kg, with reporting limits for Au of 0.0001 ppm to 10 ppm (ALS Method: Au-CN12).
- A suite of 53 elements was assayed by aqua regia digestion of 0.5 g of the sample, and analysed by ICP-MS and ICP-AES (ALS Method: ME-MS41L).

11.1.4 Stream Sediments

The original and duplicate samples were dried and disaggregated at the camp, and were submitted to the laboratory for analysis as follows:

- Sieve to −180 micron (80 mesh).
- Conduct a fire assay of the −180 micron (80 mesh) fraction for Au, using a 50 g charge (ALS Method: Au-AA24).
- Conduct a test for a suite of 53 elements by aqua regia digestion of 0.5 g of the sample, and analysis by ICP-MS and ICP-AES (ALS Method: ME-MS41L).

11.2 QUALITY ASSURANCE AND QUALITY CONTROL

Quality assurance (QA) consists of evidence to demonstrate that the assay data has precision and accuracy within generally accepted limits for the sampling and analytical method(s) used to have confidence in the resource estimations. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of sampling, preparing and assaying exploration samples.

In general, QA/QC programmes are designed to prevent or detect contamination and allow analytical precision and accuracy to be quantified. In addition, a QA/QC programme can identify the overall sampling and assaying variability of the sampling method itself. The programme can also determine the reporting accuracy for clerical and data transfer errors.

Accuracy is assessed by reviewing assays of commercially available certified reference material (CRM) or in-house standards where available, and by check assaying at external alternative accredited laboratories (referee, umpire, or check samples). Precision or repeatability is assessed by processing duplicate samples from each stage of the analytical process from the primary stage of sample splitting, through the sample preparation stages of crushing/splitting, pulverising/splitting, and assaying. Control samples can also help identify possible mix-ups or mislabels during sample preparation.

11.2.1 QA/QC Programme

Minecon has reviewed the QA/QC results for the Imbo Project, which includes the Adumbi, Kitenge and Manzako deposits. Kilo followed an industry-standard QA/QC programme with the regular submission of blanks and CRMs (standards) into the sample stream. However, there were no records of duplicates.

RPA, in their study in 2014, reviewed the QA/QC of the project data from 2010 to 2013, involving 33,230 field samples made up of adit, trench and drillhole samples, and provided a comprehensive report in their 2014 NI 43-101 Technical Report. The database included a total of 163 drillholes totalling 34.32 km of drilling. A summary of the QA/QC as provided by RPA is shown in Table 11.1.

Table 11.1: Summary of RPA 2014 QA/QC Review of the Database

Blanks		Field Duplicates	CRMs (Standards)		Referee Samples
Number	Number or Percentage of Failures	Number	Number	Number or Percentage of Failures	Number
1,107	5 or 0.5 %	139	858	82 or 10 %	296

RPA made the following comments in their 2014 NI 43-101 Technical Report:

"RPA considers an overall CRM (Commercial Reference Material or Standard) failure rate of 2% to be acceptable. The Kilo inserted CRMs have a 10% failure rate which raises serious concerns with regard to precision at the assay laboratory and/or inadequate homogenization of the commercial standard. The CRM failures have not been re-assayed. RPA recommends that if a batch of samples has a CRM failure rate of over 2%, it should be re-assayed as a whole. In addition, RPA recommends that greater care be taken when naming a standard and sufficient material is supplied for assay."

Kilo, as part of the 2014 exploration programme, followed up on RPA's recommendations.

The standards and blanks results were interrogated with a view to identifying analytical batches or parts of batches that had failed the QC criteria and warranted re-assay. The failed samples from Adumbi were then prioritised, and re-assays were carried out at the SGS Mwanza laboratory.

This section of the report describes the QC criteria adopted by the exploration team and presents the re-assay results for Adumbi and discusses its implications. Table 11.2 provides a summary of the drill core samples, standards and blanks submitted for assay from the Adumbi, Kitenge and Manzako deposits in the pre-2013 drilling programme. Table 11.3 shows the standards submitted with the Kilo drill core samples.

Table 11.2: Summary of Drill Core Samples, Standards and Blanks Submitted for Assay from the Adumbi, Kitenge and Manzako Deposits

Deposit	Samples	Standards		Blanks	
		No.	%	No.	%
Adumbi	9,121	221	2.4	338	3.7
Kitenge	12,141	402	3.3	495	4.1
Manzako	7,176	230	3.2	265	3.7
Total	**28,438**	**853**	**3.0**	**1,098**	**3.9**

Table 11.3: Standards Submitted with Kilo Drill Core Samples

Standard	Au Grade (ppm)	Deposit
OxE101	0.607	Adumbi, Kitenge, Manzako
OxE74	0.651	Adumbi, Kitenge, Manzako
OxJ95	2.337	Kitenge, Manzako
OxJ64	2.366	Adumbi, Kitenge, Manzako
SJ39	2.641	Kitenge, Manzako
OxL93	5.841	Adumbi, Kitenge, Manzako
OxL63	5.865	Adumbi, Kitenge, Manzako
OxN49	7.635	Adumbi

To determine whether an analytical result for a particular standard lies within acceptable limits, data was inserted into an MS Excel spreadsheet dedicated to that standard. A standard control sheet, unique for each standard, generates charts based on control limits defined on the same general basis. The control limits are defined as 3 × SD (med mr) above and below the mean. The “SD (med mr)” is the standard deviation based on the median moving range and provides a more robust estimate than other straight standard deviation calculations.

Every laboratory-reported grade for an inserted standard is plotted on the standard control sheet that corresponds to the specific standard.

The standard control sheet shows the standard assay results and control limits in graph format, as shown in Figure 11.1. Standards that fall outside the defined tolerance are considered to have failed. In this performance chart, the last two samples can be seen to plot outside the control limits indicated by the red lines.

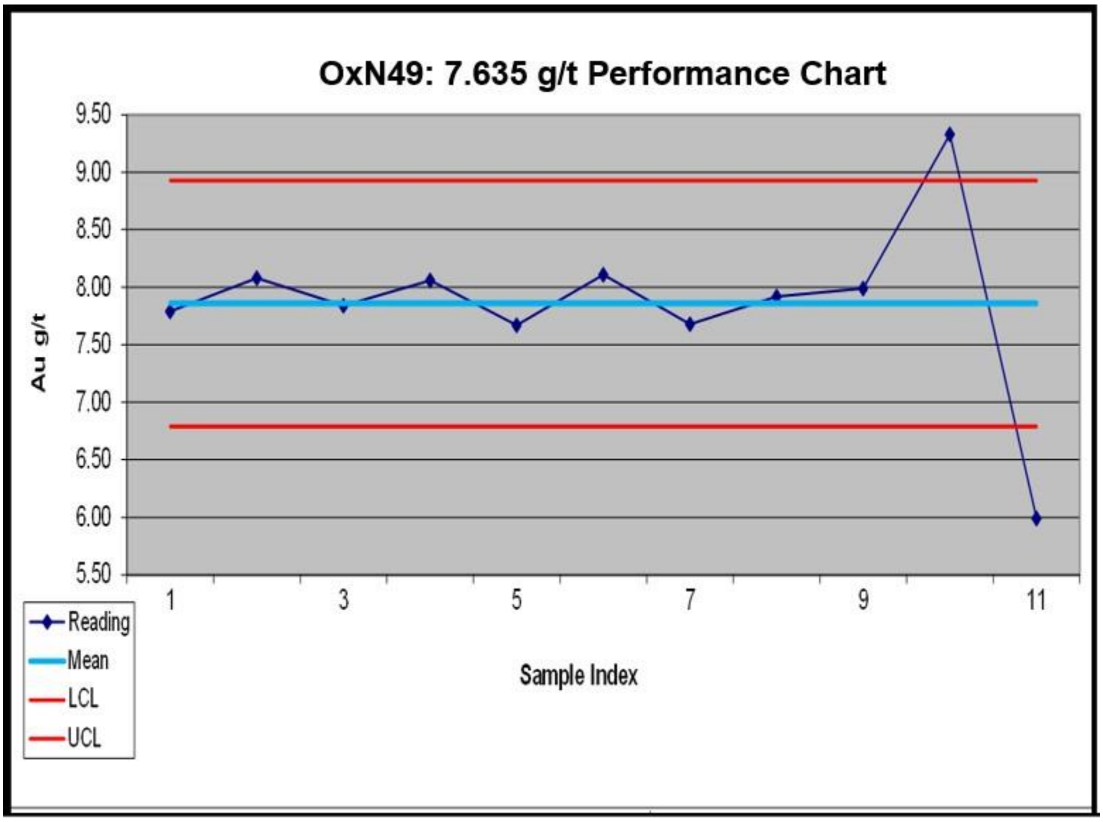


Figure 11.1: Standard Control Sheet Showing Assay Values, Mean and Control Limits for Standard OxN49

To ensure that the extent of failure is properly determined, samples that fall between the passing standard before the failed standard and the passing standard after the failed standard are selected for investigation.

The laboratory is then instructed to re-assay the samples between the first accepted standard above the failure to the first accepted standard below the failure, together with the three standards. The re-assay results for the standards are then assessed by means of the standard control sheet, and if accepted, the sample results are also accepted and entered into the project database. If any of the re-assayed standards are rejected, the procedure is repeated.

11.2.2 Accepting or Rejecting Assay Data using Standard Results

After using the standard control sheet to determine whether to accept or reject the assay result for a standard, the information is used to annotate the laboratory assay spreadsheet. As shown in the example in Figure 11.2, the accepted standard assays are highlighted in green, and the rejected standard assays are highlighted in red. This assists the process of selecting which samples are re-assayed by the laboratory as outlined in Section 11.2.1.

MW111392				
BATCH95	Au	AuR	AuR2	AuS
METHOD	FAA505	FAA505	FAA505	FAA505
LDETECTION	0.01	0.01	0.01	0.01
UDETECTION	100	100	100	100
UNITS	PPM	PPM	PPM	PPM
529135	0.03			
529136	0.69			
529137	0.02			
529138	1.35		1.24	
529139	0.02			
529140	0.02			
529141	0.02			
529142	0.02	0.02		
529143	0.03			
529144	0.04			
529145	0.02			
529146	0.03			
529147	0.02			
529148	0.05			
529149	0.27			
529150	0.02			
529151	0.04			
529152	0.04			
529153	0.02			
529154	0.02			
529155	0.31			
529156	0.03			
529157	0.03			
529158	0.03			
529159	0.03			
529160	0.03			
529161	0.05	0.04		
529162	0.03			
529163	0.02			
529164	0.02			
529165	1.03		1.01	
529166	0.04			
529167	0.03			
529168	0.02			
529169	0.03			
529170	0.02			
529171	0.02			
529172	0.02			
529173	0.02			
529174	0.03			
529175	0.05			
529176	0.02			
529177	0.04			0.03
529178	0.02			
529179	0.02			
529180	0.02			
529181	0.02			
529182	0.03	0.02		
529183	0.81		0.83	
529184	0.04			
529185	0.03			
529186	0.02			
529187	0.04			
529188	0.02			
529189	0.02			
529190	0.1			
529191	0.02			

Accepted Standard

Rejected Standard

Accepted Standard

Range of samples to be re-assayed by the laboratory: From the first accepted standard above the failure to the first accepted standard below the failure

Annotated assay results sheet showing the samples selected for re-assay based on a rejected standard

MW111392					
BATCH95	Au	AuR	AuR2	AuS	
METHOD	FAA505	FAA505	FAA505	FAA505	
LDETECTION	0.01	0.01	0.01	0.01	
UDETECTION	100	100	100	100	
UNITS	PPM	PPM	PPM	PPM	
529135	0.03				
529136	0.69				ACCEPTED RE-ASSAYS:
529137	0.02				
529138	1.35		1.24		1.27
529139	0.02				0.02
529140	0.02				0.04
529141	0.02				0.03
529142	0.02	0.02			0.03
529143	0.03				0.02
529144	0.04				0.06
529145	0.02				0.02
529146	0.03				0.04
529147	0.02				0.03
529148	0.05				0.05
529149	0.27				0.35
529150	0.02				0.03
529151	0.04				0.03
529152	0.04				0.02
529153	0.02				0.04
529154	0.02				0.03
529155	0.31				0.65
529156	0.03				0.03
529157	0.03				0.03
529158	0.03				0.03
529159	0.03				0.03
529160	0.03				0.02
529161	0.05	0.04			0.04
529162	0.03				0.02
529163	0.02				0.02
529164	0.02				0.01
529165	1.03		1.01		1.13
529166	0.04				
529167	0.03				
529168	0.02				
529169	0.03				
529170	0.02				
529171	0.02				
529172	0.02				
529173	0.02				
529174	0.03				
529175	0.05				
529176	0.02				
529177	0.04			0.03	
529178	0.02				
529179	0.02				
529180	0.02				
529181	0.02				
529182	0.03	0.02			
529183	0.81		0.83		
529184	0.04				
529185	0.03				
529186	0.02				
529187	0.04				
529188	0.02				
529189	0.02				
529190	0.1				
529191	0.02				

Standards and assay results accepted and entered into the project

Results sheet with re-assay results, showing that all the results can be accepted

Figure 11.2: Assay and Re-Assay Results Sheets

11.2.3 Blanks

Theoretically, a blank will have a gold content below the analytical detection limit, which at most laboratories is 0.01 g/t (10 ppb) for a standard fire assay with a 50 g charge. However, instrumental and analytical errors may occur, and accidental contamination by gold-bearing material is possible, any of which may give a result above the detection limit.

For the current exercise, an upper limit of 0.03 g/t (30 ppb) Au was used for blanks, i.e., results > 0.03 g/t were rejected. The failed blank and associated samples were re-assayed, using the same principles as for failed standards.

The review resulted in the need for up to 3,820 samples representing 13.4 % of the entire drilling database for Adumbi, Kitenge and Manzako to go through another QC process. Of this total number of samples, 1,014 were from the Adumbi deposit and represented 11.1 % of the entire Adumbi database. The preferred samples to be selected for the resubmissions were pulp rejects from the original samples submitted. However, efforts made at Kilo’s storage facility at Beni to retrieve the 1,014 samples as pulps yielded only 616 (61 %) pulps. For the rest of the samples, 382 (38 %) quarter cores were taken from the remaining half cores that were in Kilo’s storage facility. The remaining 16 (1 %) samples could not be obtained as the hole (SADD0017) from which they were from had already been quartered for metallurgical studies.

The Adumbi samples were renumbered for resubmission to an umpire laboratory other than SGS for analysis. Emphasis was put on using similar analytical methods (50 g fire assay charge with AAS finish) as for the original samples by ALS. The samples were submitted with an insertion of 12 % of quality control material, made up of 8 % international standards and 4 % blanks.

Once the re-assayed results were received, the Kilo team undertook assessment of both the standards and blanks using the same criteria outline above.

Once all the checks were done and the new re-assayed values were determined as passed, they were compared to the earlier assays for the samples in the earlier database. The comparison in terms of correlational studies was made differently for samples submitted as pulps and those submitted as quarter cores on different grade ranges. In conclusion, the results of the pulps correlated very well with those for the original samples whereas those for the quarter cores showed some variation. The lesser correlation between the results comparing results from quarter cores with those from pulps of an earlier half core was expected as it is a known function of volume variance as well as nugget effect.

On the basis of these observations, the Kilo team assessed the impact of substituting the new re-assayed results on the mineralisation intercepts affected in terms of both widths and overall composited intercept grades, and they concluded that it was not worth replacing the old results in the database with the new ones, as they would not have any significant impact on the overall intercept widths and grades.

Minecon is of the opinion that as the re-assays all passed the QA/QC test, they should be used to replace the old results, and the process should not have ended with the correlation exercise. The resubmitted samples, even for the quarter cores, were submitted with an entire set of samples, including pulp splits from the original half core which went through QA/QC checks and passed; hence, they should have replaced the old sample results. The re-assaying exercise affected at least seven holes, namely SADD0001, SADD0004, SADD0005, SADD0010, SADD0011, SADD0012 and SADD0017, which went through the mineralised zones and impacted the interpretation; hence, replacing the old results with the new ones was necessary.

11.3 2014 TO 2017 QA/QC PROGRAMME

During the 2014 to 2017 exploration programme, the team continued with more stringent QA/QC protocols of inserting 8 standards and 4 blanks in every 100 samples submitted, i.e. 12 % QA/QC samples. It is worth noting that between 2010 and early 2011, Kilo submitted CRMs at a rate of 4 CRMs in a sample batch of 200.

The QA/QC database for the period 2014 to 2017 includes quality control samples inserted into samples collected from diverse sampling methods. Samples included BLEG, rock chip, pit, trench, channel and diamond drillhole samples. A total of 5,973 samples were submitted to the analytical laboratory for assaying. Table 11.4 provides a summary of the samples submitted during the period. A total of 525 standards and 289 blanks were inserted during the period, and the summarised performance of these QA/QC materials is as shown in Table 11.5 to Table 11.12.

Table 11.4: Summary of the Samples in the 2014 to 2017 Exploration Period

Number of Samples						Total
BLEG	Rock Chip	Pit	Trench Channels	Other Channels	DD	
216	419	198	74	355	4,531	**5,793**

Colonial adits that had earlier been sampled were resurveyed but not resampled.

Of the 380 m of colonial trenches re-opened, 72.2 m of portions with good alteration known to be associated with mineralisation were sampled, yielding 74 samples.

The drilling data count included 998 samples from the pre-2014 drilling programme, which were sent for re-assaying.

The quality control material introduced with these samples included 525 standards and 289 blanks. The rate of standards and blanks usage per the number of samples submitted was 9.1 % and 4.9 %, respectively. During the 2014 to 2015 period, 171 standards were inserted, and in the 2016 to 2017 period, 354 standards were inserted. Diamond drilling of a total of 38 holes (6,907.64 m) was undertaken during the 2016 to 2017 period on several prospects under the Imbo Licence, including Adumbi West, Kitenge Extension, Adumbi South and the four Adumbi deep holes. Table 11.5 summarises the drilling undertaken during the period.

Table 11.5: Summary of Drilling Undertaken in 2016 to 2017

Prospect	Number of Holes Drilled	Metres
Adumbi West	11	1,555.45
Kitenge Extension	14	2,169.60
Adumbi South	9	1,406.64
Adumbi Deep	4	1,775.95
TOTAL	**38**	**6,907.64**

A summary of the performance of the QA/QC materials inserted in all the exploration activities undertaken from 2014 to 2017 is shown in Table 11.6.

Table 11.6: Summary of Performance of QA/QC Materials Inserted in 2014 to 2017

Blanks		CRMs (Standards)	
Number	Number or Percentage of Failures	Number	Number or Percentage of Failures
289	7 or 2.4 %	525	30 or 5.7 %

The source, type and other properties of the standards used are shown in Table 11.7.

Table 11.7: Source, Type, and Grade of Various Standards used in 2014 to 2017

CRM ID	Source	Material Type	Expected Grade (ppm)	95 % Confidence Interval
OxA89	Rocklabs	Oxide	0.084	0.0025
OxE106	Rocklabs	Oxide	0.606	0.004
OxG99	Rocklabs	Oxide	0.932	0.006
OxG98	Rocklabs	Oxide	1.017	0.006
Oxi96	Rocklabs	Oxide	1.802	0.012
HiSilK2	Rocklabs	Sulphide	3.474	0.034
SK62	Rocklabs	Sulphide	4.075	0.045
HiSilP1	Rocklabs	Sulphide	12.05	0.130
OxP91	Rocklabs	Oxide	14.82	0.100
SQ48	Rocklabs	Sulphide	30.25	0.170

The standards used by Kilo considered both a broad grade range and different material types, oxides and sulphides, which Minecon considers good practice. The distribution of the standards across the various prospects is shown in Table 11.8.

Table 11.8: Distribution of Standards Across the Imbo Project

Prospect	HiSilK2	HiSilP1	OxG98	Oxi96	OxP91	SK62	SQ48	OxA89	OxE106	OxG99	Total
Adumbi Deep	14	11	12	15	11	12	11				**86**
Adumbi (2014 DD Re-Assays)	19			19			18	18	19		**93**
Adumbi West (2014 to 2015)	14			12		4	13	11	13		**67**
Adumbi West (2017)	6	9	12	12	11	10	9				**69**
Adumbi South	6	13	19	13	11	10	13				**85**
Kitenge Extension	1	8	7	15	11	15	12				**69**
Imbo West (BLEG)								3	3	2	**8**
Ngazi (PE9692)	4	4	4	3	3	4	5				**27**
Dhahabu (PE9595)	1	1	2	1		1					**6**
Nane (PE140)			1	1							**2**
Gambi (PE137)		1			1						**2**
Vatican	1			2			1				**4**
Kitenge Senegal	1			1		1	2		2		**7**
TOTAL	**67**	**47**	**57**	**94**	**48**	**57**	**84**	**32**	**37**	**2**	**525**

A total of 4.9 % (30) of standards and 2.4 % (7) of blanks failed at the first submission. The overall performance of the standards is summarised in Table 11.9. Table 11.10 shows the summary of the performance of the standards across the prospects.

Table 11.9: Summary of Overall Performance of Standards Used

CRM Performance	CRM ID										TOTAL
	HiSilK2	HiSilP1	OxG98	Oxi96	OxP91	SK62	SQ48	OxA89	OxE106	OxG99	
Number of Times Used	67	47	57	94	48	57	84	32	37	2	525
Number of Passes	61	45	52	89	46	55	81	30	34	2	495
Number of Failures	6	2	5	5	2	2	3	2	3	0	30
Percentage Failure	9.0	4.3	8.8	5.3	4.2	3.5	3.6	6.3	8.1	–	5.7

Table 11.10: Summary of Overall Performance of Standards by Prospects

Prospect	CRM Performance			
	Number of Times Used	Number of Passes	Number of Failures	Percentage Failure
Adumbi Deep	86	82	4	4.7
Adumbi (2014 DD Re-Assays)	93	83	10	10.8
Adumbi West (2014 to 2015)	69	66	3	4.3

Prospect	CRM Performance			
	Number of Times Used	Number of Passes	Number of Failures	Percentage Failure
Adumbi West (2017)	67	67	0	–
Adumbi South	85	81	4	4.7
Kitenge Extension	69	65	4	5.8
Imbo West (BLEG)	8	8	0	–
Ngazi (PE9692)	27	23	4	14.8
Dhahabu (PE9595)	6	5	1	16.7
Nane (PE140)	2	2	0	–
Gambi (PE137)	2	2	0	–
Vatican	4	4	0	–
Kitenge Senegal	7	7	0	–
TOTAL	**525**	**495**	**30**	**5.7**

Figure 11.3 to Figure 11.6 are standard control charts plotted for QA/QC analyses of the various standards used in the Imbo Project.

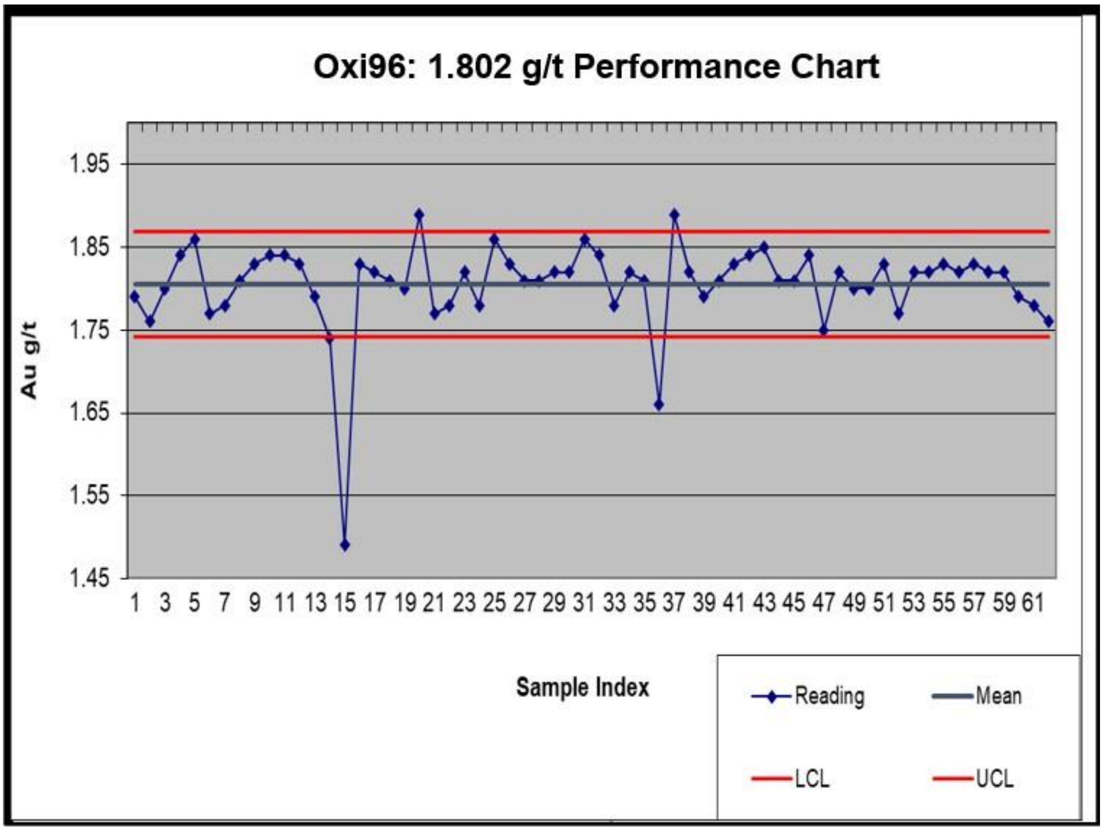


Figure 11.3: Standard Control Performance Chart for Oxi96, Imbo Project

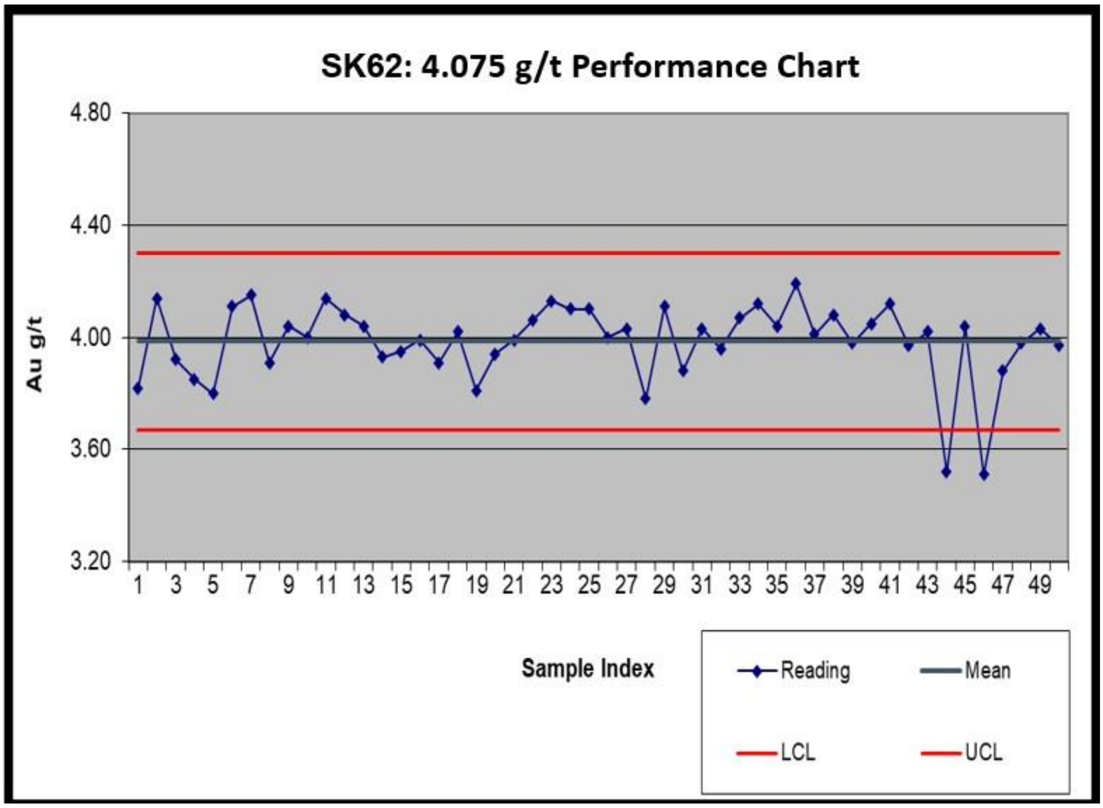


Figure 11.4: Standard Control Performance Chart for SK62, Imbo Project

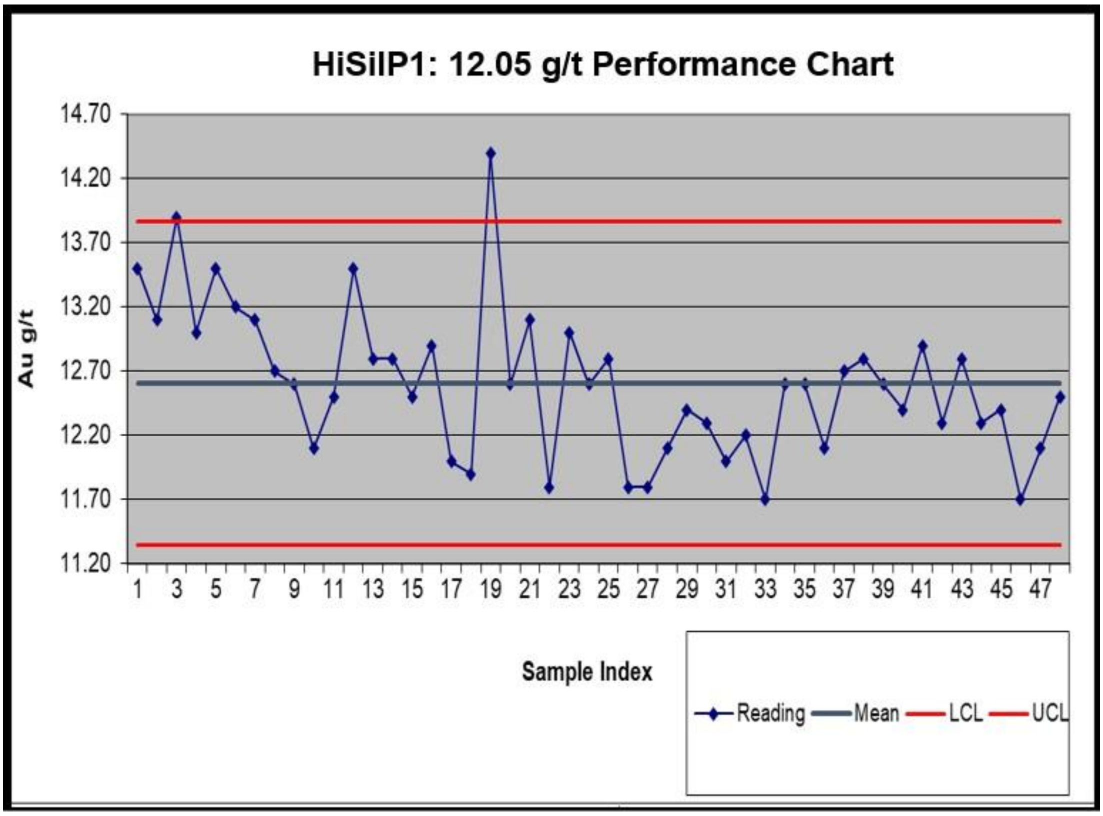


Figure 11.5: Standard Control Performance Chart for HiSilP1, Imbo Project

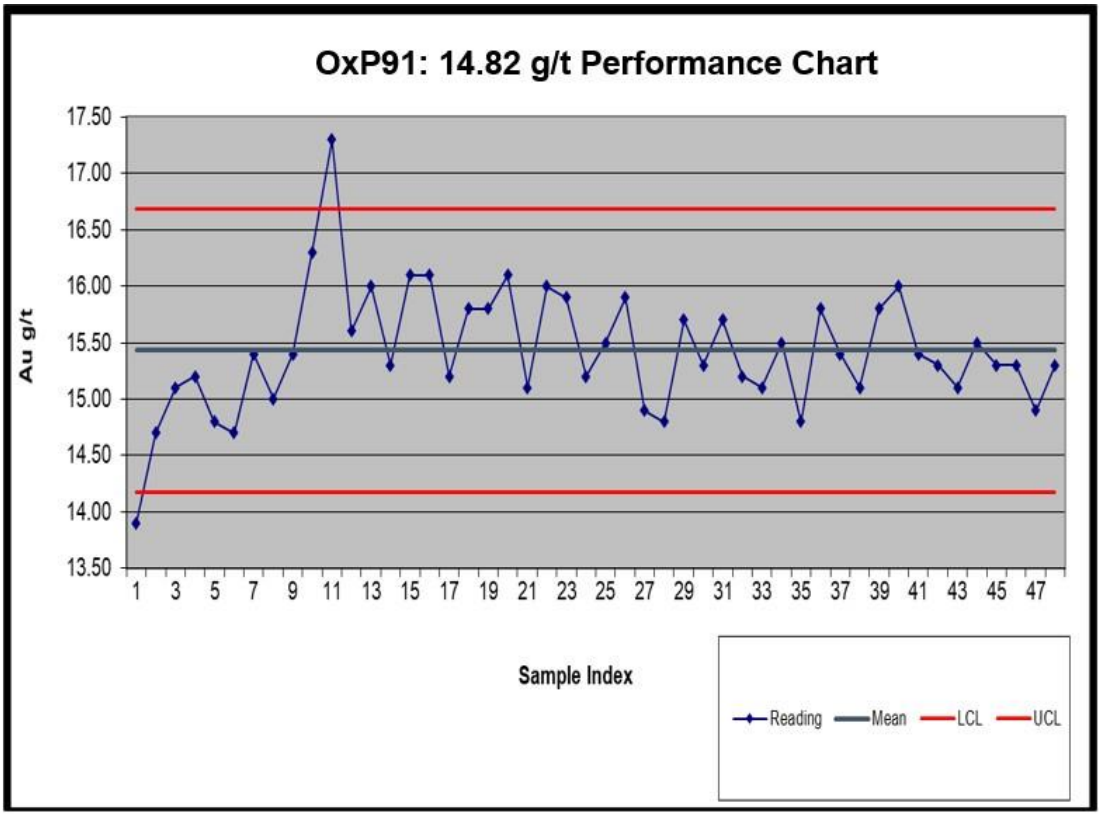


Figure 11.6: Standard Control Performance Chart for OxP91, Imbo Project

The basic statistics of the blanks submitted as part of the QA/QC process are summarised in Table 11.11.

Table 11.11: Basic Statistics of Blanks Submitted as Part of 2014 to 2017 QA/QC Programme

Field	No. of Samples	Minimum (ppm)	Maximum (ppm)	Range	Mean (ppm)	Variance	Standard Deviation
Au	288	0.005	0.09	0.090	0.014	0.000	0.011

11.3.1 Adumbi Deposit Standards Performance

Of the 525 standards inserted, 86 were inserted into the Adumbi drillhole samples submitted, which formed the core of the resource database for the Adumbi deposit.

The 86 standards represent 8 % of the 1,073 drillhole samples assayed. The summary of the standards used in the Adumbi deposit is given in Table 11.12. Table 11.13 provides a summary of the performance of the standards used for the Adumbi deposit.

Table 11.12: Summary of Standards used in QA/QC Programme for Adumbi Deposit

CRM ID	Certified Grade (ppm)	No. of Samples	Minimum (ppm)	Maximum (ppm)	Range	Mean (ppm)	Variance	Standard Deviation (Std)	3Std
HiSilK2	3.474	14	3.44	3.60	0.16	3.503	0.002	0.040	0.120
HiSilP1	12.05	11	11.70	12.90	1.20	12.491	0.119	0.345	1.035
OxG98	1.017	12	0.98	1.19	0.21	1.033	0.003	0.050	0.150
Oxi96	1.802	15	1.75	1.85	0.10	1.813	0.001	0.025	0.074
OxP91	14.82	11	14.80	16.00	1.20	15.427	0.113	0.336	1.008
SK62	4.075	12	3.52	4.19	0.67	4.012	0.026	0.160	0.480
SQ48	30.25	11	29.40	32.40	3.00	30.873	0.893	0.945	2.835

Table 11.13: Summarised Performance of Standards Used in QA/QC Programme for Adumbi Deposit

CRM ID	Count	Certified Grade (ppm)	Number Passed	Number Failed	Comment
HiSilK2	14	3.474	13	1	No re-assay submitted
HiSilP1	11	12.05	11	0	
OxG98	12	1.017	10	2	1 failed re-assayed other not re-assayed
Oxi96	15	1.802	15	0	
OxP91	11	14.82	11	0	
SK62	12	4.075	11	1	No re-assay submitted
SQ48	11	30.25	11	0	
Total	**86**		**82**	**4**	

Figure 11.7 to Figure 11.13 are standard control charts plotted for QA/QC analyses of the various standards used in the Adumbi deposit only.

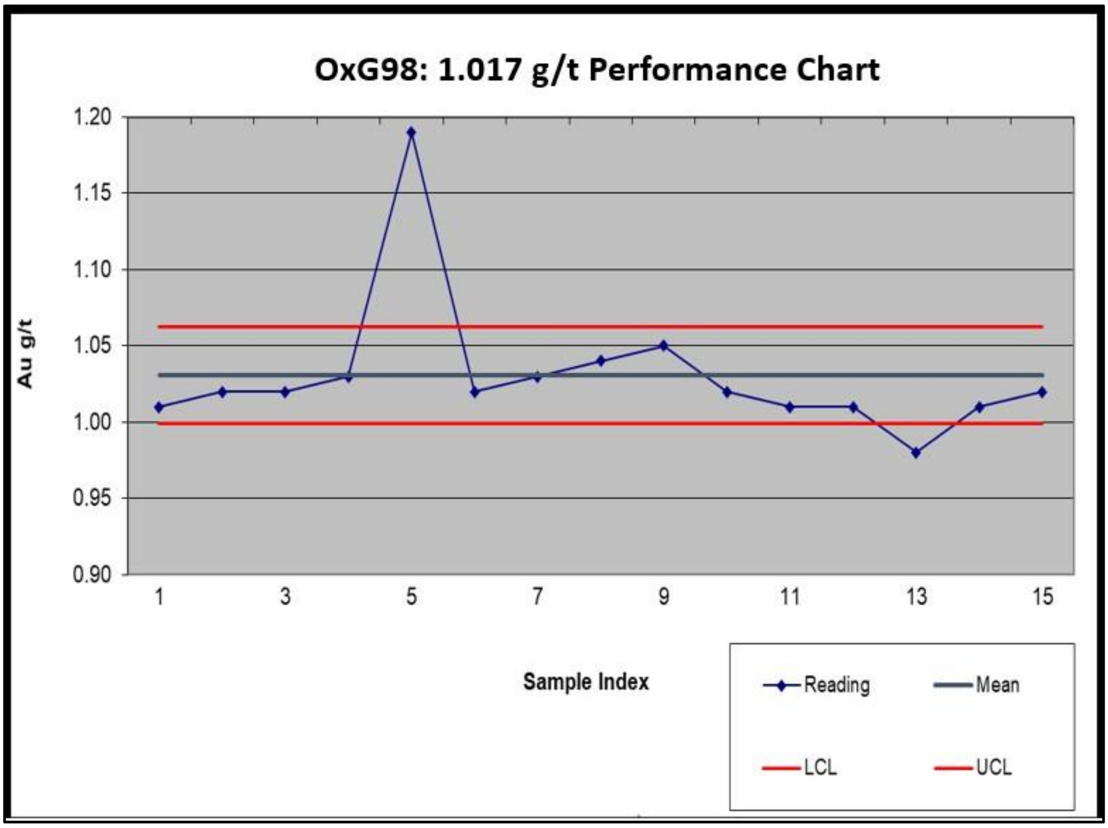


Figure 11.7: Standard Control Performance Chart for OxG98, Adumbi Deposit Only

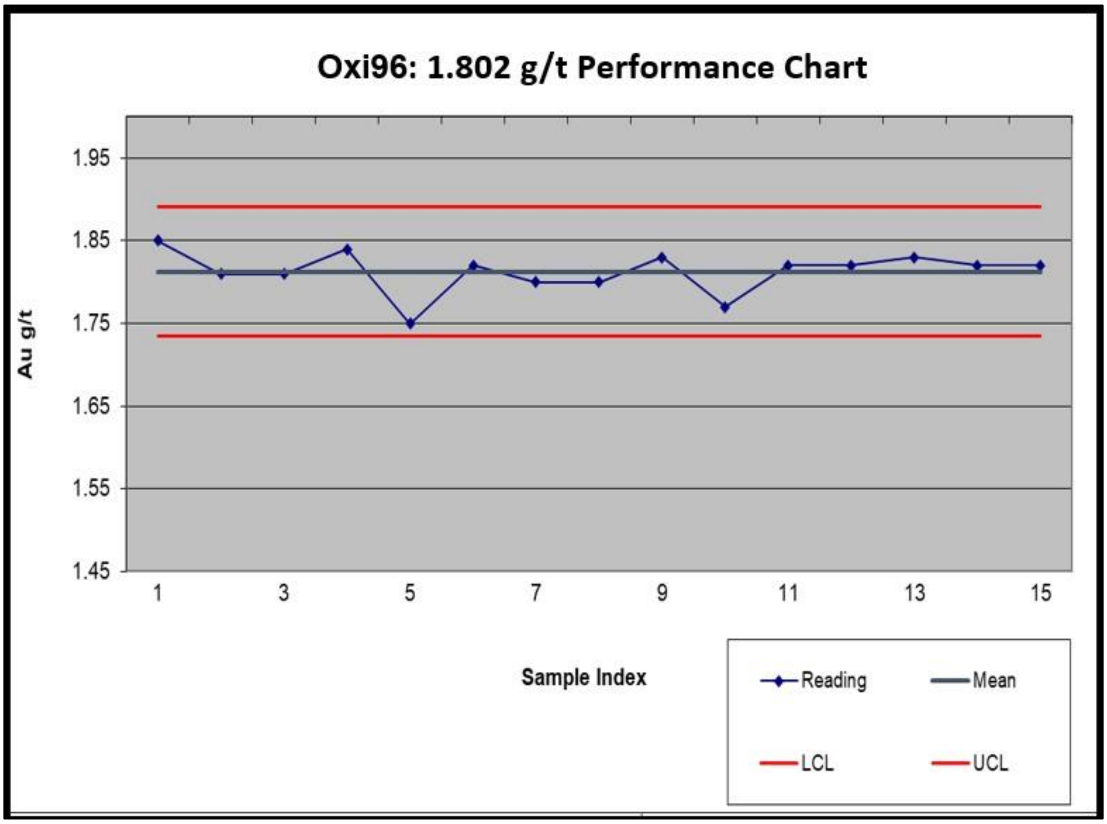


Figure 11.8: Standard Control Performance Chart for Oxi96, Adumbi Deposit Only

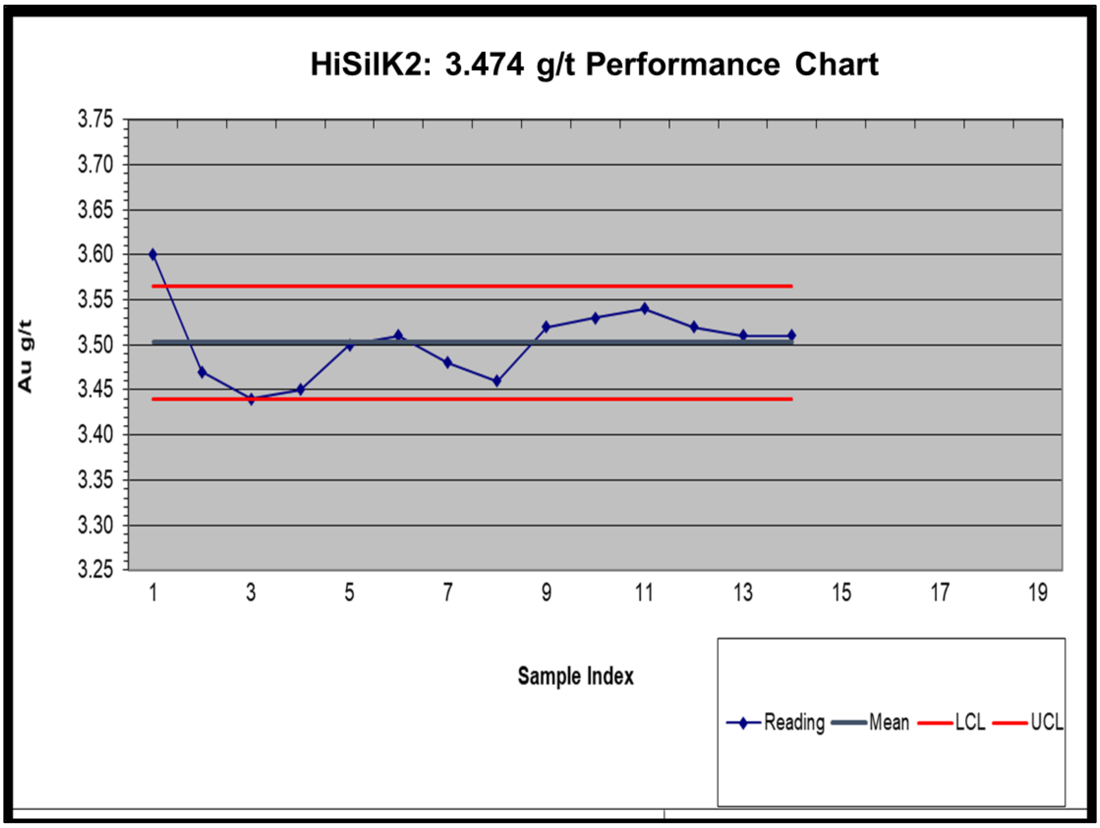


Figure 11.9: Standard Control Performance Chart for HiSilK2, Adumbi Deposit Only

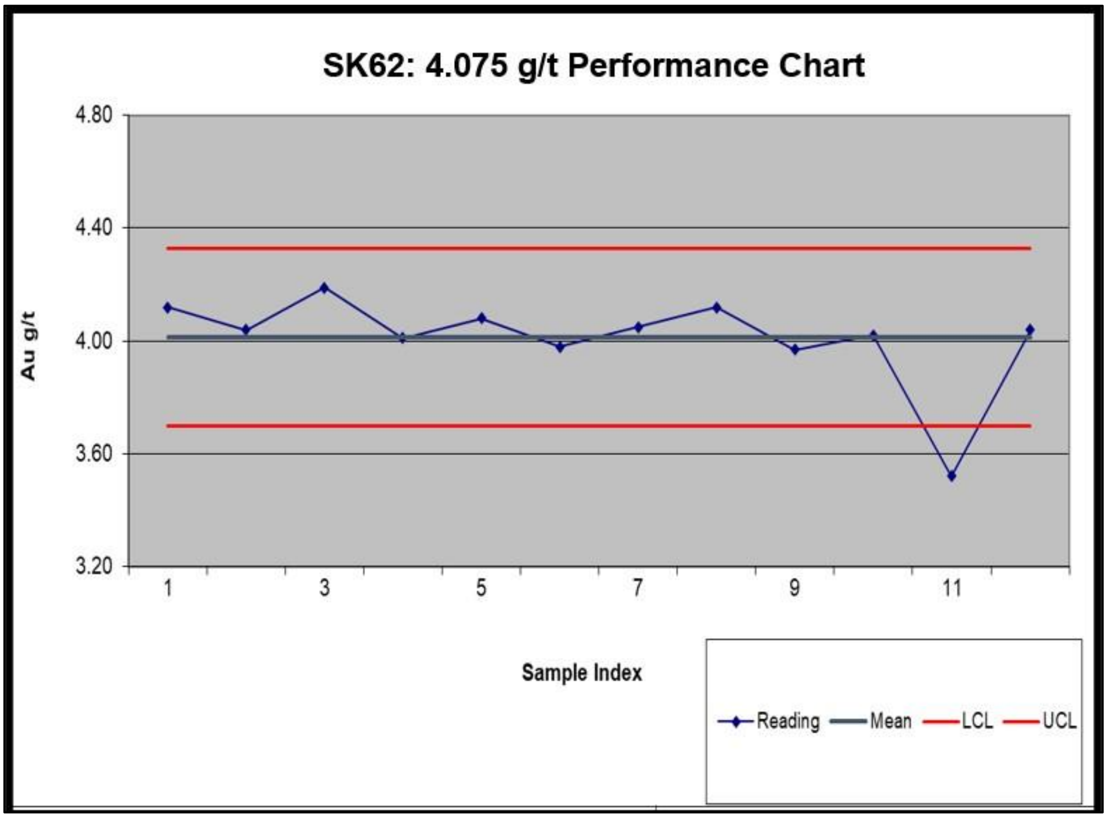


Figure 11.10: Standard Control Performance Chart for SK62, Adumbi Deposit Only

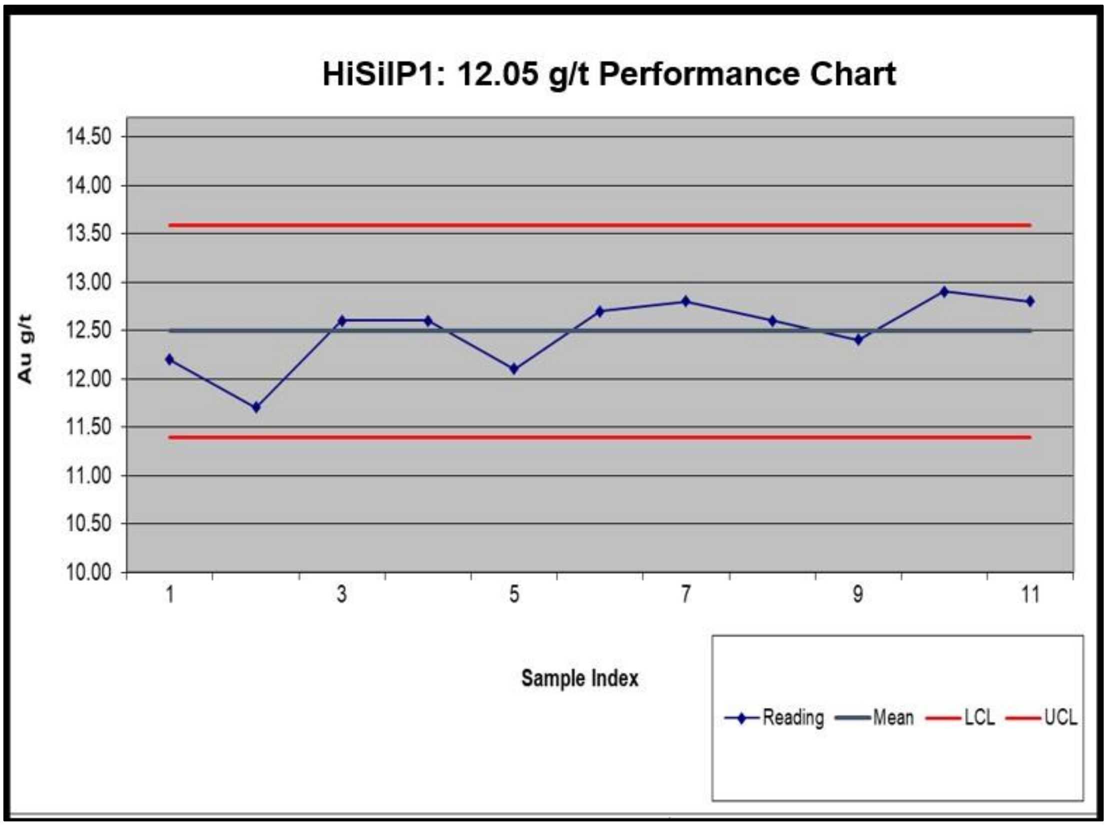


Figure 11.11: Standard Control Performance Chart for HiSilP1, Adumbi Deposit Only

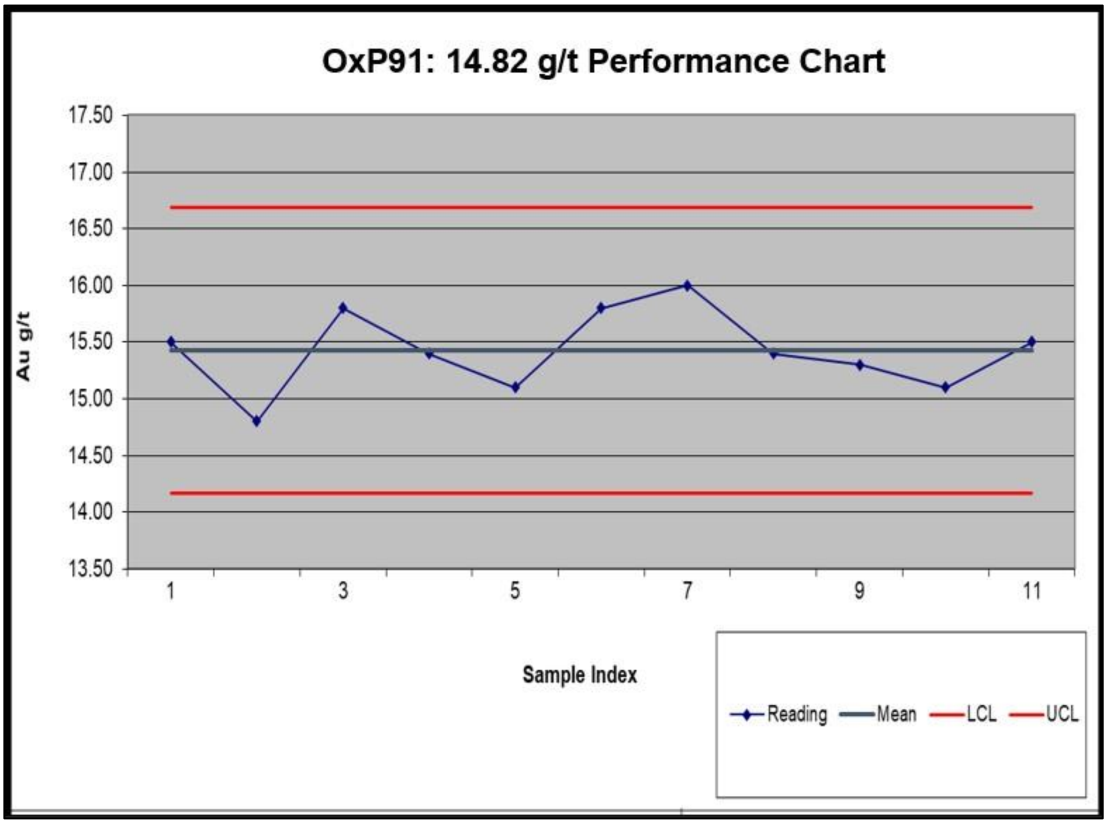


Figure 11.12: Standard Control Performance Chart for OxP91, Adumbi Deposit Only

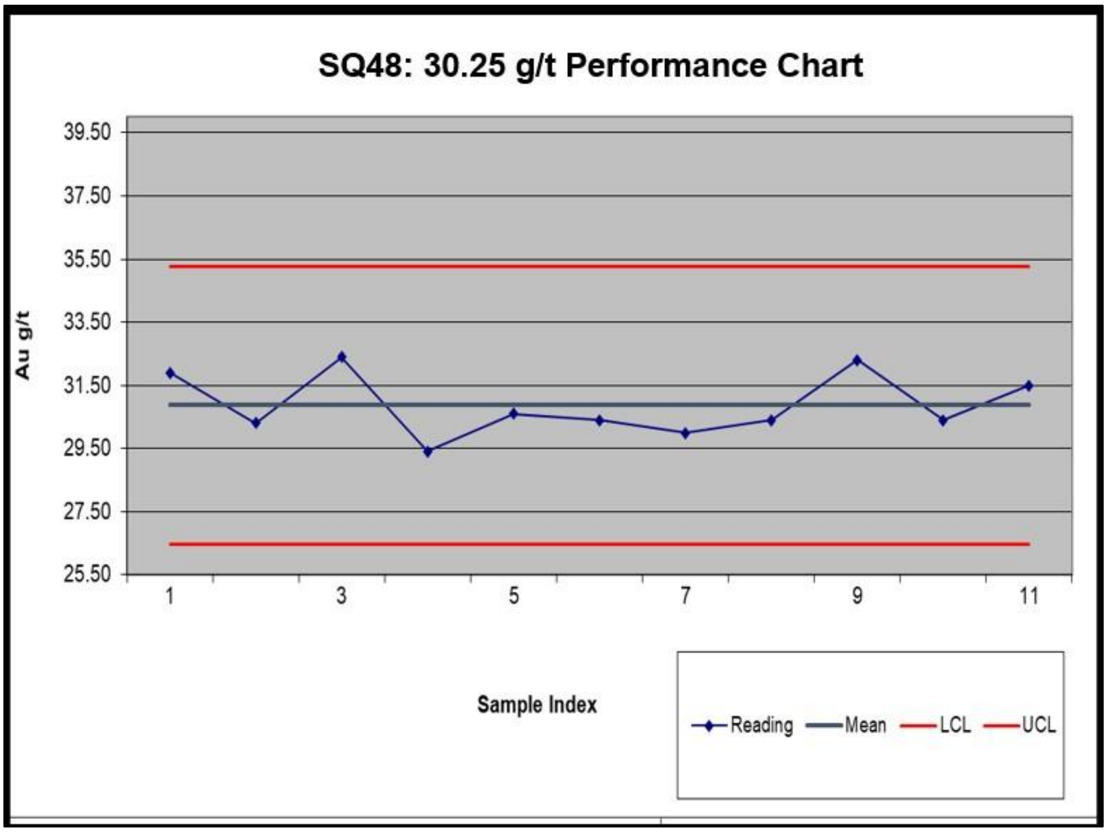


Figure 11.13: Standard Control Performance Chart for SQ48, Adumbi Deposit Only

There was a re-assay request for one of the four Adumbi standards that failed: Sample Number 61775 (OxG98) failed, and the re-assay results passed the QC check so the re-assayed result was used in the database. There were however no re-assay requests for the other three samples that failed (62207 (SK62), 62174 (OxG98) and 61325 (HiSilK2)). For 62174 (OxG98) and 61325 (HiSilK2), the Kilo team were of the view that they had passed when considered within the range of the entire standards of their kind submitted for the entire Imbo Project, hence re-assaying was not necessary. Minecon is however of the opinion that, the domain to determine the passing of the standards should have been Adumbi specific and not the entire project samples. The failure of 3 standards out of the 86 standards submitted represents a 3.5 % failure, which, in Minecon's opinion, is not fatal, but the team should have requested re-assays. In the absence of the re-assayed result, Minecon carried out visual checks on the adjacent samples to the failed standards to determine the possible impact of the failure on these nearby samples. Though no clear related impact could easily be seen, Minecon recommended that these samples be retrieved and submitted for re-assay.

The overall performance of the standards does not exhibit any bias. The frequency of the insertion of QC materials is adequate to enable the data to be used for geological modelling and resource estimation.

11.3.2 Blanks

Kilo, as part of their QA/QC programme, inserted blanks at a rate of 4 blank samples in every batch of 100 samples. The blanks sourced from Humac Laboratories Tanzania are stored at Adumbi in 50 × 20 L storage bins in a secured place.

As a way of checking the integrity of the stored blanks, the Kilo team collected blanks from 20 different bins, labelled them as normal samples, and submitted them to the SGS Mwanza laboratory for assaying. The results of the assays received are as shown in Table 11.14. All but one sample (Number 51255 from Bucket 18) returned results less or equal to 0.02 g/t, which is the accepted upper limit of a blank. The failed bucket was isolated, investigated, and not used as a blank.

Table 11.14: Results for Batch Testing of Blanks

Sample Number	Assay Result (ppm)	SGS Job No.	Kilo Batch No.	Description
51237	< 0.01	MW141778	Batch 005	Bucket No. 1
51238	< 0.01	MW141778	Batch 005	Bucket No. 2
51239	< 0.01	MW141778	Batch 005	Bucket No. 3
51240	< 0.01	MW141778	Batch 005	Bucket No. 4
51241	< 0.01	MW141778	Batch 005	Bucket No. 5
51242	< 0.01	MW141778	Batch 005	Bucket No. 6
51243	< 0.01	MW141778	Batch 005	Bucket No. 7
51244	< 0.01	MW141778	Batch 005	Bucket No. 8
51245	< 0.01	MW141778	Batch 005	Bucket No. 9
51246	< 0.01	MW141778	Batch 005	Bucket No. 10
51247	< 0.01	MW141778	Batch 005	Bucket No. 11
51248	0.01	MW141778	Batch 005	Bucket No. 12
51249	< 0.01	MW141778	Batch 005	Bucket No. 13
51250	< 0.01	MW141778	Batch 005	Bucket No. 14
51251	< 0.01	MW141778	Batch 005	Bucket No. 15
51252	0.02	MW141778	Batch 005	Bucket No. 16
51253	< 0.01	MW141778	Batch 005	Bucket No. 17
51254	0.09	MW141778	Batch 005	Bucket No. 18
51255	< 0.01	MW141778	Batch 005	Bucket No. 19
51256	< 0.01	MW141778	Batch 005	Bucket No. 20

Of the 289 blanks inserted, 7 returned grades above 0.03 g/t, which is Kilo's accepted upper limit for blanks grade. These 7 samples are indicated in blue in Table 11.15. The blanks reported minimum and maximum grades of 0.005 g/t and 1.19 g/t, respectively. One failed blank reported a grade of 1.19 g/t, which is not a typical grade for a blank. This was discarded after checking the grade of adjacent samples in the same batch (Batch 70, SGS Job Number MW141774), which reported lower grades than it or even blank grades. The sample before it reported a grade of 1.06 g/t, and the one after it was < 0.01 g/t. Minecon suspects that this could have been due to sample swapping and not contamination. Therefore, although it has been included in the list of failed blanks, it has been discarded in any calculations or plots. Kilo made re-assay requests for some of the other failed blanks. The failure of 7 blanks represents a 2.4 % failure, which Minecon considers satisfactory. Table 11.15 displays the results of the failed blanks. Figure 11.14 shows a performance chart of all the blanks inserted in the 2014 to 2017 programme.

Table 11.15: Results of Failed Blanks

Sample Number	Assay Result (ppm)	SGS Job No.	Kilo Batch No.	Prospect
61540	0.03	MW170761	Batch 076	Adumbi Deep
61990	0.03	MW171154	Batch 081	Adumbi Deep
56309	0.03	MW142179	Batch 009	Adumbi Pre-2014 Cores Re-Assays
56334	0.03	MW142179	Batch 009	Adumbi Pre-2014 Cores Re-Assays
56709	0.04	MW142183	Batch 013	Adumbi Pre-2014 Cores Re-Assays
57034	0.04	MW142186	Batch 016	Adumbi Pre-2014 Cores Re-Assays
56687	0.08	MW142182	Batch 012	Adumbi Pre-2014 Cores Re-Assays
57087	0.09	MW142192	Batch 019	Adumbi Pre-2014 Cores Re-Assays
57298	0.03	MW162448	Batch 041	Adumbi South
57824	0.03	MW162451	Batch 044	Adumbi South
59248	0.03	MW170400	Batch 059	Adumbi West
59374	0.03	MW170401	Batch 060	Adumbi West
59398	0.03	MW170401	Batch 060	Adumbi West
59916	0.03	MW170595	Batch 069	Adumbi West
61166	0.03	MW170597	Batch 071	Adumbi West
51982	0.07	MW150667	Batch 033	Adumbi West
51254	0.09	MW141778	Batch 005	Adumbi West
51162	1.19	MW141774	Batch 001	Adumbi West
66890	0.03	MW171641	Batch 092	Ngazi
NOTE: The values in blue indicate failed blanks.				

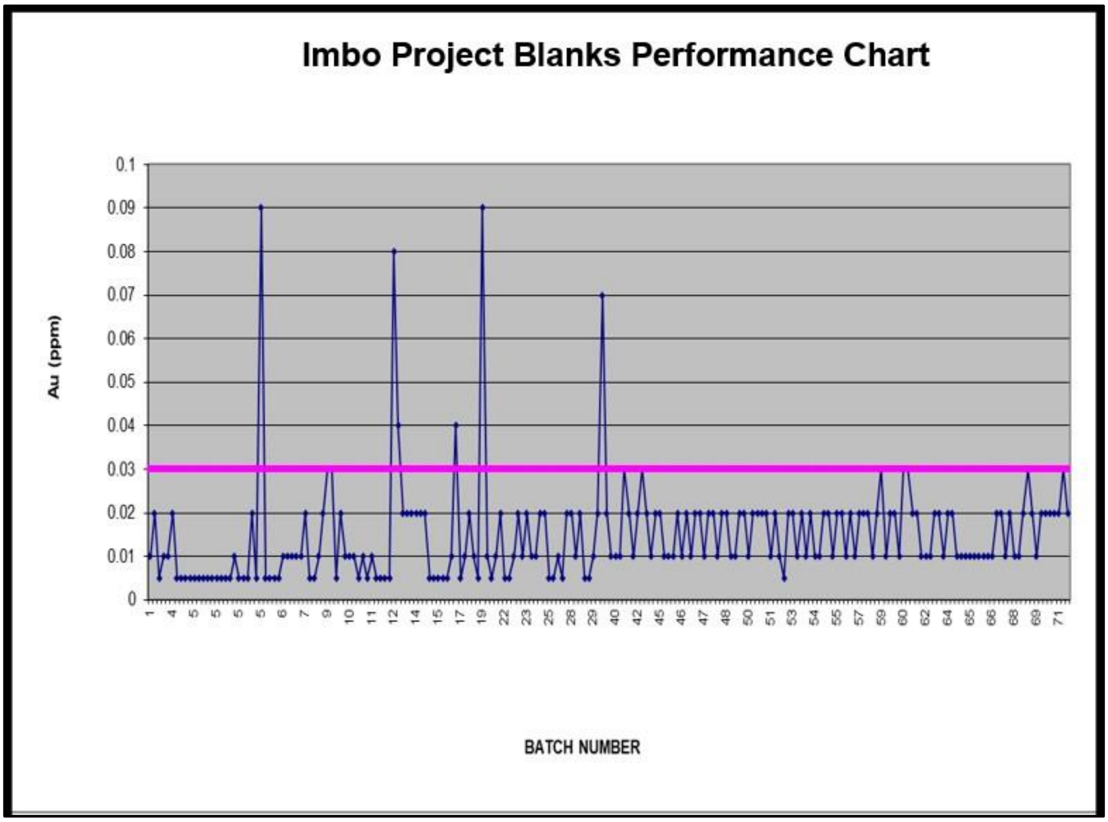


Figure 11.14: Performance Chart of all Blanks Inserted in the 2014 to 2017 Programme

It was however noted that a further 12 samples returned with a grade of 0.03 g/t (see Table 11.15), this could have sent the percentage of failed blanks to 6.6 %. Minecon considers an upper limit of 0.02 g/t as tolerable for blanks. Minecon therefore recommended that any blank reporting a grade of greater than 0.02 g/t be flagged as failed and a re-assay request made for the sample and three adjacent samples before and after the failing blank. This recommendation was implemented during the 2020 to 2021 drilling programme. After checking the neighbouring samples of the failing blanks, Minecon does not think that there was any significant cross-contamination of the samples during the sample preparation process.

Minecon also recommended that re-assay requests be sent for all failed blanks. Upon receipt of re-assayed results, a decision could be made on whether to replace the results of adjacent samples in the database. Minecon noted that there were approximately 12 extra blanks listed as inserted, for which there were no results provided in the database. These blanks were investigated with respect to their adjacent samples.

11.3.3 Duplicates

The Kilo QA/QC programme did not include the submission of any duplicates. For drill cores, half cores were sent to the SGS Mwanza laboratory for preparation and assaying, and Kilo decided to keep the other half for further studies including metallurgical studies.

Duplicates are vital in QA/QC programmes as they assist in determining the repeatability or variability even at the local stage (nugget effect) inherent with sampling the same interval and detecting sample number mix-ups and even sample swapping.

Minecon recommended that Loncor incorporate the use of duplicates in the QA/QC programme. The recommendation was implemented during the 2020 to 2021 drilling programme. Duplicates were inserted at a rate of 1 in any 50 samples submitted. In the same way that standards of variable grade ranges are used to monitor the laboratory precision in various grade ranges, the duplicates selected were within the potential mineralised zones with varying grade ranges, to test the repeatability of grades in a wider range of grades. Duplicate samples were labelled in a disguised manner so that the analytical laboratory could not detect that they were duplicates. Duplicate samples were field (core, trench or underground), coarse (crushed reject), or pulp (pulverised reject) duplicates.

11.3.4 Inter-Laboratory Checks

For the period 2014 to 2017, Kilo did not submit any samples for inter-laboratory or refereeing checks. Inter-laboratory checks are essential in comparing the repeatability of grades of different splits of the sample by different laboratories. In the pre-2014 exploration programme period, Kilo sent approximately 296 Kitenge and Manzako pulps to the SGS Johannesburg laboratory for referee or umpire checks.

Now that Loncor is using the SGS Mwanza laboratory as the main laboratory, ALS Chemex was selected as the umpire laboratory. The umpire laboratory uses the same analytical method that the principal laboratory employed to facilitate comparison of the results obtained from the two different laboratories.

11.3.5 Review of External Laboratory Internal QA/QC Programme

The SGS Mwanza laboratory uses standards, blanks, duplicates and replicates as part of their internal QA/QC checks. The frequency of the QC materials usage is as follows:

- 2 standards samples inserted in a batch of 50 samples
- 1 preparation blank (prep process blank) inserted in a batch of 50 samples
- 1 reagent blank inserted in a batch of 50 samples
- 1 weighed replicate in every 50 samples
- 1 preparation duplicate (re-split) in every 50 samples

Minecon has reviewed the internal QC reports submitted by the SGS laboratory during the period that they processed Kilo's samples and finds them all in order. Hence, there is no evidence of contamination or lack of precision in the laboratory processes.

A diverse grade range of standards from low-grade through medium to the higher-grade standards was used, and they all passed the QA/QC protocol. In addition, all the blanks inserted by SGS during the period passed, and no grade above 0.02 g/t was reported.

Duplicate correlation graphs showed high repeatability of results with a high correlation co-efficient in the 0.999 ranges.

Replicates also confirmed good repeatability.

11.4 SECURITY OF SAMPLES

For the period 2014 to 2017, the Kilo exploration team submitted all the samples to the SGS Mwanza laboratory for both sample preparation and chemical analysis. No employee, officer, director, or associate of Kilo carried out any sample preparation on samples from the Imbo exploration programme.

The drill core was transported from the drill site, by a Kilo vehicle or helicopter, to the secure core yard facility at the Adumbi base camp. Initially, all the samples collected for assaying were retained in a locked secure shed until they were dispatched by a Kilo vehicle to the administrative office in Beni. A commercial freight-forwarding agent then transported the samples from Beni to the SGS Mwanza Laboratory for sample preparation and analysis.

Dispatch forms accompany the samples from the field to the laboratory for analysis to verify each step of the process and to ensure that all samples are accounted for. The SGS laboratory sends sample reconciliation forms upon receipt of any batch of samples sent by Kilo through the forwarding agents to be sure that no sample losses or reduction occurs. All the half core was indexed and stored at the secured core storage facility at the Adumbi base camp.

11.5 2020 TO 2021 QA/QC PROGRAMME

During the 2020 to 2021 exploration programme, Loncor initiated enhanced QA/QC protocols. In a batch of 100 samples, 8 standards, 2 blanks and 2 duplicates were inserted, equivalent to 12 % of control samples. These control materials were inserted into all types of samples that were collected and processed during the period, prior to being dispatched to the SGS Mwanza laboratory for analysis.

All the analytical results received from SGS were subjected to Loncor's internal QA/QC checks. These included checking their performance with respect to the inserted control materials, made up of international CRMs, blanks, and duplicates. Batches that passed the checks were released to the database geologist for further verification and capturing into the validated master assay database. Per practice, batches that fail the internal QA/QC checks are subject to either partial or full re-assay requests, depending on the cause and extent of the failure. The re-assayed results are re-subjected to the same internal QA/QC checks. Only results that pass the QA/QC checks are entered into the master database.

By mid-October 2021, 7,675 samples had been received for processing at the sample preparation laboratory. A total of 8,020 samples were processed by the sample preparation laboratory. The processed samples included control samples such as blanks and other laboratory efficiency monitoring samples. A total of 8,743 samples of various forms, including QA/QC resubmissions, were dispatched to the SGS Mwanza laboratory for analysis during the period. These included 1,042 control samples, 708 standards, 205 blanks and 129 duplicates. The shortfall in duplicates was as a result of the delay in starting the introduction of the collection of duplicates. This represents an overall QA/QC percentage of 11.9 % with respect to the samples processed by the sample preparation laboratory by mid-October 2021.

Table 11.16: Summary of Samples sent to the Sample Preparation Laboratory for Processing

Number of Samples			Total
DD	Soils	Others	
4,748	2,586	341	7,675

At the time of compiling this report, 26 core holes totalling 10,433.64 m had been drilled since the start of the 2020 to 2021 drilling campaign. Twenty-four holes were drilled at Adumbi-Canal and two holes were drilled at Mambo Bado (see Table 11.17).

Table 11.17: Summary of Drilling Undertaken in 2020 to 2021

Prospect	Number of Holes Drilled	Metres
Adumbi	22	9,711.84
Canal	2	359.60
Mambo Bado	2	362.20
TOTAL	**26**	**10,433.64**

The performance of the QA/QC materials, based on the results received to date for the 2020 to 2021 exploration programme, is summarised in Table 11.18.

Table 11.18: Summary of Performance of QA/QC Materials Inserted in 2020 to 2021

Blanks		CRMs (Standards)	
Number	Number or Percentage of Failures	Number	Number or Percentage of Failures
205	3 or 1.5 %	708	14 or 2.0 %

The source, type and other properties of the standards inserted are shown in Table 11.19.

Table 11.19: Source, Type, and Grade of Various Standards used in 2020 to 2021

CRM ID	Source	Material Type	Expected Grade (ppm)	95 % Confidence Interval
OxA89	Rocklabs	Oxide	0.084	0.0025
OxE106	Rocklabs	Oxide	0.606	0.004
OxG99	Rocklabs	Oxide	0.932	0.006
HiSilK2	Rocklabs	Sulphide	3.474	0.034
SK62	Rocklabs	Sulphide	4.075	0.045
HiSilP1	Rocklabs	Sulphide	12.05	0.13
SQ48	Rocklabs	Sulphide	30.25	0.17
OXC109	Rocklabs	Oxide	0.201	0.002

CRM ID	Source	Material Type	Expected Grade (ppm)	95 % Confidence Interval
SE44	Rocklabs	Sulphide	0.606	0.006
SE114	Rocklabs	Sulphide	0.634	0.005
SG115	Rocklabs	Sulphide	1.017	0.005
SJ111	Rocklabs	Sulphide	2.812	0.021

The standards used by Loncor considered both a broad gold grade range and various material types: oxide and sulphide. The grade range is generally selected to match the sample types submitted, which Minecon considers good practice. The distribution of the standards across the various prospects is shown in Table 11.20.

Table 11.20: Distribution of Standards Across the Imbo Project

Project/Area	HiSilK2	HiSilP1	OXE106	OXG99	SE114	SE44	SG 115	SJ111	SK62	SQ48	OXA89	OXC109
Adumbi	86	72	1	7	56	11	52	55	87	32		
Imbo East			40	35							42	41
Imbo West			13	12							14	11
Mambo Bado	5	3				4			6	1		
Laboratory			7	5							6	4
TOTAL	**91**	**75**	**61**	**59**	**56**	**15**	**52**	**55**	**93**	**33**	**62**	**56**

A total of 2.0 % (14) of standards and 1.5 % (3) of blanks failed at the first submission. The overall performance of the standards is summarised in Table 11.21.

Table 11.21: Summary of Overall Performance of Standards Used

CRM Performance	CRM ID											
	HiSilK2	HiSilP1	OXE106	OXG99	SE114	SE44	SG 115	SJ111	SK62	SQ48	OXA89	OXC109
Number of Times Used	91	75	61	59	56	15	52	55	93	33	62	56
Number of Passes	90	75	60	57	56	15	50	51	91	33	61	55
Number of Failures	1	0	1	2	0	0	2	4	2	0	1	1
Percentage Failure	1.10	–	1.64	3.39	–	–	3.85	7.27	2.15	–	1.61	1.79

Figure 11.15 to Figure 11.18 are standard control charts plotted for QA/QC analyses of the various standards used in the Imbo Project.

