NEWCORE GOLD

NI 43-101 Technical Report

PRELIMINARY ECONOMIC ASSESSMENT FOR THE ENCHI GOLD PROJECT

Enchi, Ghana

Prepared for:
Newcore Gold Ltd.

Effective Date: June 8, 2021
Signature Date: July 13, 2021

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BBA

sems EXPLORATION

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This technical report is effective as of the 8th day of June 2021.

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This certificate applies to the Technical Report entitled "NI 43-101 Technical Report for the Preliminary Economic Assessment of the Enchi Gold Project, Ghana" prepared for Newcore Gold Ltd., dated July 13, 2021 (the "Technical Report"), and effective as of June 8, 2021.

I, Todd McCracken, P. Geo., as a co-author of the Technical Report, do hereby certify that:

1. I am the Director – Mining and Geology – Central Canada with the firm BBA E&C Inc., located at 1010 Lorne Street, Unit 101, Sudbury, ON, P3C 4R9, Canada.
2. I graduated from the University of Waterloo in 1992, with a Bachelor of Science (Honours) in Applied Earth Science.
3. I am a member in good standing of the Association of Professional Geoscientists of Ontario and License (PGO No. 0631).
4. My relevant experience includes 30 years in exploration, operations and consulting, including resource estimation on shear-hosted gold deposits. This also includes 8 years experience overseeing mining studies as department manager.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for Chapters 1 to 11, 14, 19, 20, and 22 to 27. I am also responsible for contributions to Chapters 12 and 21 of the Technical Report.
8. I visited the Enchi Property that is the subject of the Technical Report for three days from April 28 to May 1, 2014, and previously in 2010 and 2011.
9. I have prior involvement with the Property that is the subject of the Technical Report having issued technical reports in May 2010, July 2012, June 2015, January 2016 and October 2020.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

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I, Bahareh Asi, P. Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Senior Mining Engineer with the firm BBA E&C Inc., located at 10 Carlson Court, Suite 420, Toronto, ON, M9W 6L2, Canada.
2. I graduated in Mining from the Bahonar University of Kerman in 2001, with a Bachelor of Engineering and from Tarbiat Modares University in 2004 with a Master of Engineering.
3. I am a member in good standing of the Professional Engineers of Ontario (PEO No: 100203076).
4. I have been employed in consulting engineering and mining operations since my graduation and practiced my profession continuously. My relevant experience includes the mine engineering for the design, planning and estimation in technical studies and mine operations for numerous mining projects.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for Chapters 15 and 16. I am also responsible for contributions to Chapters 1, 2, 3, 21, 25, 26 and 27 of the Technical Report.
8. I have not visited the Enchi Property that is the subject of the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

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I, Mathieu Bélisle, P. Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Metallurgist with the firm BBA E&C Inc., located at 990 route de l'Église, Suite 590, Québec, QC, G1V 3V7, Canada.
2. I am a graduate of Laval University, with a Bachelor of Engineering in Metallurgy and Materials in 2002.
3. I am a member of the Ordre des Ingénieurs du Québec (OIQ 128549), Professional Engineers of Ontario (PEO 10210546), and Professional Engineers and Geoscientists of British-Colombia (EGBC 49319).
4. My relevant experience includes 20 years of experience working for mining operations and engineering consultants. I have been involved in numerous projects requiring detailed engineering design and produced several studies for the mining industry.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for Chapters 13 and 17. I am also responsible for contributions to Chapters 1, 2, 3, 21, 25, 26 and 27 of the Technical Report.
8. I have not visited the Enchi Property that is the subject of the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

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I, David Willock, P. Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Mining Engineer with the firm BBA E&C Inc., located at 1010 Lorne Street, Unit 101, Sudbury, ON, P3C 4R9, Canada.
2. I graduated from the Laurentian University in 2000, with a Bachelor of Engineering.
3. I am a member in good standing of the Professional Engineers of Ontario (PEO No: 100113931).
4. I have been employed in mining engineering, operations and projects for over 21 years. My relevant experience includes production planning, mine studies, estimation, operations supervision and project execution/construction.
5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
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8. I have not visited the Enchi Property that is the subject of the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

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I, Joe Amanor, MAusIMM(CP), as a co-author of the Technical Report, do hereby certify that:

1. I am a consultant geologist with SEMS Exploration Services Limited, located at No. 17 Orphan Cresent, North Labone, Accra, Ghana.
2. I am a graduate of the University of Ghana, Legon, Accra with a Bachelor of Science in June 1973, and an MSc, (Mineral Exploration and Mining Geology), from the Royal School of Mines, Imperial College of Science and Technology, London in 1979.
3. I am a member of the Australasian Institute of Mining and Metallurgy, Chartered Professional (CP) with License #204572.
4. My relevant experience includes over 40 years of experience in exploration, development, and operations, including more than 40 years on greenstone-hosted and structurally controlled gold deposits.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for contributions to Sections 12.1.1 to 12.1.3, 12.2.1 to 12.2.3, 12.3.1 to 12.3.2 and 12.4.1 to 12.4.3 of the Technical Report.
8. I visited the Enchi Property twice in 2017 for two days, each in June and September respectively, in November 2020 for four days, and then in June 2021 for five days.
9. I have prior involvement with the Property that is the subject of the Technical Report in my capacity as a consultant geologist of SEMS Exploration Services Limited participating in the exploration on the Enchi Gold Project from 2017-2020.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

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Joe Amanor, Consultant Geologist
SEMS Exploration Services Limited

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LIST OF ABBREVIATIONS

UNITS OF MEASURE

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in
cubic metre	m^3
cubic yard	yd^3
Coefficients of Variation	Cvs
day	d
days per week	d/wk
days per year (annum)	d/y
dead weight tonnes	DWT
decibel adjusted	Ba
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	$ or US$
dollar (Canadian)	CAN $
dry metric tonne	mt
foot	ft
gallon	gal
gallons per minute	gpm
Gigajoule	GJ
Gigapascal	GPA
Gigawatt	GW
Gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/y
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kiloton	kt
kilovolt	kV
kilovolt-ampere	kVa
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/y
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	Mpa
megavolt-ampere	Mva
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
microns	µm
milligram	mg
milligrams per litre	mg/L

millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
ounce	oz
pascal	Pa
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
short ton (2,000 lb)	st
short tons per day	st/d
short tons per year	st/y
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km2
square metre	m^2
three-dimensional	3D
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per annum (year)	t/a
tonnes seconds per hour metre cubed	ts/hm^3
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt
year	y

ACRONYMS

ADR	adsorption-desorption-recovery
Ai	Abrasion Index
AISC	All-in Sustaining Costs
ARD	Acid Rock Drainage
BOQ	Bill of Quantity
BS	Bibiani Shear
CEPA	Centre for Policy Analysis
CIC	Carbon-in-Column
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CP	Companion Policy
CRM	Certified Reference Material
CWi	Bond Crushability Work Index
DCF	Discounted Cash Flow
DIBK	di-isobutyl ketone
DTM	Digital Terrain Model
EAR	Environmental Assessment Regulations
ECG	Electricity Commission of Ghana
ECZ	East Contact Zone

EDM	Electronic Distance Measurement
EMP	Environmental Management Plan
EPA	Environmental Protection Agency
EPAA	Environmental Protection Agency Act
EPCM	Engineering, Procurement, Construction Management
ESIA	Environmental and Social Impact Assessment
F1	Form 1
HLF	Heap Leach Facility
ID^2	Inverse Distance Squared
IFC	International Finance Corporation
IRR	Internal Rate of Return
Kinross	Kinross Gold Corporation
LIDAR	Light Detection and Ranging
LOM	Life-of-Mine
Leo Shield	Leo Shield Exploration Ghana NL
LOI	Letter of Intent
MB	Volcanic
MCZ	Main Contact Zone
ML	Metal Leaching
MPSO	MinePlan Schedule Optimizer
Mutual	Mutual Ghana Ltd.
NI	National Instrument
NN	Nearest Neighbour
NPV	Net Present Value
NSR	Net Smelter Royalty
NSZ	Nyamebekyere Shear Zone
Nyam	Nyamebekyere
OK	Ordinary Krig
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PLS	Pregnant Leach Solution
the Project	Enchi Gold Project
the Property	Enchi Gold Property
PVC	Polyvinyl-chloride
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
QV	Quartz Vein
RAB	Rotary Air Blast
RBMGL	Red Back Mining Group Limited
RC	Reverse Circulation
Red Back	Red Back Mining Inc.
RF	Revenue Factor
ROM	Run-of-Mine

RTSZ .... Ridge Top Shear Zone
SETO .... Sewum-Tokosea Mine Trend
SHS .... Hilltop Shear
SPG .... Graphitic Phyllite
SPH .... Turbidite
SRM .... Standard Reference Material
SRTSZ .... Sewum Ridgetop Shear Zone
SRZ .... Road Zone
SVC .... Volcaniclastic
SWZ .... Sewum West Zone
UTM .... Universal Transverse Mercator
WCZ .... West Contact Zone
WGS .... World Geodetic System
WRC .... Water Resources Commission
WSP .... WSP Canada Inc.
WSS .... West Sewum Shear

1. SUMMARY

1.1 Introduction

BBA E&C Inc. (BBA) and SEMS Exploration Services Limited (SEMS) were retained by Newcore Gold Ltd. (Newcore) to prepare a Preliminary Economic Assessment (PEA) for the Enchi Gold Project (the Project or the Property), located in southwestern Ghana. This report has been prepared to comply with disclosure and reporting requirements set forth in National Instrument 43-101 (NI 43-101), Form 43-101F1 of NI 43 101 (NI 43-101F1) and Standards of Disclosure for Mineral Projects, Companion Policy 43 101CP (NI43-101CP) to NI 43-101. The purpose of this PEA is to:

- Update the mineral resource statement based on drilling completed since the 2020 mineral resource statement; and
- Generate a PEA based on the 2021 Mineral Resource Estimate.

This PEA is based on using mining contract services under the supervision of Newcore, whereby the open pit mining operations are undertaken by a contractor while the processing and other site operations are undertaken by the Project owner.

The Mineral Resource Estimate, mine plan, and 6.6 million tonnes per annum (Mt/a) heap leach processing scenario are described in this report.

The Project comprises seven prospecting licenses, totalling 216. km^2 located in the Enchi and Aowin Suaman Districts, in the south west of Ghana (www.ghanamissionun.org). The Project is accessed from Accra on sealed roads via the regional port city of Sekondi (Takoradi) or the mining centre of Kumasi. From either of these centres, access to Enchi (population of 11,737), the capital of Aowin-Suaman district, is available by paved roads (Elubo-Enchi Road or the Asankragua-Enchi Road). Access through the remainder of the Project area is by dirt and gravel roads.

The Project is based in the Wet Semi Equatorial Climatic Zone. The climate is typically warm and humid with a mean-monthly temperature of 27°C. There are two rainy seasons: the major rainy season from May to July, and a shorter rainy season from September to October. The district receives an annual rainfall of between 1,500 and 1,800 mm. During the dry season, predominately December to March, Harmattan winds (dry hot continental fronts from the Sahara) blow over the country resulting in drier warm days and cool nights.

Modern exploration in the form of soil sampling, surface trenching, rotary air blast drilling, reverse circulation drilling, and diamond drilling has been completed by various operators, including Leo Shield Exploration Ghana NL (Leo Shield) from 1995 to 1998, Redback Mining Inc. (Red Back (now Kinross)) from 2003 to 2010, Edgewater Exploration Ltd. (Edgewater) from 2011 to 2013, and Newcore Gold Ltd. (Newcore) from 2014 to present.

1.2 Geology

The Project is located in southwestern Ghana, in a region well known for prolific gold production, and hosts numerous historical and current operating mines located along strike to the northeast of the Project. In 2020, Ghana was the largest gold producer in Africa at 4.02 Moz (www.miningweekly.com). The Project covers a 40 km strike length of the Bibiani Shear Zone along the eastern margin of the Sefwi Belt stretching from the Côte d'Ivoire border in the southwest to neighbouring claims to the northeast. The Bibiani shear is known to host significantly large lode-gold deposits such as Bibiani and Chirano.

The Project is situated on the contact between the Sefwi Belt to the west and the Kumasi Basin to the east. The Sefwi Belt is dominated by mafic volcanics, metasediments, and intrusive granitoids. The Kumasi Basin contains wide basins of marine clastic sediments. All the rocks of the region have been extensively metamorphosed to greenschist facies.

Extensive faulting, on local and regional scales, occurs along the margins of the volcanic-sedimentary belts. These northeast-trending structures are fundamentally important in the development of the gold deposits for the region. The major shear system within the Project area is located at the boundary of the Sefwi Belt and the Kumasi Basin, and is called the Bibiani Shear Zone. Gold deposits are typically located on second or third order structured or splay off the Bibiani Shear.

The Project contains mineralized zones that are characteristic of mesothermal quartz vein style gold deposits. This type of mineralization is the most important type of gold occurring within West Africa and is commonly referred to as the Ashanti-type.

Mineralization can occur as both sulphide and non-sulphide styles. Sulphide mineralization is characterized by early stage disseminated sulphides of primarily pyrite and/or arsenopyrite, hosting significant gold content, which is overprinted by late-stage quartz veining with minor amounts of visible gold and accessory polymetallic sulphides. Non-sulphide mineralization is characterized by gold not hosted within sulphide minerals, in either the early or later stage-mineralizing event. Extensive oxidation has occurred throughout the Property.

1.3 Drilling

A total of 1,171 drillholes and trenches has been completed on the Project for a total of 110,982 m. This includes diamond drilling, reverse circulation (RC), rotary air blast, and surface trenching. Of the entire dataset, 1,153 holes or 98% has been completed within the four mineral resource areas.

In 2012, 4,058 m in 25 RC holes were completed in the Nyam and Sewum zones. In 2017, 3,406 m in 28 RC holes were completed in the Boin and Sewum zones. An ongoing 2020 -2021 RC program has completed 246 holes totalling 38,641 m as of the effective date of this report. The holes completed in 2021 after specified dates have not been used in the mineral resource estimation.

Industry standard, drilling, logging, and sampling practices were implemented during the various phases.

1.4 Sample Preparation, Analysis and Security

All RC chip samples, diamond drill core samples and trench chip samples were prepared and analyzed at an accredited laboratory.

QA/QC programs in place during the 2012, 2017, and 2020-2021 drilling programs meet industry standards practice.

1.5 Data Verification

Validation of the database has been conducted and any issues identified have been corrected in the database.

Check assays have been completed on selected samples from the 2012, 2017 and 2020-2021 drilling programs. A good correlation exists between the original samples and the check assay.

1.6 Mineral Resource Statement

The Mineral Resource Estimate was completed on the Sewum, Boin and Nyam zones using the ordinary kriging (OK) methodology on a capped and composited borehole dataset consistent with industry standards. The Mineral Resource Estimate was completed on the Kwakyekrom zone using inverse distance squared (ID^2). Validation of the results was conducted through the use of visual inspection, swath plots, and global statistical comparison of the model against drillhole composites, ID^2 and nearest neighbour (NN) models.

Table 1-1 summarizes the results of the Inferred Mineral Resource Statement (pit constrained).

Table 1-1: Enchi Inferred Mineral Resource Statement

Deposit	Tonnes	Gold (g/t)	Gold (ounce)
Sewum	41,009,000	0.55	725,200
Boin	21,807,000	0.72	504,800
Nyam	4,892,000	0.82	129,000
Kwakyekrom	2,703,000	0.64	55,600
Total	**70,411,000**	**0.62**	**1,414,600**

Notes for Inferred Mineral Resource Statement:

- CIM definition standards were followed for the Mineral Resource Estimate.
- The 2021 mineral modes used ordinary kriging (OK) grade estimation within a 3D block model with mineralized zones defined by wireframes solids and constrained by it shells for Sewum, Boin and Nyam. Kwakyekrom used inverse distance squared (ID^2).
- A $1,650/ounce gold price, open pit with heap leach operation was used to determine the cut-off grade of 0.2 g/t Au. Mining cost of $1.40 for oxide, $2.10 for transition, and $2.60 for fresh rock per mined tonne, and G&A and milling cost of $6.83/milled tonne.

- Metallurgical recoveries have been applied to four individual deposits and in each case three material types (oxide, transition and fresh rock) with average recoveries of 77% for Sewum, 79% for Boin, 60% for Nyam and 72% for Kwakyekrom.
- A density of 2.20 g/cm^3 for oxide, 2.45 g/cm^3 for transition and 2.70 g/cm^3 for fresh rock was applied.
- Numbers may not add due to rounding.
- Optimized pit slopes angles varied based on rock types.
- Mineral resources that are not mineral reserves do not have economic viability.

1.7 Mining Methods

The Enchi deposits will be mined using conventional open pit mining methods (drill, blast, load and haul) with the mining operations being outsourced to a mining contractor. Newcore will provide supporting technical services and mine management.

A pit optimization analysis was completed for each of the four deposits to determine the cut-off grades and to what extent each deposit can be mined profitably. The selected pit shells were then used to guide the pit designs that include smoothing the pit walls, adding ramps to access the pit bottom and ensuring that the pits can be mined safely and efficiently. A total of 10 individual open pits were designed in total. Table 1-2 presents the subset of mineral resources within the open pit designs for the PEA, which include 68.6 Mt of mineral resources at an average gold grade of 0.57 g/t, which can be mined at a strip ratio of 2.1:1.

Table 1-2: Subset of Mineral Resources Within the PEA Pit Designs (Above Cut-off) (1)(2)

Item	Tonnage (k tonnes)	Gold Grade (g/t)	Contained Ounces (k ounces)	Strip Ratio
Sewum	39,750	0.50	645	1.3:1
Boin	22,646	0.65	472	3.3:1
Nyam	4,520	0.73	107	3.2:1
Kwakyekrom	1,650	0.52	28	2.0:1
Total	**68,566**	**0.57**	**1,252**	**2.1:1**

(1) Including mining dilution and mining recovery.
(2) Numbers may not add due to rounding.

A mine production plan was prepared for the 10 open pits using Hexagon's MinePlan Schedule Optimizer (MPSO) tool with the objective of maximizing the Net Present Value (NPV). The mine plan resulted in an approximate 11-year mine life with approximately 6.6 Mt of mineralized material being sent to the heap leach facility annually. The maximum annual mining capacity reaches 22 Mt/a between Years 5 and 8. Since the mineralization is close to surface, very little pre-production waste stripping is required. The mine plan also considers the stockpiling of lower grade material that is rehandled during the mine life.

Operations will consist of a fleet of 6.6 m^3 hydraulic excavators and 64-tonne haul trucks, the QP estimated that a total of four excavators, one loader and 26 haul trucks will be required during peak production. A total of six production drills will also be required during peak production in addition to a fleet of support and service vehicles. The total mine workforce including contractor and owner employees is expected to reach 173 during peak production. Contractor and owner personnel consists of 144 and 29 employees respectively.

It was assumed that the workforce will largely consist of local personnel except for 5 supervisory roles that would likely be filled by expatriate personnel.

1.8 Metallurgy and Recovery

Due to the lack of test work specific for heap leach design and amenability, only a conceptual design has been studied for the purpose of the PEA. The process facility for the Project has been designed to process oxide and saprolite mineralization from the Sewum, Boin, Nyam, and Kwakyekrom zones at an average annual feed rate of 6.6 million tonnes. The process route selected is a conventional heap leaching facility. The process facilities including primary & secondary crushing, agglomeration, stacking, heap leaching and recovery plant will operate year-round. An overall average gold recovery of 79% has been estimated for the Project.

1.9 Project Infrastructure

The Project site is located in a rural area between the villages of Sewum and Achimfo with the heap leach facility and central facilities approximately 20 km by road south of the district capital of Enchi. The Project area has limited to moderate infrastructure, with a paved road crossing the central portion of the Project leading to the town of Enchi. Little infrastructure exists in the area so the Project infrastructure should be considered new construction apart from portions of existing roads that will be upgraded for use as haulage roads.

The town of Enchi is located 77 km north of the substation at Elubo, serviced by a 225 kV line, and 122 km southwest of the substation at Asawinso, serviced by a 161 kV line. The Chirano Gold Mine, owned by Kinross and located 50 km northeast of the Project, is supplied by power from a 33 kV overhead power line from an existing transformer that is also supplying the Bibiani Gold Mine plant. In addition, six diesel generators are located at the Chirano facility to provide stand-by power in case of supply issues from the Electricity Commission of Ghana (ECG).

The anticipated infrastructure for the Project includes offices and administrative facilities, mine dry facilities, equipment maintenance workshop, refuelling facilities, explosive magazine, assay laboratory, and warehouse facilities. Items required to support all site facilities and activities will include haulage and access roads, stockpile areas, surface water diversion, water supply, power supply network, diesel generators, sewage treatment plant, and waste management facilities.

No on-site accommodations have been accounted for in this study due to the proximity to the town of Enchi. Senior staff and expatriate employees will be accommodated in rental units in the town of Enchi.

1.10 Environmental Studies, Permitting, and Social Impacts

The most significant environmental and social issues are expected to be related to water management, social-economic impacts, and post mine-closure expectations. These issues are likely to be of key concern to local communities and/or have cost implications in respect of impact management during the operation and closure phases.

The Project will trigger a range of regulatory requirements and processes, which will require the application for, receipt of, and compliance with a variety of environmental permits and approvals from the relevant Ghanaian authorities.

Closure objectives should be defined early in the mine planning process and integrated into all activities throughout the Life-of-Mine (LOM).

1.11 Capital and Operating Costs

1.11.1 Capital Costs

An initial capital expenditure of $97 million (including 30% contingency on direct costs) has been estimated to construct the Project, with a further $23 million in sustaining capital during operations, $23 million for closure (including reclamation) and $14 million of salvage value. The capital cost estimate is based on an open pit mining and heap leach operation processing 6.6 Mt/a utilizing contract mining. Capital costs are detailed in Table 1-3.

Table 1-3: Capital Cost Estimate Details (4)

Description	Initial ($K)	Sustaining ($K)	Closure ($K)	LOM ($K)
Direct Costs				
Mining	$2,576	$270	$796	$3,642
Processing	$55,264	$13,405	-	$68,669
Environmental (1)	-	-	$15,053	$15,053
Infrastructure	$5,726	$1,719	-	$7,445
Salvage Value (2)	-	-	-	-$14,106
Total Direct Costs	**$63,566**	**$15,394**	**$15,849**	**$80,703**

Description	Initial ($K)	Sustaining ($K)	Closure ($K)	LOM ($K)
Indirect Costs				
Engineering and Procurement	$7,371	$1,539	$1,545	$10,456
Construction Indirect	$4,879	$1,210	$1,204	$7,293
Owner's Cost	$1,748	-	-	$1,748
Total Indirect Costs	**$13,998**	**$2,749**	**$2,750**	**$19,497**
Capital Costs Pre-Contingency	**$77,564**	**$18,144**	**$18,599**	**$100,200**
Contingency: 30% of Direct Costs [3]	$19,070	$4,618	$4,755	$28,443
Total Capital Costs	**$96,634**	**$22,762**	**$23,353**	**$128,643**

(1) Environmental includes closure and remediation works in Years 11 and 12, as well as post closure maintenance for three years.

(2) Salvage value recovered in Year 12, assumes 20% of processing costs and 5% of owner's infrastructure costs recouped.

(3) Contingency not applied to salvage value.

(4) Numbers may not add due to rounding.

This capital cost estimate is based on industry standard estimates. Capital cost estimates were developed using budgetary quotes provided by contractors experienced in Ghana and reviewing other heap leach (HL) projects in West Africa.

Construction is estimated to be 15 months. The Project benefits from relatively flat terrain (rolling hills) and simple infrastructure, limiting the amount of earthworks required. The initial capital costs reflect an estimate for the design and development of the plant and mine infrastructure that includes crushing, agglomeration, heap leaching, processing ponds and a gold recovery plant. The heap leach pads will be built in three phases, with a third of the cost upfront and the remainder included in sustaining capital in Years 3 and 6.

Reclamation and closure costs have been estimated based on the preliminary infrastructure plans and are inclusive of an allowance for rehabilitation monitoring, and care and maintenance for three years post completion of mining.

1.11.2 Operating Costs

The Project is modelled as a near surface, open pit, heap leach mine with heap leach feed material trucked from four deposits (Sewum, Boin, Nyam, Kwakyekrom) to a central crushing and heap leach facility that will be located near Sewum.

Operating costs for the Life-of-Mine are estimated at $908 million. Cash costs over that time are estimated at $1,025 million and include operating costs, royalties and refining charges. A 5% royalty on revenues is due to the Government of Ghana, and a 2% NSR royalty is due to Maverix Metals Inc. Camp costs for the Project are lower relative to other projects due to the Project's proximity to the town of Enchi where most administrative facilities can be located. Operating costs are summarized in Table 1-4.

Table 1-4: Operating Cost Estimate Details (3)

Description	Operating Cost		
	LOM ($K)	$/tonne milled	$/oz Au
Mining	$422,363	$6.16	$430
Processing	$361,817	$5.28	$368
Environment & Infrastructure	$5,241	$0.08	$5
On-Site G&A	$118,284	$1.73	$120
Total Operating Costs	$907,705	$13.24	$923
Treatment & Refining Charges	$3,933	$0.06	$4
Royalties	$113,492	$1.66	$115
Total Cash Costs	**$1,025,130**	**$14.95**	**$1,043**
Sustaining Capital (1)	$22,762	$0.33	$23
All-in Sustaining Costs (AISC) (2)	**$1,047,891**	**$15.28**	**$1,066**

(1) Sustaining capital excludes closure costs and salvage value.
(2) AISC consists of cash costs plus sustaining capital (excluding closure costs and salvage value).
(3) Numbers may not add due to rounding.

1.12 Economic Analysis

The financial results of the Project are summarized in Table 1-5, Table 1-6 and Table 1-7. On a pre-tax basis, the Project has a NPV of $332.7 M at a discount rate of 5%, an Internal Rate of Return (IRR) of 54%, and a payback period of 2.1 years. On a post-tax basis, the NPV is $212.5 M at a discount rate of 5%, the IRR is 42%, and the payback period is 2.3 years.

Table 1-5: Summary of Financial Analysis

Description	Unit	LOM	Y2 to Y5
Tonnage Mineralized Material Feed	k tonnes	68,566	26,303
Feed Grade Processed	g/t Au	0.57	0.62
Gold Recovery (average)	%	79	80
Production Period	year	10.6	4.0
Tonnage Waste Rock	k tonnes	143,490	54,216
Stripping Ratio	-	2.09	2.06
Gold Production	k ounces	983,296	416,685
Annual Gold Production (LOM)	oz/y	92,530	104,171
Gold Production (Gross Revenues)	K $	1,622	687,531
Net Revenues (1)	K $	1,505	637,770
Total Operating Cost (2)	K $	907,705	341,994
Total Cash Costs (Operating + Refining Charges and Royalties)	K $	1,025,130	391,754
Total Capital Costs with contingencies	K $	128,643	11,850
Initial Capital Costs	K $	96,634	-
Sustaining Capital Costs	K $	32,009	11,850
All-in Cost (Cash Costs + Capital Costs)	k tonnes	1,153,773	403,604

(1) Including refining charges and royalties.

(2) Including fees for mineral tenure.

Table 1-6: Pre-tax Financial Results

Description	Unit	LOM
Total Cash Flow	K $	468,665
NPV @ 5%	K $	332,710
Pre-Tax IRR	%	54
Pay-back Period (from start of construction)	year	2.1

Table 1-7: Post tax Financial Results

Description	Unit	LOM
Total Cash Flow	K $	304,326
NPV @ 5%	K $	212,466
Pre-Tax IRR	%	42
Pay-back Period (from start of construction)	year	2.3

1.13 Conclusions

The present study indicates that the Project has positive economics, within the parameters of a PEA. The key financial indicators, based on future gold prices and capital and operating cost estimates, justify advancing the Project and undertaking the work outlined in Chapter 26 (Recommendations). It is the QP's opinion that additional exploration and engineering test work expenditures are warranted to improve the understanding of the Project and delineate additional resources.

Table 1-8 summarizes key project results for the study. Table 1-9 presents the summary of the Project economics.

Table 1-8: Key Project Parameters

Description	Unit	Value
Key Assumptions		
Base Case Gold Price	$/oz	1,650
Production Profile		
Total Tonnes Processed	k tonnes	68,566
Total Tonnes Waste	k tonnes	143,490
Strip Ratio		2.1
Feed Grade Processed	g/t Au	0.57
Mine Life	year	11
Throughput	Mt/a	6.6
Gold Recovery	%	79
LOM Gold Production	k ounces	983,296
LOM Average Annual Gold Production	k ounces	89,391
Peak Gold Production in Year 10	k ounces	121,387
Average Annual Gold Production Years 2 to 5	k ounces	104,171
Unit Operating Costs		
LOM Average Operating Cost (1)	$/oz gold	923
LOM Average Cash Cost (2)	$/oz gold	1,043
LOM AISC (Cash Cost plus Sustaining Cost) (3)	$/oz gold	1,066
Capital Costs		
Initial Capital Cost	M $	97
Sustaining Capital Cost (4)	M $	23
Reclamation Cost	M $	22

(1) Operating costs consist of mining costs, processing costs, and on-site G&A.
(2) Cash costs consist of operating costs plus treatment and refining charges, and royalties.
(3) AISC consists of cash costs plus sustaining capital (excluding closure costs and salvage value).
(4) Sustaining Capital Cost excludes closure costs and salvage value. Includes $6.7 M in each of Years 3 and 6 for heap leach pad expansion.

Table 1-9: Project Economics Summary

	$1,650/oz Gold Price		$1,850/oz Gold Price	
	Pre-Tax	After-Tax	Pre-Tax	After-Tax
$NPV_{5\%}$	**$333 million**	**$212 million**	$471 million	$302 million
IRR	**54%**	**42%**	69%	54%
Payback	**2.1 years**	**2.3 years**	1.7 years	1.9 years
LOM Cash Flow	**$469 million**	**$304 million**	$652 million	$423 million

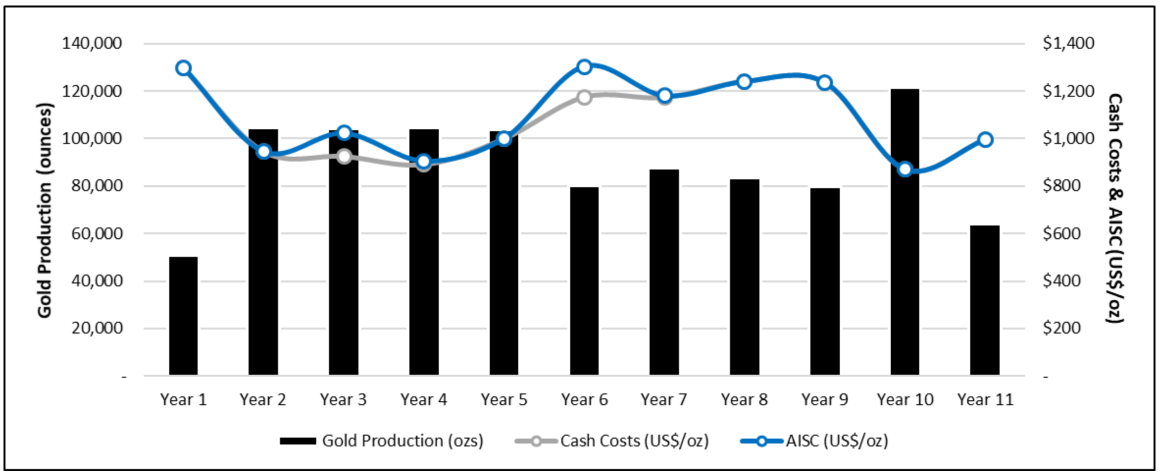


Figure 1-1: Production and Cost Profile by Year

The Mineral Resources used in the LOM plan and economic analysis include Inferred Resources. Inferred Mineral Resources are considered speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the Inferred Resources will be upgraded to a higher resource category, or that the results of this preliminary assessment will be realized.

When ranked, the sensitivity analysis indicates that the Project is most sensitive to gold price and gold recovery. From a cost perspective, the Project is more sensitive to operating expenditure than capital costs.

Some key areas of risk or uncertainty that need to be addressed in subsequent study phases include:

- Additional geological studies are required to upgrade the Inferred Resources to Indicated or Measured Resources.
- Additional metallurgical test work is required to determine heap leach amenability and metallurgical behaviour of the oxide, transition, and sulphide zones.
- Conduct a geotechnical drill program, ideally in conjunction with geological drill program, to confirm and assess pit slope angle assumptions and assess rock mass strength.

1.14 Recommendations

The results of the PEA support continued exploration on the Project. Two separate exploration programs are proposed. Phase 2 is independent of the results of Phase 1 and can be completed before or after the completion of Phase 1.

Phase 1 is designed to further expand the mineral resources of the known zones with RC and diamond drilling. A budget of $2.5 million is recommended to complete the program.

Phase 2 is designed for continued exploration on the Project with RC and diamond drilling as well as collect of data for future engineering studies. A budget of $4,800,000 is recommended.

Additional recommendation for the Project includes:

- Collection of specific gravity data; and
- Level the drillhole collar elevations to a LiDAR survey.

2. INTRODUCTION

This report was prepared and compiled by the QPs under employment contract with BBA E&C Inc. (BBA) and SEMS Exploration Services Limited (SEMS) at the request of Newcore Gold Ltd. ("Newcore" or the "Company"). The purpose of this report is to provide a technical report of the Enchi Gold Project (the "Project") in accordance with the guidelines of the Canadian Securities Administrators National Instrument 43-101 (NI 43-101) and Form 43-101 F1.

2.1 Purpose of Report

The purpose of this report is to publish a technical report on the Enchi Property summarizing the geology, past exploration activities, Mineral Resource Estimate and preliminary economic assessment on the Property.

2.2 Terms of Reference

Newcore engaged the services of the authors on January 22, 2021 to write an independent NI 43-101 Technical Report on the Enchi Property in Ghana.

This report was prepared in accordance with NI 43-101 and Form NI 43-101F1.

2.3 Newcore Gold Ltd.

Newcore's corporate offices are located at 595 Burrard St. Suite 413, Vancouver, British Columbia, V7X 1J1, Canada. Newcore is a public company listed on the TSX Venture Exchange (TSX-V: NCAU) and also trades on the OTCQX in the United States (OTCQX:NCAUF).

Newcore is a Vancouver based junior exploration company focused on the advancement and development of its Enchi Gold Project in southwest Ghana.

2.4 Qualification of Consultants

The consultants preparing this technical report are specialists in the fields of geology, mining, mineral processing and mine infrastructure.

The consultants or any associates employed in the preparation of this report have no beneficial interest in Newcore. The consultants are not insiders, associates, or affiliates of Newcore. The results of this technical report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Newcore and the consultants. The consultants were paid a fee for the services in accordance with normal professional consulting practice.

2.5 Report Responsibility and Qualified Persons

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QPs) as defined in the NI 43-101 and are members in good standing of appropriate professional institutions.

- Todd McCracken, P. Geo., BBA E&C Inc.
- Bahareh Asi, P. Eng., BBA E&C Inc.
- Mathieu Bélisle, P. Eng., BBA E&C Inc.
- David Willock, P. Eng., BBA E&C Inc.
- Joe Amanor, MAusIMM(CP), SEMS Exploration Services Limited

The preceding QPs have contributed to the writing of this Report and have provided QP certificates, included at the beginning of this Report. The information contained in the certificates outlines the sections in this Report for which each QP is responsible. Each QP has also contributed figures, tables and portions of Chapters 1 (Summary), 2, (Introduction), 3 (Reliance on other Experts), 21 (Capital and Operating Costs), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References). Table 2-1 outlines the responsibilities for the various sections of the Report and the name of the corresponding Qualified Person.

Table 2-1: Qualified Persons and Areas of Report Responsibility

Chapter	Description	Qualified Person	Company	Comments and exceptions
1.	Executive Summary	T. McCracken	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
2.	Introduction	T. McCracken	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
3.	Reliance on other Experts	T. McCracken	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
4.	Project Property Description and Location	T. McCracken	BBA	All Chapter 4
5.	Accessibility, Climate, Local Resource, Infrastructure and Physiography	T. McCracken	BBA	All Chapter 5
6.	History	T. McCracken	BBA	All Chapter 6
7.	Geological Setting and Mineralization	T. McCracken	BBA	All Chapter 7
8.	Deposit Types	T. McCracken	BBA	All Chapter 8
9.	Exploration	T. McCracken	BBA	All Chapter 9
10.	Drilling	T. McCracken	BBA	Sections 10.1 to 10.3

Chapter	Description	Qualified Person	Company	Comments and exceptions
11.	Sample Preparation, Analyses and Security	T. McCracken	BBA	All Chapter 11
12.	Data Verification	T. McCracken	BBA	Sections 12.1.1 to 12.1.3, 12.2.1 to 12.2.3, 12.3.1 to 12.3.2, 12.4.1 to 12.4.3 and 12.5
		J. Amanor	SEMS	Sections 12.1.4, 12.1.5, 12.2.4, 12.2.5, 12.3.3, 12.3.4, 12.4.4, 12.4.5 and 12.5
13.	Mineral Processing and Metallurgical Testing	M. Bélisle	BBA	All Chapter 13
14.	Mineral Resource Estimate	T. McCracken	BBA	All Chapter 14
15.	Mineral Reserve Estimate	B. Asi	BBA	All Chapter 15
16.	Mining Methods	B. Asi	BBA	All Chapter 16
17.	Recovery Methods	M. Bélisle	BBA	All Chapter 17
18.	Project Infrastructure	D. Willock	BBA	All Chapter 18
19.	Market Studies and Contracts	T. McCracken	BBA	All Chapter 19
20.	Environmental Studies, Permitting, and Social or Community Impact	T. McCracken	BBA	All Chapter 20
21.	Capital and Operating Costs	B. Asi	BBA	Sections 21.4.2.1, 21.5.2
		M. Bélisle	BBA	Sections 21.4.2.2 and 21.5.3
		D. Willock	BBA	Sections 21.4.2.3
		T. McCracken	BBA	Sections 21.1 to 21.3, 21.4.1, 21.4.2.4, 21.5.1 and 21.5.4 to 21.5.6
22.	Economic Analysis	T. McCracken	BBA	All Chapter 22
23.	Adjacent Properties	T. McCracken	BBA	All Chapter 23
24.	Other Relevant Data and Information	T. McCracken	BBA	All Chapter 24
25.	Interpretation and Conclusions	T. McCracken	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
26.	Recommendations	T. McCracken	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
27.	References	T. McCracken	BBA	All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.

2.6 Site Visit

Mr. Todd McCracken, P. Geo., is a qualified person (QP) and co-author of this report. Mr. McCracken is a professional geologist with 30 years of experience in exploration and operations, including several years working in shear hosted lode gold deposits and 20 years completing mineral resource estimations and block models. Mr. McCracken visited the Property for three days from April 28 to May 1, 2014. This was Mr. McCracken's third visit to the Property, having visited previously in 2011 and 2010. During the most recent trip, Mr. McCracken was accompanied by Mr. Vincent Dzorkpetey, a geologist with Edgewater.

Mr. Joe Amanor, MAusIMM(CP), is a qualified person (QP) and co-author of this report. Mr. Amanor, is a professional geologist with over 40 years of experience in exploration and operations, including several years working in shear hosted lode gold deposits. Mr. Amanor visited the Property for two days from June 6 to 7, 2017, and two days from September 5 to 6, 2017. Mr. Amanor visited the Property from November 5 to 8, 2020 and most recently from June 2 to 6 of 2021 (inclusive). For the 2021 visit, Mr. Amanor was accompanied by Daniel Adusei, the SEMS Exploration Services Project Geologist on site and Dan Wilson, Newcore's Country Manager.

Ms. Bahareh Asi, P. Eng., Mr. Mathieu Bélisle, P. Eng., and Mr. David Willock, P. Eng., all QPs in this report, did not visit the Property that is the subject of this technical report.

2.7 Effective Date and Declaration

The issue date of this report is July 13, 2021. The effective date of the technical report is June 8, 2021.

As of the date of this report, the authors are not aware of any material fact or material change with respect to the subject matter of this technical report that is not presented herein, or which the omission to disclose could make this report misleading.

2.8 Currency, Units of Measure, and Calculations

Unless otherwise specified or noted, the units used in this report are metric. Every effort has been made to clearly display the appropriate units being used throughout the report.

- Coordinates within this report use WGS 84 UTM Zone 30N, unless otherwise stated;
- Currency is in United States dollars (US$ or $), unless otherwise noted;
- All ounce units are reported in troy ounces, unless otherwise stated: 1 oz (troy) = 31.1035 g.

This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs consider them immaterial.

2.9 Acknowledgement

BBA and the QP to the report would like to acknowledge the following individuals for their general support provided during this assignment:

The Project benefited from the specific input of Greg Smith, P. Geo., Luke Alexander, Danny Lee, Mal Karwowska and Michael Vint of Newcore, Henry Schnell, Charlotte Athurion, P. Geo., Clovis Auger, P. Geo., Jeff Cassoff, P. Eng., Jim Barrett, and Amanda Fitch, P. Eng. from BBA, and Branden Fraser, P. Eng., independent consultant.

3. RELIANCE ON OTHER EXPERTS

The Qualified Persons have reviewed and analyzed data and reports provided by Newcore, together with publicly available data, drawing their own conclusions augmented by direct field examination.

The QPs who prepared this report relied on information provided by experts who are not QPs. The QPs believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

- Todd McCracken, P. Geo., relied upon Greg Smith, Newcore Gold's VP Exploration for matters pertaining to mineral claims and mining leases as disclosed in Chapter 4 (personal communications);
- Todd McCracken, P. Geo., relied upon Luke Alexander, Newcore Gold's President and CEO for matters pertaining to royalties as disclosed in Chapter 4 (personal communications); and
- Todd McCracken, P. Geo., relied upon Danny Lee, Newcore Gold's CFO for matters pertaining to taxes as disclosed in Chapter 22 (personal communications).