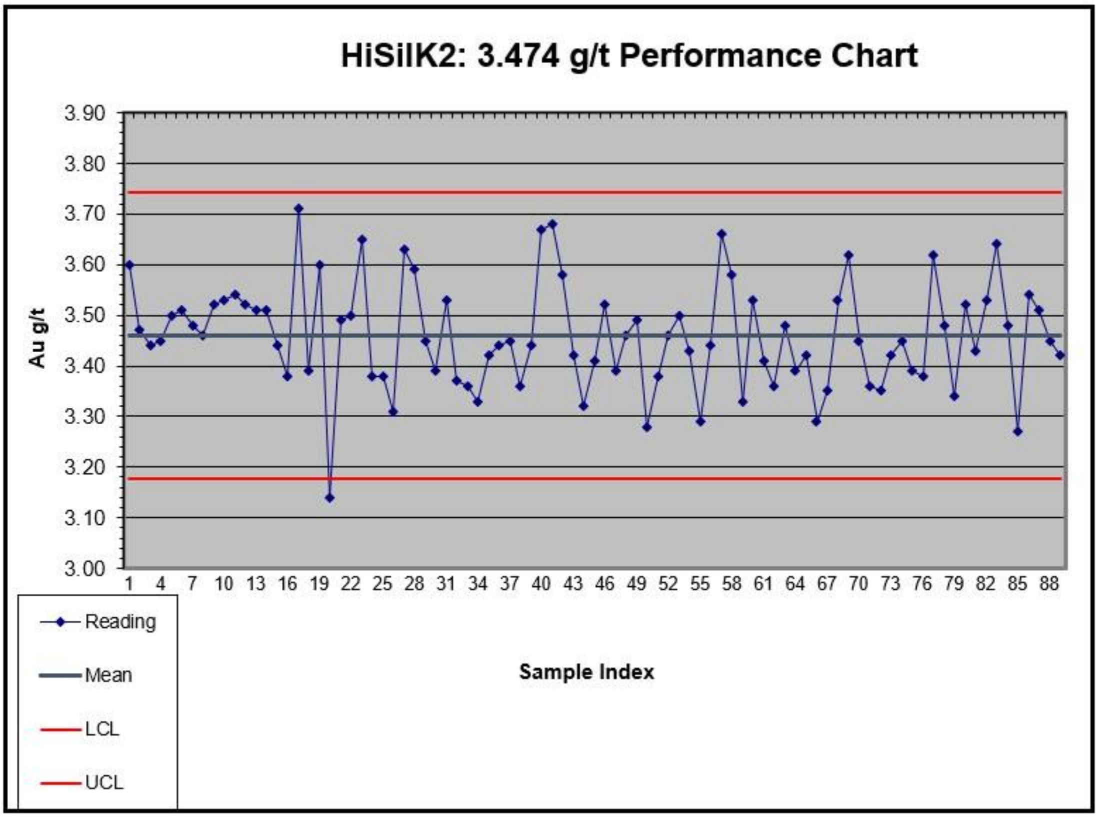


Figure 11.15: Standard Control Performance Chart for HiSilK2, Imbo Project

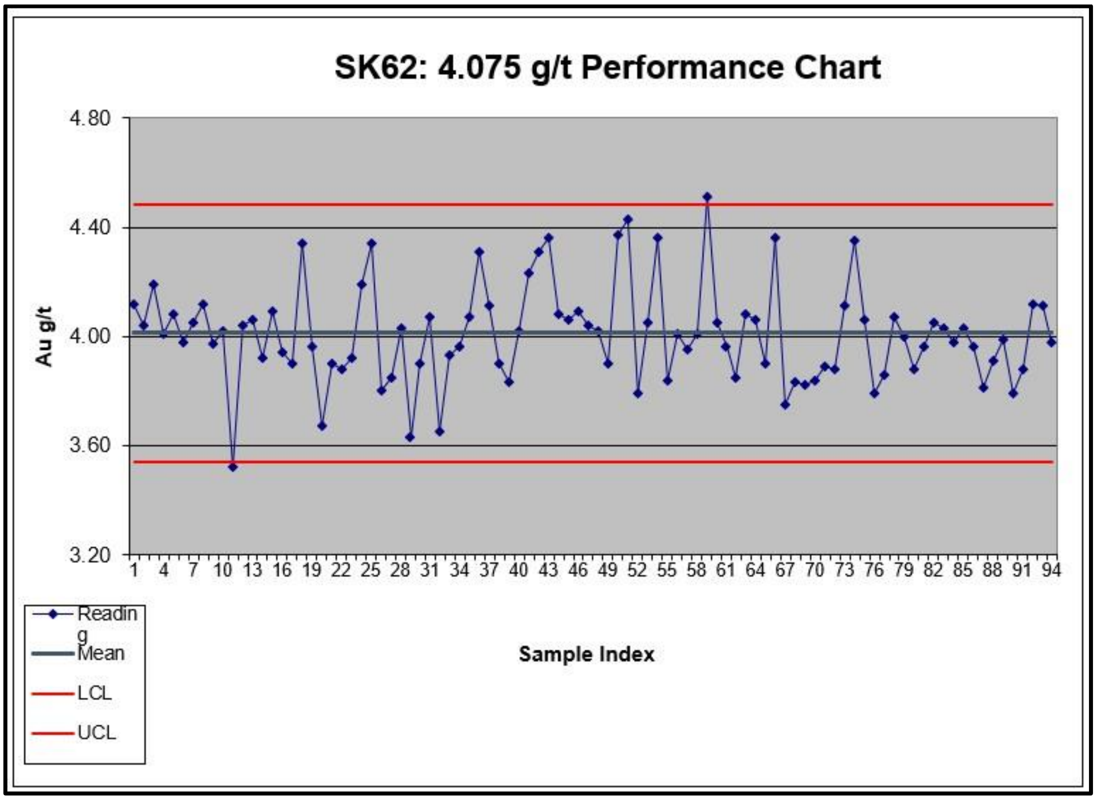


Figure 11.16: Standard Control Performance Chart for SK62, Imbo Project

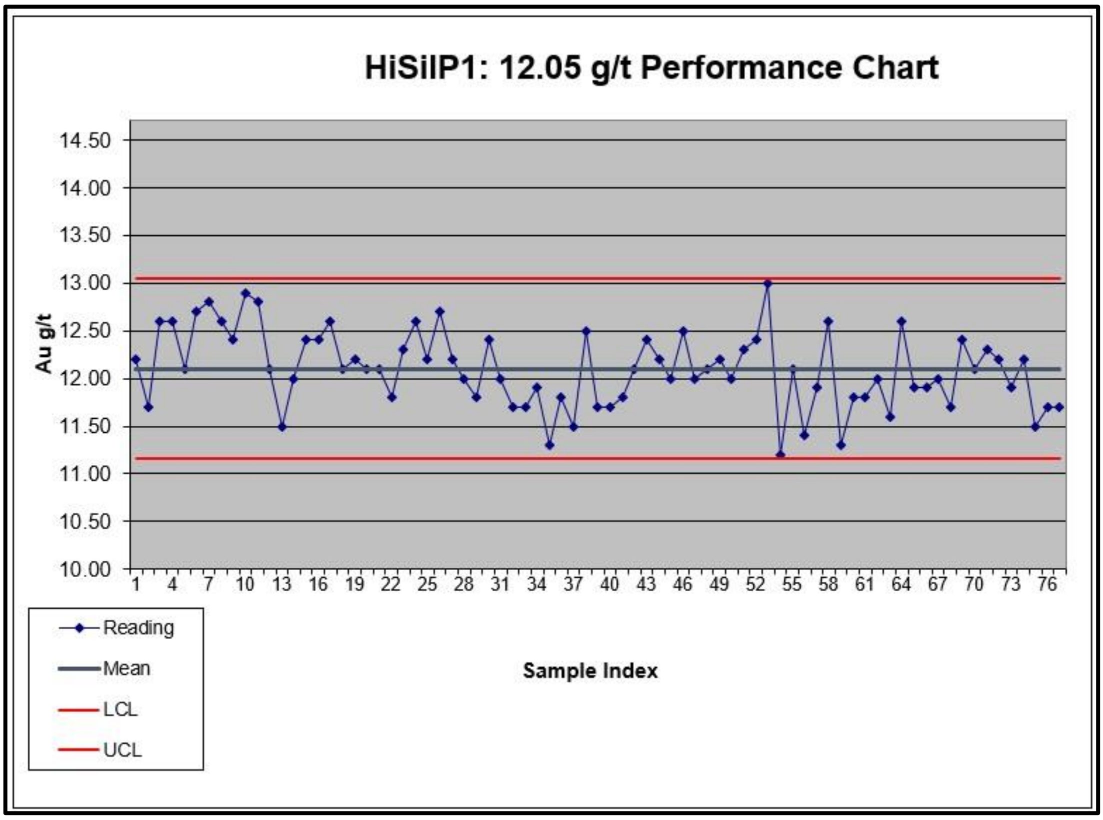


Figure 11.17: Standard Control Performance Chart for HiSilP1, Imbo Project

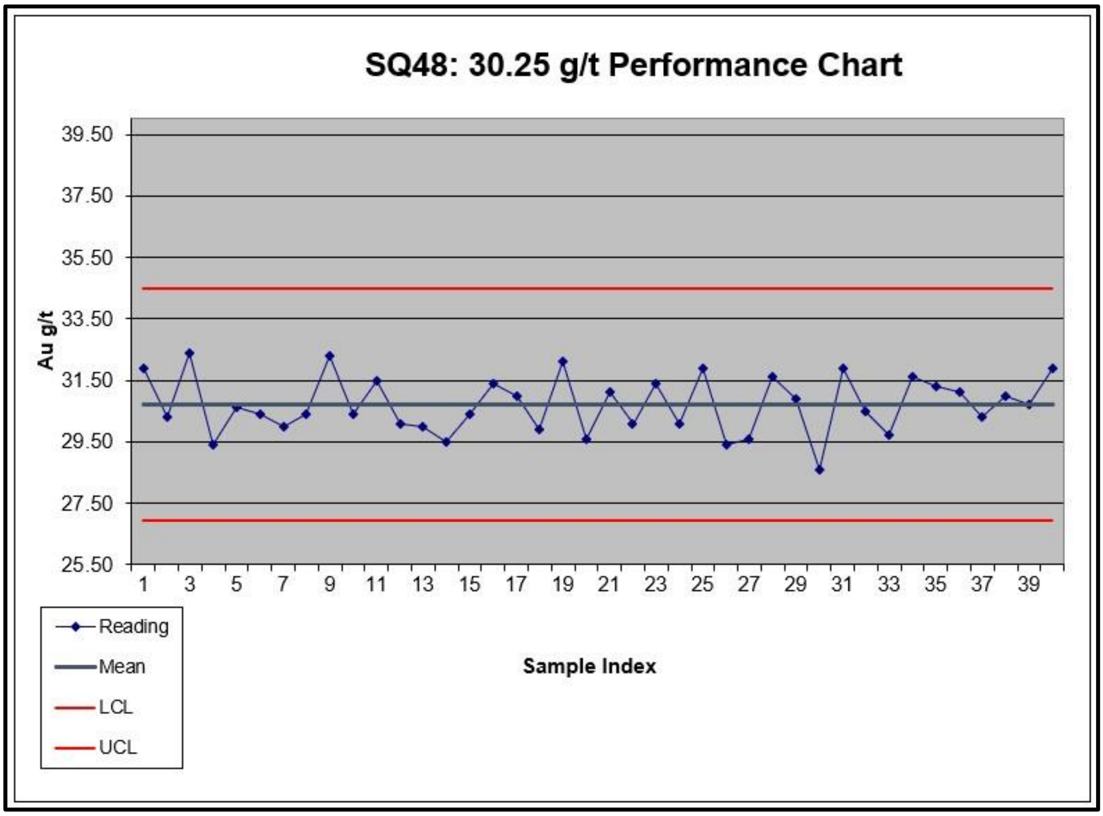


Figure 11.18: Standard Control Performance Chart for SQ48, Imbo Project

The basic statistics of the blanks submitted as part of the QA/QC process are summarised in Table 11.22.

Table 11.22: Basic Statistics of Blanks Submitted as Part of 2020 to 2021 QA/QC Programme

Field	No. of Samples	Minimum (ppm)	Maximum (ppm)	Range	Mean (ppm)	Variance	Standard Deviation
Au	205	0.005	0.07	0.065	0.01	0	0.007

11.5.1 Adumbi Deposit Standards Performance

Of the 708 standards inserted, 459 were inserted into the Adumbi drillhole samples submitted, which formed the core of the resource database for the Adumbi deposit.

The 459 standards represent 9.6 % of the 4,746 samples assayed in relation to the Adumbi drillhole samples assayed. The summary of the standards used in the Adumbi deposit is given in Table 11.23. Table 11.24 provides a summary of the performance of some of the standards used for the Adumbi deposit.

Table 11.23: Summary of Standards used in QA/QC Programme for Adumbi Deposit

CRM ID	Certified Grade (ppm)	No. of Samples	Minimum (ppm)	Maximum (ppm)	Range	Mean (ppm)	Variance	Std	3Std
HiSilK2	3.474	86	0.01	4.04	4.03	3.394	0.168	0.410	1.229
HiSilP1	12.05	72	0.02	13.2	13.18	11.829	2.257	1.502	4.507
OXG99	0.932	7	0.91	0.96	0.05	0.933	0.000	0.019	0.057
SK62	4.075	87	2.08	4.95	2.87	3.981	0.115	0.339	1.018
SQ48	30.25	32	28.6	32.1	3.5	30.713	0.808	0.899	2.697
OXE106	0.606	1	0.59	0.59	0	0.590	–	–	-
SE44	0.606	11	0.59	0.63	0.04	0.615	0.000	0.010	0.030
SE114	0.634	56	0.5	0.75	0.25	0.624	0.002	0.042	0.125
SG 115	1.017	52	0.82	1.17	0.35	0.988	0.004	0.059	0.178
SJ 111	2.812	55	2.4	3.33	0.93	2.789	0.028	0.168	0.504

Table 11.24: Summarised Performance of Standards Used in QA/QC Programme for Adumbi Deposit

CRM ID	CRM Grade (ppm)	Count	Number Passed	Number Failed	Comment
HiSilK2	3.474	91	90	1	Re-assay returned 0.01 g/t, re-submitted Sample 82428 in Batch 191AD returned 3.45 g/t
HiSilP1	12.05	75	75	0	
OXE106	0.606	61	60	1	Initial failure reporting 0.41 g/t passed upon re-assay

CRM ID	CRM Grade (ppm)	Count	Number Passed	Number Failed	Comment
OXG99	0.932	59	57	2	1 swap between standard and sample resolved after re-assay in Batch 142LB
					Other failed standard in Batch 140IW passed on re-assay
SE114	0.634	56	56	0	
SE44	0.606	15	15	0	
SG 115	1.017	52	50	2	Initial failures reporting 0.82 g/t passed upon re-assay
SJ111	2.812	55	51	4	All initial failures resolved via re-assay in their batches
SK62	4.075	93	91	2	Initial failure returned slightly higher than 2SD but within 3SD, result used
					Failed sample passed using acceptable AuR (gold assay replicate from SLIMS) value after inspecting adjacent samples
SQ48	30.25	33	33	0	

Figure 11.19 to Figure 11.22 are standard control charts plotted for QA/QC analyses of the various standards used in the Adumbi deposit only.

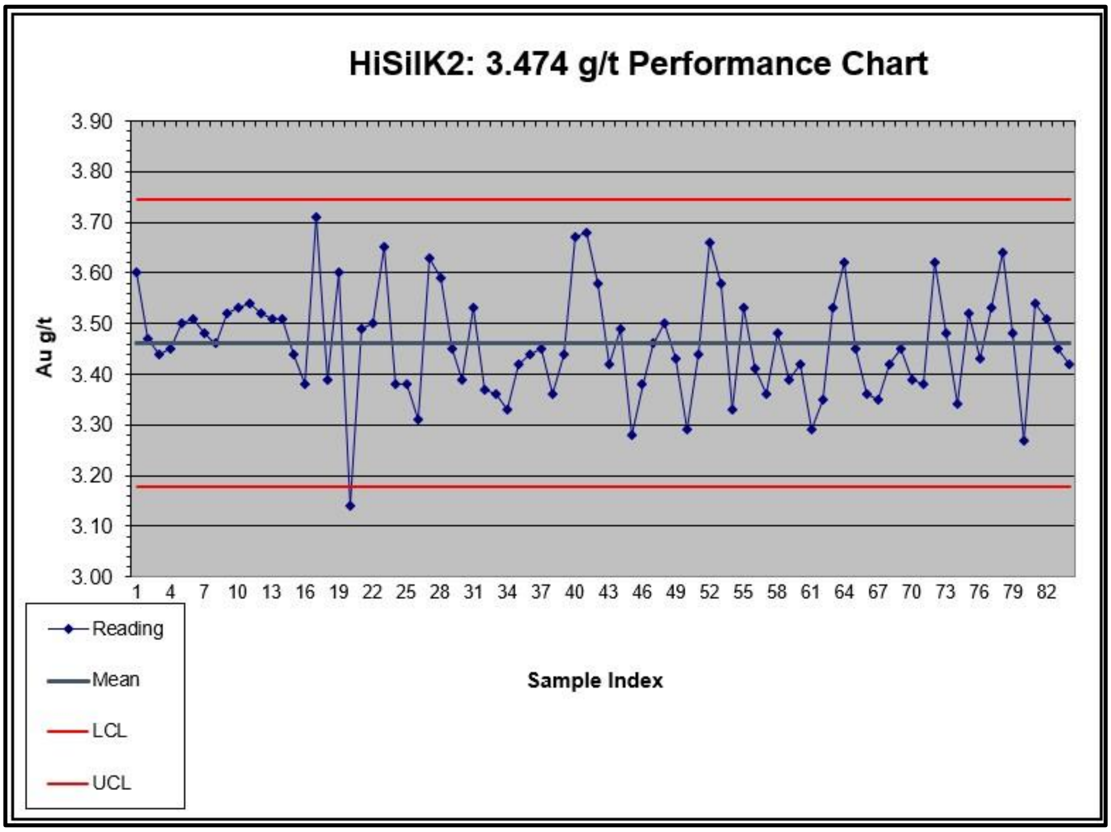


Figure 11.19: Standard Control Performance Chart for HiSilK2, Adumbi Deposit Only

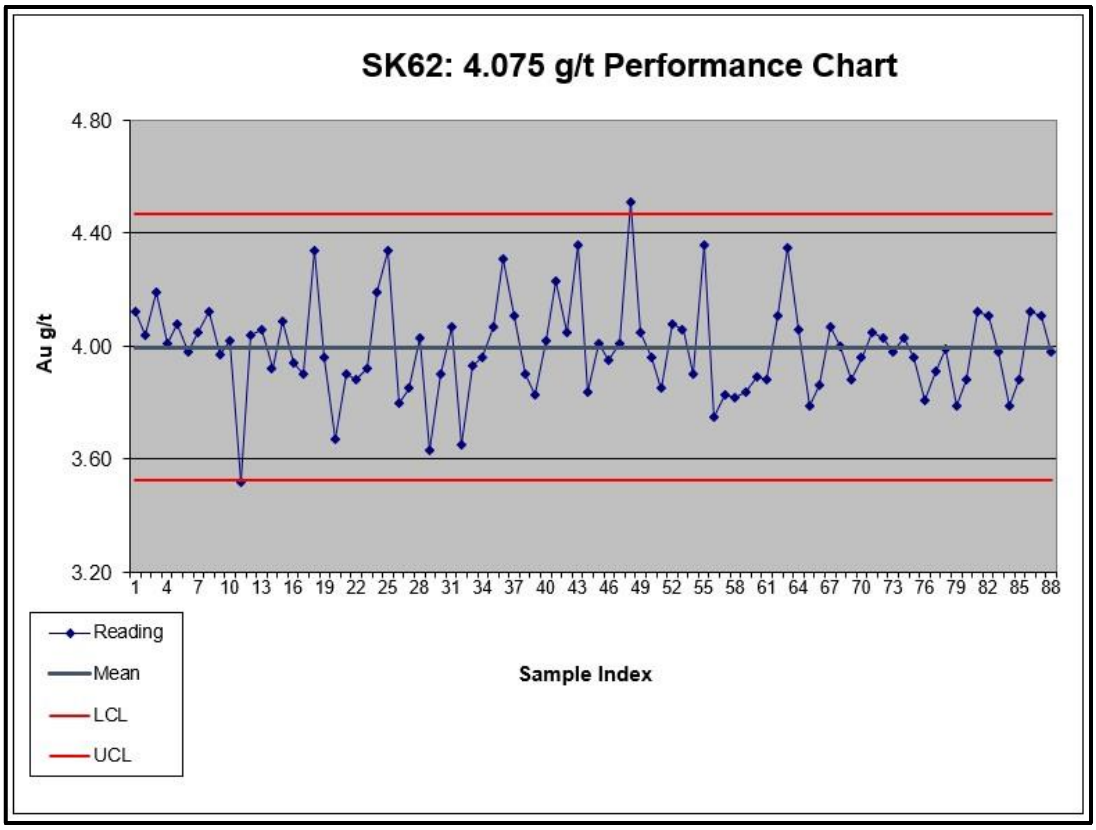


Figure 11.20: Standard Control Performance Chart for SK62, Adumbi Deposit Only

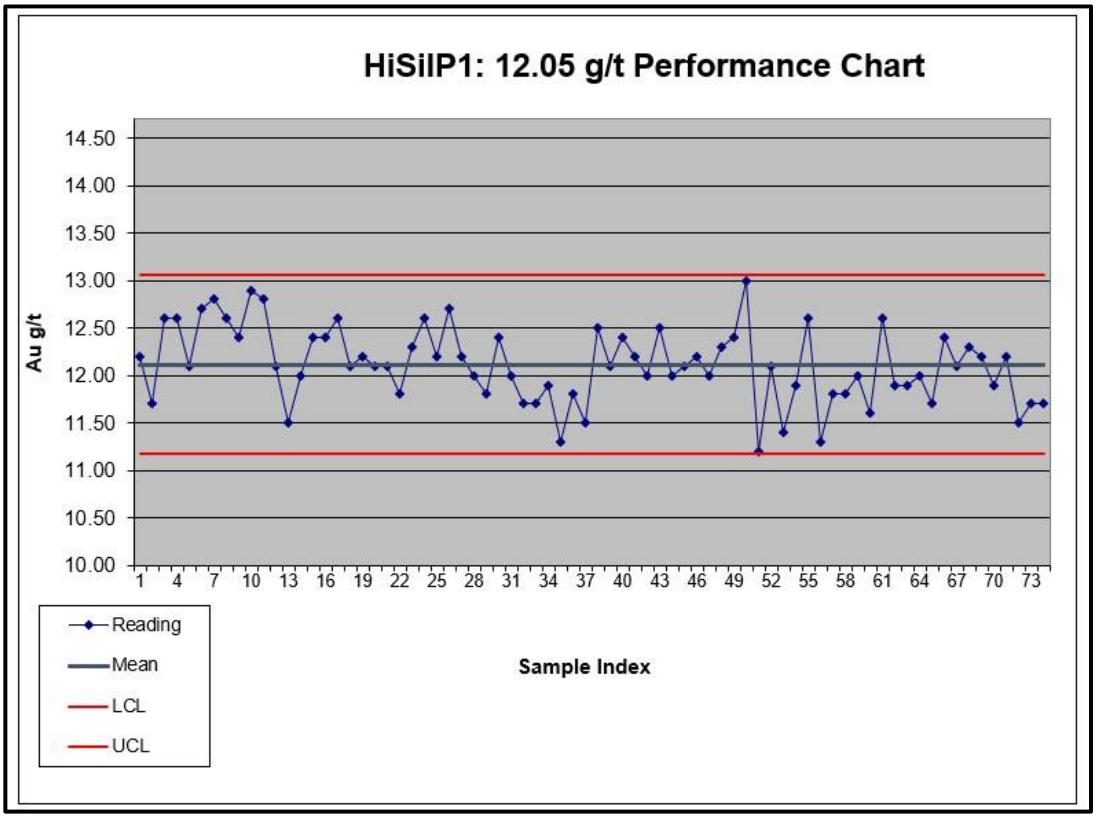


Figure 11.21: Standard Control Performance Chart for HiSilP1, Adumbi Deposit Only

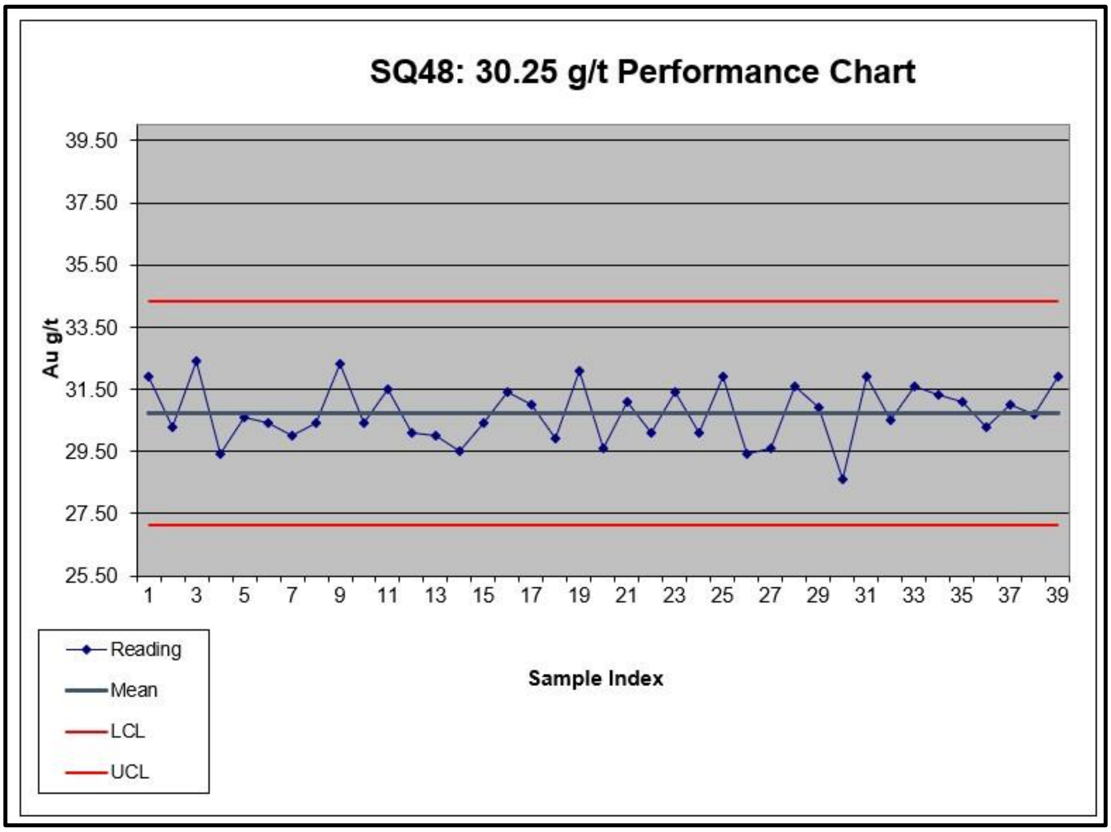


Figure 11.22: Standard Control Performance Chart for SQ48, Adumbi Deposit Only

Though the results for some batches originally sent by SGS showed failure of some of the standards as per Loncor's internal QA/QC protocols, re-assay requests were promptly sent to SGS selecting the failed standards together with samples on either side of the failed sample up to the passing standards before and after the samples for re-assaying. In almost all the cases, the re-assayed results returned grades within the accepted tolerance. In such cases, the results that accompanied the passed standard were used in the database.

For batch number 137AD, the initial results issued by SGS for sample numbers 63020 and 63021 were as follows: 63020, a standard HiSilP1 (with a certified grade of 12.05 g/t), was assigned a grade of 0.02 g/t; and 63021, a normal field sample, was assigned a grade of 12.1 g/t. This suggests a swap of the two samples, possibly during the assaying process. A re-assay request was sent to SGS, and the re-assayed result confirmed the swap. The standard was now assigned the new grade of 11.9 g/t and the normal sample a grade of 0.01 g/t. There was also a re-assay request sent for the failed standard HiSilK2 (certified grade 3.474 g/t), submitted as Sample Number 62843 in batch 128AD together with other samples initially considered failed standards to the next passing standard above and below the standard. The re-assayed result still reported a grade of 3.71 g/t, which is within tolerable limits with the accumulated standards submission. For Sample Number 71008, a standard HiSilK2 (certified grade 3.474 g/t) inserted in Batch 191AD, SGS initially reported a grade of 0.01 g/t, but upon a re-assay request they reported 3.45 g/t, which is within the acceptable range.

In the absence of the re-assayed result, Minecon carried out visual checks on the adjacent samples to the failed standards to determine the possible impact of the failure on these nearby samples. Though no clear related impact could easily be seen, Minecon recommended that these samples be retrieved and submitted for assaying as part of the inter-laboratory checks, and most of these samples were accordingly included in the samples selected for inter-laboratory checks.

The overall performance of the standards does not exhibit any bias. The frequency of the insertion of QC materials is adequate to enable the data to be used for geological modelling and resource estimation.

11.5.2 Blanks

Loncor, as part of its QA/QC programme, inserted blanks at a rate of 4 blanks in every batch of 100 samples. This was reviewed to 2 blanks in every 100 samples with the introduction of duplicates in the QA/QC protocols.

The blanks sourced from Humac Laboratories Tanzania are stored at Adumbi in 50 × 20 L storage bins in a secured place.

As a way of checking the integrity of the stored blanks, the Loncor team collected blanks from 20 different bins, labelled them as normal samples, and submitted them to the SGS Mwanza laboratory for assaying. Routinely, the integrity of the blanks was tested by fetching representative samples from each bucket and submitting them for assaying to ensure that they were reporting blank grades, which Loncor has fixed at less or equal to 0.02 g/t.

An attempt has also been made to acquire some blanks from nearer sources like Beni for use as barren material for testing.

During the 2020 to 2021 period, representative samples from some of the purchased blanks were fetched from buckets and prepared by the sample preparation laboratory, and pulps of these were submitted to SGS for analysis.

The results of the assays received are as shown in Table 11.25.

From the results in Table 11.25, 31 out of the 46 buckets tested returned grades of less than or equal to 0.02 g/t. These were considered potentially useful for barren granites and were separated from the rest, which were discarded.

Table 11.25: Results for Batch Testing of Blanks

Sample Number	Assay Result (ppm)	MW Batch	Loncor Batch No.	Description
81201	0.05	MW202297	BATCH 132IE	B3
81202	0.04	MW202297	BATCH 132IE	B8
81203	0.03	MW202297	BATCH 132IE	B9
81205	0.03	MW202297	BATCH 132IE	B12
81206	0.02	MW202297	BATCH 132IE	B13
81207	0.02	MW202297	BATCH 132IE	B14

Sample Number	Assay Result (ppm)	MW Batch	Loncor Batch No.	Description
81208	0.03	MW202297	BATCH 132IE	B15
81209	0.02	MW202297	BATCH 132IE	B16
81210	0.04	MW202297	BATCH 132IE	B17
81211	0.02	MW202297	BATCH 132IE	B18
81212	0.02	MW202297	BATCH 132IE	B19
81213	0.01	MW202297	BATCH 132IE	B20
81214	0.02	MW202297	BATCH 132IE	B21
81215	0.02	MW202297	BATCH 132IE	B22
81216	0.01	MW202297	BATCH 132IE	B23
81217	0.02	MW202297	BATCH 132IE	B24
81218	< 0.01	MW202297	BATCH 132IE	B25
81219	0.02	MW202297	BATCH 132IE	B26
81220	0.02	MW202297	BATCH 132IE	B27
81221	0.02	MW202297	BATCH 132IE	B28
81222	0.02	MW202297	BATCH 132IE	B29
81223	0.02	MW202297	BATCH 132IE	B30
81224	0.02	MW202297	BATCH 132IE	B31
81225	0.02	MW202297	BATCH 132IE	B32
81226	0.02	MW202297	BATCH 132IE	B33
81227	< 0.01	MW202297	BATCH 132IE	B34
81228	0.02	MW202297	BATCH 132IE	B35
81229	0.05	MW202297	BATCH 132IE	B36
81231	0.03	MW202297	BATCH 132IE	B37
81232	0.03	MW202297	BATCH 132IE	B38
81233	0.03	MW202297	BATCH 132IE	B39
81234	0.03	MW202297	BATCH 132IE	B40
81235	0.02	MW202297	BATCH 132IE	B41
81236	0.02	MW202297	BATCH 132IE	B42
81237	0.03	MW202297	BATCH 132IE	B43
81238	0.02	MW202297	BATCH 132IE	B44
81239	0.04	MW202297	BATCH 132IE	B45
81240	0.02	MW202297	BATCH 132IE	B46
81242	0.03	MW202297	BATCH 132IE	B47
81243	0.01	MW202297	BATCH 132IE	B48
81244	0.03	MW202297	BATCH 132IE	B49
81245	0.02	MW202297	BATCH 132IE	B50
81246	0.02	MW202297	BATCH 132IE	B51
81247	0.02	MW202297	BATCH 132IE	B52
81248	0.02	MW202297	BATCH 132IE	B53
81249	< 0.01	MW202297	BATCH 132IE	B54

For the 2020 to 2021 exploration QA/QC programme, out of the 205 blanks inserted, 3 returned grades above 0.02 g/t, which is Minecon's recommended ceiling for blanks. The blanks reported a minimum of 0.005 g/t and a maximum of 0.07 g/t.

It is worth noting that at the sample preparation laboratory, blanks are introduced into the sample processing process like any ordinary sample and thus go through the entire sample processing process that any other sample goes through. This ensures, in a way, that there is a check for cross-contamination within the sample preparation process.

The 3 blanks that failed out of 205 blanks represent 1.5 % of the blanks, which is considered satisfactory by Minecon.

Though the initial assay results that SGS reported had 14 sample grades above 0.02 g/t Au, a request for re-assay of the failed blanks and three adjacent samples on each side of the blank was made. The re-assay results that SGS reported showed that 11 of the samples passed as blanks leaving only 3 samples as true failures. Thus, the additional assaying beyond the initial report did not introduce further blank failures.

Of the three blanks that failed (sample numbers 67966 (0.03 g/t), 66598 (0.09 g/t) and 63369 (0.03 g/t) from batches 127IW, 139IW and 145AD, respectively), two of the samples failed again. Inspection of the results of the adjacent samples around Sample Number 63369 showed a grade lower than 0.03 g/t thus ruling out any possible cross-contamination. For Sample Number 66598, the samples around it reported relatively higher grades than it did, so cross-contamination cannot be completely ruled out.

Figure 11.23 shows the performance chart of all the blanks inserted for QC purposes in the 2020 to 2021 programme. Table 11.26 shows the results of the failed blanks.

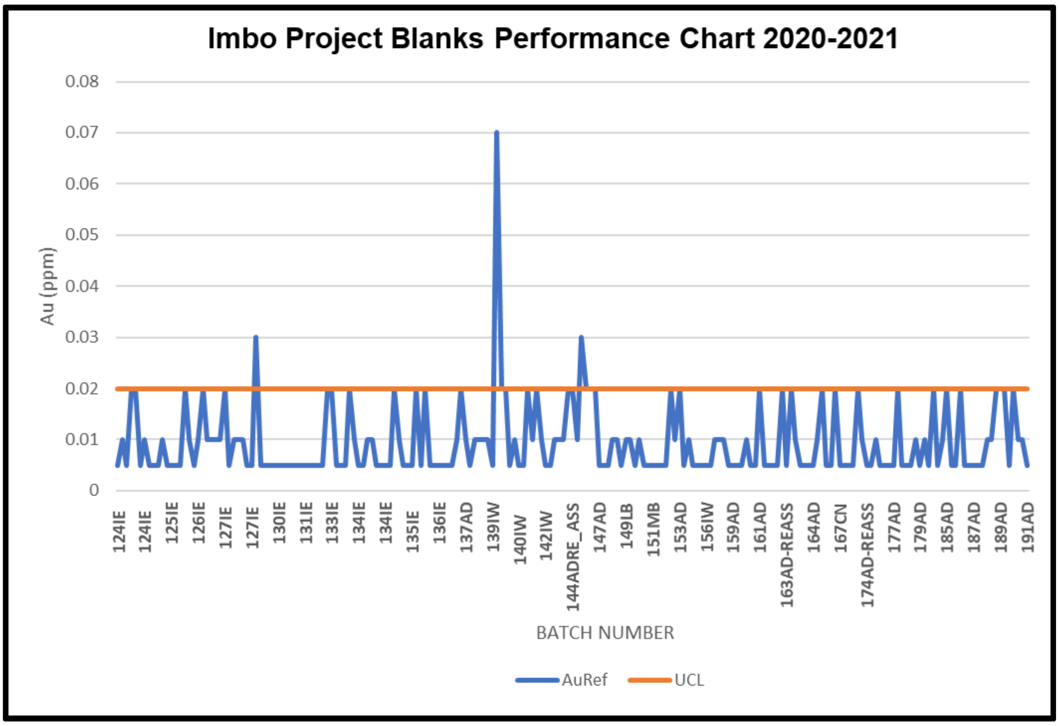


Figure 11.23: Performance Chart for All Blanks Inserted in the 2020 to 2021 Programme

Table 11.26: Results of Failed Blanks

Sample Number	Assay Result (ppm)	SGS Job No.	Loncor Batch No.	Prospect
66598	0.09	MW202438	139IW	Imbo West
66598	0.07	MW202438	139IW Re-assay	Imbo West
67966	0.03	MW202292	127IW	Imbo West
63369	0.03	MW210120	145AD	Adumbi
63369	0.03	MW210121	145AD Re-assay	Adumbi

11.5.3 Duplicates

Following from Minecon's recommendations in the April 17, 2020, NI 43-101 Technical Report on the need for duplicates to be included in Loncor's QA/QC programme, collection of duplicates was introduced into the process at the sample preparation phase in the sample preparation laboratory.

Duplicates are vital in QA/QC programmes as they assist in determining the repeatability or variability even at the local stage (nugget effect) inherent with sampling the same interval and detecting sample number mix-ups and even sample swapping. Duplicates are collected and inserted at a rate of 1 in every 50 samples.

Duplicates, like standards, are used to monitor the laboratory precision in various grade ranges; the duplicates selected should be within the potential mineralised zones with varying grade ranges to test the repeatability of grades in a wider range of grades. Duplicate samples can be field (core, trench or underground), coarse (crushed reject), or pulp (pulverised reject) duplicates.

The duplicates used in the QAQC report are second pulp splits collected at predetermined points during the sample preparation process. They are given different numbers from the original samples and submitted within the same batch to the assay laboratory for analysis.

Figure 11.24 shows the original versus duplicate sample assay plots inserted for QC in the 2020 to 2021 programme.

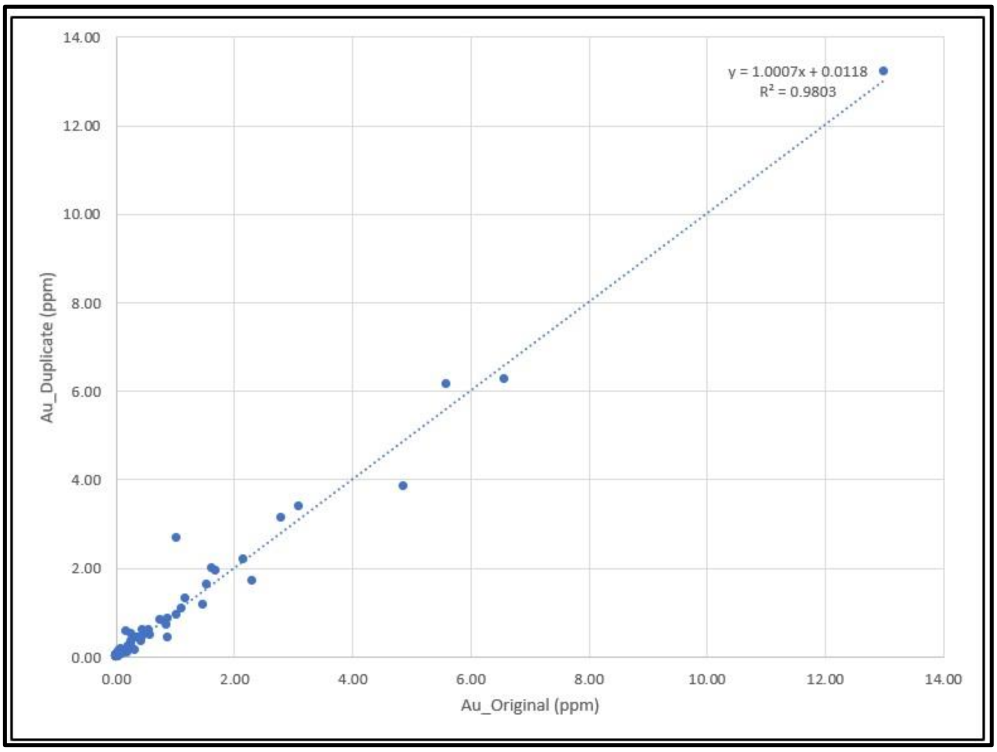


Figure 11.24: Original Versus Duplicate Assay Plots for Duplicates Inserted in the 2020 to 2021 Programme

The chart shows a good correlation between the original samples and their duplicates. This indicates a high repeatability of results with a high correlation co-efficient in the 0.98 region.

11.5.4 Inter-Laboratory Checks

Loncor submitted samples for inter-laboratory checks on a routine basis to ALS Chemex, RSA, which is independent of the company, which acts as the umpire laboratory to the primary laboratory, SGS Mwanza. Duplicate pulp samples covering the entire grade range were selected and dispatched to ALS. Loncor dispatched two batches of 200 samples each, including quality control materials for the same analytical method, for the first half of 2021, and another two batches were sent for the second half of 2021 later.

The initial results obtained have been checked and compared with the results obtained from SGS Mwanza. A comparison was done for the entire sample grade range and plotted as a chart (see Figure 11.25). The chart shows a generally good correlation between the assay results provided by the two analytical laboratories. To determine any potential bias in the higher-grade results (ore), the data was subsequently divided into less than 1.0 g/t and greater than 1.0 g/t. The mean absolute relative difference (MARD) has been calculated for the separated data. Table 11.27 shows the inter-laboratory comparison: SGS Mwanza vs ALS Chemex, RSA.

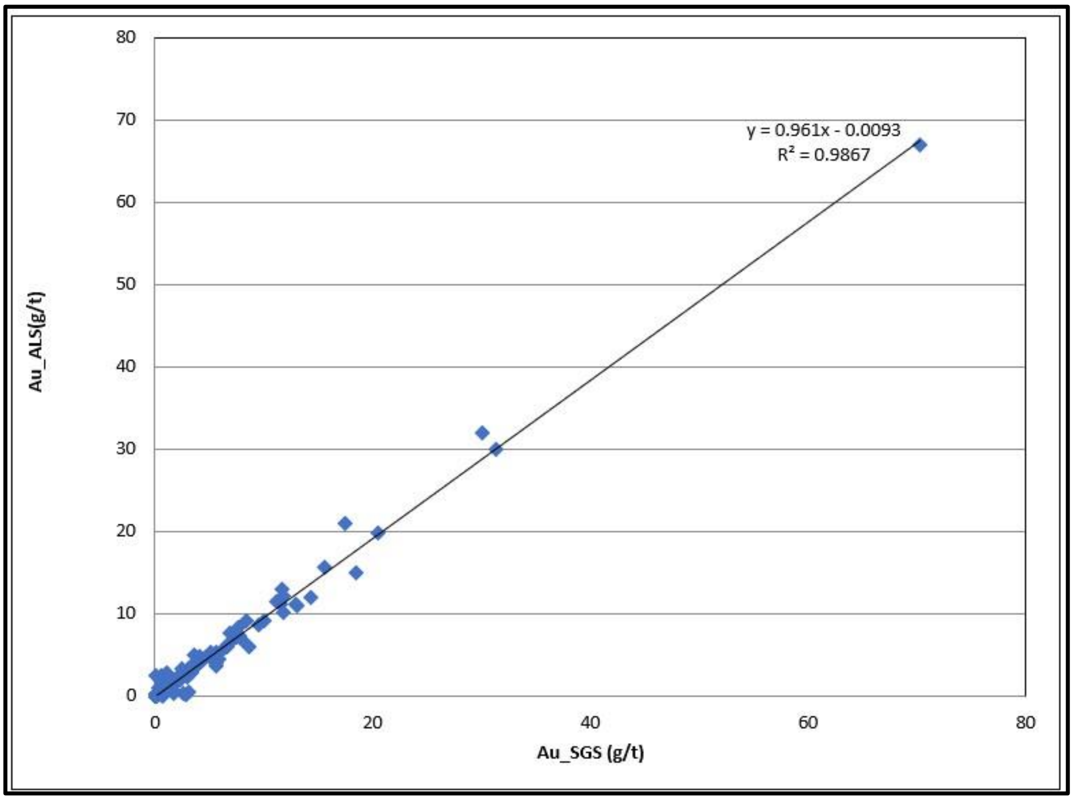


Figure 11.25: 2021 Inter-Laboratory Assay Comparison: SGS_MWZ vs ALS

Table 11.27: Inter-Laboratory Comparison: SGS Mwanza vs ALS Chemex, RSA

Au < 1.0 g/t	SGS MWZ	ALS_RSA	Au >1.0 g/t	SGS MWZ	ALS_RSA
Count	99	99	Count	88	88
Minimum (g/t)	0.005	0.005	Minimum (g/t)	1.05	1
Maximum (g/t)	0.93	1	Maximum (g/t)	70.20	67.00
Mean (g/t)	0.19	0.18	Mean (g/t)	6.46	6.23
Standard Deviation	0.23	0.24	Standard Deviation	8.99	8.70
MARD %	**3.51**		**MARD %**	**3.55**	

The results obtained do not show any significant differences or bias between the results reported by the two analytical laboratories even though for the period covered by the results, SGS appears to be reporting slightly higher grades than ALS. Upon receipt and analysis of subsequent results, further studies will be conducted to determine periodic trends.

11.5.5 Sample Preparation Laboratory External Independent Audit

The Adumbi on-site sample preparation laboratory was successfully audited by SGS in September 2021.

11.5.6 Review of External Laboratory Internal QA/QC Programme

The SGS Mwanza laboratory uses standards, blanks, duplicates and replicates as part of its internal QA/QC checks. The results of the standards and blanks used are reported below the results of the samples submitted by Loncor in their respective batches.

The frequency of the QC materials usage is as follows:

- 2 standards in a batch of 50 samples
- 1 preparation blank (prep process blank) in every 50 samples
- 1 reagent blank in every 50 samples
- 1 weighed replicate in every 50 samples
- 1 preparation duplicate (re-split) in every 50 samples

Minecon has reviewed the batch-by-batch results submitted by SGS and also the internal QC reports that SGS submitted during the period that they processed the Loncor samples. All the QA/QC materials performances are in order. Hence, there is no evidence of contamination or lack of precision in the laboratory processes.

A diverse grade range of standards from a broad grade range of 0.19 g/t Au to 16.2 g/t Au and standards of different material matrices were used by SGS Mwanza during the period under review. All except one of the standards passed their three standard deviation tolerance from the mean limit (SGS internal QA/QC protocol). In addition, all 231 blanks inserted by SGS during the period passed, reporting no grade above 0.02 g/t. Table 11.28 is a summary of the QC materials used by SGS Mwanza and the grade ranges reported.

Table 11.28: QC Materials Inserted by SGS in Samples Analysed for Loncor in 2020 to 2021

QC Material Type	Count	Minimum (g/t)	Maximum (g/t)
Blank	231	< 0.01	0.02
Standard	453	0.19	16.2

Replicates also confirmed good repeatability.

The umpire laboratory ALS Chemex RSA, used for inter-laboratory analysis, also used their internal QC material with the Loncor samples that they analysed and provided results which showed that all the materials passed their internal QA/QC protocols.

11.6 RECOMMENDATIONS

Minecon recommends the following:

- Loncor should continue with the process of umpire check on the SGS results. Although the process has commenced, more routine (quarterly) inter-laboratory check will be required to enhance the quality control process.
- Loncor should continue with the routine independent audit of the sample preparation laboratory by external auditors to maintain standards and adopt new ideas.

12 DATA VERIFICATION

Additional information regarding the Imbo Project with respect to data verification is set out in Minecon's technical report dated April 17, 2020, entitled "Independent NI 43-101 Technical Report, on the Imbo Project, Ituri Province, Democratic Republic of the Congo" (available from SEDAR at www.sedar.com).

The information in this section relates to Loncor's current exploration programme at Adumbi.

12.1 SITE VISIT

A site visit was carried out by Daniel Bansah, Chairman and Managing Director of Minecon, from February 12 to 20, 2020. Christian Bawah was also on site for a period of eight weeks from October to November 2020. Mr Bawah was accompanied by Peter Kersi, a contributing engineer to this report. Also on the trip were the following Minecon geologists and other technical personnel: Bel Mapendo, chief geologist, Patient Zamakulu, senior geologist, and three of Minecon's laboratory technical and operational staff.

Tasks undertaken during the visit included a technical inspection of the site, an inspection of the old drill core, a review of all the technical work carried out from 2014, including work carried out following RPA's 2014 recommendations but not limited to the sampling and drill site protocols and security, as well as QA/QC issues and the ALS Minerals on-site sample preparation facility.

Gordon France, Minecon's Database, GIS and IT Manager, visited the Adumbi site for seven weeks from June to July 2021. The scope of work during the visit was to ensure that the Adumbi database was migrated onto a centralised data repository (the Century Database System).

In September 2021, Mr Bansah undertook another visit to the Adumbi site. During the visit, he spent time reviewing all the field geological activities undertaken on the Adumbi deposit, the geological logging and sampling procedures, including the sampling preparation protocols carried out in the sample preparation laboratory. Mr Bansah also reviewed the geological interpretation work carried out by Minecon's site team.

The Minecon team worked in collaboration with Fabrice Matheys, Loncor's General Manager and geologist with +25 years of experience in the DRC and the African region.

The following list summarises Minecon's site visit comments with reference to the CIM Exploration Best Practices Guidelines:

- **Qualified Person** – Loncor's General Manager, Fabrice Matheys, is a very experienced geologist with many years of DRC exploration experience, particularly on the Ngayu Greenstone Belt.
- **Geological Concept** – Loncor has developed a robust geological deposit and structural model that will guide future exploration from target generation, drilling and evaluation. A review of the results of the holes drilled in Adumbi in 2020 to 2021 confirms the down-plunge extension of the mineralisation.

- **On-Site Sample Preparation and QA/QC Controls –** The on-site sample preparation laboratory was originally set up and managed by ALS Minerals with the requisite standards but was not operational for the 2013 to 2017 exploration programme. Minecon provided the needed technical skills and management to provide guidelines to recommission the on-site sample preparation laboratory and provided the needed skills to improve the QA/QC procedures to align with Industry Standards. An analysis was carried out by the SGS Mwanza analytical laboratory in Tanzania. As part of the audit trail, SGS Mwanza carried out an independent audit of the sample preparation laboratory in September 2021.
- **Data Capturing and Standard Operating Procedure –** In Minecon's opinion, Loncor has a comprehensive procedural manual SOP for all data capture. Minecon has worked with Loncor, and all the Loncor data is currently being migrated into a centralised database management system (FUSION), which is more secure than the storage of data in MS Excel format.
- **Core Photographs –** Minecon has developed a modified platform that allows core photographs to be taken from a fixed location with a stationary camera with enhanced and consistent resolution.
- **Sampling –** Sampling procedures are appropriate to the deposit style. Samples are collected under the supervision of key technical personnel who are trained by the QP. Key personnel understand why they employ the various sampling methods. Duplicate sampling has been introduced to raise the QA/QC measures to best industry standards.
- **Drilling –** Drilling procedures are appropriate to the deposit style. Core recovery in the weathered profile (oxide) is poor. For the extension and deep drilling programme, the mineralised zones were intersected at a depth which has competent rocks, and excellent recoveries were achieved. For future infill drilling, an appropriate RC rig will be secured to manage the shallow infill holes.
- **Sample Security –** Sample storage and sample security procedures are found to be robust and appropriate.
- **Database Management Audit –** Minecon identified some minor issues with the MS Excel database that was used for the previous modelling and resource estimation. The migration of the database into a more secured platform and the training and mentoring of the database administrator have improved the security of the database. Periodic independent database audits by external technical consultants are recommended to ensure that the database is in good order and that minor data issues can be identified and fixed. Following the external audit of the database, a compliance certificate can be issued.
- **Health, Safety, Environment and Community (SHEC) –** SHEC procedures currently in place on site need improvements, and site-based protocols and reporting should be better structured. The personal protective equipment (PPE) is adequate for this level of exploration programme on site. Steps should be taken to systematically backfill all the open trenches. Minecon also recommends a structured and a more routine engagement with the community and other stakeholders including government structures even though community relations at local, district, provincial and central government level appear good.

12.2 DRILLHOLE, TRENCH AND ADIT DATA

Currently, all the forms of project data that were stored in MS Excel and other data formats are being migrated into a secured industry standard database system, FUSION.

The Datamine Studio RM version 1.6.8.7.0. (Datamine) software has been applied by Minecon on the modelling data for verification, validation and manipulation of the Adumbi drillholes, adit and trench data using the inherent verification, validation and manipulation protocols within the Datamine software.

Prior to the mineral resource updates, Minecon's technical personnel consistently carried out verification and validation exercises, including "from and to intervals" and "end of hole depths". The lithological description of two of the zones on one hole was reviewed with site geologists and was modified to conform to the lithology of that section from the drill core.

Statistical manipulation of the uploaded assay data from the submitted databases showed that several samples reported Au grades of 0. Further checks need to be done to verify these as analytical laboratories do not report 0 g/t Au.

12.3 INDEPENDENT AUDIT AND WITNESS SAMPLING

Minecon independently reviewed and audited the Adumbi database. During the audit, Minecon identified that the majority of the resource database was stored on MS Excel data sheets and was in good order, and only minor data issues were identified. All the data that was flagged as having minor issues was isolated and corrected before being released and added to the database. Minecon is currently assisting Loncor to migrate the cleaned-up data into a centralised database repository system, FUSION.

No independent witness sampling was carried out on the six new holes as Minecon technical personnel were involved in the sampling process. On the previous samples, Minecon also did not carry out any independent witness sampling. For this, Minecon has relied on the previous independent witness samples collected and analysed during RPA's site visit of 2013 and concurs with the conclusions of that study.

12.4 DISCUSSION

Minecon is currently supporting Loncor to migrate all the cleaned-up data sets into an industry standard secured centralised database repository and management system. This will ensure data security and will minimise potential data errors.

A full-time database administrator has been employed by Loncor at the Adumbi site to manage the database. Minecon's database manager is helping to train the database administrator using a customised front-end application that has been designed for data entry, reporting, and viewing via open database connectivity (ODBC), which utilises the data validation procedures from the central database. All the other geological software databases on site will be linked to retrieve information from a centralised repository.

Validated assay data from the assay certificates will be imported directly from the laboratory. This task can be undertaken only by fully trained and authorised network users.

12.5 RECOMMENDATIONS

Minecon is happy with the speed of the migration of the database into the industry-standard secured centralised database repository and management system but recommends that the implementation and training process be expanded to other Loncor technical personnel and not just the database administrator.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

SENET was requested by Loncor to conduct a PEA on the Adumbi deposit. SENET developed a metallurgical test work programme the results of which were used to develop the process flowsheet and determine values to size and design the process plant.

Previous metallurgical test work was conducted on oxide and sulphide ore at the Wardell Armstrong International (WAI) laboratory, and the test work findings are in the following reports/documents:

- WAI, August 2011, “Characterization Testwork on Samples of Gold Ore from Adumbi Deposit, Democratic Republic of Congo”, Report Number MM584.
- WAI, October 2011, “Optimization Testwork on Samples of Gold Ore from Adumbi Deposit, Democratic Republic of Congo”, Report Number MM601.
- WAI, December 2011, “Flotation and Leach Optimization Testwork on Samples of Gold Sulphide Ore from Adumbi Deposit, Democratic Republic of Congo”, Report Number MM626.
- RPA, February 2014, “Technical Report on the Somituri Project Imbo Licence, Democratic Republic of the Congo – NI 43-101”.

13.2 SUMMARY

Historical metallurgical test work conducted in 2011 on oxide and fresh ore indicated the following:

- Comminution Bond ball work index (BBWi) test work indicated that the oxide and fresh ores are medium hard with BBWi values of 10.46 kWh/t and 11.76 kWh/t, respectively.
- Both the oxide and fresh ores respond well to gravity concentration.
- The oxide ore is free milling.
- The fresh ore contains both refractory and non-refractory gold.
- Further metallurgical test work was performed on the refractory fresh sample to try to improve gold recovery. The results were as follows:
    - The ore responded well to flotation, giving 96 % gold recovery to a rougher concentrate.
    - Fine milling the flotation concentrate to 80 % passing 10 µm with oxygen sparging gave a low gold extraction of 18.4 %.
    - Use of kerosene and lead nitrate did not improve gold extraction.
    - Roasting (an aggressive oxidation process) was not effective and increased gold extraction on the flotation concentrate to 63.2 % only.

The PEA test work was conducted in 2021 on oxide, transition and fresh samples. Table 13.1 shows a summary of the PEA test work results.

Table 13.1: Summary of the PEA Test Work Results

Parameter		Oxide	Transition	Fresh	
				Fresh RP	Fresh BIF
Ore Characterisation	Specific Gravity (SG) – SENET	2.85	3.07	3.09	3.17
	SG – OMC	2.70	2.80	2.90	
	As (ppm)	2,133	4,443	12,877	7,008
	Ag (g/t)	0.87	1.10	0.07	0.08
	Bulk Density	1.80	–	3.00	
	Au (g/t)	1.34	3.25	–	–
Comminution	BBWi (kWh/t) – SENET	11.58	13.6	14.6	
	BBWi (kWh/t) – OMC	11.8	13.7	14.2	
	Bond Rod Work Index (BRWi)	–	–	–	
	Uniaxial Compressive Strength (UCS)	–	–	–	
	Crushability Work Index (CWi)	–	–	–	
	Abrasion Index (Ai) (g) – SENET	0.1899	0.2519	0.3560	
	Ai (g) – OMC	0.19	0.25	0.34	
Gold Recovery	Proposed Process Route	Gravity + CIL	Gravity + CIL	Gravity + CIL	Gravity + CIL
	Gravity Recovery (%)	36.82	31.58	29.61	31.66
	Intensive Leach Reactor (ILR) (%)	96.25	94.61	83.53	85.19
	CIL on Gravity Middlings and Tailings (G M&T) (%)	89.05	88.23	75.34	63.25
	Overall Recovery – Gravity + Leach (%)	91.70	90.24	77.77	70.20
	Gold Recovery (used for Mining and Process Design) (%)	90.76	87.53	80.10	89.83
	Cyanide Consumption (kg/t)	0.87	1.19	1.45	0.91
	Lime Consumption (kg/t)	3.09	4.85	2.00	4.11
	Rougher Flotation Recovery (% Au)	–	95.52	93.03	85.13
	Rougher Mass Pull (% w/w)	–	16.83	26.59	21.76
	Rougher Flotation Recovery (% S)	–	95.52	93.03	85.13
	As-is Leach on Float Concentrate – Oxygen Sparging (%)	–	90.32	61.42	65.81
	Ultrafine Grinding (UFG) (12 µm) Leach on Float Concentrate – Oxygen Sparging (%)	–	91.27	75.53	66.74
	As-is Leach on Float Tailings – Oxygen Sparging (%)	–	86.89	73.19	66.36
	Overall Gold Recovery – Gravity – Flotation – Cyanidation (%)	–	91.77	77.73	72.57

The Adumbi ore responded well to gravity. Gravity followed by cyanidation on the oxide and transition ores gave good overall gold recoveries of 91.70 % and 90.24 %, respectively.

However, the fresh RP and BIF gave lower gold recoveries of 77.77 % and 70.20 %, respectively. Due to the low recoveries on the fresh RP and BIF, flotation was investigated to try and improve gold recoveries. Flotation on the transition, fresh RP and BIF showed rougher flotation recoveries of 95.52 %, 93.03 % and 85.13 %, respectively.

The flotation concentrate samples generated were not sufficient to enable further processing routes such as

- Fine milling followed by leaching with oxygen addition
- Fine milling followed by partial oxidation using high shear reactors and leaching
- Albion process
- Pressure oxidation
- Bio leaching
- Roasting

These recovery processes will be investigated during the next phase of the project to optimise the gold recovery in the transition and fresh ore types.

13.3 REVIEW OF HISTORICAL TEST WORK

The historical test work indicated that the oxide ore is free milling. Leach on the sulphide gravity middlings and tailings showed gold dissolution ranging from 39.5 % to 94.1 %, indicating that the sulphide ore contained both refractory and non-refractory gold.

The refractory sulphide sample was taken for flotation tests followed by leach on the float products. Fine milling the concentrate to 80 % passing 10 µm, followed by cyanidation with oxygen sparging yielded a poor gold dissolution of 18.4 %.

Pre-treating the flotation concentrate with kerosene and adding lead nitrate did not improve gold recovery on the flotation concentrate.

Roasting was investigated, and this improved gold extraction on the float concentrate to 63.2 %. Roasting was also investigated on the feed (whole ore) and yielded 50.7 % gold extraction.

A summary of the historical test work results is given below.

13.3.1 Report MM584

Table 13.2 provides a summary of WAI Report MM584: “Characterization on Samples of Gold Ore from Adumbi Deposit, Democratic Republic of Congo”, August 2011.

Table 13.2: Summary of Test Work Results – Report MM584

Test Work			Oxide				Sulphide			
			Ox 1	Ox 2	Ox 3	Ox Comp	Sulp 1	Sulp 2	Sulp 3	Sulp Comp
Head Assays		Au (g/t)	5.88	1.35	1.85	–	4.76	8.71	5.25	–
		Ag (ppm)	1.50	1.00	1.49	–	4.49	2.49	4.48	–
		S (%)	0.07	0.07	0.02	–	8.88	1.97	3.29	–
Comminution		BBWi (kWh/t)	–	–	–	10.46	–	–	–	11.76
Gravity Recovery		Au (%)	34.4	16.6	21.1	–	1.4	13.1	26.2	–
		Ag (%)	13.5	8.6	8.1	–	1.1	12.2	4.2	–
Leach on Gravity Tails	Dissolution	Au (%)	85.6	91.4	88.4	–	39.5	88.5	94.1	–
		Ag (%)	96.9	95.2	98.3	–	48.8	99.0	65.5	–
	Cyanide Consumption	kg/t	1.50	1.16	1.35	–	1.14	2.45	1.90	–
	Lime Consumption	kg/t	0.37	0.39	0.25	–	0.28	0.21	0.16	–
	Residence Time	h	24	24	24	–	36	36	36	–
Coarse Bottle Roll at 20 mm	Dissolution	Au (%)	75.6	88.2	78.9	–	–	–	–	–
		Ag (%)	57.6	33.3	73.5	–	–	–	–	–
	Cyanide Consumption	kg/t	1.52	1.46	1.37	–	–	–	–	–
	Lime Consumption	kg/t	0.15	0.14	0.14	–	–	–	–	–
Coarse Bottle Roll at 15 mm	Dissolution	Au (%)	72.4	86.4	80.8	–	–	–	–	–
		Ag (%)	55.8	39.9	53.8	–	–	–	–	–
	Cyanide Consumption	kg/t	1.66	2.68	1.47	–	–	–	–	–
	Lime Consumption	kg/t	0.08	0.14	0.12	–	–	–	–	–
Coarse Bottle Roll at 12.5 mm	Dissolution	Au (%)	76.7	85.7	78.3	–	–	–	–	–
		Ag (%)	51.9	47.9	70.8	–	–	–	–	–
	Cyanide Consumption	kg/t	1.46	1.30	1.43	–	–	–	–	–
	Lime Consumption	kg/t	0.3	0.24	0.18	–	–	–	–	–

The results indicate the following:

- Comminution test work – BBWi values were10.46 kWh/t and 11.76 kWh/t for the oxide and sulphide ore, respectively, indicating that both the oxide and sulphide ore are medium hard for ball milling.
- Gravity gold recovery on the oxide ore ranged from 16.6 % to 34.4 %, showing that the ore is likely amenable to gravity recovery.
- Gold dissolution on the oxide ore gravity middlings and tailings ranged from 85.6 % to 91.4 %.
- Gravity gold recovery on the sulphide ore ranged from 1.4 % to 26.6 %, showing that the ore is likely amenable to gravity recovery.

- Gold dissolution on the sulphide ore was low, ranging from 39.5 % to 94.1 %. The Sulphide 1 sample showed a low gravity recovery and low gold dissolution values, indicating that it is refractory. This shows that the sulphide ore comprises both refractory and non-refractory gold.

Simulated heap leach coarse bottle rolls done at varied crush sizes on the oxide ore indicated that the oxide ore could potentially be amenable to heap leaching.

13.3.2 Report MM626

Since the Sulphide 1 sample showed poor response to gravity and cyanidation, flotation followed by cyanidation was investigated on this sample to try to improve gold recoveries.

Table 13.3 provides a summary of WAI Report MM626: “Flotation and Leach Optimization Testwork on Samples of Gold Sulphide Ore from Adumbi Deposit, Democratic Republic of Congo”, December 2011.

Table 13.3: Flotation Optimisation Test Work

Test		Concentrate	Cumulative Weight (%)	Cumulative Assay		Cumulative Recovery (%)	
				Au (g/t)	S (%)	Au	S
Effect of Primary Grind	FT1 (150 µm)	1	7.96	19.46	35.14	33.60	31.08
		1 – 2	22.23	14.39	28.79	69.43	71.14
		1 – 3	28.47	13.31	26.85	82.23	84.99
		1 – 4	31.82	12.77	25.53	88.16	90.30
	FT2 (106 µm)	1	12.49	17.24	31.38	50.43	44.91
		1 – 2	25.02	12.91	27.26	75.64	78.16
		1 – 3	31.42	11.96	25.57	88.04	92.06
		1 – 4	34.54	11.48	24.36	92.95	96.48
	FT3 (53 µm)	1	10.77	11.39	29.12	31.51	35.13
		1 – 2	19.51	10.76	26.30	53.93	57.49
		1 – 3	29.12	10.63	24.90	79.51	81.26
		1 – 4	33.20	10.43	24.18	89.02	89.97
Reagent Optimisation	FT4 (200 g/t PAX)	1	13.66	17.27	32.37	53.03	46.09
		1 – 2	27.29	13.74	28.86	84.29	82.10
		1 – 3	31.83	12.80	27.58	91.59	91.50
		1 – 4	34.95	12.04	26.13	94.59	95.19
	FT5 (100 g/t PAX and 50 g/t MaxGold 900)	1	18.04	17.25	30.50	65.52	58.30
		1 – 2	31.36	13.78	27.67	91.00	91.95
		1 – 3	34.74	12.97	25.89	94.85	95.32
		1 – 4	36.88	12.38	24.85	96.14	97.12

13.3.2.1 Primary Grind Optimisation

The effect of primary grind results indicated the following:

- The concentrate grade dropped with finer grinding. An optimum grind of 80 % passing 150 µm was selected. As a result, less emphasis was placed on the grade of the concentrate in order to maximise the recovery of gold to the concentrate.
- Results showed that after the full 15 min of flotation, the highest overall recovery of gold, 93.0 %, was achieved in Test 2, which was performed at 106 µm.

The variation between maximum and minimum gold recovery was just 4.8 %; therefore, the decision was taken to proceed with the coarsest grind size (150 µm). Although the finer grind sizes achieved marginally higher recoveries, it was felt that the increased grinding cost was likely to outweigh any benefit from the increased recovery.

13.3.2.2 Reagent Optimisation

The results of the two tests showed that doubling the amount of the potassium amyl xanthate (PAX) collector that was added increased the gold recovery by 6.4 % to 94.6 %.

However, when the secondary MaxGold 900 collector was added in addition to the primary PAX collector (Test 5), gold recoveries increased by a further 1.5 % to 96.1 % overall. This result was also 3.1 % higher than that achieved at the 106 µm grind size.

In addition, the results also showed that the increase in recovery had almost no effect on the grade of gold present within the concentrate, with values ranging from 12.4 g/t Au in Test 5 to 12.8 g/t Au in Test 1.

The Test 5 regime offered the best performance in terms of gold recovery and, as such, was selected as the basis for cleaner flotation tests.

13.3.2.3 Effect of Cleaning

The results of the kinetic cleaner flotation test showed that cleaning had no appreciable benefit with respect to either the gold grade or rejection of mass from the concentrate.

Whilst higher concentrate grades were achievable, these were achieved with a corresponding reduction in gold recovery, which was in line with the grades and recoveries achieved during the initial roughing stage.

It was therefore clear that the addition of direct cleaning of the rougher concentrate was of no overall benefit, and as such, the decision was taken to proceed based on the preparation of a rougher concentrate only.

13.3.2.4 Test Work on the Float Concentrate

A rougher concentrate was produced for concentrate handling test work.

The rougher concentrate was fine milled to 80 % passing 10 µm and leached with air sparging and oxygen sparging. Cyanidation with air sparging gave 11.3 % gold dissolution while oxygen sparging gave 18.4 %.

Pre-conditioning with kerosene and adding lead nitrate did not improve gold recovery on the flotation concentrate.

Roasting (an aggressive oxidation process) was not effective and increased gold extraction on the flotation concentrate to 63.2 % only.

13.4 PEA TEST WORK

The PEA test work was performed at Maelgwyn Mineral Services Africa (MSA) in South Africa.

13.4.1 Sample Selection

The following samples were collected and were shipped to MSA for test work:

- **Oxide**: Three channel samples of approximately 16 kg each were collected along the strike of the oxide mineralisation from three separate adits and labelled LA-0-1, LA-0-2 and LA-0-3. The grade of all the oxide samples ranges from 2.5 g/t to 2.75 g/t Au.
- **Transition**: Three composite crushed cores were drilled from a shallow, strategically placed hole. Approximately 10 kg to 16 kg of three composite samples (i.e. −2 mm crushed samples in sample preparation storage) labelled LA-T-1, LA-T-2 and LA-T-3 were collected. The transition mineralised samples consist of well-defined BIF, RP and QCS host rock. The three transition composite samples collected were labelled LA-T-1BIF, LA-T-1RP, LA-T-1QCS. The grade of the samples ranges from 2.33 g/t to 4.99 g/t Au.
- **Fresh**: A total of eleven composite crushed core samples (i.e. −2 mm crushed core samples in sample preparation storage) were collected from six core holes along strike at shallow and deep parts of the fresh rock. For the six shallower fresh composites, samples were collected from holes LADD001, LADD003 and LADD008. For the five deeper fresh composites, samples were collected from LADD007, LADD009 and LADD014. For each of the six core holes, two composite (BIF, RP and/or QCS zones) samples were collected, except for LADD014 where only one sample was collected. The composite BIF/QCS grade ranges from 2.60 g/t to 3.06 g/t Au, and the composite RP grade ranges from 3.48 g/t to 5.17 g/t Au.

13.4.1.1 Abrasion and Comminution Test Work Samples

For this part of the metallurgical test work, the following samples were collected:

- For the fresh material, approximately 20 kg of quartered/half core was collected and labelled AC-F-1.
- For the transition material, approximately 20 kg of quartered/half core was collected and labelled AC-T-1.
- For the oxide material, approximately 20 kg of lumpy material from the adits was collected and labelled AC-O-1.

Details of the samples collected for the diagnostic as well as the abrasion and comminution test work are given below.

13.4.1.2 Adumbi Metallurgical Sampling

Samples were taken for the diagnostic test work as follows:

- Oxide: three composite samples from three adits (see Table 13.4)
- Transition: three composite samples from crushed cores from LADD017 (see Table 13.5)
- Fresh: 11 composite samples from crushed cores from six drillholes: LADD001, LADD003 and LADD008 (shallower holes); LADD007, LADD009 and LADD014 (deeper holes) (see Table 13.6)

Table 13.4: Diagnostic Test Work – Oxide

Adit Number	Sample Number	Sample ID	Sample Weight (kg)	Composited Grade (g/t Au)
SAAD001	78552, 78553, 78554, 78555, 78556, 78557, 78558	LA-O-1	16.2	2.75
SAAD002	78564, 78565, 78566, 78567	LA-O-2	16	2.55
SAAD006	78573, 78574, 78575, 78576	LA-O-3	16	2.50
TOTAL			**48.2**	

Table 13.5: Diagnostic Test Work – Transition

Drillhole Number	Sample Number	Composite Number	Sample Weight (kg)	Composited Grade (g/t Au)
LADD017	65213, 65214, 65215, 65220	La-T-1RP	15.3	4.14
LADD017	65221, 65222, 65223, 65224, 65225	La-T-1QCS	16.8	2.33
LADD017	65268, 65272, 65273, 65281, 65282	La-T-1BIF	11.7	4.99
Total			**43.8**	

Table 13.6: Diagnostic Test Work – Fresh

Drillhole Number	Sample Number	Composite Number	Sample Weight (kg)	Composited Grade (g/t Au)
LADD001	62756, 62765, 62811, 62812, 62828, 62832, 62869, 62890, 62928, 62932, 62937, 62938, 62939	LA-F-1BIF	14.8	2.72
LADD001	62790, 62791, 62792, 62808, 62851,62852, 62853	LA-F-1RP	16.2	3.67
LADD003	63030, 63071, 63072, 63125, 63172, 63184, 63186, 63187, 63194, 63206	LA-F-1QCS	16.1	2.60
LADD003	63119, 63120, 63121, 63122, 63123, 63127, 63128, 63154, 63156, 63158, 63165, 63188	LA-F-2RP	14.5	5.06
LADD007	63685, 63736, 63753, 63759, 63760, 63767, 63768, 63770, 63775	LA-F-3BIF	14.4	2.65
LADD007	63776, 63777, 63778, 63783, 63798, 63799, 63800, 63801	LA-F-4RP	15	5.17

Drillhole Number	Sample Number	Composite Number	Sample Weight (kg)	Composited Grade (g/t Au)
LADD008	63912, 63914, 63915, 63932, 63933, 63936, 63937, 63941, 64030	LA-F-3RP	14.8	4.57
LADD008	63899, 63904, 63909, 63910, 63925, 63975, 63976, 63977, 64015, 64023, 64029	LA-F-2BIF	16.9	2.79
LADD009	64056, 64057, 64077, 64084, 64085, 64116, 64119, 64120, 64157, 64158, 64160, 64162	LA-F-4BIF	13.9	3.02
LADD009	64104, 64107, 64109, 64149, 64150, 64151	LA-F-5RP	11.5	4.80
LADD014	64853, 64855, 64856, 64866, 64867, 64868	LA-F-6RP	13.7	3.48
TOTAL			**176.3**	

Oxide samples were taken for the abrasion and comminution test work as shown in Table 13.7.

Table 13.7: Abrasion and Comminution Test Work – Oxide

Drillhole or Adit Number	Sample Number	Sample ID	Sample Weight (kg)	Composited Grade (g/t Au)
LADD009	64056, 64057, 64084, 64116, 64119, 64157, 64158, 64160, 64162	AC-F-1	19.8	3.64
LADD017	65276, 65280, 65283, 65284, 65286, 65288, 65290, 65224, 65225	AC-T-1	20.1	3.60
SAAD001	78560, 78561, 78562, 78563	AC-O-1	20.4	4.19
TOTAL			**60.3**	

13.4.2 Test Work Conducted

SENET proposed metallurgical test work for the PEA study. The metallurgical test work was conducted on oxide, transition and fresh ore samples and covered the following:

- Comminution test work:
  - BBWi
  - Ai
- Gold recovery test work:
  - Head assays
  - Diagnostic leach
  - Gravity recovery
  - Flotation
  - Cyanidation tests
  - Flotation recovery

Figure 13.1 shows an outline of the PEA test work.

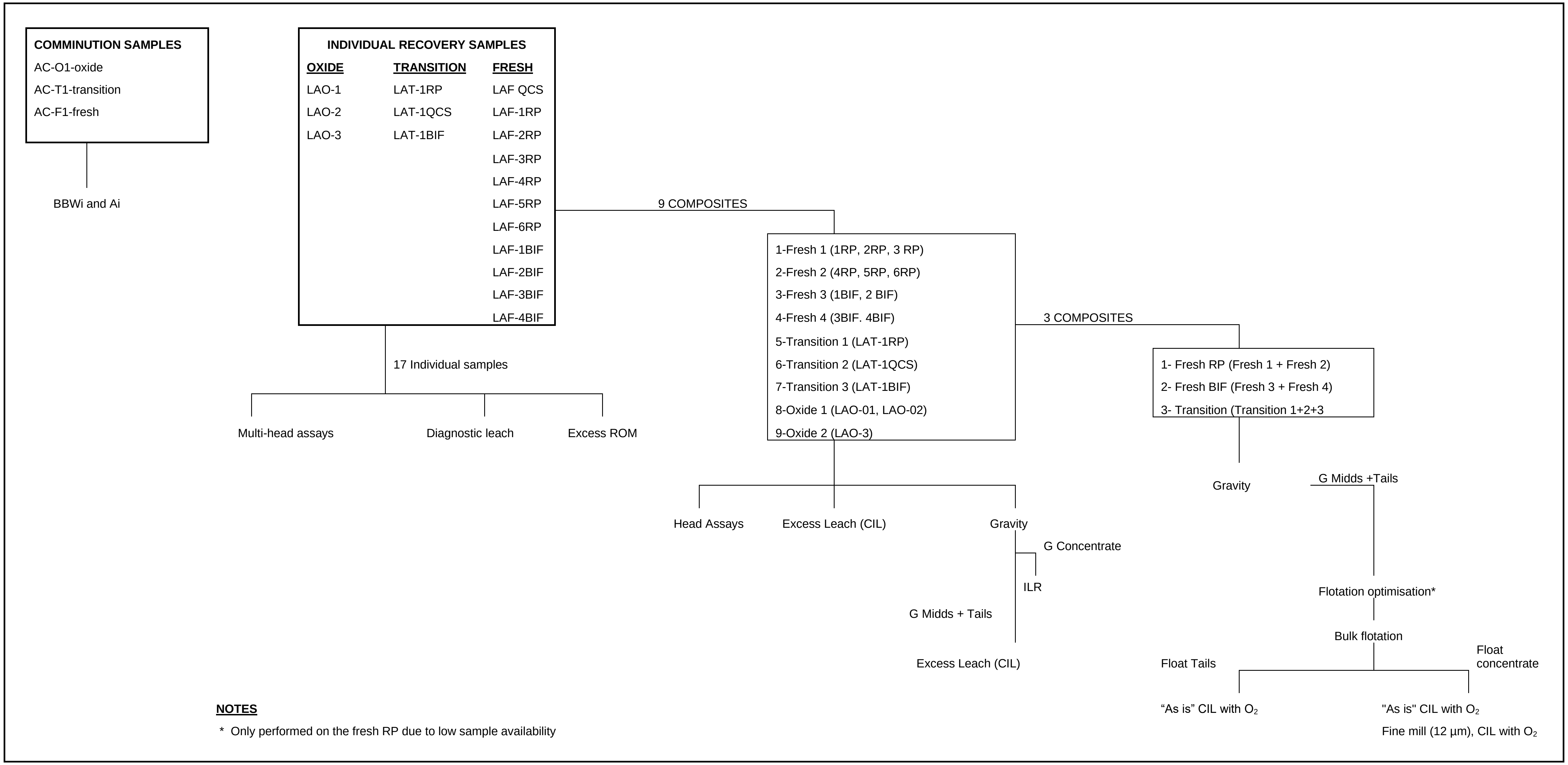


Figure 13.1: PEA Test Work Flowsheet

13.4.3 PEA Test Work Results

The detailed PEA test work results, including test work procedures, are included in the MSA test work report: "Adumbi metallurgical test work in support of a Preliminary Economic Assessment", Report No. REP 21-124 REV 0, December 2021.

13.4.3.1 Comminution Test Work

BBWi and Ai tests were performed on the oxide, transition and fresh samples. Table 13.8 shows a summary of the comminution results.

Table 13.8: Summary of Comminution Results

Sample ID	Ore Type	Ai (g)	BBWi (kWh/t)
ACF1	Fresh	0.3409	14.6
ACT1	Transition	0.2519	13.6
ACT1	Oxide	0.1878	12.7

The results indicate the following:

- As expected, ore hardness (BBWi) increased from the oxide ore to the transition ore to the fresh ore.
- The oxide and transition ores are medium hard for ball milling with BBWi values of 12.7 kWh/t and 13.6 kWh/t, respectively. The fresh ore is hard for ball milling with a BBWi value of 14.6 kWh/t. The historical BBWi comminution test work also showed that the oxide ore is medium hard. The historical BBWi tests on the sulphide ore showed that the ore is medium hard while the current results indicate that the sulphide ore is hard. The difference is an indication that there is variability in the sulphide ore hardness, and this phenomenon should be investigated further by variability comminution test work.
- The abrasion index values for the oxide, transition and fresh ore were 0.1878 g, 0.2519 g and 0.3409 g, respectively, indicating that the oxide ore is low abrasive and the transition and fresh ores are medium abrasive. Therefore, liner and media wear is not expected to be significant.

13.4.3.2 Multi-Head Assays

Multi-head assays were performed on the individual samples (see Table 13.9).

Gold assays were performed at two separate laboratories, namely Superlabs and SGS Johannesburg.

Table 13.9: Multi-Head Assays

Sample ID	Au 1	Au 2	Average Au (Superlabs)	Au 1	Au 2	Au 3	Average Au (SGS)	Adumbi Assays	SG	Ag	Hg	S^{2-}	As
	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t Au		ppm	ppm	%	ppm
LA-F-1RP	4.48	4.6	4.54	4.02	3.75	4.18	3.98	3.67	3.09	2	0.03	10.5	13,800
LA-F-1BIF	3.52	3.4	3.46	2.22	2.43	2.35	4.46	2.72	3.17	< 1	0.04	5.44	7,680
LA-F-1QCS	2.52	2.72	2.62	1.82	1.88	1.98	2.16	**2.60**	3.14	< 1	0.94	3.2	5,150
LA-F-2RP	4.56	4.36	4.46	3.49	3.64	3.67	3.60	**5.06**	3.11	3.5	0.13	7.69	18,800
LA-F-3BIF	2.32	2.36	2.34	2.47	2.49	2.55	2.50	**2.65**	3.14	< 1	0.04	3.19	4,840
LA-F-4RP	5.96	6.20	6.08	3.91	4.17	3.93	4.00	**5.17**	3.14	2.8	0.05	9.29	13,900
LA-F-3RP	3.84	4.02	3.93	3.71	3.49	3.56	3.59	4.57	3.16	2.4	0.14	8.16	24,800
LA-F-2BIF	2.2	2.04	2.12	2.27	2.19	2.31	2.26	2.79	3.18	2.1	0.04	4.46	11,800
LA-F-4BIF	2.44	2.66	2.55	2.93	2.82	2.93	2.89	3.02	3.17	1.2	0.18	4.31	5,190
LA-F-5RP	3.64	3.48	3.56	4	3.63	4.81	4.15	4.80	2.94	2.1	0.02	5.56	8,720
LA-F-6RP	2.52	2.48	2.50	2.71	2.52	2.5	2.58	3.48	3.08	< 1	0.02	5.7	6,910
LA-O-1	-	-	-	1.93	1.77	1.76	1.82	-	2.83	1.20	0.48	0.03	2,660
LA-O-2	-	-	-	2.06	1.89	2	1.98	-	2.94	1.10	0.11	0.03	2,380
LA-O-3	-	-	-	0.2	0.24	0.19	0.21	-	2.77	< 0.3	0.07	0.02	1,360
LA-T-1RP	3.08	3.00	3.04	2.98	3.38	3.03	3.13	4.14	3.16	1.10	0.42	4.4	3,210
LA-T-1QCS			-	2.19	2.09	2.21	2.16	2.33	3.13	0.40	0.05	2.48	4,590
LA-T-1BIF			-	4.38	4.62	4.37	4.46	4.99	2.91	1.80	0.5	2.66	5,530
Fresh ore	Maximum	-	6.08	-	-	-	4.46	5.17	3.18	3.5	0.94	10.50	24,800
	Minimum	-	2.12	-	-	-	2.16	2.60	2.94	< 1	0.02	3.19	4,840
	Average	-	3.47	-	-	-	3.29	3.68	3.12	2.3	0.15	6.14	11,054
Transition ore	Maximum	-	-	-	-	-	4.46	4.99	3.16	1.8	0.5	4.40	5,530
	Minimum	-	-	-	-	-	2.16	2.33	2.91	0.4	0.05	2.48	3,210
	Average	-	-	-	-	-	3.25	3.82	3.07	1.1	0.32	3.18	4,443
Oxide ore	Maximum	-	-	-	-	-	1.98	-	2.94	1.2	0.48	0.03	2,660
	Minimum	-	-	-	-	-	0.21	-	2.77	1.1	0.07	0.02	1,360
	Average	-	-	-	-	-	1.34	-	2.85	1.15	0.22	0.03	2,133

The gold assay results from the two laboratories (Superlabs and SGS Johannesburg) indicate the following:

- The gold assays (SGS) compared well to the assays provided by the geologists except for oxide sample LA-03, which showed a low head grade of 0.21 g/t Au compared to 2.50 g/t Au supplied by the geologists. The difference in head grade is likely due to coarse gold within the ore. Therefore, oxide sample LA-03 was not included in the oxide composite make-up as it would have skewed the oxide composite gold recoveries. This sample was treated individually and would present a variability scenario of a low head grade sample.
- Considering the SGS triplicate gold analysis, the oxide, transition and fresh ores showed average gold head grades of 1.34 g/t Au, 3.25 g/t Au and 3.29 g/t Au, respectively.
- The oxide, transition and fresh ores showed average silver head grades of 1.15 g/t Ag,1.10 g/t Ag and 2.30 g/t Ag, respectively. Relative to gold, the silver levels are significant and would need to be monitored. Silver may have a negative influence on leach kinetics, carbon loadings, elution and electrowinning efficiencies.
- The average SG of the oxide, transition and fresh ores was 2.85, 3.07 and 3.12, respectively.
- The oxide, transition and fresh ores showed average sulphide sulphur head grades of 0.03 %, 3.18 % and 6.14 % S^{2-}, respectively.
- The oxide, transition and fresh ores showed maximum organic carbon head grades of 0.05 %, 0.05 % and 0.22 %, respectively. Therefore, the Adumbi ore is expected to be preg-robbing.
- The oxide, transition and fresh ores showed average mercury assays of 0.22 ppm, 0.32 ppm and 0.15 ppm Hg respectively. The mercury levels are low, and no special mercury handling equipment will be required in the gold room.
- Arsenic head assays on the fresh ore ranged from 0.48 % to 2.48 %, with an average of 1.11 % As. The transition ore showed arsenic values ranging from 0.32 % to 0.55 % with an average of 0.44 %. The oxide ore showed arsenic values ranging from 0.14 % to 0.27 %, with an average of 0.21 %. The arsenic values in the Adumbi ore are relatively high. As the project progresses, it is recommended that arsenic balance tests be performed to determine if arsenic reports in the final tailings solution. If arsenic does report in the final tailings solution at > 0.1 ppm As, then an arsenic precipitation plant would need to be included in the plant design to reduce arsenic levels to environmentally acceptable levels.