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Project comprises seven prospecting licenses, totaling 216 km^2 located in the Enchi and Aowin Suaman Districts, in the southwestern region of Ghana.

The Project covers a 40 km strike length of the eastern margin of the Sefwi Belt stretching from the Côte d'Ivoire border in the southwest to neighbouring claims to the northeast. The Project is located 290 km west of the capital of Accra and 50 km southwest of the Chirano Mine operated by Kinross (Figure 4-1). The Project is centered on 5°47’ North latitude and 2°42’ West longitude.

4.2 Mineral Disposition

The seven licenses that make up the Project are summarized in Table 4-1 and are also displayed in Figure 4-2. Lease boundaries are defined by a list of latitude and longitude coordinates of the corners (pillar points) submitted to the Minerals Commission (Mincom). The boundaries are not physically marked on the ground and have not been surveyed by Newcore.

Nyame Esa and Nkwanta are license applications and are required to proceed through the full application process. These licenses were submitted in 2019. The application process for a prospecting license, which is required for drilling and excavation work, is as follows:

- Application submitted to Mincom;
- Mincom completes paperwork and checks maps;
- Mincom generates a letter that is sent to the local authorities and is posted for three weeks; this provides an opportunity for objections to the license application;
- Local authorities write back to Mincom if no objections are presented;
- Application proceeds to a technical committee for review;
- Upon technical committee approval, the license is prepared and sent to the Mincom Minister for signature.

The entire process typically takes two years or more to complete. Once an application is submitted, work under the license is allowed to proceed.

Sewum, Enkye, Nyam, Abotia and Yehikwakrom are subject to license renewal. The renewal process is similar to the application process indicated above yet does not require approval of the district and community. The applications for renewal were submitted in November 2019 and approved May 31, 2020; the licences are now in good standing until May 31, 2023. The time frame for extending the licenses is variable depending on how busy Mincom is, and can take as little as six months to as long as two years.

During the renewal process, the licenses are not subject to a reduction in land size.

Figure 4-1: Location Map (modified from McCracken and Smith, 2020)

Table 4-1: List of Project Licenses

Name	Type	Number	New Area (km^2)	Current Holding Company	Status
Sewum	PL	PL 2/424	32.55	Cape Coast Resources Ltd.	50% shed off completed. All maps and application for 3-year extension submitted November 3, 2019. License extended to May 31, 2023.
Enkye	PL	PL 2/404	34.65	Cape Coast Resources Ltd.	50% shed off completed. All maps and application for 3-year extension submitted November 3, 2019. License extended to May 31,2023.
Nyamebekyere	PL	PL 2/406	35.91	Cape Coast Resources Ltd.	50% shed off completed. All maps and application for 3-year extension submitted November 3, 2019. License extended to May 31, 2023.
Abotia	PL	PL 2/119	26.04	Cape Coast Resources Ltd.	License expired February 16, 2016. Application for 3-year extension submitted November 3, 2019. Error in processing causes delay. Maps resubmitted for further processing.
Yehikwakrom	PL	PL 2/405	29.82	Cape Coast Resources Ltd.	50% shed off completed. All maps and application for 3-year extension submitted November 3, 2019. License extended to May 31, 2023.
Nyame Esa	PL	not assigned	24.36	Boin Resources Limited	Re-application for the 50% shed off from Nyamebekyere PL by BRL. Resubmission of maps after corrections. Documents Gazetted in the newspapers.
Nkwanta	PL	not assigned	32.76	Boin Resources Limited	Re-application for the 50% shed off from Sewum PL by BRL. Documents Gazetted in the newspapers.

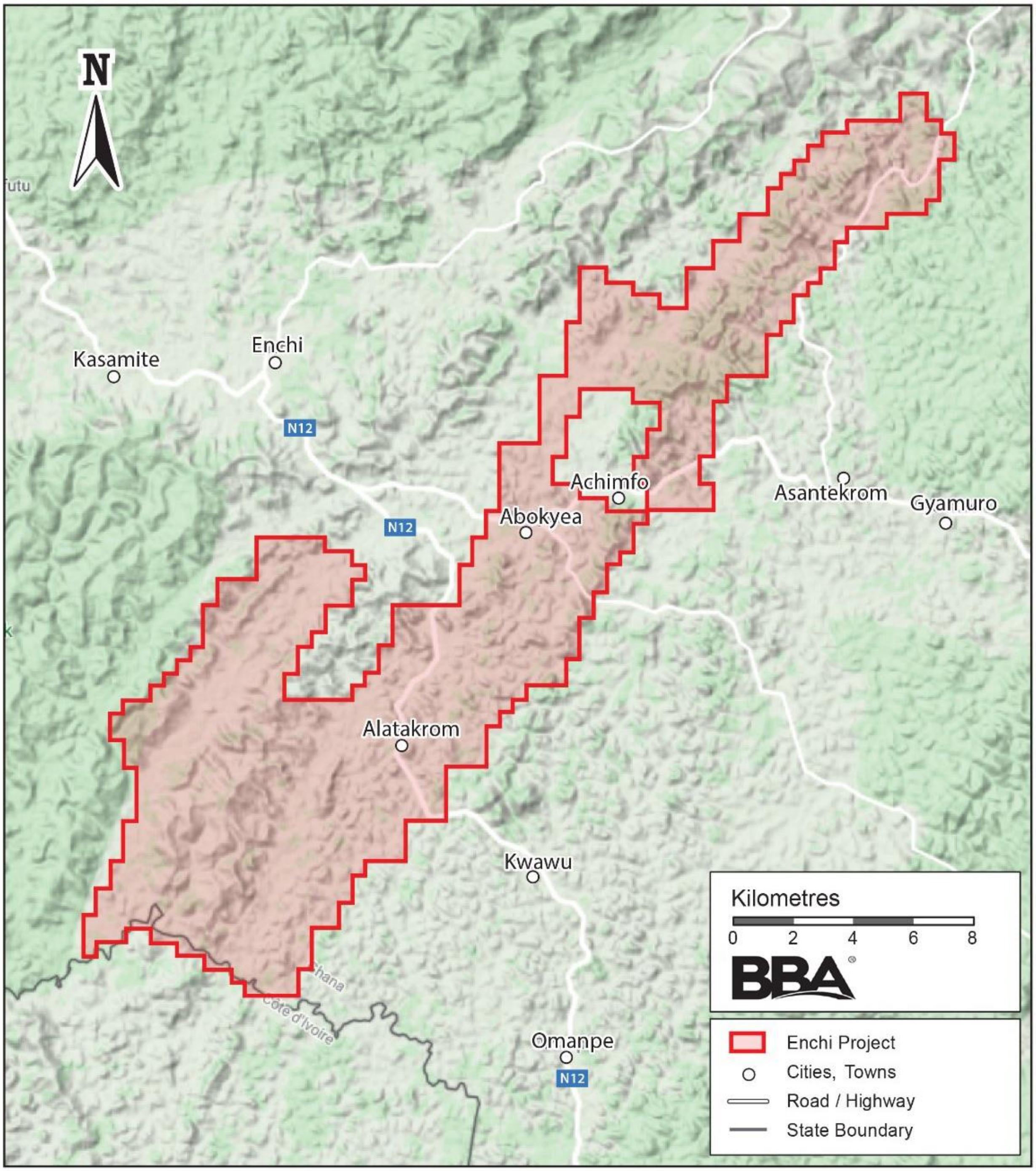


Figure 4-2: Enchi License Map

4.3 Tenure Rights

Edgewater executed a definitive Option Agreement dated May 5, 2010 that outlined the terms of an Option-Joint Venture agreement with Red Back, whereby Edgewater at the time could earn a 51% interest in Red Back's ownership interest in the Project.

In order to earn the 51% interest, Edgewater had to spend a total of CAN $5.0 million on work expenditures on the Project within 26 months, including CAN $2.0 million in the first 14 months. Edgewater would be the operator of the Option-Joint Venture agreement and would continue to be the operator of the Joint Venture as long as Edgewater held the largest equity interest in the Joint Venture.

On September 17, 2010, Kinross announced that it had successfully completed the transaction to acquire all outstanding shares of Red Back for CAN $7.1 billion, and that Red Back would become a wholly owned subsidiary of Kinross.

On May 22, 2012, Edgewater announced that it had completed the earn-in requirements of the 2010 Option Agreement with Kinross. As a result, Edgewater held a 51% interest in Kinross' ownership of the licenses and a joint venture company was to be formed.

On May 22, 2014, Newcore Gold (at the time named Pinecrest Resources Ltd.) announced that it had entered into an agreement to earn 100% interest of the Project from Kinross and Edgewater. The terms of the transaction were as follows.

For Newcore to acquire Kinross' 49% interest:

- Red Back to receive 19.9% of the issued and outstanding common shares of Newcore post closing of the transaction;
- Red Back to retain a 2% net smelter return (NSR) royalty on production from the Project with Newcore retaining the right at any time to buy back 50% (1%) of the NSR for $3.5 million. Newcore's buyback option was subsequently transferred to Sandstorm Gold Ltd. in 2014 while Kinross also sold its 2% NSR to Maverix Metals in 2019;
- Red Back to receive a payment of $10 per ounce of gold on any new NI 43-101 Measured and Indicated Resource estimate included in a Feasibility Study or any ounce of gold mined whichever occurs first. Such amount would be payable in cash or, if agreeable to Newcore, common shares of Newcore, at Newcore's sole discretion, provided that, Newcore shall not be entitled to elect to pay in common shares if such issuance would result in Red Back holding more than 20% of the issued and outstanding shares of Newcore;
- Red Back to have first right to process material from the Project at its Chirano Mill if toll processing is considered; and
- Red Back to receive 5,000,000 share purchase warrants priced at CAN $0.40 per warrant exercisable for a five-year term from closing of the transaction. Subsequently expired unexercised.

For Newcore to acquire Edgewater's 51% interest:

- Upon closing of the transaction, Edgewater to receive one Newcore post-consolidated common share (the "Acquisition Shares") for every five common shares of Edgewater issued and outstanding on the closing, which will represent approximately 40% of the issued common shares of Newcore post-closing of the transaction. All shares issued to Edgewater were to be subject to resale restrictions as follows: 25% to be free-trading six months and nine months from closing, and the remaining 50%, twelve months from closing;
- Edgewater was to agree to distribute the Acquisition Shares pro-rata to its shareholders as soon as reasonably practicable after the closing of the transaction;
- Newcore was to pay to Edgewater a cash payment of CAN $150,000; and
- The completion of the transactions contemplated by the Edgewater Letter of Intent (LOI) were subject to the execution of a definitive agreement with Newcore and the concurrent completion of the transactions contemplated by the Red Back LOI.

On December 5, 2014, Newcore announced that it had completed the acquisition of a 100% interest in the Enchi Gold Project from Edgewater Exploration Ltd. and Red Back Mining Ghana Limited, an indirect wholly-owned subsidiary of Kinross Gold Corporation. The Government of Ghana is entitled to a 10% free carried interest in the Project.

On August 6, 2020, the company officially announced a company name change from Pinecrest Resources Ltd. to Newcore Gold Ltd.

4.4 Royalties and Related Information

A 5% royalty on revenues is due to the Government of Ghana (Pricewaterhouse Coopers, 2012). Maverix Metals Inc. (Maverix) holds a 2% NSR royalty (acquired Kinross royalty in 2019) (https://maverixmetals.com).

4.5 Environmental Liabilities

The QP is not aware of any known environmental liabilities on the Property. Newcore is not responsible for small-scale artisanal and alluvial mining that has occurred across the Property and Newcore has good relations with the local communities.

4.6 Permits

All required permits for conducting exploration on the licenses have been granted or have been applied for and are awaiting government approval.

4.7 Other Relevant Factors

In areas where there is no existing surface holder, Newcore is not required to pay any compensation or fees. In areas where there is an established surface holder, Newcore is required to pay compensation when properties are disturbed, in most cases this is related to the disturbance of crops during establishment of access for exploration activities.

The risk to the Project would come in the form of the licence applications being denied by Mincom and work needing to be halted. Licences covering the mineral resources and other areas currently being completed for further work have been granted and have been extended until 2023.

There are no other significant risk factors which could affect access, title, or the right or ability to perform work. Newcore has completed successive and extensive exploration programs covering the majority of the licenses over the last ten years.

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

5.1 Site Topography, Elevation and Vegetation

The Project area is primarily drained by the Tano River and its tributaries, which flow generally in an easterly direction. Much of the Project area comprises steep topography incised by river tributaries with scattered flat plateaus with an average height of about 300 amsl.

A portion of the Project area is covered by subsistence farmland. The main food crops grown locally are cocoa, plantain, maize, cocoyam, cassava, and rice (Figure 5-1).

Figure 5-1: Cocoa Plantation (Newcore, 2021)

The northern part of the Project lies adjacent to forest reserves, and is covered by tall, primary, semi deciduous rain forest (Figure 5-2). Most of this area is reserved for commercial timber production.

Figure 5-2: Local Landscape

5.2 Access

The Project is located in the southwestern region of Ghana and is accessed from Accra on sealed roads via the regional port city of Sekondi (Takoradi) or the mining centre of Kumasi. From either of these centres, access to the town of Enchi (population of 11,737), the capital of Aowin-Suaman district, is available by paved road (Elubo-Enchi Road or the Asankragua-Enchi Road). Access through the remainder of the Project area is by dirt and gravel roads (Figure 5-3).

Accra has daily international flights to and from Europe, the US, and various African locations. Domestic flight services are available with scheduled flights between Accra and Kumasi, which is located 170 km northeast of the Project. There is no known scheduled air service to the Project area.

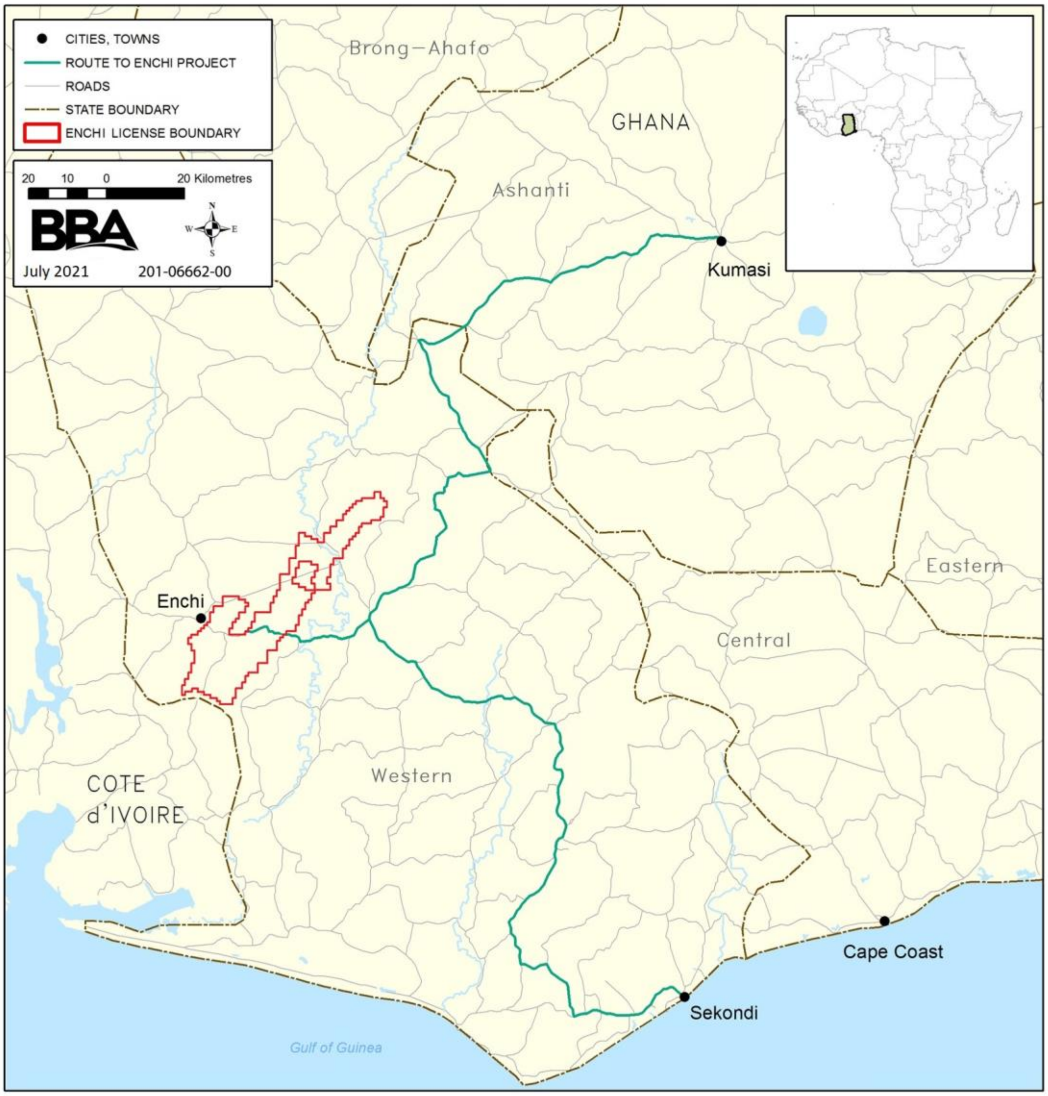


Figure 5-3: Project Access Map (modified from McCracken and Smith, 2020)

5.3 Climate

The Aowin District, within which the Project is based, is situated in the Wet Semi Equatorial Climatic Zone. The climate is typically warm and humid with a mean-monthly temperature of 27°C. There are two rainy seasons: the major rainy season from May to July, and a shorter rainy season from September to October. The district receives an annual rainfall of between 1,500 and 1,800 mm. During the dry season, predominately December to March, Harmattan winds (dry hot continental fronts from the Sahara) blow over the country resulting in drier warm days and cool nights.

Exploration and mining operation can be conducted on the Project year-round if required.

5.4 Infrastructure

The Project area has limited to moderate infrastructure. A paved road crosses the central portion of the Project leading to the town of Enchi. The remainder of the Project is serviced by a series of dirt and gravel roads. The district capital of Enchi is located 10 km west of the Project.

The town of Enchi is located 77 km north of the substation at Elubo, serviced by a 225 kV line, and 122 km southwest of the substation at Asawinso, serviced by a 161 kV line. The Chirano Gold Mine, owned by Kinross and located 50 km northeast of the Project, is supplied by power from a 33 kV overhead power line from the Asawionso main substation that is also supplying the Bibiani gold mine plant. In addition, six diesel generators are located at the Chirano facility to provide stand-by power in case of Electricity Commission of Ghana (ECG) supply issues.

Fuel, accommodations, food, and most supplies can be obtained in the city of Enchi. Potable water must either be trucked into the area or supplied through water wells. The region has a long history of mining, and there is a large population base of skilled and unskilled labour to draw upon during the exploration programs.

Modern seaports at Takoradi and Tema are located 207 km and 447 km southeast of the Project respectively and have been used for the implementation and construction of several gold mines in recent years.

6. HISTORY

The exploration activities in the entire Project area date back to colonial times, with activities completed sporadically and by various individuals and companies.

Alluvial and reef gold were prospected and exploited by several generations of galamsey (local artisanal gold miner) workings to the present day. European companies explored, developed, and mined in several phases since 1900. The result is that erratic gold in vein quartz mineralization was "opened up" in a large number of pits, shafts, and drives, notably at the Sewum, Tokosea, Alatakrom, Achimfu, Nkwanta, and Kojina Hill prospects. Only the colonial Sewum and Tokosea mines appear to have any significant development and production history although this is poorly recorded. Since the 1940s, mining activities have continued in the area on a very limited scale.

Table 6-1 summarizes the exploration activities that have taken place within the boundaries of the Project as currently held by Newcore. Due to the scattered nature of the work and the various license holders, the QP cautions that the history may not be complete. Most of the information was derived from reports and digital data acquired from Leo Shield Exploration Ghana NL (Leo Shield), Mutual Ghana Ltd. (Mutual), and Kinross. Trenching and drilling procedures and results are disclosed in Chapter 9 and Chapter 10, respectively. Metallurgical test work is disclosed in Chapter 13.

The extensive work completed by the previous landholders has resulted in the identification of at least 14 gold-bearing prospects. A summary of the results for each prospect is provided in Chapter 7.

Table 6-1: Project History

Year	Company	Activities
1987	EQ Resources	▪ 2,837 soil samples on a 100 m x 25 m spaced grid.
1993	Mt. Edon	▪ 3,260 soil samples on a 6 km by 3 km, followed by a 100 m x 25 m spaced grid; and ▪ 250 rock chip and float samples.
1994-1997	Mutual	▪ Spot imagery; ▪ Helicopter magnetic and electromagnetics on 100 m spaced lines; ▪ Fix wing magnetic and radiometric on 200 m spaced lines; ▪ 2,837 soil samples on 100 m by 25 m grid spacing; ▪ 2,257 soil samples on 200 m x 40 m grid spacing; ▪ 34 trenches totalling 2,396 m; ▪ Six diamond drillholes totalling 464 m; and ▪ RC drill program totalling 1,202 m.
1995-1998	Leo Shield	▪ 14,470 soil samples in 400 m by 50 m grid; ▪ 89 trenches totalling 10,240 m; ▪ Audit sampling at Kojina Hill and Achimfu; ▪ Stream sediment sampling (76 pits); ▪ 121 RC holes totalling 7,621 m; and ▪ 49 RAB holes totalling 2,028 m.

Year	Company	Activities
2003	Red Back	▪ Assess historical data.
2004	Red Back	▪ 237 regional stream sediment samples; ▪ 16,728 soil samples; and ▪ 148 rock chip samples.
2005	Red Back	▪ 695 soil samples; ▪ 69 trenches totalling 5,750 m; ▪ 102 RAB holes totalling 5,261 m; and ▪ 80 RC holes totalling 9,715 m.
2006	Red Back	▪ Ground magnetic survey; ▪ IP survey; ▪ 2,221 soil samples; ▪ 38 trenches totalling 3,564 m; ▪ 217 RAB holes totalling 7,182 m; and ▪ 73 RC holes totalling 7,403 m.
2011	Edgewater	▪ 9,441 soil samples over 461-line km; ▪ Twelve trenches at Nyam totalling 396 m; ▪ Three trenches at Sewum totalling 781 m; ▪ Eight trenches at Boin totalling 359 m; ▪ Seven trenches at Eradi totalling 1,294 m; ▪ VTEM/magnetic/radiometric survey totaling 3,084-line km; ▪ 182 diamond drillholes and 13 RC holes totalling 23,697 m; and ▪ Resource estimation completed on Boin, Sewum and Nyam.
2012	Edgewater	▪ Completion of 25 RC holes totalling 4,058 m; ▪ Bottle roll tests; and ▪ Soil and rock sampling, auger drilling, and trenching.
2014	Pinecrest	▪ Completes acquisition of the Project from Edgewater and Kinross.
2015	Pinecrest	▪ Completion of a PEA.
2017	Pinecrest	▪ Completion of 28 RC holes totalling 3,406 m.
2020	Newcore	▪ Company changes name from Pinecrest Resource to Newcore Gold; ▪ 25 RC holes at Sewum totalling 4,269 m; ▪ 53 RC holes at Boin totalling 8,812 m; ▪ 8 RC holes at Nyam totalling 1,030 m; ▪ 18 RC holes at Kwakyekrom totalling 2,457 m; ▪ Resource estimation completed on Boin, Sewum and Nyam; and ▪ Bottle roll tests.
2021	Newcore	▪ 6 RC holes at Sewum totalling 1,194 m; ▪ 20 RC holes at Kwakyekrom totalling 3,026 m; ▪ Bottle roll and column tests; ▪ Resource estimation completed on Boin, Sewum, Nyam and Kwakyekrom; results summarized in Section 14.11 of this technical report; and ▪ Completion of a PEA.

Resource estimations were completed on the Project in 2012, 2014 and 2020. The resources are considered historic and Newcore is not treating the resource statements in Table 6-2 as current. The changes in the resource statements is attributed to additional drilling, the application of lower cut-off grades based on adjustments to the operating costs and a higher gold selling price. The resource statements in 2012 and 2014 were not pit constrained.

Table 6-2: Historic Resource Statements

Zone	Cut-off (g/t)	Tonnes	Grade Au (g.t)	Contained Gold (ounces)
2012 (McCracken, 2012)				
Sewum	0.7	7,549,000	1.09	264,500
Boin		9,551,000	1.20	368,500
Nyam		3,716,000	1.13	135,000
Total		**20,816,000**	**1.15**	**768,000**
2014 (McCracken et al, 2015)				
Sewum	0.5	16,135,000	0.82	423,700
Boin		15,872,000	0.96	489,900
Nyam		5,350,000	0.96	165,100
Total		**37,357,000**	**0.90**	**1,078,700**
2020 (McCracken and Smith, 2020)				
Sewum	0.3	27,600,000	0.60	535,800
Boin		19,837,000	0.84	533,000
Nyam		5,489,000	0.88	155,000
Total		**52,926,000**	**0.72**	**1,223,800**

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Enchi concession is located within southwest Ghana and straddles the boundary between the Sefwi Volcanic Belt to the west and the Kumasi Sedimentary Basin to the east. The Sefwi Belt and Kumasi Basin are comprised predominantly of Birimian-age rocks (2.17 to 2.18 Ga) (Davis et al., 1994) (Figure 7-1).

The Sefwi Belt is dominated by mafic volcanics, metasediments, and intrusive granitoids that are sandwiched between sedimentary basins (Sunyani Basin to the west and the Kumasi Basin to the east). The Sefwi Belt is traceable for hundreds of kilometres along strike, yet is usually only 20 to 60 km wide. The metavolcanic and metasedimentary sequences are believed to be contemporaneous, with the sediment deposited in basins eroded from the adjacent volcanic terrains (Asiedu et al., 2004).

The Kumasi Basin is characterized by wide sequences of marine clastic sediments (quartzite, conglomerates, and phyllites). Both the Birimian sediments and volcanics have been extensively metamorphosed to greenschist facies, locally to amphibolite facies. The boundary between the volcanic belts and basins can be gradational, yet is typically faulted with the faults most likely representing basin margin growth faults along which basin subsidence occurred (Hirdes and Leube, 1989).

Granitoid intrusions are common within the belt and basin terrains and can be divided into two types: Belt Type (Dixcove) and Basin Type (Cape Coast) granitoids. Belt type granitoids (2,180 Ma) range from tonalite to granodiorite in composition and are confined to the metavolcanic belts. Basin granitoids (approximately 2,116 to 2,088 Ma) are mainly granodiorite in character and contain more potassium and rubidium relative to the belt granitoids and are concentrated in the central portions of the Birimian metasedimentary basins (Hirdes and Leube, 1989).

Extensive faulting occurs along the margins of the volcanic-sediment belts. Observed at local and regional scales, these northeast-trending structures are fundamentally important in the development of gold deposits for the region. The major shear system within the Enchi concession at the boundary of the Sefwi Belt and Kumasi Basin is termed the Bibiani Shear Zone. Gold deposits are located in third-order structures that splay off the second-order structures and sub-parallel to the overall trend of the Bibiani Shear Zone. The Bibiani Shear Zone has been traced for 40 km on the Project area. Major structures within the concession are named from west to east, the Bibiani Shear (BS), the West Sewum Shear (WSS), and the Nyam Shear (NS).

The Obuasi-Enchi lineament, a major east-west crustal scale feature, deflects the Bibiani Shear Zone at the north end of the Property in the vicinity of the Eradi gold prospect. This lineament is associated with the major Ashanti and Akyem gold deposits in the Ashanti Belt, 100 to 200 km to the east.

Multiple tectonic events have affected virtually all Birimian rocks. The dominant event is compressional folding and thrusting from the Eburnean Orogeny (2.1 to 2.2 Ga) (Schofield, 2006; Eisenlohr, 1989).

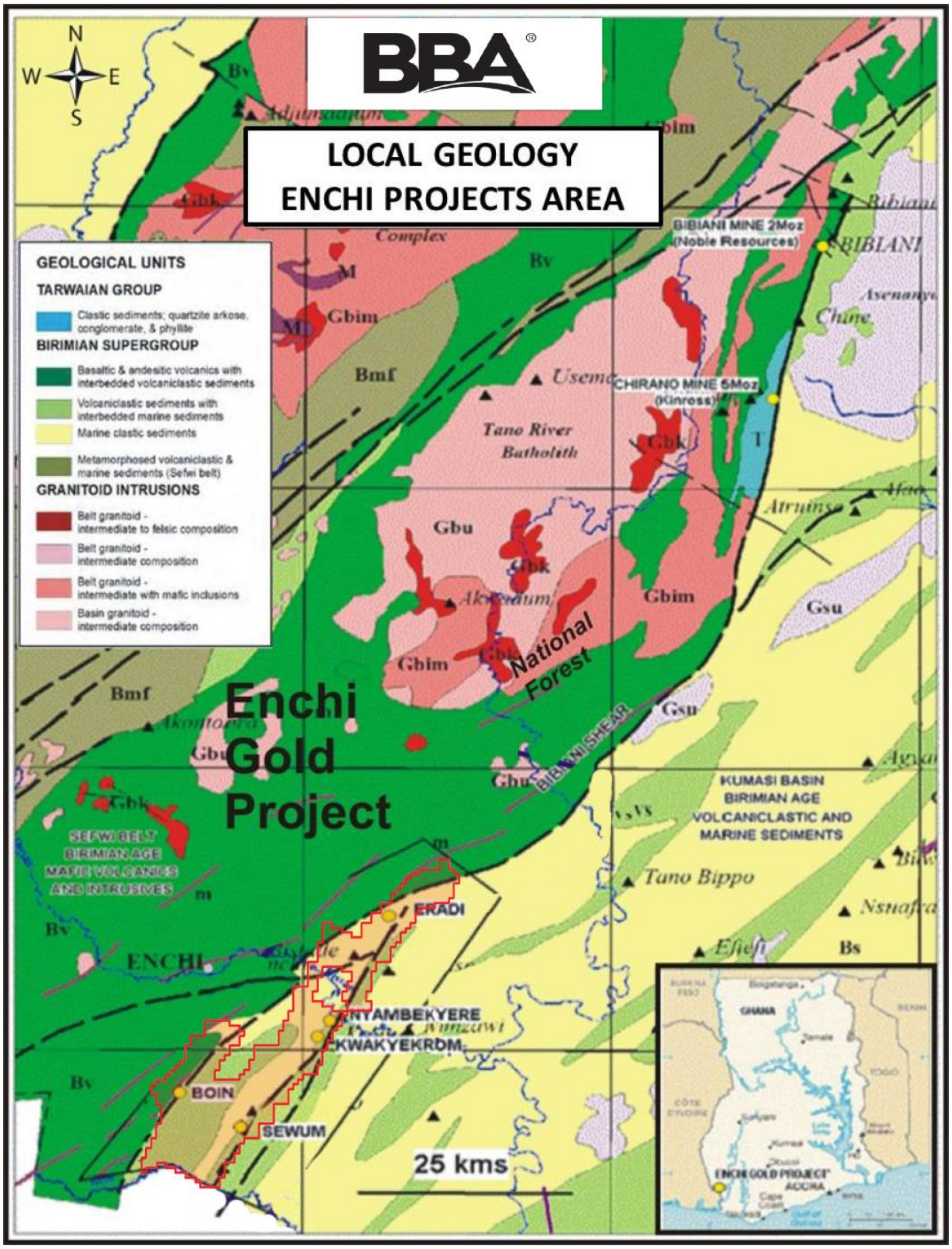


Figure 7-1: Regional Geology (modified from Davis et al, 1994)

7.2 Project Geology

The Project overlaps 40 km of the belt-basin contact on the east side of the Sefwi Volcanic Belt north of the Côte d'Ivoire border. The contact is marked by a major fault known as the Bibiani Shear Zone, which also hosts the Chirano and Bibiani Gold mines located 50 km north of the Enchi licenses (Figure 7-2).

The Project is characterized by variably degraded laterite to residual soil profiles with minor caps of indurated ferro-duricrust across the main hilltops. Rock outcrops are rare due to the thick tropical weathering and jungle cover. Most rock exposures are found in road cuttings and by trenching.

Numerous other major faults splay off the Bibiani Shear Zone pass through the license area, e.g. Boin Fault, Sewum Fault, and Nyam Fault. Many gold deposits in the Enchi district are localized along or adjacent to these structures.

The regional scale shears are believed to have been originally formed as thrusts during northwest-southeast compression with later movements dominated by left lateral strike slip shearing (Griffis, 2002).

The principal rock types found on the Project are defined below.

- **Volcanics (MB):** massive, very fine-grained, textureless, weathered white to brown, to deep pink and red, igneous rock generally evident as un-deformed rafts, fault-bound, within foliated and sheared volcaniclastics and pelitic sediments.
- **Volcaniclastics (SVC):** hanging wall, fine- to medium-grained, lithic to crystal volcaniclastic wacke, with a characteristic porous, spongy, honeycombed texture. It weathers to light pink and is variably graphitized and foliated to sheared, proximal to the late faults.
- **Turbidites (SPH):** footwall, metre-thick, cyclically bedded, turbidite sequence of graded, fine- to medium-grained, grey to black, phyllitic pelite-psammite beds. The finer pelite horizons are more preferentially strained and the coarser units are more preferentially fractured.
- **Graphitic Phyllites (SPG):** black, very fine- to fine-grained carbonaceous and graphitic altered phyllites and schists. Each of the host rock-types may be preferentially graphitized ±silicified and sheared proximal to the reactivating faults and shears, becoming increasingly assimilated to SPG. Within and proximal to the main SPG deformation zones, texture was the main discriminating feature used to distinguish and map the SVC-SPH contact.
- **Quartz Veins (QV):** massive 0.5 to 5 m wide, white to smoky, blue polyphase quartz veins variably faulted and graphitized and mineralized. The major quartz zones represent the main hanging wall deformation zone developed as a result of the progressive movement along the basal contact shear zone.

- **Basic, Intermediate and Felsic Dykes and Sills:** coarse-grained granodiorite to diorite and finer grained equivalent andesites to dolerites have been logged. The felsic and intermediate dykes tend to be layered parallel, altered and structurally deformed within the surrounding host volcanics and sediments. The dolerites are generally much later, crosscutting. They were traditionally mapped as post-deformational, though they are often crosscut and displaced by late reactivation. There is evidence for multiple generations of dolerites through to post-Cretaceous times.

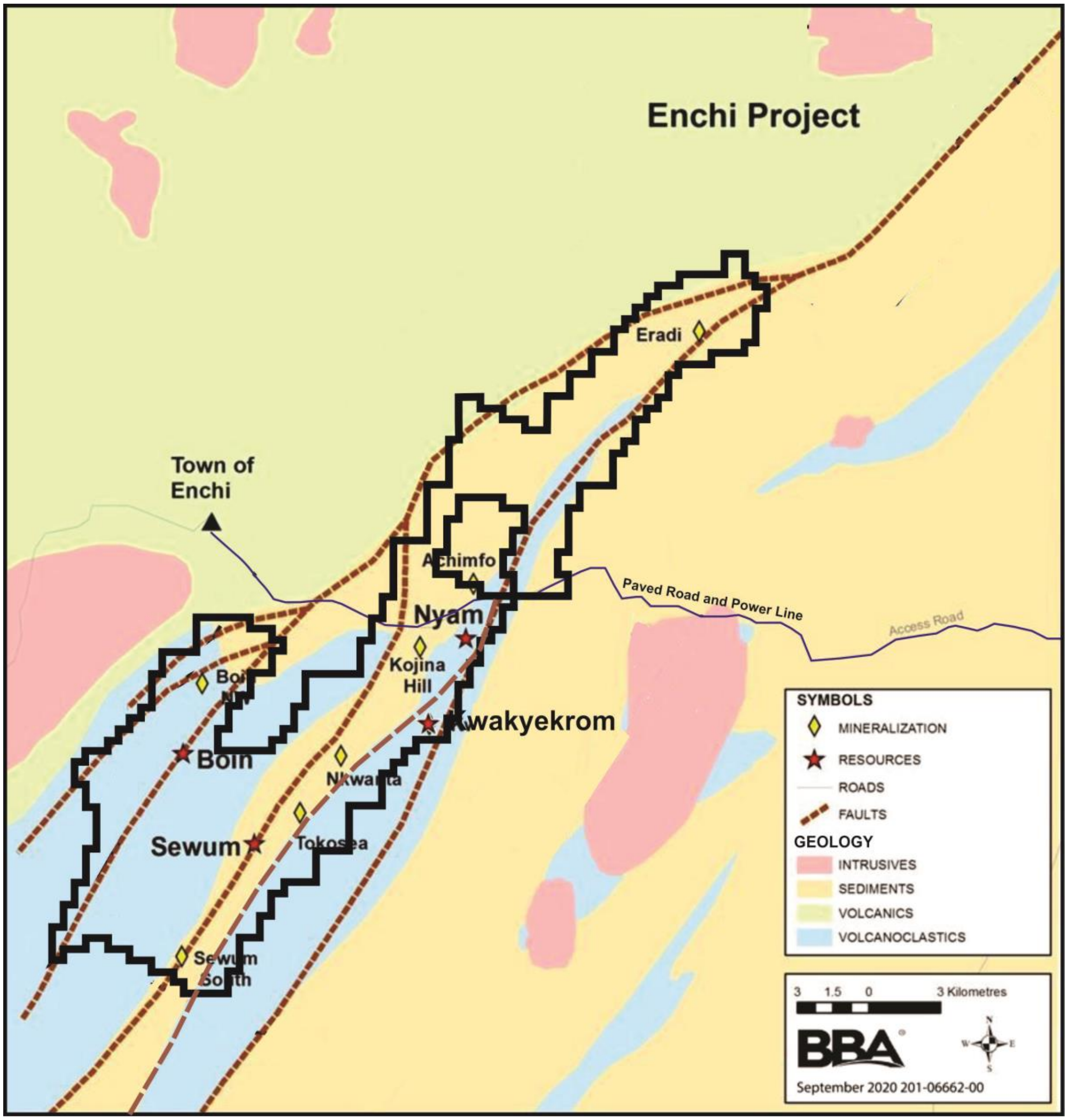


Figure 7-2: Project Geology (modified from McCracken and Smith, 2020)

7.3 Mineralization

Fifteen gold zones or prospects have been identified on the Project to date. The locations of the zones are illustrated on Figure 7-3.

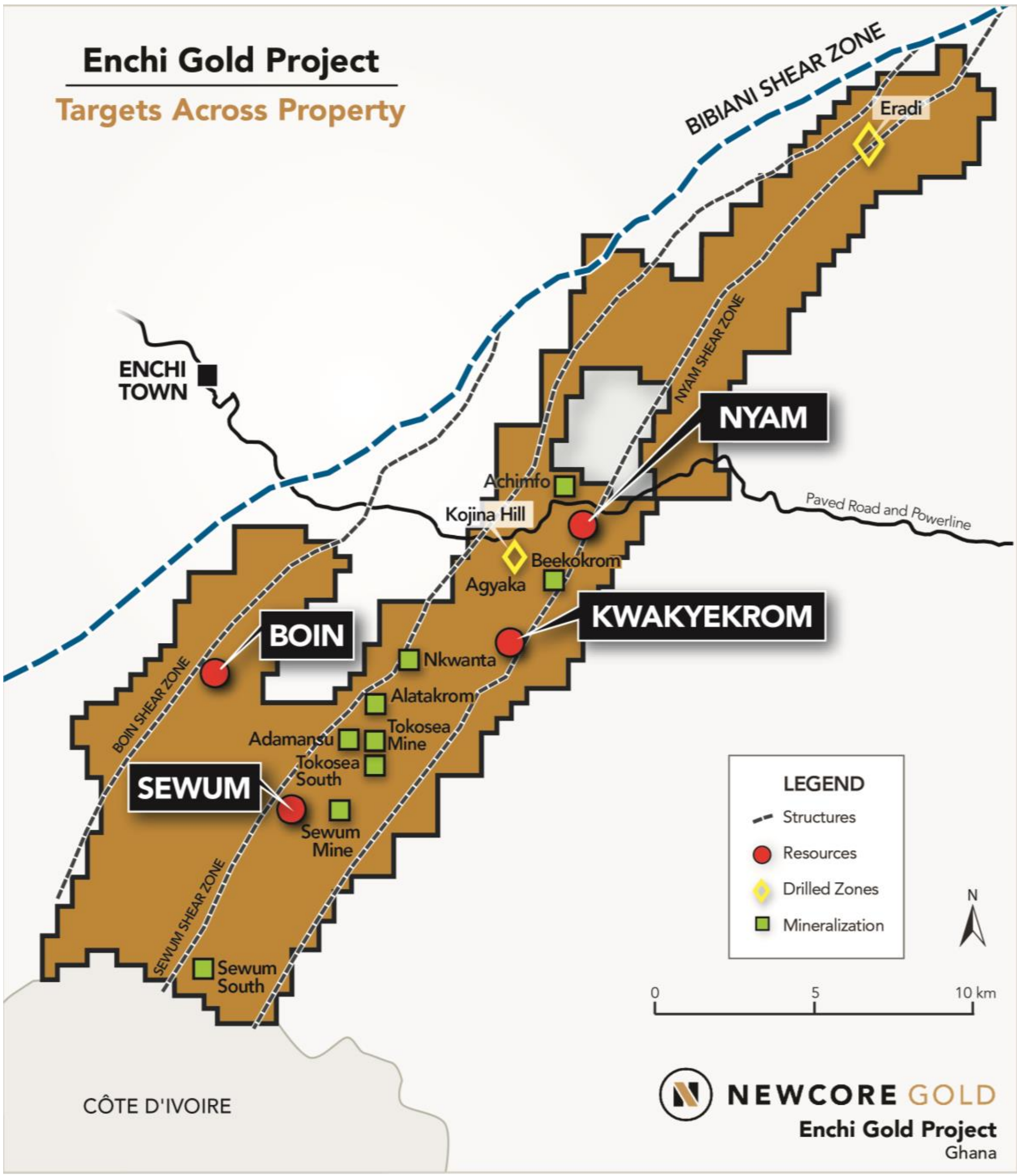


Figure 7-3: Mineral Zones (Newcore, 2021)

7.3.1 Sewum Zone

The Sewum West and South prospects are found along the eastern contact of a thrust-bounded volcanic sliver, outcropping 6 km to the east of the Boin Zone on the NS. The gold mineralization is associated with late D2 to D4 deformation phases. It is structurally controlled within, and adjacent to, late graphitic faults focused on the margins of poly-phase quartz veins within faults. The veins developed along the axial planes of hinges and limbs of earlier hanging wall D3 drag folds ± intrusives.