13.4.3.3 Diagnostic Leach Test Work

Table 13.10, Table 13.11, and Table 13.12 show the oxide, transition and fresh ore diagnostic leach results, respectively.

Table 13.10: Diagnostic Leach Results – Oxide Ore

Stage	Oxide Ore					
	LAO-1		LAO-2		LAO-3	
	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%
Assayed Head	1.72		1.90		0.52	
Calculated Head	1.72		1.90		0.52	
Cyanide Soluble	1.45	84.32	1.61	84.77	0.22	42.57
CIL Recovery	1.60	93.04	1.62	85.03	0.23	43.43
HCl Leach	0.03	1.89	0.13	6.78	0.22	41.44
HNO_3 Leach	0.01	0.45	0.04	1.92	0.03	5.56
Roast	0.06	3.48	0.05	2.71	0.03	5.83
Silica/Gangue	0.02	1.14	0.07	3.57	0.02	3.75

The results indicate the following:

- Samples LAO-1 and LAO-2 showed 93.04 % and 85.03 % gold amenable to CIL. Sample LAO-1 showed significant preg-robbing of 8.71 % while sample LAO-2 had low preg-robbing of 0.26 % Au.
- Sample LAO-3, which had a low head grade, showed a low CIL gold extraction of 43.43 %. Strangely, this sample showed significant gold locked in sulphides, which does not make sense. Maybe this ore is not a true oxide.

Table 13.11: Diagnostic Leach Results – Transition Ore

Stage	Transition Ore					
	LA-T-1RP		LA-T-1BIF		LAT-1QCS	
	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%
Assayed Head	3.04		5.30		1.90	
Calculated Head	3.04		5.30		1.90	
Cyanide Soluble	2.96	97.39	4.76	89.85	1.18	62.17
CIL Recovery	2.96	97.44	4.90	92.48	1.38	72.66
HCl Leach	0.02	0.66	0.03	0.53	0.02	1.24
HNO_3 Leach	0.00	0.04	0.09	1.64	0.22	11.65
Roast	0.01	0.25	0.15	2.79	0.10	5.17
Silica/Gangue	0.05	1.61	0.14	2.56	0.18	9.27

The results indicate the following:

- The transition RP and BIF are not refractory and gave a CIL gold extraction of 97.44 % and 92.48 %, respectively, with corresponding preg-robbing of 0.05 % and 2.63 %.
- The transition QCS showed a lower CIL gold extraction of 72.66 %. This sample is highly preg-robbing, with 10.45 % gold lost to preg-robbers. Significant gold (12.90 %) in this sample is associated with sulphide minerals.

Table 13.12: Diagnostic Leach Results – Fresh Ore

Stage	Fresh Ore																					
	LA-F-1BIF		LA-F-1QCS		LA-F-1RP		LA-F-2RP		LA-F-3BIF		LA-F-4RP		LA-F-3RP		LA-F-4BIF		LA-F-2BIF		LA-F-5RP		LA-F-6RP	
	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%	Au (g/t)	%
Assayed Head	3.46		2.62		4.54		4.46		2.34		6.08		3.93		2.52		2.12		3.56		2.50	
Calculated Head	3.46		2.62		4.54	100.00	4.46		2.34		6.08		3.93		2.52		2.12		3.56		2.50	
Cyanide Soluble	3.24	93.68	1.77	67.44	2.29	50.51	2.32	52.07	2.26	96.60	4.73	77.85	1.89	48.21	2.44	96.84	1.07	50.40	3.48	97.76	2.42	96.81
CIL Recovery	3.38	97.70	2.51	95.82	3.55	78.28	2.83	63.44	2.26	96.68	4.79	78.77	2.58	65.55	2.44	96.84	1.44	68.11	3.48	97.76	2.42	96.81
HCl Leach	0.02	0.50	0.03	1.18	0.30	6.54	0.37	8.27	0.02	0.95	0.36	5.93	0.41	10.33	0.00	0.04	0.06	2.86	0.02	0.61	0.02	0.65
HNO_3 Leach	0.03	0.92	0.04	1.35	0.60	13.32	0.64	14.24	0.01	0.56	0.03	0.55	0.28	7.02	0.00	0.00	0.07	3.33	0.00	0.09	0.01	0.34
Roast	0.00	0.05	0.00	0.00	0.01	0.14	0.55	12.33	0.00	0.15	0.43	7.11	0.03	0.64	0.00	0.08	0.01	0.24	0.01	0.17	0.00	0.00
Silica/ Gangue	0.03	0.84	0.04	1.66	0.08	1.72	0.08	1.72	0.04	1.67	0.46	7.64	0.65	16.46	0.08	3.07	0.54	25.46	0.05	1.38	0.06	2.23

The results indicate the following:

- CIL gold extraction varied from 63.44 % (RP) to 97.70 % (BIF), showing that sulphide ore contains both refractory and non-refractory gold.
- The sample with the lowest CIL gold extraction has 28.37 % gold locked in sulphides meaning that flotation can be an alternative process route for this sample.
- Preg-robbing ranged from 0 % to 28.37 %, with an average of 9.78 %, implying that gold cyanidation needs to be via CIL.

13.4.3.4 Excess Leach Test Work (Whole Ore leach)

Excess leach tests (CIL) were performed on nine composites made from the individual samples. Table 13.13 shows a summary of the excess leach results.

Table 13.13: Excess Leach Results

Sample ID	Head		Reagent Consumption		Dissolution (Calculated)	
	Assayed	Calculated	CaO	NaCN	Solution + Carbon	Solid
	Au (g/t)	Au (g/t)	kg/t	kg/t	Au %	Au %
Fresh 1	4.06	4.12	0.55	3.15	58.71	58.71
Fresh 2	5.86	4.96	0.45	3.16	82.67	82.67
Fresh 3	2.58	2.69	0.49	3.20	63.57	63.57
Fresh 4	2.26	4.19	0.34	3.23	80.41	80.41
Transition 1	3.04	5.02	1.27	1.96	65.31	65.31
Transition 2	2.46	2.58	0.81	2.87	94.57	94.57
Transition 3	2.28	2.35	1.42	2.65	85.09	85.09
Oxide 1	1.00	1.07	1.86	3.22	55.04	55.04
Oxide 2	1.06	1.06	1.25	2.22	71.80	71.80

The results indicate the following:

- Gold dissolution on the fresh samples ranged from 58.71 % to 80.41 %. These values are lower than those achieved in the diagnostic leach tests. The reason could be that the diagnostic tests were performed at a higher cyanide addition of 10 kg/t and a lower percentage of solids of 30 % w/w.
- Gold dissolution (CIL) on the transition ore ranged from 65.31 % to 94.57 % compared to the 63.44 % to 97.70 % achieved in the diagnostic leach tests.

13.4.3.5 Gravity and Cyanidation Test Work

Gravity recovery, followed by ILR on the gravity concentrate and excess leach (CIL) on the gravity middlings and tailings, was performed on each of the nine samples. Table 13.14 shows a summary of the test work results.

Table 13.14: Gravity and Cyanidation Test Work Results

Sample ID	Head Assay	Gravity Recovery	ILR	CIL	Reagent Consumption (kg/t)		Overall Au Recovery
	Au g/t	% Au	% Au	% Au	Cyanide	Lime	%
Fresh 1 (1RP, 2RP, 3 RP)	3.48	28.00	76.00	65.44	1.45	2.96	68.46
Fresh 2 (4RP, 5RP, 6RP)	3.84	31.18	90.84	85.24	1.45	1.03	86.99
Fresh 3 (1BIF, 2 BIF)	2.74	26.29	81.38	59.08	1.00	5.63	64.94
Fresh 4 (3BIF. 4BIF)	2.36	37.04	89.00	67.43	0.83	2.60	75.42
Transition 1 (LAT-1RP)	3.13	36.61	94.11	95.07	1.20	2.96	94.72
Transition 2 (LAT-1QCS)	2.16	35.66	93.13	75.81	0.59	5.98	81.98
Transition 3 (LAT-1BIF)	4.46	22.46	96.59	93.80	1.78	5.62	94.43
Oxide 1 (LAO-01, LAO-02)	1.73	31.27	97.09	92.04	1.10	3.01	93.62
Oxide 2 (LAO-3)	0.21	42.38	95.42	86.07	0.64	3.16	90.03

The results indicate the following:

- All the ore types are amenable to gravity concentration. The oxide, transition and fresh ore showed average gravity recoveries of 36.82 %, 31.58 % and 31.00 %, respectively. It should be noted that these are laboratory-scale gravity recoveries. For plant recoveries, extended gravity recoverable gold (EGRG) test work needs to be performed, followed by modelling to obtain the predicted plant gravity gold recovery.
- Average overall gold recoveries via gravity and cyanidation were 91.82 %, 90.38 % and 73.95 % for the oxide, transition and fresh ores, respectively. The oxide and transition ores responded well to gravity followed by cyanidation. The fresh ore showed lower overall gold recovery via gravity followed by cyanidation; this is as a result of the low gold dissolution on the gravity middlings and tails and on the gravity concentrate.

13.4.3.6 Gravity – Flotation and Cyanidation Test Work

Since the fresh ore was giving low or poor gold recoveries via cyanidation, flotation was investigated as an alternative recovery method to improve gold recoveries.

Initial scouting flotation tests were performed on the fresh and transition samples using flotation conditions from previous flotation optimisation tests. Table 13.15 shows a summary of the scouting flotation results.

Table 13.15: Summary of Scouting Flotation Test Work Results

Sample ID	Mass Pull (%)	Rougher Concentrate	
		Concentrate Grade (g/t Au)	Au Recovery (%)
Fresh 1	19.84	11.66	69.27
Fresh 2	14.16	23.97	80.79
Fresh 3	15.68	11.44	29.28

Sample ID	Mass Pull (%)	Rougher Concentrate	
		Concentrate Grade (g/t Au)	Au Recovery (%)
Fresh 4	13.53	15.71	80.90
Transition 1	18.32	18.69	90.11
Transition 2	18.83	9.72	80.10
Transition 3	12.60	4.60	29.28

The results indicated poor flotation recoveries for both the fresh and transition samples. Therefore, additional flotation optimisation tests were performed to try to improve flotation recoveries. There was only enough sample mass available to do the flotation optimisation on the fresh RP sample. Primary grind and reagent optimisation tests were performed. The flotation optimisation gave the following optimum conditions for rougher flotation:

- Primary Grind: 80 % passing 75 µm
- Residence time: 15 min
- PAX staged addition: 70 g/t
- pH: Natural

The optimisation yielded a gold recovery of 97.48 % at a grade of 14.98 g/t Au and mass pull of 25.57 %.

13.4.3.7 Bulk Flotation Test Work

Bulk flotation was performed on the following samples to produce a flotation concentrate and flotation tailings for the cyanidation test work:

- Fresh RP
- Fresh BIF
- Transition composite

Table 13.16 shows a summary of the bulk flotation test work results.

Table 13.16: Summary of Bulk Flotation Test Work Results

Sample ID	Mass Pull (%)	Rougher Concentrate			
		Au Grade (g/t Au)	Au Recovery (% Au)	S Grade (%)	S Recovery (% S)
Fresh RP G.T	26.59	9.57	93.03	25.07	93.03
Fresh BIF G.T	21.76	8.30	85.13	17.90	85.13
Transition G.T	16.83	11.82	95.52	15.80	95.52

The results indicate gold recoveries of 93.03 %, 85.13 % and 95.52 % for the fresh RP, fresh BIF and transition composite, respectively.

13.4.3.8 Flotation Concentrate Leach Test Work

The flotation concentrates were leached under excess leach conditions with oxygen sparging. The leach tests were performed on the "as-is" float concentrate and on a float concentrate fine milled to 80 % passing 12 µm. Table 13.17 shows a summary of the leach results.

Table 13.17: Summary of Leach Tests on Flotation Concentrates

Grind	Sample ID	Au Dissolution (%)
"As is" (80 % passing 75 µm)	Fresh RP	61.42
	Fresh BIF	65.81
	Transition	90.32
80 % passing 12 µm	Fresh RP	75.53
	Fresh BIF	66.74
	Transition	91.27

The results indicate that fine milling to 80 % passing 12 µm improved gold extraction, but the increase was not significant enough to give +90 % gold extraction on the fresh ore flotation concentrate.

13.4.3.9 Flotation Tailings Leach Test Work

Excess leach rate tests were performed on the "as-is" flotation tails without fine milling. Table 13.18 shows a summary of the leach results.

Table 13.18: Summary of Leach Rate Tests on Flotation Tailings

Residence Time (h)	Fresh RP (%)	Fresh BIF (%)	Transition (%)
2	47.13	60.06	81.19
4	61.04	64.39	81.19
8	65.72	66.60	83.59
16	71.55	66.36	86.75
24	73.19	66.36	86.89

13.4.3.10 Overall Gold Recoveries

Table 13.19 shows a summary of the gold recoveries achieved through the various methods.

Table 13.19: Summary of Gold Recovery Values

Average	Diagnostic (%)	Excess Leach (%)	Gravity + Leach (%)	Gravity-Flotation + Leach (%)	Reagent Consumption (kg/t)	
					Cyanide*	Lime
Fresh RP	80.10	70.69	77.73	77.82	1.45	2.00
Fresh BIF	89.83	69.77	70.18	72.57	0.91	4.11
Transition	87.53	81.66	90.38	91.77	1.19	4.85
Oxide	90.76	70.07	91.82	-	0.87	3.09
* Cyanide consumption values are only indicative. No cyanide optimisation tests were done since this was a PEA study.						

The results indicate the following:

- For the oxide and transition ores, the best process route is gravity followed by cyanidation.
- For the fresh ore, the gravity-flotation-leach process route gave lower than expected overall gold recoveries. This was because lower (< 90 %) gold extraction was achieved on the flotation concentrates. Gold extractions > 90 % are required on the flotation concentrate to achieve overall gold recoveries above 90 %. Due to a limited sample mass, there were not enough flotation concentrate samples to perform further test work on the flotation concentrate. Further tests can be done in the next phase of the study when more samples are available from further drilling. To progress with the PEA study, gold recovery values from the diagnostic test work were used for the mine pit design and optimisation.

13.4.3.11 Proposed Design Values

From the test work results, preliminary design values were selected. At a PEA study level, these design values are preliminary. Final values for design will be obtained by additional detailed test work that can be done in the feasibility stages of the project. Table 13.20 shows the proposed design values.

Table 13.20: Recommended Design Values

Parameter		Oxide	Transition	Fresh	
				Fresh RP	Fresh BIF
Ore Characterisation	SG – SENET	2.85	3.07	3.09	3.17
	SG – OMC	2.70	2.80	2.90	
	As (ppm)	2,133	4,443	12,877	7,008
	Ag (g/t)	0.87	1.10	0.07	0.08
	Bulk Density	1.80	–	3.00	
	Au (g/t)	1.34	3.25	–	–
Comminution	BBWi (kWh/t) – SENET	11.58	13.6	14.6	
	BBWi (kWh/t) – OMC	11.8	13.7	14.2	
	BRWi	–	–	–	

<table>
<tr><th colspan="2" rowspan="2">Parameter</th><th rowspan="2">Oxide</th><th rowspan="2">Transition</th><th colspan="2">Fresh</th></tr>
<tr><th>Fresh RP</th><th>Fresh BIF</th></tr>
<tr><td rowspan="3"></td><td>UCS</td><td>–</td><td>–</td><td colspan="2">–</td></tr>
<tr><td>CWi</td><td>–</td><td>–</td><td colspan="2">–</td></tr>
<tr><td>Ai (g)</td><td>0.1899</td><td>0.2519</td><td colspan="2">0.3560</td></tr>
<tr><td rowspan="4">Gold Recovery</td><td>Proposed Process Route</td><td>Gravity + CIL</td><td>Gravity + CIL</td><td>Gravity + CIL</td><td>Gravity + CIL</td></tr>
<tr><td>Gold Recovery (%)</td><td>90.76</td><td>87.53</td><td>80.10</td><td>89.83</td></tr>
<tr><td>Cyanide Consumption (kg/t)</td><td>0.87</td><td>1.19</td><td>1.45</td><td>0.91</td></tr>
<tr><td>Lime Consumption (kg/t)</td><td>3.09</td><td>4.85</td><td>2.00</td><td>4.11</td></tr>
</table>

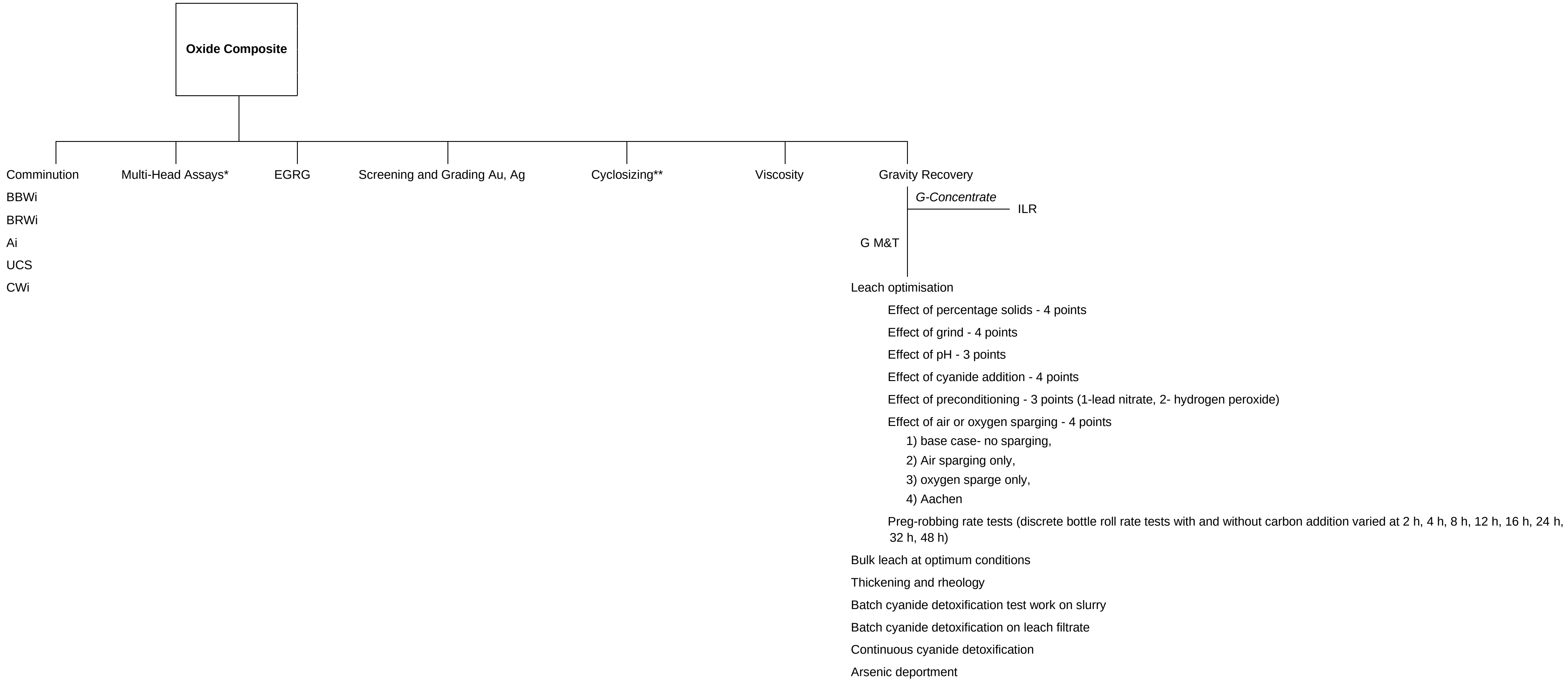
13.5 RECOMMENDATIONS

Flotation test work indicated that gold can be recovered into a rougher flotation concentrate. The concentrate samples that were generated were not sufficient to enable further processing routes such as

- Fine milling followed by leaching with oxygen addition
- Fine milling followed by partial oxidation using high shear reactors and leaching
- Albion process
- Pressure oxidation
- Bio leaching
- Roasting

These recovery processes will be investigated during the next phase of the project to optimise the gold recovery in the transition and fresh ore types.

Comminution and gold recovery test work was proposed for the pre-feasibility study (PFS). Figure 13.2 and Figure 13.3 show an overview of the proposed oxide, transition and fresh PFS test work.



NOTES

* Au (×3), Ag, SG, As, Hg, Sb, Sulphur package, carbon package, Full ICP

** Based on the outcome of the screening and grading results. Cyclosizing test work to be performed if screening and grading results indicate that there is significant gold in the fines.

Figure 13.2: Oxide Ore PFS Test Work Flowsheet

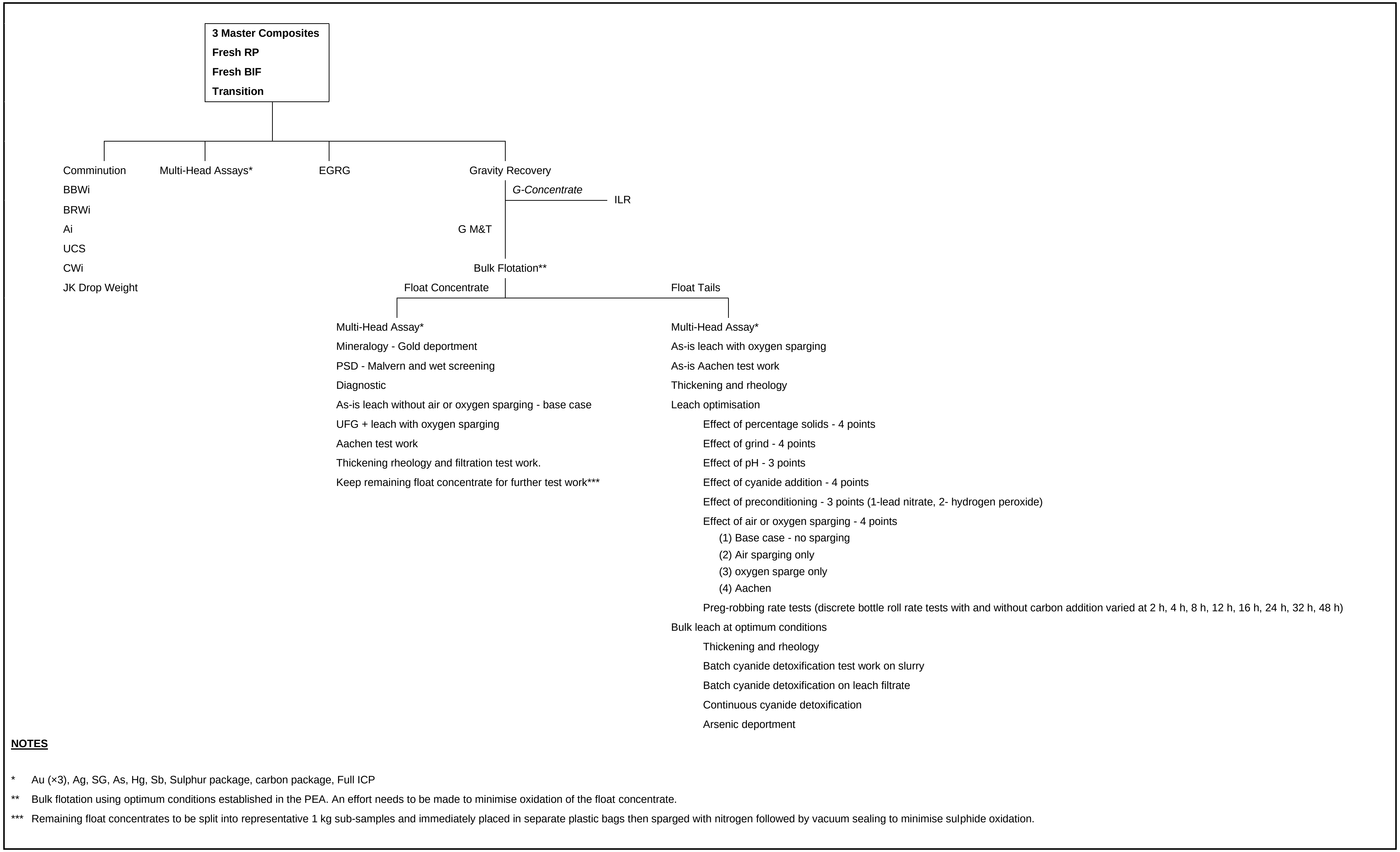


Figure 13.3: Transition and Fresh Ore PFS Test Work Flowsheet

Table 13.21 shows a summary of the quotations received for the PFS test work.

Table 13.21: Quotations for PFS Test Work

Ore Type	Test Work	Costs		Sample Requirements	
		Laboratory Currency	US Dollars	Mass (kg)	Type
Oxide	Comminution and gold recovery	ZAR688,752.00	US$44,406.96	200*	1/2 or full HQ or PQ core
Sulphide	Concentrate generation Leach and Aachen	ZAR3,938,210.00	US$253,914.25	1,500**	1/2 or full HQ or PQ core
	Albion	AU$34,635.15	US$24,917.37	10***	Flotation concentrate
	Biological oxidation (BIOX)	ZAR656,677.90	US$42,339.00	10***	Flotation concentrate
	Pressure oxidation (POX)	US$43,000.00	US$43,000.00	50***	Flotation concentrate
	Roast	US$78,700.00	US$78,700.00		
TOTAL			**US$487,277.59**		

* 1 × 200 kg of a master oxide composite required
** Amount per composite. Three separate composites of the following are required: Fresh RP, Fresh BIF and Transition
*** Float concentrate produced from the 1,500 kg

NOTE: The following exchange rates were used: ZAR15.51/US$1 and AU$1.39/US$1.

14 MINERAL RESOURCE ESTIMATES

14.1 APPROACH

The Adumbi three-dimensional updated model was constructed by Minecon in collaboration with on-site geologists using cross-sectional and horizontal flitch plans of the geology and mineralisation to assist in constraining the 3D geological model. The mineralisation model was constrained within a wireframe at a 0.5 g/t Au cut-off grade. Grade interpolation was undertaken using the following:

- 2 m sample composites capped at 18 g/t Au to improve the reliability of the block grade estimates. Capping affected approximately 1 % of the samples.
- Ordinary Kriging to interpolate grades into the block model.
- Relative densities of 2.45 for oxide, 2.82 for transition and 3.05 for fresh rock applied to the block model for tonnage estimation.

After grade interpolation, Minecon used visual inspection in sections and plan views, together with other validation methods, to ensure that the resultant model reflected the drilling database used.

To constrain the depth extent of the geological model and any mineral resources, an open pit was constructed for the Adumbi deposit based on the following pit optimisation parameters:

- A gold price of US$1,600/oz
- A block size of 16 m × 16 m × 8 m
- A 32 m minimum mining width and a maximum of 4 m of internal waste was applied
- A mining dilution of 100 % of the tonnes at 95 % of the grade
- An ultimate pit slope angle of 45°
- Metallurgical recoveries of 91 % for oxide, 88 % for transition and 90 % for sulphide (based on diagnostic metallurgical test work as part of the PEA)
- An average reference mining cost of US$3.29/t mined
- An average processing cost of US$14.63/t for oxide, US$16.30/t for transition and US$18.43/t for sulphide
- An average general and administration cost of US$4.20/t
- Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material, constrained by a US$1,600/oz optimised pit shell
- Transport of gold and refining costs equivalent to 4.5 % of the gold price
- No additional studies on depletion by artisanal activity have been undertaken since the RPA 2014 study, and the same total amount of material was used by Minecon

All the blocks with grades above the cut-off grade within the Whittle open-pit shell truncated at the surface by the topography were reported as open-pit mineral inventory. Historical mining based on estimates used in the RPA 2014 NI 43-101 technical report was depleted from the final resource estimates as there have been no further studies undertaken on depletion by artisanal mining since the RPA 2014 NI 43-101 technical report.

The definitions for Mineral Resource categories used in this estimate are consistent with those set out in the CIM 2014 Definition Standards as incorporated in NI 43-101.

14.2 RESOURCE DATABASE

A total of 73 diamond drillholes made up of 46 re-logged, 4 holes drilled in 2017, and the 23 newly drilled holes in the 2020 to 2021 drilling programme were used in the updated mineral resource estimate. These holes totalled 20,806.97 m and provided 12,415 assays, which were used in the Adumbi mineralisation and geological interpretation, and resource model creation. A 24th hole, LADD0026, was also drilled up to 705 m. Hole LADD0026 had very good intercepts when the results were received. Assays for Hole LADD0026 were not used in the resource modelling due to the delay in receiving the results for this hole. However, the geological data captured for this hole, coupled with the team's understanding of the mineralisation controls at Adumbi, assisted in defining the geometry of the orebody and lithological model in the vicinity of the hole. The holes drilled in the 2020 to 2021 programme were drilled with the initial focus in areas within the pit shell where insufficient drilling had been undertaken to outline mineral resources. The upper parts were infilled at closer spacing in order to upgrade portions of the mineral resources into a higher confidence category. Later drilling has been and is being undertaken at depth below the open-pit shell to outline potential underground mineral resources. The ongoing exploration programme intersected significant grades that supported the down-dip/plunge extension of the mineralisation.

Table 14.1 shows some of the significant intercepts from the holes drilled in 2020 to 2021 which have been incorporated into the current model.

Table 14.1: Significant Intercepts from Drillholes Drilled in 2020 to 2021

Borehole	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
LADD001	202.58	223.35	20.77	1.72
LADD001	231.27	237.17	5.9	1.89
LADD001	251.27	258.6	7.33	5.8
LADD001	295.25	298.7	3.45	2.1
LADD001	301.62	321.95	20.33	2.47
LADD001	Including 317.11	321.95	4.84	5.4
LADD003	224.55	235	10.45	3.88
LADD003	253.5	286.8	33.3	3.25
LADD003	Including 253.50	259.2	5.7	7
LADD003	Including 277.73	286.8	9.07	5.11
LADD004	429	457	28	3.26
LADD004	Including 432.00	436.9	4.90	6.96
LADD004	Including 450.62	454.15	3.53	8.3
LADD004	473.8	478.4	4.60	2.07
LADD004	505.85	526.15	20.3	2.83

Borehole	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
LADD004	Including 506.85	513.4	6.55	4.64
LADD004	Including 523.85	526.15	2.30	7.25
LADD006	299.37	302.25	2.88	2.64
LADD006	308	309	1	21.2
LADD006	322.1	337.3	15.2	1.67
LADD006	353.35	357.85	4.5	3.25
LADD007	99.95	107.8	7.85	1.45
LADD007	540.62	596.05	55.43	2.76
LADD007	Including 583.60	596.05	12.45	8.11
LADD007	607.9	611.27	3.37	4.61
LADD008	235.05	278.15	43.1	1.68
LADD008	291.8	298.9	7.1	1.34
LADD008	305.15	305.93	0.78	21.8
LADD008	323.8	338.78	14.98	3.62
LADD008	Including 335.75	338.78	3.09	13.28
LADD009	559.76	564.76	5	3.17
LADD009	581.9	614.05	32.15	6.17
LADD009	Including 599.05	600.51	1.46	94.77
LADD009	629.56	644.92	15.36	3.73
LADD009	Including 632	637.89	5.89	6.56
LADD009	650.5	657.95	7.45	1.48
LADD012	784.35	797.8	13.45	3.63
LADD012	Including 784.35	786.35	2	9.56
LADD012	806.3	810.35	4.05	4.73
LADD013	394.06	401.1	7.04	2.68
LADD013	418.65	438.65	20	4.21
LADD013	Including 419.75	430.75	11	6.91
LADD013	452.3	469.6	17.3	2.48
LADD013	Including 457.35	465.55	8.2	4.71
LADD014	670	681.8	11.8	2.97
LADD014	Including 670	673.53	3.53	6.44
LADD015	24.43	31.5	6.07	1.77

Borehole	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)
LADD016	672.85	680.94	8.09	1.9
LADD016	731.51	757.1	25.59	2.39
LADD016	Including 737.18	743.27	6.09	4.78
LADD016	Including 749.67	752.56	2.89	4.98
LADD016	672.85	680.94	8.09	1.9
LADD017	45.55	62.7	17.15	1.9
LADD017	92.68	118.45	25.77	6.24
LADD017	Including 100.76	110.05	9.29	9.68
LADD017	Including 112.95	118.45	5.5	9.75
LADD018	93.34	113.7	20.36	0.93
LADD018	152.48	178.2	25.72	2.26
LADD019	4.57	11.6	7.03	2.13
LADD021	75.21	88.17	12.96	2.09
LADD021	99.74	106	6.26	1.09
LADD021	144.78	160.51	15.73	5.28
LADD021	Including 144.78	149.78	5	13.7
LADD022	20.5	42	21.5	2.23
LADD022	Including 25.5	34	8.5	4.23
LADD023	227.1	261.73	34.63	3.12
LADD023	Including 231.65	237.4	5.75	7.23
LADD023	Including 248.1	255.25	7.15	5.55
LADD023	270.43	300.25	29.82	1.77
LADD024	216.15	227.65	11.5	3.47
LADD024	Including 224.1	227.65	3.55	7.79
LADD024	235.97	253.75	17.78	3.2
LADD025	258.38	266	7.62	1.16
LADD025	279.5	286.35	6.85	3.44
LADD025	301.1	311.57	10.47	1.74
LADD025	321.6	336.2	14.6	2.11
LADD025	342.65	361.75	19.1	4.11
LADD025	Including 349	357.75	8.75	5.4
NOTES:				

Borehole	From (m)	To (m)	Intercept Width (m)	Grade (g/t Au)

1. Core holes LADD002 and LADD005 were discontinued before intersecting the mineralised zone.
2. Core hole LADD026, which reported 22.03 m grading 5.11 g/t Au (including 14.70 m grading 7.19 g/t Au) and 11.20 m grading 4.93 g/t Au, was not included in the current mineral resource update due to timing.
3. It is estimated that the true widths of the mineralised sections for the drillholes are as follows: LADD001 (82 %), LADD003 (80 %), LADD004 (81 %), LADD006 (95 %), LADD007 (89 %), LADD008 (62 %), LADD009 (82 %), LADD012 (86 %), LADD013 (85 %), LADD014 (78 %), LADD015 (65 %), LADD016 (69 %), LADD017 (71 %), LADD018 (75 %), LADD019 (65 %), LADD021 (73 %), LADD022 (58 %), LADD023 (76 %), LADD024 (77 %) and LADD025 (78 %) of the intercepted widths given in this table.

The database included nine resurveyed adits with a total length of 1,121 m yielding 868 assayed samples. Trench and adit data has been used to support the geological and mineralisation interpretation and in the grade interpolation process. All 73 drillholes intersected the interpreted mineralisation, within which 4,740 samples (38.2 % of all drillhole assays) were selected by the mineralisation wireframe.

Table 14.2 shows some basic statistics of the number of samples in the database that informed the interpretation, and the number of each type of sample that has been captured in the mineralisation wireframe.

Table 14.3 shows a simple count of the distribution of mineral intercepts over the various lithologies at Adumbi.

Table 14.2: Basic Statistics of All Adumbi Samples and Selected Samples within Wireframe Model

Field	No. of Samples	Min. (g/t)	Max. (g/t)	Mean (g/t)	Variance	Logvar	Cov	Description
Au	14,403	0.01	170	0.84	13	4.45	4.3	All Adumbi DD log samples
Au	4,740	0.01	170	2.17	31	2.97	2.56	Selected Adumbi DD samples within ore wireframe
Au	1,731	0.01	90	2.11	18	2.32	2.01	Selected Adumbi DD samples, composited 2 m uncapped
Au	1,731	0.01	18	1.97	7.35	2.29	1.38	Selected Adumbi DD samples, composited 2 m capped at 18 g/t
Au	868	0	12.4	0.29	0.7	3.57	2.87	All Adumbi resurveyed adit log samples
Au	1,010	0	12.8	0.37	0.82	2.04	2.42	All Adumbi trench log samples
Au	264	0.02	12.8	0.98	2.43	1.7	1.58	Selected Adumbi trench samples within ore wireframe
Au	130	0	12.4	0.79	2.39	2.49	1.97	Selected Adumbi resurveyed adit samples within ore wireframe

Table 14.3: Distribution of Mineral Intercepts over Various Lithologies at Adumbi

Description	Lithology									
	BIF	QCS	RP	CBS	CS	QV	IQCS	CBS-AS	ICQS	BCH
Intercept Count	1,851	772	592	412	394	207	181	160	135	42

14.3 BULK DENSITY

Minecon applied the revised relative densities of 2.45 for oxide, 2.82 for transition, and 3.05 for fresh rock to the block model for tonnage estimation.

Additional information regarding the Imbo Project with respect to the determination and application of bulk density is set out in Minecon's technical report dated April 17, 2020, and entitled "Independent National Instrument 43-101 Technical Report on the Imbo Project, Ituri Province, Democratic Republic of the Congo" (available from SEDAR at www.sedar.com).

Table 14.4 shows the relative density measurements used for the Minecon resource estimation.

Table 14.4: Relative Density used for Minecon Resource Estimation

Type	Mineralised		Unmineralised	
	No. of Samples*	Relative Density	No. of Samples*	Relative Density
Oxide	297	2.45	882	2.26
Transition	178	2.82	601	2.54
Sulphide	796	3.05	1953	2.83
* Excludes samples which were not assayed				

14.4 WIREFRAME AND 3D MODELLING

Wireframe models of the geological domains aided in the interpretation and modelling of the mineralisation and grade continuity studies as well as to constrain the block model interpolation. A joint team of Minecon resource evaluation personnel and on-site geologists undertook the interpretation of the various zones, which aided the creation of the Adumbi model. The software used to build the model was Datamine. The mineralisation is structurally controlled. Other models, including the redox surfaces and the digital terrain, were modelled using the triangulation tools available in Datamine.

14.4.1 Geological Wireframe and Modelling

A lithological model was created and used to guide the mineralisation modelling. In creating the lithological model, drillhole logging data for Hole LADD0026 (for which assay data was not available at the time of the modelling process) was used to assist in defining the geometry within the vicinity of the hole. Figure 14.1 is a 3D view of the Adumbi deposit lithological model.

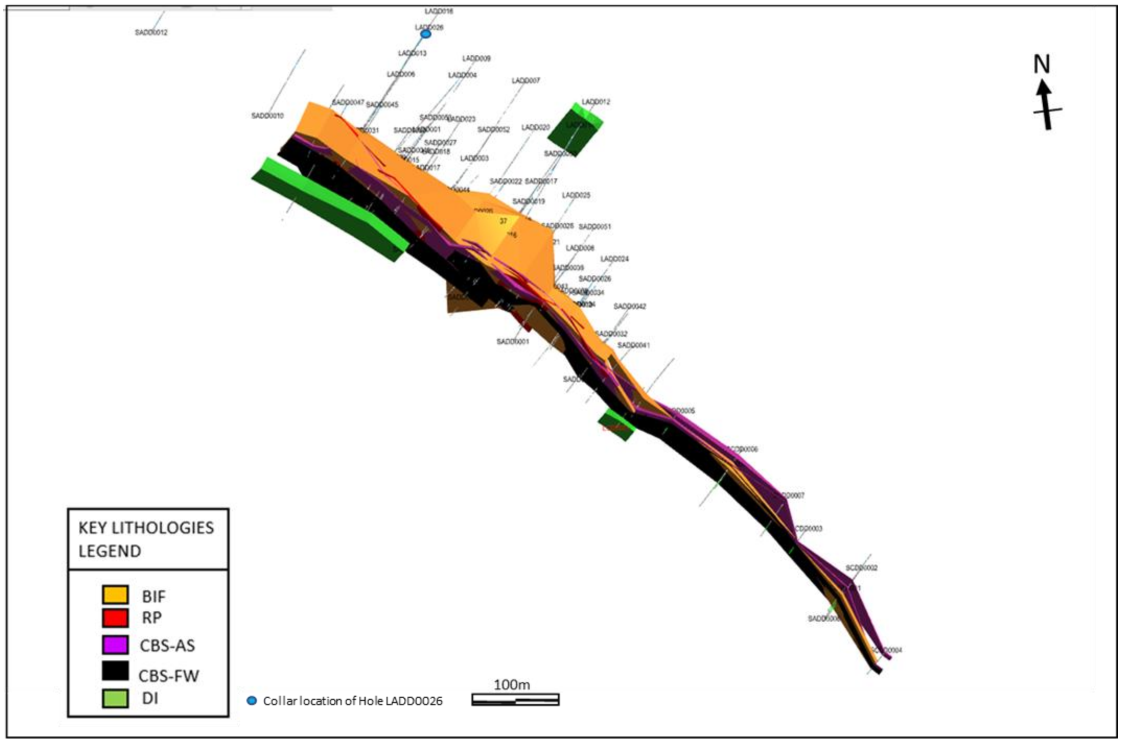


Figure 14.1: Adumbi Deposit – 3D of Lithological Model

It is worth noting that, all the major lithologies contained some level of mineralisation but with variable average grades, some of which were below the mineralisation cut-off grade.

The primary Adumbi database was made up of a combination of drillholes and trenches, and the resurveyed adits were desurveyed in the Datamine software and plotted. Geological and mineralisation interpretation was undertaken in both sections and flitches by Minecon's combined technical team.

Interpretation of the Adumbi mineralisation was developed using a 0.5 g/t Au sample cut-off. Cross sections were generated on a 040 bearing along a mineralisation trending 130°. Section lines were on drill fences spaced between 60 m and 95 m, with an average spacing of 75 m. The interpretations were digitised in Datamine software, and strings were snapped to drillholes. Where necessary, a simplification of the mineralised outlines was undertaken using assay values lower than the cut-off grade of the material to ensure geological continuity, tolerating up to 4 m of internal waste. Three main mineralised zones (Zones 1, 2 and 3) were observed in the Adumbi central area (counting the zones from the footwall). Whilst digitising the ore perimeter strings, Zone 2 was split into two zones named 2U and 2L, thus making a total of four zones. This split was necessary to avoid the inclusion of wider than 4 m internal low-grade bands. Zones 1 and 2 are separated by the carbonaceous marker, which is essentially unmineralised. Generally, Zone 1 is within the Lower BIF sequence, Zone 2 is in the lower part of the Upper BIF Sequence, and Zone 3 is a weaker zone in the upper part of the Upper BIF Sequence. Figure 14.2 is a section through Boreholes SADD0005, 0049, 0053, LADD0015, 001, 004 and 009 showing the 2020 to 2021 interpreted ore outlines.

Towards the southeastern end around the Canal prospect, there is another footwall-mineralised zone thus making five main zones. Figure 14.3 is a flitch at RL560 showing the ore outline interpretation.

The trench and adit information was used to assist with the up-dip continuity of the interpretation where drillhole information was lacking but trench or adit data indicated the continuity of the mineralisation. Down-dip extrapolations beyond the limits of drilling were done to ensure consistency in shape and orientation with due consideration to available geological knowledge. In such instances, up to 100 m extensions were done on some sections, and to ensure continuity along strike extrapolations were 40 m. All the digitised strings were linked to create the 3D mineralised wireframe. The strike length of the mineralised wireframe is 2.3 km. Figure 14.4 is a 3D view of the Adumbi mineralisation wireframe.

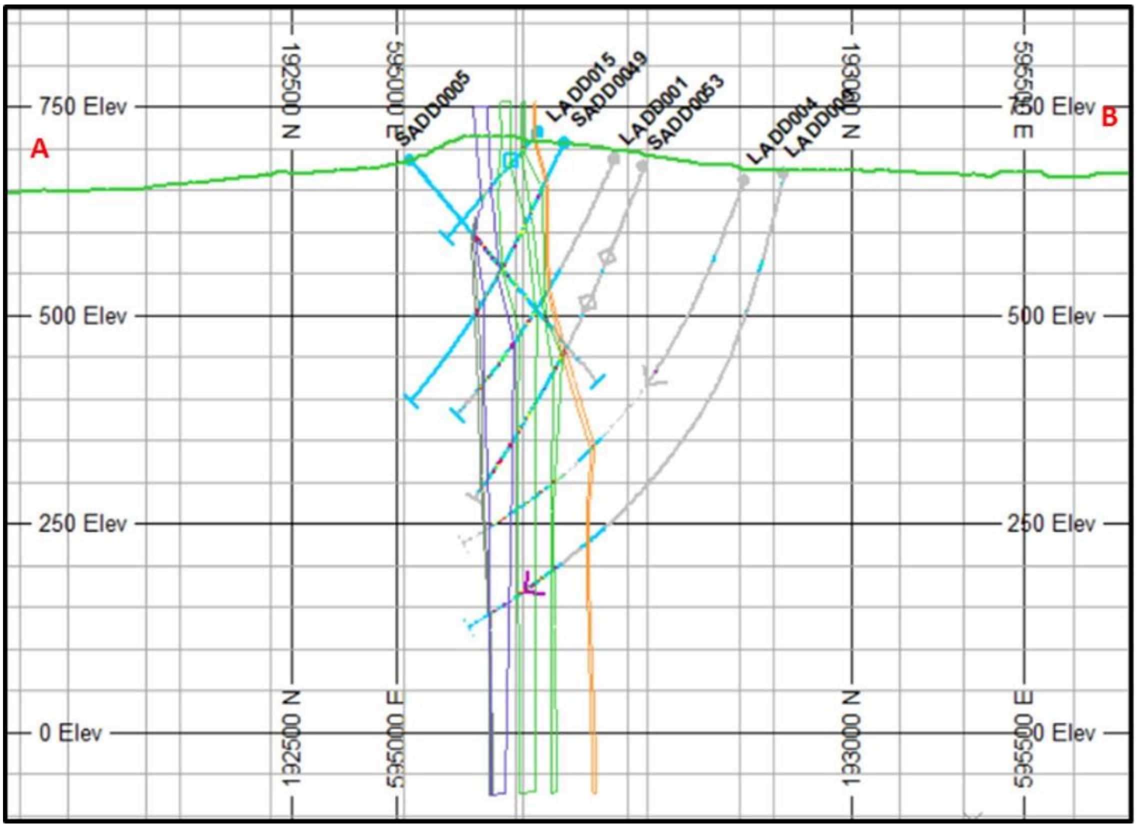


Figure 14.2: Sections through SADD0005, 0049, 0053, LADD0015, 001, 004 and 009 showing 2020-21 Interpreted Mineralised Outlines

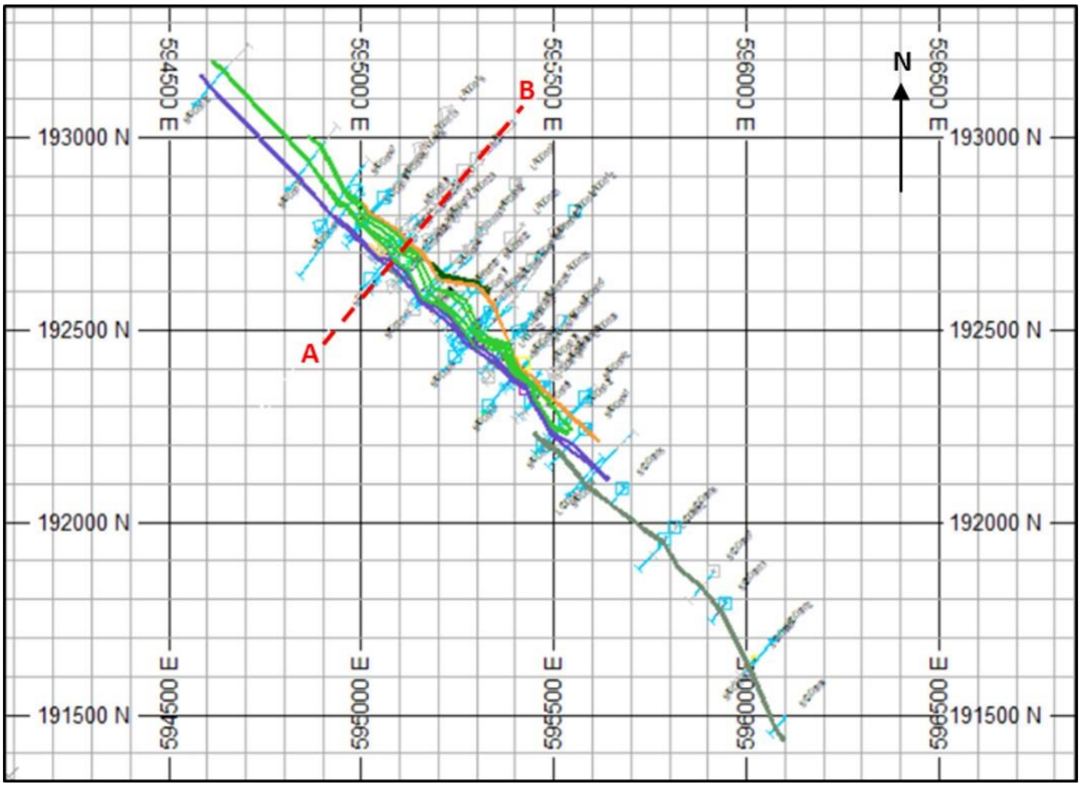


Figure 14.3: Flitch at RL560 showing Interpreted 2021 Ore Outline

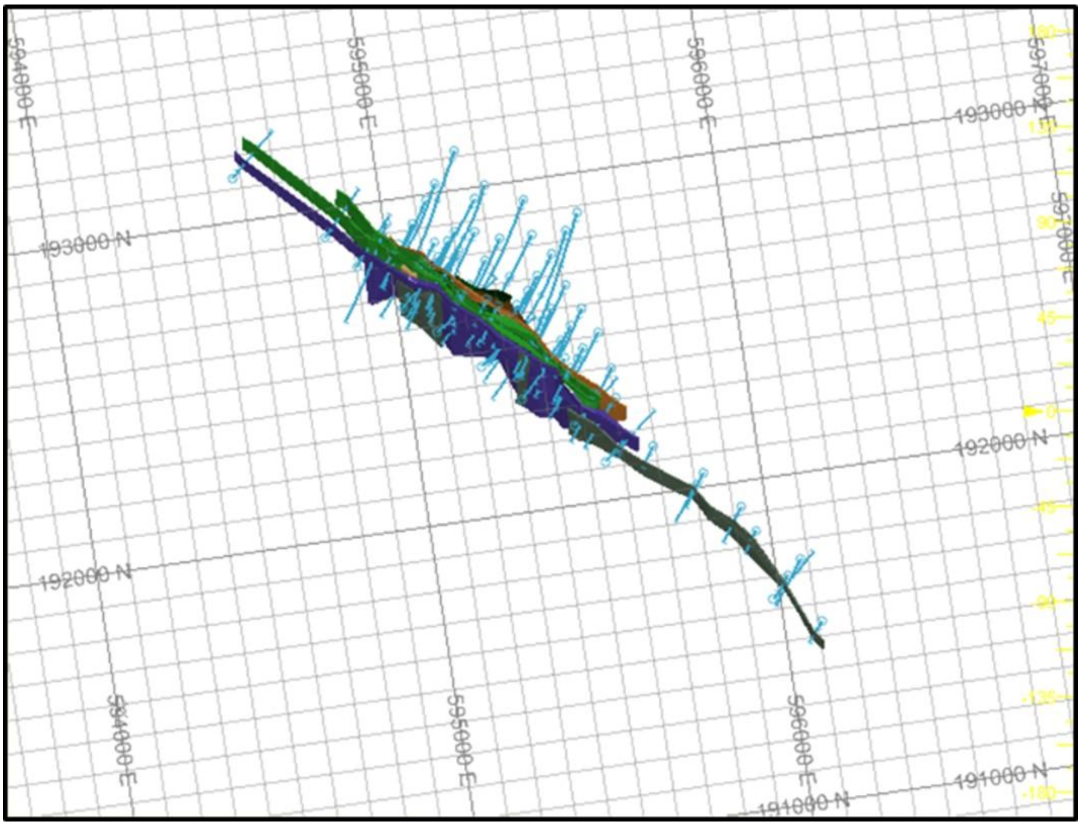


Figure 14.4: 3D View of Adumbi Mineralisation Wireframe

14.4.2 Digital Terrain Model

At Adumbi, Minecon used 10 m interval contours to generate a DTM in Datamine software.

This DTM model was used for the geological modelling and model depletion for the estimation of resources.

14.4.3 Redox Surfaces and Modelling

The BOCO and TOFR surface models were created from each of the 73 drillholes by digitising them in cross sections and wireframing to create models for each surface. The previous surface models were refined by the new information from the extra 18 holes drilled in the 2020 to 2021 programme. There were minor modifications as a result of the newly drilled holes but no significant impact. The digitised surface interpretation strings from each of the sections were linked to create wireframe surfaces in Datamine.

Figure 14.5 shows a typical section of the location of redox surfaces used by Minecon in the April 2021 model compared with the updated November 2021 redox surfaces.

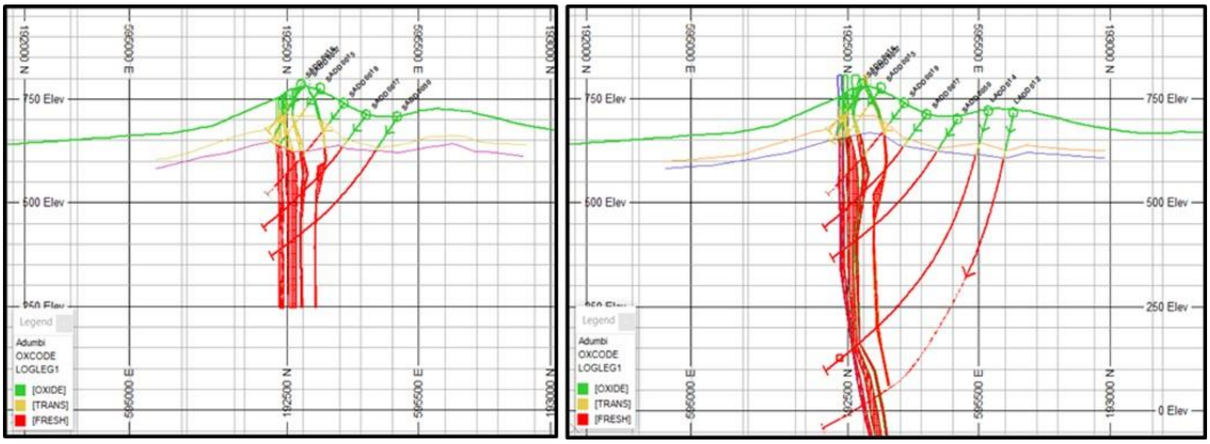


Figure 14.5: Sections through Adumbi Model showing Relative Location of Redox Surfaces used by Minecon in April 2021 vs November 2021

14.5 ASSAY CAPPING

To avoid undue influence of random anomalous high grades on the resource determination, Minecon prepared histograms, probability plots and other graphs and used these to study the various grade distributions of the selected samples. Selected samples within the Adumbi mineralisation wireframe were composited to 2 m. The assay grades appear reasonably independent of sample length (see Figure 14.6) and thus allow for capping based on grades. A suitable capping of 18 g/t Au of the selected samples was applied after studying the distribution from the histogram (see Figure 14.7), frequency log grade graph (see Figure 14.8) and probability plot (see Figure 14.9) to improve the reliability of the block grade estimates.

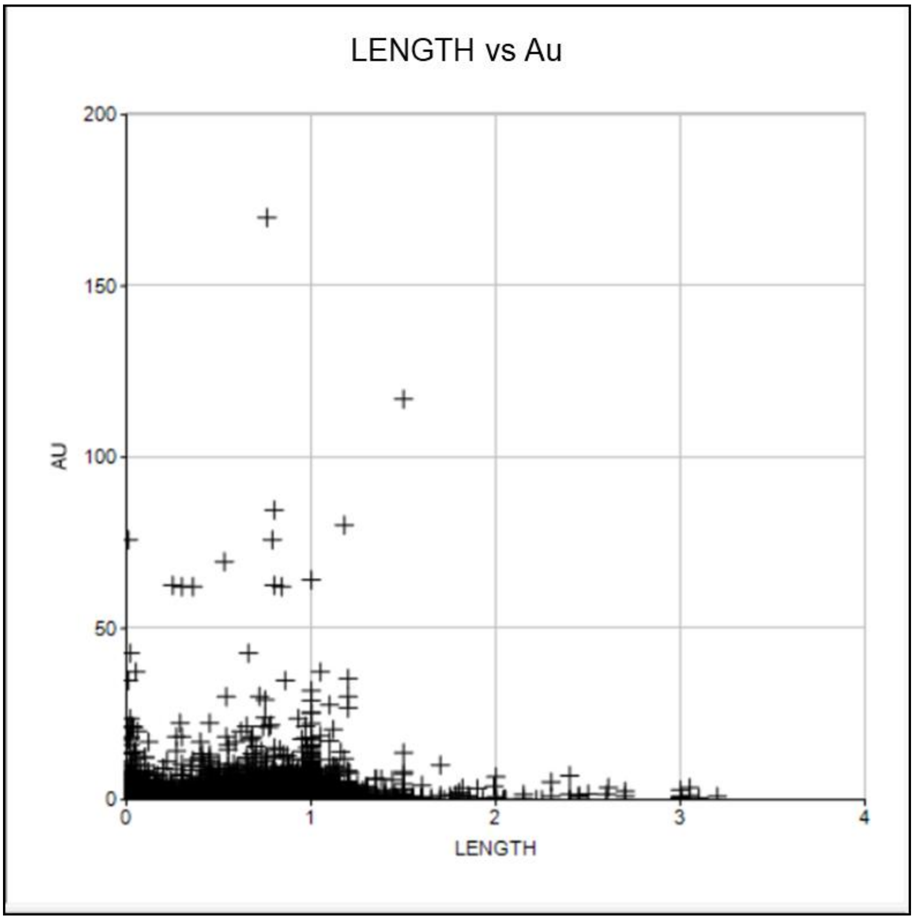


Figure 14.6: Plot of Adumbi Selected Sample Grades vs Sample Lengths

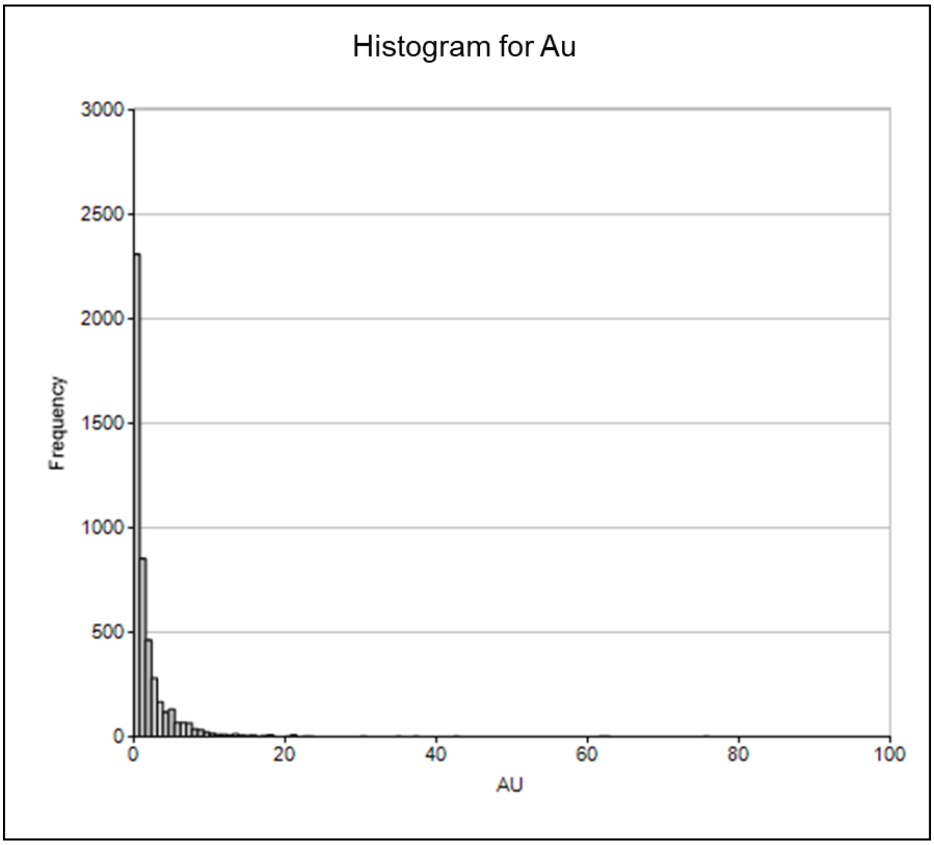


Figure 14.7: Histogram of Selected Au Distribution

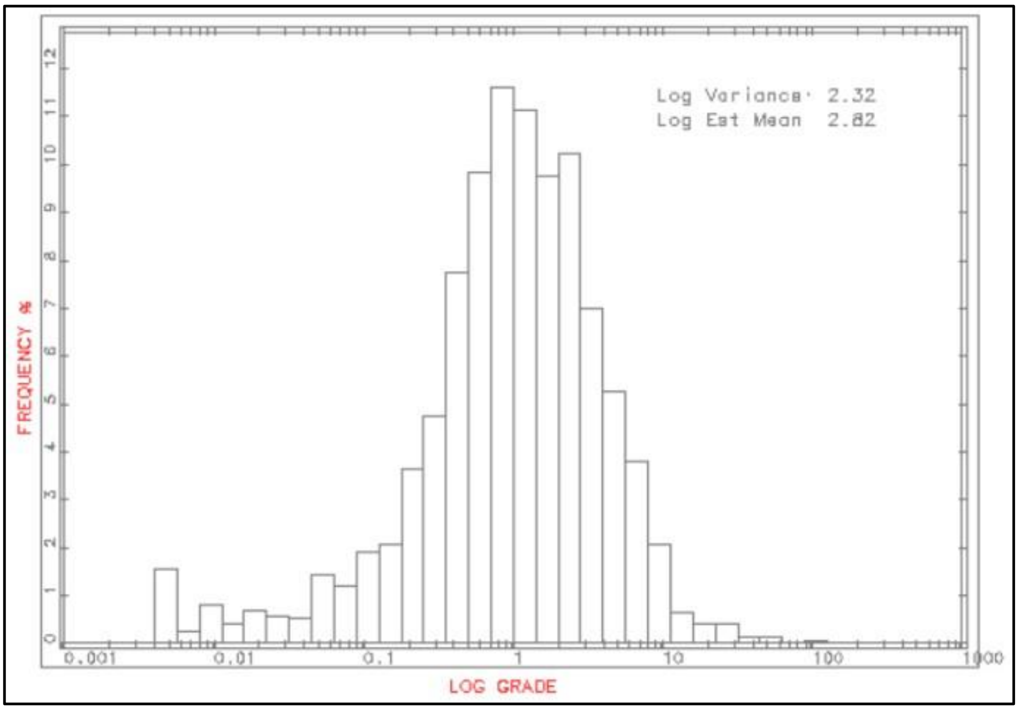


Figure 14.8: Frequency vs Log Grade Plot of Selected Samples

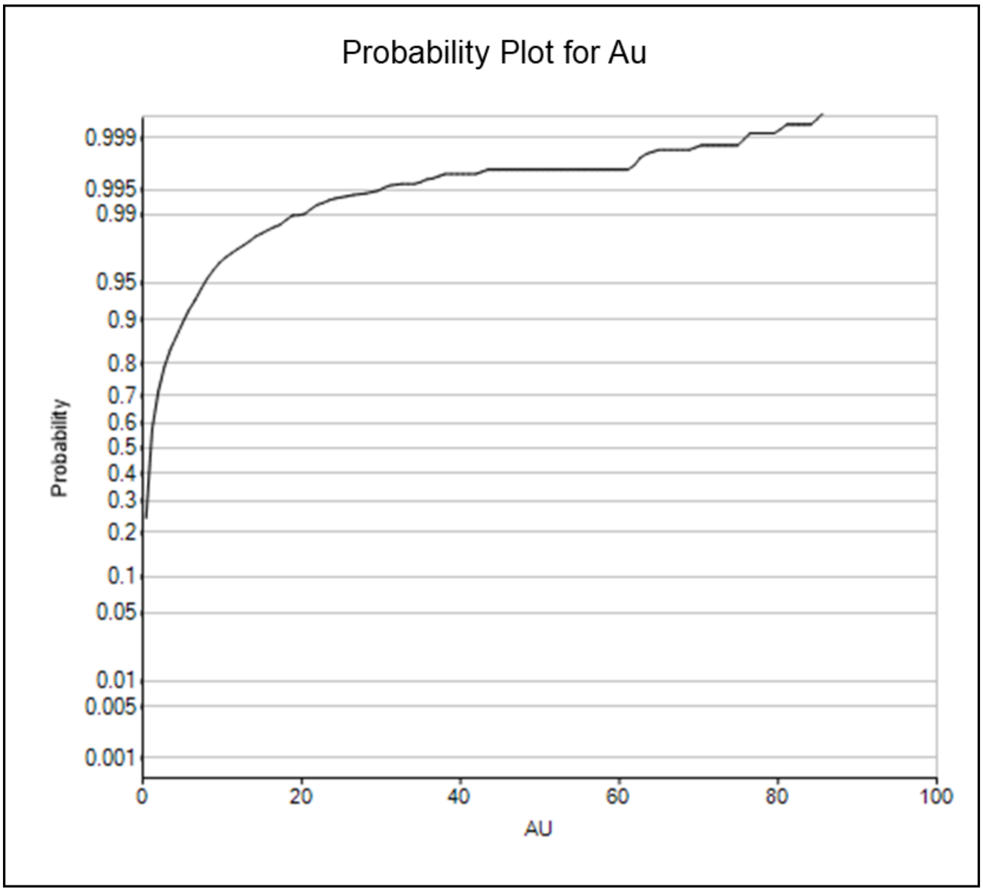


Figure 14.9: Probability Plot of all the Selected Gold Assays

The application of the capping significantly reduced the noise in the assay grade database as seen in the significant drop of both the variance and co-efficient of variation (see Table 14.5). The 18 g/t Au capping affected 18 samples (1 %) of the composited samples. Most of the samples affected by capping were in Zone 2 (BIF mineralised zone), though their concentration in Zone 2 suggests that they could be real and not discrete such that capping was utilised as a conservative control. It is worth noting also that the Adumbi gold grades do not show any direct correlation with the sample length, so capping is permissible. Minecon did further reviews on the impact of using a lower cap on the resource but decided to use the 18 g/t Au cap as lower capped grades affected a greater number of samples and thus impacted the overall resource.

Table 14.5: Descriptive Statistics of Selected and 2 m Composite and Capped Samples within Mineralised Zones

Field	No. of Samples	Min. (g/t)	Max. (g/t)	Mean (g/t)	Variance	Logvar	Cov	Description
Au	4,740	0.01	170	2.17	30.96	2.97	2.56	Selected Adumbi DD samples within ore wireframe
Au	1,731	0.01	90	2.11	17.98	2.32	2.01	Selected Adumbi DD samples, composited 2 m uncapped
Au	1,731	0.01	18	1.97	7.35	2.29	1.38	Selected Adumbi DD samples, composited 2 m capped at 18 g/t
Au	868	0	12.4	0.29	0.70	3.57	2.87	All Adumbi resurveyed adit log samples
Au	130	0	12.4	0.79	2.39	2.49	1.97	Selected Adumbi resurveyed adit samples within ore wireframe
Au	264	0.02	12.8	0.98	2.43	1.70	1.58	Selected Adumbi trench samples within ore wireframe
Au	92	0.01	24.1	2.21	10.11	3.29	1.44	All Zone 5 2 m composite samples
Au	442	0.01	23.8	2.61	12.21	3.65	1.34	All Zone 1 2 m composite samples
Au	636	0.01	62.4	2.20	22.19	2.18	2.15	All Zone 2L 2 m composite samples
Au	351	0.01	90	1.83	28.26	2.27	2.9	All Zone 2U 2 m composite samples
Au	135	0.01	7.59	1.21	1.85	2.60	1.13	All Zone 3 2 m composite samples
Au	92	0.01	18	2.15	7.58	3.27	1.28	All Zone 5 2 m composite capped at 18 g/t samples
Au	442	0.01	18	2.55	10.10	2.24	1.25	All Zone 1 2 m composite capped at18 g/t samples
Au	636	0.01	18	1.98	8.06	2.13	1.43	All Zone 2L 2 m composite capped at 18 g/t samples
Au	351	0.01	18	1.59	4.67	2.22	1.36	All Zone 2U 2 m composite capped at 18 g/t samples
Au	135	0.01	7.59	1.21	1.85	2.60	1.13	All Zone 3 2 m composite capped at 18 g/t samples

14.6 ASSAY INTERVAL COMPOSITING

The dominant sample length in the Adumbi drillhole database is 1 m. Figure 14.10 shows the select sample length versus count. The mean sample length is 0.75 m. Approximately 70 % of the selected samples had sample lengths in the range 0.5 m to 1.5 m. Minecon applied 2 m down-the-hole sample compositing to reduce the variability of the data for samples selected within the mineralised wireframe. Compositing of the selected samples was restricted to the individual zones within the wireframe. The restrictions ensured that the geological and mineralisation definition was maintained. The minimum composite length was set to 1 m. The

Datamine compositing parameter (MODE) was set to Value 1 to ensure that every sample fitted into one of the composites. The descriptive statistics of the samples selected within the mineralisation prior to compositing and after compositing are shown in Table 14.6. A histogram of the resulting 2 m composite lengths at MODE=1 is illustrated in Figure 14.11.

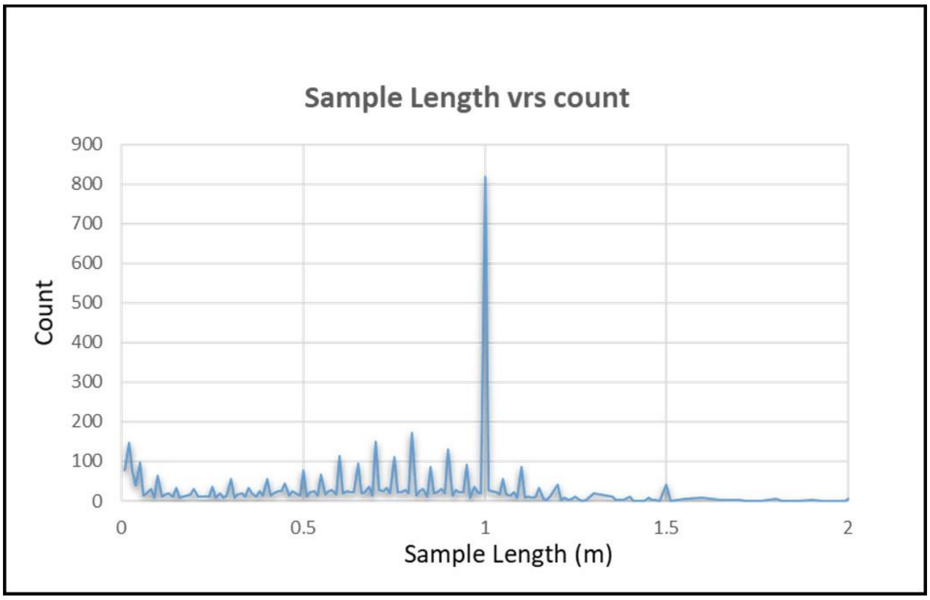


Figure 14.10: Select Sample Length vs Count

Table 14.6: Descriptive Statistics of Selected Samples within Mineralised Zones from Wireframes

Field	No. of samples	Min. (g/t)	Max. (g/t)	Mean (g/t)	Variance	Logvar	Cov	Description
Au	222	0.01	26.80	2.64	20.02	3.51	1.69	All Zone 5 samples
Au	1,242	0.01	80.20	2.76	24.36	2.99	1.79	All Zone 1 samples
Au	1,779	0.01	170.00	2.15	44.24	3.07	3.09	All Zone 2L samples
Au	939	0.01	117.00	1.80	31.95	2.67	3.15	All Zone 2U samples
Au	350	0.01	13.30	1.28	2.85	2.11	1.32	All Zone 3 samples
Au	92	0.01	24.14	2.21	10.11	3.29	1.44	All Zone 5 2 m composite samples
Au	442	0.01	23.76	2.61	12.21	3.65	1.34	All Zone 1 2 m composite samples
Au	636	0.01	62.43	2.20	22.19	2.18	2.15	All Zone 2L 2 m composite samples
Au	351	0.01	90.01	1.83	28.26	2.27	2.90	All Zone 2U 2 m composite samples
Au	135	0.01	7.59	1.21	1.85	2.60	1.13	All Zone 3 2 m composite samples

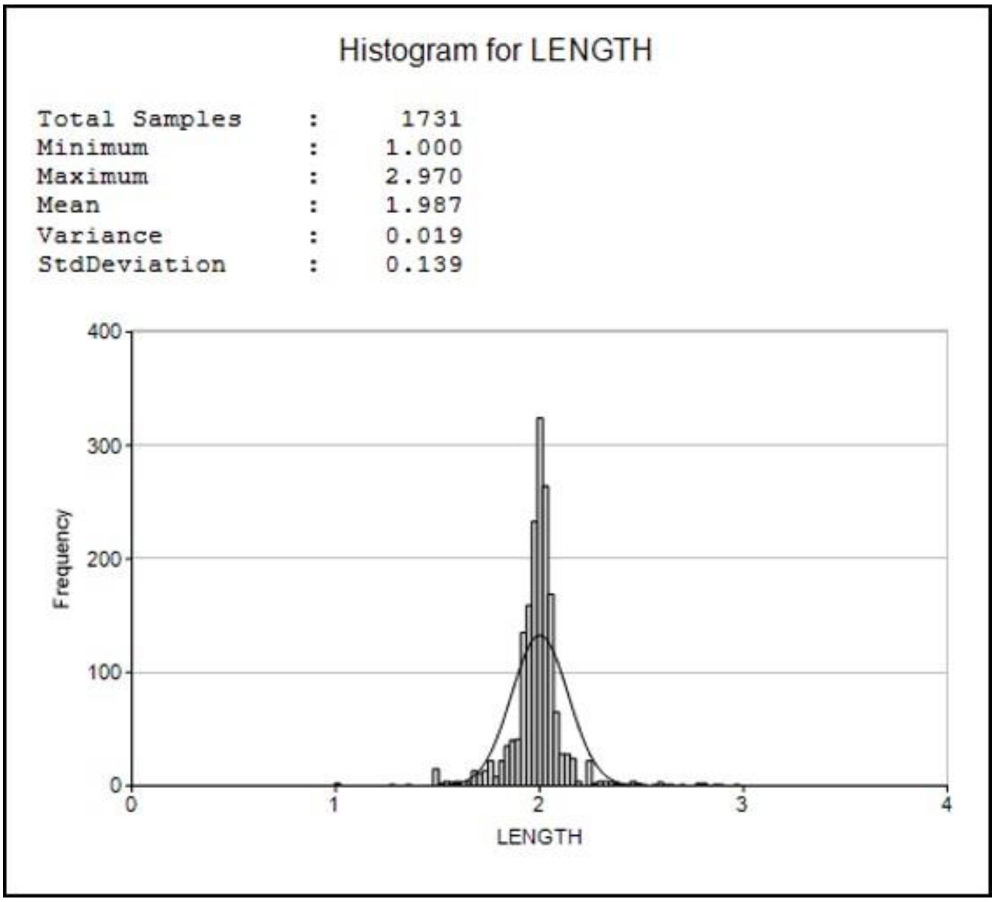


Figure 14.11: Histogram of the resulting 2 m Composite Lengths at MODE=1

14.7 MINERALISATION CONTINUITY AND VARIOGRAPHY

For the variography analysis, Minecon used the selected samples within the mineralisation wireframe, composited into 2 m and capped at 18 g/t Au as input data into Datamine to generate and study the variograms in several directions: downhole, along strike, down-dip and cross structure. The capping was to aid in getting smoother variograms.

Variograms were re-modelled for mineralised zones with sufficient samples to support meaningful variograms, and the parameters obtained were applied to all the mineralisation. The nugget value derived from the downhole variogram was fixed at 0.17. A typical example of the variograms, the along strike variogram, is shown in Figure 14.12. Variograms will be reviewed as and when more drilling data becomes available in future.

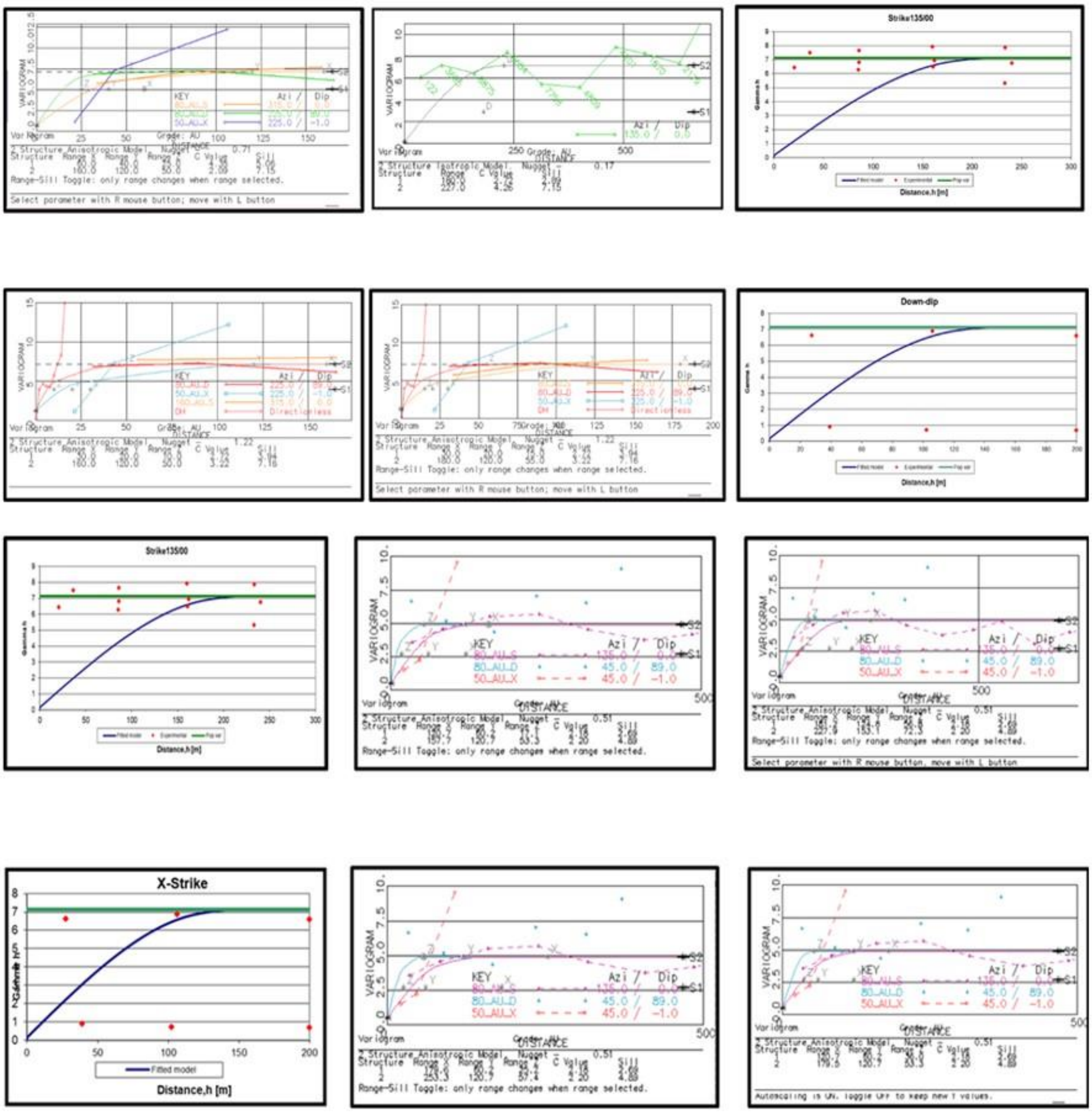


Figure 14.12: Adumbi Variograms and Models in Different Directions

The parameters used in the volume model parameter file are as listed in Table 14.7.