The Sewum Gold Prospects form a continuous 40 km strike length of prospects from Sewum Hill northeast through Kojina Hill and Nyam Zone to the north.

The main relief of Sewum West Hill is characterized by a relict indurated, duricrust, or ferricrete plateau along the main hilltop, degraded breakaways forming the slope crests and variably mixed and transported upper-slope soils progressing into residual mid- and lower-slope soils. The duricrust mantle is geochemically subdued and potentially ferricrete bearing. Various surrounding hilltops have similar remnant duricrust caps and should be evaluated with care to understand and develop the regolith model for the region. Sewum West Hill has a very significant deep weathering profile.

The Sewum setting differs, however, in the scale of shear zones as compared to those expressed at Boin and has proportionally more igneous volcanic rocks and late-stage, intrusive intermediate and felsic dykes or sills.

The Sewum prospects are situated along several major thrust zones that crop out across the regional 3-km wide north-south corridor, south of Tokosea. The structures comprise (west to east) (Figure 7-4):

- Road Zone (SRZ);
- Hilltop Shears (SHS);
- Main Contact Zone (MCZ);
- Sewum West Zone (SWZ);
- Sewum-Tokosea Mine Trend (SETO).

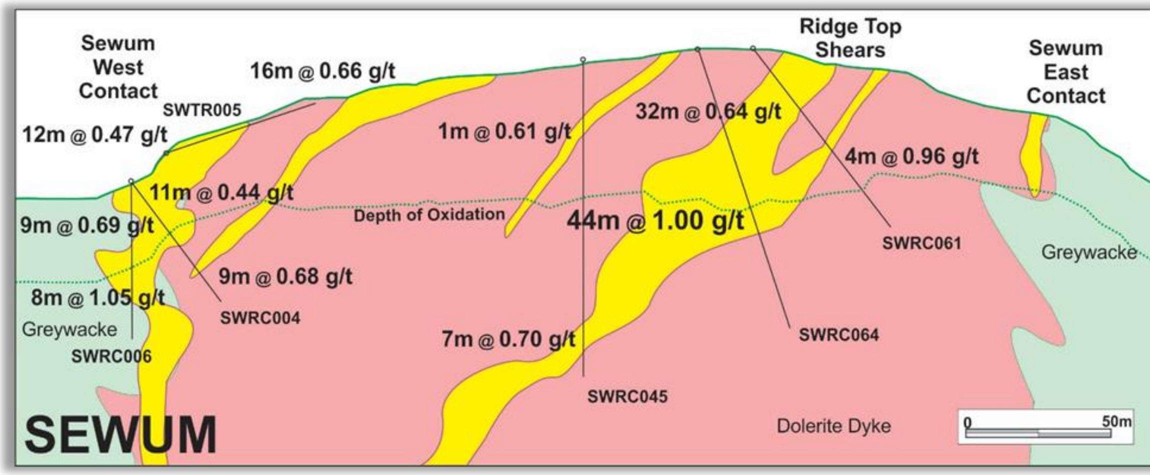


Figure 7-4: Sewum General Section (Newcore, 2020)

The host rocks at Sewum include interbedded carbonaceous siltstone and sandstone (turbidite). The sediments have been regionally deformed to greenschist facies, are steeply dipping, and typically strike north-northeast (30º) parallel to the regional structural grain. A steeply dipping dolerite dyke 3 km long and up to 500 m wide has been intersected in the drilling and acts as an important host to gold mineralization in the Sewum area.

Three styles of mineralization have been identified at Sewum:

- Disseminated arsenopyrite associated with quartz veining and silicification in sheared sediment, e.g. Sewum Shear Zone.
- Quartz – sericite – carbonate replacement of sheared dolerite and sediment localized along moderately (40°) dipping shears hosted within dolerite, e.g. Sewum Ridge Top Shears (SRTSZ).
- Brecciated and stockworked sediment and dolerite developed at the margin of the dolerite dyke and replaced and in-filled by quartz-sericite ankerite and minor sulphides, e.g. Checkerboard Hill, East Contact Zone (ECZ), and West Contact Zone (WCZ).

The Sewum Shear Zone represents a major regional structure that can be traced within Ghana for 25 km south from where the shear branches off the Bibiani Shear Zone and continues across the Ghana border into Côte d'Ivoire. The shear has a complex anastomosing geometry with numerous splays and has played a major role in localizing gold mineralization in the Sewum area, e.g. Adamansu, Sewum, and Tokosea goldmines currently operating small-scale mines.

Striking north-northeast, the Sewum Shear is typically vertical to steep west dipping and can be up to 100 m wide. Mylonitic fabric has been observed within the shear zone in places. Gold mineralization within the Sewum Shear is related to a phase of quartz veining with associated arsenopyrite.

Mineralization is discontinuous and appears to be related to an early phase of quartz veining that has been brecciated by later movement along the Sewum Shear.

The dolerite dyke at Sewum has acted as a solid “node” with the bulk of the regional scale deformation absorbed by the surrounding host fine-grained carbonaceous sediment. Branches of the Sewum Shear have anastomosed around the dolerite dyke and in places mark the contact.

The most significant zone of continuous gold mineralization identified in drilling at Sewum is the Ridge Top Shears Zone (RTSZ), related to moderately dipping shears up to 20 m thick hosted within the dolerite dyke.

The relationship of these shears with the Sewum Shear Zone is unclear but they are most likely temporally related. The shears within the dolerite may be thrust faults or faults that link between the steep shears that anastomose around the dolerite dyke.

The dolerite intrusive has not been faulted into place as along the dyke’s west margin features typical of intrusive contacts such as frictional “intrusive breccia”, hornfelsing of adjacent sediment and chilled margins within the intrusive have been observed. The age of the dolerite dyke is not certain, however the partially sheared east contact, spatial relationship with gold mineralization and some drill core features indicative of soft sediment deformation at the intrusive contact, indicate the intrusive was probably emplaced during the Eburnean Orogeny similar to most other mafic intrusives in the region. It is also possible the dyke may have been intruded as a sill along bedding planes and later tilted vertical during region deformation along with the host sediment.

The size and composition of the intrusive at Sewum is more akin to the “belt” style intrusives than the “basin” style intrusives which tend to be larger, coarser grained, and felsic in composition (Griffis, et al. 2002).

The presence of the dolerite body within the Sewum Shear Zone is significant in that the intrusive represents a more competent rock type compared to the surrounding sediment and is more likely to deform in a brittle manner during faulting and deformation, potentially making a better (more permeable) host to mineralization similar to the Chirano Gold Mine (brecciated granite host).

Mineralized breccia and stockworking is commonly found along the margin of the dolerite dyke (ECZ and WCZ). The breccia is composed of angular clasts of siltstone and dolerite in a clast support fabric cemented by quartz, carbonate, and minor pyrite. The breccia texture indicates very little milling and mixing of fragments has occurred and was formed by hydraulic fracturing, probably in response to fault movement near the intrusive contact.

7.3.2 Boin Zone

The Boin Shear Zone is one of a number of major structures that splay off the BS and pass through the Project. The Boin Shear Zone is interpreted as a thrust fault, dipping moderately west and is responsible for the development of the zone of mineralized quartz veins at Boin. Eleven kilometres of the Boin Shear Zone has been drill tested at shallow depths over regular intervals across the structure. A generalized section is shown on Figure 7-5.

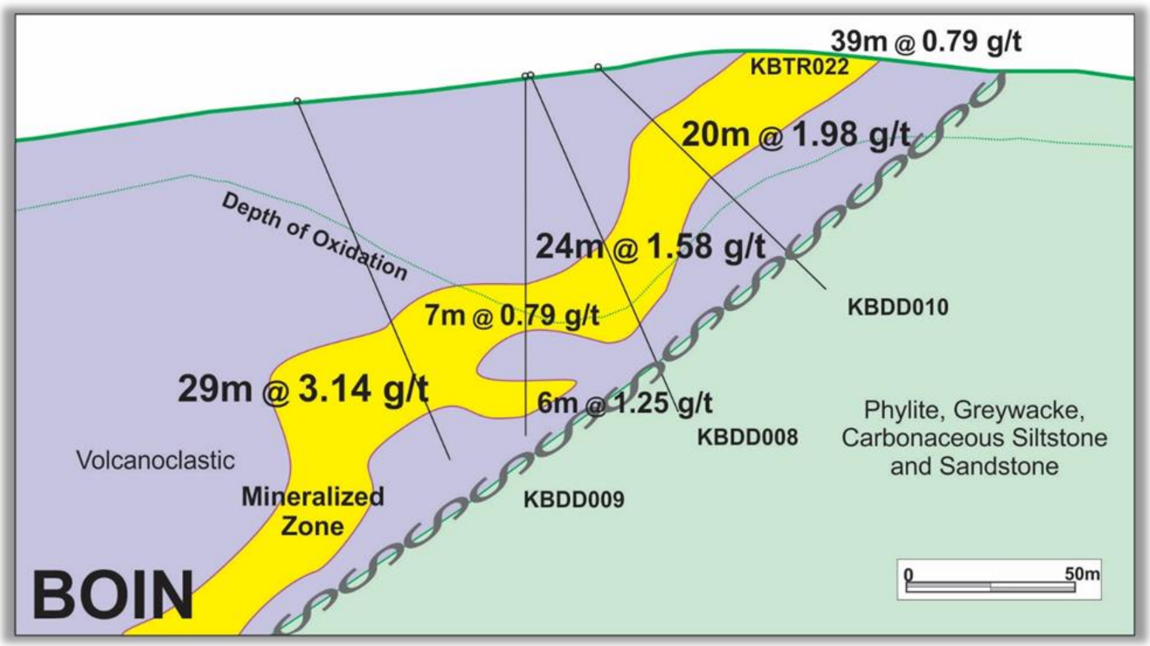


Figure 7-5: Boin General Section (Newcore, 2020)

The Boin Shear Zone is formed in the west hanging wall of this major second-order, west-dipping, thrust contact between mafic volcanic ±volcaniclastic sediments which over-thrust turbidites to the east. The whole contact is expressed as a 10 to 30 m wide graphitic shear zone, which trends 025° to 040° and dips west 30° to 70°. The Boin thrust is an early, regionally second-order splay or replication off the main basin-boundary contact further to the west. Multiple sets of crosscutting fabrics, veins, and faults have been recorded within the core and trench logging. The gold is mostly found in the hanging wall quartz zone and is characterized by massive 20 to 30 m wide zones of intensive quartz veining cut and fractured by late, graphitic faults.

There are multiple generations of pyrite developed within the Boin structures. The early, barren, non auriferous pyrite tends to be intense, well-formed, coarse, and cubic. The later, possibly remobilized, auriferous pyrite tends to form as fine to very fine, disseminated cubic crystals within graphitic fault margins, or amorphous ribbons, rims or coatings within quartz veins.

Hydrothermal alteration displays a typical greenschist assemblage (gold + quartz + sericite ±graphite ±chlorite ±epidote ±ankerite). Chlorite + epidote clots are observed within, or proximal to, the gold mineralization within the brecciated quartz veins. These probably result from remobilization associated with regional alteration.

No visible bleaching or other styles of alteration have been observed in the host sediment related to the quartz veining apart from narrow silicified vein selvedges. At the Boin Zone, the depth of intense weathering is up to 100 m in places. Weathering is deepest where the mineralization is best developed suggesting the greater intensity of veining and fracturing may have enhanced the weathering over the deposit.

7.3.3 Nyam Zone

The Nyam Zone strikes over a distance of 1.8 km, hosted by altered phyllite, 200 to 300 m west of the interpreted position of the second order NS. The zone of mineralization lies in the hanging wall of a northeast-striking shear that dips 70º east and is up to 30 m thick. Nyam mineralization is part of a continuous 15 km strike length of gold prospects on the Project from Nyam southwest through Kojina Hill to Sewum in the south. An extensive envelope of weak gold mineralization (more than 0.25 g/t) dips sub-vertically and strikes 030° (Figure 7-6).

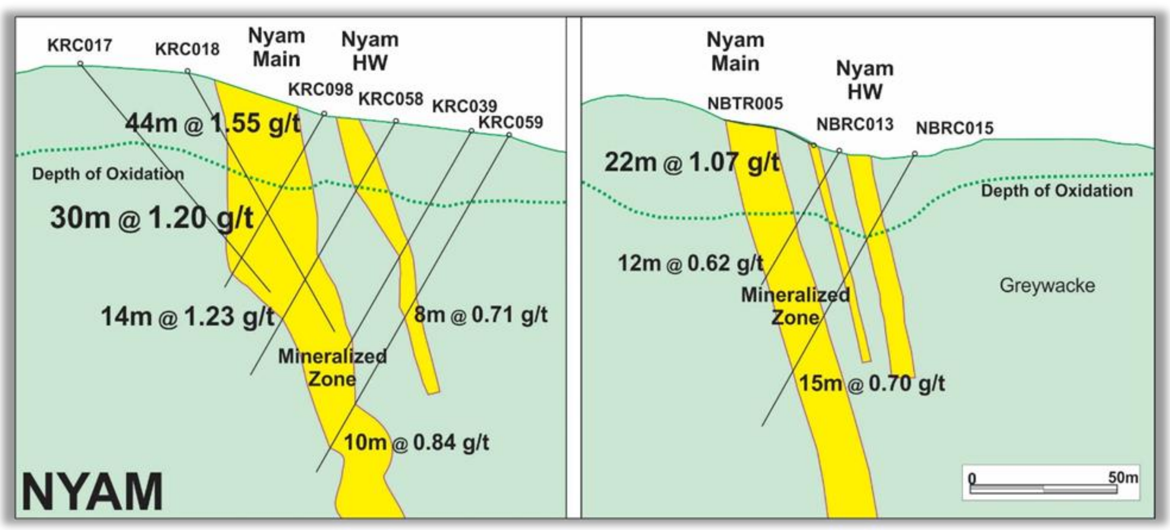


Figure 7-6: Nyam General Section (Newcore, 2020)

Mineralization at the Nyam Zone is composed of veined and brecciated sediment and phyllite cemented by quartz, carbonate (ankerite), and albite and has been traced continuously in trenching and drilling for over 2 km along strike.

Alteration associated with the zone of veining and brecciation consists of bleaching due to replacement by sericite, quartz, ankerite, albite, rutile, and minor pyrite. Pyrite typically makes up less than 1% of the infill and alteration minerals. No visible gold or arsenopyrite or base metal sulphides have been identified in any core samples to date.

The footwall of the mineralization is marked by carbonaceous shears and a 2 to 3 m wide zone of green coloured fuchsite-magnesium chlorite alteration. The fuchsite is believed to represent an alteration front where chromium leached from the altered volcaniclastic sandstone beds and has been re-deposited in micas, replacing the basal shear adjacent to the quartz-carbonate-sericite alteration zone.

The zone of quartz-sericite-carbonate bleaching has a gradational upper contact and is not always mineralized. Carbonaceous shears cut through the mineralization indicating that the shear zone has continued to move after the mineralization event. Post-mineralization deformation is also supported by petrologic studies that describe stylolites, recrystallization, strained and sutured quartz, and albite grains in the vein material (England, 2011).

Rare sphalerite and anhedral grains of chalcopyrite less than 0.1 mm in size, rimmed by tetrahedrite – tennantite have been observed in the quartz veins during petrological studies (England, 2011).

7.3.4 Kwakyekrom Zone

The Kwakyekrom Zone is located 3 km south of the Nyam Zone and is interpreted to be related to the extension of the same structure. Drilling has tested the Kwakyekrom Zone over a strike distance of 1.3 km and to a depth of approximately 150 m. The zone is hosted by altered phyllite, 700 to 800 m west of the interpreted position of the second order NS structure. The phyllite has been intersected by metre-scale dolerite dykes similar in composition to the larger intrusive bodies encountered at Sewum.

Kwakyekrom mineralization is part of a continuous 15 km strike length of gold prospects on the Project from Nyam southwest through Kwakyekrom to Sewum in the south. Gold mineralization is hosted in a series of sub-parallel zones (more than 0.20 g/t) ranging in width between 5 and 25 m and dipping sub-vertically and striking 030° (Figure 7-7).

Kwakyekrom mineralization is associated with sediments showing intense ductile strain, with centimetre- to metre-scale quartz veins focused within brittle-ductile deformation zones. Additionally, sediment-dolerite contacts are often the site of quartz veins and variable gold mineralization.

The NNE-SSW-striking metasedimentary package steep to moderate dips consistent with high degrees of ductile strain and possibly the presence of tight folds in the stratigraphy. As with Nyam, the main fabric is overprinted by a moderately developed crenulation that dips to the NW.

The alteration associated with the zone of veining and brecciation consists of bleaching due to replacement by sericite, quartz, ankerite, albite, and minor pyrite but is not as well developed as at Nyam. Fine-grained pyrite is focused around discrete quartz veins ranging in width from <0.1 m to more than 1.5 m. No visible gold or arsenopyrite or base metal sulphides have been identified in any samples to date. A series of crosscutting graphitic sheared structures ranging in width between 0.2 and >1.5 m, are present.

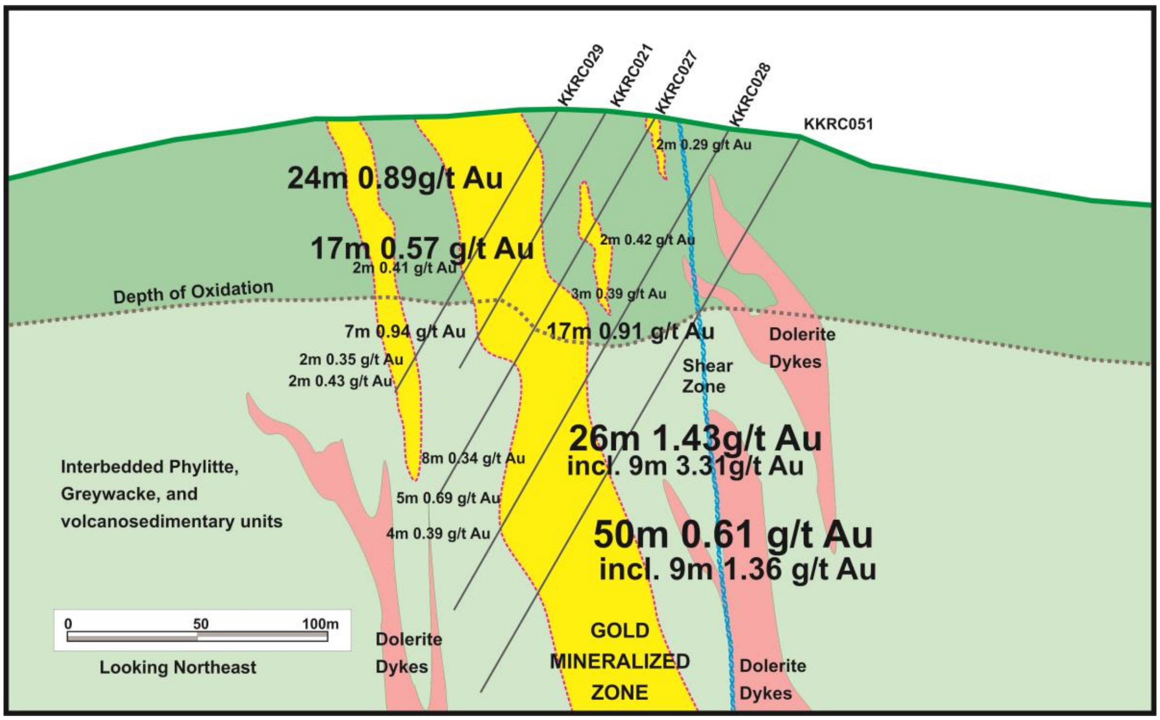


Figure 7-7: Kwakyekrom General Section (Newcore, 2021)

7.3.5 Eradi

The Eradi prospect is located in the north of the Enchi license area where the regional structures converge and gradually change strike from north-northeast to northeast. Very little outcrop exposure is present at Eradi due to the thick weathering profile and laterite development. All geology mapped comes from trenches and drillholes.

The Nyam Shear Zone (NSZ) is one of a number of major structures that splay off the Bibiani Shear and pass through the Enchi license area. Mineralization at Eradi is developed within a second order shear that parallels and lies 300 m west of the NSZ. Gold mineralization at Eradi is entirely hosted in quartz veins. The veins are very irregular in shape, size and orientation, rarely exceeding 1 m in thickness and tend to dip moderately (20° to 60°) east. The intensity of veining varies markedly between drill sections. Quartz in the veins is composed of white, less than 10 mm anhedral grains that are often fractured and recrystallized by later shearing. The quartz veins are generally quite pure, containing rare carbonate minerals and no sulphides.