Table 14.7: Variogram Model Parameters

VREFNUM	VANGLE1	VANGLE2	VANGLE3	VAXIS1	VAXIS2	VAXIS3	NUGGET	ST1
1	225	89	0	3	1	0	0.17	1
ST1PAR1	**ST1PAR2**	**ST1PAR3**	**ST1PAR4**	**ST2**	**ST2PAR1**	**ST2PAR2**	**ST2PAR3**	**ST2PAR4**
180	120	55	0.38	1	227	151	72	0.45

14.8 BLOCK MODELS

The Adumbi block model origin and block size are outlined in Table 14.8.

Table 14.8: Adumbi Block Model Origin and Block Size

Parameter	Easting	Northing	RL
Model Origin	594,200	191,200	−300
Parent Block Sizes (m)	4	4	4
Subcells	2	2	2

The model limits are in Table 14.9.

Table 14.9: Adumbi Model Limits

Field	Minimum	Maximum	Range
Easting	594,584	596,101	1,516
Northing	191,432	193,196	1,763
RL	−101	780	881

The orientation of the model is 135° along the strike of the mineralisation. The number of blocks in the various dimensions as per the above model limits are Easting (380), Northing (440) and vertical (220). The along strike length of the model is 2,300 m.

14.9 INTERPOLATION SEARCH PARAMETERS AND GRADE INTERPOLATION

The Adumbi deposit mineral resource was estimated by Minecon using Ordinary Kriging with the ellipsoidal search parameters as listed in Table 14.10.

Table 14.10: Ellipsoidal Search Parameters

DIST1	SDIST2	SDIST3	SANGLE1	SANGLE2	SANGLE3	SAXIS1	SAXIS2	SAXIS3
180	120	55	225	89	0	3	1	0

The search ellipsoid was aligned along the strike of the mineralisation with a long axis search range along the strike of 180 m, a down-dip search range of 120 m, and a cross-structure search range of 55 m based on the average ranges obtained from the principal direction through variography. The dip of the mineralisation is almost vertical and hence set to 89°.

A minimum of 2 samples and maximum of 24 samples were used to effect the grade interpolation. Zonal restriction was applied. A two times expansion of the search volume was utilised by setting the SVOLFAC to 2 to ensure that most blocks had grades interpolations into them.

A block model prototype (see Table 14.11) was prepared and used to fill the Adumbi closed-volume geological wireframe with cells.

Table 14.11: Adumbi Block Model Prototype

Parameter	Easting	Northing	RL
Model Origin	594,200	191,200	−300
Parent Block Sizes (m)	4	4	4
Number of Blocks in Different Directions	575	575	280

The surface topography DTM was used to trim the upper part of the model. Subcell splitting was used along other surfaces, including BOCO and TOFR, to preserve the shape of the mineralisation. Each cell of the prototype was uniquely assigned one of the three oxidation states. The wireframe interpretation of the various mineralised zones, though continuous, shows considerable variability in the local strike directions. The estimation process used the Dynamic Anisotropy optional feature of Datamine. True dip and dip azimuths were calculated from the wireframe triangles. These were then angle-estimated into the blocks using inverse distance squared interpolation (with adaption for circular data). Appropriate constraints were applied to avoid inappropriate angles from the edges of truncated wireframes. Block grades were estimated using Ordinary Kriging, which used the local orientation of the search ellipsoid. Grades were estimated into parent cells. Two passes were made for grade interpolation. Restrictions were employed so that only grades within particular zones influenced that zone grade interpolation. The BOCO and TOFR model surfaces were used to control the assignment of relative densities to the various material types in the model: oxide (2.45), transition (2.82) and fresh (3.05).

14.10 HISTORICAL AND ARTISANAL MINING DEPLETION

No additional studies on depletion by artisanal activity have been undertaken since the RPA study. Minecon has therefore subtracted the same amount of material reported as depletion by RPA in the 2014 studies from the final estimates, assuming that all the material is oxide. A total of 19,361 oz of gold, 457,000 t at a grade of 1.32 g/t was subtracted as depletion due to historical mining. Minecon was unable to verify depletion due to historical and artisanal mining activities.

It is important that further works be undertaken to help better estimate depletion due to historical and recent artisanal mining.

14.11 RESOURCE CLASSIFICATION

Using the CIM Definition Standards on Mineral Resources and Mineral Reserves:

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic material, or natural solid fossilised organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has "reasonable prospects for economic extraction."

Mineral Resources are classified into Measured, Indicated and Inferred categories based upon increasing geological confidence. In addition, resource classification within mineralisation envelopes are generally based on drillhole spacing, grade continuity, and overall geological continuity. The distance to the nearest composite, amount of extrapolation from last drillhole, number of samples used to interpolate grades into blocks, and the number of drillholes are also considered in the classification.

There is increased understanding of the geology and mineralisation controls of the Adumbi deposit following the technical works undertaken between 2010 and 2021.

The Adumbi Mineral Resource has been classified into Indicated and Inferred Resources. This was informed by improved confidence in the geological knowledge of the Adumbi deposit, well established mineralisation and geological continuity, increased drilling data density and increased reliability of the database, amongst other considerations.

In portions of the model where drilling data spacing consistency reaches 80 m, block cells estimated from sampling within a one-variogram range search ellipsoid, supported by positive Kriging efficiency (KEF), were identified and confined using sectional digitised strings. The strings were linked to form a wireframe surface which was used to select and flag confidence levels into the orebody. Cells falling within this wireframe surface were classified as Indicated and those falling within a two-variogram range, supported by adequate number of samples for valid local estimates and lying within the US$1,600/oz optimised pit shell, were classified as Inferred. Figure 14.13 shows a section through the model coloured on the KEF values and classified as Indicated and Inferred.

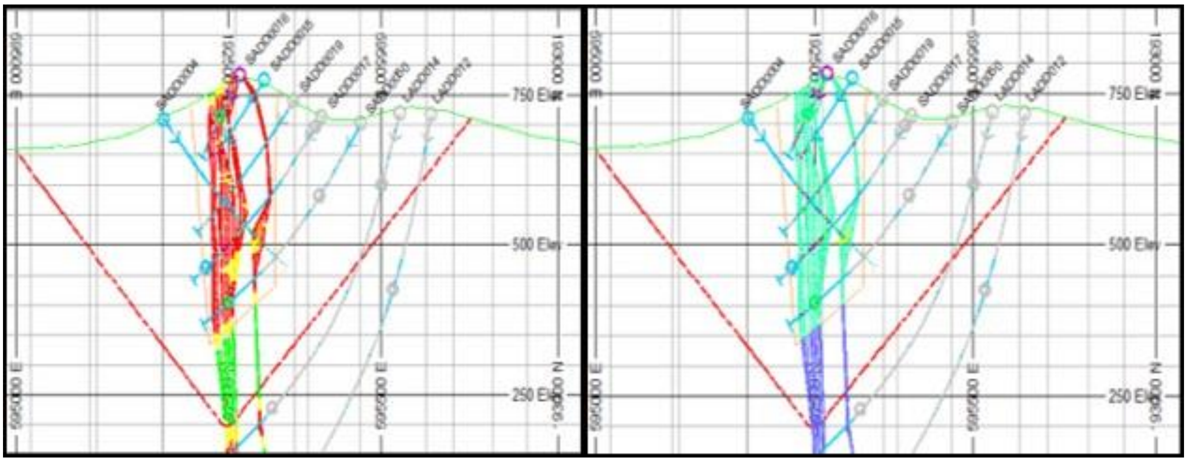


Figure 14.13: Section through Model Coloured on KEF Values and Classified as Indicated and Inferred Resource

14.12 CUT-OFF GRADE PARAMETERS

Minecon, in consultation with Loncor Management, employed a gold price of US$1,600/oz for in-pit optimisations to limit and constrain the Adumbi deposit in-pit resources.

Pit Optimisation Parameters

To constrain the depth extent of the geological model and any mineral resources, an open pit was constructed for the Adumbi deposit based on the following pit optimisation parameters:

- A gold price of US$1,600/oz
- A block size of 16 m × 16 m × 8 m
- A 32 m minimum mining width and a maximum of 4 m of internal waste was applied
- A mining dilution of 100 % of the tonnes at 95 % of the grade
- An ultimate pit slope angle of 45°
- An average mining cost of US$3.29/t mined
- Metallurgical recoveries of 91 % for oxide, 88 % for transition and 90 % for sulphide
- An average general and administration cost of US$4.20/t
- Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material, constrained by a US$1,600/oz optimised pit shell
- Transport of gold and refining costs equivalent to 4.5 % of the gold price
- No additional studies on depletion by artisanal activity have been undertaken since the RPA 2014 study, and the same total amount of material was used by Minecon

The preliminary open-pit shell provided a constraint for the reported open-pit resources based on the 2014 CIM requirement for Mineral Resources to have “reasonable prospects for economic extraction”.

All the model blocks with grades above the block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t for transition and 0.63 g/t Au for fresh material within the US$1,600/oz pit shell and truncated at the surface by the topography were reported as a mineral resource for Adumbi (see Figure 14.14).

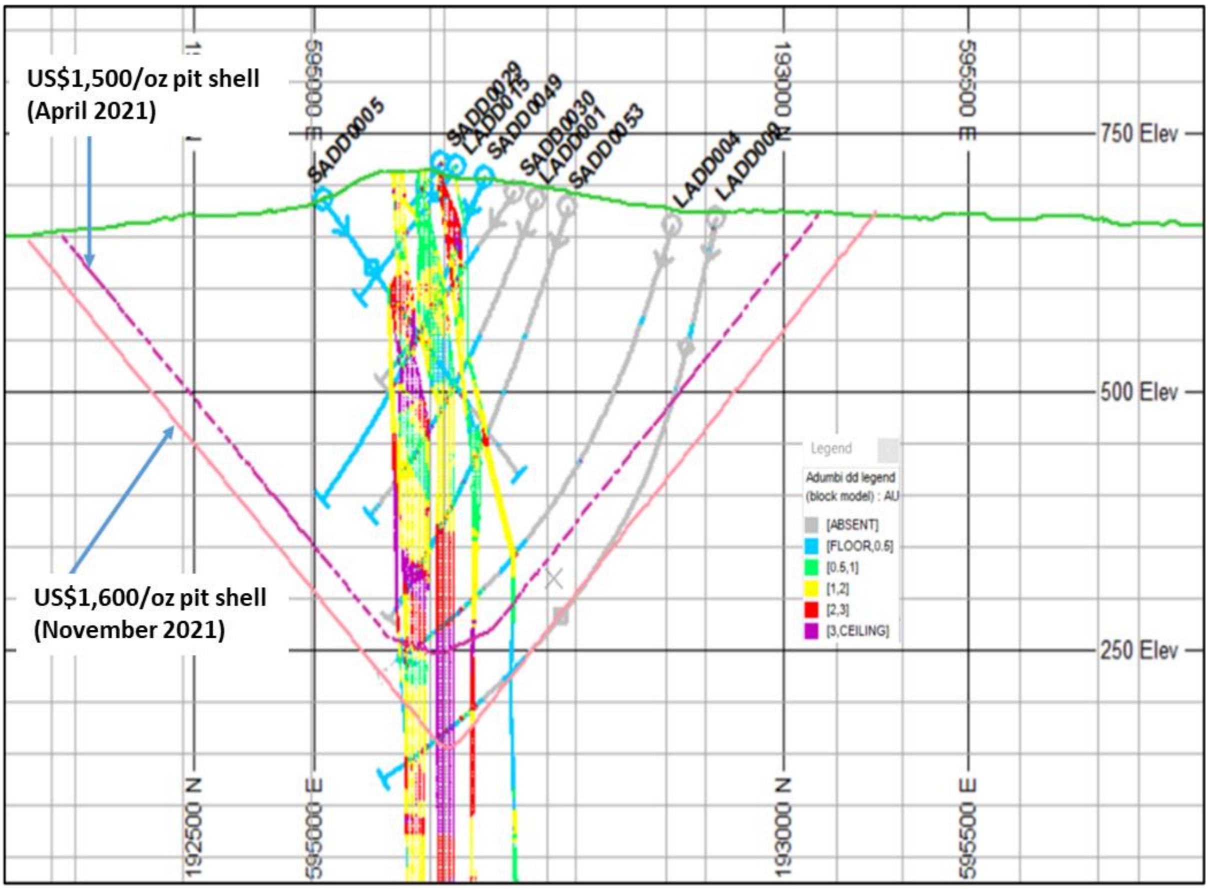


Figure 14.14: Adumbi Model Section showing the US$1,500/oz April 2021 Inferred Resource Pit Shell and the US$1,600/oz November 2021 Pit Shell

The results of the Adumbi pit optimisation (see Figure 14.15) resulted in 1.88 Moz of gold (28.19 Mt grading 2.08 g/t Au) for the Indicated Mineral Resource and 1.78 Moz of gold (20.83 Mt grading 2.65 g/t Au) for the Inferred Mineral Resource constrained within the US$1,600/oz pit shell. Figure 14.16 shows the Adumbi model coloured by material type.

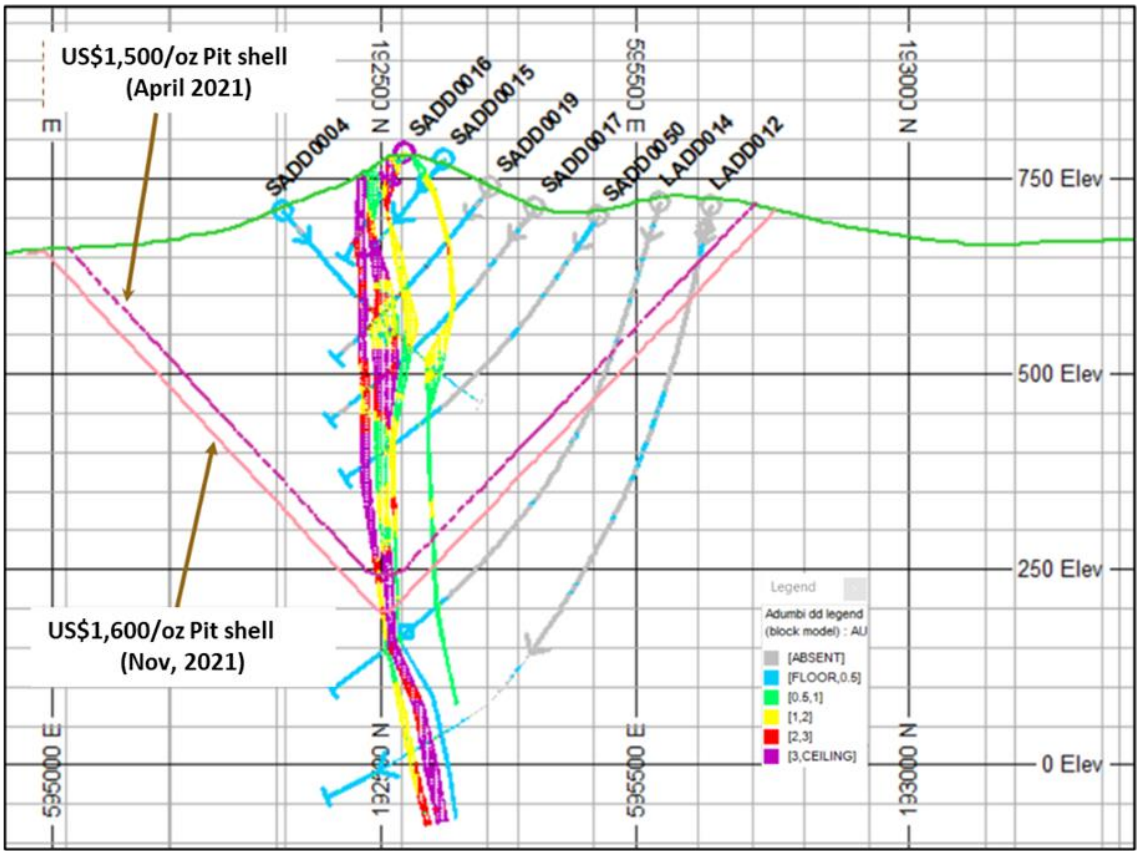


Figure 14.15: Adumbi Section showing Resource Model with Holes coloured on Grade and the US$1,500/oz April 2021 Pit Shell and the US$1,600/oz November Pit Shell

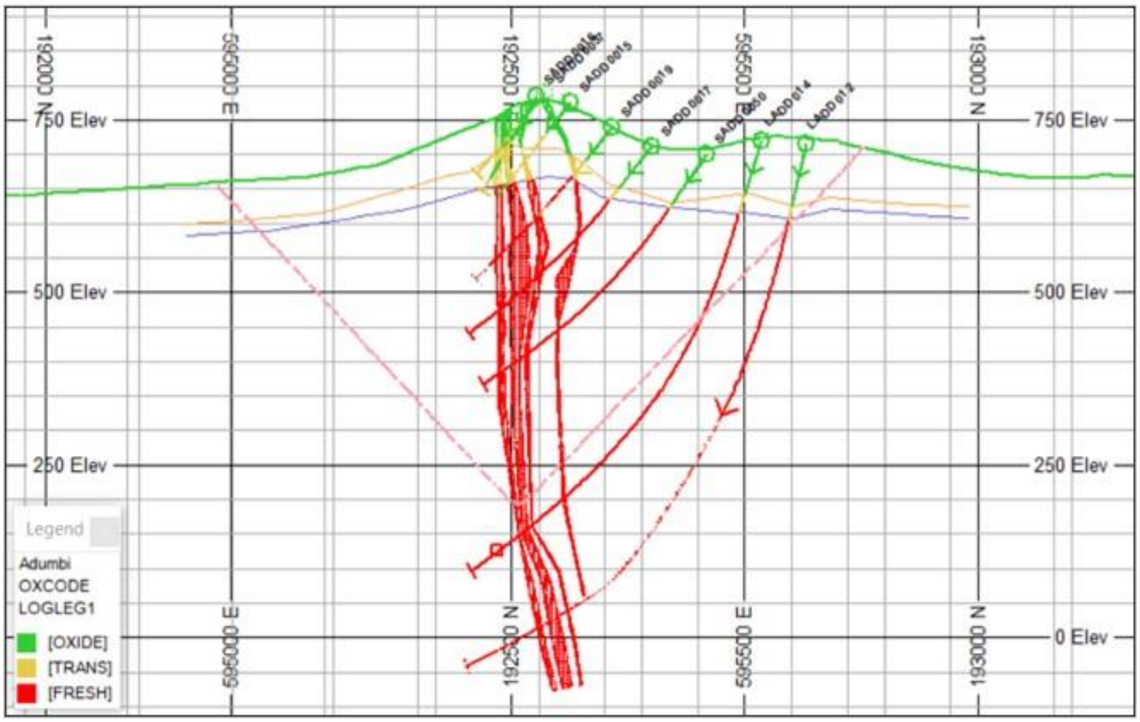


Figure 14.16: Adumbi Block Model coloured by Material Type: Oxide, Transition and Fresh

Figure 14.17 shows the 3D grade model illustrating the previous Minecon US$1,500/oz pit shell (April 2021) and the current US$1,600/oz pit shell (November 2021).

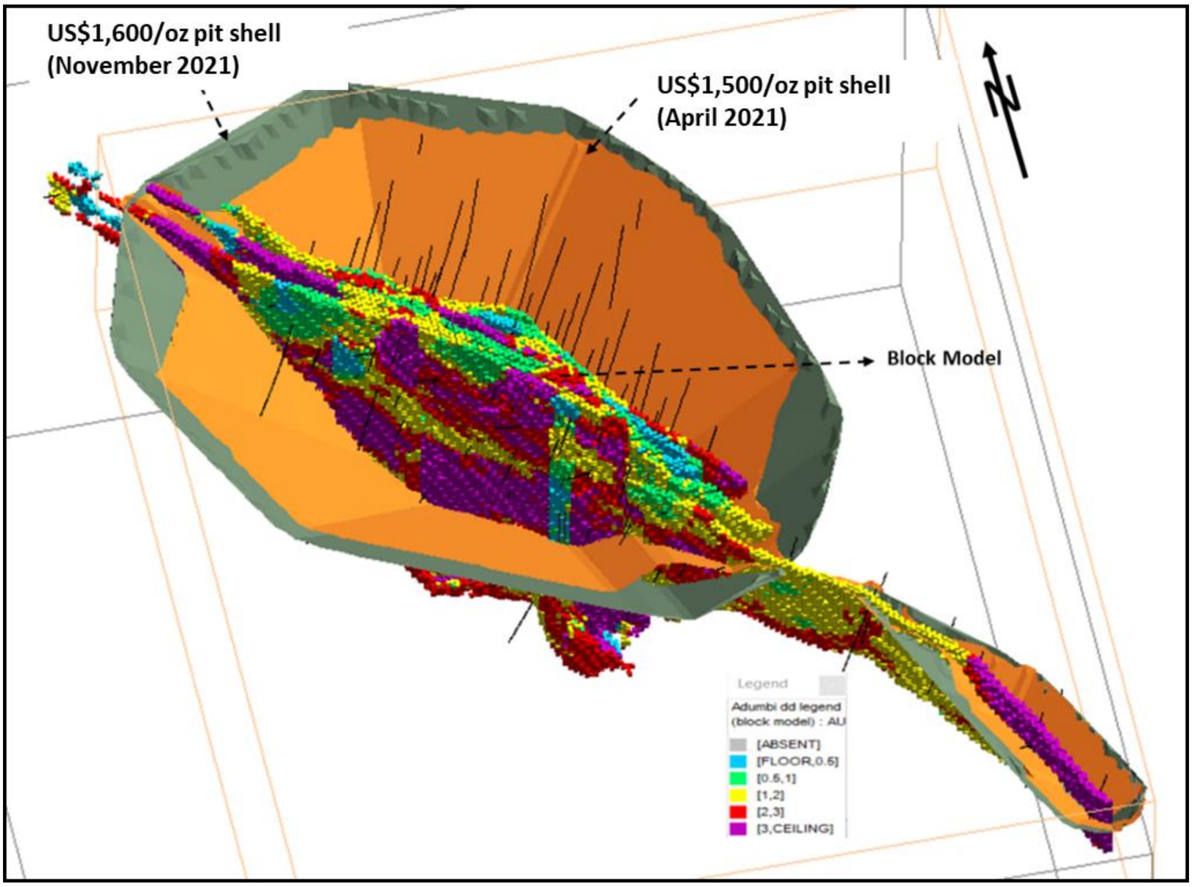


Figure 14.17: 3D Grade Model showing the April 2021 US$1,500/oz and November 2021 US$1,600/oz Pit Shell

The grade-tonnage curves for the Adumbi mineral resources at various gold cut-offs are summarised in Table 14.12 and shown in Figure 14.18.

Table 14.12: Adumbi Mineral Resource Sensitivity by Cut-Off Grade

Block Cut-Off	Tonnage	Grade	Contained Gold
g/t Au	Mt	g/t Au	Moz
0.0	51.60	2.23	3.70
0.5	50.10	2.29	3.68
1.0	41.15	2.61	3.45
1.5	29.07	3.17	2.97
2.0	21.76	3.66	2.56
2.5	16.06	4.17	2.15
3.0	12.12	4.63	1.80

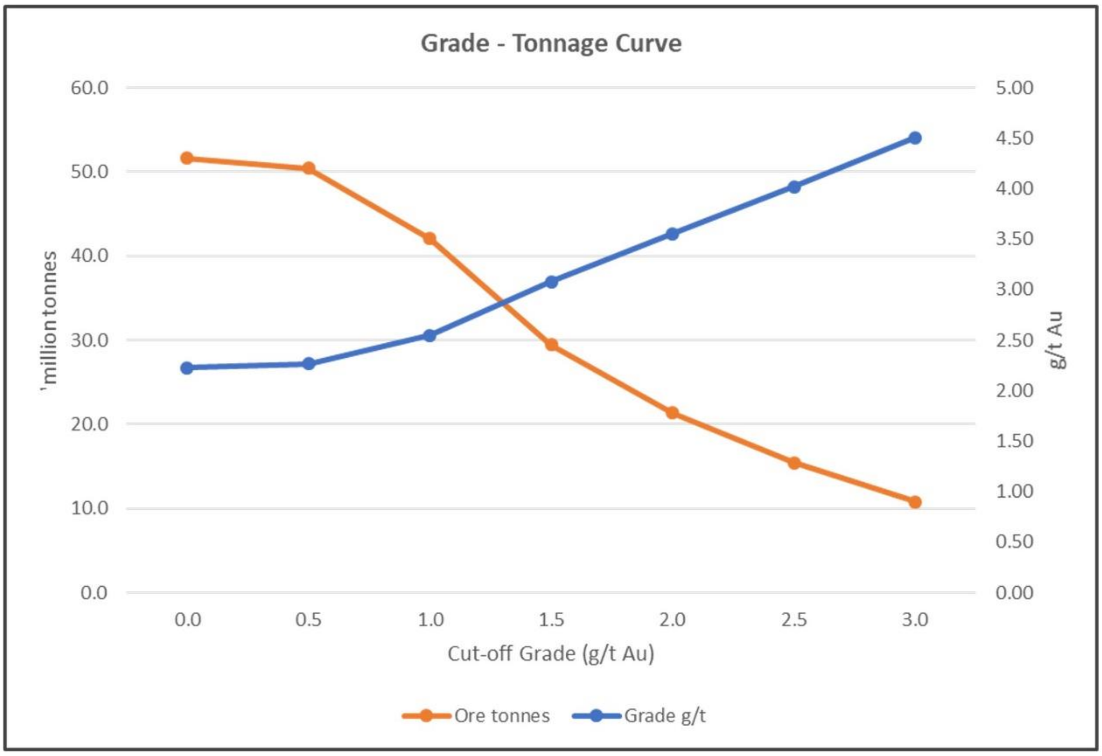


Figure 14.18: Grade-Tonnage Curve for Adumbi Mineral Resource

14.13 MODEL VALIDATION

Minecon carried out various block modelling validation procedures to check the robustness of the model. These included the following:

- Visual comparison of the block grades versus the composited grades used to interpolate the grades into the block in section and plan
- Statistical comparison
- Comparison of individual blocks and composite grades
- Model extent comparison
- Cross validation
- Check conducted on search ellipsoid orientations

A visual comparison of the block model grades with the adjacent composite drillhole grades that were used to interpolate grades into them showed a good correlation. Figure 14.19 to Figure 14.21 show Minecon's block model with the US$1,500/oz April 2021 Inferred Resource pit shell outline and the current November US$1,600/oz pit shell outline.

A statistical comparison of the mean grade of the block model with the mean composited grades of the selected samples within the mineralised wireframe showed a good correlation, suggesting that there was not much bias in the estimation process (see Table 14.13).

The model and wireframe extents compared well (see Table 14.14).

The overall volumes of the mineralised wireframe and the block model compared very well.

The cross-validation graph that was generated also showed that there was a good correlation between the means of the actual grades and the estimate grades, thus also supporting the estimation parameters used (see Figure 14.22).

All the blocks within the block model were checked to ensure that they have been assigned a reasonable grade, the appropriate density, and material type classification based on inputs used.

Checks were conducted on search ellipsoid orientations to ensure that it followed expected orientations during grade interpolation (see Figure 14.23).

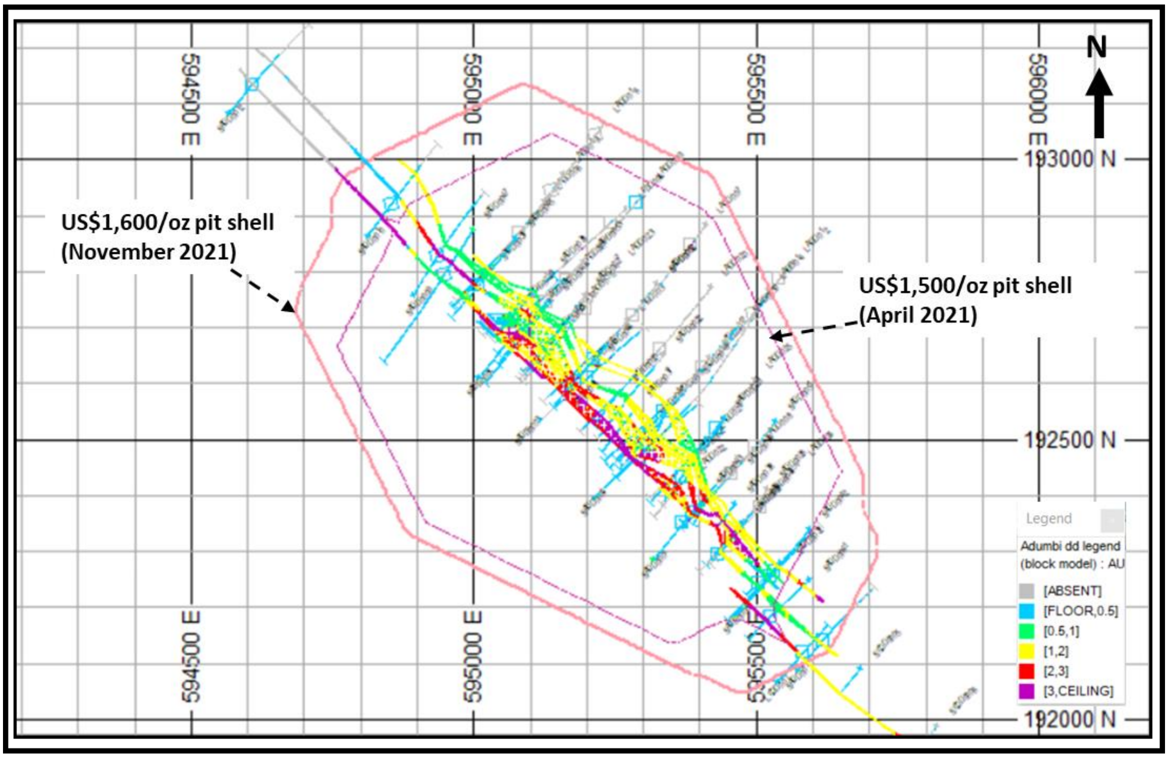


Figure 14.19: Adumbi Deposit Model Flitch at RL560 Coloured on Grade US$1,500/oz April 2021 Pit Shell and US$1,600/oz November 2021 Pit Shell

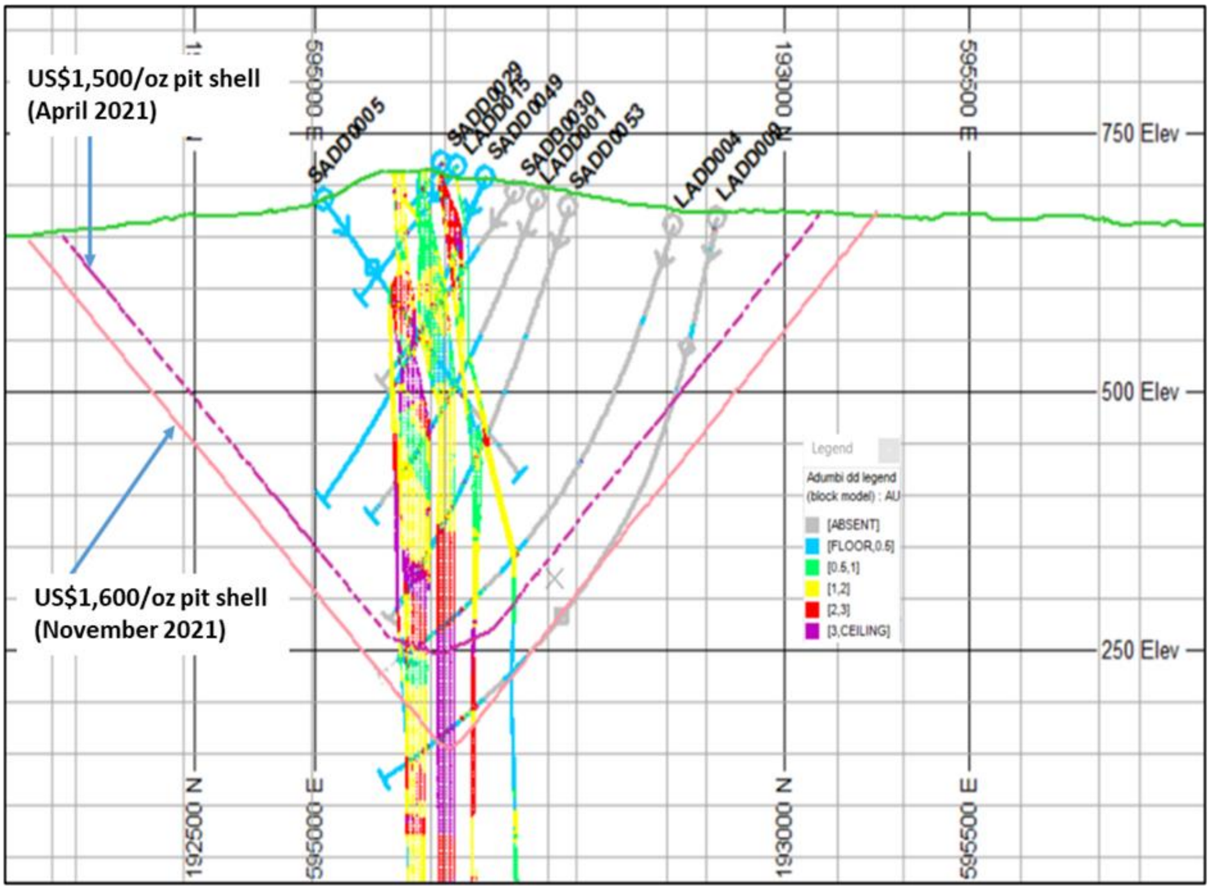


Figure 14.20: Adumbi Model Section showing the US$1,500/oz April 2021 Inferred Resource Pit Shell and the US$1,600/oz November 2021 Pit Shell

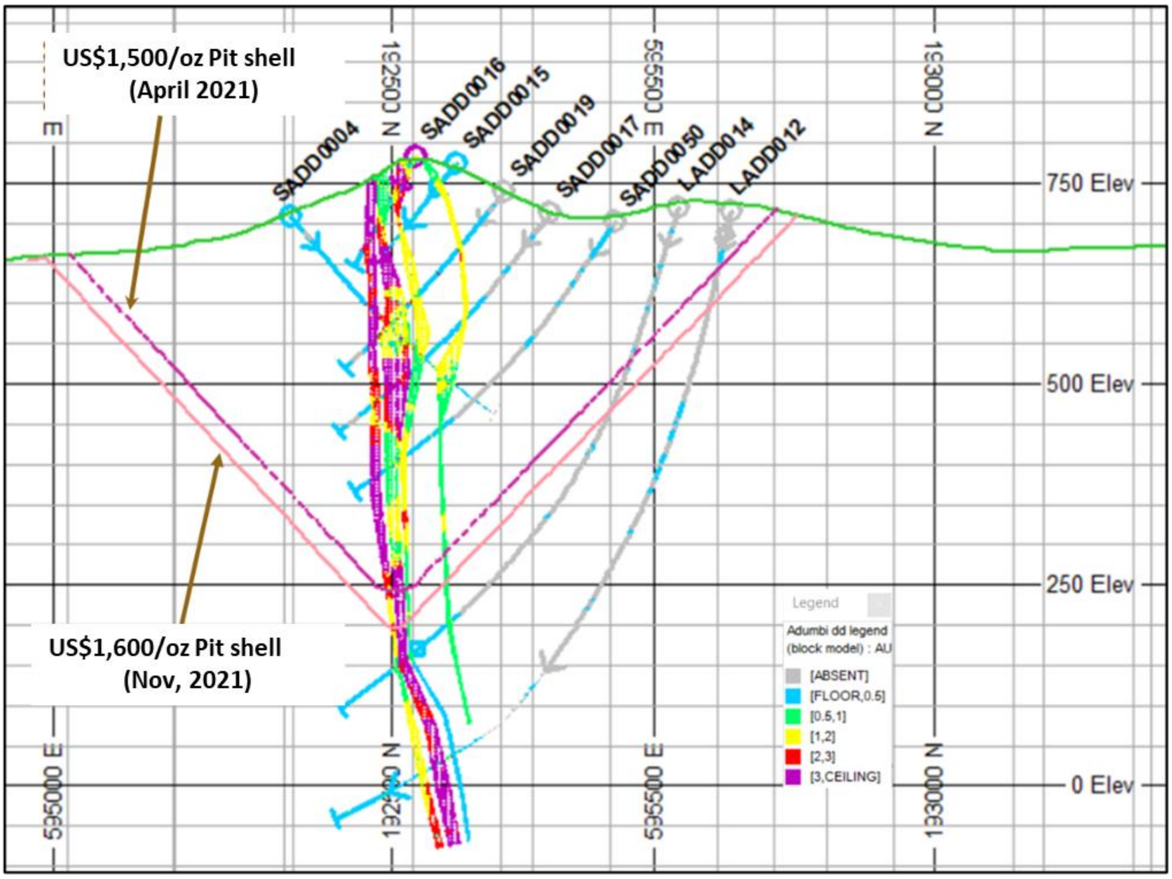


Figure 14.21: Adumbi Section showing Resource Model with Holes Coloured on Grade and the US$1,500/oz April 2021 Pit Shell and the US$1,600/oz November Pit Shell

Table 14.13: Statistical Comparison of Block Model and Selected Samples within Wireframe

Field	No. of Samples	Min. (g/t)	Max. (g/t)	Mean (g/t)	Variance	Logvar	Cov	Description
Au	4,742	0.01	170	2.18	30.95	2.96	2.56	Selected Adumbi DD samples within ore wireframe
Au	1,731	0.01	90.01	2.11	17.98	2.32	2.01	Selected Adumbi DD samples, composited 2 m uncapped
Au	1,731	0.01	18	1.97	7.35	2.29	1.38	Selected Adumbi DD samples, composited 2 m capped at 18 g/t
Au	5,391,813	0.01	15.16	2.09	2.45	0.76	0.75	Block model samples

Table 14.14: Model vs Ore Wireframe Extent Comparison

Field	Block Model	Ore Wireframe	Difference	% Difference
	Minimum	**Minimum**		
X	594,584	594,584	0.1	0.0
Y	191,433	191,429	3.0	0.0
Z	(101)	(101)	0.4	(0.4)
	Maximum	**Maximum**		
X	596,101	596,102	−0.9	(0.0)
Y	193,196	193,198	−2.0	(0.0)
Z	780	780	0.0	(0.0)

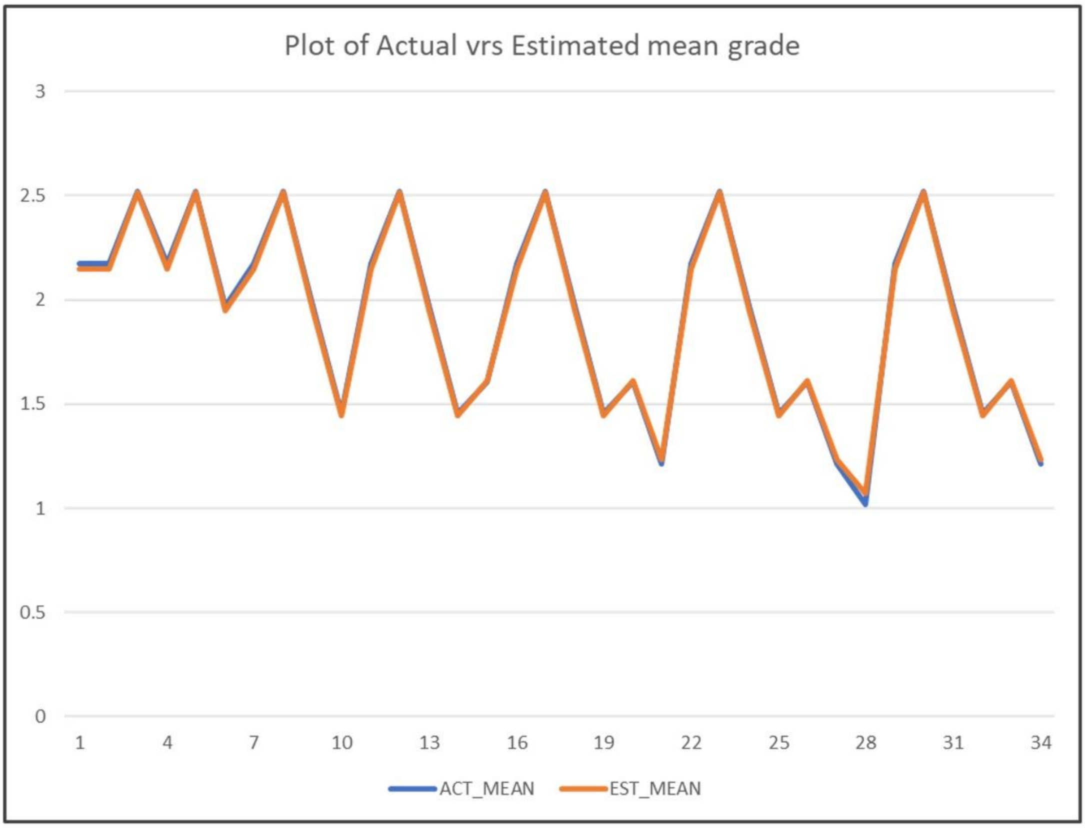


Figure 14.22: Cross-Validation Graph

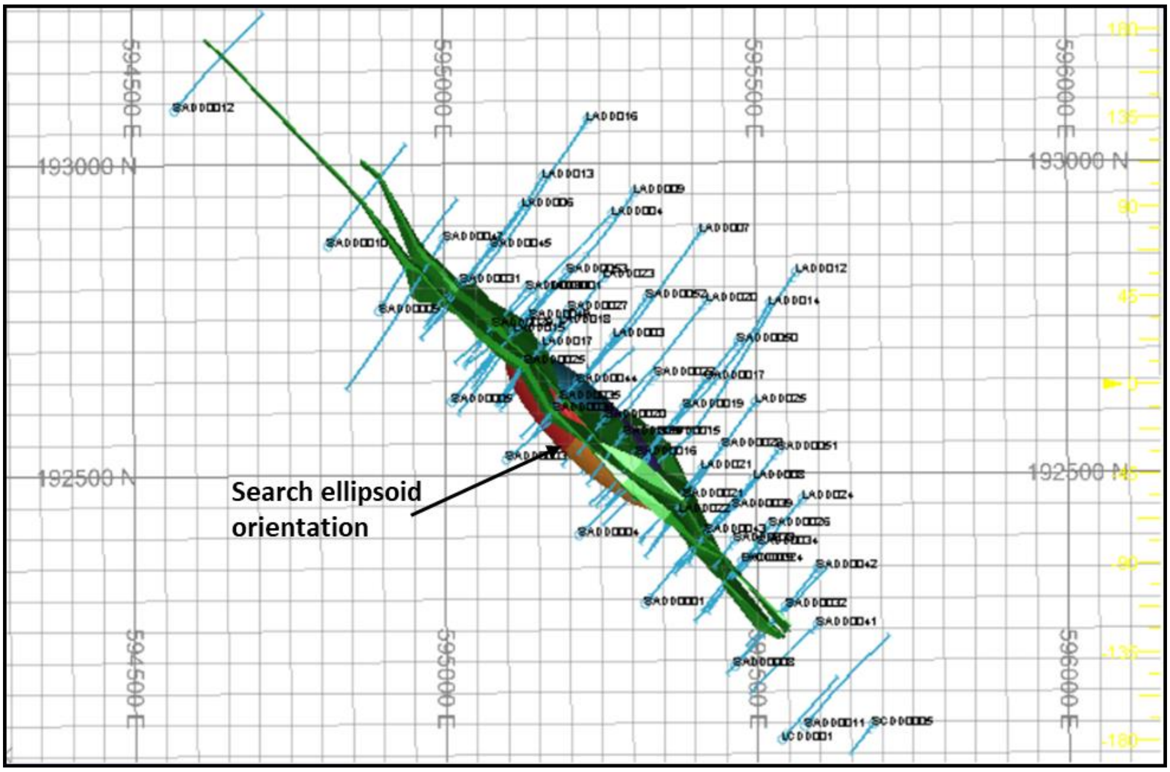


Figure 14.23: Search Ellipsoid Orientation for Grade Interpolation

14.14 MINERAL RESOURCE REPORTING

Minecon has prepared this Mineral Resource estimate for the Adumbi deposit, with a drillhole database cut-off date of October 10, 2021.

The Adumbi Mineral Resource estimate has an effective date of November 17, 2021. The resource is made up of the resources contained in the US$1,600/oz optimised pit with a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material. Table 14.15 summarises the Adumbi Mineral Resources. A total of 84.68 % of the Adumbi mineral resources are attributable to Loncor via its 84.68 % interest in the Imbo Project.

Table 14.15: Adumbi Deposit Indicated and Inferred Mineral Resources (Effective Date: November 17, 2021)

Mineral Resource Category	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Indicated	28,185,000	2.08	1,883,000
Inferred	20,828,000	2.65	1,777,000
NOTES: 1. Mineral resources are not mineral reserves and do not have demonstrated economic viability. 2. Numbers might not add up due to rounding.			

Table 14.16 summarises the Indicated and Inferred category mineral resources in terms of material type.

Table 14.16: Adumbi Mineral Resources by Material Type
(Effective Date: November 17, 2021)

Material Type	Indicated Mineral Resource			Inferred Mineral Resource		
	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Oxide	3,169,000	2.05	208,000	458,000	3.39	49,000
Transition	3,401,000	2.51	274,000	280,000	2.74	24,000
Fresh (Sulphide)	21,614,000	2.02	1,400,000	20,089,000	2.64	1,703,000
TOTAL	**28,185,000**	**2.08**	**1,883,000**	**20,828,000**	**2.65**	**1,777,000**

NOTES:
1. The CIM definitions were followed for Mineral Resources.
2. Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material constrained by a Whittle pit.
3. Mineral Resources for Adumbi were estimated using a long-term gold price of US$1,600/oz.
4. A minimum mining width of 32 m horizontal was used.
5. A maximum of 4 m internal waste was used.
6. Adumbi bulk densities of 2.45 for oxide, 2.82 for transition and 3.05 for fresh rock were used.
7. High gold assays were capped at 18 g/t Au for Adumbi, prior to compositing at 2 m intervals.
8. Numbers might not add up due to rounding.

The Imbo Project Indicated and Inferred Mineral Resource for the combined Adumbi, Manzako and Kitenge deposits now respectively totals 1.88 Moz of gold (28.19 Mt grading 2.08 g/t Au) and 2.09 Moz of gold (22.50 Mt grading 2.89 g/t Au). The total Inferred Resource is summarised in Table 14.17.

Table 14.17: Inferred Mineral Resource for the Imbo Project
(Effective Date: November 17, 2021)

Deposit	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Adumbi	20,828,000	2.65	1,777,000
Kitenge	910,000	6.60	191,000
Manzako	770,000	5.00	122,000
TOTAL	**22,508,000**	**2.89**	**2,090,000**

NOTES:
1. The CIM definitions were followed for Mineral Resources.
2. Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material constrained by a Whittle pit.
3. Mineral Resources for Adumbi were estimated using a long-term gold price of US$1,600/oz.
4. A minimum mining width of 32 m horizontal was used.
5. A maximum of 4 m internal waste was used.
6. Adumbi bulk densities of 2.45 for oxide, 2.82 for transition and 3.05 for fresh rock were used. For Kitenge and Manzako, reference is made to the RPA Technical Report, where bulk densities of 1.7 for oxide, 2.2 for transition and 2.7 for sulphide material were used.
7. High gold assays were capped at 18 g/t Au for Adumbi, prior to compositing at 2 m intervals. For Kitenge and Manzako, reference is made to the RPA Technical Report where assays were capped at 50 g/t Au, prior to compositing at 2 m intervals.
8. Estimated historical mining has been removed.
9. Numbers might not add up due to rounding.

A total of 84.68% of the Imbo Project mineral resources are attributable to Loncor via its 84.68% interest in the Imbo Project. The resource estimates at Kitenge and Manzako, which were undertaken by RPA in its February 2014 NI 43-101 technical report, have not been reviewed in this study but are based on underground mining scenarios and at a cut-off grade of 2.70 g/t Au. Reference is made to the estimates reported for Kitenge and Manzako in the RPA February 2014 NI 43-101 technical report.

14.15 DISCUSSION

The 17 additional new holes that were completed before the start of the modelling targeted the following:

- Inferred Resources within the then US$1,500/oz limiting pit shell
- Plunge and depth extension of the mineralisation
- Confirming the geometry of the mineralised bodies at depth with increased confidence

Minecon's updated model for this estimate is deeper than the previous model, incorporating additional lower-grade material because of the improved modifying factors and lower breakeven grade and therefore producing a slightly lower grade block cut-off than the previous model. Minecon has completed a full review of the modifying factors used in developing the current estimates and updated them as appropriate based on the new drilling information. It is Minecon's view that the changes in the cost inputs to the modifying factors have limited influence on the estimation of block grades due to the fact that the lower breakeven grade (0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material), constrained within a US$1,600/oz optimised pit shell, impacted only volume estimation as more lower-grade blocks were captured in this evaluation than in the previous model.

The latest Mineral Resource for the Adumbi deposit represents an increase of 15 %, with increased confidence in the resource as the limiting economic pit shell pushes significantly deeper in the fresh rock. The increased Mineral Resource at Adumbi is mostly in the fresh rock material. Reconciliation work between the previous Minecon model and the current estimate shows that the significant increase in the resources is due to the additional drilling programme intersecting certain additional higher-grade intersections at depth, which has resulted in material being transferred from the unclassified categories within the previous pit into the Inferred Mineral Resource category as well as bringing in material from the down plunge extension to the mineralisation.

In summary, for the Imbo Project, the mineral resources for the Adumbi, Manzako and Kitenge deposits now total 1,883,000 oz of gold (28,185,000 t grading 2.08 g/t Au) in the Indicated category and 2,090,000 oz of gold (22,508,000 t grading 2.89 g/t Au) in the Inferred category, a 15 % increase in the contained gold on the previous resource outlined by Minecon on April, 27, 2021, as well as a conversion of approximately 51.4 % of the Adumbi deposit resource into the higher confidence Indicated Resource category. A total of 84.68 % of the Imbo Project mineral resources are attributable to Loncor via its 84.68 % interest in the Imbo Project. The resource estimates at Kitenge and Manzako, which were undertaken by RPA in its February 2014 NI 43-101 technical report, have not been reviewed in this study and are based on underground mining scenarios and at a cut-off grade of 2.70 g/t Au. Reference is made to the estimates reported for Kitenge and Manzako in the RPA 2014 NI 43-101 technical report.

14.16 RECOMMENDATIONS FOR FURTHER WORK

There is significant additional resource potential at depth and along the strike extension to the southwest (Canal area) within the Adumbi deposit. Minecon recommends that an expanded drilling programme encompassing over 20 km of the planned infill and deep drilling be undertaken at Adumbi to unearth the full potential of the deposit and to advance the project up the value curve. The main recommendations include but are not limited to the following:

- At the Adumbi deposit, the gold mineralisation is still open at depth and along strike to the southwest. Minecon proposes that the deep drilling programme be expanded to delineate additional resources to the southwest (Canal area) of the deposit. Furthermore, infill drilling is required to increase the confidence of the Inferred Mineral Resources reported at this deposit into the Indicated and Measured categories.
- Along trend from Adumbi, the Manzako and Kitenge deposits have Inferred Mineral Resources of 313,000 oz of gold (1.68 Mt grading 5.80 g/t Au) and remain open along strike and at depth. An infill drilling programme of 5,000 m is proposed by Minecon.
- At a distance of 8 km to 13 km along the structural trend to the southeast across the Imbo river and within the Imbo Project, four prospects (Esio Wapi, Paradis, Museveni and Mungo Iko) have been outlined with soil, rock and trench geochemical sampling with similar host lithologies to those at Adumbi. An initial programme of 2,000 m of scout drilling is recommended on these four prospects to determine their mineral resource potential.
- Following from the above drilling programmes and with increased confidence in the mineral resources, Minecon recommends that a pre-feasibility study (PFS) be undertaken at Adumbi and other prospects within the Imbo Project. This would include undertaking

    - Further metallurgical test work
    - Open-pit and potential underground mining studies
    - Improved metallurgical plant processing design
    - Power studies
    - Infrastructural studies
    - Economic and financial studies
    - Environmental and social impact study (ESIS)

- The additional drilling may include close spaced drilling clusters or crosses in three or four parts of the Adumbi deposit to confirm short-scale continuity of the mineralisation and to allow a conditional simulation to be completed if necessary. A total of 24,000 m of drilling (including 7,600 m reverse circulation (RC) drilling and 16,400 m of coring in the mineralised zone) is recommended by Minecon. This would include infill, deep and extension drilling, and further drilling for metallurgical and geotechnical studies.
- The proposed drilling programme should be undertaken in sequential phases: Priority 1 and 2. All the shallow holes will be undertaken using RC drilling. The deep holes (Priority 1) will be pre-collared with RC drilling and drilled off using core drilling. The Priority 2 holes will include slightly deep and shallow holes. The slightly deep holes will also be pre-collared with RC drilling and tailed off using core drilling while the shallow holes will be drilled by RC.

- Further studies should be undertaken to assist proper estimations of historical depletions and depletions by recent artisanal mining. This will allow for increased confidence in the estimates of the open cavities.
- Compilation of the geological and sampling database into a secure central repository database system and a move away from the storage of files in Microsoft Excel are also recommended. The creation of a central repository will ensure that the data has passed QA/QC validation and has replaced the old data set in the database with the appropriate paper trail to support any changes made.

The recommended infill, extension and deep drilling programme has the potential to significantly increase the Adumbi mineral resource with increased confidence for both open-pit and underground mining scenarios (see Figure 14.24).

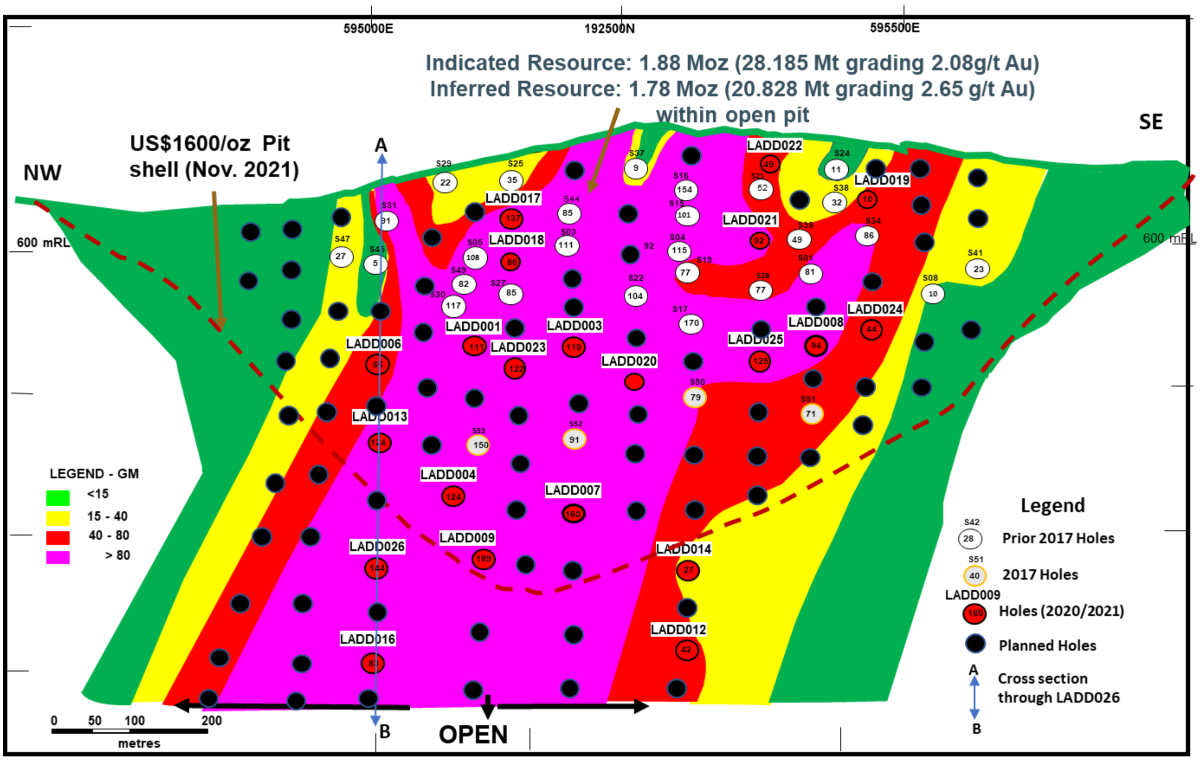


Figure 14.24: Adumbi Deposit Long Section with Existing and Recommended Drillholes

15 MINERAL RESERVE ESTIMATES

No Mineral Reserves have been estimated for the Imbo Project.

16 MINING METHODS

The gold mineralisation at Imbo is found within three identified deposits: Adumbi, Manzako and Kitenge.

This section details the key assumptions, parameters and methods used by Minecon to generate the results of an open-pit optimisation, to develop the practical pit designs, and to determine the inventory of the Mineral Resource within a practical pit design, as well as the production scheduling carried out by Minecon based on the Mineral Resources that were estimated by Minecon and published by Loncor in a press release dated November 17, 2021. It also details the proposed mining methods and provides a summary of the relevant information used to establish the amenability or potential amenability of the mineral resources to the proposed mining methods.

The base case assumed is that electricity will be supplied by the construction of a hydroelectric power plant close to the operation. The proposed Adumbi process plant is based on a 5.0 Mt/a CIL circuit, consisting of crushing, milling and gravity recovery of gold by the cyanidation process. The processing parameters for the mineralised material have been received from SENET, and the study details the optimisation results for the Adumbi deposit only.

In this section, the following has been detailed:

- The results of the practical pit design and production scheduling carried out by Minecon based on the Mineral Resources that were optimised by Minecon in November 2021
- The outcomes of the mining study and production schedule based on a process throughput of 5.0 Mt/a
- Mining equipment requirement and machinery, including but not limited to the selection and size of the mining fleet, and manpower requirements

16.1 MINERAL RESOURCE CLASSIFICATION

Minecon has assumed that a conventional open-pit shovel-and-truck method will be used. In general, the production schedule produced has been driven by two factors: the requirement to pre-strip the Adumbi pit to maintain a consistent and practical waste schedule and the requirement to feed the oxide and transition material through the process plant during the initial years.

Table 16.1 shows the Adumbi deposit Mineral Resource. The pit optimisation and the production schedule take the Indicated and Inferred categories of the Mineral Resource into account.

Table 16.1: Adumbi Deposit Indicated and Inferred Mineral Resources (Effective Date: November 17, 2021)

Mineral Resource Category	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Indicated	28,185,000	2.08	1,883,000
Inferred	20,828,000	2.65	1,777,000

NOTES:

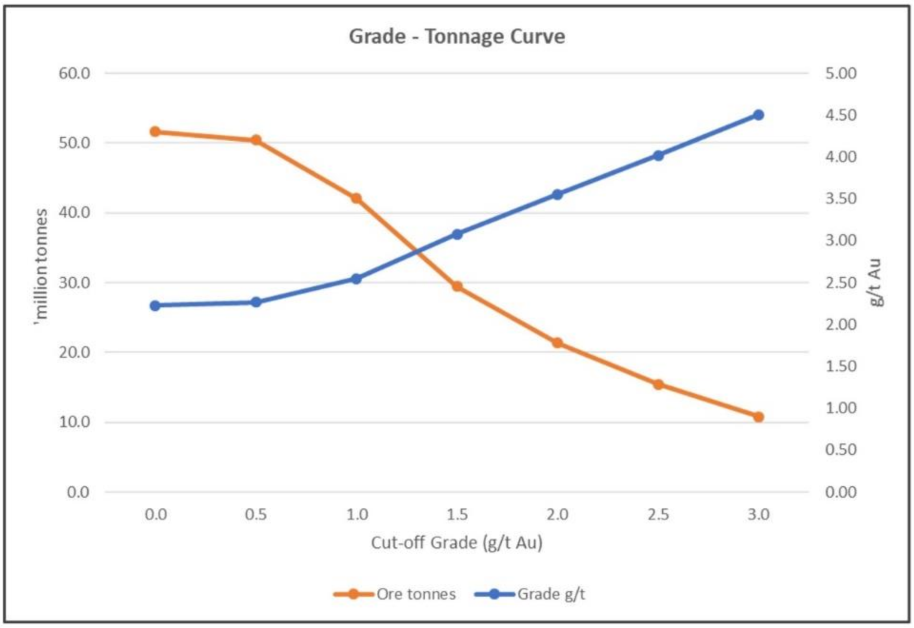
1. The CIM definitions were followed for Mineral Resources.
2. Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material constrained by a Whittle pit.
3. Mineral Resources for Adumbi were estimated using a long-term gold price of US$1,600/oz.
4. A minimum mining width of 32 m horizontal was used.
5. A maximum of 4 m internal waste was used.
6. Adumbi bulk densities of 2.45 for oxide, 2.82 for transition and 3.05 for fresh rock were used.
7. High gold assays were capped at 18 g/t Au for Adumbi, prior to compositing at 2 m intervals.
8. Numbers might not add up due to rounding.

16.2 GRADE-TONNAGE CURVE

Grade-tonnage curves for the Adumbi mineral resources at various gold cut-offs are summarised in Table 14.12 and Figure 16.1.

Table 16.2: Adumbi Mineral Resource Sensitivity by Cut-Off Grade

Block Cut-Off	Tonnage	Grade	Contained Gold
g/t Au	Mt	g/t Au	Moz
0.0	51.60	2.23	3.70
0.5	50.10	2.29	3.68
1.0	41.15	2.61	3.45
1.5	29.07	3.17	2.97
2.0	21.76	3.66	2.56
2.5	16.06	4.17	2.15
3.0	12.12	4.63	1.80



Source: Minecon (2021)

Figure 16.1: Grade-Tonnage Curves for the Adumbi Mineral Resources

16.3 GEOTECHNICAL INVESTIGATION AND SLOPE STABILITY ANALYSIS

There has not been any detail geotechnical investigation and slope stability analysis for the project. The parameters employed in the study are based on information from similar deposits and the initial geotechnical work carried out on the drill core. The slope angles, as employed in the PEA, are based on an assumption of a 45° ultimate pit slope angle.

16.4 OPEN-PIT OPTIMISATION

16.4.1 Introduction

The open-pit optimisation addresses the mining components of the 5.0 Mt/a study and is based on the geological block model generated by Minecon. Note that the Adumbi deposit material in the Indicated and Inferred categories is used in the optimisation. The Kitenge and Manzako materials have not been considered in the study.

A conventional open-pit shovel-and-truck method will be used for the mining of sufficient ore to supply a 5.0 Mt/a of oxide, transition and fresh ore throughput. The mining functions of the operation will be carried out by a mining contractor.

The open-pit optimisation study was performed using the Whittle Four-X Analyser (Whittle 4X) software package to provide guidance to the potential economic final pit geometries. Whittle 4X compares the estimated value of the individual mining blocks at the pit boundary versus the cost for waste stripping. It establishes the pit walls where the ore revenue and waste stripping cost balance for maximum net revenue.

The selected optimum pit shell is then engineered to generate a practical pit design that incorporates the design slope angles and access ramps/haul roads for an operating pit.

The ratio of ore/waste tonnages in the practical pit can be calculated and scheduled to determine the ore production and the waste stripping requirements.

The Whittle process requires various input data including the mineral resource block model, unit costs and other physical parameters such as the slope angles at which the pit can be mined. Appropriate unit costs specific to the Adumbi operation were provided by the relevant parties and benchmarked with previous and similar projects undertaken by Minecon.

The following sections describe the methodology and derivation of the initial Whittle input parameters and assumptions.

16.4.2 Resource Block Model

A single computer block model (Adumbi model) was used as the basic resource model for the pit optimisation study. The resource model was originally built using Datamine Studio software and imported into Surpac for verification to ensure that no losses occurred during the process. The Surpac model was then exported to Whittle 4X. All the mineralised material in the Indicated and Inferred categories was taken into account.

16.4.3 Geotechnical Aspects

Based on the slope parameters derived by Minecon and approved by Loncor for the PEA, the following overall slope angles were used for the pit optimisation:

- General slope angle of 45° in oxide/transition/fresh material
- General slope angle of 45° in all directions

These parameters used assumed dry mining conditions.

16.4.4 Cost Inputs

Cost input parameters were obtained from the initial review of Adumbi undertaken by Minecon and from other open-pit gold projects undertaken by Minecon in Africa. For the purposes of the Whittle optimisation, capital costs, depreciation, amortisation and other interest/finance charges have been excluded.

16.4.4.1 Mining Costs

For the pit optimisation analysis, Minecon has used a reference total mining cost of US$3.29/t with a mining cost adjustment factor of 0.92 applied to the oxide material due to the reduced drill and blast costs. It would have been more accurate to use the cost charged by the mining contractor, but this information is not available.

16.4.4.2 Processing and General and Administration Cost

SENET provided the metallurgical processing recovery factors and costs for the hydro as well as the diesel option with regard to the Adumbi deposit, and the overall general and administration (G&A) costs based on a throughput of 5.0 Mt/a.

A breakdown of the costs and parameters used in the Whittle optimisation runs is shown in Table 16.3.

Table 16.3: Whittle Parameters for the Open-Pit Optimisation

Parameter	Adumbi Deposit
Recovery	Oxide = 91 %, Transition = 88 %, Fresh = 90 %
Processing Cost	Oxide = US$17.81/t processed, Transition = US$18.92/t processed, Fresh = US$22.13/t processed
General and Administration	US$4.2/t processed
Selling Cost (Refinery, Shipment and Government Royalties)	US$72.00/oz

16.4.5 Mining Factors

In consideration of the geo-modelling techniques, the nature of the geological contacts, the grade smoothing incorporated in the resource estimation methodology and the mining equipment selection, it was decided that the mining dilution factor should be set at 5 % (at 0 g/t grade) and mining recovery at 100 %, based on the block size used in the mining model.

16.4.6 Gold Price, Royalties and Selling Costs

A gold price of US$1,600/oz was used for the optimisation, and a range of gold prices was used in the sensitivities: US$1,200/oz, US$1,300/oz, US$1,400/oz, US$1,500/oz, US$1,700/oz, US$1,800/oz, US$1,900/oz and US$2,000/oz (see Table 16.4 and Figure 16.2).

Table 16.4: Optimisation Sensitivities on Gold Price

Gold Price	Total Mined	Ore Mined	Strip Ratio	Diluted Au	Diluted Au	Contained Gold
US$/oz	kt	kt		kg	g/t Au	koz
1,200	156,882	23,256	5.75	49,018	2.108	1,576
1,300	453,342	42,416	9.69	92,122	2.172	2,962
1,400	479,721	44,267	9.84	95,701	2.162	3,077
1,500	521,559	46,526	10.21	100,579	2.162	3,234
1,600	595,812	49,828	10.96	108,262	2.173	3,481
1,700	645,597	51,919	11.43	113,014	2.177	3,633
1,800	665,646	52,881	11.59	114,886	2.173	3,694
1,900	709,117	54,445	12.02	118,503	2.177	3,810
2,000	741,964	55,590	12.35	121,039	2.177	3,891

Source: Minecon (2021)

Figure 16.2: Optimisation Sensitivities on Gold Price

Applying the 2018 mining code, royalties have been set at 3.5 % of the gold price, according to information provided by Loncor. In addition, the revenue calculations were offset by a US$4.00/oz refining cost.

16.4.7 Cut-Off Grade Calculations

The cut-off grade was derived by calculating the overall cost of producing 1 t of ore divided by the recovered value of the gold contained therein. This was determined for oxide, transition and fresh ore types as shown in Table 16.5.

Table 16.5: Cut-Off Grade per Ore Type

Ore Material Type	Recoverable Au Cut-Off (g/t Au)	Equivalent In-Situ Au Cut-Off (g/t Au)
Oxide	0.45	0.52
Transition	0.47	0.57
Fresh	0.54	0.63

16.4.8 Pit Optimisation Assumptions and Parameters

The following summarises the pit optimisation assumptions and parameters used to constrain the depth extent of the geological model to generate the mineral inventory of the open pit for the Adumbi deposit:

- A gold price of US$1,600/oz
- A block size of 16 m × 16 m × 8 m
- A 32 m minimum mining width and a maximum of 4 m of internal waste was applied
- A mining dilution of 100 % of the tonnes at 95 % of the grade
- An ultimate slope angle of 45°
- An average mining cost of US$3.29/t mined
- Metallurgical recoveries of 91 % for oxide, 88 % for transition and 90 % for fresh
- An average G&A cost of US$4.20/t

- Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh materials, constrained by a US$1,600/oz optimised pit shell
- Transport of gold and refining costs equivalent to 4.5 % of the gold price

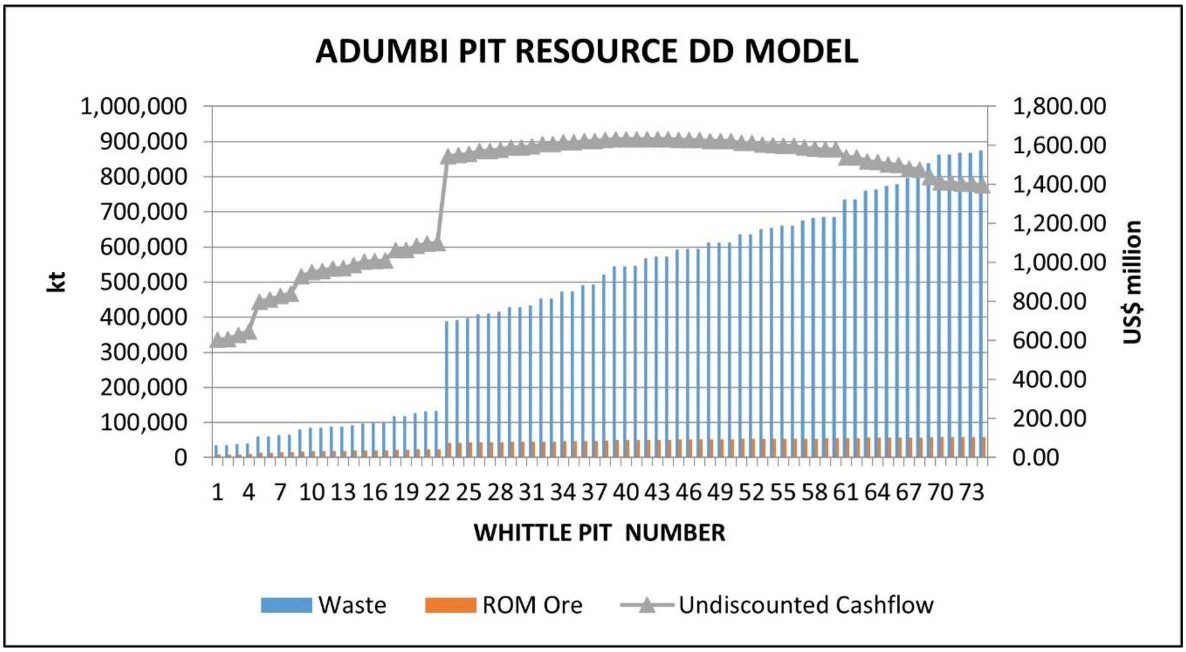
16.5 WHITTLE RESULTS

The results from the Adumbi Whittle pit optimisation for the gold price of US$1,600/oz and a mining cost of US$3.29/t are shown in Table 16.6 and Figure 16.3.

Table 16.6: Adumbi Whittle Pit Optimisation Results

Whittle Pit Number	Recovery Factor	ROM		Waste	Marginal Material		Strip Ratio	Undiscounted Cash Flow
		kt	g/t Au	kt	kt	g/t Au		US$ Million
1	0.5	8,971	2.45	35,772	23	0.55	3.98	605.39
2	0.5125	9,065	2.44	36,028	23	0.55	3.97	608.38
3	0.525	9,554	2.42	38,706	41	0.56	4.05	628.90
4	0.5375	10,084	2.38	40,898	71	0.56	4.05	646.92
5	0.55	14,080	2.22	59,776	93	0.56	4.24	799.71
6	0.5625	14,396	2.21	61,119	96	0.56	4.24	810.33
7	0.575	14,902	2.20	63,728	96	0.56	4.27	827.86
8	0.5875	15,338	2.18	65,715	98	0.56	4.28	841.66
9	0.6	18,050	2.12	80,756	171	0.56	4.46	931.01
10	0.6125	18,623	2.11	84,555	177	0.56	4.53	950.54
11	0.625	18,857	2.11	85,951	179	0.56	4.55	957.59
12	0.6375	19,252	2.10	88,283	179	0.56	4.58	969.03
13	0.65	19,351	2.10	88,676	179	0.56	4.57	971.00
14	0.6625	19,917	2.09	92,355	184	0.56	4.63	986.65
15	0.675	20,546	2.08	97,745	184	0.56	4.75	1,004.62
16	0.6875	20,591	2.08	98,344	184	0.56	4.77	1,006.02
17	0.7	20,837	2.08	99,910	184	0.56	4.79	1,011.67
18	0.7125	23,008	2.05	118,301	235	0.55	5.13	1,065.18
19	0.725	23,010	2.05	118,305	235	0.55	5.13	1,065.22
20	0.7375	23,762	2.05	126,358	244	0.55	5.31	1,083.89
21	0.75	24,373	2.04	132,313	246	0.55	5.42	1,097.64
22	0.7625	24,469	2.04	133,038	253	0.55	5.43	1,099.43
23	0.775	42,753	2.11	388,712	443	0.55	9.08	1,545.31
24	0.7875	42,931	2.11	391,859	447	0.55	9.12	1,550.05
25	0.8	43,225	2.12	397,557	454	0.55	9.19	1,557.57
26	0.8125	43,800	2.12	409,098	454	0.55	9.33	1,571.38
27	0.825	43,881	2.12	410,204	454	0.55	9.34	1,572.74
28	0.8375	44,093	2.13	415,343	462	0.55	9.41	1,577.51
29	0.85	44,757	2.13	428,362	471	0.55	9.56	1,589.90
30	0.8625	44,771	2.13	428,512	471	0.55	9.56	1,590.03
31	0.875	44,991	2.14	434,266	479	0.55	9.64	1,594.14
32	0.8875	45,865	2.15	453,269	488	0.55	9.87	1,606.25
33	0.9	45,883	2.15	453,688	488	0.55	9.88	1,606.42

Whittle Pit Number	Recovery Factor	ROM		Waste	Marginal Material		Strip Ratio	Undiscounted Cash Flow
		kt	g/t Au	kt	kt	g/t Au		US$ Million
34	0.9125	46,823	2.15	474,102	511	0.55	10.11	1,616.18
35	0.925	46,828	2.15	474,208	511	0.55	10.12	1,616.21
36	0.9375	47,533	2.16	492,102	520	0.55	10.34	1,621.96
37	0.95	47,655	2.16	494,036	527	0.55	10.36	1,622.56
38	0.9625	48,901	2.16	521,813	566	0.55	10.66	1,628.82
39	0.975	49,816	2.17	545,217	584	0.55	10.93	1,632.32
40	0.9875	49,828	2.17	545,358	584	0.55	10.93	1,632.28
41	1	49,881	2.17	546,121	584	0.55	10.94	1,632.36
42	1.0125	50,673	2.18	567,863	588	0.55	11.19	1,632.13
43	1.025	50,873	2.18	572,134	590	0.55	11.23	1,631.88
44	1.0375	50,875	2.18	572,156	590	0.55	11.23	1,631.87
45	1.05	51,614	2.19	593,251	600	0.55	11.48	1,628.58
46	1.0625	51,650	2.19	593,827	600	0.55	11.49	1,628.46
47	1.075	51,704	2.19	595,266	600	0.55	11.50	1,628.09
48	1.0875	52,335	2.19	612,131	608	0.55	11.68	1,622.72
49	1.1	52,335	2.19	612,139	608	0.55	11.68	1,622.72
50	1.1125	52,358	2.19	612,527	608	0.55	11.69	1,622.50
51	1.125	53,154	2.19	636,286	615	0.55	11.96	1,612.35
52	1.1375	53,155	2.19	636,347	615	0.55	11.96	1,612.32
53	1.15	53,687	2.20	651,064	621	0.55	12.12	1,604.35
54	1.1625	53,782	2.20	654,500	621	0.55	12.16	1,602.48
55	1.175	53,948	2.20	661,133	622	0.55	12.24	1,598.39
56	1.1875	53,950	2.20	661,187	622	0.55	12.24	1,598.34
57	1.2	54,378	2.20	675,098	626	0.55	12.40	1,588.44
58	1.2125	54,565	2.20	682,355	626	0.55	12.49	1,583.15
59	1.225	54,732	2.20	686,320	632	0.55	12.53	1,579.91
60	1.2375	54,736	2.20	686,359	632	0.55	12.53	1,579.87
61	1.25	56,041	2.21	735,308	637	0.55	13.11	1,539.53
62	1.275	56,062	2.21	735,802	637	0.55	13.11	1,539.02
63	1.2875	56,678	2.22	759,540	641	0.55	13.39	1,518.16
64	1.3	56,838	2.22	763,464	643	0.55	13.42	1,514.20
65	1.3125	57,178	2.22	774,086	657	0.55	13.53	1,503.94
66	1.325	57,263	2.22	777,808	657	0.55	13.57	1,500.46
67	1.3375	57,701	2.23	796,743	659	0.55	13.80	1,482.06
68	1.35	57,813	2.23	801,489	661	0.55	13.85	1,477.11
69	1.3625	58,748	2.23	838,831	672	0.55	14.27	1,437.34
70	1.375	59,356	2.24	863,308	679	0.55	14.53	1,409.85
71	1.3875	59,365	2.24	863,512	679	0.55	14.53	1,409.57
72	1.4	59,476	2.24	868,514	683	0.55	14.59	1,403.79
73	1.4125	59,489	2.24	868,875	683	0.55	14.59	1,403.30
74	1.425	59,608	2.24	874,136	685	0.55	14.65	1,396.74

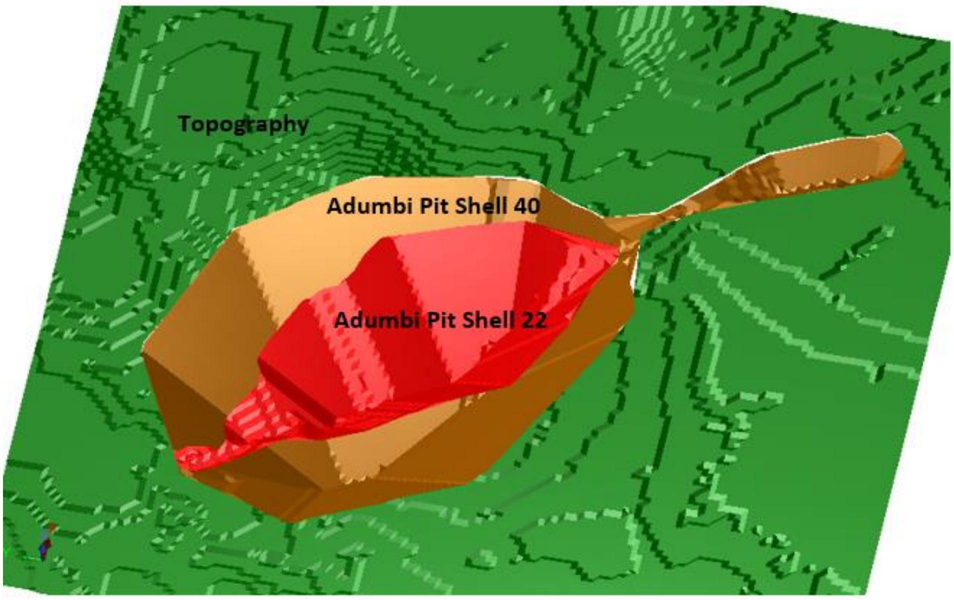


Source: Minecon (2021)

Figure 16.3: Adumbi Whittle Pit Optimisation Results

16.6 SELECTION OF OPTIMISED PIT SHELL

The selection of the optimised final pit shell was based on the maximum undiscounted cash flow. In this instance, Pit Shell 40 was selected for the practical pit design as shown in Figure 16.4.



Source: Minecon (2021)

Figure 16.4: Optimised Pit Shell Selected for Practical Design

16.7 MINERAL INVENTORY

The Mineral Inventory Estimate has been managed by Daniel Bansah, who is a qualified person. Mr Bansah is a graduate with a Master of Science with Distinction in Mineral Exploration gained from the University of Leicester in 1999. Mr Bansah is a Fellow of the West African Institute of Mining, Metallurgy and Petroleum (WAIMM), a Chartered Professional Member of AusIMM, and a member of the Ghana Institute of Geoscientists.

The Mineral Inventory Statement uses the definitions and guidelines given in the CIM Definition Standards on Mineral Resources and Mineral Reserves and is reported in accordance with NI 43-101 requirements.

Table 16.7shows a summary of the Adumbi Mineral Inventory for the various material types (oxide, transition and fresh) contained within the practical pit designs.

Table 16.7: Summary of Adumbi Mineral Inventory (Effective Date: November 17, 2021)

Material Type	Indicated Mineral Resource			Inferred Mineral Resource		
	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)	Tonnage (t)	Grade (g/t Au)	Contained Gold (oz)
Oxide	3,169,000	2.05	208,000	458,000	3.39	49,000
Transition	3,401,000	2.51	274,000	280,000	2.74	24,000
Fresh (Sulphide)	21,614,000	2.02	1,400,000	20,089,000	2.64	1,703,000
TOTAL	**28,185,000**	**2.08**	**1,883,000**	**20,828,000**	**2.65**	**1,777,000**

NOTES:
1. The CIM definitions were followed for Mineral Resources.
2. Mineral resources were estimated at a block cut-off grade of 0.52 g/t Au for oxide, 0.57 g/t Au for transition and 0.63 g/t Au for fresh material constrained by a Whittle pit.
3. Mineral Resources for Adumbi were estimated using a long-term gold price of US$1,600/oz.
4. A minimum mining width of 32 m horizontal was used.
5. A maximum of 4 m internal waste was used.
6. Adumbi bulk densities of 2.45 for oxide, 2.82 for transition and 3.05 for fresh rock were used.
7. High gold assays were capped at 18 g/t Au for Adumbi, prior to compositing at 2 m intervals.
8. Numbers might not add up due to rounding.

16.8 PRACTICAL PIT DESIGNS

The practical pit designs were prepared using the optimised pit shells as templates. The relevant Whittle pit shells were exported from the GEMS to Surpac software, where the practical pit designs were prepared. The practical pit design incorporates the ramps together with the appropriate inter-ramp slope angles. No practical pit design was prepared for the Final Pit; hence, the optimised pit shell (Pit 40) was used to define Cut 3 for the blocks to be scheduled.

The open-pit design criteria were as follows:

- A nominal bench height of 8 m

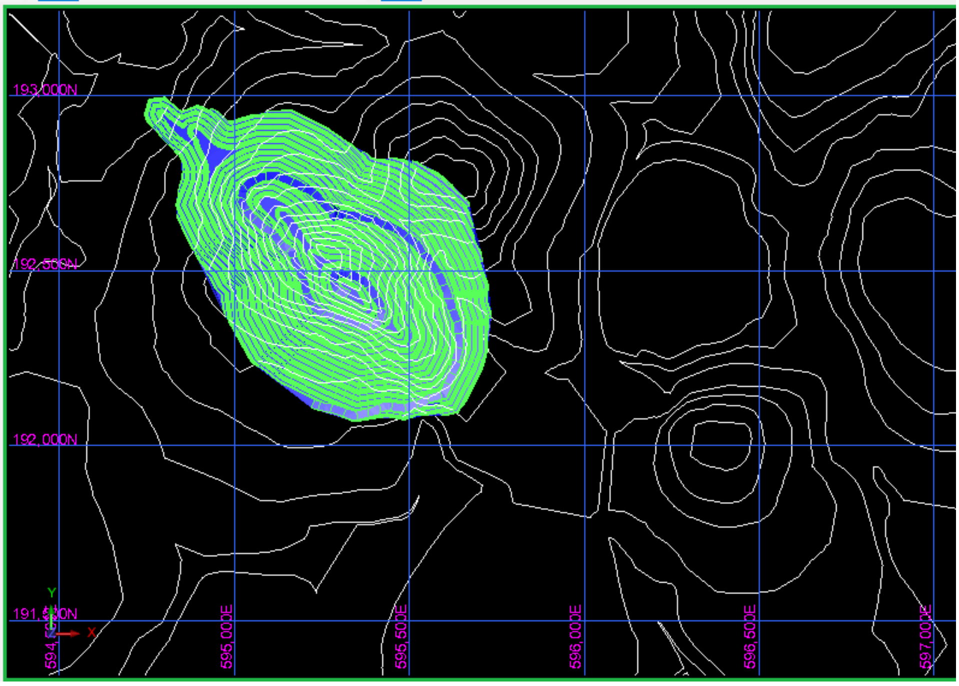
- The inclusion of the ramping systems, resulting in the recommended overall slope angles of approximately 40° to 45° being achieved
- Haul road widths of 25 m, including safety berms, providing sufficient room for two-way traffic for the planned 140 t capacity truck fleet
- Haul road grades at a 10 % continuous gradient

A total of two pits (Cut 1 and Cut 2) have been designed as follows:

- Cut 1 – this pit exploits the orebody using Whittle Pit 22 as a template. A plan view is shown in Figure 16.5.
- Cut 2 – this pit exploits the Adumbi orebody using Whittle Pit 40 as a template, as shown in Figure 16.6.

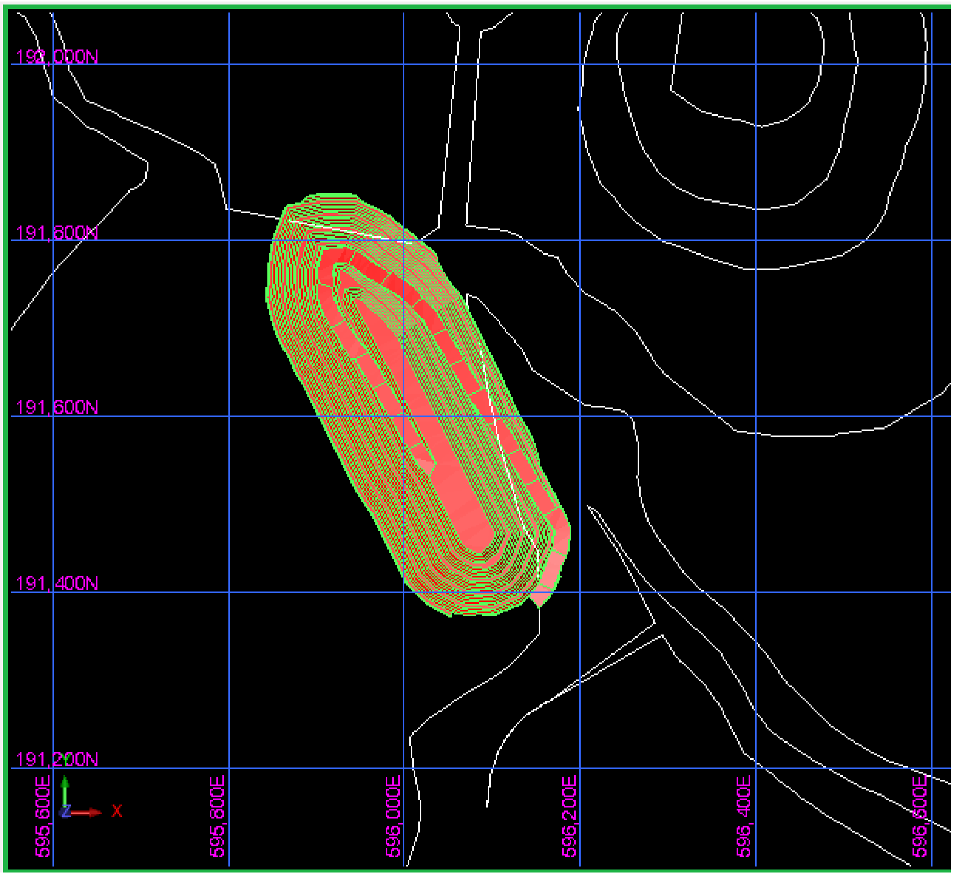
The Cut 3 pit exploits the Adumbi orebody using Whittle Pit 40 without designing, as shown in Figure 16.7. Figure 16.8 shows the combined pits of Adumbi.

Principal haul roads shall be designed and constructed to connect the pit working areas to the primary crusher and the waste dumps.



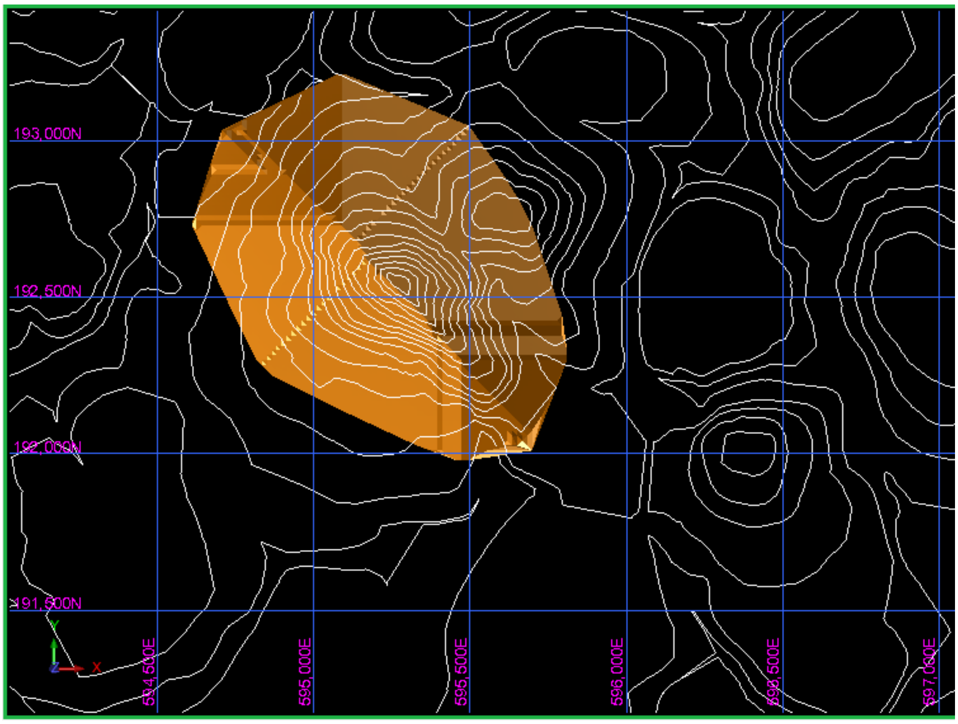
Source: Minecon (2021)

Figure 16.5: Adumbi Cut 1 Pit Design



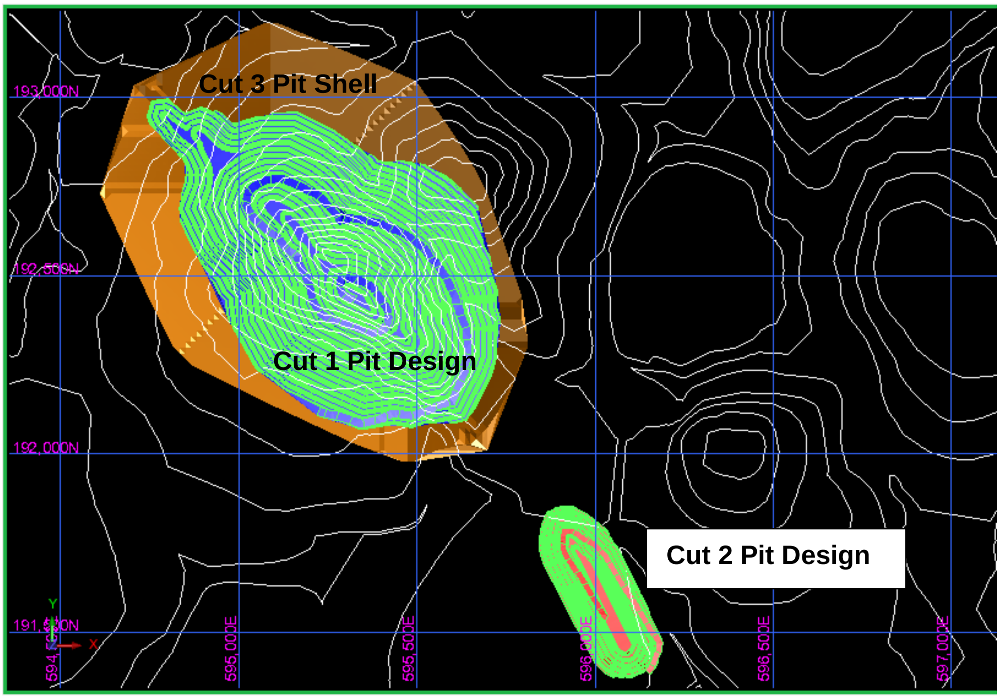
Source: Minecon (2021)

Figure 16.6: Adumbi Cut 2 Pit Design



Source: Minecon (2021)

Figure 16.7: Adumbi Cut 3 Pit Shell



Source: Minecon (2021)

Figure 16.8: Adumbi – All Pits

16.9 LOM PRODUCTION SCHEDULE

The scheduling process consists of developing a mine plan that is economically optimal using the inventory included in the practical pit designs and pit shell. The production schedule was based on a methodology of block selections from the block model inside the pits.

Table 16.8 shows the proposed LOM production schedule.

Table 16.8: LOM Production Schedule

Description	Unit	Year	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Period Number		**TOTAL**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
ROM Tonnes	t	**29,886,695**	3,080,333	1,980,361	2,633,447	2,887,295	4,215,615	3,884,952	2,523,768	2,291,908	4,274,748	2,114,268	-
ROM In-Situ Grade	g/t	**3.06**	3.05	3.45	2.77	2.68	2.76	2.68	2.95	3.14	3.42	4.14	-
ROM Diluted Grade	g/t	**2.9**	2.9	3.28	2.63	2.55	2.62	2.55	2.8	2.98	3.25	3.94	-
ROM Recoverable Grade	g/t	**2.61**	2.59	2.92	2.36	2.29	2.36	2.29	2.52	2.68	2.92	3.54	-
LG Tonnes	t	**19,884,165**	2,281,482	1,489,862	1,824,614	2,944,307	3,338,833	2,557,117	1,822,242	1,545,813	1,588,504	491,390	-
LG In-Situ Grade	g/t	**1.13**	1.07	1.02	1.09	1.18	1.13	1.17	1.14	1.16	1.21	1.11	-
LG Diluted Grade	g/t	**1.07**	1.01	0.96	1.04	1.12	1.07	1.11	1.08	1.1	1.14	1.05	-
LG Recoverable Grade	g/t	**0.96**	0.91	0.86	0.93	1	0.96	1	0.97	0.99	1.03	0.94	-
Waste Tonnes	t	**547,232,488**	67,178,178	82,619,771	85,738,796	74,468,394	62,670,499	49,166,557	41,165,882	39,962,277	37,936,746	6,325,388	-
Total Material (Ore + Waste)	t	**597,003,348**	72,539,994	86,089,994	90,196,857	80,299,996	70,224,947	55,608,626	45,511,892	43,799,998	43,799,998	8,931,047	-
Plant Feed Tonnes	**t**	**49,770,861**	**2,985,000**	**5,018,750**	**5,032,500**	**5,018,750**	**5,018,750**	**5,018,750**	**5,032,500**	**5,018,750**	**5,018,750**	**5,018,750**	**1,589,611**
Feed Grade (Undiluted)	g/t	**2.29**	3.04	2.04	1.96	2.04	2.5	2.34	2.05	2.06	3.09	2.42	1.16
Feed Grade (Diluted)	g/t	**2.17**	2.89	1.94	1.86	1.94	2.37	2.22	1.95	1.96	2.93	2.3	1.1
Feed Grade (Recoverable)	g/t	**1.95**	2.59	1.73	1.67	1.74	2.13	2	1.75	1.76	2.64	2.07	0.99
Recovery	%	**89.79**	**89.44**	**89.38**	**89.56**	**89.82**	**89.92**	**89.91**	**89.91**	**89.9**	**89.92**	**89.92**	**89.91**
Gold Produced	**oz**	**3,121,324**	**248,189**	**279,209**	**270,170**	**280,875**	**344,336**	**321,976**	**283,422**	**283,806**	**425,296**	**333,331**	**50,715**
Strip Ratio	t/t	**11**	12.53	23.81	19.23	12.77	8.3	7.63	9.47	10.41	6.47	2.43	-
Operational Waste	t	**443,675,980**	55,158,163	38,155,236	49,016,548	64,118,684	62,670,499	49,166,557	41,165,882	39,962,277	37,936,746	6,325,388	-
Capitalised Waste	t	**103,556,508**	12,020,015	44,464,534	36,722,248	10,349,710	-	-	-	-	-	-	-

16.10 MINE PLANNING AND PUSHBACK

The mining production schedule sequence is planned to mine the Cut 1 and Cut 2 pit designs from the beginning of Year 1 to get quick access to the ore and manage the strip ratio. Mining of the Cut 3 pit shell starts from the beginning of Year 2.

16.11 MINING EQUIPMENT

The mining equipment in Table 16.9 has been selected based upon the annual mine production schedule, the mine work schedule and equipment shift production estimates. The size and type of mining equipment is consistent with the size of a project for which the annual peak material movement is approximately 5.0 Mt/a (see Table 16.9 and Table 16.10).

Table 16.9: Mining Major Equipment Requirements

Major Equipment Required	Year 1	Year 2 To LOM
Caterpillar 6020 Excavator	6	7
Caterpillar 785 Dump Truck	34	40
Pantera 1500 Drill Rig	10	12
Caterpillar D9N Track Dozer	4	5
Caterpillar 824 Wheel Dozer	1	1

Table 16.10: Mining Minor Equipment Requirements

Ancillary Equipment Required	Year 1	Year 2 To LOM
Caterpillar 16H Grader	4	4
Caterpillar 773 Water Bowser	2	3
Hitachi 670 Excavator	2	3
Lighting Plant	10	12
Fuel Truck	2	3
Service Truck	2	3
Grade Control Drill Rig	2	2

The equipment units and costs have been selected from world-renowned manufacturers. Although quotes have not been specifically requested for this project, the capital and transportation costs have been derived from similar projects from the relevant equipment suppliers. Outputs have been derived from generic calculations.

16.12 MINING MANPOWER

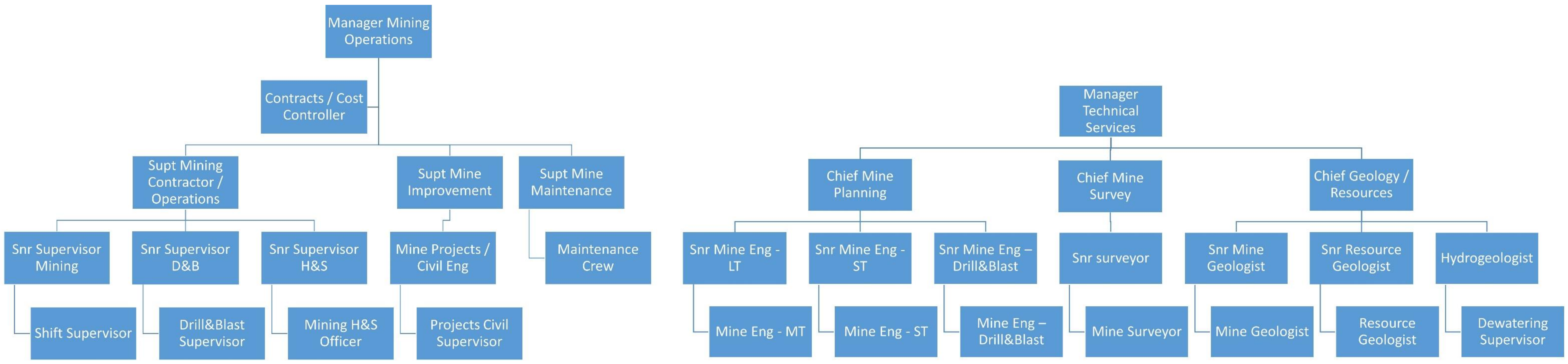
The mining personnel include all the salaried staff working in the mine operations and mining engineering departments, and the operational labour required to operate drilling, blasting, loading, hauling and mine support activities.

Minecon has generated manpower and staffing requirements from first principles and with knowledge of similar operations in remote conditions. These requirements have been based on the mining operations being carried out by contract mining as opposed to owner mining (see Table 16.11).

Table 16.11: Mining Manpower Summary

Year	Equipment Operator (EOP)	EOP + Leave + Sick Leave	Local					Total
			Engineering		Mining	Management (Expatriates and Nationals)		
			Artisans	Supervisors	Supervisors	Engineering	Mining	
Year 1	201	221	9	2	3	1	1	237
Year 2	243	267	11	3	4	1	1	287
Year 3	243	267	11	3	4	1	1	287
Year 4	216	238	9	2	3	1	1	254
Year 5	192	211	9	2	3	1	1	227
Year 6	159	175	9	2	3	1	1	191
Year 7	129	142	9	2	3	1	1	158
Year 8	120	132	9	2	3	1	1	148
Year 9	120	132	9	2	3	1	1	148
Year 10	84	92	9	2	3	0	1	107
Year 11	84	92	9	2	3	0	1	107

The proposed organisational structure for the mining operation is detailed in the organisational chart given in Figure 16.9.



Source: Minecon (2021)

Figure 16.9: Organisational Chart for Mining

16.13 CAPEX AND OPEX FOR MINING

Minecon has estimated the pre-strip CAPEX for mining (including a 20 % OPEX for contingency) to be US$59.99 million.

The OPEX over the LOM is US$1,559.2 million and a deferred stripping amount of US$328.2 million, which equates to US$3.29 per total tonnes mined.

16.14 INTERPRETATION AND CONCLUSIONS

The Adumbi PEA was completed on December 15, 2021, by SENET and Minecon, and the resulting study was summarised in the Loncor PEA Press Release of December 15, 2021. An economic model and financial analysis were also undertaken.

The Whittle pit optimisation and design results demonstrate a robust economic outlook, and with further exploration work, the Adumbi deposit can be advanced up the development curve.

The Adumbi deposit is planned to be exploited at an average mining rate of 7.5 Mt per month to provide a throughput of 5.0 Mt/a of ore to the processing plant. The Adumbi deposit has an average stripping ratio of 1:11, which is an important contributing factor to the high operating costs.

16.15 RECOMMENDATIONS

Further related works and additional studies in the following areas are necessary in order to optimise the PEA study to a PFS level:

- Geotechnical and hydrogeological investigations need to be incorporated into the slope stability analysis to refine the geotechnical inputs into the pit optimisation process and mine design studies. This is also required for the stability of the waste dumps.
- The principal haul roads design needs to be refined to improve the cycle time to the ROM pad and waste dump.
- An investigation of the acid rock drainage potential of the stockpile and the waste dumps needs to be undertaken.
- Further cost investigations need to be undertaken with regard to contractor mining.

17 RECOVERY METHODS

17.1 DESIGN PHILOSOPHY

The proposed process plant design for the Imbo Gold Project is based on a metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is constructed from unit operations that are well known, proven and established in industry.

The Imbo plant will process a range of ore types (oxide, transition and sulphide) with variable ore characteristics, gold grades and metallurgical treatment requirements. The sulphide ores are significantly more competent than the oxide ores.

Oxide will be processed using single-stage crushing and a combination of a primary semi-autogenous grinding (SAG) mill and a secondary ball mill circuit. Transition and sulphide ores will also be processed in a primary SAG mill and a secondary ball mill circuit in combination with a pebble crusher.

The key project and ore-specific design criteria that the plant design must meet are as follows:

- 5 Mt/a of ore throughput
- Mechanical availability of 65 % for the crushing plant
- Mechanical availability of 91.3 % for the remainder of the plant, supported by crushed ore storage and standby equipment in critical areas
- Sufficient automated plant control to minimise the need for continuous operator intervention and allow manual override and control when required

A process design criteria document has been prepared incorporating the engineering and key metallurgical design criteria derived from the results of metallurgical test work and comminution circuit modelling. The design document forms the basis for the design of the processing plant and required site services.

The selected milling circuit has a surplus of available grinding power when treating oxide, and it is planned to treat sulphide ore at the same design rate of 5 Mt/a.

17.2 SELECTED PROCESS FLOWSHEET

The Imbo process plant has been designed to process 5 Mt/a.

The process plant was developed from the interpretation of the results of the PEA test work programmes (described in Section 13).

The interpretation of these metallurgical test work results and simulations conducted by Orway Mineral Consultants (OMC) led to the development of the process flowsheet, which includes the following:

- Single-stage crushing facility, utilising a primary crusher to produce a crushed product size suitable for SAG milling
- A crushed ore surge bin which, when full, allows crushed ore to be diverted to an emergency stockpile from which ore can be reclaimed by front-end loader (FEL) to feed the mill during the periods when the crushing circuit is offline

- Two-stage SAG and ball mill combination, operating in closed circuit with the cyclone cluster to produce the required size
- Additional pebble crushing system to crush pebbles from the SAG mill when treating transition and sulphide ores, i.e. the SABC circuit
- Gravity concentration and removal of coarse gold from the milling circuit recirculating load and the treatment of gravity concentrate by intensive cyanidation and electrowinning to recover gold to bullion
- CIL circuit incorporating 1 pre-leach tank and 11 CIL tanks containing carbon for gold adsorption
- INCO air/SO_2 cyanide detoxification, arsenic precipitation and pumping of the detoxified tailings to the dam
- TSF and the return water pumping system
- 12 t acid wash, pressure Zadra elution circuit, electrowinning, carbon regeneration and gold smelting to recover gold from the loaded carbon to produce bullion
- Consumables and reagents
- Air and water services

A simplified flow diagram depicting the unit operations incorporated in the selected process flowsheet is shown in Figure 17.1.

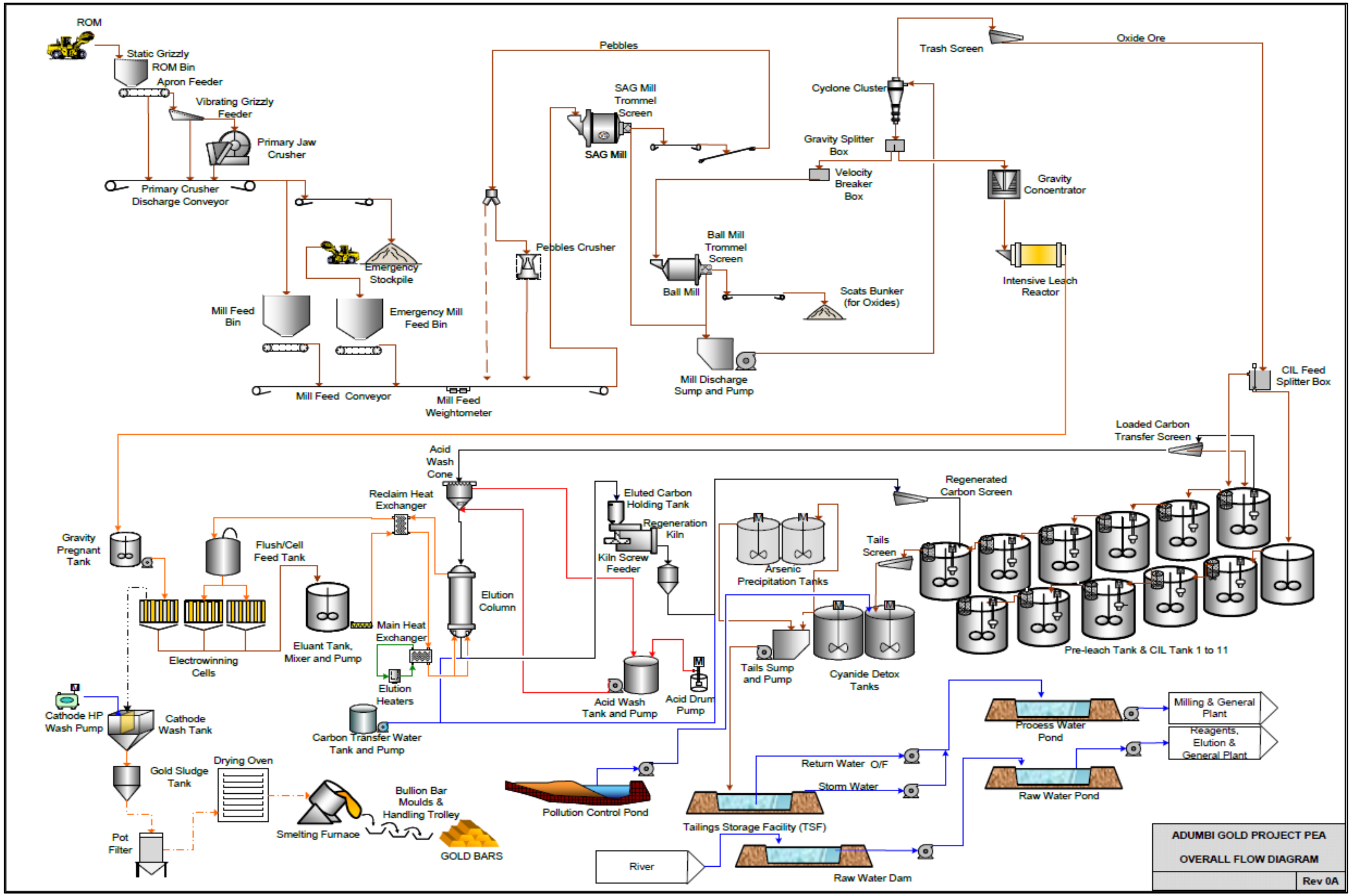


Figure 17.1: Plant Overview Process Flow Diagram

17.3 PLANT DESIGN BASIS

The key issues considered in the process and equipment selection are discussed below.

17.3.1 Process Plant

The plant design has been based on a nominal capacity of 5 Mt/a for all ore types (oxide, transition and sulphide). The plant design is based on the initial separate treatment of the oxide ore, followed by the introduction of a pebble crushing facility into the comminution circuit as the transition and sulphide ores are introduced into the plant feed.

17.3.2 ROM Pad

The ROM pad will be used to provide a buffer between the mine and the plant. The ROM stockpile will allow blending of feed stocks and ensure a consistent feed type and rate to the plant.

17.3.3 Comminution Circuit Selection

OMC was requested to conduct simulations based on the comminution test work results. The results are summarised in Table 17.1 (see Section 13 for the detailed results). The simulations were conducted at 625 t/h to give a product of P_{80} passing 75 µm.

Table 17.1: Comminution Test Work Data

Parameter	Unit	Oxide	Transition	Sulphide
Ai	g	0.1878	0.2519	0.3409
CWi	kWh/t	–	–	15
BRWi	kWh/t	12.7	12.7	14.6
BBWi	kWh/t	11.8	13.7	14.2
SG		2.85	3.07	3.12

The selected comminution circuit will consist of single-stage primary crushing, followed by the two-stage SAG and ball mill circuit when processing the oxide ore, i.e. SAB circuit. The pebble crusher will be added to the circuit to treat pebbles generated by the SAG mill when treating transition and sulphide ores, i.e. SABC circuit. The mill discharge will be pumped to the cyclone, and the cyclone underflow will be recycled back to the ball mill while the overflow will gravitate to the CIL circuit.

Table 17.2 summarises the sizing of the equipment suitable for the selected comminution circuit.

Table 17.2: Simulated Comminution Circuit

Parameter	Unit	Oxide	Transition	Sulphide
Primary Crusher				
Type		Jaw Crusher or Equivalent		
Model		C160		
Installed Power	kW	250		
Pebble Crusher				
Type			Cone Crusher	
Model			HP200 or Equivalent	
Installed Power	kW		132	
SAG Mill				
Diameter	m	9.15		
Effective Grinding Length	m	4.35		
Discharge Arrangement		Grate		
Installed Power	kW	7,400		
Ball Mill				
Diameter	m	6.70		
Effective Grinding Length	m	11.00		
Discharge Arrangement		Overflow		
Installed Power	kW	9,000		
Milling Circuit Classification				
Type		Cluster		
Model		400 CVD10		
Diameter	mm	400		
Number of Installed Cyclones		18		

17.4 KEY PROCESS DESIGN CRITERIA

Based on the above test work results, the Imbo process plant design was configured utilising well-known, proven and established gravity and CIL technologies to recover gold from blends of oxide, transition and sulphide ores that will be processed at a rate of 5 Mt/a.

The key process design criteria used for the plant design are summarised in Table 17.3.

Table 17.3: Key Process Design Criteria Used for Plant Design

Item	Unit	Oxide	Transition	Sulphide
Plant Throughput	Mt/a	5.0	5.0	5.0
Gold Head Grade	g/t Au	2.25	3.2	4.00
Design Gold Recovery	%	91.82	90.38	80.1 to 89.83
Crushing Plant Utilisation	%	65.0	65.0	65.0
Plant Availability	%	91.32	91.32	91.32
ROM Ore Top Size	mm	1,000	1,000	1,000

Item	Unit	Oxide	Transition	Sulphide
CWi	kWh/t	–	–	15
BBWi	kWh/t	11.8	13.7	14.2
A × b		130.0	88.6	47.2
SG		2.85	3.07	3.12
Ai		0.19	0.25	0.34
Comminution Circuit		Primary Crushing and SAB	Primary Crushing and SABC	Primary Crushing and SABC
Crush Size, P_{80}	mm	180	180	180
Grind Size, P_{80}	µm	75	75	75
Mill Pinion Power	kW	5,112	4,075	3,670
Grinding Media Consumption	kg/t	0.780	0.547	0.503
Gravity Gold Recovery	%	36.8	31.6	26.6
Leach/CIL Residence Time	h	24	24	24
Leach Slurry Density	% w/w	40	40	50
Number of Pre-Leach Tanks		1	1	1
Number of CIL Tanks/Stages		11	11	11
Cyanide Consumption	kg/t	0.99	1.32	1.31
Lime Consumption	kg/t	3.64	5.40	3.61
Elution Circuit Type		Pressure Zadra	Pressure Zadra	Pressure Zadra
Elution Circuit Size	t	12	12	12
Frequency of Elution (Number of Strips per Week)		7	7	7

17.5 PROCESS DESCRIPTION

The Imbo process plant design utilises well-known, proven and established gravity and CIL technologies to recover gold from oxide, transition and sulphide ores that will be processed at a rate of 5 Mt/a. These technologies will be utilised in combination with the adsorption-desorption-recovery (ADR) process for the recovery of gold from active carbon.

17.5.1 Primary Crushing and Crushed Ore Storage

The ROM ore will be transported to the plant by dump trucks, which will tip into the ROM bin. The ROM bin (with a capacity of at least one and a half times the truck size) will be equipped with a static grizzly to ensure that the oversize material does not report to the primary crusher. The crushing circuit will consist of the single-stage jaw crusher.

An apron feeder located under the ROM bin will be used to withdraw the ore from the bin at a controlled rate and will discharge onto a vibrating grizzly feeder to scalp off the fines while the oversize from the feeder reports to the crusher. The feeder undersize (fines) will discharge onto the sacrificial conveyor. The jaw crusher will reduce the oversize material to the required product size and discharge onto the sacrificial conveyor. The feed to the crushing circuit is measured using the crushing feed weightometer located on the bin feed conveyor.

The scalped feeder undersize and the jaw crusher product will be combined and will be transferred from the sacrificial conveyor onto the bin feed conveyor, where it will be conveyed to the mill feed bin. The mill feed bin ensures a constant feed to the milling circuit. When the mill feed bin is full or if maintenance work is being carried out on the bin, a provision has been made for the crushed ore to be diverted to an emergency stockpile from where it can be reclaimed and fed to the milling circuit via an emergency feed facility.

The crushed ore will be withdrawn from the mill feed bin at a controlled rate using the apron feeder onto the mill feed conveyor feeding the SAG mill. An FEL will be used to reclaim the ore from the emergency stockpile onto the mill feed conveyor via an emergency feed bin, thus ensuring that the feed to the milling circuit is not disrupted during maintenance of the crushing system.

The feed to the mill is measured using the mill feed weightometer located on the mill feed conveyor.

The mill feed weightometer will control the feed to the mill by varying the speed of the mill feed bin apron feeder. The mill feed conveyor weightometer will also be used for metallurgical accounting purposes.

Conveyor skirting and dust enclosures, together with dust suppression systems, are included in the design as a means of containing the dust produced by the crushing circuit. The dust suppression system uses fine water sprays, which will be used to suppress dust at the main dust-generating points.

An electric hoist will be located above the jaw crusher and will be used to remove any boulders from the crusher in the event of a choke and will also be useful for minor maintenance. Other electric hoists are included in the crushing section to facilitate maintenance.

17.5.2 Pebble Crushing

The SAG mill scalped oversized scats will discharge onto the scats discharge conveyor. The scats will then be conveyed into the pebble crusher via the pebble crusher feed bin. A weightometer will measure the amount of scats for process control.

An overband electromagnet on the scats discharge conveyor will remove the steel balls or steel discharged together with the scats ahead of the pebble crusher. In addition, a metal detector will be installed, which will enable bypass of the pebble crusher in the event that metal is detected in the feed to the pebble crusher, which would otherwise damage the crusher’s lining. The pebble crusher product will be discharged onto the SAG mill feed conveyor and fed back to the mill.

17.5.3 Milling

The milling circuit will consist of the primary SAG mill, secondary ball mill and a pebble crusher to reduce the size of the pebbles generated by the SAG mill.

The SAG mill will be equipped with a variable speed drive, which allows the power input to the mill to be varied when processing ore requiring different milling energy inputs and for the protection of shell liners in the event of low-load conditions and worn-out liners.

Process water is added at the inlet of the SAG mill in order to control the mill discharge slurry density. The mill feed dilution water is ratio-controlled using the ore mill feed weightometer located on the mill feed conveyor and the flowmeter on the process water line. The discharge product from the SAG mill will be passed through the slotted apertures of a trommel screen to scalp off pebbles which would otherwise damage the mill discharge pumps. Part of the process water is also added as high-pressure spray water on the mill trommel screen to ensure efficient wet screening.

The trommel screen undersize from both the SAG and ball mills will gravitate to the mill discharge sump where sump dilution water will be added at a controlled rate to achieve the required solids feed density to the cyclone cluster. The diluted slurry is pumped to the cyclone cluster using the variable speed cyclone feed pumps.

The underflow from the cyclone cluster will gravitate back to the ball mill through the splitter box with an option of gravitating back a portion to the SAG mill. A portion of the cyclone underflow is directed to feed the gravity recovery circuit.

The cyclone overflow slurry will gravitate directly to the trash screen. The trash screen removes oversize material (such as misplaced oversize particles, vegetal debris, plastic fragments, blast fuses, and wires) from the cyclone overflow stream.

Milling circuit spillage will be contained in a bunded area with a sloped-end floor to direct spillage to the mill feed and mill discharge end spillage sumps. Each sump will have a vertical spindle pump that will pump the spillage into the mill discharge sump.

A safety shower is provided in this area. It is activated by a foot pedal and is equipped with an eye bath.

Grinding media (steel balls) are added to the SAG mill using the ball addition hopper that discharges onto the mill feed conveyor. The drums of steel balls are opened, and the balls are emptied into the hopper. Then the balls roll onto the mill feed conveyor.

Steel balls will also be added to the ball mill using the ball loading hopper kibble. The kibble is lifted by the crane (hoist) into the grinding media feed chute. Steel balls will be discharged into the mill by lowering the ball loading hopper to its rest position located directly above the mill feed chute(s).

17.5.4 Gravity and Intensive Leach Reactor (ILR)

Due to the presence of "free gold" in the ore, the removal of coarse gold prior to the CIL circuit reduces the possibility of gold lock-up and improves the leach kinetics of the ore. A portion of the cyclone underflow slurry will be bled off to the gravity concentrator feed vibrating screen to scalp off the oversize material. The screen oversize material will gravitate to the mill discharge sump while the screen undersize will gravitate to the gravity concentrator to recover any free gold.

Process water is also added as high-pressure spray water to ensure efficient wet screening and to dilute the slurry to the required solids concentration prior to feeding the concentrator. Tails from the gravity concentrator will gravitate to the mill discharge sump.

The centrifugal gravity concentrator operates in a batch mode on a set operating cycle. The slurry is fed into the concentrator for a pre-set time determined by the on-site optimisation. After the unit has undergone the flush cycle, it discharges the batch of concentrate into the concentrate holding tank. The length of the concentrating cycle can be adjusted to suit the ore being treated by using the machine countdown timer. The gravity concentrator subjects the ore particles to a centrifugal force, and clean raw water is injected into the rotating concentrate cone to keep the concentrating bed fluidised. Constant fluidisation of the concentrating bed results in efficient classification of the heavier gravity gold from the lighter gangue material.

The gravity concentrate recovered will be stored in the leach reactor holding tank and treated using the intensive cyanidation process. Once a full concentrate batch is collected and excess water removed through an overflow pot, a fully automated intensive leach process is initiated. At the beginning of the batch leach cycle, the entire contents of the feed cone will be discharged into the reactor drum, and excess water from the drum during the loading cycle will overflow into the ILR sump.

The level in the storage tank will be adjusted by adding raw water, caustic, and cyanide solution before commencing the leaching step. Leaching will be achieved by recirculating the cyanide solution through the rotating reactor drum. The overflow will gravitate to the sump and be pumped back to the solution storage tank. Hydrogen peroxide or oxygen will be added to the ILR sump to provide oxygen for the leaching step.

At the end of the leach cycle, the drum will be stopped, and the solution in the drum will be allowed to drain to the ILR sump. Thereafter, it will be pumped to the solution storage tank where it will be clarified by adding flocculant. The clarified solution will be pumped to the gravity pregnant solution tank, and wash water will be added to the drum to wash the entrained solution from the solids before being pumped to the CIL circuit.

The leached and washed solids will be emptied into the ILR sump by running the reactor drum in reverse, and from there, they will be pumped to the mill discharge sump.

Spillage in the gravity area is contained in a bunded area and is pumped, using the area spillage pump, to the intensive cyanidation system or mill discharge sump.

A safety shower is provided in this area. It is activated by a foot pedal and is equipped with an eye bath.

A hoist is provided for maintenance of the concentrator.

17.5.5 Trash Removal

A vibrating screen will be used to remove trash material from the cyclone overflow slurry. The trash screen undersize slurry will gravitate directly to the CIL circuit via a splitter feed box and slurry sampler. Oversize from the trash screen will be collected in a trash bucket.

17.5.6 CIL

The CIL circuit feed box receives the feed slurry from the trash screen. A CIL feed sampler is installed in the CIL feed slurry stream and will cut samples at set time intervals to collect a shift composite sample for metallurgical accounting purposes.

The slurry gravitates from the CIL feed box to either the pre-leach tank or the first CIL tank when the pre-leach tank is bypassed. The slurry gravitates, via interconnecting launders, through the cascading train of tanks to the final CIL tank. The CIL slurry tails gravitate from the final CIL tank to the carbon safety screen in the cyanide detoxification section. The CIL section will consists of 12 tanks in series: 1 pre-leach tank, followed by 11 CIL tanks.

The CIL section is designed with the ability to bypass any tank when required. There are two launder outlets from each tank, equipped with launder gates for easy isolation and bypassing. The discharge side of each CIL tank is equipped with an interstage screen to prevent the migration of carbon from one CIL tank to another. The transfer of slurry from one CIL tank to the next is achieved by the pumping action of the internal impeller mechanisms of the interstage screens.

The interstage screens periodically become blocked with the near-size carbon and are lifted from the CIL tanks onto the interstage screen wash bay for periodic cleaning. A spare interstage screen is provided to replace any screen that is removed for cleaning or repairs. A high-pressure, low-volume wash pump is used to clean the blocked interstage screens while in the wash frame. A tower crane located in the CIL area will be used to lift the interstage screens to a washing bay and for general maintenance purposes.

Oxygen gas required for gold leaching will be introduced into the pre-leach tank through sparging into the tank. Further sparging of oxygen will be allowed for through the CIL tanks to maintain the required dissolved oxygen levels.

The cyanide solution will be introduced to the CIL tanks through a ring main using the dedicated cyanide dosing pumps to the leach feed splitter box. An automatic cyanide analyser will be used to ensure the efficient use of cyanide as some of the samples displayed high cyanide consumptions during the laboratory tests. In the event that the determined cyanide concentrations in the tanks are low, a provision has been made to dose cyanide into all the CIL tanks except the last CIL tank.

Lime slurry will be pumped through a ring main from the lime dosing tank to the CIL circuit to maintain the slurry pH between 9.5 and 10.5. Lime is added to the CIL feed splitter box and in the first two CIL tanks if the slurry pH is below the operating set-point.

The loaded carbon will be transferred from CIL Tank 1 to the acid wash cone through the loaded carbon screen. The slurry-bearing loaded carbon will be pumped to the loaded carbon screen using the recessed impeller vertical spindle carbon transfer pump in CIL Tank 1. The interstage carbon transfer pump in CIL Tank 2 will be sized to be able to transfer the loaded carbon from CIL Tank 2 to the loaded carbon screen in the event that CIL Tank 1 is offline.

The loaded carbon screen undersize slurry gravitates back to CIL Tank 1 with the provision to gravitate to CIL Tank 2 when CIL Tank 1 is offline. High-pressure spray water will be added to the loaded carbon screen. Screen oversize, which will be clean loaded carbon, will gravitate to the acid wash cone.

Regenerated or fresh carbon will be added to the last CIL tank, and a provision has been made to add carbon to the second last tank if the last tank is being bypassed. Carbon is moved upstream from the last CIL tank, countercurrent to the leach slurry flow, using the recessed impeller vertical spindle interstage carbon transfer pumps.

Spillage in the CIL circuit will be contained in the CIL bunded area. The bunded area is equipped with two spillage pumps that will pump spillage from either side of the CIL area train back into the CIL tanks. In the event of the CIL bund failing to contain all the spillage, an event pond will be provided to cater for those unusual occasions.

Due to the use of strong sodium cyanide solution in the CIL, a foot-activated safety shower with eye bath will be provided in this area.

17.5.7 Carbon Safety and Detoxification

The CIL tails slurry from the last CIL tank gravitates to the carbon safety screen. The screen recovers the fugitive carbon from the CIL tails slurry before it gravitates to the detoxification tanks. The safety screen oversize discharges into the carbon basket.

Periodically, the carbon basket will be lifted using the CIL tower crane and its contents emptied into the last CIL tank. The carbon will not be introduced back to the CIL circuit if is predominantly fine and platy. If the carbon is suitable to be re-introduced into the CIL circuit, the fine carbon is screened out before the carbon is returned to the last CIL tank.

The undersize slurry from the carbon safety screen will gravitate to the first detoxification tank or the second detoxification tank, if the first detoxification tank is offline.

The detoxification process occurs in two stages. First, the undersize slurry from the tails screen gravitates to the first detoxification tank, where sodium metabisulphite (SMBS) and copper sulphate solution are added, and the process of detoxification is initiated. Then the slurry will overflow into the second detoxification tank, where the process of detoxification will be completed. Low-pressure blower air (for oxygen supply) is fed into the detoxification tank via a sparging ring to ensure maximum dispersion throughout the slurry.

The cyanide species present in the tailings slurry are oxidised to the more stable cyanates by the addition of sulphur dioxide and oxygen according to the following reaction:

$$CN\text{-}WAD + SO_2 + O_2 + H_2O = OCN^- + H_2SO_4$$

Sulphur dioxide is provided by the SMBS solution. Oxygen is provided by the air blown through the slurry in the detoxification reactor. The cyanide detoxification reaction results in the formation of sulphuric acid; therefore, lime slurry is added to the tank to maintain an optimal operating pH range of 8 to 10. The detoxification reaction requires the presence of copper ions acting as a catalyst at concentrations of approximately 50 mg/L. The copper ions are added to the process in the form of copper sulphate solution.

Slurry from the second detoxification tank gravitates to the arsenic precipitation circuit.

Spillage in the detoxification and tailings area is contained within a bunded area that is equipped with a spillage pump that pumps the spillage back to the detoxification tanks.

The plant will be designed with a pollution control pond. Should the detoxification and tailings bunded area overflow, the pollution control pond will be available to contain the excess spillage. Once the detoxification and tailings bunded area has been cleared, the excess spillage in the pollution control pond is pumped to the detoxification and tailings bunded area using the pollution control pond pump.

One safety shower is provided in the cyanide detoxification bund area.

17.5.8 Arsenic Precipitation

The slurry leaving the last detoxification tank overflows into the arsenic precipitation tanks. The slurry still contains arsenic levels that need to be lowered to ~0.1 ppm prior to discharge to the TSF. The tailings slurry is thus subjected to arsenic precipitation using ferrous sulphate and air.

Sulphuric acid is also added to lower the pH to initiate the reaction. Thereafter, the reaction will maintain the required pH. A pH analyser will be used to measure the pH in the arsenic tank and control the sulphuric acid dosing pump flow. An allowance has been made to raise the tails slurry pH above 8 through the addition of caustic in the arsenic precipitation tank if required.

The slurry from the second arsenic precipitation tank gravitates to the final tails tank, from where the final tails are pumped to the TSF using the tailings feed pumps.

A slurry sampler is installed on the final tails line transferring the slurry to the tailings dam. The sampler collects samples at regular intervals. A shift composite sample is collected at the end of each shift for analysis. The sampler is used for metallurgical accounting purposes.

Spillage in the arsenic precipitation area is contained within a bunded area that is equipped with a spillage pump that pumps the spillage back to the detoxification tanks.

One safety shower is provided in the arsenic precipitation bund area.

17.5.9 Tailings Storage and Return Water

The slurry exiting either the last detoxification tank or the last arsenic precipitation tank will gravitate to the final tails’ sump, from where it will be pumped to the tailings disposal and storage facilities. The return water gravitates from the tailings dam to the return water pond via a penstock.

Return water will be pumped back to the process water pond using the return water pump. Storm water is collected separately in the storm water pond and will be pumped to the return water pond when needed during the dry season. An allowance will be made to pump from the storm water directly into the return water line to the plant.

17.5.10 Acid Wash

Loaded carbon is moved from the first CIL tank into the acid wash cone through the loaded carbon screen. The loaded carbon batch is washed with spray water on the loaded carbon screen and is discharged directly into the acid wash cone. Once a batch (12 t of carbon) has been accumulated, the carbon is washed with dilute hydrochloric acid solution to remove the scale that builds up on the carbon in the CIL circuit.

The acid wash process is carried out in four steps: elutriation, acid wash, rinse, and spent acid disposal. At the end of the acid wash process, a batch of loaded carbon is ready for transfer into the elution column for gold stripping.

17.5.10.1 Elutriation

Elutriation is the process of partially fluidising the carbon bed in order to free and remove trapped light contaminants from the loaded carbon. The carbon batch is washed with raw water to remove any light trash such as slimes, plastic and organic fibrous material captured in the bed. Blowing small amounts of air through the acid wash column often assists in lifting trapped trash during the elutriation process. The trash is flushed to the tails screen feed box via a gravity pipeline.

17.5.10.2 Acid Wash

The scale that builds up on the carbon in the CIL circuit, consisting predominantly of calcium carbonates and/or sulphates, is removed by soaking the carbon in dilute hydrochloric acid.

A concentrated hydrochloric acid solution is diluted to approximately 3 % strength, using raw water in the acid wash tank. The dilute acid solution is pumped to the bottom of the acid wash cone and through the loaded carbon bed. The upflow of the dilute acid solution expands the carbon bed and allows for optimum surface area contact between the loaded carbon particles and the solution. An acid wash cycle usually takes between 1 h and 2 h, depending on the scaling severity.

17.5.10.3 Rinse

After the acid circulation has taken place, the carbon bed is rinsed with fresh water. Fresh water is circulated through the acid wash cone to remove any residual acid. The effluent produced from the rinsing process will be neutralised in the acid neutralisation tank and is pumped to the detoxification tails transfer sump.

17.5.10.4 Spent Acid Disposal

A batch of dilute acid solution can only be used to wash a certain number of loaded carbon batches, after which the dilute acid batch normally becomes too contaminated. The contaminated acid wash solution is neutralised with an excess of sodium hydroxide solution in the acid neutralisation tank and is pumped to the detoxification tails transfer tank.

Once the process of acid wash and rinsing has been completed (the pH of the rinse solution is equal to that of the wash water), the acid-washed carbon batch is drained by gravity into the transfer line and hydraulically transferred into the elution column.

A spillage pump is provided in the acid wash area to pump spillage to the acid neutralisation tank.

One safety shower will be provided in the acid wash bund area.

17.5.11 Elution

The elution process will utilise the Pressurised Zadra Elution System.

Gold is stripped from the loaded carbon by circulating a hot caustic cyanide solution, approximately 3 % NaOH and 2 % NaCN, through the column at 125 °C under a pressure of 300 kPa to 350 kPa. The eluate solution from the elution column is passed through the electrowinning cells to recover the gold from the circulating eluate stream, and the electrowinning return solution is directed to the eluant tank before being pumped back through the elution column.

The elution process involves two main stages:

- Stage 1: Heating the eluant solution, elution column and contents to operating temperature
- Stage 2: Circulating the eluate through the electrowinning cells and back to the eluant tank

17.5.11.1 Stage 1

The eluant solution is pumped through the elution column through a system of recovery and primary heat exchangers, and back to the eluant tank. The secondary heat exchanger recovers heat from the solution exiting the elution column. The eluant solution, preheated by the recovery heat exchanger, is passed through the primary heat exchanger where the solution is heated to the required temperature prior to entering the elution column.

Hot oil is circulated through the primary heat exchanger from the elution heater bank to heat the eluant solution to the temperature required for the elution process. The solution (eluate) exits the elution column through the externally mounted (duty and standby) elution strainers. The strainers prevent the carbon from migrating out of the column to the secondary heat exchanger.

The eluant is recycled back to the elution tank during the heating stage. Once the column, its contents and the eluate exiting the column reach an operating temperature of approximately 120 °C to 125 °C, the eluate is directed to the electrowinning cells and Stage 2 of the elution process commences.

17.5.11.2 Stage 2

The eluant solution in the eluate tank is pumped through the column, and the cooled eluate is passed through the electrowinning cells. The electrowinning return electrolyte is circulated back to the eluant tank. This step will typically last approximately 14 h and will only be stopped once the feed solution to electrowinning reaches gold levels of 5 ppm Au or less.

At the end of the elution cycle, the heaters are switched off. Eluant continues to be pumped through the column until the solution exiting the column is approximately 90 °C. The cooled stripped carbon is hydraulically transferred to the carbon regeneration section. The carbon transport water required to transfer the eluted carbon from the elution column to the regeneration circuit is pumped from the carbon transfer tank.

A fresh batch of eluant is reused to strip up to four batches of loaded carbon, after which the level of contamination in the eluant becomes too high for the eluant to be used for the elution process. The spent eluant is drained from the eluant tank into the elution bunded area and is pumped using the spillage pump to the leach feed splitter box. The cyanide available in the spent eluant is utilised in the CIL circuit, and the residual gold values in the eluant are recovered onto the carbon.

In order to reduce the calcium scaling levels in the elution system, an antiscalant system is employed. A small amount of antiscalant is continuously added to the eluant. Scaling issues in the heat exchangers are reduced by regular opening and cleaning (descaling) of these units.

A spillage pump is provided in the elution area to pump spillage to the leach feed splitter.

Two safety showers will be provided in the elution bund area.

17.5.12 Electrowinning

The electrowinning (EW) circuit is made up of two dedicated circuits:

- CIL gold EW circuit comprising three EW cells
- Gravity gold EW circuit comprising one EW cell

17.5.12.1 CIL Electrowinning

The pregnant electrolyte from elution is transferred to the CIL EW cell feed tank. This arrangement allows the de-aeration of the electrolyte to the cells and distributes the solution among the cells. The electrolyte gravitates from the feed tank and is equally distributed to all the sets of electrowinning cells.

The return spent electrolyte from the CIL electrowinning cells will be directed back to the eluate tank. Any excess electrolyte from the cell feed tank will overflow and be combined with the return electrolyte from the cells and gravitate to the gravity pregnant electrolyte tank.

Stainless-steel mesh cathodes are utilised to electrowin gold from the pregnant electrolyte. An electric current is applied across the cell electrodes, and gold is deposited as a fine sludge, loosely adhering to the stainless steel knit mesh contained in the cathode basket. The electrowinning cycle takes place over a determined period in close circuit with elution.

Samples of the electrolyte tails are taken at regular intervals during the electrowinning process. These are analysed for gold, caustic, and cyanide concentrations in solution. Electrowinning is complete once the gold feed tenor to electrowinning reaches the required level of 5 ppm Au or less.

17.5.12.2 Gravity Electrowinning

The pregnant solution from the intensive cyanidation circuit is pumped to the gravity pregnant tank. The pregnant solution is then pumped from the gravity pregnant tank through the gravity electrowinning cell using the gravity EW pregnant pumps. Gold is deposited onto the stainless-steel mesh of the cell cathodes as a weakly bound fine sludge. The return electrolyte from the gravity electrowinning cell overflows back to the gravity pregnant liquor tank. The electrolyte continues to circulate until a final gold feed tenor to electrowinning reaches the required level of 5 ppm Au. When the electrowinning cycle is complete, the barren electrolyte is pumped to the leach feed splitter box.

17.5.12.3 Gold Sludge Handling

Cathodes are periodically lifted from the cells and washed of the deposited gold sludge. The electrowinning circuit is equipped with a hoist to lift the loaded cathodes from the cells onto the cathode wash tank. The cathode wash pump provides high-pressure spray water to remove the gold sludge adhering to the cathode mesh, and the cleaned-out cathodes are recycled. The sludge accumulated on the floors of the electrowinning cells is washed into the gold sludge tank.

The sludge from the cathode wash tank is washed into the gold sludge tank where the sludge is allowed to settle at the bottom of the tank in the cone section. The sludge is then manually tapped off from the gold sludge tank into a bucket. The solution in the gold sludge tank will overflow into the gold trap tank. The remaining solids in the gold trap tank are allowed to settle and are also manually tapped into the bucket.

The gold sludge collected in the buckets is taken to the gold room for drying, where the sludge is placed in trays and taken to the calcining furnace to remove surface moisture prior to smelting. The solution in the gold trap tank is pumped back to the CIL circuit.

A fume extraction system on the electrowinning cells extracts potentially poisonous and explosive gases that evolve during the electrowinning process. A fresh air fan is installed to force air into the gold room to improve ventilation inside the building.

A spillage pump is provided in the electrowinning area to pump spillage to the leach feed splitter box or to the gravity pregnant tank.

Two safety showers will be provided in the electrowinning bund area.

17.5.13 Carbon Regeneration

Eluted carbon is transferred hydraulically from the elution column to the kiln feed hopper at the end of the elution cycle. Excess water and carbon fines drain through strainers fitted at the bottom and overflow sections of the hopper and discharge into the carbon transfer tank. Provision will be made to bypass the regeneration facility so that eluted carbon can be transferred directly to the last CIL tank if the kiln facility is not available.

Carbon is withdrawn from the kiln feed hopper to the regeneration kiln using a variable speed screw feeder. Excess water, which might not have drained through the kiln feed hopper strainers, will be drained through the wedge wire screen mounted on the screw feeder. The screw feeder moves carbon from the feed hopper into the kiln at a constant rate (as set on the vendor control panel). Upon entering the kiln, the wet carbon, which still contains approximately 50 % moisture, will be dried and heated to the required temperatures that promote carbon regeneration.

The reactivated carbon exiting the kiln is immediately quenched with water in the quench pan to prevent oxidation reactions with atmospheric oxygen after exiting the kiln. The quenched carbon is passed over the reactivated carbon screen, where spray water is applied to help remove fines that would have been generated during transportation and the regeneration process. The regenerated carbon is hydraulically transferred from the reactivated carbon screen to the last CIL tank. The fines and spray water from the quench screen gravitate to the carbon fines transfer sump prior to being pumped onto the carbon safety screen.

New batches of activated carbon are discharged into the carbon attritioning tank, where the carbon is mixed, wetted and attritioned using the carbon attritioning mixer. From the attritioning tank, the carbon gravitates to the quench screen for the removal of fine carbon.

A spillage pump is provided in the carbon regeneration area to pump spillage to the leach feed splitter box.

17.5.14 Gold Room

Gold sludge from the electrowinning section will be placed onto the calcining trays, and the trays will be loaded into the electric calcining oven operated at approximately 800 °C for drying.

The dry cake exiting the cathode sludge drying oven is allowed to cool and is mixed with smelting fluxes in the specified ratios. The fluxed and dried gold sludge is then loaded into the smelting crucible, which is fitted to the smelting furnace.

The smelting furnace operates at temperatures between 1,200 °C and 1,400 °C. The furnace can be hydraulically tilted to pour the molten gold. Once the smelt is completed, the furnace firing system is switched off, and the molten gold is poured into bullion moulds mounted on a cascade trolley. Slag overflow from the last cascade will be collected in a slag granulation launder.

Within the moulds, the heavier metallic phase sinks to the bottom, and the lighter slag phase floats to the top. Upon cooling and solidification, the glassy slag phase is easily broken off the remaining gold bullion bar. The bullion bar is then cleaned further by chipping off and wire-brushing the remaining adhering slag.

The slag granulation plant comprises three sections: granulation, dewatering and storage. The slag will gravitate to the slag sump for the granulation step. In the granulation section, the liquid slag is quenched by pouring onto it an excess water stream. After granulation, dewatering is required to lower the moisture content to approximately 15 %. The slag granules will be dewatered and discharged into two slag collection bins, from where they will be bagged and stored.

The cleaned bullion bar is sampled using a prill drill. Samples are drilled out from two opposing long faces of the bar. The bar is then labelled, weighed and stored in a safe prior to dispatch to the refinery.

The gold room is equipped with two scales:

- Bullion scale, which measures the weight of the bullion and bullion samples
- Flux scale, which measures the weight of the flux

Fresh air is introduced into the gold room by means of a ventilation system, which includes the smelt house fresh air fan and the gold room extraction fan.

Spillage from the gold room gravitates to a central drainpipe and discharges into the bunded area below the electrowinning cells. The collective spillage inside the gold room is pumped to the gold trap tank, with the option of being pumped to the CIL circuit.

17.5.15 Reagents

17.5.15.1 Cyanide

Sodium cyanide is delivered to site in bulk bags packed onto wooden crates. The wooden crates are transported from the cyanide storage area to the cyanide make-up area using a forklift. The reagent hoist is used to lift the cyanide bags individually onto the bag breaker, which is fitted onto the cyanide make-up tank.

Prior to the addition of cyanide briquettes, the pH of the water in the make-up tank will be adjusted to approximately pH 10, using caustic soda solution to prevent the formation of hydrogen cyanide, which is generated at low pH values.

The cyanide bags are broken into the cyanide make-up tank, which is half-filled with raw water, and mixed using the cyanide make-up tank mixer. Cyanide mixing usually continues for a set period to ensure that the cyanide is completely dissolved. Raw water is added to the mixed cyanide solution to achieve a concentration of 25 % cyanide by weight. The completed cyanide solution is pumped to the cyanide dosing tank using the cyanide transfer pump.

Duty and standby cyanide dosing pumps will be used to pump the cyanide solution into the ring main, from which cyanide is dosed into the CIL tanks, ILR, concentrate leach tanks and the eluant make-up tank.

Spillage in the cyanide make-up process is hosed down with water to the cyanide spillage sump, and the cyanide spillage pump will pump the spillage back to either the make-up tank or CIL.

Due to the use of sodium cyanide solution, safety showers will be provided in this area.

17.5.15.2 Caustic

Caustic is delivered to site in 25 kg bags packed onto pallets. The pallets are transported from the caustic storage area to the caustic make-up area using a forklift. The reagent hoist is used to lift the caustic bags into the caustic feed hopper fitted with the bag breaker.

The caustic bags are manually placed onto the bag breaker, and the bag is emptied into the make-up/dosing tank, which will be half-filled with raw water. The solution is mixed using the tank mixer until the caustic is completely dissolved. Raw water is added to the mixed solution to fill up the tank and achieve a concentration of 20 % caustic by weight.

The duty and standby caustic dosing pumps will be used when specific batch volumes of caustic solution are required at specific distribution points: cyanide make-up, acid wash, elution and intensive cyanidation.

Spillage in the caustic make-up process flows to the sump in the caustic bunded area and is pumped back to the caustic make-up tank or CIL feed box.

Safety showers will be installed in the caustic make-up area.

17.5.15.3 Lime

Hydrated lime will be delivered to site in 1 t bulk bags and will be kept in the lime store. The bags will be transported using a forklift from the lime storage area to the lime mixing area for lime make-up, where hydrated lime powder will be made to 20 % w/w lime slurry.

The lime section will consist of two make-up tanks and two dosing tanks. The lime bags are broken into the lime make-up tank, which is half-filled with raw water and mixed homogeneously using the lime make-up tank mixer. The lime slurry from the mixing tank will be pumped to a lime dosing tank using the lime transfer pumps. The solids in the dosing tank will be kept in suspension by a lime dosing mixer.

Lime slurry will be pumped to the ring main using the lime dosing pumps to the CIL and concentrate leach circuits. The lime ring main will also supply lime to the detoxification circuit if required.

Spillage in the lime make-up area is contained in a bunded area equipped with a spillage pump that pumps the spillage back into the lime make-up tank.

Safety showers will be installed in the lime make-up area.

17.5.15.4 Detoxification Reagents

These reagent dosing tanks will be located adjacent to the detoxification facility.

17.5.15.4.1 Sodium Metabisulphite (SMBS)

SMBS is delivered to site in bulk bags. The bulk bags required for batch make-up are transported using a forklift from the storage area to the SMBS mixing and dosing area.

The SMBS make-up tank is half-filled with raw water. The reagent hoist is used to lift the bags onto the bag breaker, and the SMBS powder is discharged into the SMBS make-up tank. The SMBS powder is dissolved batch-wise to a 25 % concentration by weight. The tank is topped up to level with raw water, and the SMBS make-up mixer ensures adequate mixing.

The SMBS solution is transferred to the SMBS dosing tank using the transfer pump. Variable speed SMBS dosing pumps are used to pump the SMBS solution to the detoxification circuit at a controlled rate.

17.5.15.4.2 Copper Sulphate

Copper sulphate is delivered to site in bulk bags. The bags are transported using a forklift from the storage area to the copper sulphate mixing and dosing area for make-up. The copper sulphate crystals are dissolved in raw water to make up a solution batch of 15 % concentration by weight.

The copper sulphate bags delivered to the make-up area are lifted onto the platform on top of the copper sulphate make-up tank using the reagent hoist. The operator lifts the bags onto the bag breaker, which discharges the copper sulphate crystals into the copper sulphate make-up tank half-filled with raw water. Once the required number of bags has been added to the make-up tank, the tank is topped up to level with raw water. The copper sulphate tank is equipped with a mixer, which ensures that the crystals are dissolved completely during the make-up process.

The copper sulphate solution is pumped to the copper sulphate dosing tank using the single duty transfer pump. Variable speed copper sulphate dosing pumps, one duty and one standby, are used to pump copper sulphate solution to the detoxification circuit at a controlled rate.

A safety shower is provided in the detoxification reagent make-up area, close to the tanks.

Spillage is contained in a bunded area. The spillage pump will pump the spillage from the detoxification reagent make-up area to the tails carbon safety screen.

17.5.15.5 Arsenic Precipitation Reagents

17.5.15.5.1 Ferrous Sulphate

The ferrous sulphate bags delivered to the make-up area are lifted onto the top of the make-up tank using the reagent area hoist. The operator lifts the bags onto the bag breaker, which discharges the ferrous sulphate crystals into the ferrous sulphate make-up tank, half-filled with raw water. Once the required number of bags has been added to the make-up tank, the tank is topped up to the maximum make-up level with raw water.

The ferrous sulphate tank is equipped with a mixer, which ensures that the crystals are dissolved completely during the make-up process.

After the ferrous sulphate crystals are dissolved completely during the make-up process, the solution is pumped to the ferrous sulphate dosing tank using a transfer pump. The ferrous sulphate dosing pumps are used to pump ferrous sulphate solution to the arsenic precipitation tanks.

Spillage in the ferrous sulphate make-up area is contained in a bunded area equipped with a spillage pump that pumps the spillage to the arsenic precipitation tanks.

17.5.15.5.2 Sulphuric Acid

Concentrated sulphuric acid is used for pH adjustment in the arsenic precipitation process. The arsenic precipitation process is optimal at approximately pH 5.5. The process normally produces enough acid to sustain the optimal pH; however, there may be conditions where additional acid is required to attain the optimum operating conditions.

Concentrated sulphuric acid is delivered to site in a 1,000 L intermediate bulk container (IBC). The full IBC is lifted onto a bunded platform within the detoxification area bund. The IBC is connected to the concentrated sulphuric acid pump, which is used to pump a regulated amount of sulphuric acid to the arsenic precipitation reactor when required.

The spillage from the sulphuric acid dosing bund is pumped to the arsenic precipitation tank using the sulphuric acid spillage pump.

A safety shower will be installed in the sulphuric acid dosing area.

17.5.15.6 Other Consumables

17.5.15.6.1 Mill Balls

Grinding media sizes of 125 mm and 50 mm will be used in the SAG and ball mills, respectively. For ease of transportability, grinding media will be delivered in drums (200 L) and stored in separate concrete bunkers.

17.5.15.6.2 Carbon

Carbon will be delivered to site in 500 kg bulk bags. The bulk bags will be transported using a forklift from the storage area to the carbon attritioning area located in the carbon regeneration circuit.

17.5.16 Air Services – Plant Air and Oxygen Plant

The plant and instrument air high-pressure compressors, one duty and one standby, supply the instrument air required for instrumentation, workshops and for plant-wide general use. Instrument air is passed through one pair of air filters and the instrument air dryer. Dried instrument air is filtered again through another pair of air filters before it is stored in the instrument air receiver. The instrument air receiver distributes the instrument air to all the air-operated instruments throughout the plant. The plant air take-off is after the first pair of filters, and air is stored in the general plant air receiver.

Oxygen to the CIL and concentrate leach tanks will be supplied by a dedicated oxygen plant.

The detoxification air blowers, one duty and one standby, will supply the low-pressure air required for the detoxification and arsenic precipitation tanks.

17.5.17 Water Services

17.5.17.1 Raw Water Supply

A weir will be installed across the raw water supply nearby river to enable the abstraction of water using submersible pumps, one working and one standby, to a raw water storage pond. Details regarding this facility will be confirmed during the subsequent project development stages.

17.5.17.2 Process Water Distribution

A process water pond, located close to the raw water pond, will be used to store the water required for the process water to the plant. The return water pumped from the tailings facility and raw water top-up pumped from the raw water pond will constitute process water. Additional sources of process water are the pre-leach thickener overflow and concentrate thickener overflow.

Duty and standby low-pressure high-volume process water pumps will be used in milling, gravity scalping screen dilution, detoxification, and lime make-up.

A duty and standby set of high-pressure pumps will be installed for spray water requirements. Spray water is used on the mill trommel screens, trash screen, gravity scalping screen, carbon safety screen, loaded carbon screen, and quench screen.

17.5.17.3 Raw Water Storage and Distribution

The raw water pond located close to the processing plant will be used to supply raw water to the process plant. Plant raw water pumped from the nearby river (water supply source) will be used to feed the raw water pond.

The fluidising water requirements of the gravity concentrator will be provided using dedicated pumps.

Raw water pumps will be used to distribute water for the following: dust suppression in the crushing area, mill seal water, reagent make-up, potable water plant supply, carbon transfer water make-up, gland water top-up, high-pressure wash pumps, acid water, carbon regeneration, and the gold room.

The process water top-up system is supplied by dedicated duty and standby pumps.

The raw water pond will have a reserve for firefighting water to be distributed throughout the plant. Electric and diesel-driven pumps will supply water to the fire water system and a high-pressure jockey pump to maintain the fire system continually pressurised under no-flow conditions.

17.5.17.4 Potable Water Distribution

Raw water is supplied to the water treatment plant and is treated for potable water distribution. The potable water is stored in the potable water storage tank and delivered to the two potable water hydrospheres using the potable water pumps.

The first potable water hydrosphere will be used to maintain the required pressure in the potable water distribution header. The second potable water hydrosphere will be used to maintain water pressure to the plant safety showers.

17.5.17.5 Gland Seal Water

Gland seal water will be supplied to the slurry pumps in the milling and detoxification areas, to both the duty and standby pumps.

Gland water tanks will be used to store and distribute gland water using the dedicated pumps for each system. Raw water will be used to feed the gland water tanks.

18 PROJECT INFRASTRUCTURE

The following existing and proposed project infrastructure will support the proposed mining and plant operations:

- Existing infrastructure:
  - Exploration camp
  - Airstrip
- Proposed infrastructure:
  - In-plant infrastructure:
    - Plant buildings
    - Plant reagents and consumables stores
  - Other infrastructure:
    - Mining infrastructure
    - Access road to the project site
    - TSF
    - Water supply
    - Off-site power supply and distribution
    - Diesel fuel storage and supply
    - Accommodation camps

Figure 18.1 indicates the layout of the existing and proposed infrastructure.

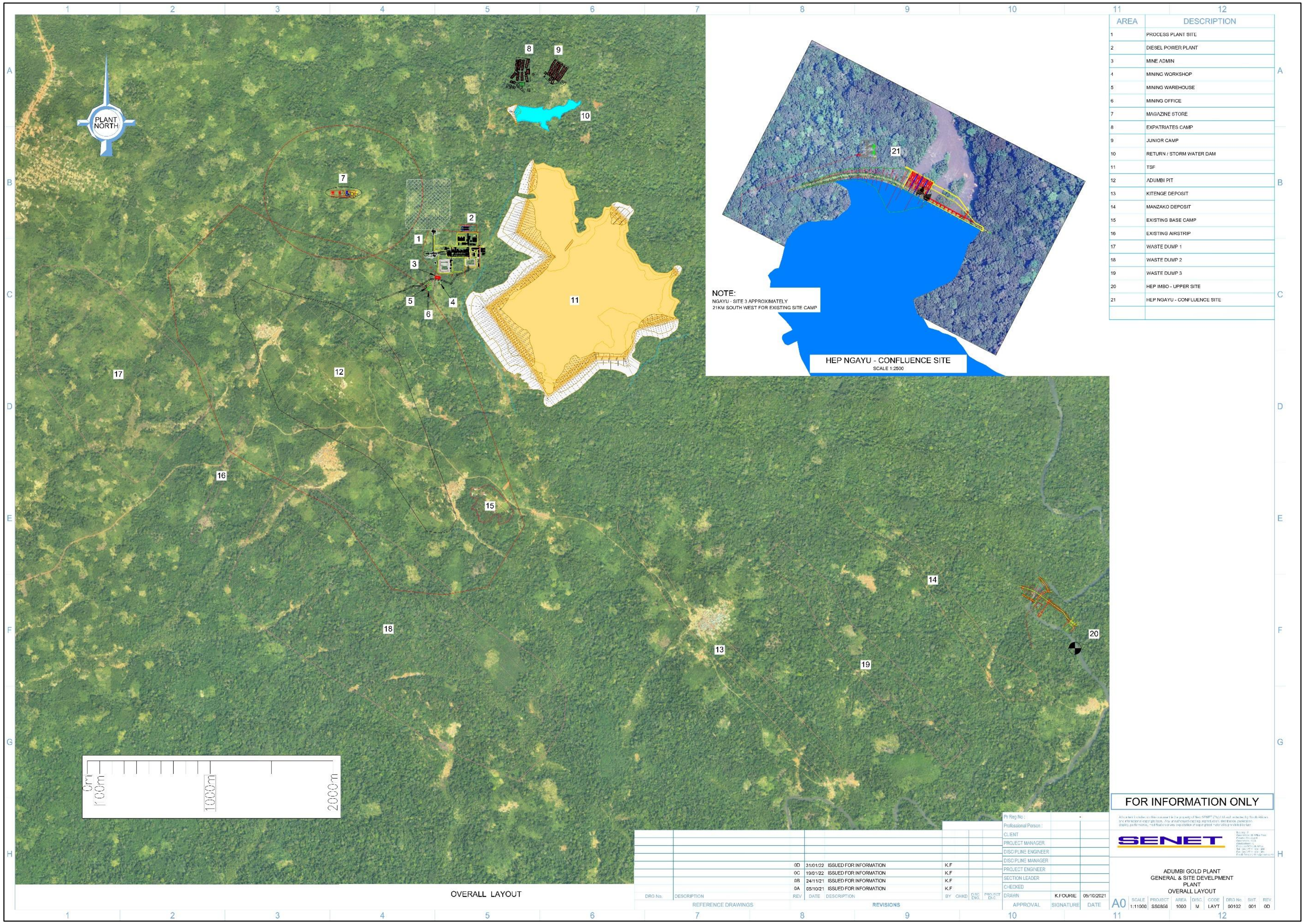


Figure 18.1: Layout of Existing and Proposed Infrastructure

18.1 EXISTING INFRASTRUCTURE

18.1.1 Exploration Camp

The existing exploration camp details are discussed in Section 5.3. It is envisaged that this camp will be utilised to house the initial construction management team. This camp will have to be decommissioned on plant start-up, due to the consequence risk should the TSF flood.

18.1.2 Airstrip

The airstrip at Adumbi is expected to be commissioned in January 2022 and can accommodate propeller aircraft of up to 8.1 t payloads. Figure 18.2 shows the current status of the airstrip.

Figure 18.2: Adumbi Airstrip

18.2 PROPOSED INFRASTRUCTURE

18.2.1 In-Plant Infrastructure

18.2.1.1 Plant Buildings

The plant buildings will consist of the following:

- Security office and change house
- Plant control room
- Plant workshop
- Plant office building

- Assay laboratory
- Plant warehouse

These block buildings will be founded on a separate terrace within the process plant perimeter. The layout of these buildings has been conceptualised utilising SENET's in-house database for similar, previously executed projects.

18.2.1.2 Plant Reagents and Consumables Stores

These pre-engineered structural steel buildings will be constructed on a separate terrace within the process plant perimeter.

The buildings will house the reagents and consumables required for the operations of the process plant.

The layout of these buildings has been conceptualised utilising SENET's in-house database for similar, previously executed projects.

18.2.2 Other Infrastructure

18.2.2.1 Mining Infrastructure

The mining infrastructure will comprise the following:

- In-pit mining haul roads
- Access haul road between the Adumbi pit and process plant

This infrastructure has been conceptualised utilising Minecon's in-house database for similar, previously executed projects.

18.2.2.2 Access Road to the Project Site

Loncor engaged with a local DRC engineer to assess the works required to upgrade the access road to allow suitable access to the project site. Loncor produced an initial route survey, "Report_PreliminaryRoadAssesment_Mombasa_Adumbi", assessing the potential routes to site, and it was agreed that the preferred route for the transportation of road cargo to site is from the Port of Mombasa following the roadway to the Kibali Mine, shown in Figure 18.3. The route then continues from Arua to Bafwabango and then to the project site (see Figure 18.4).