No visible bleaching or other styles of alteration have been observed in the host sediment related to the quartz veining apart from narrow silicified vein selvedges. No intrusives have been identified in trenches or drill core at Eradi.

The host rocks at Eradi are dominated by interbedded carbonaceous siltstone and sandstone (turbidite). The sediments have been regionally deformed to greenschist facies, are steeply dipping, and typically strike northeast (040°) parallel to the regional structural grain. Gold mineralization at Eradi is hosted in irregular quartz veining localized along northeast striking shear zones with a near vertical dip.

7.3.6 Achimfo

Several thin (less than 1 m wide) quartz veined structures are hosted by phyllite exposed in old workings including small shafts and galamsey workings over strike-lengths of up to 400 m and depths of up to 40 m. Erratic high-grade gold is hosted by quartz veining. The vein hosting structures are considered steep southeast dipping thrusts that juxtaposed folded finer- and coarser-grained metasediments (carbonate altered siltstones, pyrite altered quartzite, and greywacke).

7.3.7 Adamansu

Quartz veining is hosted by phyllite, within a contact zone, with volcaniclastics to the west. The contact zone is possibly the fault displaced strike extension of that at the Sewum mine, and the southern extension of that at the Tokosea Mine.

7.3.8 Alatakrom

The Alatakrom prospect is along strike, northeast of the Tokosea East prospect. Several conformable sub-vertical gold mineralized quartz vein zones are hosted by phyllite, within 50 m of a contact with volcaniclastics to the west.

7.3.9 Beekokrom

The prospect straddles projected strike positions of mineralized structures defined at the Kwakyekrom prospect, 2 km to the southeast.

7.3.10 Kojina Hill

Mineralization is hosted by a zone of deeply weathered quartz-veined phyllite. Fuchsite-altered greywacke is also noted. Mineralized zone dips west at 80° and plunges steeply to moderately north.

7.3.11 Nkwanta

An adit at the Nkwanta prospect tests a weakly mineralized narrow quartz vein over a strike of 300 m. The quartz vein is hosted by phyllite, within a contact zone, with volcaniclastics to the west. The contact zone is possibly the strike extension of that in the Tokosea Mine 3 km to the south.

7.3.12 Sewum Mine

The Sewum Mine developed two narrow (0.5 to 1 m wide) quartz veins, the Main Reef and West Reef, over a strike of 450 m. The veins dip southeast at 45 to 60° within a strongly deformed carbonaceous phyllite near a contact with less deformed volcaniclastics to the west. The Sewum Mine is possibly hosted by a bedding concordant splay from the second order splay.

From 1940 to 1951, the Kwahu Mining Co. deepened the main shaft to 120 m and developed the 45 m and 78 m levels. No production was recorded (Kesse, 1985).

7.3.13 Sewum South

The Sewum South Target is located 3 km south of the Sewum Deposit. Soil sampling has generated the largest individual anomaly on the Enchi Project measuring 3.5 km by 1.5 km. The anomaly is associated with a wide and complex conductive zone in the airborne electromagnetic survey suggesting structural and geological similarities to the Sewum Deposit area. Some of the lower lying portions of the Sewum South area have been the site of artisanal gold mining activity.

7.3.14 Tokosea Mine

The Tokosea prospect is located on the same phyllite/volcaniclastic contact as that located west of the Sewum Mine, although offset by faulting south of Adamansu. The prospect includes the workings of the Tokosea Mine along with several parallel and en-echelon gold mineralized quartz veined zones some 30 m to the east, including the Tokosea East prospect. The mine has development on the 18 m, 27 m, and 45 m levels.

All the significant gold mineralization is hosted by sub-vertical quartz veined structures in phyllite with some gold in quartz veinlets within the volcaniclastic unit. The main structure developed in the Tokosea Mine is a shear hosted, thin (0.3 to 1 m) lenticular quartz vein, dipping 85° northwest, and following a contact between a dominantly argillaceous (phyllite) footwall (eastern) and a dominantly volcaniclastic hanging wall (western) unit. The immediate host rock is a black carbonaceous phyllite. The general strike is 030°.

7.3.15 Tokosea South

An adit intersected patchy gold (the best result was 1 m at 2.04 g/t) on a shear zone with quartz stringers and veins hosted by phyllite, within a contact zone, with volcaniclastics to the west. The contact zone is the strike extension of that in the Tokosea Mine, 800 m to the northeast.

8. DEPOSIT TYPES

The Project's mineralized zones have the characteristics of epigenetic, mesothermal quartz vein style gold deposits with an overlying gold-bearing saprolite. This type of mineralization is the most important type of gold occurrence in West Africa and is commonly referred to as the Ashanti-type.

Mesothermal mineralization has a strong structural control and brittle-ductile deformational style that is related to large tectonic corridors (more than 50 km long and several kilometres wide). These deformational zones display evidence of complex multi-phase displacement with mineralization typically associated with second and third order structures (Roberts, 1988). Auriferous veins are best developed at dilatational sites where structural or compositional irregularities occur within the shear structure. Favourable sites include conjugate or branching shear zone intersections, major flexures within the shear plane, and compositional variations associated with major lithological contacts or incorporated dyke material.

The most common host rock is usually a fine-grained metasediment in close proximity to graphitic or siliceous chemical sediments. However, in some areas, mafic volcanic and intrusive rocks are known to host significant gold mineralization as at Kinross' Chirano Gold Mine located 70 km north-east of the Project.

Mesothermal alteration is generally more visible within greenschist facies settings. Alteration usually occurs as chloritization, pyritization, silicification, and tourmalinization, with minor amounts of potassic and alkali feldspar alteration as well as potassic phyllosilicate (sericite, muscovite, and biotite) alteration. Carbonate alteration is pervasive (ankerite and calcite) on regional and deposit scales (Vu et al., 1987).

Mineralization can occur as both refractory and non-refractory styles. Refractory mineralization is characterized by early-stage, disseminated sulphides of primarily pyrite, and/or arsenopyrite hosting significant gold content, which is overprinted by late-stage quartz veining with minor amounts of visible gold and accessory polymetallic sulphides. Examples of the refractory-style deposits include Obuasi (AngloGold), and Bogoso-Prostea (Golden Star Resources). Non-refractory ore is described as gold not hosted within sulphide minerals of either the early or late stage mineralization events. Examples of non refractory mineralization include Chirano (Kinross), and Ahafo (Newmont).

The gold mineralization that occurs in the oxidized zone is released from the hypogene orebody by physical disaggregation and chemical dissolution. Dissolution and reprecipitation of gold in the saprolite appears to take place in situ with little evidence of supergene enrichment. The mineralization can be concealed by metres of kaolinite-mica forest soils. The saprolite zone of leached rock can extend down 60-70 m (Bowell, 1992).

9. EXPLORATION

Exploration, consisting of line cutting, soil sampling, trenching, and auger drilling, was completed by Edgewater in 2012–2013 (McCracken, 2014). The principal targets were anomalies generated from the airborne geophysical surface. The work included both wide-spaced and detailed surveys. Results included anomalous gold in soils, trenches, and auger, which warrant additional follow-up work.

The procedures for each exploration method were summaries from the “Geologist’s Procedures Manual, Version 1.0, October 1, 2005” generated by Red Back Mining Inc. (Red Back, 2005).

9.1 Soil

All soil sampling was conducted in the presence of a geologist and was not carried out by technicians alone. Samples were collected from ±50 cm depth and were 2 to 3 kg of material. Duplicate samples were collected every 25 samples. To collect the duplicate, a larger hole had to be dug to collect 5 to 6 kg of sample and mixed thoroughly on a plastic sheet. The material was then coned and quartered into two samples. The results of the soil survey were disclosed in a previous technical report (McCracken et al., 2016). Table 9-1 summarizes the soil work completed.

Table 9-1: Soil Survey Summary

Prospect	Area Covered (km^2)	No. of Lines	Grid Spacing	Total Line Length (km)	No. of Samples	Type of Sample
Enkye	35.00	10	400 m x 50 m	60.0	986	Soil

The Enkye Grid area results of the soil samples defined a moderately anomalous zone 2.5 km long and averaging 1.25 km wide on trend from the Nyam Anomaly located 4 km south. The anomalous area is defined by a series of results greater than 50 ppb gold, with common results greater than 100 ppb gold, and isolated results greater than 500 ppb gold. The samples are representative of the material tested and generally no sample bias has been identified beyond the normal variability of the weathered and soil profile.

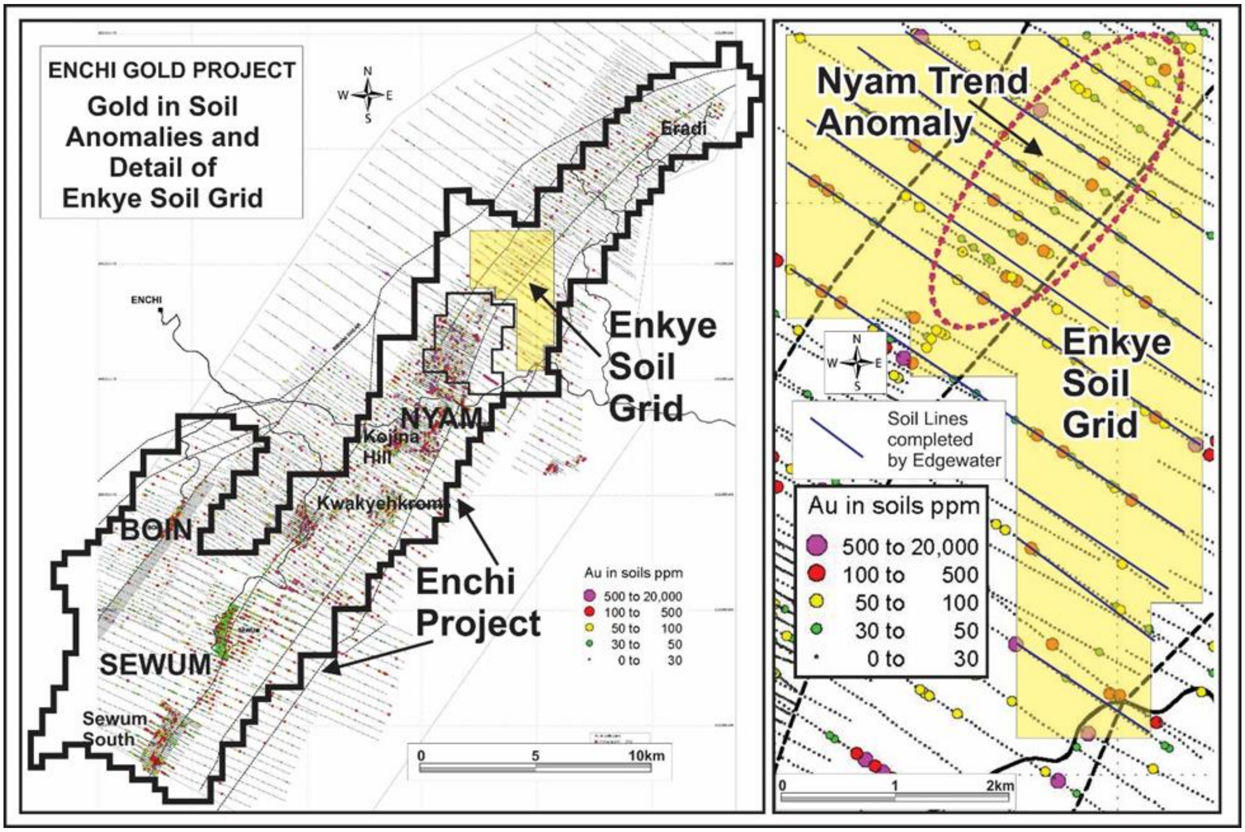


Figure 9-1: Significant Soil Results (Newcore 2020)

9.2 Trenching

The trenches were dug 80 cm to 100 cm wide with a maximum depth of 3.5 m.

The name of a trench consists of a two-letter prospect prefix, followed by "TR" and then a sequential numbering.

For consistency, trenches start at the western end (collar) and intervals are measured along the surface using slope distance, not horizontal distance. This allows correct plotting of the trench as a three-dimensional entity. To allow routine plotting of the trench as a drillhole, each segment must be considered to be a separate trench, with its own collar, and with its sample intervals starting at zero at its western end. The segments of a trench are identified by suffixes, for example CHTR798A, CHTR798B, from west to east.

Completed trenches are measured by marking out intervals along the surface starting from zero at the western end. Strings may be dropped down the sides of the trench to help the marking of the 1 or 2 m sampling intervals near the base of the trench.

The trenches are surveyed as a three-dimensional entity, and trench data is stored in the standard drilling tables of the database (collar, survey, assay, geology). The collar coordinates are determined by tape and compass, GPS, DGPS, or EDM survey depending on the stage of the Project.

The surface trace of the trench is surveyed from the collar to the end using tape, compass, and clinometer to produce a 'downhole' survey file. The intervals are chosen to match inflection points in the trench trace.

The 'from and to' measurements are slope measurements along the surface and are not corrected to horizontal distances.

The survey is usually done by a geologist and an assistant. The assistant holds a pole with a mark at the geologist's eye height. The geologist stands at the collar, the assistant at the first inflection point, and the geologist sights on the mark on the pole to record the inclination and azimuth.

Continuous channel samples are cut from the centre line of the floor of the trench. The trench must be checked by a geologist prior to sampling to ensure saprolite has been reached. The base of the trench must be cleaned by brushing or using a spade prior to sampling. Trenches are sampled by lithology, routinely using 2 m intervals with a minimum interval 0.5 m.

Duplicates were taken every 25 samples. This is a second channel cut either just above or just below the original sample.

One hundred and eighty (180) trenches totalling 17,019 m have been completed on the Project. Table 9-2 summarizes significant trench results. The criteria for reporting trench results were 4 m minimum length and minimum 0.3 g/t average grade over the interval. Figure 9-2 shows the location of the trenches at the Project site.

Table 9-2: Trench Summary

Trench	From (m)	To (m)	Length (m)	Au (g/t)
Boin				
KBTR002	58.3	63.4	5.1	1.43
KBTR003	2.0	6.0	4.0	0.42
KBTR004	0.0	6.0	6.0	0.68
KBTR004	108.0	112.0	4.0	0.74
KBTR004	140.0	144.0	4.0	0.44
KBTR007	12.0	16.0	4.0	1.29
KBTR008	38.0	58.0	20.0	0.31
KBTR009C	47.0	66.0	19.0	1.83
KBTR010	88.0	143.0	55.0	0.83
includes	*94.0*	*98.0*	*4.0*	*2.21*
includes	*124.0*	*127.0*	*3.0*	*3.63*

Trench	From (m)	To (m)	Length (m)	Au (g/t)
KBTR011	3.0	6.2	3.2	2.92
KBTR011	81.3	105.7	24.4	0.51
includes	*85.3*	*87.3*	*2.0*	*3.00*
KBTR012	162.4	167.7	5.3	0.48
KBTR014	216.5	221.0	4.5	0.87
KBTR016	11.6	13.6	2.0	1.39
KBTR018	107.0	113.0	6.0	0.54
KBTR023	66.4	93.0	26.6	1.53
includes	*66.4*	*70.4*	*4.0*	*3.49*
includes	*90.4*	*93.0*	*2.6*	*6.21*
KBTR024	95.8	103.8	8.0	1.40
KBTR024	142.5	144.5	2.0	1.07
KBTR024C	55.0	63.0	8.0	1.40
includes	*59.0*	*63.0*	*4.0*	*2.19*
KBTR029	2.0	41.0	39.0	0.78
includes	*16.0*	*19.2*	*3.2*	*2.03*
KBTR030	0.0	42.0	42.0	0.56
includes	*14.0*	*18.0*	*4.0*	*2.21*
KBTR031	26.0	30.0	4.0	1.32
KBTR032	26.0	34.0	8.0	1.30
KBTR033	28.0	45.0	17.0	0.63
KBTR034	14.0	36.0	22.0	1.21
includes	*22.0*	*26.0*	*4.0*	*4.01*
KBTR036	66.0	90.0	24.0	0.66
includes	*76.0*	*82.0*	*6.0*	*1.46*
KBTR037	34.0	42.0	8.0	2.00
includes	*36.0*	*40.0*	*4.0*	*3.47*
KBTR037	148.0	154.0	6.0	0.78
Sewum				
SWTR001B	0.0	104.0	104.0	0.68
includes	*20.0*	*32.0*	*12.0*	*1.71*
includes	*86.0*	*92.0*	*6.0*	*1.44*
SWTR003A	10.0	18.0	8.0	0.46
SWTR003B	36.0	42.0	6.0	0.42
SWTR004B	0.0	48.0	48.0	0.54
includes	*0.0*	*4.0*	*4.0*	*1.28*
SWTR005	10.0	20.0	10.0	0.47
SWTR005	42.0	58.0	16.0	0.66

Trench	From (m)	To (m)	Length (m)	Au (g/t)
SWTR006	18.0	64.0	46.0	0.46
includes	*44.0*	*48.0*	*4.0*	*1.14*
SWTR008	97.0	104.0	7.0	3.41
includes	*100.0*	*102.0*	*2.0*	*11.13*
SWTR009	176.0	190.0	14.0	1.18
includes	*178.0*	*186.0*	*8.0*	*1.78*
SWTR010	62.0	84.0	22.0	0.59
includes	*72.0*	*74.0*	*2.0*	*2.63*
SWTR010	100.0	126.0	26.0	0.58
includes	*110.0*	*112.0*	*2.0*	*3.04*
SWTR010	160.0	205.0	45.0	0.44
includes	*164.0*	*170.0*	*6.0*	*0.88*
SWTR010B	0.0	25.0	25.0	0.75
includes	*1.0*	*11.0*	*10.0*	*1.05*
SWTR010D	11.0	39.0	28.0	0.85
SWTR013A	80.0	149.0	69.0	1.27
includes	*86.0*	*102.0*	*16.0*	*1.99*
includes	*122.0*	*140.0*	*18.0*	*1.81*
SWTR013B	0.0	73.0	73.0	0.54
includes	*26.0*	*32.0*	*6.0*	*1.09*
includes	*50.0*	*60.0*	*10.0*	*0.98*
SWTR013D	14.0	30.0	16.0	0.49
SWTR015	28.0	78.0	50.0	0.71
includes	*32.0*	*38.0*	*6.0*	*1.91*
SWTR015	98.0	120.0	22.0	0.47
SWTR017	2.0	56.0	54.0	1.32
includes	*28.0*	*40.0*	*12.0*	*3.39*
SWTR017	67.0	126.0	59.0	0.90
includes	*75.0*	*89.0*	*14.0*	*1.70*
SWTR017B	4.0	14.0	10.0	0.38
SWTR017D	20.0	52.0	32.0	0.89
includes	*40.0*	*44.0*	*4.0*	*1.38*
SWTR019A	47.0	53.0	6.0	0.80
SWTR022	234.0	240.0	6.0	0.34
SWTR024A	0.0	11.0	11.0	0.41
SWTR024B	25.0	70.0	45.0	0.56
SWTR025A	32.0	48.0	16.0	0.98
includes	*44.0*	*47.0*	*3.0*	*3.48*

Trench	From (m)	To (m)	Length (m)	Au (g/t)
SWTR026C	52.0	74.0	22.0	0.79
includes	*56.0*	*62.0*	*6.0*	*1.21*
SWTR027	37.0	40.0	3.0	0.64
SWTR029	20.0	28.0	8.0	1.57
SWTR033	17.0	23.0	6.0	1.10
SWTR034	13.0	22.0	9.0	2.14
includes	*17.0*	*19.0*	*2.0*	*4.05*
SWTR038	21.0	64.0	43.0	0.80
includes	*40.0*	*45.0*	*5.0*	*1.57*
SWTR039B	6.0	17.0	11.0	0.81
SWTR045	0.0	18.0	18.0	1.83
includes	*7.0*	*17.0*	*10.0*	*2.43*
Nyam				
NBTR001	16.0	42.0	26.0	0.33
NBTR002	20.0	26.5	6.5	1.26
NBTR003	12.0	22.0	10.0	2.34
NBTR004	26.0	46.0	20.0	1.67
NBTR005	2.0	24.0	22.0	1.07
includes	*4.0*	*8.0*	*4.0*	*2.14*
NBTR006	0.0	10.0	10.0	0.98
NBTR006	34.0	42.0	8.0	0.63
NBTR007	8.0	20.0	12.0	1.20
NBTR008	14.0	24.0	10.0	1.12
NBTR009	4.0	12.0	8.0	1.89
NBTR009	22.0	38.0	16.0	0.77
NBTR011	14.0	30.0	16.0	0.73
NBTR012	12.0	18.0	6.0	2.33
NBTR015	72.0	76.0	4.0	9.05
includes	72.0	*74.0*	*2.0*	*18.08*
Kwakyekrom				
ADT001	84.0	92.0	8.0	1.31
includes	88.0	*92.0*	*4.0*	*1.99*
ADT002	100.0	104.0	4.0	0.37
ADT003	40.0	96.0	56.0	0.47
includes	*40.0*	*44.0*	*4.0*	*1.67*
includes	*76.0*	*80.0*	*4.0*	*1.08*
includes	*92.0*	*96.0*	*4.0*	*1.09*