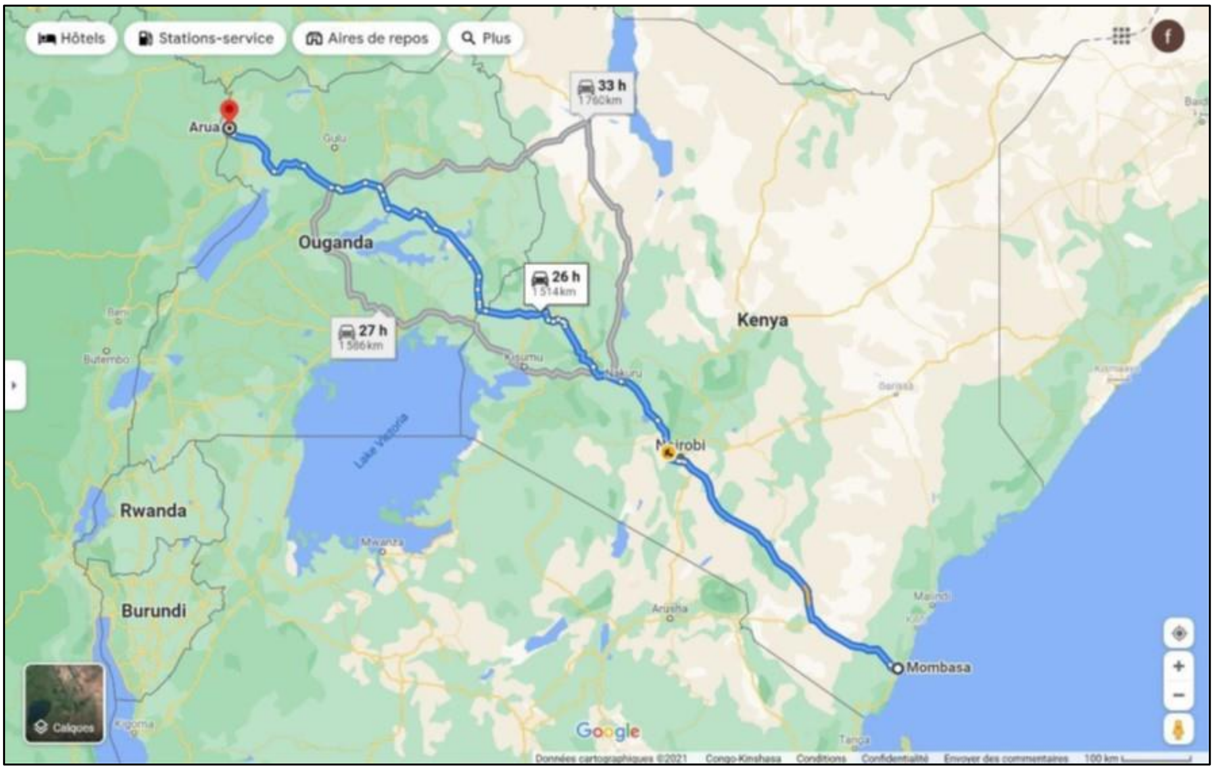


Figure 18.3: Route from Port of Mombasa to Arua (near the Kibali Mine)

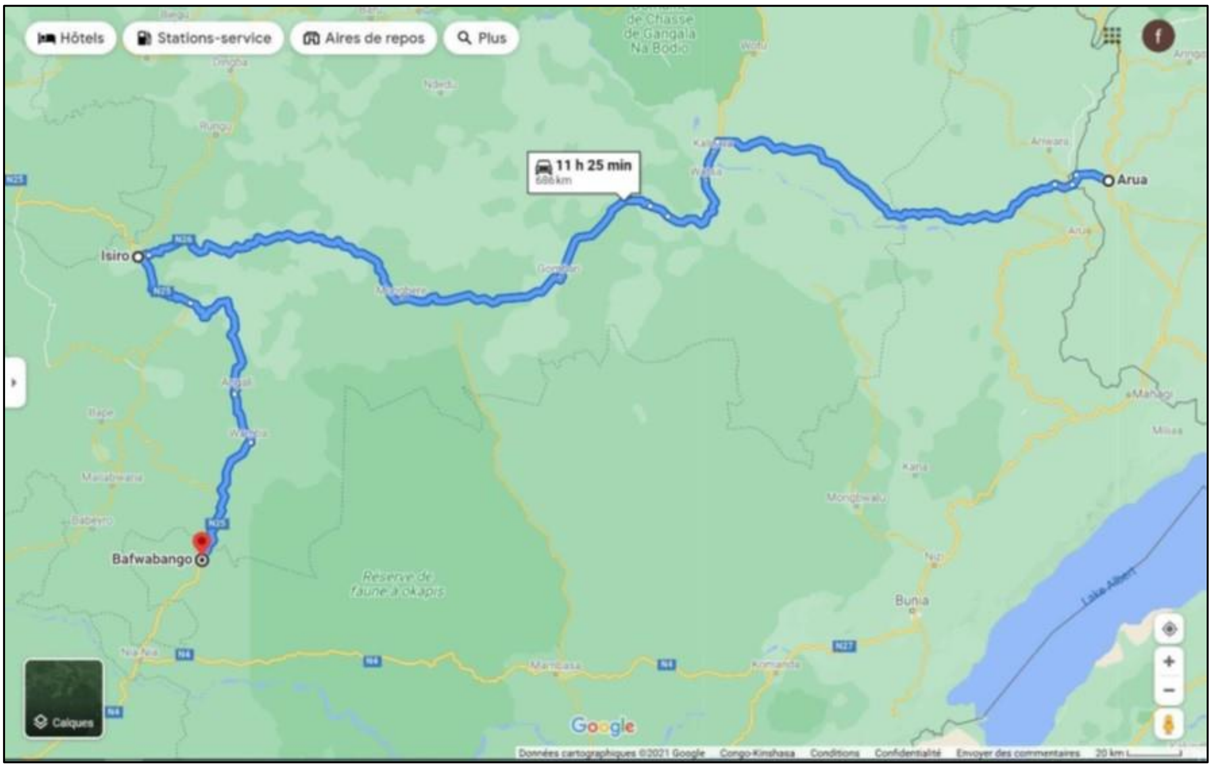


Figure 18.4: Route from Arua to Bafwabango

It was deemed that the road between Bafwabango and the project site would require rehabilitation. Therefore, a further assessment, Road Assessment Executive Summary_Bafwabango – Durba_20211030, was carried out to assess the magnitude of the required rehabilitation work.

18.2.2.3 Tailings Storage Facility (TSF)

Epoch Resources carried out the design and costing of the TSF at a PEA level of accuracy. The design of the TSF comprises the following:

- An HDPE-lined full-containment tailings dam (TD)
- An HDPE-lined return water dam (RWD) associated with the TD
- The storm water management trenches and infrastructure for the above-mentioned facilities

For the PEA, the following legislation, regulations and design standards have been taken into account:

- DRC laws relating to mine tailings storage stockpile areas
- International Cyanide Management Code's Standards of Practice
- International Financial Corporation Guidelines
- Global Industry Standard on Tailings Management (GISTM)

The design criteria associated with the Imbo Project TSF are summarised in Table 18.1.

Table 18.1: Imbo Project TSF Design Criteria

Item	Design Criteria	Value	Source	Comment
1	Tailings material	Gold	SENET	
2	Design life of facility	10 Years	SENET	
3	Tailings deposition rate	5 Mt/a	SENET	
4	Total tonnage	50 Mt	SENET	Combination of oxide, transition and sulphide
5	Tailings SG	Oxide – 2.45 Transition – 2.82 Sulphide – 3.05	SENET	
6	Particle size distribution (PSD)	75 % passing 75 µm sieve	SENET	
7	Slurry % solids by mass	40 % w/w	SENET	
8	Tailings void ratio	1	Epoch – Assumed	
9	Placed dry density	Oxide – 1.23 t/m^3 Transition – 1.41 t/m^3 Sulphide – 1.52 t/m^3	Epoch – Calculated from assumed void ratio and SG	

Item	Design Criteria	Value	Source	Comment
10	Design freeboard on TD and Water Dams	Minimum 1.5 m	DRC Regulations	Minimum of 1.5 m for dams and dykes should TSF/water dams be upstream of fauna/communities/water source. Acidifying/cyaniding/highly hazardous waste to account for 1:1,000-year storm event. Other waste/tailings to account for 1:100-year storm event.
11	Geochemical characteristics of tailings	Tailings slurry containing residual cyanide up to 50 ppm WAD cyanide – Detoxification facility in the plant can reduce the WAD cyanide to 5 ppm	SENET	Including hazardous rating and material type
12	TSF lining system	HPDE-lined facility with an underlying clay layer	DRC Regulations	Total cyanide is > 1 ppm; therefore, it is considered Highly Hazardous and Level B tightness measures are required
13	Depositional methodology	Open-end deposition	Epoch	
14	Type of facility	Full-containment facility	Epoch	Side slopes have to contain a synthetic liner
15	Return water and storm water management strategy	Project and water balance dependent	Epoch	
16	Water dam designs	RWD – 5 d to 7 d slurry water and 1:1,000-year 24-hour storm Storm water dam (SWD) – project and water balance dependent	DRC Regulations	Acidifying/cyaniding/highly hazardous waste to account for 1:1,000-year storm event. Other waste/tailings to account for 1:100-year storm event.
17	TSF decant system	Turret pumping system	Epoch	For the purposes of the PEA, it is assumed that supernatant pond water on the TD will be pumped to the RWD.
18	Rate of rise of limit of the TSF	N/A	Epoch	Full-containment facility
19	Maximum height of TSF	44 m	Epoch	
20	Seismic acceleration	0.195 g Maximum Considered Earthquake (MCE)	Epoch	Value from a nearby operational seismic assessment, Probabilistic Seismic Hazard Assessment (PSHA)

Item	Design Criteria	Value	Source	Comment
21	Survey information	Contour interval of 2 m to an accuracy of 0.5 m	SENET	Based on LiDAR
22	Topographical/ infrastructure locations	Location of pits, waste rock dumps, plant, communities, and existing infrastructure	SENET	Area constraints/"No Go" zones/mine boundaries and existing infrastructure
23	Design storm events	Recurrence interval: 1:5, 1:10, 1:20, 1:50, 1:100, 1:200; 1:1,000,1:10,000 and Probable Maximum Precipitation (PMP) Time period: 1 d to 7d	Epoch	Available information from a nearby operational site was used
24	Climatic data (evaporation and rainfall)	Wet, dry and average year	Epoch	
25	Government legislation	Laws Relating to Mine Tailings Stockpile Areas (DRC)	Epoch	

Within the current mine boundary, 11 potential TSF sites were identified for consideration. These options were considered for the safety classification of the dam according to the GISTM. The GISTM considers the population at risk, loss of life, environmental and cultural values and infrastructure and economics, and based on these aspects, the TSF is classified from low to extreme. Of the 11 options categorised, 2 options were classified as Extreme facilities due to the potential loss of life and were thus omitted from the qualitative risk assessment.

From the safety classification, the aspects are further broken down to complete a qualitative risk assessment whereby the TSF sites are rated against any flaws they may present. From this assessment, Option 2 (classified as Very High) was taken as the preferred TSF site for the Imbo Project and was advanced to complete the PEA.

The design of the TD and RWD was concluded as per the design criteria provided in Table 18.1.

The detailed PEA report, "Pre-Economic Assessment for Adumbi Gold Tailings Storage Facility", provides the full details of the design and costing of the TSF.

18.2.2.4 Water Supply

An RWD will be constructed to ensure uninterrupted supply of water to the plant. This dam will form part of the TSF, which is discussed in more detail in Section 18.2.2.3.

A raw water abstraction system will be constructed to pump raw water to the process plant for process requirements.

18.2.2.5 Off-Site Power Supply and Distribution

Two options were evaluated based on reliability, utilisation, and redundancy to achieve the best cost of energy:

- Diesel-only power generation
- Hydroelectric power (HEP) hybrid system with diesel, solar photovoltaic (PV) power generation and battery energy storage system (BESS)

The operational cost of diesel power plants is significantly high due to their fuel consumption. Commercial solar PV power has been proven to provide energy at a lower cost for longer life cycle projects. A hybrid system uses PV power generation to reduce the demand from thermal generators, which results in a considerable saving on fuel consumption and lowers the environmental impact of the emissions produced by the plant.

18.2.2.5.1 Power Demand

An outline of the electrical power demand is summarised below, based on the mechanical equipment list:

- Absorbed apparent power: 23,742 kW
- Maximum start-up power demand: 26,380 kW
- Annual consumption: 173,637,413 kWh

18.2.2.5.2 Diesel-Only Option

The thermal diesel generators are configured to operate in a prime operating mode. This will ensure that a reliable and steady power supply is provided throughout the contract period, and that the equipment supplier's warranty and service requirements for the generators are not compromised. The diesel generator parameters are provided in Table 18.2.

Table 18.2: Diesel Generator Sets

Item	Description
Total Generation Capacity	31.1 MW at 0.8 pf, Prime
Fuel Consumption, 100 % Load	0.274 L/kWh
Fuel Consumption, 75 % Load	0.270 L/kWh

The cost of fuel is based on a diesel cost of US$0.90/L.

18.2.2.5.3 Hydroelectric Power

For the HEP option, multiple scenarios were initially considered for different capacities comprising facilities at Imbo Upper, Ngayu Falls, Ngayu Confluence and Ituri Catchment. These options were further refined, with Imbo Upper and Ngayu Confluence being selected as the most feasible HEP sites.

Full details are provided in the Knight Piésold report entitled "Hydropower screening options and assessment".

Table 18.3 compares the hydropower options which were recommended for the mine.

Table 18.3: Hydropower Options (Knight Piésold, 01/2022)

Name	Catchment Area (km²)	Average Power (MW)	Peak Power/ Installed Capacity (MW)	Annual Energy (GWh/a)	Capacity Factor	Overhead Line Length (km)	Average Capital Costs (U$$ Million)
Imbo Upper Site 3	1,619	3.7	5.0	28.6	0.65	6	32.00
Ngayu Confluence	4,787	16.3	21.0	127.4	0.69	21	104.0

Figure 18.5 and Figure 18.6 show the average monthly power generation for the Imbo Upper Site 3 and Ngayu Confluence, respectively.

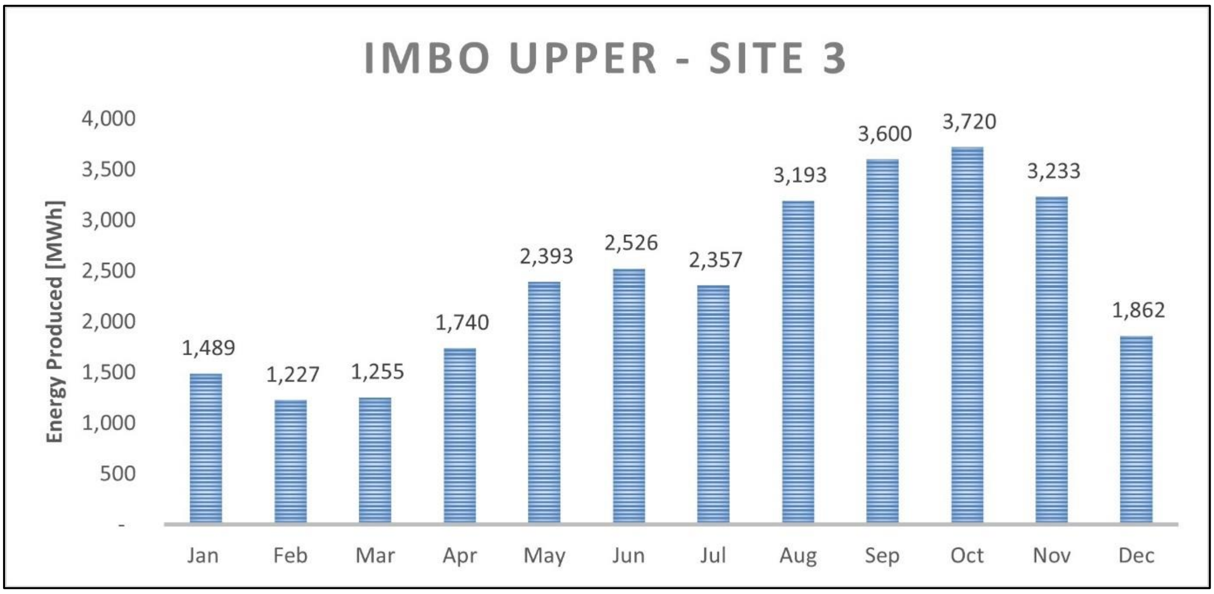


Figure 18.5: Imbo Upper Site 3 Average Monthly Generation

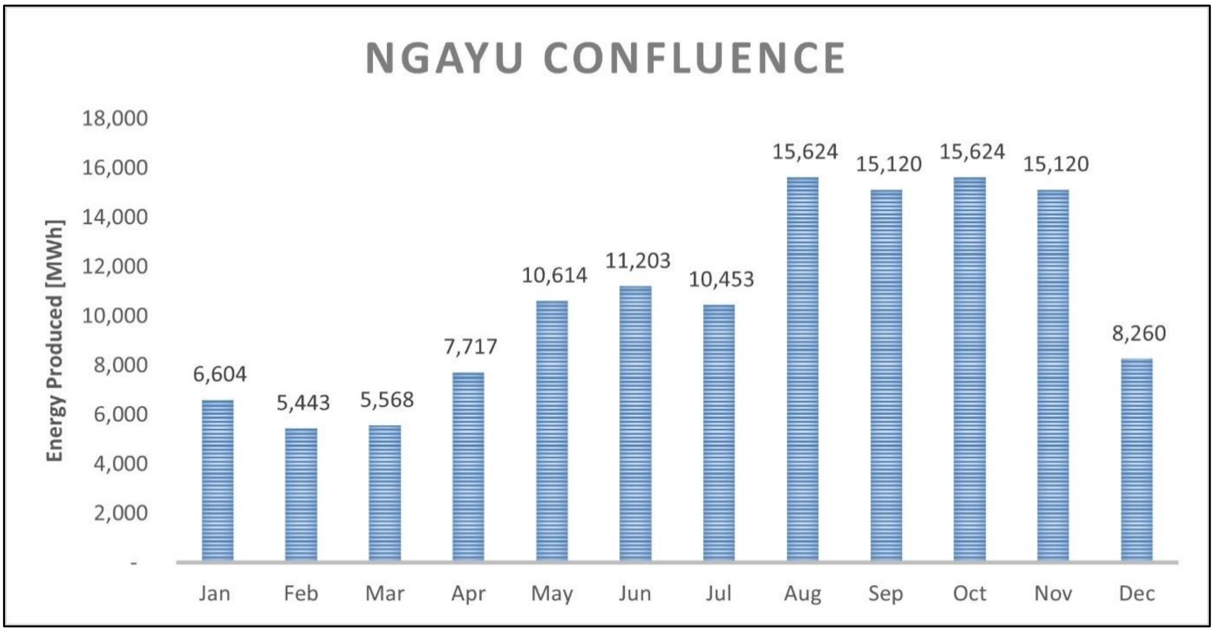


Figure 18.6: Ngayu Confluence Average Monthly Generation

From the seasonal variance shown in the hydro resources above, the power solution will require a second base power generation option for the dry season and for standby purposes if the hydro supply is interrupted.

Following a site visit and initial techno-economic modelling, a combined scenario with the Ngayu Confluence, high-speed diesel generators, solar PV and BESS provided the lowest cost of energy solution.

18.2.2.5.4 HEP Hybrid Plant Option

A solar PV plant was considered to reduce the thermal plant fuel consumption.

18.2.2.5.5 PV Plant Yield

The yield assessment was done by PVsyst software version 7.2.7, utilising long-term mean satellite-derived meteorological data from the Meteonorm 7.2 database.

The parameters of the PV plant are shown in Table 18.4, with an expected first year annual P50 yield of 48,893 MWh at the medium-voltage output terminals of the PV plant. This yield is based on operating the PV inverters at a 0.98 leading power factor. The active energy will vary for different power factor conditions.

Table 18.4: Solar PV Plant

Item	Description
AC power rating	20.0 MW
DC power rating	24.0 MWp
Production, Year 1 – P50	48,893 MWh

Table 18.5 indicates the expected annual yield for the first 10 years of the PV plant operation.

Table 18.5: Annual PV Yield

Year	P50 Yield (MWh)
1	48,892,781
2	48,672,764
3	48,453,736
4	48,235,695
5	48,018,634
6	47,802,550
7	47,587,439
8	47,373,295
9	47,160,115
10	46,947,895

18.2.2.5.6 Battery Energy Storage System (BESS)

A BESS was considered to

- Provide dispatchable power in case of sudden load variation (operating reserve).
- Store excess energy (energy shifting).
- Optimise the loading of the thermal plant to allow the thermal plant to operate at its optimal efficiency.

The BESS consists of batteries and a power conversion system that charges and discharges the batteries.

The parameters of the BESS are shown in Table 18.6.

Table 18.6: Battery Energy Storage System

Item	Description
Technology	Lithium Iron Phosphate (LFP)
AC power rating	12.4 MW
Nominal storage, beginning of life	18.4 MWh
Annual throughput, Year 1	6,354 MWh

The selected power conversion system is of the grid-forming (black-start) type, which will add additional redundancy to the thermal plant as the systems do not require a reference grid to function, and they can generate a reference grid for solar PV inverters.

The final solution is containerised with all the required cooling and protection systems to allow for maximum reliability and ease of maintenance.

18.2.2.5.7 Benefits of HEP Hybrid Power Plant Option

The hybrid system with thermal, solar PV power generation and BESS option was found to achieve the best balance of capital and operating cost.

This option is designed to deliver the lowest cost of energy and reliable electricity, but it also includes a strong renewable energy component that will significantly reduce the carbon footprint of the mine.

A hydropower system is aimed at minimising the cost of energy and emissions.

Figure 18.7 shows a comparison of the annual fuel consumption and the resulting emissions in the first year of operation, for a total energy consumption of 173,674 MWh.

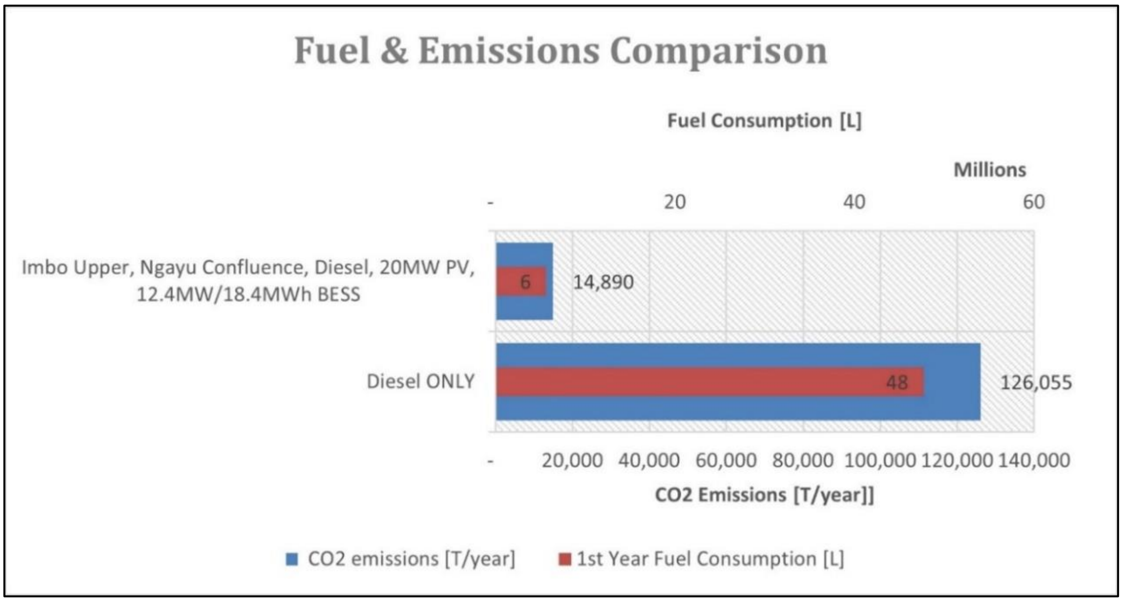


Figure 18.7: Fuel and Emissions Comparison with HEP

The HEP hybrid solution will enable significant emission reductions compared to a conventional thermal plant (see Table 18.7).

Table 18.7: Comparison of Emission Reductions between the Diesel-Only and HEP Hybrid Options

Description	Reduction
Fuel consumption	88 %
Carbon dioxide	88 %
Carbon monoxide	88 %
Unburned hydrocarbons	88 %
Sulphur dioxide	88 %
Nitrogen oxides	88 %

In addition to the environmental benefits, the significant fuel offset from the renewable energy component of the hybrid power system also de-risks the mine against fuel cost fluctuations.

18.2.2.5.8 Fuel Consumption

Table 18.8 summarises the expected fuel consumption for the respective power supply options.

Table 18.8: Annual Fuel Consumption for Respective Power Supply Options

Year	Maximum Demand (kW)	Annual Average Demand (kW)	Annual Energy (MWh)	Annual Diesel Fuel Consumption from Diesel-Only Option (L)	Annual Diesel Fuel Consumption from HEP Hybrid Option (L)
1	26,379	19,826	173,674	47,694,632	5,633,779
2	26,379	19,826	173,674	47,694,632	5,639,249
3	26,379	19,826	173,674	47,694,632	5,641,831
4	26,379	19,826	173,674	47,694,632	5,644,777
5	26,379	19,826	173,674	47,694,632	5,648,806
6	26,379	19,826	173,674	47,694,632	5,651,444
7	26,379	19,826	173,674	47,694,632	5,654,778
8	26,379	19,826	173,674	47,694,632	5,656,067
9	26,379	19,826	173,674	47,694,632	5,659,819
10	26,379	19,826	173,674	47,694,632	5,662,893

18.2.2.6 Diesel Fuel Supply and Storage

Various suppliers have been identified to supply diesel for the power generation plant. Due to the demand for diesel, more than one fuel supplier will be contracted. Loncor has engaged with the suppliers who are currently supplying the Kibali Mine.

The storage of the diesel will be at the diesel power plant.

18.2.2.7 Accommodation Camps

18.2.2.7.1 Senior Expatriates' Camp

The expatriates' camp will be constructed of plastered blockwork and will consist of the following:

- 75 rooms for technical staff (including visitors), and 10 rooms for directors (including VIP visitors)
- Entertainment club
- Restaurant and kitchen
- Laundry

- Gymnasium
- Gate and gatehouse for security control

The buildings are envisaged to be constructed utilising local labour and materials, which will provide the local people with an opportunity for employment.

18.2.2.7.2 Junior Camp

The junior camp will be constructed of plastered blockwork and will consist of the following:

- Housing for junior staff consisting of 108 rooms
- Gymnasium
- Gate and gatehouse for security control

The buildings are envisaged to be constructed utilising local labour and materials, which will provide the local people with an opportunity for employment.

19 MARKET STUDIES AND CONTRACTS

19.1 INTRODUCTION

Adumbi is planned to produce 303,000 oz/a of gold contained in doré over a 10.3-year mine life. The combined gold and silver content is expected to be 98 % (the remaining 2 % is likely to consist of impurities such as copper, iron and zinc). The gold content is expected to account for approximately 95 % to 97 % of the precious metals content, with the remaining amount being silver content. No deleterious elements are indicated in the ore head grade assayed, and as such these are not expected to be in the doré.

Doré bars are planned to be cast in 804 oz (~25 kg) bricks with approximate dimensions of 190 mm length, 120 mm width and 80 mm height. Fifteen to twenty pours per month are planned, with the doré being transported weekly from the mine site to the export facilities. After weighing, sampling and assaying, the doré will be packed and secured in high-security tamper-evident carry boxes (see Figure 19.1) in the gold room in the presence of mine production and security personnel.

Industry-standard gold room and strong room facilities will be constructed on site to hold the doré until it is ready for export. Appropriate checks and balances, security cameras, alarms, insurance and security procedures will be implemented to cover the production and storage at the mine. Maximum storage on site will be no more than two weeks' production (i.e. 12,000 oz doré) unless exceptional circumstances dictate otherwise.

MEGA FORTRIS
GROUP

Secure Tamper-Evident Carry Box

Manufactured with high-impact ABS and fitted with plastic coated reinforced metal handles to carry heavy contents, the GB Box has a tamper-resistant slide-on security lid, with one secure sealing point.

The sealing point on the GB Box can be used with either metal barrier seals or plastic indicative seals, making the GB Box easy and cost effective to seal.

The GB Box can be used to carry a wide range of high security goods, including bulk coin, cash handling, gold bullion, jewellery, high value retail goods, duty-free goods, medical and pharmaceutical drugs.

Features

1. The slide-on lid is tamper-resistant and can be sealed.
2. Plastic coated metal fold-down handles.
3. High-impact ABS plastic box and lid.
4. Lid design allows for stacking boxes on top of one another and nesting to save on space in transit.
5. One secure sealing point which can be used with either metal barrier seals or plastic indicative seals.

Technical Specifications

PRODUCT

Product	Material	Dimensions	Box Weight	Marking Area	Max Marking Digits
GB Box	High-impact ABS	265 x 140 x 95 mm	0.675 kgs	**Cover:** 250 x 130 mm **Side:** 95 x 155 mm	*According to customers' requests*

OUTER CARTON

Product	Carton Quantity	Dimensions / mm	Weight kg/carton	Volume m^3/carton	Standard Colours
GB Box	10	615 x 280 x 300 mm	7.1 kgs	0.05166	*Customisable colour – MOQ 2000 pcs*

Source: https://mfgroupmedia.blob.core.windows.net/newdatasheets/datasheet_GB_Box.pdf

Figure 19.1: High-Security Tamper-Evident Carry Box

19.2 GOLD REFINERS ACCESSIBILITY

The gold bullion bars, typically no greater than 25 kg each, produced at the Adumbi site may be sent to any of the active gold refiners in the world for toll refining. There are several refineries suitable for transforming the doré into refined gold bullion, including facilities in South Africa, Europe, North America and the Middle East. The key determinants in choosing the refinery will be credit standing, refining experience, pricing and refining terms, transport and insurance costs, and ease of logistics. There are several refiners in the world whose bars are accepted as “good delivery” by the following associations:

- London Bullion Market Association (LBMA)
- Istanbul Gold Exchange (IGE)
- Shanghai Gold Exchange (SGE)
- The Chinese Gold and Silver Exchange Society in Hong Kong (CGSE)

Table 19.1 provides a list of some of the active refiners in Europe published by the LBMA and automatically accepted by other associations. The rest of the LBMA Good Delivery list can be accessed from www.goldbarsworldwide.com.

Table 19.1: LBMA-Accredited Active Refiners in Europe

Country	Refinery Location	Active Refiner	Year of LBMA Accreditation
Belgium	Hoboken	Umicore SA	1930
Germany	Hanau	WC Heraeus GmbH	1958
Germany	Hamburg	Norddeutsche Affinerie AG	1934
Italy	Badia al Pino Arezzo	Chimet SpA	1996
Italy	Milan	Metalli Preziosi SpA	1962
Netherlands	Amsterdam	Schone Edelmetaalberivjen NV	1934
Spain	Madrid	SEMPSA Joyeria Plateria SA	1984
Sweden	Ronnskar	Boliden Mineral AB	1984
Switzerland	Mendrisio, Ticino	Argor-Heraeus SA	1961
Switzerland	Berne	Cendres & Métaux SA	1981
Switzerland	Neuchatel	Metalor Technologies SA	1934
Switzerland	Castel San Pietro	PAMP SA	1987
Switzerland	Balerna, Ticino	Valcambi SA	1968

19.3 REFINING CHARGES, GOLD PRICING AND REVENUE

The refining process will be completed within two to three working days after receipt of the bullion. The refiner will charge refining costs, which cover melting, assaying, refining, and the provision of bars accredited by LBMA.

Generally, 99.92 % of the gold contained in the bullion will be returned to Loncor, and the sale price will be fixed on the day of the refinery outrun with a settlement of two to three working days.

The refinery will refine the product to the LBMA Good Delivery standard and credit the payable gold and silver to the Project’s account three business days after arrival at the refinery.

Gold and silver will be priced as follows:

- Gold: London AM, PM or spot price
- Silver: Prevailing London market spot price

Transport, insurance and refining costs from site to the refinery, such as one in Johannesburg, will be US$3.50/oz. Shipping time should not typically exceed three days.

19.4 BULLION WEIGHING, MELTING AND ASSAYING

Weighing, melting and assaying will be performed at the refinery within one business day after receipt of the doré. Samples will be taken, and some will be used for assays and others retained and sealed for an umpire assay if required.

The material will be analysed using X-ray fluorescence (XRF) analysis as part of the refinery's standard procedure. The material will be evaluated using the weight after the melt procedure, with the following splitting limits:

- Weight: 0.10 %
- Gold: 0.5 %
- Silver: 1 %

When the assay result falls within the splitting limits, the refinery's value will be used for the final settlement. When the assay result exceeds the limits, samples may be submitted to an independent umpire at Loncor's request. A written notice period of at least 72 h should be given to the refinery. Should the umpire's result be within the limits of Loncor's result and the refinery's assay results or coincide with either, then the arithmetic mean of the umpire assay and the party closest to the umpire assay shall be used for the final settlement. Otherwise, the median of the three shall be taken as the final settlement.

Several elements, when present in feed materials, may cause health or environmental problems or damage to the refining process. There are, therefore, certain limits for the deleterious element quantities in the feed material to the refinery. There is zero tolerance for any radioactive isotopes and mercury, and the refinery specification for the other elements is given in Table 19.2.

Table 19.2: Bullion Specification

Deleterious Element	Symbol	Maximum Permitted Level (%)
Iron	Fe	2.00
Copper	Cu	10.00
Zinc	Zn	5.00
Lead	Pb	5.00
Nickel	Ni	2.00
Arsenic	As	0.20
Mercury	Hg	Not accepted

Should the maximum limits of a deleterious element be exceeded, a charge/penalty is applied. This is typically US$0.05 for every 0.10 % or part thereof in excess of these levels per dry kilogram of the material. Should mercury or radioactive isotopes be present, the bullion will not be accepted and will be returned at the mine's cost.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 OBJECTIVES

The objectives of this environmental and social component of the Adumbi PEA are to

- Evaluate the adequacy of the environmental and social work undertaken to date as input to the PEA.
- Determine, at a screening level, the potential environmental and social issues, both positive and negative, associated with the proposed mine development by
    - Classifying the development proposal in terms of the level of environmental assessment required
    - Identifying key issues
    - Identifying fatal flaws
- Identify high-level environmental and social risks that may affect the project and require further investigation.
- Identify costs for further environmental and social work required to a scoping-level accuracy.

20.2 SCOPE OF WORK

The material presented is to meet the requirements as per the DRC *Code Minier* or Mining Code, Law No. 007/2002 of July 11, 2002, as amended and completed by Law No. 18/001 of March 9, 2018. Information is drawn from the September 2021 site visit and Loncor's documentations, and an overview of the very limited environmental and social baseline data available. The following documentations were made available to Minecon for review:

- Correspondence and records of meetings with stakeholders supplied by Loncor
- Adumbi Mining Licensing documentation
- Data supplied by Loncor relating to resettlement and compensation for another project in the DRC

20.3 LEGISLATIVE CONTEXT

The DRC's environmental and social legal requirements and good international industry practice (GIIP) provide the framework, with Loncor having stated its intention to follow GIIP in developing mines, including the World Bank Group's (WBG's) Equator Principles (EPs) and the International Finance Corporation's (IFC's) Performance Standards (PSs). Further detail on the DRC Legal and International Best Practice Frameworks is summarised below.

20.3.1 National Legislation

Key legislation includes the 2006 Constitution of the Third Republic (The Constitution), the 1997 National Environmental Action Plan (NEAP) providing a framework for the management of the DRC's natural resources, as well as the Mining Code governing commercial and artisanal mining activities. The latter requires, inter alia, an Environmental and Social Impact

Study (ESIS), a Mitigation and Rehabilitation Plan (MRP), and an Environmental and Social Management Plan of the Project (ESMPP). Mining Regulations (Decree No. 038/2003 of March 26, 2003, as amended and completed by Decree No. 18/024 of June 8, 2018) give effect to the Code in terms of environmental social obligations, public consultation and requirements for ESIS and ESMPP reports. Schedules/annexes include financial surety for rehabilitation and closure measures. Exploration and exploitation permits are necessary for mining activities which are required to be compliant with environmental protection regulations. The Framework Law on the Environment (Act No.11/009 of July 9, 2011) aims to define the protection of the environment and the direct management of natural resources, and to reduce pollution and serve as a basis for sectors impacting the environment.

20.3.2 International Best Practice

The EPs updated in 2012 form the accepted framework for the management of social and environmental issues that apply to all new projects seeking project finance from Equator Principles Financial Institutions (EPFIs) with total project costs of US$10 million or more. A particular focus is placed on ensuring that adequate public consultation and disclosure are carried out so that affected communities are fully informed about the project, and their views and concerns are taken into account. All the PSs are applicable to the proposed project and are aimed at providing guidance on how to identify risks and impacts. They are designed to help avoid, mitigate and manage risks and impacts as a way of doing business in a sustainable way.

The IFC also has a suite of tools to assist with the application of the PSs, including guidance notes on each PS, general and sector-specific Environment, Health and Safety (EHS) Guidelines, and good practice manuals for the mining environment.

20.3.3 Corporate Governance

Loncor has a Business Conduct Policy, which is applicable to all directors, officers, and employees. The Policy establishes a common set of expectations and standards for Loncor and its people with respect to ethical business practices, international business, personal conduct, health, safety and the environment, and the disclosure of information. The Board currently has an audit committee which is conversant with best practices. The Board has also enacted a whistler-blower policy to protect employees who report violations of law, regulations, or corporate policy and to ensure that their concerns are acted upon as appropriate. Loncor is further considering subscribing to the Voluntary Principles on Security and Human Rights, which were developed in 2000 by companies in the extractive and energy sector, in partnership with governments and non-governmental organisations.

20.4 BASELINE ENVIRONMENT

This information has been drawn from Minecon's knowledge of the region, specifically from the Kibali Gold Mine, data from the suspended AngloGold Mongbwalu Gold Project, desktop research, as well as from the on-site assessment during Minecon's September 2021 site visit. Sensitive environmental receptors include the rainforests and streams, which are in relatively pristine condition. Infrastructure, including roads, is very limited, which has contributed to the low levels of disturbance in the area. The presence of artisanal miners and indigenous

(Pygmy) communities who carry out hunting and make use of other ecosystem services are identified as sensitive social receptors. A synopsis of the biophysical and socio-economic environment of the active exploration area in the Ngayu concession and more specifically the Imbo Project is presented below.

20.4.1 Location and Access

The Imbo Project comprises one exploitation licence, encompassing approximately 122 km^2 of land within the Ituri forest in the northeast of the DRC. The Ituri forest is 63,000 km^2 and forms part of one of the world's largest tropical forests outside of the Amazon and is home to the Okapi/Epulu Wildlife Reserve, a World Heritage Site. The Okapi reserve covers 13,726 km^2 of land and, at its western edge, borders the Imbo Licence.

Logistics access for people and goods is either by air or by road. By air, access is by fixed-wing aircraft to Nia-Nia or Isiro via Bunia, Butembo and/or Beni or by helicopter from Beni (225 km). By road, access is via the Ugandan border post of Kasindi and the A109 to Beni (77 km). From Beni, access is currently via the N4 to Komanda and Nia-Nia and then the N25 to Isiro, a distance of approximately 450 km (see Figure 20.1). In moving the project forward, and to facilitate the movement of personnel and supplies, Loncor has constructed a 1.0 km long fixed-wing landing strip on the project site.

All the roads used to access the site in the DRC are historical and of lateritic construction. More recently, the N4 and the N25 have been upgraded by Chinese contractors.

Figure 20.1 illustrates the road network in the area and the bridge section on the N25 between Nia-Nia and PK 51.

It is important to note that

- Numerous small to medium villages are dispersed along the major arterial access routes to the Imbo Project.
- The Kasindi border post is a key access point for goods entering the northeast DRC from Uganda.
- All the roads from Mombasa in Kenya to the border post at Kasindi in Uganda are paved.
- Beni is a key logistics hub for petroleum products sourced from Mombasa in Kenya. Longer-term petroleum products may be sourced from the new oil fields of Uganda.

The location of the Adumbi deposit and the Adumbi camp relative to key nearby artisanal villages is illustrated in Figure 20.2.

Figure 20.1: Regional Location of and Access to the Imbo Project

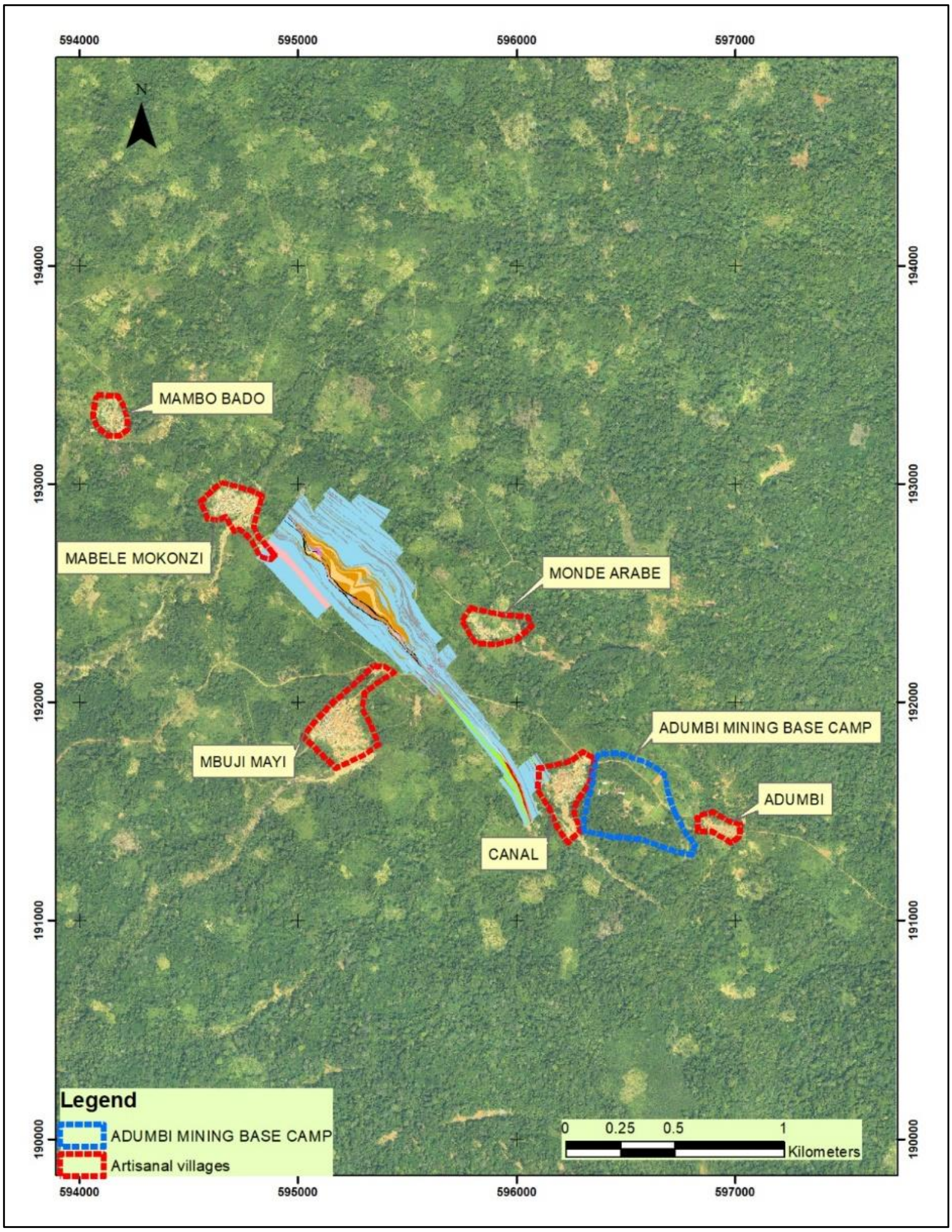


Figure 20.2: Location of the Adumbi Deposit and Surrounding Villages

20.4.2 Relief and Altitude

The Ituri landscape and forest system, within which the Imbo Project falls, comprises a slightly undulating peneplain at an altitude of 700 m to 900 m, rising up to 1,000 m in the east. The mostly gentle relief is punctuated by low massifs covering 20 km² or more and rising to 50 m to 300 m above the peneplain along old fracture lines in the Gondwanian shield. These massifs join to form a spectacular chain of granite inselbergs exposing large stretches of naked rock.

This extends for over 100 km from east to west along the Ituri and Nepoko watersheds in the north of the landscape and small isolated massifs extending over 50 km in the central part of the forest landscape.

20.4.3 Geology and Soils

The soils of the landscape are mostly derived from degraded granite and quartzite of the Gondwanian shield. The soils range from red oxysol, fine and highly degraded, to yellow or brown sandy clay. Alluvium deposits occupy the banks of the watercourses and poorly drained basins of the heads of numerous rivers. The soils are generally very acidic, and this acidity is associated with low fertility, as well as a shortage of available nitrogen and phosphorus. More fertile areas exist, particularly in association with red oxysol soils.

20.4.4 Hydrology

Almost all the landscape is part of the Congo Basin and is covered with a dense network of permanent watercourses which flow into the Upper Ituri and its main tributaries: the Epulu, Nepoko, Nduye, Lenda, Ebiena and Ngayu rivers.

The region's rivers have moderately high water flows, with the maximums reached between September and November. Flood plains are rare in the landscape and are limited to the largest rivers in the west, especially the Ituri, the Lower Ngayu and the Lower Lenda. The heads of numerous streams have poorly drained areas that create dendritic networks of marshy environments. So far, the heads of most of the basins draining the landscape have been very little affected by human activities, unlike the case with rivers originating outside the landscape. The latter often have more turbid waters associated with deforestation and other changes.

There is no hydroelectric development within the Ituri forest area. With its extremely high gradients and vast volumes of water, the Upper Ituri and its main tributaries represent a substantial potential in this field.

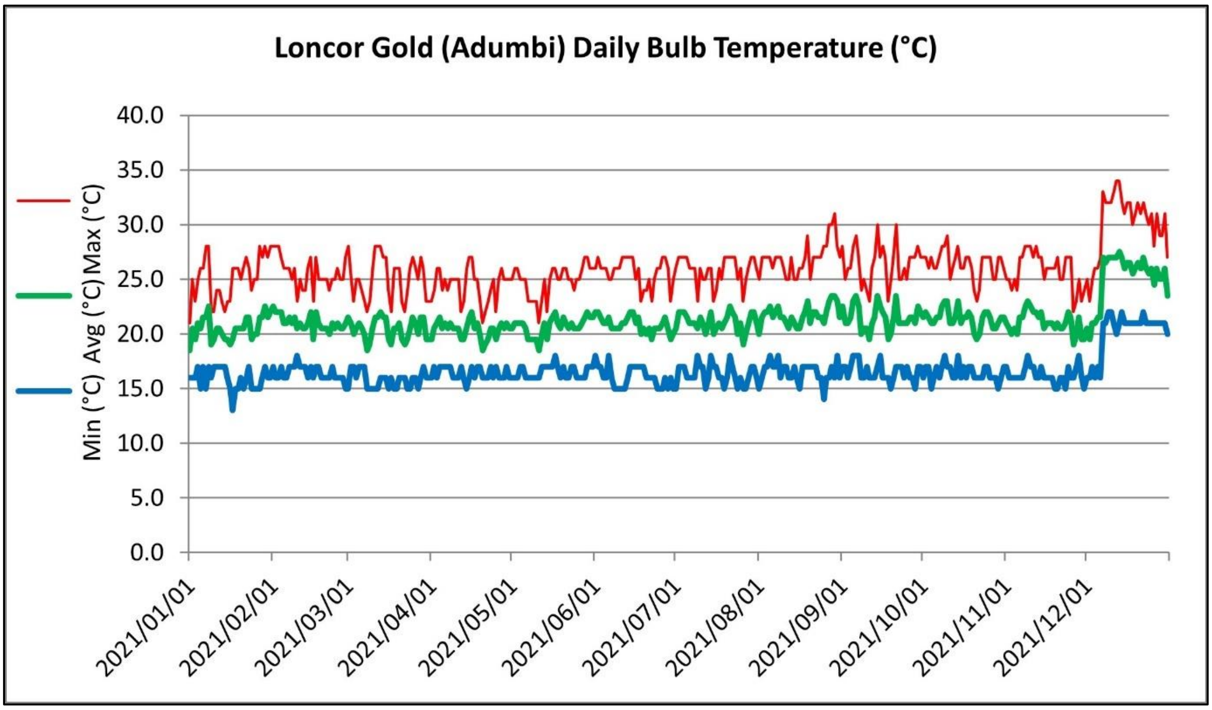
20.4.5 Climate

The climate data as presented herein is derived from the Adumbi Camp Weather Station, which has been in operation on Loncor's Imbo Licence since March of 2013, as well as from the AngloGold Ashanti Mongbwalu Weather Station and information sourced from intergovernmental organisations.

20.4.5.1 Temperature and Humidity

Based on the data collected (March 2013 to October 2013), the average daily dry bulb temperature is largely uniform throughout the year, varying between 19 °C and 23 °C, with daily lows and highs of 16 °C and 33 °C, respectively.

Humidity is typically high and, over the aforementioned period, typically averaged between 75 % and 99 % on any given day. Consequently, wet bulb temperatures are high and relatively uniform over the data set collected, averaging between 20 °C and 22°C on a daily basis. The daily dry bulb temperature, wet bulb temperature and relative humidity are illustrated in Figure 20.3, Figure 20.4 and Figure 20.5, respectively.



Source: [Adumbi, Democratic Republic of the Congo Weather and Radar Map - The Weather Channel | Weather.com](https://weather.com/weather/monthly/l/121c60e3aab63a926b031387d3b1765643ab4ed37c12c1f0b69ea313d5a923d6) (https://weather.com/weather/monthly/l/121c60e3aab63a926b031387d3b1765643ab4ed37c12c1f0b69ea313d5a923d6)

Figure 20.3: Adumbi Dry Bulb Temperature

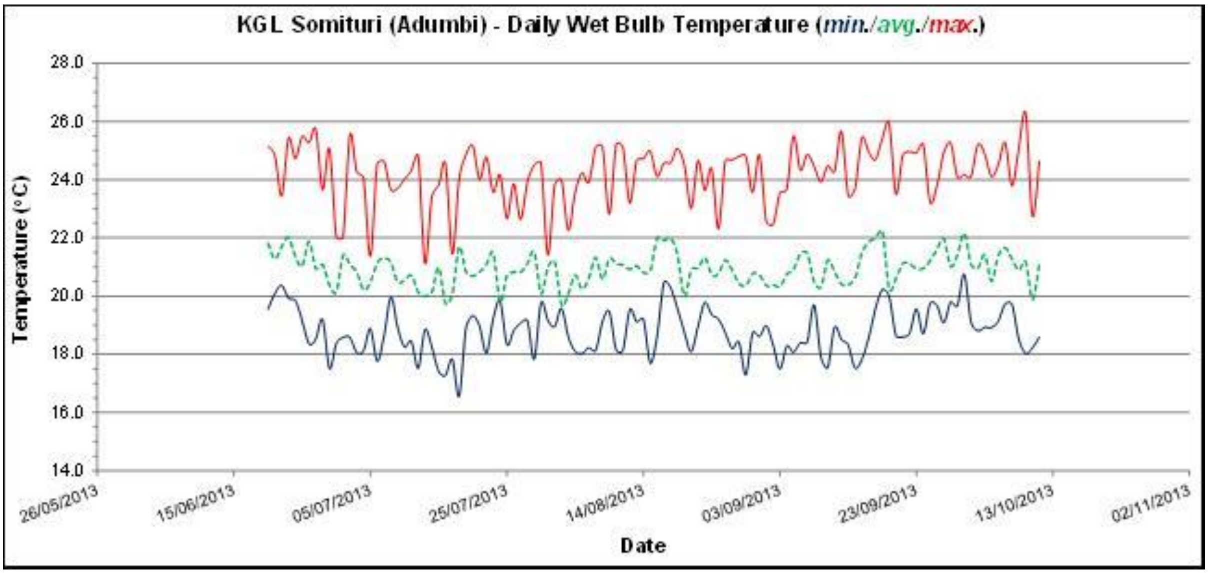


Figure 20.4: Adumbi Camp Weather Station – Wet Bulb Temperature

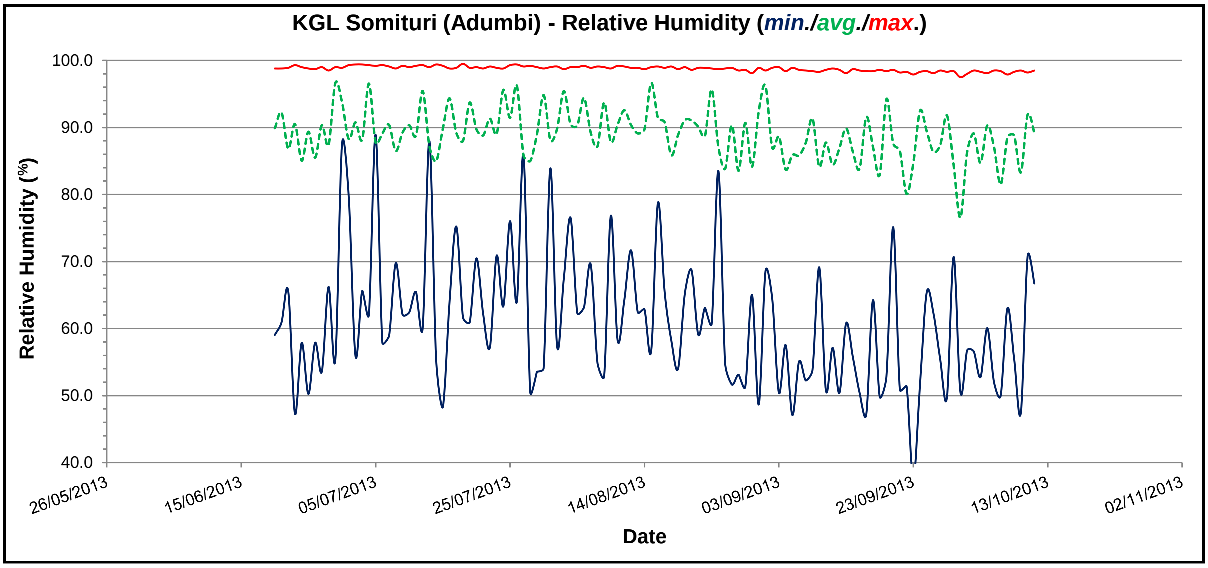


Figure 20.5: Adumbi Camp Weather Station – Relative Humidity

20.4.5.2 Rainfall

Rainfall is bimodal, with the rainy season centred on the equinoxes, and dry periods centred on the solstices. Interannual variations can be considerable and are partly linked to the variability of the passing of the intertropical convergence. The climatic system is largely controlled by the Congo Basin. However, the monsoon influences of the Indian Ocean and other factors cannot be excluded as having an influence on rainfall. It is relevant to note that at the end of the dry season in 1990 to 1991, black rain fell on the Ituri forest as a result of oilfield fires during the first gulf war. These observations suggest an eastern influence on the Ituri climatic system.

Periods of drought are not unknown in the Epulu National Park, and over the past 25 years, at least five years have had sufficiently long dry periods for the development of forest fires.

The United Nation's Food and Agricultural Organisation (FAO) rainfall charts indicate that the area lies in a region which has an annual rainfall of between 1,475 mm/a and 2,474 mm/a. These values are in alignment with the AngloGold Ashanti Weather Station data at Mongbwalu.

The driest period is often, but not always, between the months of December and January. Rainfall over this period typically varies between 30 mm and 60 mm per month (see Figure 20.6). Outside of this period, monthly rainfall figures typically vary between 100 mm and 400 mm per month.

Rainfall intensity is high, with in excess of 60 mm of rainfall falling within a 24 h period and up to 44 mm and 15 mm falling within 60 min and 10 min periods, respectively.

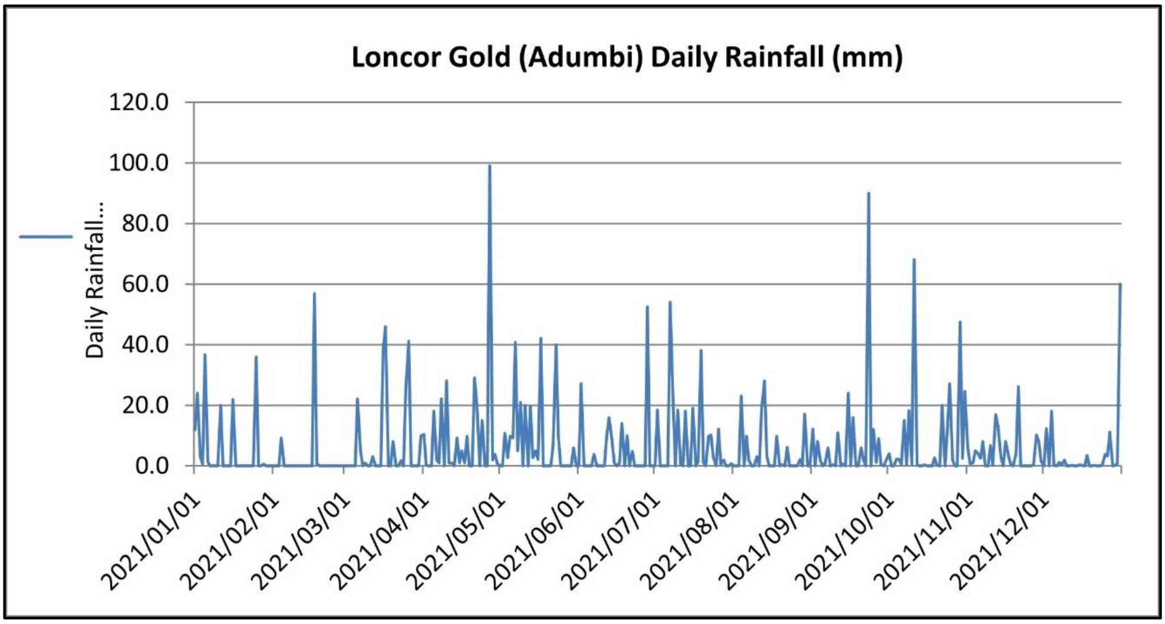


Figure 20.6: Adumbi Camp Rainfall Data (2021) from the Rain Gauge

20.4.5.3 Evaporation

A preliminary evaporation rate has been calculated using the previous data from the Adumbi Camp Weather Station’s tipping bucket rain gauge readings from 0800 to 0800 and the corresponding change in the A-Pan depth over this period. With the evaporation rate averaging 4.6 mm/d, extrapolation over a year is 1,680 mm/a.

Based on this preliminary data, it may be assumed that annual rainfall probably exceeds the rate of evaporation.

20.4.5.4 Solar Radiation

Being on the equator, daylight hours at Adumbi are fairly consistent year-round, with limited to no seasonal variation. Based on readings from the Adumbi Camp Weather Station’s solar sensor, daylight is defined as the period between 0630 and 1830.

For the weather station data previously collected, daily solar radiation averaged 160 W/m^2 over a 24 h period or if extrapolated over a day or a year, the average gross power available from the sun is 3.9 kWh/d and 1426 kWh/a, respectively. It is important to note that daily averages over this period were as low as 30 W/m^2/d and as high as 326 W/m^2/d.

20.4.5.5 Atmospheric Pressure

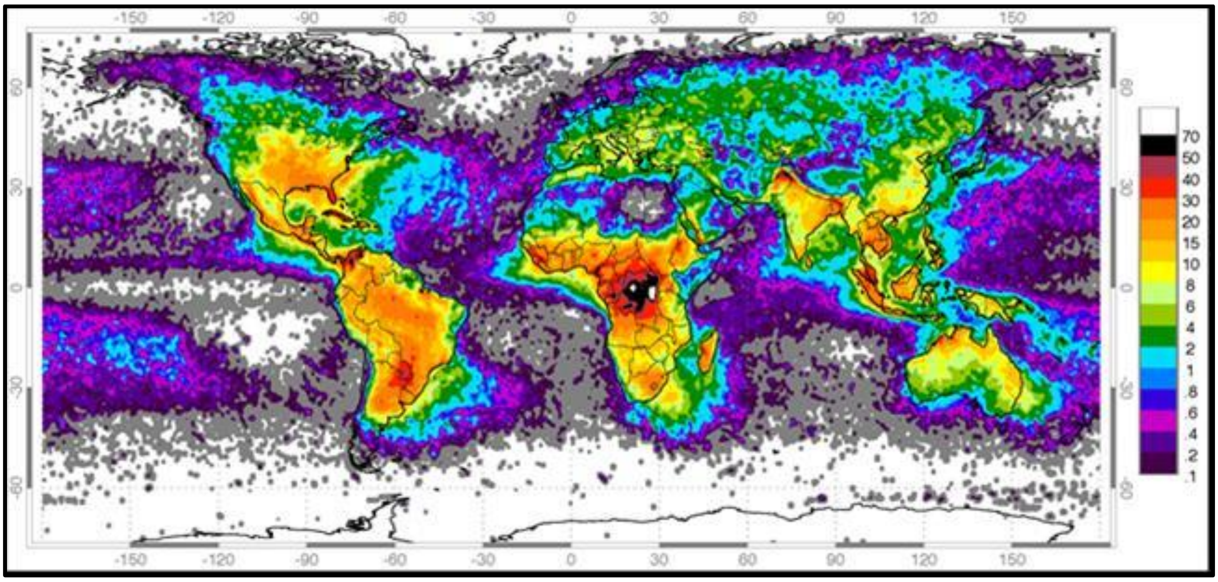
Atmospheric pressure is largely governed by elevation (approximately 900 m), with some variation expected as a result of approaching weather fronts. Daily atmospheric pressure typically averages 93.8 kPa (a), with minimum and maximum recorded pressures of 93.3 kPa (a) and 94.5 kPa (a), respectively.

20.4.5.6 Dust

No information on atmospheric and ground-level dust loads has been found for this region. Minecon recommends that a number of dust collection points be installed in the project area without further delay.

20.4.5.7 Lightning

The Central Africa region and the northeast DRC have the highest rate of lightning strikes per square kilometre in the world, with in excess of 50 flashes/km2/a as illustrated in Figure 20.7. In order to build up a profile of lightning occurrences and their associated impact on construction and operation of Adumbi. It is recommended that Loncor procure and install a lightning sensor for the Adumbi Camp Weather Station.



NOTE: The colours indicate the number of flashes per square kilometre per year.

Figure 20.7: Annual Lightning Intensity

20.4.5.8 Wind

Data collected from the Adumbi Camp Weather Station from March 2013 to October 2013 indicates that the prevailing wind direction is from the southeast, with the maximum wind velocity and average daily wind velocities being relatively low, approximately 12 m/s[1] and 0.5 m/s, respectively. Notwithstanding this, historical evidence indicates that the area can be hit with severe storms, damaging swathes of forest.

Based on the wind data collected to date, it is not techno-economically feasible to derive power from wind-based systems.

[1] Category 5 on the Beaufort scale (fresh breeze)

20.4.6 Seismicity

20.4.6.1 Regional Overview

The occurrence of earthquakes in the DRC and adjacent areas is mainly controlled by the Western Rift Valley of Africa (WRA). A concentration of epicentres follows the rift structures, starting from southern Sudan and terminating in southern Malawi. The WRA is occupied by several large lakes, viz. Albert, Edward, Kivu, Tanganyika, Rukwa and Malawi. Furthermore, the region south of the Lake Tanganyika Rift, which includes the Katanga province of south-eastern DRC and north-western Zambia, is of considerable tectonic interest since geological and geophysical studies have revealed seismically active areas that may be related to the WRA. The region is marked by a large negative Bouguer anomaly, a northeast to southwest trending zone of seismic activity (although active fault structures are poorly exposed at the surface), and several young Cenozoic rifts, viz. Upemba, Luano, Lukusashi, Luangwa Karoo and Moero. Diffuse seismicity is also observed in the Congo Basin.

20.4.6.2 Congo Basin

The tectonic origin of this intracratonic basin is unknown. No surface ruptures have been documented even though some large and damaging shocks have occurred in the area. Four earthquakes with magnitudes ranging from Mb=5.4 to Mb=5.6, which occurred in the Congo Basin during the period 1976 to 1998, were studied by Fairhead and Stuart (1983) and Dziewonski et al. (1996). They demonstrated that the Congo Basin is predominantly in a state of horizontal compression. The fault mechanisms of these earthquakes show approximately east-west oriented P-axes, which could be explained by a compression of the African Plate due to ridge push forces originating from the Mid-Atlantic Ridge and the East African Rift System (EARS).

The seismic hazard in the Congo Basin diminishes with the distance from the Western Rift Valley until, at approximately 450 km, the chance of exceeding 0.05 g (the threshold value of engineering interest) is less than 10 % in 50 years. The Adumbi deposit is situated approximately 280 km from the Ruwenzori-Lake Edward trough.

Where relevant local seismic conditions are unknown, a ground acceleration rate may be interpolated from Figure 20.8. The peak ground acceleration (PGA) figures given represent the value at which there is a 10 % probability of exceeding the stated value over a 50-year period. For Adumbi, a ground acceleration value of 0.4 m/s^2 would in all likelihood be used, albeit it would appear that Adumbi falls on the edge of the white and blue shaded areas in Figure 20.8.

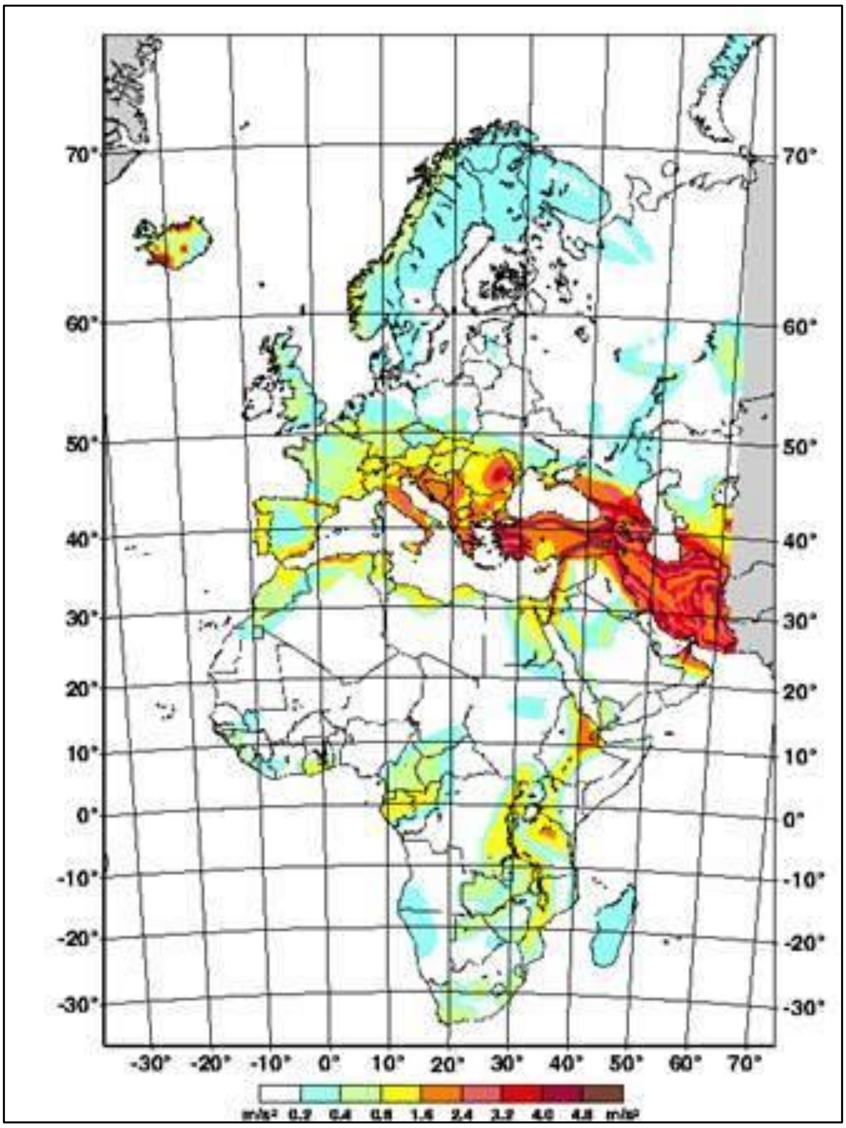


Figure 20.8: Peak Ground Acceleration (PGA) Global Seismic Assessment Programme

20.4.7 Flora

Most of the landscape is covered with dense evergreen terra firma forests with a closed canopy. They comprise forests with a monodominance of *Gilbertiodendron dewevrei* (mbau), which forms both the canopy and the undergrowth, and mixed forests in which no species is predominant, but where other Caesalpinioideae, such as *Julbernardia seretii* and *Cynometra alexandri,* are abundant.

In the north and the east of the landscape and on the dry slopes, there are semi-deciduous forests whose canopies contain more light-demanding species, such as *Entandrophragma* spp., *Khaya anthotheca, Albizia* spp. and *Canarium schweinfurthii,* and a growing proportion of dendritic Euphorbiaceae and Rubiaceae. At the northern and eastern ends of the forests, the dense forests turn into a mosaic of dry forests, evergreen forest galleries and wooded savannahs. Swamp forests grow along watercourses or in poorly drained areas. They are characterised by the presence of *Hallea stipulosa* and *Uapaca.*

On the shallow and rocky soils on the granite inselbergs are highly specialised xerophile plant formations comprising many species of plants that have a limited distribution and are of global importance for conservation.

Throughout the landscape, there are also clearings, called "edo" locally, which are maintained by elephants but used by a wide variety of fauna. The size of the clearings varies from less than a hectare to several hectares. The clearings are recolonised by the forest when the influence of the elephants disappears.

Secondary forests of varying ages cover large parts of the landscape, partly as a result of natural causes. Violent storms effectively tear large holes in the forest, and a mosaic of primary and secondary forest develops. These mosaics can cover an area of more than 10,000 ha. Over the last 25 years, three violent storms that affected the canopy over an area of more than 1,000 ha were recorded within an area of 500 km^2 around the Epulu station. Secondary forests are also the result of human activities: shifting agriculture and to a lesser extent, small-scale logging. Approximately 2 % of the land in the Epulu reserve, as well as land outside the reserve, is covered with anthropogenic environments of different ages.

20.4.8 Fauna

20.4.8.1 Mammals

The Ituri forest is exceptionally rich in mammals and a total of 90 species have been found in the central sector. This forest is home to thirteen species of diurnal primates, the highest number for an African forest, and six species of duiker. The landscape contains populations of world importance for several species with a limited distribution, endemic or almost endemic to the DRC: the okapi (*Okapia johnstoni*), the aquatic genet *(Osbornictis piscivora*), the giant genet (*Genetta victoriae*) and Hamlyn's monkey (*Cercopithecus hamlyni*). It also has large populations of globally threatened species, such as the forest elephant (*Loxodonta africana cyclotis*) and the chimpanzee (*Pan troglodytes*).

Other important species are L'Hoest's monkey (*Cercopithecus lhoesti*), the leopard (*Panthera pardus*), the Cape buffalo (*Syncerus caffer nanus*), the bongo (*Tragelaphus eurycerus*), the sitatunga (*Tragelaphus spekei*), the African golden cat (*Felis aurata*), the giant forest hog (*Hylochoerus meinertzhageni*), the red river hog (*Potamochoerus porcus*), the water chevrotain (*Hyemoschus aquaticus*) and the forest aardvark (*Orycteropus afer eriksonni*).

The forest-savannah ecotone has not yet been systematically inventoried, but reports by missionaries before the recent civil war mention the lion (*Panthera leo*), the spotted hyena (*Crocuta crocuta*), the hippopotamus (*Hippopotamus amphibius*), the East African Defassa waterbuck (*Kobus ellipsiprymmus defassa*), the bongo (*Tragelaphus eurycerus*), the bushbuck (*Tragelaphus scriptus*), the bohor reedbuck (*Redunca redunca*) and the vervet monkey (*Cercopithecus aethiopicus*). The skin of a little-known meerkat, Dyboswki's meerkat (*Dologale dybowski*), a species which lives at the edge of forests, was collected in 2013. This suggests that the forest-savannah mosaic could contain specialised fauna that is rare or absent in other parts of the Congo Basin and absent from the more arid regions to the east and the north.

20.4.8.2 Ichthyofauna

The Ituri River and its tributaries contain an ichthyofauna that is still largely unknown. It is fished locally, although not intensively at present.

Given the fact that the Ituri Basin is well upstream in the Congo Basin, its fauna is not as rich as in the central basin. Furthermore, some major rapids on the middle course of the Ituri create a bio-geographical barrier which isolates this river from the Congo River. Preliminary inventories carried out at the beginning of the 1980s showed that the ichthyofauna consisted primarily of generalist species that were usually widely distributed; it also included some species that have not yet been found elsewhere and specialist species such as rock browsers, which live in torrents and are probably endemic to the Ituri Basin. Some of these species were unknown to local fishermen, whose methods are inappropriate for catching these specialist species.

20.4.8.3 Invertebrates

Diurnal butterflies are the only invertebrates that have been the subject of systematic inventories in the landscape. Collections have been made in three places. In Epulu (altitude 750 m), the 6,251 species collected belong to 116 genera and 48 species. They include the most eastern collections for five species previously known only in the west or Central-West Africa. These species are absent in the eastern landscape where altitudes exceed 900 m. These results confirm the importance of the Ituri forest as an area where species from separate bio-geographical regions come together.

20.4.8.4 Herpetofauna

There are no recent inventories, and research will be necessary before the value of the landscape for this group of animals can be estimated. However, collections in museums suggest that the Ituri forest is rich in reptiles, with three species of crocodiles, but that they do not constitute a “hot spot” in this field. Very little is known of the amphibians.

20.4.8.5 Birds

Ornithological inventories have only covered small portions of the landscape, but at least 333 species have been observed in the central sector of the Epulu reserve. Systematic observations in the Epulu area have revealed a rich avifauna and a large number of specialised forest species, particularly among the ground thrushes (*Zoothera* sp.), Timalidae and Accipitridae.

20.4.8.6 Humans

Until recently, the Ituri forest was one of the least populated areas in the northeast of the DRC despite a very long history of human occupation. Cut stone tools found at the eastern edge of the landscape indicate human presence in the Middle Stone Age. However, it is not certain if the region was covered in forests at that time. Recent excavations in sheltered areas under rocks in the north of the landscape show that a few millennia ago, the forest was inhabited, but played only a minor role in the development of human cultures, particularly in the expansion of ironworking.

When the first Europeans arrived at the end of the 19th century, and the first documents were written, the forests of Upper Ituri contained only small, scattered villages, and vast areas were not inhabited on a permanent basis.

The human populations in the landscape increased during the colonial period, following the opening up of the first roads and the development of mining and agricultural plantations in the region. Over the last 60 years, and most notably in the last 30, considerable migratory movements have invaded large portions of the landscape.

This immigration continued even during the conflicts between 1996 and 2003 and in spite of the clashes between rival militias who were present throughout the landscape. Some immigrants were fleeing insecurity in their home region; however, even during the periods of conflict, most of them were motivated by economic opportunities. These opportunities included easy access to cultivable land, jobs in mining or small-scale forestry, and the small businesses that these activities generated.

20.4.8.7 Ethnic Groups

The semi-nomadic Mbuti and Efe, colloquially know as pygmies, were probably the first inhabitants of the region, but the date of their arrival is unknown. While their way of life today is similar to what it was originally, these pygmies were not able to live inside the forest independently of other ethnic groups and had to colonise the forest at the same time as groups practising migratory agriculture. The latter are believed to have come to the region 2,000 to 3,000 years ago. At present, the number of pygmies in the landscape is estimated at 30,000. They continue to lead a semi-nomadic lifestyle, but still depend heavily on the Bantu populations.

Among the forest farmers, the main ethnic groups are the Bila, Ndaka, Lese, Mbo and Mamvu. They depend on migratory agriculture, supplemented by fishing and hunting. The Ngwana arrived in the landscape with the Arab slave trade in the 19th century. During the colonial era, new populations were added. The most numerous are the Nande, from the mountains to the east of the landscape, and the Budu, from densely populated regions to the north and west. Many of the newcomers came to rejoin members of their family or clan. Today, all the centres contain several ethnic groups. The population of the Epulu forest reserve comprises 2,000 inhabitants and over 30 different ethnic groups.

20.4.9 Human Activities

20.4.9.1 Fishing

Small-scale fishing using nets, traps, lines and hooks is the most important activity after agriculture. In some communities near watercourses, fishing is the main activity. Immigrants from lower down in the hydrological basin bring new methods and fish more intensively. An inventory of the ichthyofauna and its productivity is an essential priority in the landscape.

20.4.9.2 Hunting

Hunting is practised throughout the landscape and is the primary activity of the Mbuti and Efe, who generally practise hunting by tracking. The use of nets and rounding-up methods is practiced in the centre and the south of the landscape, while hunting with bows and arrows is dominant in the north and the east. Eight species of small ungulates are the main targets of this hunting. Hunters using dogs also catch several species of primates and small animals such as meerkats or rodents. When hunting with nets or bows and arrows, many animals manage to escape. This reduces the impact of hunting and makes it profitable only when there are large numbers of animals. Mbuti hunters have hunting territories with more or less fixed camp locations, which are used on a periodic basis by clan or family groups.

Village hunters primarily use snares, which ensure a degree of success even when the animal populations are low. Budu hunters recently extended hunting with snares deep into the landscape from the west. Few recent immigrants from Kivu are specialised hunters.

The use of firearms was not common until the civil war, when rival militia, joined by the national police, set themselves up in the landscape to control access to gold and timber or to organise commercial and ivory hunting. At present, military arms are still circulating in the landscape.

20.4.9.3 Subsistence Agriculture

Cassava, banana-plantain, rain-fed rice, cocoyam, yams and groundnuts are the main crops in the landscape. Maize is used for the local production of alcohol. The agriculture practised by the groups who traditionally live in the forest is based on a rotation system of two crop years and ten fallow years. Fields are small, generally less than 2 ha, and represent only a small proportion of the agricultural mosaic. The long fallow periods allow the soil to regain its fertility and provide good habitats for fauna. The populations of certain animal species are denser in these secondary environments than in the adjacent forests. In areas where fallow periods are still long, clearing of the primary forest is very limited. The mosaic of secondary forests is rich in palm trees (*Elaeis guineensis* and *Raphia* sp.), which are rare in dense forests.

The recent immigrants practise a more intensive agriculture, with larger fields, shorter fallow periods and more extensive clearing of the primary forest. Although more research is necessary, studies show that fallow periods of five years or less lead to the soil becoming depleted, the regeneration of the forest coming to a halt, and the forest being replaced by prairies of *Imperata* or thickets of bushes and lianas.

20.4.9.4 Cash-Crop Agriculture

During the late colonial period and up to the 1970s, there was substantial production in the landscape of products for marketing: rain-fed rice and palm oil were intended for the urban and mining centres and coffee for export. The traditional forest populations had very few cash crops. Coffee was produced by small family businesses, generally belonging to recent immigrants, or vast plantations typically owned by Europeans. In the 1970s, the expatriates' companies were nationalised and subsequently quickly abandoned. By the end of the 1970s, the road network began to deteriorate, access to markets became difficult, prices fell and crops were gradually abandoned. Today, production is minimal, and there are no longer any coffee plantations. These plantations have been converted into land for subsistence agriculture or have been invaded by the forest.

20.4.9.5 Mining

Ituri is rich in minerals: gold, coltan and diamonds. Gold mining dates back to the colonial period. Kilo-Moto gold mines was one of the main companies at that time and is located near the eastern edge of the landscape.

Access to mining resources was a major issue in the civil war and is still at the heart of the present conflicts. Hundreds of small permanent or semi-permanent mines have become active since the legalisation of small-scale mining in the 1980s. No landscape-wide survey exists, but at least 25 camps are operational in the Epulu reserve.

Today, most mining operations in the region are on a small scale, even in the Kilo-Moto concession where miners work under contract. Most operations are concerned with alluvial gold, but a growing number of artisanal miners are trying to extract gold from the primary rock after grinding and washing it with mercury. This trend will increase if the price of gold remains high.

However, in 2013, the Kibali gold mine jointly owned by Barrick Gold and AngloGold Ashanti came into production. In 2019 and 2020, the Kibali mine (managed by Barrick Gold Corporation) produced a record 814,027 oz and 808,134 oz of gold, respectively, demonstrating the ability to successfully develop and operate a modern top-tier gold mine in one of the world’s remotest and most infrastructurally under-endowed regions. The Kibali mine, which is approximately 220 km from Loncor’s Adumbi deposit, is now Africa’s largest gold producer. Besides Kibali, a number of other international gold exploration companies have been exploring for gold in this part of the DRC.

20.4.9.6 Logging

Logging is concentrated in semi-deciduous forests, near the transitional area between the dense forests and the wooded savannahs in the east, where valuable species such as *Milicia excelsa, Entandrophragma* sp. and *Khaya anthotheca* are more abundant.

There were no concessions in the initial landscape, but in 2004, the limits of the landscape were extended to encompass the only legally registered logging area in eastern DRC. It covers approximately 52,000 ha and has been allocated to the Enzyme Refiners Association (ENRA), which has its base in Beni. This small company produces 5,000 m^3 to 7,000 m^3 of sawn timber a year. Another company, Dara Forest, was active during the period of the rebellion, but it was unable to legalise its activities, most likely due to the fact that it was involved in illegal exploitation of the country’s resources during the war. It is continuing with its illegal activities.

Most logging in the landscape is in the form of small-scale activities stretching from the south and east right across the landscape. There is virtually no logging in the north or west, because of the very bad state of the roads. This illegal small-scale logging and agriculture are major threats for the ENRA concession.

20.4.10 Adumbi

The Adumbi deposit is found within the Imbo exploitation licence of 122 km^2 and dissects, in part, the administrative districts of Mambasa and Wamba. Complicating matters is the fact that whilst the licence and surrounding villages (Nia-Nia, PK 25, PK 47 and PK 51) fall under the Mambasa territorial authority and by association the Ndaka tribe, the majority of the people in these villages are of the Budu tribe and look for guidance and assistance from the Wamba territorial authority and associated government departments

Notwithstanding this, regional migration from the colonial period has resulted in an amalgam of people from different ethnic Bantu groups along with indigenous populations of pygmies residing in areas immediately adjacent to and along key transit routes to the Imbo Licence.

Key regional towns and the population demographics of the Nia-Nia region are illustrated in Figure 20.9 and summarised below:

- Key regional cities/towns and provinces:[2]
    - Butembo (North Kivu): 217,625 (2012)
    - Beni (North Kivu): 95,407 (2010)
    - Bunia (Orientale): 366,126 (2012)
    - Komanda (Ituri): Unknown
    - Mambasa (Ituri): ≈ 20,000
    - Nia-Nia[3] (Ituri): ≈ 2,300
    - PK 25[4] (Ituri): ≈ 9,532
    - PK 47 (Ituri): ≈ 6,116 (2013)
    - PK 51[5] (Ituri): ≈ 22,659 (2013)
    - Wamba[6] and its surrounds (Haut Uele): ≈ 130,000
    - Isiro (Isiro):[7] ≈ 185,000 (2012)
    - Kisangani (Tshopo): ≈ 935,000 (2012)
- Territories:
    - Wamba (423,000)
    - Mambasa

[2] The values given are based on discussions between Loncor and the District Government Doctor for Wamba. No official census has been cited.

[3] ≈ 50 km from Adumbi Camp

[4] The numerical values after the term PK refer to the distance from Nia-Nia along the N25

[5] PK 51 and PK 47 fall within a 6 km radius of the Adumbi base camp

[6] ≈ 50 km from the Adumbi base camp

[7] ≈ 180 km from the Adumbi base camp

Ethnological demographics for Nia-Nia and its surrounds, including PK47 and PK51, are summarised below:

- 85 % Ndaka (Mambasa) and Budu (Wamba)
- 5 % Nande (Butembo – control regional trade)
- 5 % Beru (subset of Budu)
- 2 % to 5 % Mbuti/pygmies (subset of Budu chieftaincy)
- Bombo chieftaincy (subset of Ndaka chieftaincy)
- Lokele (fishermen from Kisangani)
- Hema (Uganda alliance) and Lendu – small percentage

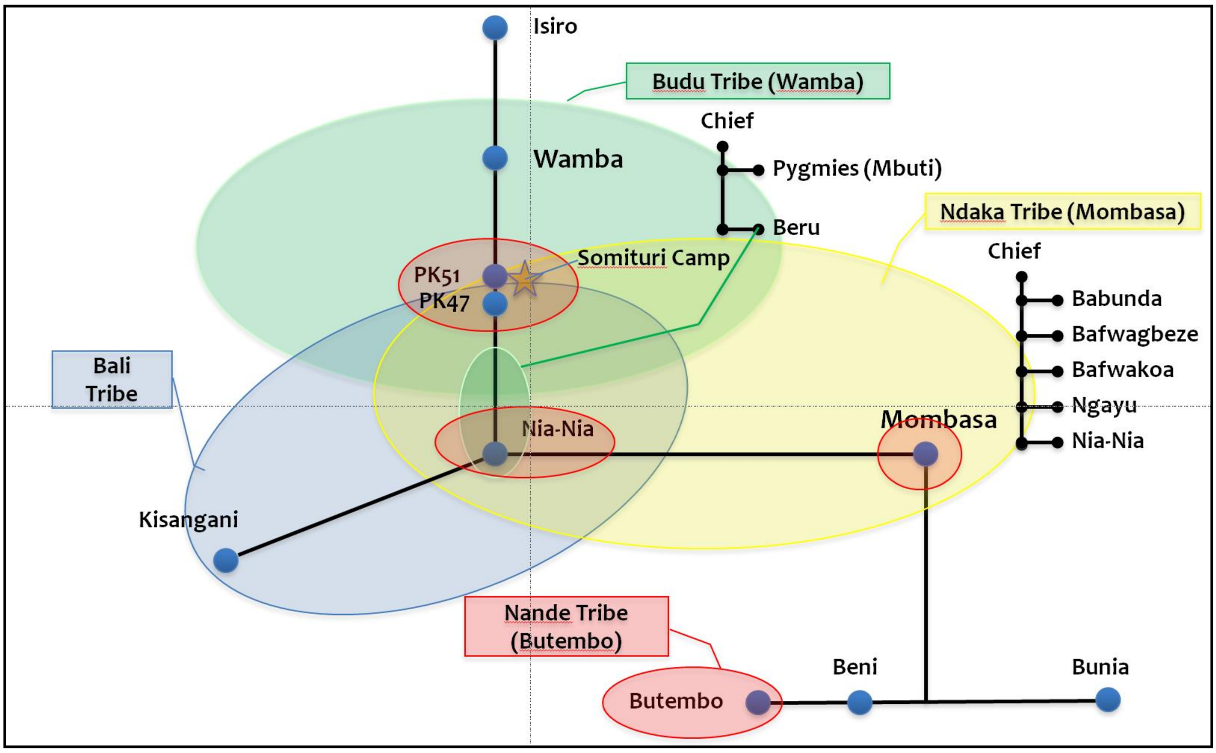


Figure 20.9: Local Tribal Structures and Demography around Adumbi (Somituri)

Economic activity in Nia-Nia and its surrounds, and the various role players are summarised below:

- General trade and trade in gold and diamonds controlled by Nande
- FARDC (Armed Forces of the Democratic Republic of Congo) (Wamba controlled) oversee and control artisanal mining groups
- Diamonds (Kisangani) and gold (Bunia)
- Ituri district: 60,000 to 150,000 people employed in artisanal mining
- Agriculture (cassava, sugar cane and sombe), hunting and fishing
- Hardwood exploitation (Beru Chief)

20.5 ENVIRONMENTAL AND SOCIAL ISSUES AND RISKS

Risks can arise as a result of project activities but may also be posed by the broader context in the DRC and the local environment. Risks associated with the issues identified broadly fall into categories including the following:

- Compliance (enforcement/transgressions)
- Scheduling (delays to the project)
- Liabilities
- Financial (cost implications for the project including access to funding)
- Reputational risks posed by factors including the presence of active NGOs in the DRC

The key environmental and social issues identified during this initial screening study as posing potential project risks are outlined in Table 20.1.

Table 20.1: Possible Impacts and Associated Project Risks

Issue	Potential Risks
Biophysical Environment	
Surface water, groundwater and air quality	Impacts associated with the proposed development pose risks to livelihoods, including individuals and communities that depend on ecosystem services. There is a risk of liabilities arising from impacts on human health from cyanide toxicity. This would generate reputational risk for the project.
Biodiversity	Rich biodiversity in the project area (Ituri Rainforest) provides ecosystem services to the human population in the area, and upgrading road access has the potential for large-scale impacts from commercial logging and bushmeat trade. A reputational risk is posed from contributing to the alteration of biodiversity in the broader project area, and the ecosystem services which they provide.
Climate	High rainfall poses a risk to scheduling, and failure to cater for storm water volumes during the prolonged wet season will carry cost and reputational implications.
Socio-Economic Environment	
Infrastructure	Mine-related traffic will impact road safety and road capacity, with liabilities and reputational risk arising from collisions with pedestrians, children, motorcyclists and other traffic. Inadequate social services and infrastructure pose a risk to the project in terms of schedule delays, cost implications, and liabilities from inadequate emergency preparedness and response.
Safety and security	Ongoing conflict in the region poses a risk to the health and safety of project employees and contractors.
Community development	There is a risk that the project will generate dependency on authorities and local communities. resulting in undermined self-sufficiency. This could result in reputational risks for Loncor.
Resettlement, land acquisition, land replacement and compensation	Risks from inadequately addressing these issues include delays to project implementation, liabilities arising from poor planning of resettlement locations, and reputational risks from impacts on livelihoods. Failure to implement international good practice on the re-positioning and or resettlement and compensation for the project could result in reputational risk such as through NGO action, increased cost implications, and delays through prolonged issues raised by dissatisfied stakeholders.

Issue	Potential Risks
Health and safety	Ineffective health and safety measures pose a reputational risk. The remote location away from advanced health infrastructure poses risks from project-induced impacts. This means high costs could be anticipated for developing infrastructure and for effective emergency evacuations.
Labour	Failure to identify suitably trained manpower for the project poses risks of increased costs from training, delays to project schedule from the need to bring staff or spend time training, as well as liabilities from employment of underqualified and inexperienced staff.
Vulnerable groups and artisanal miners	Insufficient attention to vulnerable people, including indigenous groups (pygmies), poses a reputational as well as financial risk to the project, including through potential action by NGOs. Conflict with artisanal miners poses a reputational as well as potential safety and security risk to the project.
Sense of place	The project poses reputational risks owing to potential impacts on sense of place.
Environmental Impact Assessment Process	
In-country authorisation process	Project authorisation processes pose a significant risk of project delays. Regulatory capacity could also have implications in terms of authorisation timing.
Area of influence	Inaccurate definition, as well as changes to the area of influence, could cause delays in the completion of feasibility documentation.
Alternatives	Insufficient consideration of alternatives could pose risks through non-compliance with IFC requirements.
Stakeholder Engagement	
Project engagement	Insufficient expectation management poses a reputational and financial risk to the project. If not well managed, there are potential security threats for the mine. Unrealistic expectations of benefits on the one hand, and insufficient understanding of potential negative impacts on the other hand by surrounding communities pose a reputational risk to the project.
Policies	Insufficient attention to the implementation of international best practice poses a reputational risk to Loncor. It could also result in an increased need for expenditure downstream should red flags and key potential issues not be addressed at the earliest possible stage.

20.6 CONCLUSIONS AND RECOMMENDATIONS

20.6.1 Conclusions

On the basis of currently available information, screening undertaken as part of this exercise has not identified any fatal flaws from an environmental and socio-economic perspective. Key issues have been identified which will need to be investigated further during the PFS and feasibility study (FS) phases of the project. The list of issues is considered preliminary and is expected to be expanded once the first round of stakeholder engagement has been undertaken.

In terms of DRC legal requirements, an ESIS and ESMPP are required to support an application for a mining licence as well as other permitting requirements. The current Imbo Permit is an mining licence, as such, it would not be requirement for Loncor to complete such ESIS and ESMPP studies. However, should the project require financing by an EPFI, Minecon believes the project will be categorised as a Category A project and as such will require that a full ESIS and ESMPP be undertaken for the mine and all associated infrastructure and facilities, including roads, power and water supply, housing, and host-resettlement site.

All the IFC PSs, including PS 7, which deals with indigenous people, are relevant to the project.

20.6.2 Recommendations for Project Design

Based on Minecon's understanding of the latest project layout as shown in Figure 18.1, the following recommendations have been identified at this early stage from an environmental and social perspective:

- The TSF and the proposed plant are sited in the same catchment.
- Six artisanal mining villages identified within the project footprint have been assessed for social repositioning. The total population of these villages is estimated to be approximately 5,800.

20.6.3 Recommendations for Further Work

Adumbi will require a full ESIS and ESMPP to be undertaken. However, it is recognised that it is too early in the project's development life cycle to commence immediately with the ESIS and ESMPP. There are, however, activities that could be undertaken in the early project stages which would provide important input to the PFS and FS. This approach could also reduce the overall schedule of the ESIS and ESMPP once they are commenced. This approach provides the following:

- Key steps of a full ESIS and ESMPP
- Environmental studies typically undertaken during the PFS and FS stages of the project development life cycle
- Environmental and social studies that should commence during the PFS to either provide input to the PFS or address scheduling issues associated with these being long-lead items that will contribute to the FS and ESIS/ESMPP

20.6.3.1 ESIA (Feasibility) Process

It is recommended that the ESIS/ESMPP be undertaken in a phased approach to meet the requirements of both the DRC government and the IFC, as well as to provide meaningful input into the FS to cover the following:

- Preparation of design criteria for ambient and workplace environmental conditions and emissions and effluents to inform the FS
- Preparation of a preliminary environmental impact assessment and preliminary environmental management plan in support of the potential financing application
- Baseline data collection and preparation of an IFC-compliant ESIS and ESMPP. Key elements of the process itself are as follows:
    - Scoping (engagement of interested and affected parties to identify key issues, from which final terms of reference for the detailed ESIS and ESMPP may be drawn up)
    - Project description (site layouts, designs to feasibility-level detail)

- Analysis of alternatives
- Pre-project environmental and socio-economic baseline
- Assessment of biophysical and socio-economic impacts and identification of mitigation measures
- Record of public consultation
- Environmental and social management plan specifying how the mitigation measures for significant impacts are to be implemented and monitored
- Where resettlement will be required, drawing up a resettlement action plan (RAP)

20.6.3.2 Resettlement Plan Process

The RAP forms part of the resettlement planning process and outlines the procedures and actions that Loncor will follow to enhance development opportunities and to mitigate negative impacts associated with physical and/or economic displacement. In this report, Minecon has employed a conservative approach in the estimation of the social repositioning of the six artisanal settlements that will be impacted by the project. In particular, Minecon has assumed that Loncor may need to relocate artisanal mining huts in these settlements even though this might not turn out to be the case. The legal situation regarding compliance with the DRC resettlement plan process, assessment and reporting requirements is unclear on the repositioning of artisanal mining huts but in the development of the RAP in the Twangiza and Namoya Gold Projects in eastern DRC, artisanal mining huts were not compensated for by Banro Corporation (Banro). Banro provided livelihood restoration to the artisanal miners and relocated only households that were outside the artisanal mining areas but that were impacted by the project. Minecon believes that if the livelihood restoration programmes are accepted for the repositioning of artisanal miners within the Adumbi mine footprint, the capital budget could be significantly lower than that stated in Section 20.7.2. At this early stage, Minecon has also proposed a conservative approach with the construction of a number of communal infrastructures: schools, clinics, churches, etc., which are currently not present in the artisanal communities because the communities are mostly inhabited by transient artisanal miners. Minecon believes that the final budget for the communal infrastructure that would need to be constructed during the projected development could be significantly lower than that stated in Section 20.7.2.

Key aspects of relevance to the resettlement planning and the development of a RAP for Adumbi include the following:

- Indigenous (pygmy) groupings present within and around the project area. Due to their biodiversity reliance (services provided by ecosystems), they are particularly vulnerable to developmental changes in the area.
- Artisanal miners and others settled in homesteads within the project area, with artisanal communities being situated within the footprint of the mine pit/processing plant, namely Mambo Bado, Monde Arabe, Mbuji Mayi, Canal, Adumbi and Mobele Mokonzi. Both economic and physical displacement as a result of the project are sensitive issues that need to be addressed.
- Subsistence agriculture being practised within and around the project site. Information on the extent of crop production within the project area is currently very limited.

20.6.3.3 Pre-Feasibility RAP Framework

At a pre-feasibility level, a RAP framework would normally be prepared as a basis for estimating the costs of implementation. Minecon has already prepared a preliminary framework with some assumptions and estimates having been made at this PEA level. The scope of the preliminary RAP includes the following:

- Broad identification of project affected persons (PAPs)
- Preliminary identification of potential host areas, but no agreement reached with hosts
- Identification of household livelihood strategies and socio-economic status
- Listing in broad terms of the types and levels of household assets and resources that will have to be replaced or compensated for
- Formulation of a draft resettlement and compensation framework with limited policy and procedures
- Preliminary cost estimate for the implementation of a resettlement and compensation programme, which is more than is possible at this stage (see Table 20.1)

20.6.3.4 Feasibility RAP

The detailed RAP would be completed at the feasibility stage, following the completion of the ESIS and ESMPP.

The RAP would cover aspects including the following:

- Identification and description of all categories of affected people at the site and in the proposed service corridors
- Development of an inventory of all types of affected assets at the site and in the proposed service corridors
- Description of the type of compensation and assistance to be provided to affected households
- Presentation of a framework for compensation, including details about the quantity of replacement land required, timing and administration of replacement assets, and compensation payments
- The identification of stakeholders affected by linear infrastructure servicing the project
- Details of valuations and individual landowner agreements

20.7 ESTIMATED COSTS FOR FUTURE WORK

Assumptions and exclusions upon which costing has been developed are provided in Table 20.2.

Table 20.2: Assumptions on which Resettlement Implementation Costs for Adumbi are based

Infrastructure Type	Area (m^2)	Costs (US$)
House Type A	27	10,444.00
House Type B	32	11,276.00
House Type C	40	15,199.00

Infrastructure Type	Area (m^2)	Costs (US$)
House Type D	51	17,040.00
House Type E	73	19,851.00
House Type F	80	22,376.00
House Type G	102	29,724.00
Kitchen	6	3,017.00
Toilet	3.6	1,741.00
School	500	98,686.00
Church	202	44,533.00
Market	446	63,730.00
Clinic	149	45,644.00
Government Office	126	33,792.00

20.7.1 Costs per Project Phase

Outlined in Table 20.3 is a high-level estimate of the costs per project phase for carrying out the PFS, ESIS and ESMPP, development of the RAP report, as well as project management costs.

Table 20.3: Estimated Costs per Project Phases

Component	Low-Range Estimate (US$)	High-Range Estimate (US$)
PFS – Project initiation, screening, scoping and public participation	140,000.00	185,000.00
ESIS and ESMPP – Stakeholder engagement, specialist studies (baselines, impacts and management), ESIS, ESMPP and action plans excluding the RAP, ESIS/ESMPP project management	250,000.00	300,000.00
RAP report development	850,000.00	1,800,000.00

20.7.2 Resettlement Implementation Costs

The estimate for the RAP report development is provided in Table 20.3 while this section deals with the costs associated with the implementation of the RAP. Variables which will influence costing include the following:

- Assets owned by community members
- Cultural heritage, including the presence of graves which may need to be moved
- Number of people to be moved
- Livelihoods of resettled communities, including subsistence agriculture (growing of crops, keeping of livestock and need for grazing land)
- Farming replacement land
- Extent of reliance on biodiversity and ecosystem goods and services resettlement in different communities

- RAP report and community development programme
- Carrying out of relocations and/or compensation

As noted above, the villages identified at this stage of the project as being potentially affected by resettlement are Mambo Bado, Monde Arabe, Mbuji Mayi, Canal, Adumbi and Mobele Mokonzi where both physical and economic displacement could be experienced. Farther villages may be affected, depending on the footprint of the mine and its associated impacts during all the mine phases. Table 20.4 provides an indication of the resources which may need to be allocated for the resettlement of Adumbi and other concerned areas while Table 20.5 outlines the identified social infrastructure and estimated number of households and costs.

Table 20.4: Indication of Resource Allocation Required for Resettlement for Adumbi and Other Areas

Item	Costs (US$)
Operation	
Salaries (four dedicated staff members)	To be determined by Loncor
RAP report development (as per Table 20.3), including public participation materials, public meetings and sitting allowances	US$140 thousand to US$180 thousand, incorporating approximately US$70 thousand to US$80 thousand for public participation
Office/administration	To be determined by Loncor
Construction of replacement dwellings (based on local cost estimate)	See Table 20.5
Relocation of graves and religious ceremonies associated with sacred sites, and landscapes and natural features	To be discussed with relevant stakeholders once ESIS and ESMPP are completed
Compensation	
Compensation for lost crops (based on extent and type of cultivation, and average price for crops)	Unknown – insufficient information on livelihoods and crops per household
Compensation for lost access to fishing/hunting	Unknown – insufficient information on livelihoods and crops per household
Compensation for enterprises	Unknown – insufficient information on livelihoods and crops per household
Compensation for communal infrastructure	See Table 20.5
Transitional support (including removal of households to host site and business loss)	To be discussed with relevant stakeholders
Land Acquisition and Resettlement Site Planning	
Site plan surveying and land use assessment	Depending on the host site selected
Infrastructure	Depending on the host area site selected. Will probably include upgrading of health centre and water provision and additional school
Monitoring and evaluation	Framework to be developed
ESIS and ESMPP Host Areas: Community Development	
Income restoration programmes	To be explored with relevant stakeholders, depending on type of programmes decided upon

Item	Costs (US$)
Agricultural extension	Extent of need for agricultural extension to be determined
Artisanal miner support	To be explored with relevant stakeholders, depending on type of support decided upon
Contingencies (20 % of total budget)	To be calculated once all costs have been estimated

Table 20.5: Costs of Implementing Construction Associated with Social Repositioning of Adumbi

House Type		Total No.	Adumbi	Canal	Mbuji Mayi	Mobele Mokonzi	Monde Arabe	Estimated Cost of Replacement (US$)	Estimated Replacement Costs for Physical Displacement (US$)	Assumptions and Comments
Housing										
Estimated number of households per village		781	93	87	292	281	44			Estimated by counting from six artisanal mining villages
Types of houses	A	530	51	49	178	215	37	10,444.00	5,535,320.00	Estimate based on relative numbers of household type in similar villages elsewhere in the DRC
	B	179	22	26	67	59	5	11,276.00	2,018,404.00	
	C	64		11	45	6	2	15,199.00	972,736.00	
	D	5	4	1	–	–	–	17,040.00	85,200.00	
	E	2	–	–	2	–	–	19,851.00	39,702.00	
	F	1	–	–	–	1	–	22,376.00	22,376.00	
	G	–	–	–	–	–	–	29,724.00	–	
Artisanal and Small-Scale Miners (ASM) Repositioning Projects										
Road Maintenance, Agriculture, Livestock, Skills Development and Training Programmes followed by direct engagement by Loncor, etc.								8,759,692.00	8,759,692.00	Estimate based on relative numbers of ASM in the surrounding villages
Communal Infrastructure										
School		10	1	1	3	4	1	98,686.00	986,860.00	Information provided by Loncor
Church		12	2	1	5	4	–	44,533.00	89,066.00	
Market		1	–	–	1	–	–	63,730.00	–	
Clinic		4	1	–	2	1	–	45,644.00	45,644.00	
Government Office		–	–	–	–	–	–	33,792.00	–	
TOTAL									**18,555,000.00**	

21 CAPITAL AND OPERATING COSTS

21.1 PROJECT REQUIREMENTS

21.1.1 Introduction

The purpose of the capital cost (CAPEX) and operating cost (OPEX) estimates is to provide costs to an accuracy of −35 % to +45 % that can be utilised to assess the economics of the Adumbi deposit when treating 5 Mt/a to target a gold production in excess of 303,000 oz/a.

21.1.2 Scope of the Estimates

The CAPEX and OPEX estimates were developed for a conventional open-pit mining CIL process plant and the supporting infrastructure for an operation capable of treating 5 Mt of ore per annum.

Two options for power supply have been presented for the PEA: a diesel power plant and a hydroelectric power (HEP) hybrid option, consisting of two HEP plants, a solar power generation plant, and a backup diesel power plant with a battery energy storage system (BESS). The CAPEX and OPEX estimates for the two options are presented separately in this section.

21.1.3 Responsibilities

The responsibilities for the CAPEX and OPEX estimates are listed below:

- Process Plant: SENET
- Plant On-Site Infrastructure: SENET
- Plant Off-Site Infrastructure: Loncor/SENET
- Mining: Minecon
- TSF: Epoch
- Environmental Management: Minecon
- Owner's costs: Loncor

21.1.4 Estimate Accuracy

The level of accuracy of the CAPEX and OPEX estimates is within −35 % to +45 % of the overall project costs as of the last quarter of 2021 and does not include any escalation factors.

21.1.5 Contingency

An average contingency of 20 % has been included in the CAPEX to cover items that are included in the scope of work but that cannot be adequately defined at this time due to lack of accurate detailed design information.

No contingency has been allowed for when calculating the OPEX.

21.1.6 Exclusions

The following were not included in the CAPEX and OPEX estimates:

- Scope changes
- Escalation beyond the last quarter of 2021
- Financing costs
- Schedule delays such as those caused by
    - Unidentified ground conditions
    - Labour disputes
    - Environmental permitting activities
    - Receipt of information beyond the control of the EPCM contractor
- Taxes and duties (These have been accounted for in the economic model.)
- Permits
- Sunk costs
- Currency fluctuations
- Force majeure
- Import duties (In the 2018 Mining Code, there are exemptions from duties during the construction period.)

21.1.7 Exchange Rates

The exchange rates given in Table 21.1 were used.

Table 21.1: Exchange Rates

Currency Description	Rate of Exchange
ZAR/US$	15.11
US$/£	1.16
AU$/US$	1.39
US$/C$	0.74
CFA/€	2,260

21.2 CAPITAL COST ESTIMATE

21.2.1 CAPEX Summary

The total estimated cost of bringing the project into production is **US$392,010,000 for the diesel option and US$530,058,000 for the HEP hybrid option**, which is inclusive of contingency and EPCM and is summarised in Table 21.2. This cost excludes any costs for feasibility studies scheduled prior to the start of basic engineering.

The estimated costs are based on the two options that were considered for the project, with the determining factor being the power-supply solution:

- Option 1: Diesel generation power plant
- Option 2: HEP hybrid option, constituting a diesel generation power plant, an HEP plant located at the Ngayu Confluence, a solar photovoltaic (PV) plant and a BESS

Table 21.2: CAPEX Summary

Item	Diesel Option Value (US$ Thousand)	HEP Hybrid Option Value (US$ Thousand)
Mining Development Costs	59,986	59,986
Process Plant	171,369	171,369
Power Plant	14,405	152,452
TSF	65,880	65,880
Infrastructure	27,461	27,461
Access Road	7,800	7,800
Owner's Costs	45,110	45,110
INITIAL CAPITAL	**392,010**	**530,058**

21.2.2 Process Plant, and Associated Infrastructure

The process plant and associated infrastructure capital costs are summarised in Table 21.3.

Table 21.3: Process Plant and Associated Infrastructure CAPEX Estimate Summary

Description	Total CAPEX (US$)
Direct Costs	
Earthworks	9,486,244
Civil Works	11,383,492
Structural Steel	6,830,095
Plate Work	4,553,397
Tankage	5,947,355
Machinery and Equipment	37,944,975
Piping	3,035,598
Valves	758,899

Description	Total CAPEX (US$)
Electricals	9,486,244
Instrumentation	3,415,048
Commissioning Spares	423,719
Vendor Services	1,138,349
Transport	17,992,903
First Fills	1,138,349
Electrical and Instrumentation (E&I) Installation	5,805,581
Structural, Mechanical, Piping, and Plate Work (SMPP) Installation	26,581,644
Subtotal	**145,921,892**
Associated Infrastructure	
Earthworks	893,661
Civil Works	488,448
Infrastructure buildings (block buildings)	3,250,000
Infrastructure buildings (structural steel buildings)	574,645
Electrical (supply)	893,661
Transport	220,246
Installation (steel structure and electrical)	660,738
Subtotal	**6,981,399**
Other Costs	
2-Year Operational Spares	2,118,594
Insurance and Critical Spares	2,542,313
Subtotal	**4,660,908**
Management Costs	
Plant and associated Infrastructure EPCM Costs	20,786,060
Subtotal	**20,786,060**
TOTAL	**178,350,259**

21.2.2.1 Process Plant Basis of Estimate and Assumptions

The estimate was based on the design basis, block flow diagram, mass balance and process flow diagrams (PFDs), from which a mechanical equipment list was developed, and equipment was sized including supporting infrastructure. Equipment supply pricing was sourced from vendors via budget pricing enquiries.

Other costs including structural steel, plate work, pipework, electrical and instrumentation, mechanical equipment installation, earthworks and civil works, EPCM services and other infrastructure were estimated by factorisation using standard industry factors for similar plants.

The following assumptions were made in the preparation of this estimate:

- The work will be executed as a single EPCM contract.
- The site work will be continuous and not be constrained by the Owner.

- The construction schedule will run approximately 24 months.
- The chosen site is suitable for the foundations, and there are no specific problems due to excess precipitation or groundwater, and no rock excavation will be required during excavation.
- Piling will not be required.
- Construction will not be in an earthquake zone.

21.2.2.2 Infrastructure Buildings

The CAPEX of the infrastructure buildings includes the following:

- Plant buildings
- Pre-engineered steel buildings

The civil works and earthworks CAPEX associated with the plant infrastructure buildings as described above has been included within the infrastructure CAPEX section.

21.2.2.2.1 Plant Buildings

The following process plant buildings are envisaged for the project:

- Change house block
- Plant office building
- Gatehouse building
- Plant laboratory
- Clinic
- Administration building

The plant buildings costs are based on obtaining quotes from local contractors in the area. The quotes were interrogated and benchmarked against costs from previously executed projects.

The buildings are envisaged to be constructed utilising local labour and materials, which will provide the local people with an opportunity for employment.

21.2.2.2.2 Pre-Engineered Steel Buildings

The following pre-engineered steel buildings will be supplied:

- Plant Workshop
- Plant Warehouse
- Reagent stores

The pre-engineered steel buildings package includes the design, fabrication and supply of the steel buildings as described above. Site installation of the buildings and all internal electrical reticulation and fittings are included in this package.

Supply and installation costs were derived utilising SENET's in-house database and benchmarked against similar, previously executed projects.

21.2.3 Other Infrastructure

Other infrastructure includes the following:

- Mining development costs
- Access road between the Bafwabango village and the project site
- TSF
- Off-site power supply and distribution
- Senior expatriates' camp
- Junior camp
- Owner's costs

The other infrastructure CAPEX estimate is given in Table 21.4.

Table 21.4: Other Infrastructure CAPEX

Description	Diesel Option Total Cost (US$ Thousand)	HEP Hybrid Option Total Cost (US$ Thousand)
Mining Development Costs	59,986	59,986
Initial TSF	65,880	65,880
Diesel Power Plant (Option 1)	15,780	
HEP Hybrid Power Plant (Option 2)		152,452
Senior Expatriates' and Junior Camps	20,479	20,479
Access Transport Road	7,800	7,800
Total Initial CAPEX	**169,925**	**306,597**

21.2.3.1 Mining Development Costs

The mining capital costs for the project are US$59,986,000. These costs include contractor mobilisation, haul road construction, clearing and grubbing, topsoil stripping, mine development, and the purchase of survey equipment and mine design software.

21.2.3.2 Access Road between the Bafwabango Village and the Project Site

There is currently a gravel road accessing the site area. Loncor engaged with a local DRC engineer to assess the works required to upgrade the access road to allow suitable access to the project site.

It was deemed that the road between Bafwabango and the project site would require rehabilitation, and costs were estimated for inclusion into the CAPEX.

21.2.3.3 TSF

To estimate the CAPEX associated with the TSF, a high-level bill of quantities (BOQ) was generated in order to determine the quantities for certain schedule items, mainly pertaining to site clearance, earthworks and drainage items such as HDPE lining and bidim® layers. All the rates pertaining to these items were obtained from a recent project currently under construction, situated in the same region of the DRC as the Imbo Project. The CAPEX for the TSF has been determined to a Class 4 Association for the Advancement of Cost Engineering (AACE) accuracy (+50 % to −15 %). In accordance with the current study, the BOQ was limited to a number of measured line items. The cost of the non-measured items was approximated as a ratio of the measured items.

The measured items include the following:

- Turret pumping system: utilised on the TSF to allow for pumping from the TSF to the RWD
- TSF wall quantities, assuming the use of borrow material for Phase 1
- TSF wall quantities, assuming the use of a combination of borrow and waste rock material for Phases 2 and 3
- Excavating, loading and hauling quantities (The mining fleet will excavate, load and haul the required waste rock material to the TSF during this time, and the contractor will only be required to place the material.)
- TSF HDPE liner and geotextile costs
- Clearance of the TSF area

For CAPEX purposes, the non-measured items include but are not limited to the following:

- Additional drains for the TSF, such as
  - Under drains
  - Toe drains
  - Curtain drains
- Slurry delivery lines
- Concrete works
- Spillway construction

21.2.3.4 Off-Site Power Supply and Distribution

A desktop study was undertaken by DRA Energy, South Africa, assessing potential hydroelectric, diesel and PV power sources for Adumbi. Knight Piésold Ltd from South Africa also undertook a desktop study on a number of potential hydroelectric power sites in and around the Adumbi deposit area and was part of the team of engineers from South Africa who visited potential sites on the ground. The total installed power required for the Adumbi deposit is estimated at 32 MW.

Table 21.5 indicates the different priority power generation options that were investigated.

Table 21.5: Priority Power Options for the Imbo Project

Option	Power Option	Power Plant CAPEX (US$)	Power Cost (US$/kWh)	Approximate Distance from Plant Site (km)
1	Diesel Only 32 MW	15,708,000	0.2768	0
2	Hybrid, 32 MW Diesel, 20 MW PV, 2.5 MW/3.7 MWh BESS	36,845,015	0.2459	0
3	Imbo Upper Site 3, 3.7 MW HEP, 32 MW Diesel, 20 MW PV, 12.4 MW/18.4 MWh BESS	73,593,075	0.2133	5
4	Ngayu Confluence, 16.3 MW HEP, 32 MW Diesel, 20 MW PV, 12.4 MW/18.4 MWh BESS	138,593,075	0.1201	23

For the Adumbi deposit PEA, two financial model options were examined: the diesel-only option (Option 1 in Table 21.5) of generating power for essential processing plant equipment and infrastructure; and the HEP hybrid option (Option 4 in Table 21.5), which is a hybrid system consisting of HEP supplemented by diesel and solar PV power generation with a BESS. Although the capital costs are higher for the HEP hybrid case, operating costs, especially processing power costs, are significantly reduced, and subsequent project economics are enhanced compared to the diesel-only power generation option.

It was assumed that the capital cost for the HEP option would be funded by Loncor. However, Loncor is already in discussion with potential power suppliers with experience in the DRC to project finance and build an HEP facility near the Adumbi deposit area and then have an offtake agreement with Loncor to supply power for the operation. Any hydroelectric power scheme could also have the potential to obtain carbon credits.

21.2.3.5 Senior Expatriates' Camp

The expatriates and local managers' camp will be constructed of plastered blockwork and will consist of the following:

- 75 rooms for technical staff (including visitors), and 10 rooms for directors (including VIP visitors)
- Entertainment club
- Restaurant and kitchen
- Laundry
- Gymnasium
- Gate and gatehouse for security control

The buildings costs are based on obtaining quotes from local contractors in the area. The quotes were interrogated and benchmarked against costs from previously executed projects.

The buildings are envisaged to be constructed utilising local labour and materials, which will provide the local people with an opportunity for employment.

21.2.3.6 Junior Camp

The junior camp will be constructed of plastered blockwork and will consist of the following:

- Housing for local technicians consisting of 108 rooms
- Gymnasium
- Gate and gatehouse for security control

The buildings costs are based on obtaining quotes from local contractors in the area. The quotes were interrogated and benchmarked against costs from previously executed projects.

The buildings are envisaged to be constructed utilising local labour and materials, which will provide the local people with an opportunity for employment.

21.2.3.7 Owner's Costs

21.2.3.7.1 Pre-Production Costs

The owner's pre-production costs are based on the costs that will be incurred from the start of the project implementation phase, up to the commissioning and handover to plant operations. A cost of US$45,110,000 has been estimated, based on plants of a similar throughput, and benchmarked against previously executed projects.

21.2.3.7.2 Sustaining Capital

Sustaining capital is defined as the capital expenditure occurring beyond the initial period leading to gold production and including the purchase of new equipment for the support of mining operations, haul roads, pit dewatering, replacement of mining pickup trucks, camp maintenance, and the process plant replacement equipment. Table 21.6 and Table 21.7 show a summary of the sustaining capital to be incurred during the LOM for the diesel-only option and the HEP hybrid option, respectively.

Table 21.6: Sustaining Capital for Diesel-Only Option

Description	Total CAPEX (US$ Thousand)
Mining Capitalised Waste	328,215
Power Plant	14,122
TSF	66,329
Rehabilitation and Closure Costs	30,678
Equipment Salvage Value	−28,789
Total Sustaining Capital	**410,556**

Table 21.7: Sustaining Capital for HEP Hybrid Option

Description	Total CAPEX (US$ Thousand)
Mining Capitalised Waste	328,215
Power Plant	0
TSF	66,329
Rehabilitation and Closure Costs	30,678
Equipment Salvage Value	−120,260
Total Sustaining Capital	**304,962**

21.2.3.7.3 Working Capital

Working capital is defined as those fixed and variable costs incurred by the mine from commissioning to the point where the realised revenue from the gold sales can pay for the entire mine’s operating costs.

An initial working capital of US$11,687,000 has been estimated, based on plants of a similar throughput, and benchmarked against previously executed projects.

21.3 OPERATING COST ESTIMATE

21.3.1 OPEX Summary

The purpose of this OPEX estimate is to provide the operating costs, and the associated general and administration (G&A) costs, that will be utilised for the economic analysis of Adumbi. Two OPEX options for the process plant were developed, using diesel and HEP hybrid options as the source of power.

The project's annual OPEX estimate for the LOM consists of the following:

- Mining operating costs estimated by Minecon
- Process plant operating costs estimated by SENET
- TSF operating costs estimated by Epoch
- Site G&A costs estimated by SENET
- Bullion transport, insurance and refining costs estimated by SENET

The overall OPEX summaries for the Adumbi deposit for both the diesel and the HEP hybrid options are given in Table 21.8 and Table 21.9, respectively, with the respective cost distributions shown in Figure 21.1 and Figure 21.2.

Table 21.8: Overall Plant OPEX Summary for Diesel Option

Description	LOM		
	US$/t processed	US$/oz	Distribution (%)
Mining	31.33	499.56	55.01
Processing	17.95	286.21	31.52
TSF	0.55	8.75	0.96
G&A	3.40	54.19	5.97
Refining and Transport	0.22	3.50	0.39
Royalties	3.51	55.96	6.16
Total	**56.95**	**908.17**	**100**

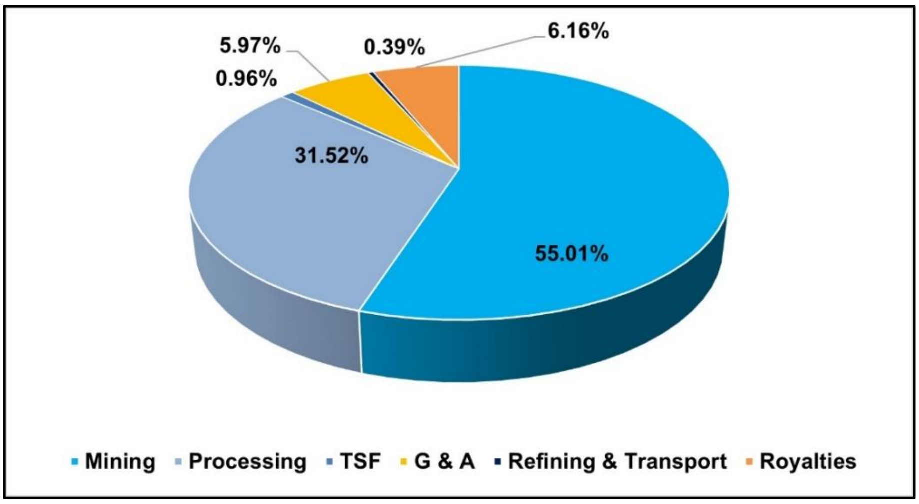


Figure 21.1: OPEX Summary Distribution for Diesel Option

Table 21.9: Overall Plant OPEX Summary for HEP Hybrid Option

Description	LOM		
	US$/t processed	US$/oz	Distribution (%)
Mining	31.33	499.56	58.62
Processing	14.44	230.22	27.02
TSF	0.55	8.75	1.03
G&A	3.40	54.19	6.36
Refining and Transport	0.22	3.50	0.41
Royalties	3.51	55.96	6.57
Total	**53.44**	**852.18**	**100**

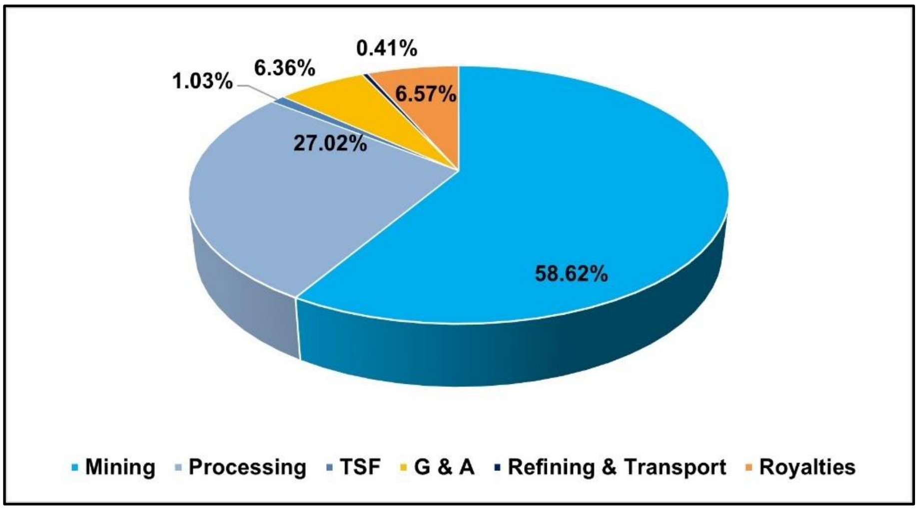


Figure 21.2: OPEX Summary Distribution for HEP Hybrid Option

21.3.2 Exchange Rates

The costs of the project are reported in the United States dollar. The exchange rates used are given in Table 21.1.

21.3.3 Process Plant Operating Costs

21.3.3.1 Basis of Estimate

The process plant operating costs were compiled from a variety of sources, notably the following:

- First principles, where applicable
- Supplier quotations on reagents and consumables
- SENET's in-house experience and database where applicable
- Client input
- Test work

The following are the main cost elements of the process plant:

- Reagents and consumables
- Power
- Process plant operating and maintenance labour
- Maintenance parts and supplies
- Assay
- General and administration (G&A)

21.3.3.2 Process Plant OPEX Summary

The overall process plant OPEX is summarised in Table 21.10 for the diesel power option and in Table 21.11 for the HEP hybrid power option. The distribution of the costs is shown in Figure 21.3 for the diesel power option and in Figure 21.4 for the HEP hybrid power option.

Table 21.10: Process Plant OPEX Summary – Diesel Power Option

Description	Oxide Ore Cost		Transition Ore Cost		Sulphide Ore Cost	
	US$/a	US$/t	US$/a	US$/t	US$/a	US$/t
Labour	4,244,072	0.85	4,244,072	0.85	4,244,072	0.85
Power	34,741,595	6.95	39,632,623	7.93	45,348,999	9.07
Maintenance	2,276,698	0.46	2,276,698	0.46	2,276,698	0.46
Assay	840,000	0.17	840,000	0.17	840,000	0.17
Consumables	5,706,788	1.14	6,770,895	1.35	11,572,564	2.31
Reagents	31,082,906	6.22	34,643,742	6.93	34,677,268	6.94
TSF	2,732,486	0.55	2,732,486	0.55	2,732,486	0.55
TOTAL	**81,624,546**	**16.32**	**91,140,516**	**18.23**	**101,692,087**	**20.34**

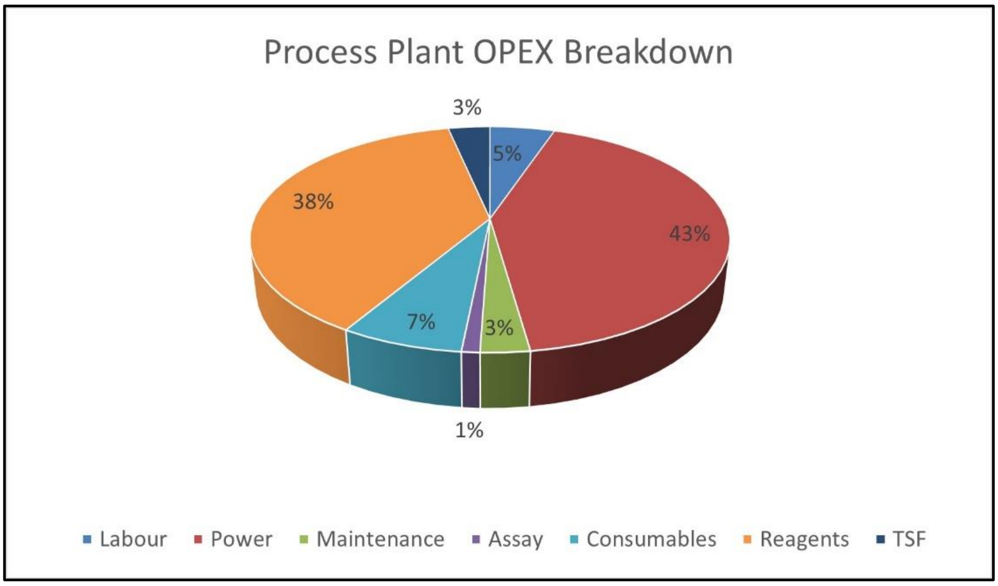


Figure 21.3: Process Plant OPEX Summary – Diesel Power Option

Table 21.11: Process Plant OPEX Summary – HEP Hybrid Power Option

Description	Oxide Ore Cost		Transition Ore Cost		Sulphide Ore Cost	
	US$/a	US$/t	US$/a	US$/t	US$/a	US$/t
Labour	4,244,072	0.85	4,244,072	0.85	4,244,072	0.85
Power	15,075,482	3.02	17,197,854	3.44	19,678,371	3.94
Maintenance	2,276,698	0.46	2,276,698	0.46	2,276,698	0.46
Assay	840,000	0.17	840,000	0.17	840,000	0.17
Consumables	5,706,788	1.14	6,770,895	1.35	11,572,564	2.31
Reagents	31,082,906	6.22	34,643,742	6.93	34,677,268	6.94
TSF	2,732,486	0.55	2,732,486	0.55	2,732,486	0.55
TOTAL	**61,958,432**	**12.39**	**68,705,747**	**13.74**	**76,021,460**	**15.20**

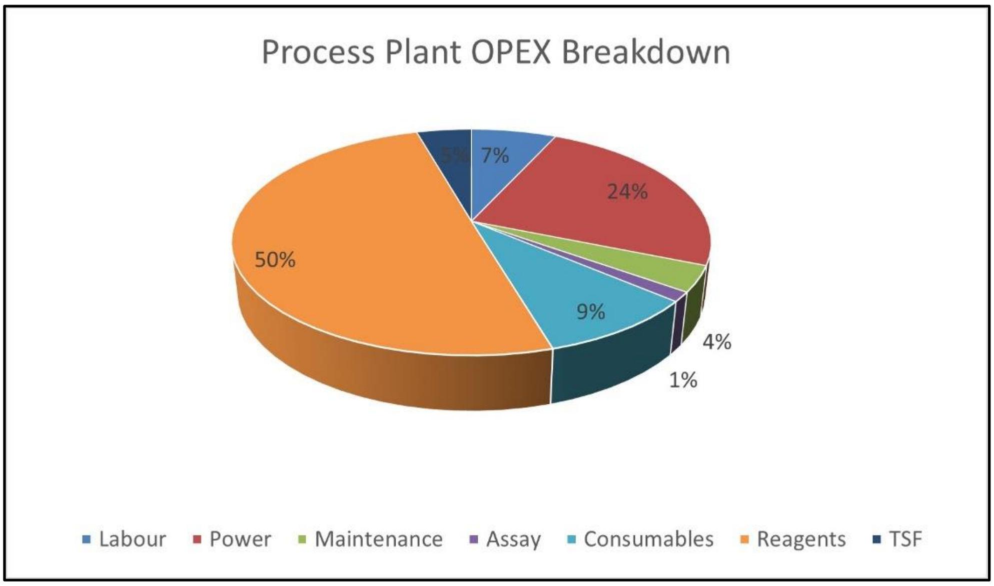


Figure 21.4: Process Plant OPEX Summary – HEP Hybrid Power Option

21.3.3.3 Reagents and Consumables

The reagents and consumables costs were calculated using vendor supply prices and the consumptions of the respective reagents, based on the test work results or consumables estimated by OMC. The reagents and consumables supplied costs are shown in Table 21.12.

Table 21.12: Reagents and Consumables Supplied Costs

Description	Unit	Cost
Grinding Media – Primary SAG Mill	US$/t	1,494
Grinding Media – Secondary Ball Mill	US$/t	1,495
Crusher Liners – Stationary Jaw	US$/set	21,333
Crusher Liners – Swing Jaw	US$/set	19,123

Description	Unit	Cost
Mill Liners – Primary SAG Mill	US$/set	162,025
Mill Liners – Secondary Ball Mill	US$/set	162,025
Pebble Crusher Liners	US$/set	25,060
Activated Carbon	US$/t	3,640
Copper Sulphate	US$/t	3,255
Ferrous Sulphate	US$/t	700
Hydrochloric Acid	US$/t	1,040
Lime – Hydrated	US$/t	510
Silica	US$/t	640
Sodium Borate (Borax)	US$/t	1,650
Sodium Carbonate	US$/t	720
Sodium Cyanide	US$/t	2,640
Sodium Hydroxide (Caustic)	US$/t	1,050
Sodium Metabisulphite (SMBS)	US$/t	1,040
Antiscalant	US$/t	2,590
Sulfamic Acid	US$/t	2,050
Sodium Hypochlorite	US$/t	2,040

The reagents and consumables consumptions and operating costs are provided in Table 21.13.

Table 21.13: Reagents and Consumables Consumptions and Cost Summary

Description	Consumption	Cost
	t/a	US$/a
Grinding Media – Primary SAG Mill	110	201,236
Grinding Media – Secondary Ball Mill	2,515	4,603,498
Crusher Liners – Stationary Jaw	5.7	62,576
Crusher Liners – Swing Jaw	3.9	34,508
Mill Liners – Primary SAG Mill	55.5	380,803
Mill Liners – Secondary Ball Mill	61.8	424,167
Pebble Crusher Liners	15	172,365
Activated Carbon	269	455,000
Copper Sulphate	316	1,133,138
Ferrous Sulphate	432	302,386
Hydrochloric Acid	229	238,255
Lime – Hydrated	18,200	9,282,000
Silica	4	2,586
Sodium Borate (Borax)	8	13,334
Sodium Carbonate	4	2,909

Description	Consumption	Cost
	t/a	US$/a
Sodium Cyanide	4,951	13,071,594
Sodium Hydroxide (Caustic)	269	282,449
Sodium Metabisulphite (SMBS)	4,514	6,253,099
Antiscalant	3	8,936
Sulfamic Acid	17	34,146
Sodium Hypochlorite	2	3,435

21.3.3.3.1 Crusher Liners

The primary crusher and pebble crusher liner costs were obtained from vendor information, by estimating the number of liner changes per annum using the abrasion indices obtained from metallurgical tests and the expected liner life for a given throughput. Quotations for the crusher liners, including the weights of the liners, were obtained from a vendor.

21.3.3.3.2 Mill Liners

The SAG and ball mill liner costs were based on estimating the liner consumption by using the abrasion index results obtained from test work. OMC used the test work data to simulate the expected wear rates. This was further cross-referenced by wear rates from other operating mines using the same type of liners and grinding media size based on SENET's experience. The current pricing for a set of rubber liners for the ball mill was obtained from a liner supply vendor and used in the cost estimate based on the number of liner changes per annum for a given throughput. The delivered costs were estimated using the customs clearance, port handling and transport costs to site.

21.3.3.3.3 Mill Grinding Media

The grinding media costs were obtained by estimating the consumption in the SAG and ball mills based on Bond's estimating method and using the standard method abrasion index results that were obtained from laboratory tests. OMC used the test work data to estimate the expected mill grinding media consumptions. In addition, the mill throughputs and quotations for mill balls were obtained from suppliers. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the grinding media costs.

21.3.3.3.4 Hydrated Lime

Hydrated lime consumption in CIL leaching was estimated from test work. The hydrated lime consumption for cyanide destruction was estimated from test work results and industry practice. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the quicklime costs.

21.3.3.3.5 Sodium Cyanide (Cyanide)

The cyanide consumptions for the CIL and ILR circuits were estimated from test work results. The cyanide consumption associated with elution was calculated from first principles, taking into account the designed consumption usage in these three circuits over a given period, and was then used, together with quotations obtained from reagent suppliers, to estimate the delivered cyanide costs.

21.3.3.3.6 Activated Carbon

Carbon consumption was based on industry practice for CIL plants. Budget quotations obtained from suppliers were then used to estimate the carbon costs per annum for a given throughput.

21.3.3.3.7 Copper Sulphate

The amount of copper sulphate used for the purposes of cyanide destruction in the plant tails slurry was estimated from test work results and first principles. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the copper sulphate costs.

21.3.3.3.8 Sodium Metabisulphite

The amount of sodium metabisulphite used for the purposes of cyanide destruction in the plant tails slurry was estimated from test work results and first principles. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the sodium metabisulphite costs.

21.3.3.3.9 Ferrous Sulphate

The amount of ferrous sulphate used for the purposes of arsenic precipitation in the detoxed plant tails slurry was estimated from test work results and first principles. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the ferrous sulphate costs.

21.3.3.3.10 Hydrochloric Acid

The hydrochloric acid (33 %) consumption associated with acid washing was calculated from first principles, taking into account the designed number of acid washes and expected acid strength in the solution. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the hydrochloric acid costs.

21.3.3.3.11 Sodium Hydroxide (Caustic)

Caustic consumption associated with elution/electrowinning and intensive cyanidation was calculated from first principles, taking into account the following parameters:

- Designed number of elutions
- Volume of gravity concentrates to be treated
- Expected caustic strength in these solutions

Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the caustic costs.

21.3.3.3.12 Smelting Reagents

Silica, sodium borate (borax) and sodium carbonate are reagents used in the smelting process.

The reagents consumptions were determined from first principles using the number of smelts per month, a flux to sludge ratio of 1:1, and the following flux ratios:

- Borax: 50 %
- Silica: 25 %
- Sodium carbonate: 25 %

Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the smelting reagents costs per annum for a given throughput.

21.3.3.3.13 Antiscalant

Antiscalant (Scaletrol PDC9339) consumption associated with elution was calculated from first principles, taking into account the designed number of elutions to reduce scaling in the heat exchangers. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the antiscalant costs.

21.3.3.3.14 Sulfamic Acid (Descalant)

Sulfamic acid consumption for descaling the elution heat exchanger was estimated from first principles, taking into account the solution's fouling nature. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the descalant costs.

21.3.3.3.15 Sodium Hypochlorite

Sodium hypochlorite consumption in the potable water treatment plant was estimated from first principles. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the hypochlorite costs.

21.3.3.4 Power

The power consumption can be categorised into two forms: fixed and variable power. The average continuous fixed power consumption was determined by taking into account the installed power rating of each of the equipment in the plant and infrastructure, excluding standbys and the projected running times. The fixed power draw includes the absorbed operating loads associated with the process plant equipment as detailed in the mechanical equipment list and on-site infrastructure, including the following buildings:

- Sewage Treatment Plant
- Fuel Farm
- Change House
- First-Aid Building
- Plant Offices
- Assay Laboratory

- Administration Building
- Weighbridge
- Control Room
- Gatehouse
- Warehouse
- Reagents Stores
- Workshop

The variable power was estimated using the gross specific energy for the crushers and mills reported by OMC as detailed in the process design criteria. The specific energy was used with the crushing and milling throughput to calculate the annual power usage (kilowatt hours per annum).

The power draw and power costs for the process plant and infrastructure were determined using the power consumption basis detailed above and the unit energy cost of US$0.270/kWh.

Table 21.14 and Table 21.15 show the power draw summaries for fixed and variable power, respectively.

Table 21.14: Power Draw Summary – Fixed Power

Process Area	Process Area Description	Oxide			Transition/Sulphide		
		Installed Power	Operating Power	Power Consumption	Installed Power	Operating Power	Power Consumption
		kW	kW	kWh/a	kW	kW	kWh/a
1000	Infrastructure	409	186	1,154,101	409	186	1,154,101
2100	Primary Crushing	767.7	422.96	2,534,706	768	423	2,534,706
2300	Mill Feed Bin and Emergency Stockpile	171.5	137.2	627,510	172	137	627,510
2400	Pebble Crushing				206.7	161.36	289,888
3100	Milling	18,324	13,864	5,657,700	18,324	13,864	5,657,700
3300	Gravity and Intensive Cyanidation	135.5	102.45	669,309	136	102	669,309
3500	Trash Handling	67	53.6	313,696	67	54	313,696
4100	CIL	2,512	2,009.6	11,711,676	2,512	2,010	11,711,676
5100	Carbon Safety and Detoxification	1,067	558	4,383,600	1,067	558	4,383,600
5200	Arsenic Precipitation	264	211.2	1,689,600	264	211	1,689,600
5500	Tailings Dam and Return Water	338	137.6	1,100,800	338	138	1,100,800
6100	Acid Wash	54.2	22.56	79,802	54	23	79,802
6200	Elution	4,858	2,908.8	12,760,164	4,858	2,909	12,760,164
6300	Electrowinning	142	107.6	820,088	142	108	820,088
6400	Carbon Reactivation	120.55	66.84	447,480	121	67	447,480
6500	Gold Room	404.4	323.52	748,979	404	324	748,979
7100	Cyanide and Caustic	107.6	54.08	143,836	108	54	143,836
7200	Lime	264.4	133.92	896,880	264	134	896,880
7400	Detoxification and Arsenic Precipitation Reagents	48.4	33.88	89,632	48	34	89,632
8100	Compressed and Blower Air and Oxygen Plant	2,466	1,424.8	11,398,400	2,466	1,425	11,398,400
8300	Raw Water Supply	440	180	1,440,000	440	180	1,440,000
8400	Process Water	620	248	1,984,000	620	248	1,984,000
8500	Raw Water Storage	484	267.6	1,006,285	484	268	1,006,285
8600	Potable Water	91	32.4	259,200	91	32	259,200
8700	Gland Seal Water	82	32.8	262,400	82	33	262,400
	Total	**34,238**	**23,519**	**62,179,845**	**34,445**	**23,681**	**62,469,733**

Table 21.15: Power Draw Summary – Variable Power

Process Area	Description	Oxide				Transition				Sulphide			
		Installed Power	Specific Energy	Operating Power	Power Consumption	Installed Power	Specific Energy	Operating Power	Power Consumption	Installed Power	Specific Energy	Operating Power	Power Consumption
		kW	kWh/t	kW	kWh/a	kW	kWh/t	kW	kWh/a	kW	kWh/t	kW	kWh/a
2200	Primary Crusher	250	0.09	200	450,000	250	0.10	200	500,000	250	0.13	200	650,000
2400	Pebble Crusher					132	0.10	105.6	83,000	132	0.10	105.6	83,000
3100	Primary SAG Mill	7,400	4.2	5,920	21,000,000	7,400	5.7	5,920	28,500,000	7,400	8.2	5,920	41,000,000
3100	Secondary Ball Mill	9,000	8.6	7,200	43,000,000	9,000	10.4	7,200	52,000,000	9,000	12	7,200	60,000,000
Total		**16,650**	**12.89**	**13,320**	**64,450,000**	**16,782**	**16.30**	**13,426**	**81,083,000**	**16,782**	**20.43**	**13,426**	**101,733,000**

21.3.3.5 Plant Operating and Maintenance Labour

The annual plant operating and maintenance labour cost was estimated at **US$4,244,072**. The cost was derived from first principles and SENET's in-house database based on SENET's experience in a similar region where the actual labour complement for each plant area and maintenance function was identified, and the required number of personnel and their levels were established.

The operating and maintenance labour cost was broken down as follows:

- Expatriate Personnel:
    - Labour remuneration
    - Medical
    - Income tax as a percentage of net salary
    - In-country taxation
    - Social security
- National Personnel:
    - Labour remuneration
    - Medical
    - Thirteenth cheque
    - Social security
    - Vacation/sick leave
    - Housing allowance

The following costs were excluded as they have been captured in the G&A operating costs:

- Camp food and catering facility
- Expatriate travel
- Safety supplies
- Training
- Consultants' fees

The categories selected for the various positions are described in Table 21.16.

Table 21.16: Process Plant Labour Categories

Description	Employee Category
Expatriates	Ex
Plant Manager	A10
Metallurgist	A5
Metallurgical Foreman/Trainer	A2
Supervisor	B7
Artisan	C
Operator	D1
Attendant	E3

Table 21.17 shows the labour cost summary.

Table 21.17: Process Plant Labour Cost Summary

Position Description	Total Number of Employees	Total Cost	Total Cost
		US$/month	US$/a
Plant Management	10	41,579	498,942
Maintenance Management	16	55,100	661,205
Subtotal – Management	**26**	**96,679**	**1,160,147**
Operational Labour	109	102,615	1,231,380
Plant Maintenance Labour	97	133,419	1,601,028
TSF and Return Water Ponds Operators	27	20,960	251,517
Subtotal – Plant, Maintenance and TSF Labour	**233**	**256,994**	**3,083,925**
Total	**259**	**353,673**	**4,244,072**

The detailed breakdown is given in Table 21.18.

Table 21.18: Detailed Breakdown of Labour

Position Description	Employee Category	Total Number of Employees	Rate/ Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
Plant Management					
Process Manager	Ex	1	14,547	14,547	174,564
Metallurgical Superintendent	Ex	1	7,440	7,440	89,280
Metallurgical Superintendent	Ex	1	7,440	7,440	89,280
Senior Metallurgist	Ex	1	7,440	7,440	89,280
Plant Metallurgist	A2	1	1,386	1,386	16,632
Cyanide Champion	B7	1	851	851	10,206
Metallurgical Safety Officer	B7	1	851	851	10,206
Metallurgical Training Officer	B7	1	851	851	10,206
Metallurgical Training Assistant	D1	1	387	387	4,644
Metallurgical Statistician	D1	1	387	387	4,644
Subtotal – Plant Management		**10**	**41,579**	**41,579**	**498,942**
Plant Operational Labour					
Laboratory					
Laboratory Technician	A2	1	1,386	1,386	16,632
Laboratory Technician	A2	8	2,820	22,558	270,700
Shift Operator	D1	8	787	6,299	75,585
Shift Attendant	E3	8	528	4,224	50,683
Reagents					
Supervisor	B7	1	851	851	10,206
Operator/Driver	D1	18	787	14,172	170,066

Position Description	Employee Category	Total Number of Employees	Rate/ Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
Attendant	E3	6	528	3,168	38,012
Attendant	E3	5	528	2,640	31,677
Gold Room					
Metallurgical Foreman	A2	1	1,386	1,386	16,632
Operator	D1	1	387	387	4,644
Attendant	E3	4	260	1,038	12,456
Tailings Storage Facility (TSF)					
Metallurgical Foreman	A2	1	1,386	1,386	16,632
Shift Supervisor	B7	4	1,730	6,921	83,056
Operator	D1	4	787	3,149	37,793
Attendant	E3	18	528	9,503	114,037
Crushing					
Shift Supervisor	B7	4	1,730	6,921	83,056
Shift Attendant	E3	4	528	2,112	25,342
Shift Attendant	E3	8	260	2,076	24,912
Milling					
Shift Supervisor	B7	4	1,730	6,921	83,056
Shift Operator	D1	4	787	3,149	37,793
Shift Attendant	E3	4	528	2,112	25,342
Shift Attendant	E3	4	528	2,112	25,342
CIL					
Shift Supervisor	B7	4	1,730	6,921	83,056
Shift Supervisor	B7	4	1,730	6,921	83,056
Shift Operator	D1	4	787	3,149	37,793
Shift Attendant	E3	4	528	2,112	25,342
Subtotal – Plant Operational Labour		**136**	**25,546**	**123,575**	**1,482,897**
Plant Maintenance Management					
Engineering Manager	Ex	1	14,547	14,547	174,564
Maintenance Engineer Process	Ex	1	7,908	7,908	94,896
Senior Training Officer	Ex	1	5,347	5,347	64,164
Training Officer	A2	2	1,386	2,772	33,264
Training Assistant	C	2	785	1,569	18,833
Planning Coordinator	Ex	1	5,347	5,347	64,164
Maintenance Superintendent Mining	Ex	1	7,908	7,908	94,896
IT Coordinator	A2	2	1,386	2,772	33,264
Planner (Process Maintenance)	A2	2	1,386	2,772	33,264
Planner (Mining Maintenance)	A2	1	1,386	1,386	16,632
Mechanical Engineer	A2	1	1,386	1,386	16,632

Position Description	Employee Category	Total Number of Employees	Rate/ Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
Electrical Engineer	A2	1	1,386	1,386	16,632
Subtotal – Plant Maintenance Management		**16**	**50,158**	**55,100**	**661,205**
Outside Services					
Fitter	Ex	1	5,347	5,347	64,164
Boilermaker	Ex	1	5,347	5,347	64,164
Rigger	Ex	1	5,347	5,347	64,164
Light Vehicle Supervisor	Ex	1	5,347	5,347	64,164
Heavy Machine Supervisor	Ex	1	5,347	5,347	64,164
Fitter Operative	D1	2	520	1,041	12,491
Fitter Assistant	E3	2	260	519	6,228
Boilermaker Operative	D1	3	520	1,561	18,736
Mobile Crane Operator	A2	3	1,386	4,158	49,896
Rigger Assistant	E3	3	260	779	9,342
Auto-Electrical Operative	D1	1	520	520	6,245
Four-Track Operative	D1	1	520	520	6,245
Tyreman	E3	1	260	260	3,114
Apprentice	D1	2	520	1,041	12,491
Heavy Machine Operative	D1	2	520	1,041	12,491
Service Operative	D1	2	520	1,041	12,491
Driver	D1	2	520	1,041	12,491
Mechanic Assistant	E3	2	260	519	6,228
Plumber	D1	1	520	520	6,245
Plumber Mason Assistant	E3	3	260	779	9,342
Electrical					
Electrical Foreman	Ex	1	5,830	5,830	69,960
Electrician	C	4	785	3,139	37,666
Electrical Operatives	D1	4	520	2,082	24,982
Electrical Assistant	D1	4	520	2,082	24,982
Air Conditioner Technician	Ex	1	5,347	5,347	64,164
Air Conditioner Technical Assistant	D1	1	520	520	6,245
Scheduler	C	1	785	785	9,416
Instrumentation					
Instrumentation Foreman	Ex	1	5,830	5,830	69,960
Operative	D1	2	520	1,041	12,491
Assistants	E3	2	260	519	6,228
Process Plant					
Mechanical Foreman	Ex	1	5,830	5,830	69,960

Position Description	Employee Category	Total Number of Employees	Rate/ Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
Fitter	Ex	1	5,347	5,347	64,164
Boilermaker	Ex	1	5,347	5,347	64,164
Fitter Operative	D1	1	520	520	6,245
Fitter Assistant	E3	2	260	519	6,228
Boilermaker Operative	D1	1	520	520	6,245
Boilermaker Assistant	E3	2	260	519	6,228
Rubber Liner Installer	D1	4	520	2,082	24,982
Scheduler	A2	1	1,386	1,386	16,632
Fitter	Ex	2	5,347	10,694	128,328
Boilermaker	Ex	2	5,347	10,694	128,328
Fitter Operative	D1	3	520	1,561	18,736
Greaser	E3	1	260	260	3,114
Fitter Assistant	E3	2	260	519	6,228
Boilermaker Operative	D1	4	520	2,082	24,982
Fitter	Ex	1	5,347	5,347	64,164
Boilermaker	Ex	2	5,347	10,694	128,328
Boilermaker Operative	D1	1	520	520	6,245
Boilermaker Assistant	E3	1	260	260	3,114
Tower Crane Operator	D1	4	520	2,082	24,982
Rigger Assistant	E3	4	260	1,038	12,456
Subtotal – Plant Maintenance Labour		**97**	**99,518**	**137,070**	**1,601,028**
Total		**259**	**216,801**	**357,324**	**4,244,072**

21.3.3.6 Maintenance Parts and Supplies

The plant maintenance parts and supplies annual costs were estimated at **US$2,246,316** when treating oxide ore and **US$2,276,698** when treating transition and sulphide ores. Plant maintenance and supplies costs refer to the costs of operating spares, lubricants, and other maintenance-related consumables for the plant. It has been assumed that the plant will experience a moderate amount of wear.

An average annual maintenance cost was calculated using the maintenance cost factor of 5 % as shown in Table 21.19.

Table 21.19: Plant Maintenance, Parts and Supplies Cost

Description	Unit	Oxide	Transition/Sulphide
Machinery and Equipment	US$	31,620,812	31,620,812
Piping	US$	2,529,665	2,529,665
Valves	US$	632,416	632,416
Electricals	US$	7,905,203	7,905,203
Instrumentation	US$	2,845,873	2,845,873
Total Cost	US$	**44,926,323**	**45,533,970**
Plant Maintenance Cost Factor	%	5.00	5.00
Maintenance Parts and Supplies Cost	**US$/a**	**2,246,316**	**2,276,698**
Maintenance Parts and Supplies Cost	**US$/t feed**	**0.45**	**0.46**

21.3.3.7 Assay

The annual assay operational cost was estimated from SENET's database costs and was estimated at **US$840,000**.

The assay costs consist of the mining/grade control and process plant assay requirements. The process plant assay operating costs will be based on the samples to be collected, the frequency of the sampling, and the type of analysis required. The number of metal accounting samples will also be based on a shift cycle of three 8-hour shifts per day. The metallurgical control samples will be collected when required, and an allowance for the number of control samples per annum has been made.

21.3.3.8 General and Administration (G&A) Costs

The G&A annual operating costs were estimated at **US$8,500,000**. The costs were best estimates when benchmarked against similar operations in the region.

21.3.4 TSF Operating Cost

To estimate the associated OPEX for the TSF, the considerations based on the available information regarding the TSF area and the deposition rates, as well as the nature of the facility, were considered.

The OPEX is broken down into two sections:

- Operational supervision including but not limited to the following:
  - Full-Time Site Manager – Expatriate
  - Full-Time Operational Supervisor as support – Local
  - Reliever for Site Manager – Expatriate
  - Safety Officer
  - Data Collection Clerk

- Operational upkeep including but not limited to the following:
    - Roads and ramps
    - Trench and drain cleaning
    - Liner repairs
    - Pipework service kits and additional pipe pieces
    - Grassing and landscaping
    - Freeboard pole marker setting up

22 ECONOMIC ANALYSIS

22.1 CAUTIONARY STATEMENT

The Adumbi PEA is preliminary in nature and includes Inferred Mineral Resources in the open-pit outlines that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves. There is no certainty that all the conclusions reached in the Adumbi PEA will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The results of the economic analysis discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes the following:

- Mineral resource estimates
- Assumed commodity prices and exchange rates
- Proposed mine production plan
- Projected mining and process recovery rates
- Assumptions as to mining dilution and ability to mine in areas previously exploited using mining methods as envisaged
- Sustaining costs and proposed operating costs
- Assumptions as to closure costs and closure requirements
- Assumptions as to environmental, permitting and social risks

Additional risks to the forward-looking information include the following:

- Changes to costs of production from what was assumed
- Unrecognised environmental risks
- Unanticipated reclamation expenses
- Unexpected variations in quantity of mineralised material, grade or recovery rates
- Geotechnical or hydrogeological considerations during mining being different from what was assumed
- Failure of mining methods to operate as anticipated
- Failure of plant, equipment or processes to operate as anticipated
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- Ability to maintain the social licence to operate
- Accidents, labour disputes and other risks of the mining industry
- Changes to interest rates
- Changes to tax rates

Calendar years used in the financial analysis are provided for conceptual purposes only.

Permits still have to be obtained in support of operations.

22.2 EVALUATION PRINCIPLES AND METHOD

The financial results presented in this report have been derived with the application of valuation methodologies that are aligned with international standards on project evaluation. No material non-compliant issues are known to exist in the valuation of the project.

All financial numbers are presented in constant money terms (CMT), using the US dollar (US$) over the LOM, unless otherwise stated.

The primary project evaluation method is a discounted unlevered free cash flow (DUFCF) analysis, which determines the net present value (NPV) of all expected cash inflows and cash outflows over Adumbi's LOM of 10 years from start of production. Cash inflows consist of annual revenue projections while cash outflows consist of initial capital costs, sustaining capital costs, operating costs, taxes and royalties.

The project investment has been evaluated on a stand-alone basis and assuming 100 % ownership; no incremental cash flow was calculated.

Two economic models were run: one using diesel generating sets as a source of power and the other using the HEP hybrid option.

22.3 ASSUMPTIONS

The economic analysis is based on several technical and economic input assumptions, as presented in Table 22.1.

Table 22.1: Financial Analysis Assumptions

Description	Unit	Assumption
Evaluation Start Date		January 1, 2022
Revenue		
Gold Price	US$/oz	1,600
Refining Losses	%	0.08
Discount Rate	%	5.0
Fuel Prices		
Diesel Price	US$/L	0.9
Fiscal		
Government Royalty	%	3.5
Tax Holiday	Years	2
Tax Rate (after tax holiday)	% of profits	30
Tax Rate if there is a loss	% of annual turnover	1
Dividend Tax	%	10
Depreciation	%	10 % over 10 years
Kilograms to Ounces	kg/oz	32.1505
Other Charges		
Bullion Transport and Refining Costs	US$/oz	3.50

Description	Unit	Assumption
Exchange Rates	ZAR/US$	15.11
	US$/£	1.16
	AU$/US$	1.39
	US$/C$	0.74
	CFA/€	2,260
	CFA/US$	2,004

22.4 EVALUATION DATE

The NPV results presented were based on an evaluation and start date of January 1, 2022.

22.5 GOLD PRICE

For the purposes of the economic analysis, the assumed gold price for the LOM is US$1,600/oz in 2022 CMT. A sensitivity analysis was performed to address the impact of various financial and operating variables on the overall project economic results.

22.6 DISCOUNT RATE

For the economic analysis, a real discount rate of 5 % was applied as this rate is commonly used to evaluate gold projects of this nature. The sensitivity of the project NPV to a range of discount rates is provided.

22.7 ESCALATION AND INFLATION

There is no adjustment for inflation and escalation in the financial model, which is in line with mineral industry economic analyses; all cash flows are in US dollars.

22.8 TAXATION AND ROYALTIES

22.8.1 Taxation

The main taxation consideration applied in the financial model, as provided by Loncor, is that 30 % of the taxable income will be applicable after one year of production, based on the DRC Tax Code.

22.8.2 Government Royalties

The Adumbi property is subject to a royalty, which is currently set at 3.5 % of the gross revenue, payable to the DRC government on production.

22.9 DEPRECIATION

A straight-line method over 10 years from the time production commences was used as the basis for depreciation.

22.10 WORKING CAPITAL

The calculation of the NPV considered the working capital needed to cater for costs associated with production up to the period when gold sales are sufficient to sustain operations.

22.11 PRODUCTION SCHEDULE AND OPERATING CASH COSTS

The production schedule and OPEX assumptions used in both models are summarised in Table 22.2 and Table 22.3.

Table 22.2: Summary of Production Schedule and Operating Cash Costs – Diesel Option

Description	Unit	LOM Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Tonnes Treated	kt	49,771	2,985	5,019	5,033	5,019	5,019	5,019	5,033	5,019	5,019	5,019	1,590
Grade	g/t	2.17	2.89	1.94	1.86	1.94	2.37	2.22	1.95	1.96	2.93	2.30	1.10
Recovery	%	89.8	89.4	89.4	89.6	89.8	89.9	89.9	89.9	89.9	89.9	89.9	89.9
Gold Production	koz	3,121	248	279	270	281	344	322	283	284	425	333	51
Payable Gold	%	99.9	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92
Payable Gold	koz	3,119	247.99	278.98	269.95	280.65	344.06	321.72	283.19	283.58	424.95	333.06	50.67
Gold Price	US$/oz	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600
Revenue	US$ thousand	4,990,084	396,781	446,373	431,923	449,037	550,493	514,746	453,108	453,722	679,924	532,898	81,079
Operating Costs													
Mining	US$ thousand	1,559,272	170,998	123,788	169,507	225,835	227,413	179,229	146,086	142,939	144,095	29,383	0
Plant and TSF	US$ thousand	893,358	53,579	90,084	90,330	90,084	90,084	90,084	90,330	90,084	90,084	90,084	28,533
G&A	US$ thousand	169,141	12,600	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	5,341
Refining Charges	US$ thousand	10,925	869	977	946	983	1,205	1,127	992	993	1,489	1,167	178
Government Royalties	US$ thousand	0	0	0	0	0	0	0	0	0	0	0	0
Total Cash Costs	US$ thousand	2,834,673	253,903	249,373	294,950	351,836	357,314	308,029	273,041	269,892	279,687	159,758	36,889
Total Cash Costs	US$/oz	908	1,023	893	1,092	1,253	1,038	957	963	951	658	479	727

Table 22.3: Summary of Production Schedule and Operating Cash Costs – HEP Hybrid Option

Description		LOM Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Tonnes Treated	kt	49,771	2,985	5,019	5,033	5,019	5,019	5,019	5,033	5,019	5,019	5,019	1,590
Grade	g/t	2.17	2.89	1.94	1.86	1.94	2.37	2.22	1.95	1.96	2.93	2.30	1.10
Recovery	%	89.8	89.8	89.4	89.4	89.6	89.8	89.9	89.9	89.9	89.9	89.9	89.9
Gold Production	koz	3,121	248	279	270	281	344	322	283	284	425	333	51
Payable Gold	%	99.9	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92	99.92
Payable Gold	koz	3,119	247.99	278.98	269.95	280.65	344.06	321.72	283.19	283.58	424.95	333.06	50.67
Gold Price	US$/oz	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600
Revenue	US$ thousand	4,990,084	396,781	446,373	431,923	449,037	550,493	514,746	453,108	453,722	679,924	532,898	81,079
Operating Costs													
Mining	US$ thousand	1,559,272	170,998	123,788	169,507	225,835	227,413	179,229	146,086	142,939	144,095	29,383	0
Plant and TSF	US$ thousand	718,581	43,097	72,460	72,658	72,460	72,460	72,460	72,658	72,460	72,460	72,460	22,950
G&A	US$ thousand	169,141	12,600	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	5,341
Refining Charges	US$ thousand	10,925	869	977	946	983	1,205	1,127	992	993	1,489	1,167	178
Government Royalties	US$ thousand	0	0	0	0	0	0	0	0	0	0	0	0
Total Cash Costs	US$ thousand	2,659,897	243,421	231,749	277,278	334,212	339,690	290,405	255,369	252,268	262,063	142,134	31,307
Total Cash Costs	US$/oz	852	981	830	1,026	1,190	987	902	901	889	616	426	617

22.12 INITIAL CAPITAL AND SUSTAINING COSTS

The initial and sustaining capital required for the project for the two options is given in Table 22.4.

Table 22.4: Summary of Capital Costs

Item	Diesel Option Value (US$ Thousand)	HEP Hybrid Option Value (US$ Thousand)
INITIAL CAPITAL	**392,010**	**530,058**
Mining	**59,986**	**59,986**
Mining Development Costs	59,986	59,986
Process Plant and TSF	**332,024**	**470,072**
Process Plant	171,369	171,369
Power Plant	14,405	152,452
TSF	65,880	65,880
Infrastructure	27,461	27,461
Access Road	7,800	7,800
Owner's Costs	45,110	45,110
SUSTAINING CAPITAL	**410,556**	**304,962**
Mining	**328,215**	**328,215**
Capitalised Waste	328,215	328,215
Process Plant and TSF	**80,452**	**66,329**
Power Plant	14,122	0
TSF	66,329	66,329
Mine Wide and G&A	**1,890**	**−89,582**
Rehabilitation and Closure	30,678	30,678
Process Plant Salvage Value	−28,789	−120,260

22.13 RESULTS OF FINANCIAL ANALYSIS

Using the assumptions shown in Table 22.1 and the production schedules, capital and operating costs, an economic analysis was carried out. Sunk costs were excluded from this analysis.

22.13.1 Summary of Financial Results

Table 22.5 summarises the key financial outputs from the financial model.