Trench	From (m)	To (m)	Length (m)	Au (g/t)
ADT004	196.0	200.0	4.0	0.61
ADT005	89.0	104.0	15.0	0.95
includes	*90.0*	*95.0*	*5.0*	2.03
ADT009	40.0	44.0	4.0	1.26
ADT009	104.0	108.0	4.0	0.68
ADT010	116.0	120.0	4.0	0.42
ADT011	16.0	20.0	4.0	0.32
ADT014	56.0	60.0	4.0	0.30
ADT018	116.0	120.0	4.0	0.32
Achimfo				
ACHTR001A	134.0	154.0	20.0	0.97
ACHTR005	4.0	24.0	20.0	0.66

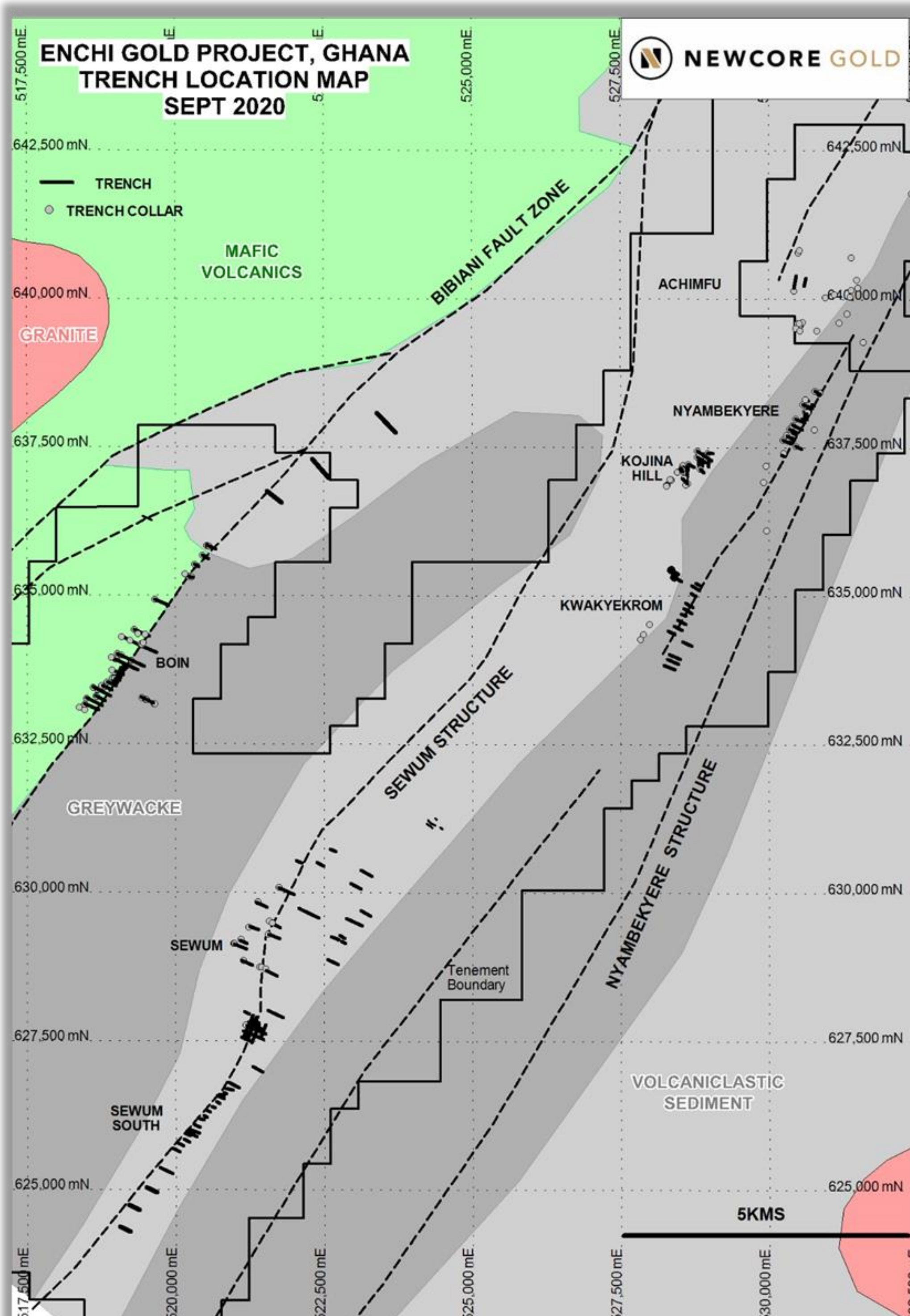


Figure 9-2: Enchi Trench Locations (Newcore, 2020)

9.3 Auger

Auger holes are vertical (-90º) and therefore no azimuth is required in the collar file. In the survey file, a -90º dip will be required at 0 m and at end of hole in the downhole survey file. The average sample depth was 3 m.

Sampling should be carried out on the basis of regolith geology. Lateritic soils, mottled clays, and saprolite were sampled separately. The A soil horizon was not sampled.

Duplicates were taken every 25 samples.

The results of the auger survey were disclosed in a previous technical report (McCracken et al., 2016). Table 9-3 summarizes the auger work completed.

Table 9-3: Auger Summary

Prospect	Area Covered (km^2)	No. of Samples	No. of Holes	Total Depth (m)	Significant Results (ppm)	Type of Sample
Achimfo	1.00	587	264	776.0	assays to 0.5g/t Au	Auger
Gyasikrom	1.55	1,051	278	949.0	assays to 0.5 g/t Au	Auger

Figure 9-3 is a map summarizing the significant auger results from around the Achimfo and Gyasikrom Prospects.

In the Achimfo and Gyasikrom area, individual auger sample results returned irregularly spaced values considered to be anomalous with greater than 500 ppb gold. No anomalous areas of significant size were outlined by the augers.

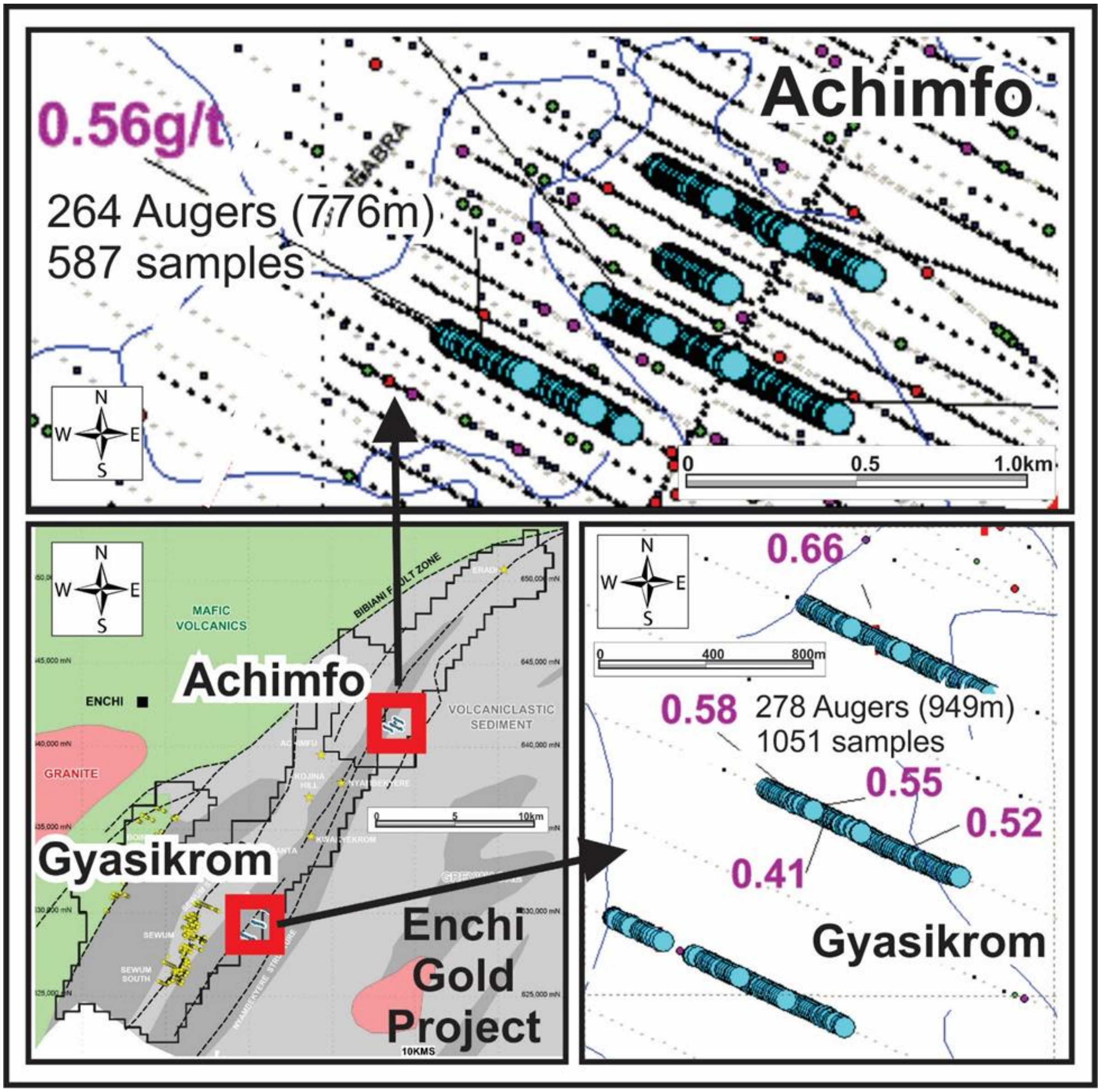


Figure 9-3: Significant Auger Results (Newcore 2020)

9.4 Exploration Results

9.4.1 Achimfo

The Achimfo Target is located in the central portion of the Project, approximately 1 km north of the Nyam Zone (Figure 7.1). The mineralized zone is located along the same structure, which hosts the Nyam mineralization. Previous soil sampling at Achimfo had generated moderate anomalous results and the airborne geophysical survey highlighted a linear radiometric anomaly. Work completed consisted of five trenches for 615.6 m and 264 holes for 776 m (587 samples). The trenches and augers were completed over an area of 1 km by 1 km. Significant results for trenches included 24 m @ 0.84 g/t Au and 14.0 m @ 0.49 g/t Au (Figure 9-1).

9.4.2 Boin NW

The Boin NW Target is located in the west-central portion of the Project, approximately 1 km west-northwest of the Boin deposit (Figure 7.2). The mineralized zone is located along a structure sub-parallel to the one hosting the Boin mineralization. Previous soil sampling at Boin NW had generated moderate anomalous results, and the airborne geophysical survey highlighted a linear radiometric anomaly. Work completed consisted of 11 trenches for 1,563.7 m. The trenches exposed silicified and brecciated volcanoclastic sediments and quartz veining with disseminated sulphides. The trenches were completed over an area 2 km long and 200 m wide. Weakly to moderately anomalous results were returned in most trenches with significant results including 10 m @ 1.64 g/t Au, and 8.0 m @ 0.49 g/t Au (Figure 9-1).

9.4.3 Sewum-Tokosea

The Sewum-Tokosea target is located in the south-central portion of the Project, approximately 500 to 1,500 m north of the Sewum deposit (Figure 7.2). The mineralized zone is located along a series of structures sub-parallel and along trend from the structure that hosts the Sewum mineralization. Previous soil sampling at Sewum-Tokosea had generated moderate anomalous results, and the airborne geophysical survey outlined a series of linear radiometric anomalies. Work completed consisted of seven trenches for 992 m. The trenches were completed over an area of 2 km by 200 m. The trenches exposed highly oxidized volcanoclastic sediments with brecciated quartz veins and foliated graphitic phyllites. Weakly anomalous results were returned in most trenches with significant results including 34 m @ 0.31 g/t Au (Figure 9-1).

9.4.4 Gyasikrom

The Gyasikrom target is located in the south-eastern portion of the Project, approximately 2 km east of the Sewum deposit (Figure 7.2). Limited previous work had been completed in this area; this work consisted of wide-spaced soil sampling that generated isolated weakly anomalous results. The airborne geophysical survey outlined a large 1 km by 3 km area interpreted to be underlain by intrusive rocks. Work completed consisted of three trenches for 540 m, and 278 auger holes for 949.1 m (1,051 samples) completed over an area of 4 km by 1 km. The trenches did not return anomalous values, and the auger returned isolated moderately anomalous values including 0.66 and 0.58 g/t Au.

9.4.5 Enkye

The Enkye target is located in the west-central portion of the Project, approximately 5 km northwest of the Nyam Zone (Figure 7.2). Limited previous work had been completed in this area; this work consisted of wide-spaced soil sampling that generated isolated anomalous results. The airborne geophysical survey outlined a large area with moderately complex radiometric and magnetic anomalies. The line cutting and soil sampling program was completed over the interpreted volcanic–sediment contact within the Enkye license on 400 m x 50 m spacing infilling the original 800 m x 50 m. Work completed consisted of 986 soil samples completed along 10 lines of length totalling 60.0 km over an area 7 km by 5 km. The soil sampling generated a limited number of weakly anomalous results as well as three spot highs including 354, 1,386, and 6,453 ppb Au.

10. DRILLING

The Project is considered an advanced project by definition of NI 43-101. As such, this technical report does not need to meet NI 43-101F1 Item 10(c). Location maps are provided in this Chapter to disclose the collar locations of the drillholes. Generalized cross sections of the drilling and geology for Sewum, Boin, Nyam (Nyamebekyere) and Kwakyekrom as disclosed in Chapter 7 Geological Setting and Mineralization.

Any drill results expressed in Chapter 10 are expressed in downhole length in metres. The orientation of the mineralization was not fully understood and the various dips of the holes result in unknown true thickness.

10.1 Pre-2011 Drilling

The 2005-06 Red Back Reverse Circulation (RC) and Rotary Air Blast (RAB) drilling program was undertaken on the Project from January 6, 2005 to December 4, 2006. A total of 153 RC holes were completed for a total of 17,120 m. A total of 320 RAB holes (including re-drills) were completed for a total of 12,443 m (Figure 10-1). The RC holes were completed with a 5.5-inch hole drilled using either a UDR KL900 or SCHRAMM. The RAB holes were completed with a 3.5-inch hole drilled using a UDR KL150RAB. Drilling was completed by GEODRILL Ghana Ltd. or African Mining Services of Ghana.

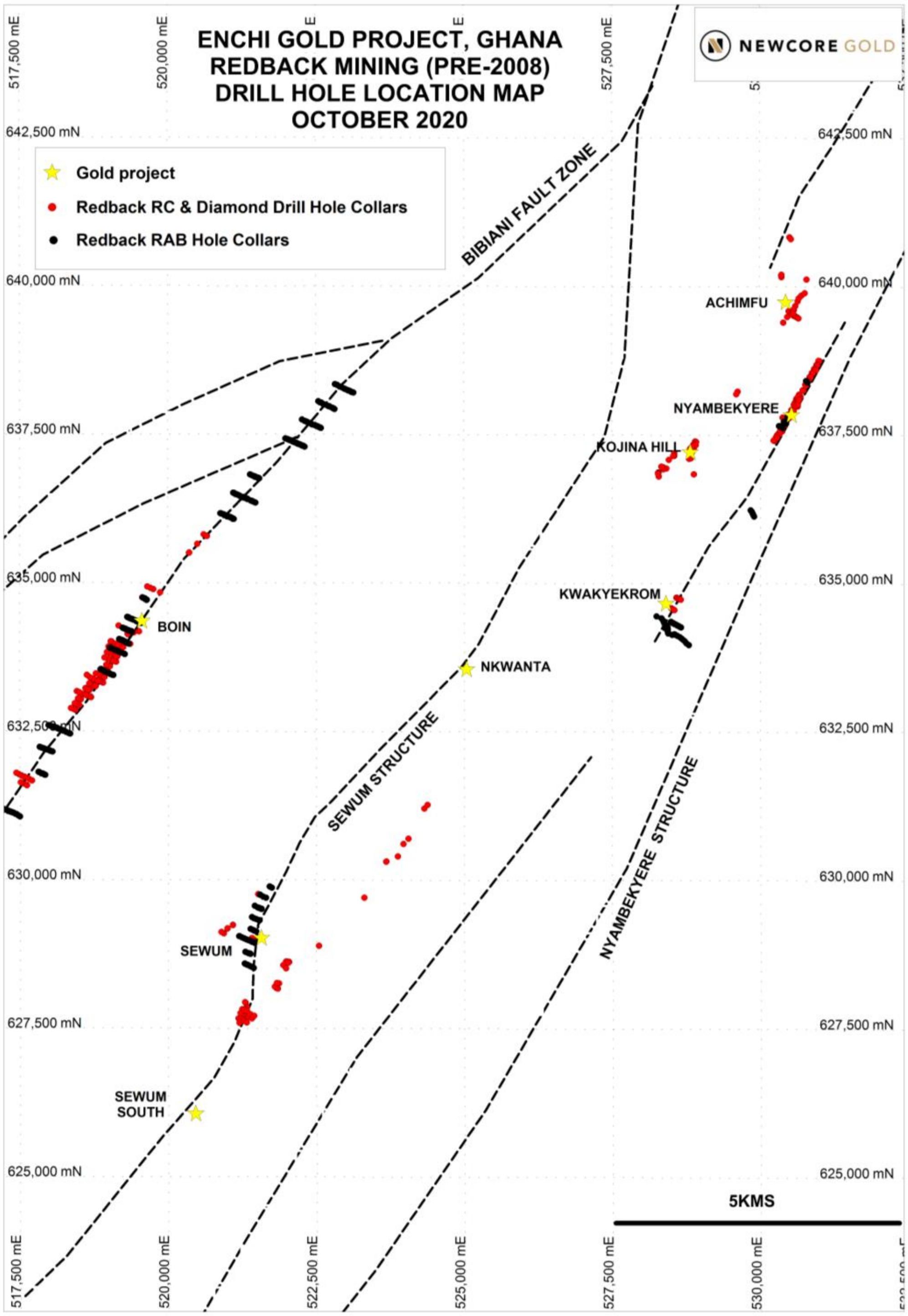


Figure 10-1: Drillhole Locations Pre-2011 (Newcore, 2020)

10.2 2011 Edgewater Drilling

The 2011 drilling program undertaken on the Project commenced in January 2011 and was completed in November 2011. A total of 180 diamond drillholes and 13 reverse circulation holes were completed for a total of 23,697 m (Figure 10-2).

10.2.1 Boin

Edgewater completed 62 diamond and 7 reverse circulation drillholes totalling 8,087 m at Boin (Figure 10-3). The aim of the program was to confirm results from the Red Back RC drilling, reduce the drill section spacing over the main part of the deposit from 100 to 50 m, and expand the gold resources by drill testing along strike and down dip.

Drilling at Boin (KBDDH001 to 033) was completed initially with an Energold Drilling Corp. (Energold) man-portable diamond rig operated by E Global Drilling Corp, a division of Energold based in London England, from February to July 2011. Most holes were completed using thin-walled HQTW (61.1 mm core diameter), reducing to NQTW (50.6 mm core diameter), if necessary. The second, deeper phase of drilling at Boin (KBDDH034 to 060) completed from September to October 2011 was done using a track mounted LF90 operated by Boart Longyear. To ensure optimum recoveries in the mineralization that was intensely weathered and hosted in clay, all holes in the second phase of drilling were cored from surface using PQ (85.0 mm core diameter, reducing to HQ (63.5 mm core diameter) when competent ground was reached.

Seven reverse circulation drillholes totalling 524 m were drilled to test mineralization along the Boin Shear Zone north of the main resource area. The first three holes had to be abandoned and re-drilled using a diamond drill owing to collapse of the collars due to thick surface clay.

The results of Edgewater's first phase of diamond drilling at Boin confirmed the continuity of the zone of gold mineralization defined by Red Back's reverse circulation drilling in 2005 and 2006. The second phase of diamond drilling, completed in 2011, extended the length of the main zone of mineralization to 1,800 m and proved continuity to a depth of 200 m down dip.

Depth of intense weathering is typically 20 or 30 m in the Enchi area. However, within the zone of mineralization at Boin, intense weathering and complete transformation of the host sediment to clay can reach 100 m depth in places. Low core recoveries were encountered in some drillholes as the mineralized quartz veins broke up into gravel-sized pieces that were spun ahead of the diamond bit, grinding up the host clay which was then flushed out with the drilling fluids. A number of methods were employed to overcome this problem, such as using thin walled drill rods and bits, larger diameter core, i.e. PQ, increasing the weight on the rod string while reducing the speed of rotation, and using bentonite and thick mixes of high-quality polymers.