Table 22.5: Financial Analysis Summary

Description	Unit	HEP Hybrid Option		Diesel-Only Option	
		Pre-Tax	After Tax	Pre-Tax	After Tax
LOM Tonnage Ore Processed	kt	49,771	49,771	49,771	49,771
LOM Feed Grade Processed	g/t	2.17	2.17	2.17	2.17
Production Period	Years	10.3	10.3	10.3	10.3
LOM Gold Recovery	%	89.8	89.8	89.8	89.8
LOM Gold Production	koz	3,121	3,121	3,121	3,121
LOM Payable Gold After Refining Losses	koz	3,119	3,119	3,119	3,119
Gold Price	US$/oz	1,600	1,600	1,600	1,600
Revenue	US$ million	4,990	4,990	4,990	4,990
Site Operating Costs	US$/oz	793	793	849	849
Total Cash Costs	US$/oz	852	852	908	908
All-In Sustaining Costs (AISC)	US$/oz	950	950	1,040	1,040
NPV (5 % Discount)	US$ million	895	624	843	600
Internal Rate of Return (IRR)	%	25.2	20.7	30.3	25.2
Discount Rate	%	5	5	5	5
Payback Period (from start of production)	Years	4.16	4.98	3.16	4.06
Project Net Cash	US$ million	1,495	1,087	1,353	992

22.13.2 Project LOM Cash Flow

The annual project cash flows for the two options are presented in Table 22.6 and Table 22.7.

Table 22.6: LOM Project Cash Flow – Diesel Option

Description	Unit	LOM Totals	2022-12-31	2023-12-31	2024-12-31	2025-12-31	2026-12-31	2027-12-31	2028-12-31	2029-12-31	2030-12-31	2031-12-31	2032-12-31	2033-12-31	2034-12-31
Tonnes Treated	kt	49,771			2,985	5,019	5,033	5,019	5,019	5,019	5,033	5,019	5,019	5,019	1,590
Grade	g/t	2.17			2.89	1.94	1.86	1.94	2.37	2.22	1.95	1.96	2.93	2.30	1.10
Recovery	%	89.8%			89.4%	89.4%	89.6%	89.8%	89.9%	89.9%	89.9%	89.9%	89.9%	89.9%	89.9%
Gold Production	koz	3,121			248.19	279.21	270.17	280.87	344.33	321.97	283.42	283.80	425.29	333.33	50.71
Payable Gold	%	99.9			99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9
Payable Gold	koz	3,119			248	279	270	281	344	322	283	284	425	333	51
Gold Price	US$/oz	1,600			1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600
Revenue	US$ thousand	4,990,084			396,781	446,373	431,923	449,037	550,493	514,746	453,108	453,722	679,924	532,898	81,079
Operating Costs															
Mining	US$ thousand	1,559,272			170,998	123,788	169,507	225,835	227,413	179,229	146,086	142,939	144,095	29,383	0
Plant	US$ thousand	893,358			53,579	90,084	90,330	90,084	90,084	90,084	90,330	90,084	90,084	90,084	28,533
TSF	US$ thousand	27,325			1,970	2,101	2,250	2,418	2,546	2,774	2,974	3,196	3,422	3,674	0
G&A	US$ thousand	169,141			12,600	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	5,341
Refining Charges	US$ thousand	10,925			869	977	946	983	1,205	1,127	992	993	1,489	1,167	178
Government Royalties	US$ thousand	174,653			13,887	15,623	15,117	15,716	19,267	18,016	15,859	15,880	23,797	18,651	2,838
Total Cash Costs	US$ thousand	2,834,673			253,903	249,373	294,950	351,836	357,314	308,029	273,041	269,892	279,687	159,758	36,889
Total Cash Costs	US$/oz	908			1,023	893	1,092	1,253	1,038	957	963	951	658	479	727
After-Tax Cash Flow															
Cash Flow Before CAPEX	US$ thousand	2,155,411			142,878	197,001	136,973	97,201	193,178	206,717	180,067	183,830	400,237	373,140	44,190
Initial Capital															
Mining	US$ thousand	49,988	0	49,988											
Process Plant	US$ thousand	143,655	77,200	66,455											
Power Plant	US$ thousand	12,004	6,451	5,553											
TSF	US$ thousand	54,900	29,503	25,397											
Preproduction	US$ thousand	68,498	25,719	42,779											
Contingencies	US$ thousand	62,965	30,114	32,851											
Initial Capital Expenditure	US$ thousand	392,010	*168,987*	*223,022*											
Other Capitalised Items															
Working Capital	US$ thousand	11,687	0	11,687											
Sustaining Capital															
Mining Sustaining Capital	US$ thousand	328,215			0	0	0	0	67,086	71,275	21,775	7,349	87,283	73,446	0
Power Plant	US$ thousand	14,122			0	0	0	6,214	565	0	5,649	1,130	565	0	0
TSF	US$ thousand	66,329			0	290	36,994	0	0	290	28,756	0	0	0	0
Rehabilitation and Closure	US$ thousand	30,678			347	0	263	473	394	294	604	294	1,699	9,693	16,616
Equipment Salvage Value	US$ thousand	(28,789)													(28,789)
Sustaining Capital	US$ thousand	410,556			347	290	37,257	6,687	68,045	71,860	56,784	8,773	89,547	83,138	-12,172
Cash Flow After CAPEX	US$ thousand	*1,352,845*	*(168,987)*	*(223,022)*	*142,531*	*196,711*	*99,716*	*90,514*	*125,133*	*134,857*	*123,283*	*175,058*	*310,689*	*290,001*	*56,362*
Depreciation	US$ thousand	(406,116)			(39,201)	(39,201)	(39,201)	(39,201)	(39,201)	(45,910)	(53,037)	(55,215)	(55,950)	0	0
Taxable Income	US$ thousand	1,133,925			0	0	153,046	56,196	46,823	80,427	83,800	65,715	115,306	247,940	284,672
Tax Rate	%	30%			0	0	30	30	30	30	30	30	30	30	30
Tax	US$ thousand	340,177			0	0	45,914	16,859	14,047	24,128	25,140	19,714	34,592	74,382	85,402
After-Tax Cash Flow	US$ thousand	1,012,668	(168,987)	(223,022)	142,531	196,711	53,802	73,655	111,086	110,729	98,143	155,343	276,097	215,619	(29,039)

Description	Unit	LOM Totals	2022-12-31	2023-12-31	2024-12-31	2025-12-31	2026-12-31	2027-12-31	2028-12-31	2029-12-31	2030-12-31	2031-12-31	2032-12-31	2033-12-31	2034-12-31
Undiscounted Cumulative Cash Flow After Tax	US$ thousand		(168,987)	(392,010)	(249,479)	(52,768)	1,034	74,689	185,775	296,504	394,647	549,990	826,088	1,041,707	1,012,668
Discount Rate	%	5													
Discounted Cash Flow After Tax	US$ thousand	571,859	(160,940)	(212,888)	123,123	124,061	64,921	57,061	71,782	74,261	66,761	86,234	138,164	113,929	25,391
Cumulative Discounted Cash Flow After Tax	US$ thousand		(160,940)	(373,829)	(250,705)	(126,644)	(61,724)	(4,662)	67,120	141,381	208,142	294,376	432,539	546,468	571,859
AISC															
AISC	US$ thousand	3,245,229			254,250	249,663	332,207	358,523	425,360	379,889	329,826	278,665	369,235	242,897	24,716
AISC	US$/oz	1,040			1,024	894	1,230	1,276	1,235	1,180	1,164	982	868	729	487
Discount Rate	%	5													
Post-Tax NPV	(US$ thousand)	600,199													
Post-Tax IRR	(%)	25.2													
Undiscounted Payback Period from Start of Production	Years	3.16													

Table 22.7: LOM Project Cash Flow – HEP Hybrid Option

Description		LOM Totals	2022-12-31	2023-12-31	2024-12-31	2025-12-31	2026-12-31	2027-12-31	2028-12-31	2029-12-31	2030-12-31	2031-12-31	2032-12-31	2033-12-31	2034-12-31
Tonnes Treated	kt	49,771			2,985	5,019	5,033	5,019	5,019	5,019	5,033	5,019	5,019	5,019	1,590
Grade	g/t	2.17			2.89	1.94	1.86	1.94	2.37	2.22	1.95	1.96	2.93	2.30	1.10
Recovery	%	89.8%			89.4%	89.4%	89.6%	89.8%	89.9%	89.9%	89.9%	89.9%	89.9%	89.9%	89.9%
Gold Production	koz	3,121			248.19	279.21	270.17	280.87	344.33	321.97	283.42	283.80	425.29	333.33	50.71
Payable Gold	%	99.9			99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9
Payable Gold	koz	3,119			248	279	270	281	344	322	283	284	425	333	51
Gold Price	US$/oz	1,600			1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600
Revenue	US$ thousand	4,990,084			396,781	446,373	431,923	449,037	550,493	514,746	453,108	453,722	679,924	532,898	81,079
Operating Costs															
Mining	US$ thousand	1,559,272			170,998	123,788	169,507	225,835	227,413	179,229	146,086	142,939	144,095	29,383	0
Plant	US$ thousand	718,581			43,097	72,460	72,658	72,460	72,460	72,460	72,658	72,460	72,460	72,460	22,950
TSF	US$ thousand	27,325			1,970	2,101	2,250	2,418	2,546	2,774	2,974	3,196	3,422	3,674	0
G&A	US$ thousand	169,141			12,600	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	16,800	5,341
Refining Charges	US$ thousand	10,925			869	977	946	983	1,205	1,127	992	993	1,489	1,167	178
Government Royalties	US$ thousand	174,653			13,887	15,623	15,117	15,716	19,267	18,016	15,859	15,880	23,797	18,651	2,838
Total Cash Costs	US$ thousand	2,659,897			243,421	231,749	277,278	334,212	339,690	290,405	255,369	252,268	262,063	142,134	31,307
Total Cash Costs	US$/oz	852			981	830	1,026	1,190	987	902	901	889	616	426	617
After Tax Cash Flow															
Cash Flow Before CAPEX	US$ thousand	2,330,187			153,360	214,625	154,645	114,825	210,802	224,341	197,739	201,454	417,860	390,764	49,772
Initial Capital															
Mining	US$ thousand	49,988	0	49,988											
Process Plant	US$ thousand	143,655	77,200	66,455											
Power Plant	US$ thousand	138,593	74,480	64,113											
TSF	US$ thousand	54,900	29,503	25,397											
Preproduction	US$ thousand	68,498	25,719	42,779											

Description		LOM Totals	2022-12-31	2023-12-31	2024-12-31	2025-12-31	2026-12-31	2027-12-31	2028-12-31	2029-12-31	2030-12-31	2031-12-31	2032-12-31	2033-12-31	2034-12-31
Contingencies	US$ thousand	74,423	35,594	38,830											
Initial Capital Expenditure	US$ thousand	530,058	*242,497*	*287,561*											
Other Capitalised Items															
Working Capital	US$ thousand	11,687	0	11,687											
Sustaining Capital															
Mining Sustaining Capital	US$ thousand	328,215			0	0	0	0	67,086	71,275	21,775	7,349	87,283	73,446	0
Power Plant	US$ thousand	0			0	0	0	0	0	0	0	0	0	0	0
TSF	US$ thousand	66,329			0	290	36,994	0	0	290	28,756	0	0	0	0
Rehabilitation and Closure	US$ thousand	30,678			347	0	263	473	394	294	604	294	1,699	9,693	16,616
Equipment Salvage Value	US$ thousand	(120,260)													(120,260)
Sustaining Capital	US$ thousand	304,962			347	290	37,257	473	67,480	71,860	51,135	7,643	88,983	83,138	-103,644
Cash Flow After CAPEX	US$ thousand	*1,495,168*	*(242,497)*	*(287,561)*	*153,013*	*214,335*	*117,388*	*114,352*	*143,322*	*152,481*	*146,604*	*193,811*	*328,878*	*307,625*	*153,416*
Depreciation	US$ thousand	(530,359)			(53,006)	(53,006)	(53,006)	(53,006)	(53,006)	(59,714)	(66,842)	(69,019)	(69,754)	0	0
Taxable Income	US$ thousand	1,196,321			0	0	156,865	60,063	56,856	84,811	87,619	75,231	120,255	252,324	302,296
Tax Rate	%	30			0	0	30	30	30	30	30	30	30	30	30
Tax	US$ thousand	358,896			0	0	47,060	18,019	17,057	25,443	26,286	22,569	36,076	75,697	90,689
After-Tax Cash Flow	US$ thousand	1,136,271	(242,497)	(287,561)	153,013	214,335	70,329	96,333	126,265	127,038	120,318	171,242	292,802	231,928	62,727
Undiscounted Cumulative Cash Flow After Tax	US$ thousand		(242,497)	(530,058)	(377,045)	(162,710)	(92,381)	3,952	130,217	257,254	377,573	548,815	841,616	1,073,544	1,136,271
Discount Rate	%	5													
Discounted Cash Flow After Tax	US$ thousand	594,106	(230,949)	(271,427)	132,178	137,618	77,858	72,603	83,774	85,414	79,954	96,836	148,029	120,798	61,419
Cumulative Discounted Cash Flow After Tax	US$ thousand		(230,949)	(502,376)	(370,198)	(232,580)	(154,721)	(82,118)	1,656	87,070	167,024	263,859	411,889	532,687	594,106
AISC															
AISC	US$ thousand	2,964,859			243,768	232,039	314,535	334,685	407,171	362,265	306,504	259,911	351,046	225,273	(72,337)
AISC	US$/oz	950			982	831	1,164	1,192	1,182	1,125	1,081	916	825	676	-1,426
Discount Rate	%	5													
Post-Tax NPV	(US$ thousand)	623,516													
Post-Tax IRR	(%)	20.7													
Undiscounted Payback Period from Start of Production	Years	4.16													

22.13.3 Sensitivities

A sensitivity analysis was performed on each individual variable, measuring the impact of each input variable independently on the target forecast (NPV).

An overall sensitivity analysis was also performed, considering all the variables on the target forecast.

22.13.3.1 Key Project Metric Sensitivity to Gold Price

Table 22.8 and Table 22.9 assess the sensitivity of some of the key financial indicators of the project to variations on the average gold price for both options. The results reflect a robust project even with a low gold price, with more than 15 % IRR achieved. At the time of the study, the gold price was fluctuating at approximately US$1,810/oz. A strong gold price, which is anticipated in the short to medium period, will significantly improve the NPV and IRR.

Table 22.8: Key Project Metric Sensitivity to Gold Price – Diesel Option

Description	Unit	Value				
Change in Gold Price	%	−15	−10	0	10	15
Gold Price	US$/oz	1,360	1,440	**1,600**	1,760	1,840
NPV at 5 % – Pre-Tax	US$ million	321	495	843	1,191	1,365
NPV at 5 % – After Tax	US$ million	211	345	600	855	983
IRR – Pre-Tax	%	15.58	20.83	30.34	38.99	43.07
IRR – After Tax	%	12.68	17.21	25.18	32.57	36.11
Total Cash Costs	US$/oz	900	903	908	914	917
AISC	US$/oz	1,031	1,034	1,040	1,045	1,048
Cash Flow Payback – Pre-Tax	Years	7.53	5.46	3.16	2.18	1.93
Cash Flow Payback – After Tax	Years	8.02	6.28	4.06	2.77	2.39
NOTE: The value in bold reflects the base case.						

Table 22.9: Key Project Metric Sensitivity to Gold Price – HEP Hybrid Option

Description	Unit	Value				
Change in Gold Price	%	−15	−10	0	10	15
Gold Price	US$/oz	1,360	1,440	**1,600**	1,760	1,840
NPV at 5 % – Pre-Tax	US$ million	373	547	895	1,243	1,417
NPV at 5 % – After Tax	US$ million	238	368	624	879	1,006
IRR – Pre-Tax	%	14.1	18.0	25.2	31.9	35.1
IRR – After Tax	%	11.4	14.7	20.7	26.4	29.1
Total Cash Costs	US$/oz	844	847	852	858	861
AISC	US$/oz	941	944	950	955	958
Cash Flow Payback – Pre-Tax	Years	7.88	6.26	4.16	3.00	2.64
Cash Flow Payback – After Tax	Years	8.28	7.10	4.98	3.76	3.30
NOTE: The value in bold reflects the base case.						

22.13.3.2 Gold Price and Discount Rate Sensitivity Analysis

In assessing the combined impact of the gold price and the discount rate on the value of the project, the two variables were varied concurrently, and the results of their combined sensitivity analysis are shown in Table 22.10 and Table 22.11.

Table 22.10: Gold Price and Discount Rate Sensitivity Analysis – Diesel Option

Gold Price (US$/oz)	NPV Pre-Tax (US$ Million)		
	Discount Rate		
	0 %	5 %	10 %
1,360	631	321	131
1,440	871	495	262
1,600	1,353	**843**	522
1,760	1,854	1,191	782
1,840	2,075	1,365	912
NOTE: The values in bold reflect the base case.			

Table 22.11: Gold Price and Discount Rate Sensitivity Analysis – HEP Hybrid Option

Gold Price (US$/oz)	NPV Pre-Tax (US$ Million)		
	Discount Rate		
	0 %	5 %	10 %
1,360	773	373	130
1,440	1,014	597	260
1,600	1,495	**895**	521
1,760	1,977	1,243	781
1,840	2,217	1,417	911
NOTE: The values in bold reflect the base case.			

22.13.3.3 Gold Price and Head Grade Sensitivity Analysis

The combined impact of varying gold price and the head grade on the project value was assessed, and the results of their combined sensitivity analysis are shown in Table 22.12 and Table 22.13. The results show that the project can accommodate a low head grade and low metal price.

Table 22.12: Gold Feed Grade and Gold Price Sensitivity – Diesel Option

Head Grade (g/t)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
1.847	−121	27	323	619	767
1.955	26	183	496	809	966
2.172	321	495	**843**	1,191	1,365
2.390	615	807	1,190	1,572	1,764
2.498	762	963	1,363	1,763	1,963
NOTE: The values in bold reflect the base case.					

Table 22.13: Gold Feed Grade and Gold Price Sensitivity – HEP Hybrid Option

Head Grade (g/t)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
1.847	−69	79	375	671	819
1.955	78	235	549	862	1,018
2.172	373	547	**895**	1,243	1,417
2.390	668	859	1,242	1,625	1,618
2.498	815	1,015	1,416	1,816	2,016
NOTE: The values in bold reflect the base case.					

22.13.3.4 Gold Price and OPEX Sensitivity Analysis

The gold price and the total OPEX were varied, and the results of their combined sensitivity are shown in Table 22.14Table 22.14 and Table 22.15.

Table 22.14: OPEX and Gold Price Sensitivity – Diesel Option

Change in OPEX (%)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
−15	617	791	1,139	1,487	1,661
−10	519	693	1,041	1,389	1,563
0	321	495	**843**	1,191	1,365
10	122	296	645,41	993	1,167
15	22	197	546	895	1,069
NOTE: The values in bold reflect the base case.					

Table 22.15: OPEX and Gold Price Sensitivity – HEP Hybrid Option

Change in OPEX (%)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
−15	651	824	1,173	1,521	1,695
−10	558	732	1,080	1,428	1,602
0	373	547	**895**	1,243	1,412
10	197	361	710	1,058	1,232
15	93	268	618	966	1,140
NOTE: The values in bold reflect the base case.					

22.13.3.5 Gold Price and CAPEX Sensitivity Analysis

The gold price and the total CAPEX were varied, and the impact on the project economics is shown in Table 22.16 and Table 22.17.

Table 22.16: CAPEX and Gold Price Sensitivity – Diesel Option

Change in CAPEX (%)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
−15	378	552	900	1,248	1,422
−10	359	533	881	1,229	1,403
0	321	495	**843**	1,191	1,365
10	282	457	805	1,153	1,327
15	263	438	785	1,137	1,308
NOTE: The values in bold reflect the base case.					

Table 22.17: CAPEX and Gold Price Sensitivity – HEP Hybrid Option

Change in CAPEX (%)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
−15	450	625	973	1,321	1,495
−10	425	599	947	1,295	1,469
0	373	547	**895**	1,243	1,417
10	321	496	844	1,192	1,366
15	295	470	818	1,166	1,340
NOTE: The values in bold reflect the base case.					

22.13.3.6 Gold Price and Gold Recovery Sensitivity Analysis

In assessing the combined impact of the gold price and the gold recovery on the value of the project, these two variables were varied concurrently, and the results of their combined sensitivity analysis are shown in Table 22.18 and Table 22.19.

Table 22.18: Recovery and Gold Price Sensitivity – Diesel Option

Recovery (%)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
74.8	−121	27	323	619	767
79.8	26	183	496	809	966
89.8	321	495	**843**	1,191	1,385
90.8	350	526	878	1,229	1,405
92.8	409	589	947	1,305	1,485
NOTE: The values in bold reflect the base case.					

Table 22.19: Recovery and Gold Price Sensitivity – HEP Hybrid Option

Recovery (%)	NPV Pre-Tax (US$ Million)				
	Gold Price (US$/oz)				
	1,360	1,440	1,600	1,760	1,840
74.8	−69	79	375	671	819
79.8	78	235	549	862	1,018
89.8	373	547	**895**	1,243	1,417
90.8	402	579	930	1,282	1,457
92.8	461	641	999	1,358	1,537
NOTE: The values in bold reflect the base case.					

22.13.4 Discussion and Opportunities

When ranked, the sensitivity analysis indicates that the project is most sensitive to gold price, followed by head grade, recovery, OPEX and then CAPEX.

23 ADJACENT PROPERTIES

In addition to the Imbo Project, there have been other mineral exploration activities in the Ngayu Greenstone Belt in recent times, and mineral resources have been defined within the belt. Since 2010, Loncor has been the largest permit holder in the Ngayu belt and has been exploring a number of prospects on its own since 2010 or in joint venture with Barrick Gold Congo SARL (formerly Randgold Resources Congo SARL) (Barrick) from 2016 to 2021 (see Figure 23.1). Rio Tinto had agreements with Loncor and Kilo Goldmines for iron ore in the Ngayu belt since 2010 and undertook initial exploration and some drilling, but terminated these agreements in 2015 due to limited exploration success.

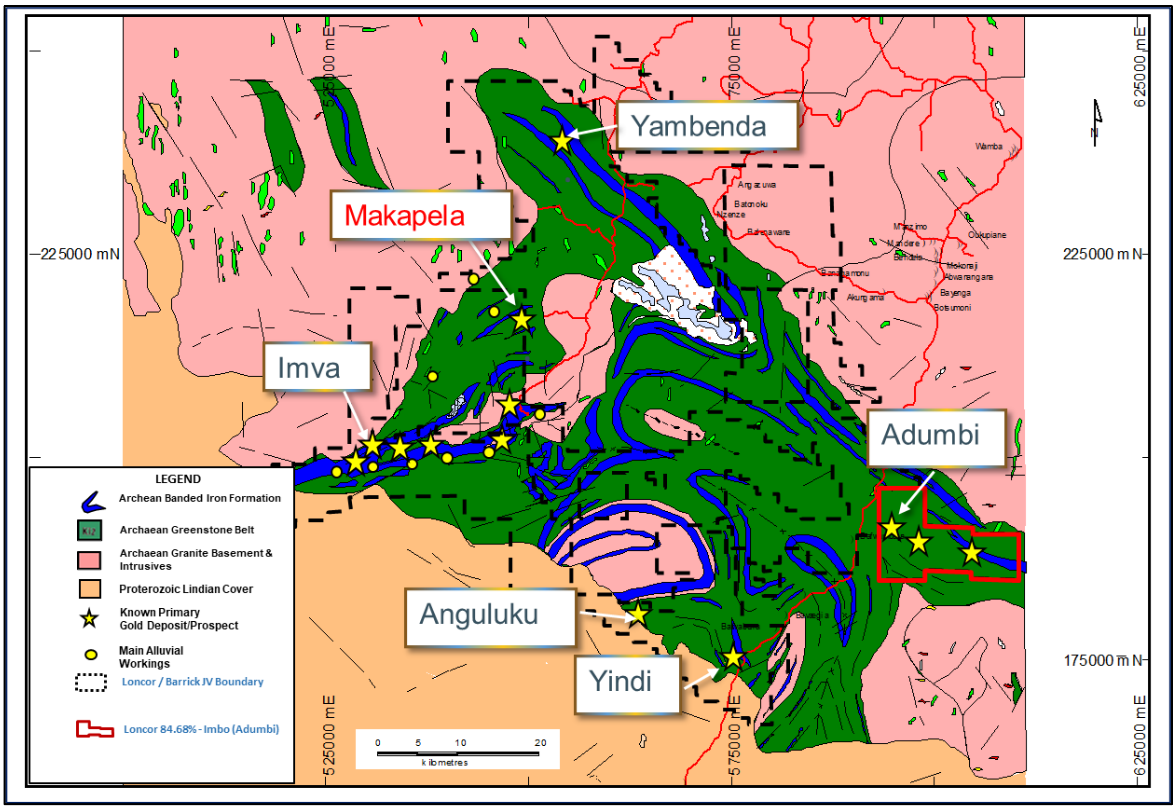


Figure 23.1: Main Gold Projects and Prospects within the Ngayu Greenstone Belt

23.1 NGAYU BELT EXPLORATION (2010 TO 2016)

Loncor commenced its exploration activities in early 2010, and a base camp was established at Yindi. Due to its large landholdings for gold of 4,500 km^2 at that time, it was decided to divide the exploration into two concurrent programmes:

- Assessment of areas of known gold mineralisation (Yindi and Makapela) with the potential to rapidly reach the drilling stage and provide a mineral resource. Soil sampling, augering, rock chip and channel sampling were carried out prior to diamond drilling.

- Regional programmes aimed at assessing the remainder of the large land package as quickly and cost effectively as possible, in order to identify and prioritise mineralised target areas for follow-up, and enable less-prospective ground to be relinquished with confidence. This programme mainly entailed a regional BLEG survey and detailed interpretation of regional aeromagnetic data, which were carried out under a technology consultation services agreement between Loncor and Newmont (a shareholder in Loncor), which was entered into in February 2011 (but is no longer in place).

During 2012, Loncor undertook more detailed aeromagnetic and radiometric surveys over priority target areas (i.e. Imva Fold area). Grids were established at the Yindi, Makapela, Itali, Matete, Nagasa, Mondarabe, Anguluku and Adumbi West prospects with airborne magnetic and radiometric surveys, geological mapping, stream sediment sampling, soil and rock sampling, trenching, augering, ground geophysical surveys (induced polarisation) and core drilling being undertaken. During the period of 2010 to 2013, Loncor undertook drilling programmes on a number of prospects in the Ngayu belt and outlined mineral resources at Makapela in the west of the belt.

Loncor holds 100 % of the Makapela project. After undertaking soil and channel sampling, a core drilling programme at Makapela was commenced in November 2010 with the objective of testing along strike and at depth the subvertical vein mineralised system being exploited by the artisanal miners at the Main, North and Sele Sele pits, which returned significant results from soil and channel sampling. Drill results at Makapela were announced by Loncor via a number of press releases in 2011 and 2012. Significant drill intersections included 7.19 m grading 64 g/t Au, 4.28 m at 32.6 g/t Au, 3.47 m grading 24.9 g/t Au, 4.09 m at 21.7 g/t Au and 4.35 m grading 17.5 g/t Au.

After conducting preliminary metallurgical test work in May 2012, Loncor announced a maiden mineral resource estimate for Loncor’s Makapela prospect of 4.10 Mt grading 7.59 g/t Au (using a 2.75 g/t Au cut-off) for an Inferred Mineral Resource of 1.0 Moz of gold to a maximum vertical depth of 500 m below surface with gold mineralisation open at depth. The resource was updated in April 2013 when Loncor announced updated mineral resource estimates for Loncor’s Makapela prospect of an Indicated Mineral Resource of 0.61 Moz of gold (2.20 Mt grading at 8.66 g/t Au) and an Inferred Mineral Resource of 0.55 Moz of gold (3.22 Mt grading at 5.30 g/t Au).

A total of 56 core holes (18,091 m) were completed in the vicinity of the Main and North pits, and 15 holes (3,594 m) were drilled at Sele Sele. In addition to the above resource drilling programme, a total of 12 holes (1,560 m) were drilled to locate potential extensions to the known reefs and new mineralised structures indicated by soil, rock chip and auger sampling. Several units of BIF are interlayered within basalts and range up to 13 m in thickness, although the width is generally less than 6 m. Quartz porphyry and quartz-feldspar porphyry dykes and sills are also present. In the vicinity of the mineralised zones, the intrusive units are generally no more than a few metres in width.

Three styles of gold mineralisation are present at Makapela:

- The first style contains quartz veins emplaced into shear zones within the basalt sequence. The best developed and economically significant vein (Reef 1) is exploited in the Main pit and consists of white quartz with irregularly distributed pyrite. Visible gold is quite common, occurring in 28 % of the intersections as isolated specks and small aggregates up to 2 mm across. Reef 1 has been intersected over a strike length of 480 m and to a vertical depth of 480 m, and dips to the WNW at 80° to 90°. It has an average true width of 2.15 m grading at 11.15 g/t Au. A characteristic of Reef 1 is the good geological continuity between drill sections. Although the width and grade are variable, the vein was present in almost all the holes, approximately in the expected position. The basalt-hosting Reef 1 shows intense hydrothermal alteration for several metres into the hanging wall and footwall.
- A second style contains strike-parallel mineralisation up to 6 m in width, which is closely associated with shearing within and on the margins of narrow BIF units. The most important zone (Reef 2) is exploited in the North pit. Visible gold is much less common than in Reef 1, occurring in 5 % of the intersections. Mineralisation in the Sele Sele pit, 2 km NNE of the North pit, has similar characteristics to those of Reef 2, and is interpreted to be on the same BIF unit. However, the Sele Sele zone is generally wider and of a lower grade than the North pit area, with the best intersection drilled being 15.68 m at 5.35 g/t Au. The mineralisation plunges to the SSE at approximately 40°.
- A third area of the Reef 2 style mineralisation occurs in the Bamako area where channel sampling returned an intersection of 4.60 m at 11.42 g/t Au. The mineralisation is associated with a 2 km long soil anomaly, and although the best intersection from preliminary drilling was of relatively low grade (3.60 m at 4.43 g/t Au), further work is warranted.

The deposit at Makapela is open down plunge, creating the prospect of drilling to below the current 500 m depth to extend the resources as well as potentially exploring for additional resources between the main target areas delineated and further along the regional structure. Loncor also considers it unlikely that all the mineralised bodies are outcropping, and a good potential exists for locating blind mineralised shoots along well-defined structures with an aggregate strike of over 5 km. Loncor is currently undertaking a feasibility study at Makapela as part of converting Makapela’s exploration permit into an exploitation permit.

Besides Makapela, Loncor drilled other prospects during this period, and significant intersections were obtained at Yindi (21.3 m grading 3.3 g/t Au, 24.0 m grading 1.5 g/t Au and 10.3 m grading 4.1 g/t Au) and at Itali (38.82 m at 2.66 g/t Au, 14.70 m at 1.68 g/t Au and 3.95 m at 19.5 g/t Au)

At the end of 2013, due to a significant drop in the gold price, exploration was reduced, and no further drilling was undertaken at Ngayu.

23.2 NGAYU EXPLORATION (2016 TO 2021)

In January 2016, Loncor entered into a joint venture agreement with Barrick. This agreement provided for a joint venture (Joint Venture) between Loncor and Barrick with respect to exploration permits held by Loncor now covering 1,894 km^2 of ground in the Ngayu belt and excluding certain parcels of land surrounding and including the Makapela and Yindi projects, which were retained by Loncor and did not form part of the Joint Venture (the Imbo Project is also not part of the Joint Venture). Under the joint venture agreement, Barrick managed and funded all exploration of the permit areas until the completion of a pre-feasibility study on any gold discovery meeting Barrick's investment criteria. Once the Joint Venture determined to move ahead with a full feasibility study, a special purpose vehicle (SPV) would be created to hold the specific discovery areas. Subject to the DRC's free carried interest requirements, Barrick would retain 65 % of the SPV with Loncor holding the balance of 35 %. Loncor would be required, from that point forward, to fund its pro-rata share of the SPV in order to maintain its 35 % interest or be diluted.

In January 2017, Loncor announced preliminary results of the geophysical airborne survey undertaken by Randgold as part of its Joint Venture with Loncor (it is noted that Randgold and Barrick merged under Barrick's name in early 2019). A 10,013 line-kilometre helicopter-borne electromagnetic VTEM (versatile time domain electromagnetic) survey (JV Survey) was completed over the Ngayu belt. The JV Survey provided a valuable additional layer of geological information through mapping the conductivity nature of the belt. The new data assisted with resolving the lithological nature of the belt as well as assisting in identifying major structures and areas of structural complexity.

During 2020, the Barrick-Loncor joint venture ground increased to approximately 2,000 km^2 with additional exploration permits and exploitation licences controlled by Loncor, as well as certain exploration permits held by Barrick, ceded into the Joint Venture.

By June 2020, several priority targets had been outlined by Barrick, including Anguluku, Bakpau, Medere (Itali), Mokepa and Yambenda, and two portable core rigs commenced scout drilling on these targets (see Figure 23.1). By May 2021, Barrick had completed 27 core holes (3,844 m) on several targets in the Ngayu belt, including Anguluku, Medere (Itali), Medere, Mokepa and Yambenda (see Figure 23.1). At Yambenda, four drill sections tested a 3.6 km portion of the 9.5 km long anomalous soil corridor. All the holes intersected mineralisation associated with WNW shear structures developed as a contact zone between BIF and volcano-sediments, including conglomerates (similar host rock assemblage found at Kibali mine). The best drill intercepts included 14 m grading 0.85 g/t Au in YBDD0001, 49 m grading 0.52 g/t Au and 14.5 m grading 1.38 g/t Au in YBDD0002, and 35.05 m at 0.60 g/t Au in YBDD0006. At Mokepa, six scout holes were drilled, with the best holes assaying 19 m grading 1.04 g/t Au in borehole ADDD0001 and 46.7 m grading 1.32 g/t Au in hole ADDD0002.

In May 2021, Barrick informed Loncor that it would not be continuing exploration on the Joint Venture ground. Loncor is assessing the results of the Joint Venture programme. In particular, the Mongaliema and Mokepa prospects, which are close to Makapela, are planned to be further investigated by Loncor. At Mongaliema, the target area is a west-northwest trending shear zone hosted within altered metasediments with cherty units near the contact of a dolerite intrusive. Pitting has demonstrated that much of the area is covered by thick transported cover which hinders near-surface exploration. Pitting was undertaken to the southwest of the trench, which graded 32 m at 1.37 g/t Au. Results from pits in excess of 5 m deep confirmed the southwestern extension beneath thick transported alluvial material, with an average high grade of 18.13 g/t Au from 11 samples. Further work is warranted from the results received to date at Mongaliema. Mongaliema will be evaluated to determine whether it has the resource potential to be combined with the nearby Makapela deposit.

In terms of producing gold mines, the Kibali Gold Mine, approximately 220 km northeast by air from the Imbo Project, is located within the Archean aged Moto greenstone belt and commenced gold production in September 2013. The mine is owned by Kibali Goldmines SA (Kibali), which is a joint venture company with 45 % owned by Barrick Gold, 45 % by AngloGold Ashanti, and 10 % by Société Minière de Kilo-Moto (SOKIMO). Barrick is the operator and in 2020, Kibali produced 808,134 oz of gold at an AISC of US$778/oz of gold. The mine is an open-pit and underground operation, and in 2020, 7.62 Mt of ore was processed at an average grade of 3.6 g/t gold and a metallurgical recovery of 90 %. Kibali had Measured and Indicated Mineral Resources of 15.5 Moz of gold, Inferred Mineral Resources of 1.5 Moz and Proven and Probable ore reserves at the end of 2020 of 9.33 Moz (from Barrick Gold 2020 Annual Report).

24 OTHER RELEVANT DATA AND INFORMATION

The DRC covers 2,344,858 km^2 of land in the centre of Africa, making it the twelfth largest country in the world, approximately two-thirds the size of Western Europe. With an estimated population of 89.50 million (2020), DRC is the fourth most populous country in Africa. Approximately 45 % of the population lives in cities, and the capital Kinshasa is by far the largest, with approximately 15 million inhabitants (2020). DRC has approximately 200 ethnic identities with approximately 45 % of the population belonging to the Kongo, Luba, Mongo and Mangbetu-Azande groups.

24.1 DRC POLITICAL AND ECONOMIC CLIMATE

The Belgian Congo gained independence from Belgium in June 1960. In 1971, the country was renamed Zaire. Following a rebellion started in mid-1996, President Mobutu Sese Seko was toppled in May 1997 by Laurent Désiré Kabila after 32 years of power and Zaire was renamed the Democratic Republic of the Congo. In 1998, a civil war broke out with the east and north of the country controlled by rebel factions allegedly supported by Rwanda and Uganda. In January 2001, Laurent Kabila was assassinated and succeeded by his son, Joseph Kabila. Whereas Laurent Kabila had a conflicted relationship with the international community, Joseph Kabila re-established various engagements and commenced overtures for peace. In June 2003, a formal peace agreement was signed and the country reunited through a transition government. In 2006, the first multi-party elections in 40 years were held, with Joseph Kabila winning the second voting round. Elections were held again in 2011 won by Joseph Kabila and in 2018 where Joseph Kabila was replaced by Felix Antoine Tshisekedi Tshilombo in a contested election.

The country is divided into 26 provinces, each with a governor and provincial parliament. The national parliament consists of a lower house where representatives are directly elected from the provinces, and a senate with members voted by provincial parliaments. The country is by and large unified and at peace. The east remains troubled by local ethnic rebellions which have little popular support. The main rebel group, the 23rd of March Movement (M23), consisted of army defectors grouped around leaders from the Kivu region bordering Rwanda, accused by the international community of supporting this group. The goals of M23 were unclear but were ostensibly motivated by control of natural resources in the area they occupy. In early November 2013, the M23 rebels were defeated by the Congolese army with support of a United Nations brigade consisting of soldiers from Tanzania and South Africa. The rebel group thereafter dissolved itself and said it was ready to disarm, demobilise and integrate into the Congolese army. Since the 1990s, the Allied Democratic Forces (ADF), an Islamic rebel group from Uganda, has operated in northeastern DRC and has been blamed for numerous civilian massacres and attacks against DRC security forces, triggering flights of refugees inside the DRC and across the border into neighbouring countries. Other smaller rebel groups are also present in the east, but have no popular support, and appear to have only guiding control of trade and commerce in areas they are established. Due to the country's lack of infrastructure, these groups remain fairly isolated. In April 2021, the government of the DRC declared a state of siege over the provinces of North Kivu and Ituri in an effort to end insecurity and restore peace in Eastern DRC. Lieutenant General Johnny N'Kashama Luboya was appointed governor of Ituri to oversee these operations.

Following the peace accords of 2003, the international community embarked on significant economic investment programmes via various bilateral and multilateral agreements, such as with the World Bank, the European Union (EU), and various other international institutions and individual countries. China in particular has committed significant funds and has undertaken various large infrastructure projects mostly focused on rehabilitation of the road network.

Since 2003, the United Nations Organization Stabilization Mission in the DRC (MONUSCO) has been addressing the threat posed by armed groups and advancing peace and stability in the DRC. The UN Security Council resolution 2502 (December 2019) authorised a troop ceiling of 14,000 military personnel to be stationed throughout the country, mostly in the east. Although its mandate is mostly for monitoring the stability of the country, MONUSCO was authorised by the UN Security Council in June 2013 to be reinforced by a brigade with a mandate under Chapter Seven to actively neutralise rebel groups. This brigade was mainly constituted of troops from Tanzania and South Africa. A major UN base is located in the city of Beni (North Kivu province).

With the installation of a transitional government in 2003 after peace accords, economic conditions slowly began to improve as the government reopened relations with international financial institutions and international donors, and the DRC government began implementing reforms. The country's GDP growth averaged 6 % from 2005 to 2017 while the inflation rate has averaged 17 % for the same period with a remarkable inflation rate of 1 % between 2012 and 2015. After reaching 5.8 % in 2018, economic growth slowed to 4.4 % in 2019, owing to the decline in commodity prices, particularly for cobalt and copper, which account for over 80 % of the country's exports. In 2020, the DRC experienced its first recession in 18 years as a result of the adverse impacts of the coronavirus pandemic (COVID-19) across the world. The DRC's real GDP contracted by 1.7 % in 2020 after increasing by 4.4 % in 2019, stemming from mobility restriction, constrained government spending, and weaker exports caused by the global economic downturn.

The banking sector has been reinforced over the past 15 years with a host of international banks, mostly of African origin, having established operations. There are currently 18 commercial banks in the DRC. The official currency is the Congolese franc, although approximately 90 % of banking deposits and lending are in US dollars, and the prices of some goods, services and financial activities are indexed to the US dollar. More progress is needed in developing banking payment systems and in facilitating the use of financial activities.

Communications have vastly improved, with several major multinational networks having established themselves in the DRC, and growth in the international aviation network attests to growing investment in the country. Mining, agriculture, telecommunications, and manufacturing are steadily growing and developing.

24.2 DRC COMMUNITY AND SOCIAL ASPECTS

Socio-economic conditions in the DRC are still profoundly affected by the years of conflict in the country. Much of the DRC's population continues to live on a subsistence basis, primarily from cultivation of crops such as cassava, or fishing and hunting. Health and education services are poor or non-existent in many areas, although steady investment and assistance through various international organisations and non-government organisations (NGOs) are slowly improving the situation in some areas. Although much of the country is still agrarian, various urban centres are being revitalised via domestic and foreign investments and offer professional opportunities. A growing number of the Congolese Diaspora are returning to the DRC to pursue opportunities deemed to be more lucrative than in their adopted countries.

24.3 STATUS OF THE DRC MINERALS INDUSTRY

The DRC has historically been a significant minerals producer, mostly of gold, diamonds, copper, cobalt, and tin. The industry was started by private investments during the colonial period from 1885 to 1960, resulting in some very large industrial mining complexes which established entire towns through the country such as Mbjui Mayi, Lubumbashi, Kolwezi, Likasi, and others.

After independence, many of the large mining operations were nationalised and suffered from mismanagement and lack of reinvestment, such has been the case of Gecamines (focused on copper and cobalt in the Haut Katanga and Lualaba provinces), Okimo (focused on gold in Ituri, Haut Uélé and Tshopo Provinces), Sakima (focused on tin in South Kivu, North Kivu and Maniema provinces), and others. Production in these parastatal mining corporations collapsed and by the late 1990s was virtually non-existent.

In 2002, the DRC adopted a new mining law (the “2002 Mining Code”), whose redaction was sponsored by the World Bank. In March 2018, the 2002 Mining Code was amended and a new mining law was enacted (the “2018 Mining Code”). Along with the 2003 peace agreement, the 2002 Mining Code became a catalyst for a massive influx of mining and exploration capital into the country, with an estimated eight billion dollars having been invested since 2004. Much of this capital was focused on joint ventures with Gecamines in the Katanga region, but other provinces also saw significant investments. In 2019, the DRC became the world’s fourth largest copper producer, on a par with the United States of America and behind China, Peru and Chile. The world’s highest-grade copper deposit, Kamoa-Kakula, achieved full production in 2021 by Ivanhoe Mines and Zijin Mining Group in the Katanga province. The DRC is also by far the world’s largest producer of cobalt, accounting for roughly 60 % of global production. The DRC is also now a significant tin producer with the world’s highest-grade tin mine at Mpama North in North Kivu Province being brought into production by Alphamin Resources in 2020.

In the Haut Uélé province, the Kibali deposit, discovered in 2003 and having achieved first production in September 2013, has since been developed into one of the world’s largest gold mines and a significant catalyst for further exploration and development in the province. In 2019 and 2020, the Kibali mine (managed by Barrick Gold Corporation) produced a record 814,027 oz and 808,134 oz of gold, respectively, demonstrating the ability to successfully develop and operate a modern top-tier gold mine in one of the world’s most remote and infrastructurally under-endowed regions. The Kibali mine, which is approximately 220 km from Loncor’s Adumbi deposit, is now Africa’s largest gold producer.

24.4 DRC MINERALS INDUSTRY POLICIES

Approximately 10 years after the DRC 2002 Mining Code was originally adopted, a revision process, which started in 2012, led to a bill that was finally approved by both houses in January 2018 and signed into law in March 2018.

A summary of the key changes introduced by the 2018 Mining Code is given below.

24.4.1 Available Mining Rights

Mining rights available under the 2018 Mining Code include the following:

- An exploration permit (PR), standardised to all minerals and granted for five years, renewable once for the same term
- A mining permit (PE), granted for 25 years, renewable for periods of up to 15 years

These mining rights can now only be granted to legal entities and not to natural persons.

24.4.2 Royalties and Taxes

An increase in royalties and taxes was among the principal innovations of the 2018 Mining Code, which include the following:

- Royalty rates increased from 2 % to 3.5 % for non-ferrous and base metals and from 2.5 % to 3.5 % for precious metals, while precious stones royalties increased from 4 % to 6 % and are calculated on the gross market value of the products.
- A special 10 % royalty was created on minerals deemed by the State to be “strategic substances”, defined as minerals which, on the basis of the State’s opinion of the prevailing economic environment, were of special interest given the critical nature of such mineral and the geo-strategical context. It is anticipated that the list would include cobalt, coltan, lithium and germanium, which have become leading mining commodities with the increased demand for electric vehicles and grid storage technology. DRC is a major global producer of these substances.
- While corporate income tax remained at a reduced rate of 30 % for mining companies, a new “super profits” tax of 50 % was created on excess profits, defined as profits made when a commodity exceeds by 25 % the price used in the bankable feasibility study.
- The holder of a PE (or a PR) in the DRC is subject to a tax on the surface area of the PE (PR) payable in Congolese francs at a rate equivalent to US$0.40/ha for the first year (US$0.20/ha for PRs); US$0.60/ha for the second year (US$0.30/ha for PRs); US$0.70/ha for the third year (US$0.35/ha for PRs); and US$0.80/ha for each subsequent year (US$0.40/ha for PRs).
- In addition to the surface area tax, the holders of a PE are subject to an annual area rights tax of the equivalent in Congolese francs of US\$588.96/m^2.The annual area rights tax for the holder of a PR is as follows: US$3.53/ha for the first two years; US$36.52/ha for each year following the first two years, and US$60.04/ha for every year of renewal of the PR.

24.4.3 Contracting Requirements

The 2018 Mining Code requires mining companies to comply with Local Law 17/001 of February 2017, requiring contractors to be Congolese and contracting companies to be owned by Congolese shareholders. While unclear, it is generally accepted that this means the Congolese contractor’s company must be majority owned by Congolese shareholders. Furthermore, in concluding services contracts for mining activities (not including contracts for the sale of goods), priority must also be given to Congolese companies. In this regard, any services contracts concluded with a foreign company are subject to a 14 % tax on amounts paid under such contract.

24.4.4 Other Notable Amendments

Other notable amendments are as follows:

- The State's free-carried shareholding in the mining company was increased from 5 % to 10 %, increased by 5 % each time the permit is renewed. Furthermore, at least 10 % of the capital must be owned by Congolese citizens, which is a development that has attracted industry concern.
- The exportation of raw minerals was forbidden, and mining permit holders must now present a plan for the refinement of their minerals to the mining authorities. A one-year derogation may be obtained if a company shows that it is impossible to transform the minerals locally.
- The requirements relating to State approvals for transfers, farm-outs and option contracts were expanded, including a new requirement that changes of control (including certain share transfers) in companies holding a mining permit are subject to State approval.
- Access to a documented state-studied deposit, secured by tender, will be subject to the payment to the State of an entry fee amounting to 1 % of the price paid for the tendered deposit.
- The stability period during which taxes and customs cannot be modified was reduced from 10 to 5 years. While existing mining rights are subject to the provisions of the new law, it is unclear to what extent existing mining agreements with stabilisation provisions will be affected.
- Companies must now establish a provision of 0.5 % of turnover for mine rehabilitation.

24.5 DRC POLITICAL RISK

The following is taken from Loncor’s 2020 Annual Report on Form 20-F publicly filed by Loncor on SEDAR and EDGAR.

Loncor's projects are located in the DRC. Loncor’s assets and operations are therefore subject to various political, economic and other uncertainties, including, among other things, the risks of war and civil unrest, hostage taking, expropriation, nationalisation, renegotiation or nullification of existing licences, permits, approvals and contracts, taxation policies, foreign exchange and repatriation restrictions, changing political conditions, international monetary fluctuations, currency controls and foreign governmental regulations that favour or require the awarding of contracts to local contractors or require foreign contractors to employ citizens of, or purchase supplies from, a particular jurisdiction. Changes, if any, in mining or investment policies or shifts in political climate in the DRC may adversely affect Loncor's operations or profitability. Operations may be affected in varying degrees by government regulations with respect to, but not limited to, restrictions on production, price controls, export controls, currency remittance, income taxes, foreign investment, maintenance of claims, environmental legislation, land use, land claims of local people, water use and mine safety. Failure to comply strictly with applicable laws, regulations and local practices relating to mineral rights could result in loss, reduction or expropriation of entitlements. In addition, in the event of a dispute arising from operations in the DRC, Loncor may be subject to the exclusive jurisdiction of foreign courts or may not be successful in subjecting foreign persons to the jurisdiction of courts in Canada. Loncor also may be hindered or prevented from enforcing its rights with respect to a governmental instrumentality because of the doctrine of sovereign immunity. It is not possible for Loncor to accurately predict such developments or changes in laws or policy or to what extent any such developments or changes may have a material adverse effect on Loncor's operations. Should Loncor's rights or its titles not be honoured or become unenforceable for any reason, or if any material term of these agreements is arbitrarily changed by the government of the DRC, Loncor's business, financial condition and prospects will be materially adversely affected.

Some or all of Loncor's properties are located in regions where political instability and violence are ongoing. Some or all of Loncor's properties are inhabited by artisanal miners. These conditions may interfere with work on Loncor's properties and present a potential security threat to Loncor's employees. There is a risk that activities at Loncor’s properties may be delayed or interfered with, due to the conditions of political instability, violence, hostage taking or the inhabitation of the properties by artisanal miners. Loncor uses its best efforts to maintain good relations with the local communities in order to minimise such risks.

The DRC is a developing nation emerging from a period of civil war and conflict. Physical and institutional infrastructure throughout the DRC is in a debilitated condition. The DRC is in transition from a largely state-controlled economy to one based on free market principles, and from a non-democratic political system with a centralised ethnic power base, to one based on more democratic principles. There can be no assurance that these changes will be effected or that the achievement of these objectives will not have material adverse consequences for Loncor and its operations. The DRC continues to experience instability in parts of the country due to certain militia and criminal elements. While the government and United Nations forces are working to support the extension of central government authority throughout the country, there can be no assurance that such efforts will be successful.

No assurance can be given that Loncor will be able to maintain effective security in connection with its assets or personnel in the DRC where civil war and conflict have disrupted exploration and mining activities in the past and may affect Loncor's operations or plans in the future.

HIV/AIDS, malaria and other diseases represent a serious threat to maintaining a skilled workforce in the mining industry in the DRC. HIV/AIDS is a major healthcare challenge faced by Loncor's operations in the country. There can be no assurance that Loncor will not lose members of its workforce or workforce man-hours or incur increased medical costs, which may have a material adverse effect on Loncor's operations.

The DRC has historically experienced relatively high rates of inflation.

25 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

The Qualified Persons (QPs) note the following interpretations and conclusions in their respective areas of expertise, based on the review of the information available for this technical report.

25.2 GEOLOGY AND MINERALISATION

The Adumbi gold deposit is found within the Ngayu Archean greenstone belt, one of a number of Archean-aged, granite-greenstone belts that extend from northern Tanzania, into northeastern DRC and then into the Central African Republic. These gold belts contain a number of major gold mines including Kibali (DRC) and Geita, North Mara and Bulyanhulu (Tanzania). Gold deposits within these belts are associated with the globally important Neo-Archean orogenic gold deposits, examples of which are found in most Neo-Archean cratons around the world.

Gold mineralisation in these greenstone belts is commonly of the fracture and vein type in brittle fracture to ductile dislocation zones. At the Adumbi deposit, the gold mineralisation is generally associated with quartz and quartz-carbonate-pyrite ± pyrrhotite ± arsenopyrite veins in a BIF unit. Examples of similar type BIF hosted gold deposits to Adumbi include the major Geita mine in Tanzania and Kibali mine in northeastern DRC.

Most of the gold occurrences within the Ngayu belt are located within or close to the contact of the BIF. Historically, only two deposits were exploited on any significant scale, Yindi and Adumbi, where gold mineralisation is found within the BIF units. Artisanal mining of weathered gold mineralisation preserved as alluvial, eluvial or colluvial material is widespread throughout the Ngayu belt. Within the Imbo Project area in the eastern part of the Ngayu belt, there is a strong association between gold mineralisation and the presence of the BIF, the BIF either constituting the host rock (e.g. Adumbi and Yindi) or forming a significant part of the local stratigraphy. The BIF forms favourable physical and chemical traps for mineralising hydrothermal fluids. Besides Adumbi, there are a number of smaller gold deposits (Kitenge and Manzako) and prospects within the Imbo Project.

25.3 EXPLORATION, DRILLING AND ANALYTICAL DATA COLLECTION IN SUPPORT OF MINERAL RESOURCE ESTIMATION

Systematic exploration has been conducted on the Adumbi deposit and Imbo Project area, including airborne LiDAR and geophysical surveys, gridding, geological mapping, soil, trench, adit and auger sampling together with a number of core drilling programmes. Sampling, sample storage, security, sample preparation and geochemical analyses and verification are considered appropriate for the resource estimate at Adumbi.

25.4 MINERAL RESOURCE METHODOLOGY AND ESTIMATION

The mineral resource estimate for Adumbi has been undertaken according to the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves, as incorporated in NI 43-101 requirements. The data used for the resource estimate and methods employed are considered reasonable for the level of study for this PEA.

25.5 METALLURGICAL TEST WORK

For the Adumbi deposit, representative core sample composites were selected for metallurgical test work from a range of depths and along strike and for the various mineralised host lithologies and styles of mineralisation. These representative samples were then submitted to an independent metallurgical laboratory for diagnostic test work to determine metallurgical recovery estimates using appropriate processing routes. The metallurgical test work undertaken is considered appropriate for the level of this study.

25.6 OPEN-PIT OPTIMISATION AND MINERAL INVENTORY

The Mineral Inventory Statement undertaken on the Adumbi deposit uses the definitions and guidelines given in the CIM Definition Standards on Mineral Resources and Mineral Reserves and is reported in accordance with NI 43-101 requirements. The Adumbi Mineral Inventory for the various material types (oxide, transition and fresh) contained within the Adumbi practical pit designs consists of 1.883 Moz (28.185 Mt grading 2.08 g/t Au) of Indicated mineral resources and 1.777 Moz (20.828 Mt grading 2.65 g/t Au) of Inferred mineral resources.

Pit optimisation assumptions and parameters used to constrain the depth extent of the geological model to generate the mineral inventory of the open pit for the Adumbi deposit are considered appropriate for its location and infrastructural setting with appropriate metallurgical recoveries used from the test work and a gold price of US$1,600/oz, which is below current levels.

In the QP’s opinion, the parameters used in the Mineral Resource to Mineral Inventory conversion process are reasonable.

25.7 MINING PLAN AND METHODS

Over the LOM of Adumbi, a total of 49.77 Mt grading 2.17 g/t Au is expected to be mined from the open pit and delivered to the processing plant. The scheduling process consisted of developing a mine plan that is economically optimal using the inventory included in the practical pit designs and pit shell. The production schedule was based on a methodology of block selections from the block model inside the pit outlines. The Adumbi deposit is planned to provide a throughput of 5.0 Mt/a of ore to the processing plant.

The size and type of mining equipment is consistent with the size of the project, and equipment units and costs have been selected from major mining manufacturers. Although quotes have not been specifically requested for this project, the capital and transportation costs have been derived from similar projects from the relevant equipment suppliers. Outputs have been derived from generic calculations. The mine plan and estimated mine capital and operating costs are considered reasonable for this level of study. Contract mining and associated costs were estimated for Adumbi.

25.8 PROCESS RECOVERY PLANT

The processing plant is planned to process 49.77 Mt of material with an average head grade of 2.17 g/t Au and an average metallurgical recovery of 89.79 % to produce 3.12 Moz of gold. The process plant flowsheet designs were based on the metallurgical test work results and industry-standard practices. The flowsheet was developed for optimum recovery while minimising CAPEX and LOM OPEX. The process methods are conventional, with comminution and recovery processes widely used in the industry.

25.9 INFRASTRUCTURE AND ACCESSIBILITY

The Adumbi deposit is situated in a remote part of the northeastern DRC, and infrastructure is limited. As part of the PEA study, a team of consulting engineers carried out site investigations at Adumbi to assess optimal positions for key infrastructure components of the mine site as well as assessing potential tailings storage sites and HEP, diesel and PV power sources. The site selection process to design a tailings management system to cater for 50 Mt over the LOM was based on a multi-criteria analysis and qualitative risk analysis, which aimed to determine the most favourable location for the TSF footprint. Preliminary HEP sites were visited around Adumbi, indicating that a significant component for the estimated 32 MW requirement for the Adumbi operation could be obtained from an HEP facility.

A number of potential access land routes were assessed from the major Port of Mombasa in Kenya via Uganda to Adumbi. The preferred route chosen is via Kenya and Uganda to Durba in DRC where Barrick Gold/AngloGold Ashanti's Kibali Gold Mine is situated and then via Isiro and Wamba to Adumbi. Raw water for the project will be abstracted from the rivers in the area, which have significant flow throughout the year.

In the QP's opinion, the estimated infrastructure CAPEX and OPEX are considered reasonable for this level of study.

25.10 ENVIRONMENTAL AND SOCIAL CONSIDERATIONS

The objectives of the environmental and social component of the Adumbi PEA were to evaluate the adequacy of the environmental and social studies undertaken to date and determine, at a screening level, the potential environmental and social issues associated with the proposed mine development. This included identifying key issues, fatal flaws, high-level environmental and social risks that may affect the project and require further investigation, and identifying costs for further environmental and social studies.

The material presented was to meet the requirements as per the DRC *Code Minier* or Mining Code, Law No. 007/2002 of July 11, 2002, as amended and completed by Law No. 18/001 of March 9, 2018. Information was obtained from the September 2021 site visit by the environmental consultant where an overview of the very limited environmental and social baseline data was assessed. Key legislation includes the 2006 Constitution of the Third Republic (The Constitution), the 1997 National Environmental Action Plan (NEAP) providing a framework for the management of the DRC's natural resources, as well as the Mining Code governing commercial and artisanal mining activities. The latter requires, inter alia, an Environmental and Social Impact Study (ESIS), a Mitigation and Rehabilitation Plan (MRP), and an Environmental and Social Management Plan of the Project (ESMPP). Mining Regulations (Decree No. 038/2003 of March 26, 2003, as amended and completed by Decree No. 18/024 of June 8, 2018) give effect to the Code in terms of environmental social obligations, public consultation and requirements for ESIS and ESMPP reports. The Framework Law on the Environment (Act No.11/009 of July 9, 2011) aims to define the protection of the environment and the direct management of natural resources, and to reduce pollution and serve as a basis for sectors impacting the environment.

The environmental assessment of the project included knowledge from other recent operating gold mines and other development projects in eastern DRC. The key environmental and social issues identified during this initial screening study that have possible impacts included surface water, air quality, biodiversity, climate, infrastructure, safety and security, community development, resettlement, land acquisition, land replacement and compensation, health and safety, labour and artisanal mining, project engagement and policies. On the basis of the available information, screening studies undertaken did not identify any fatal flaws from an environmental and socio-economic perspective. Key issues were identified that would need to be investigated further during the PFS and FS phases of the project.

Six artisanal mining villages identified within the project footprint have been assessed for social repositioning. The total population of these villages is estimated to be approximately 5,800. The legal situation regarding compliance with the DRC resettlement plan process, assessment and reporting requirements is unclear on the repositioning of artisanal mining huts and in other mine developments in DRC, artisanal miners were not compensated or resettled. Future studies (ESIS, ESMPP and MRP) will be undertaken to clarify this situation. However, a conservative approach has been taken in the estimation of costs for social repositioning of the six artisanal settlements that will be impacted by the project development. It is believed that if the livelihood restoration programmes are accepted for the repositioning of artisanal miners within the Adumbi mine footprint, the capital included in the pre-production capital costs under Owner's costs could be significantly lower than estimated in this PEA study for environmental and social considerations.

25.11 MARKET STUDIES AND CONTRACTS

Terms and conditions included as part of sales contracts are expected to be typical of similar contracts for the sale of doré throughout the world. Loncor plans to contract out the transportation, security, insurance and refining of the doré gold bars.

25.12 CAPITAL AND OPERATING COSTS

25.12.1 CAPEX Estimate

Pre-production capital costs are estimated for two power cases: HEP hybrid and diesel only.

For the HEP hybrid case, pre-production capital costs are estimated at US$530.05 million including US$74.42 million of contingencies while sustaining capital costs are estimated at US$304.962 million. Of the initial pre-production capital total, US$152.45 million is related to an HEP facility. However, Loncor is already in discussion with potential power suppliers with experience in the DRC to project finance and build a hydroelectric facility at Adumbi and then have an offtake agreement with Loncor to supply power for the mining operation. Such an agreement would improve the financial economics of the project by reducing the initial owner CAPEX for the HEP hybrid case.

For the diesel-only case, pre-production capital costs are estimated at US$392.01 million including US$62.96 million of contingencies. Sustaining capital for the diesel-only case is estimated at US$410.55 million.

In the QP's opinion, the CAPEX for Adumbi is considered reasonable for this level of study.

25.12.2 OPEX Estimate

The average LOM total cash costs for Adumbi are estimated at US$852/oz for the HEP hybrid case and US$908/oz for the diesel-only case, based on a diesel price of US$0.90/L.

The AISC are estimated at US$950/oz for the HEP hybrid case and US$1,040/oz for the diesel-only case.

25.13 ECONOMIC ANALYSIS

Cash flow valuation models for the Adumbi HEP hybrid and diesel-only options were developed by the QPs by taking into account estimated annual processed tonnages, grades and recoveries, metal prices, site operating and total cash costs including royalties and refining charges, and initial pre-production and sustaining CAPEX estimates. The financial assessment of Adumbi has been carried out on a "100 % equity" basis for both pre-tax and post-tax considerations. The financial analysis assumed a base gold price of US$1,600/oz of gold, a two-year construction period, and an operating mine life of 10.3 years.

The pre-tax net present value discounted at 5 % (NPV at 5 %) is US$895 million for the HEP hybrid case and US$843 million for the diesel-only case.

The post-tax net present value discounted at 5 % (NPV at 5 %) is US$624 million for the HEP hybrid case and US$600 million for the diesel-only case.

Calculated sensitivities to the Adumbi base case gold price of US$1,600/oz show significant upside leverage to the gold price and the robust nature of the projected economics to the CAPEX and OPEX assumptions.

25.14 POSSIBLE RISKS AND UNCERTAINTIES TO THE FUTURE DEVELOPMENT OF ADUMBI

Considering Adumbi is at an early stage of development, with PFS and FS required before the project can advance to the development stage, a number of risks have been outlined below:

- Risks to the resource estimate resulting from future drilling
- Risks related to the interpretations of mineralisation geometry and continuity in the mineralised zones
- Changes to geotechnical and hydrogeological assumptions
- Changes to power assumptions
- Changes to historical mining assumptions

- Future metallurgical test work yielding results that vary from the current test work undertaken with lower metallurgical recoveries
- Estimated capital and operating costs varying from the current estimates
- Changes to the assumed gold price of US$1,600/oz used in this PEA model
- Delays in progressing the project due to security problems
- Inability of Loncor to obtain sufficient financing to advance Adumbi to the PFS and FS stages. Also, any future mine construction at Adumbi would require significant financing. No assurance can be provided that such financing would be available to Loncor.

The future PFS and FS stages will aim to reduce the risks and uncertainties associated with future development of Adumbi.

26 RECOMMENDATIONS

Based on the positive results of this PEA, further work is warranted at Adumbi to advance the project up the value curve by completing follow-up feasibility studies on the project. A number of opportunities have been identified to increase the mineral resources and enhance and increase the economics and financial returns at Adumbi. It is recommended that Loncor follow up on these opportunities, which include the following:

- **Increasing and Upgrading Mineral Resources at Adumbi and within the Imbo Project**
  There is excellent exploration potential to further increase the mineral resources at Adumbi and within the Imbo Project. At Adumbi, the mineralised BIF host sequence increases in thickness below the open-pit shell, and wide-spaced drilling has already intersected grades and thicknesses amenable to underground mining. Further drilling is required to initially outline a significant underground Inferred Mineral Resource which can then be combined with the open-pit mineral resource so that studies can be undertaken for a combined open-pit and underground mining scenario at Adumbi. It is also recommended that infill drilling be undertaken in the deeper part of the open-pit shell to upgrade the current Inferred resources into the Indicated category.

  Minecon recommends that any deep boreholes at Adumbi be initially drilled and cased by a reverse circulation (RC) rig and followed up in the mineralised zone with a core rig. This drilling procedure should reduce the metreage unit costs and time to complete the drilling at Adumbi.

  Besides increasing the resource base, a combined open pit/underground project could increase grade throughput and reduce strip ratios with the higher grade, deeper mineral resources being mined more economically by underground mining methods, which could increase annual gold production and drive down operating costs. Minecon also recommends that further studies should be undertaken to assist in estimating historical depletions and depletions by recent artisanal mining.

  Additional deposits and prospects occur close to Adumbi and have the potential to add mineral resources and feed to the Adumbi operation. Along trend from Adumbi, the Manzako and Kitenge deposits have Inferred Mineral Resources of 303,000 oz of gold (1.68 Mt grading 5.80 g/t Au) and remain open along strike and at depth. Further drilling is warranted on these two deposits.

  Along the structural trend, 8 km to 13 km to the southeast across the Imbo River and within the Imbo Project, four prospects (Esio Wapi, Paradis, Museveni and Mungo Iko) with similar host lithologies to Adumbi have been outlined with soil, rock and trench geochemical sampling. An initial shallow scout drilling programme should be undertaken on these four prospects to determine their mineral resource potential.

  A total of 24,000 m of drilling (including 7,600 m RC drilling and 16,400 m of coring in the mineralised zone) is recommended by Minecon. This would include infill, deep and extension drilling. The proposed drilling programme should be undertaken in two sequential phases:

- Phase 1: Deep holes outside the pit outline will be pre-collared with RC drilling and completed in the mineralised zone using core drilling.
- Phase 2. Infill, moderately deep and shallow holes within the pit outline will be drilled, with the deeper holes pre-collared with RC drilling and tailed off using core drilling while the shallow holes are drilled by RC.

- **Additional Mineral Resources within the Ngayu Greenstone Belt**
  Additional feed for the Adumbi processing plant could also come from Loncor's 100 % owned high-grade Makapela deposit, where Indicated Mineral Resources of 2.20 Mt grading 8.66 g/t Au (614,200 oz of gold) and Inferred Mineral Resources of 3.22 Mt grading 5.30 g/t Au (549,600 oz of gold) have been outlined to date, with the high-grade material being able to be transported economically to Adumbi.

- **Additional geotechnical investigations:**
  - Additional geotechnical investigations including drilling are recommended to optimise and potentially steepen pit slopes especially for the competent fresh BIF host rock which could reduce the strip ratio and thereby lower mining costs at Adumbi.
  - An in-depth analysis of the waste rock should be conducted to ascertain whether it can be utilised for construction of the TSF embankment.

- **Further metallurgical test work:**
  - Additional metallurgical test work, including additional flotation and petrographic studies, is recommended to confirm recoveries and reagent consumptions, and to optimise the flowsheet design.
  - Test work on tailings is recommended to understand the chemistry and rheology of the tailings.

- **Concluding HEP Hybrid Power Agreement with Third Parties**
  As described in this PEA study, hydroelectric sites have already been identified close to Adumbi, and further studies are required to optimise the power generation concept for the operation. Loncor is already in discussion with potential power suppliers with experience in the DRC to project finance and build a hydroelectric facility at Adumbi and then have an offtake agreement with Loncor to supply power for the mining operation. Such an agreement would improve the financial economics of the project.

  To improve the inputs into the hydrological models, it is recommended that flow/level gauges be installed at critical points along the Ngayu and Imbo rivers.

SENET and Minecon estimate that the recommended drilling and other studies will cost approximately US$17.551 million and take 12 months to complete. As part of this work plan, it is recommended that a PFS be undertaken in the final quarter of 2022 to outline updated mineral resources and ore reserves, as well as to update this PEA. The recommended scope and budget are detailed in Table 26.1.

Table 26.1: Proposed Budget for Follow-Up Work on the Adumbi Deposit and Imbo Project

Description	Amount (US$)
Adumbi drilling to outline Inferred Resources below pit shell (10 boreholes totalling 7,400 m DD and 2,600 m RC)	2,716,000
Adumbi drilling within pit shell to upgrade Inferred Resources into the Indicated category (51 drillholes totalling 6,600 m DD and 6,400 m RC)	3,972,000
Imbo East scout drilling on four prospects (1,600 m – 12 boreholes)	360,000
Sample preparation and geochemical analysis	740,000
Metallurgical test work and petrographic studies	510,000
Monitoring of water levels at Ngayu and Imbo rivers	180 000
Modelling Mineral Resource/Reserve Estimation	175,000
Geotechnical drilling	100,000
Environmental and Social Impact Assessment (ESIA) – PFS Level	390,000
Independent engineering PFS	500,000
Salaries and wages	2,404,000
Management fees	360,000
Camp support (security, travel, camp, communications, vehicle, etc.)	3,378,000
Capital equipment	171,000
Subtotal	**15,956,000**
Contingency (10 %)	1,595,000
Total	**17,551,000**

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DATE AND SIGNATURE PAGE

This report entitled “NI 43-101 Preliminary Economic Assessment of the Adumbi Deposit in the Democratic Republic of the Congo” was prepared for Loncor Gold Inc. by Minecon Resources and Services Limited and New SENET (Pty) Ltd. This report, the effective date of which is December 15, 2021, is compliant with Canadian National Instrument 43-101 (NI 43-101) and Form 43-101F1. The qualified persons (within the meaning of NI 43-101) for the purposes of this report are Daniel Bansah, Christian Bawah and Philemon Bundo, who have signed below.

Signed on January 28, 2022

(signed) *“Daniel Bansah”*

Daniel Bansah

MSc (MinEx), MAusIMM (CP), FWAIMM, MGhIG

Chairman and Managing Director

Minecon Resources and Services Limited

Accra, Ghana

(signed) *“Christian Bawah”*

Christian Bawah

BSc (Hons) Geology, MBA (Finance), MAusIMM (CP), MMCC, FWAIMM, MGhIG

Director, Geology and Mineral Exploration

Minecon Resources and Services Limited

Accra, Ghana

(signed) *“Philemon Bundo”*

Philemon Bundo

BSc Eng Hons (Metallurgy), FAusIMM (Aus)

Senior Vice President

New SENET (Pty) Ltd

Johannesburg, South Africa

CERTIFICATE OF QUALIFIED PERSON – DANIEL BANSAH

I, Daniel Bansah, do hereby certify that

1. I reside at No. 8, Kweku Mensah Street, Adjiringanor, Accra, Ghana. Box CT 4096 Cantonments, Accra, Ghana.

2. I am a graduate with a Master of Science with Distinction in Mineral Exploration gained from the University of Leicester, UK, in 1998, and I have practised my profession continuously since July 1988.

3. I am a Fellow of the West African Institute of Mining, Metallurgy and Petroleum (Membership Number 074), a chartered professional member of the Australasian Institute of Mining and Metallurgy (Membership Number 208213), and a professional member of the Ghana Institute of Geoscientists (Membership Number 188).

4. I am the Chairman and Managing Director of Minecon Resources and Services Limited, a firm of consulting geology, mining and petroleum engineers.

5. I have experience with precious metal deposits and resource estimation techniques. I have worked as a geologist for over 30 years since my graduation. My relevant experience for the purposes of the technical report (the "Technical Report") dated December 15, 2021, entitled "NI 43-101 Preliminary Economic Assessment of the Adumbi Deposit in the Democratic Republic of the Congo" is as follows:

    - Reviewed various reports as a consultant on numerous exploration and mining projects in Ghana and the African region for due diligence studies.
    - Head of Projects and Operations (from 2013 to 2018) with a Canadian gold mining company exploring and developing world-class gold assets in northeastern DRC and responsible for the management of two operating gold mines, two advanced exploration projects, and an extensive regional exploration portfolio with over 16 prospective targets.
    - Vice President – Exploration (2007 to 2013) with a Canadian gold exploration and development company, exploring and developing world-class gold assets in northeastern DRC and responsible for the management of two development gold projects, two advanced exploration projects and an extensive regional exploration portfolio with over 16 prospective targets.
    - Group Mineral Resources Manager (from 2004 to 2007) with a Canadian gold exploration company exploring world-class gold assets in northeastern DRC and responsible for mineral resource development and management.
    - Group Mineral Resources Manager (from 1998 to 2004) with a Ghanaian gold mining, development and exploration company, with 7 world-class operations and an extensive development and exploration portfolio in 17 African countries, and responsible for mineral resource and mineral reserve development, auditing, management and training.
    - Senior Mineral Resources/Senior Geologist Exploration/Project Geologist/Geologist (from 1989 to 1998) with a Ghanaian gold mining, development and exploration company, with 7 world-class operations and an extensive development and exploration portfolio in 17 African countries, and responsible for the mineral resource modelling and grade estimation and mineral exploration project management.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

7. I am responsible for Sections 2 to 5, Sections 11 and 12, Sections 14 to 16, Section 20, Section 23 and part of Sections 1 and 24 to 27 of the Technical Report.

8. I have visited the Imbo Project including the Adumbi deposit, many times, with the most recent visit being in September 2021.
9. I am independent of Loncor Gold Inc. as described in Section 1.5 of NI 43-101.
10. I have not had any prior involvement with the property which is the subject of the Technical Report.
11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101, including Form 43-101F1, and in conformity with generally accepted international mining industry practices.
12. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 28th day of January 2022.

(Signed) *"Daniel Bansah"*

____

DANIEL BANSAH

MSc (MinEx), MAusIMM (CP), FWAIMM, MGhIG

Chairman and Managing Director

Minecon Resources and Services Limited

CERTIFICATE OF QUALIFIED PERSON – CHRISTIAN BAWAH

I, Christian Bawah, do hereby certify that

1. I reside at K506 Nii Obaayoo Street, Adjiringanor, East Legon, Accra, Ghana. Box YK 431 Kanda, Accra, Ghana.
2. I am a graduate with a Master of Business Administration with Merit in Finance gained from the University of Leicester Business School, UK, in 2013, a holder of a Mine Managers Certificate of Competency from the Inspectorate Division of the Minerals Commission of Ghana in 2012, and a Bachelor of Science (Honours) in Geology with Physics from the University of Ghana in 1994. I have practised my profession as a geologist continuously since August 1994.
3. I am a chartered professional member (in Geology) of the Australasian Institute of Mining and Metallurgy (Membership Number 227522), a Fellow of the West African Institute of Mining, Metallurgy and Petroleum (Membership Number 1377), and a professional member of the Ghana Institution of Geoscientists (Membership Number 189).
4. I am the Executive Director, Geology and Mineral Exploration of Minecon Resources and Services Limited, a firm of consulting geology, mining and petroleum engineers.
5. I have considerable experience in gold exploration techniques in Africa, as well as mining project development and operations. I have worked in the mining industry for over 26 years since my graduation. My relevant experience for the purposes of the technical report (the "Technical Report") dated December 15, 2021, entitled "NI 43-101 Preliminary Economic Assessment of the Adumbi Deposit in the Democratic Republic of the Congo" is as follows:

    - Have been involved with geological consultancy work and have reviewed various reports on numerous exploration and mining projects in Ghana and the African region for due diligence studies.
    - General Manager (from 2013 to 2018) with a Canadian gold mining, exploration and development company exploring, developing and operating world-class gold assets in northeastern DRC and responsible for overseeing the redesign and completion of project development, commissioning, and running the operations.
    - Deputy General Manager (2012 to 2013) with a Canadian gold mining, exploration, and development company, exploring, developing, and operating world-class gold assets in northeastern DRC and responsible for mining operations, mining geology and near mine exploration.
    - Mineral Resources Manager (from 2011 to 2013) with a Canadian gold mining, and exploration and development company exploring, developing, and operating world-class gold assets in northeastern DRC and responsible for mineral resource development and management, mining production geology, mine to mill reconciliation, and near mine exploration.
    - Chief Geologist (from 2007 to 2011) with a Canadian gold mining and exploration and development company exploring world-class gold assets in northeastern DRC and responsible for exploration from grass roots through scoping, pre-feasibility and full-feasibility studies. Was part of the project development team during the mine construction.
    - Senior Project Geologist (from 2004 to 2007) with a Canadian gold exploration company exploring world-class gold assets in northeastern DRC and responsible for setting up and running two of the company's key exploration projects.
    - Exploration Geologist (from 1996 to 2004) with a Ghanaian gold mining, development and exploration company, with 7 world-class operations and an extensive development and exploration portfolio in 17 African countries and supervised exploration projects in Ghana, Mali Côte d'Ivoire, and Guinea.
    - Geologist (from 1995 to 1996) with a Ghanaian gold mining and exploration company, a global multinational precious metal producer presently the largest gold producer in Ghana. Was involved with near mine exploration activities.

- Teaching/Research Assistant (from 1994 to 1995) with the Geology Department of the University of Ghana, and was responsible for students' tutorials and practical lessons, filing of mapping exercises, and assisting lectures with research work.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am responsible for Sections 6 to 10 and part of Section 27 of the Technical Report.
8. I visited the Imbo Project including the Adumbi deposit, from October 8 to November 27, 2020.
9. I remotely managed and reviewed drilling activities on site from November 28, 2020, to end November 2021.
10. I am independent of Loncor Gold Inc. as described in Section 1.5 of NI 43-101.
11. I have not had any prior involvement with the property which is the subject of the Technical Report.
12. I have read NI 43-101, and the part of the Technical Report that I am responsible for has been prepared in compliance with NI 43-101, including Form 43-101F1, and in conformity with generally accepted international mining industry practices.
13. At the effective date of the Technical Report, to the best of my knowledge, information and belief, the part of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 28th day of January 2022.

(Signed) *"Christian Bawah"*

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CHRISTIAN BAWAH

BSc (Hons) Geology, MBA (Finance), MAusIMM (CP), MMCC, FWAIMM, MGhIG

Director, Geology and Mineral Exploration

Minecon Resources and Services Limited

CERTIFICATE OF QUALIFIED PERSON – PHILEMON BUNDO

I, Philemon Bundo, do hereby certify that

1. I am a Senior Vice President – Process at New SENET (Pty) Ltd, Building 12, Greenstone Hill Office Park, Emerald Boulevard, Greenstone Hill, Greenstone 1609, Modderfontein, Gauteng, South Africa.
2. I am a qualified person for the purposes of the technical report (the "Technical Report") entitled "NI 43-101 Preliminary Economic Assessment of the Adumbi Deposit in the Democratic Republic of the Congo", prepared for Loncor Gold Inc. (Loncor), with an effective date of December 15, 2021.
3. I am a graduate of the University of Zimbabwe, with a BSc Honours in Metallurgical Engineering.
4. I am a Fellow of the Australasian Institute of Mining and Metallurgy.
5. I have practised my profession continuously since 1991. I have over 31 years diversified experience in the operational management and technical services areas of the mineral processing industry. I have been involved in the process operation (production) and plant design, from conceptualisation to complete project execution, of more than 50 mineral process projects, as well as more than 100 process plant studies for major commodities including cobalt, copper, gold, and uranium. I have assisted in or compiled National Instrument 43-101 (NI 43-101) Reports for various projects that have been listed on the TSX stock exchange.
6. I have read the definition of "qualified person" set out in NI 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
7. I have not visited the project site.
8. I performed consulting services and reviewed files and data supplied by Loncor between July 2021 and December 2021.
9. I am responsible for the preparation of Sections 13, 17, 18, 19, 21 and 22 and part of Sections 1, 24, 25, 26 and 27 of the Technical Report.
10. I have had no previous involvement with this project or any other project on this property.
11. I am independent of Loncor as independence is described in Section 1.5 of NI 43-101. I do not have nor do I expect to receive a direct or indirect interest in the Mineral Properties of Loncor, and I do not beneficially own, directly or indirectly, any securities of Loncor or any associate or affiliate of such company.
12. I have read NI 43-101 and Form 43-101F1, and the part of the Technical Report for which I am responsible has been prepared in compliance therewith.
13. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to ensure that the Technical Report is not misleading.

Signed at New SENET (Pty) Ltd, Johannesburg, South Africa on the 28th day of January 2022.

(Signed) *"Philemon Bundo"*

PHILEMON BUNDO

BSc Eng Hon (Metallurgy) FAusIMM (Aus)