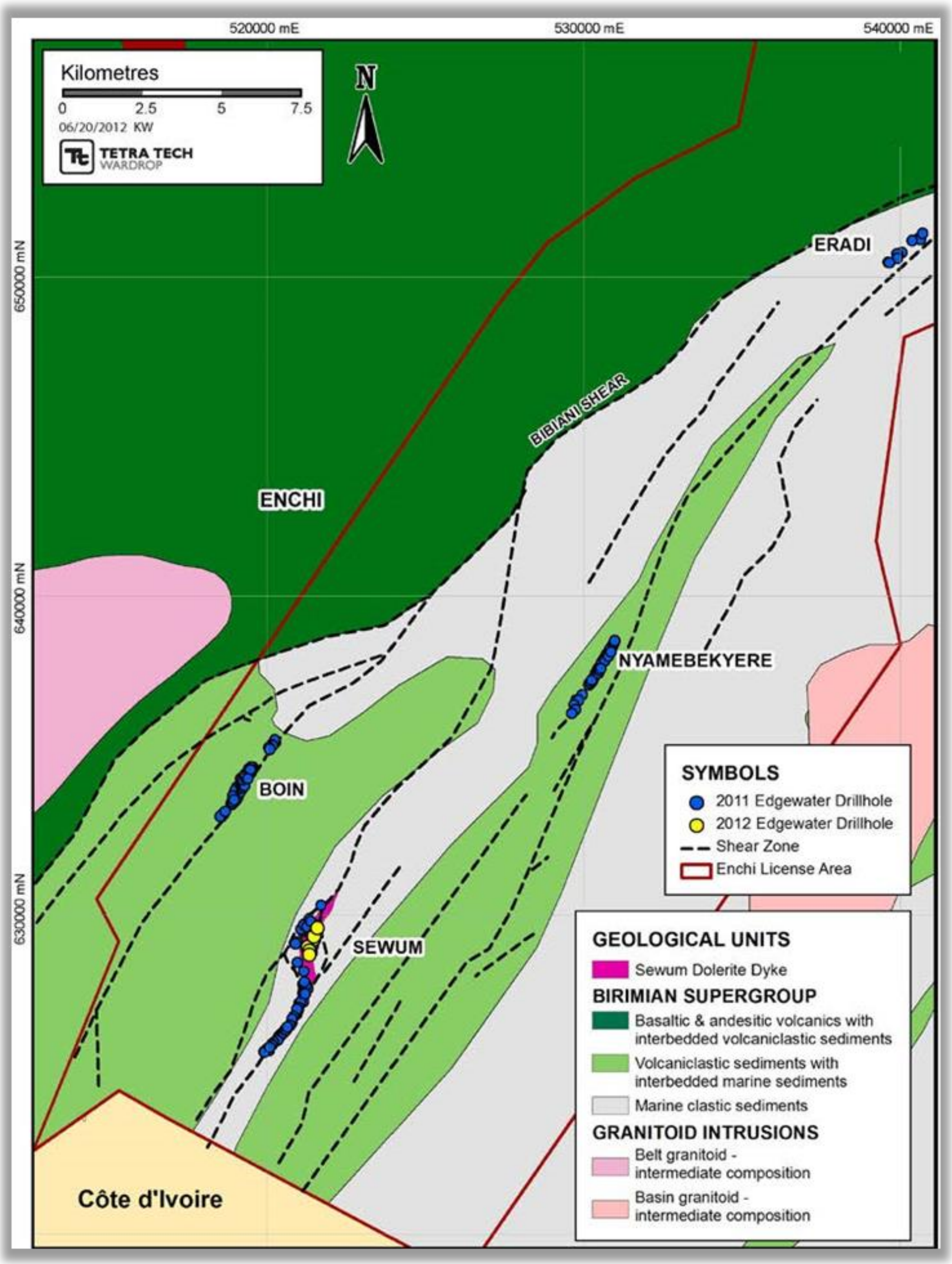


Figure 10-2: Drillhole Location 2011-2012 (McCracken, 2012)

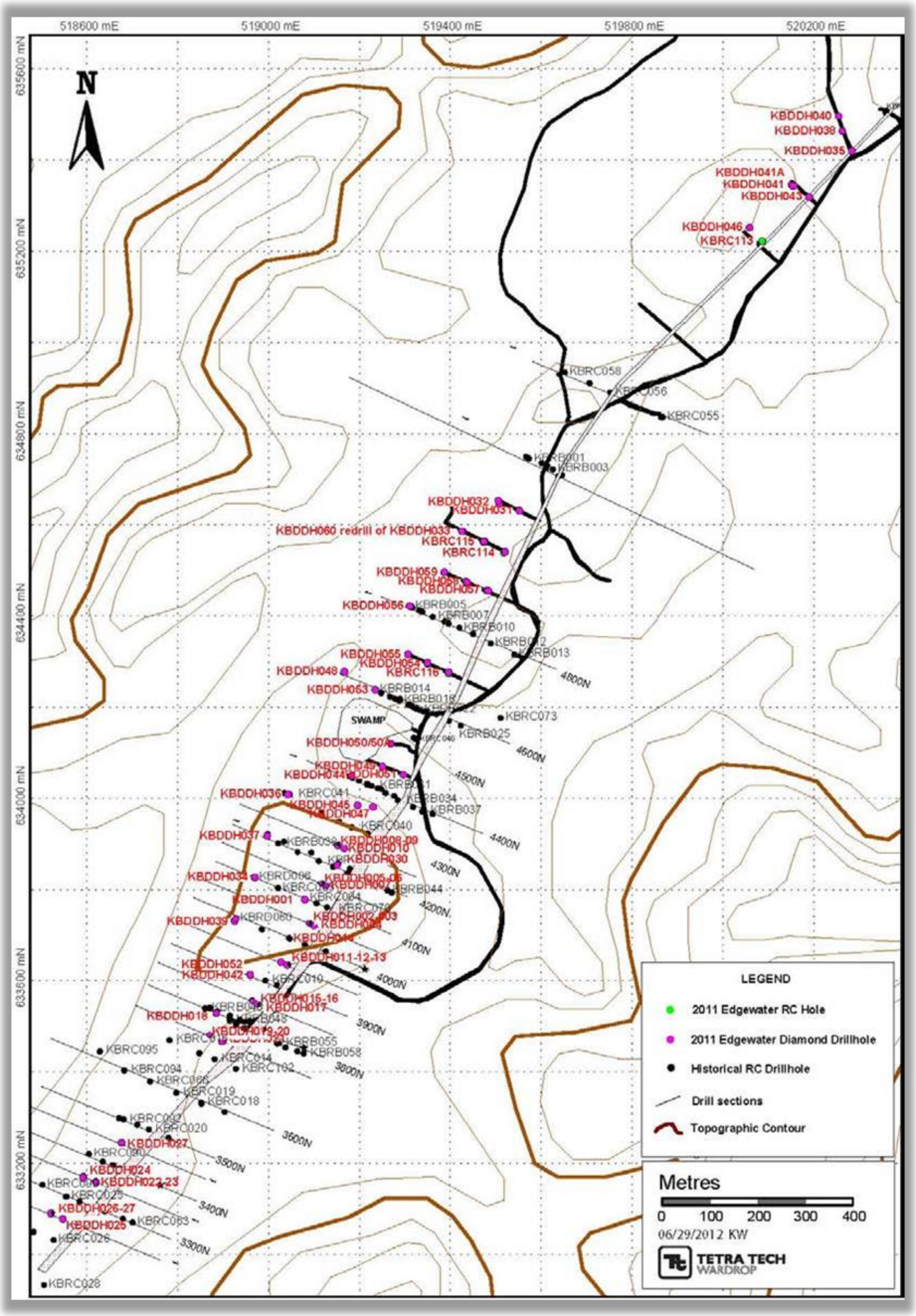


Figure 10-3: Boin Drillhole Location 2011 (McCracken, 2012)

10.2.2 Nyam

Edgewater completed two phases of drilling at Nyam. The first phase of drilling was conducted between January 24 and April 9, 2011 and consisted of 42 diamond holes totalling 3,969 m. The aim of the first program was to confirm results from Leo Shield's reverse circulation drilling (1996) by twinning selected holes and conducting infill drilling. The second drill program was conducted from October 31 to November 16, 2001 and consisted of five diamond holes totalling 1,164 m and six reverse circulation holes totalling 662 m (Figure 10-4). The second program was designed to test the down plunge extensions of mineralized material shoots identified in the first phase of drilling and to use reverse circulation drillholes to test the southern strike extension to the zone of mineralization.

The first phase of drilling at Nyam (NBDDH001 to 042) was completed with a track-mounted Longyear LM55 diamond drill. No reverse circulation rigs were available in Ghana at the time the decision to commence drilling was made. The longest hole drilled (NBDDH014) was only 129.8 m. HQ sized core was drilled from surface through the weathered zone until competent rock was encountered, the HQ was cased-off and the hole continued in NQ (47.6 mm core diameter). The deepest weathering was typically found on the tops of hills, up to 92 m deep (NBDDH014), whereas the weathering was much shallower in the valleys. Some core recovery issues were encountered in deeply weathered areas.

The second phase of diamond drilling (NBDDH043 to 047) was completed using a track-mounted Longyear LF90. To improve recoveries in the weathered zone and increase the sample size, coring was done in PQ from surface reducing to HQ once competent rock was reached.

The results of Edgewater's first phase of diamond drilling at Nyam confirmed the continuity of the zone of gold mineralization, reinforcing the results of the reverse circulation drilling completed by Leo Shield in 1996 and extending the length of known mineralization to more than 2 km.

Generally, the width and grade of the mineralization intersected in the near surface, clayey, weathered zone were better than in fresh rock, suggesting some supergene enrichment has occurred.

The reverse circulation drilling used a track-mounted Schramm HD 450 operated by Longyear. Reverse circulation drilling was designed to test the southern strike extension of the Nyam mineralized zone. All six reverse circulation holes drilled to test the southern extension to the Nyam mineralized zone intersected quartz veining and zones of bleaching caused by quartz – sericite – carbonate alteration; however, only two holes intersected anomalous gold results (NBRC001 and NBRC0060).

NBRC001 was drilled 400 m south-southwest of the southern most diamond holes NBDDH040 to 042. This large step-out along strike was due to the presence of a swamp. The reverse circulation rig had a 5.5-inch hammer and a 1,050 cfm at 350 psi compressor. The air was sufficient to keep samples dry to around 90 m. Most reverse circulation holes at Nyam were drilled at -50°.

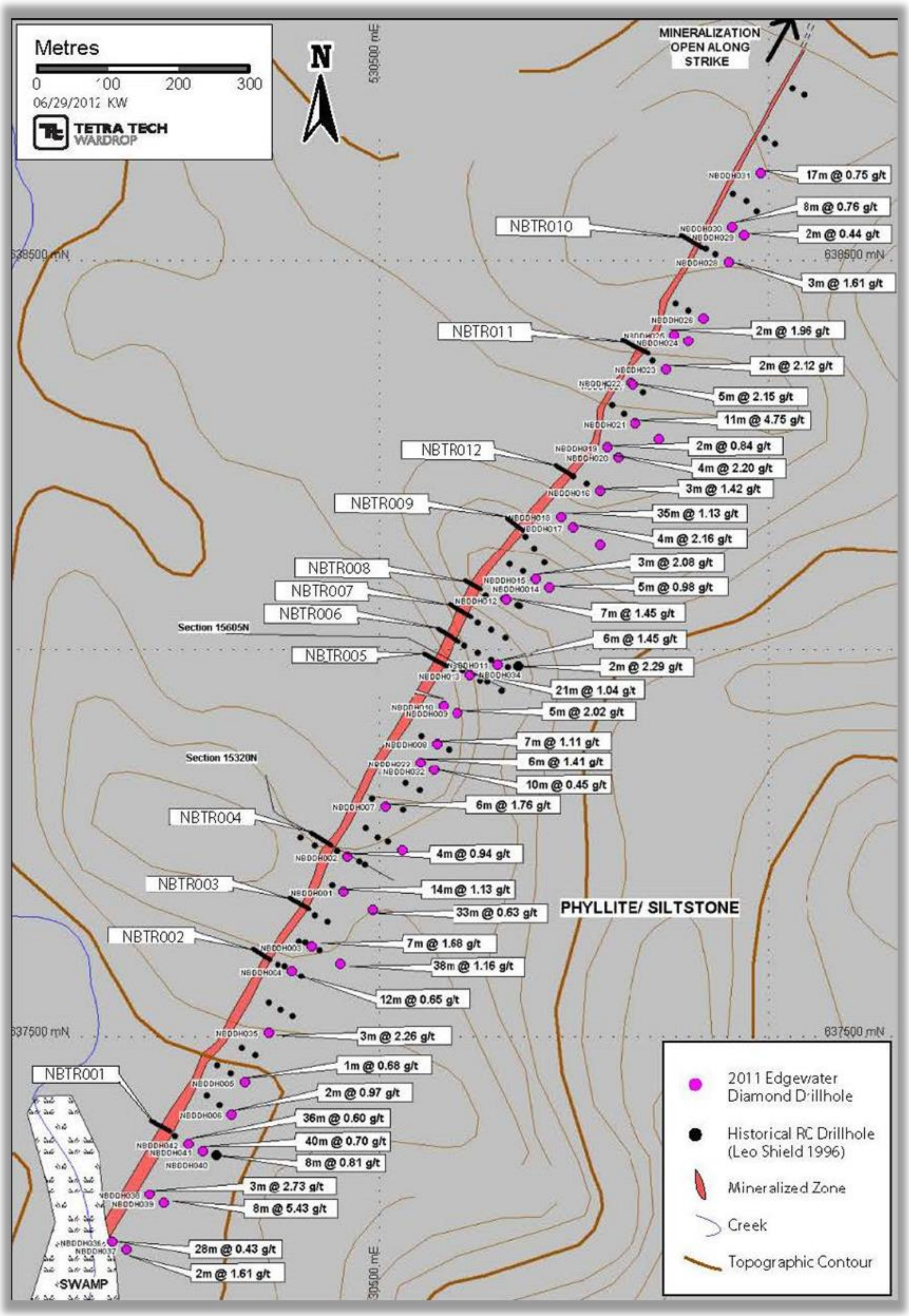


Figure 10-4: Nyam Drillhole Location 2011 (McCracken, 2012)

10.3 2012 Drilling

The 2012 Reverse Circulation (RC) drilling program undertaken on the Project commenced in March 2012 and was completed in April 2012. A total of 25 RC drillholes were completed for a total of 4,058 m (Figure 10-2).

10.3.1 Nyam

Edgewater completed an RC drilling program at Nyam from between April 13 and 24, 2012. The program consisted of nine reverse circulation holes totalling 1,524 m. Seven of the RC holes targeted the known resource area while two of the holes were exploration holes, well outside the resource area. The program used a truck-mounted LC 36 operated by Boart Longyear (Figure 10-5).

Figure 10-5: Reverse Circulation Drill (Newcore, 2012)

Reverse circulation drilling within the resource area was designed to test the eastern shear system and the northern strike extension of the Nyam mineralized zone. All seven reverse circulation holes drilled to test the Nyam mineralized zone intersected quartz veining and zones of bleaching caused by quartz – sericite – carbonate alteration with anomalous gold results. The intervals stated in the results table reflect downhole intervals and do not reflect true thickness of the mineralization.

Generally, the width and grade of the mineralization intersected in the near-surface, weathered zone were better than in fresh rock, suggesting some supergene enrichment has occurred.

10.3.2 Sewum

Edgewater completed 16 RC holes totalling 2,534 m at Sewum. The aim of the program was to better delineate the mineralization associated with the Ridge Top Shear Zone (RTSZ).

Drilling at Sewum was completed using a truck-mounted LC 36 operated by Boart Longyear.

No new drilling was conducted on the Sewum South, East Contact Zone (ECZ), West Contact Zone (WCZ), or Checker Board areas.

The 2012 RC holes drilled to target the RTSZ were drilled either vertically or dipping east. The RTSZ is hosted within the dolerite intrusive and is situated on top of the main Sewum Ridge. The zone consists of several stacked shallow dipping shears that average 20 m in thickness and has been traced in drilling for over 1 km along strike and remains along strike to the south toward the Checker Board Zone.

The northern strike extension of the RTSZ appears to be significantly thinner as evident in SWRC056 and SWRC057, with intervals approximately 7 m thick. The down dip extension of the RTSZ would be limited by the width of the dolerite intrusion..

10.4 2017-2018 Drilling

The 2017-2018 Reverse Circulation (RC) drilling program undertaken by Newcore on the Project commenced in November 2017 and was completed in February 2018. A total of 28 RC drillholes were completed for a total of 3,406 m (Figure 10-6).

10.4.1 Boin

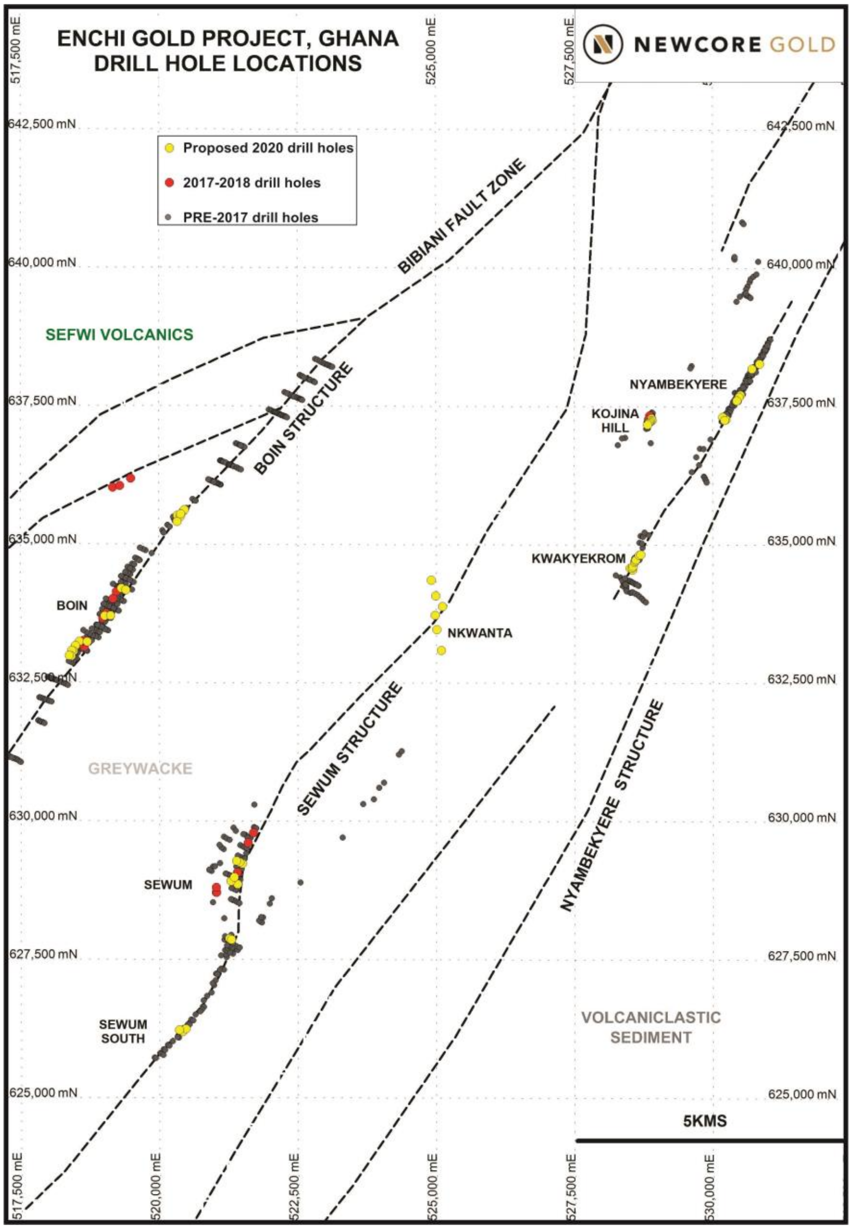


Figure 10-6: Drillhole Location 2017 (Newcore, 2020)

The 2017-2018 RC drill program was targeting infill and expansion drilling along the mineralized zone. The drilling extended the Boin Zone between 25 and 50 m to depth on several sections. The program consisted of 12 holes totalling 1,445 m, intersecting mineralization to a depth of 150 m below surface and successfully extended the main zone of continuous gold mineralization to approximately 2.5 km in length and 150 m depth.

The Boin NW target is located approximately 300 m west of the main Boin structure. Mineralization is interpreted to be associated with a splay off the main Boin structure. The 2017-2018 program consisted of three holes totalling 365 m, intersecting mineralization to a depth of 120 m below surface.

10.4.2 Sewum

In 2017, eleven RC holes totalling 1,396 m were drilled at Sewum to test extensions of known mineralized zones aimed at expanding resources. Wide intervals of gold mineralization were intersected in several holes extending the Ridge Top shear mineralization to depth toward the West Contact Zone and along strike to the south.

10.4.3 Kojina Hill

Drilling completed at Kojina Hill prior to 2017 included eight RC holes and one diamond drillhole. The drilling outlined a steeply dipping, northeast striking gold zone approximately 100 m long and up to 30 m wide. Results included near surface intercepts of: 37 m grading 1.34 g/t Au (11 to 48 m) and a second zone of 13 m grading 1.76 g/t Au (54 to 67 m), and 8 m grading 2.22 g/t Au (36 to 44 m) and a second zone of 17 m grading 0.94 g/t Au (50 to 67 m).

The 2017-2018 program consisted of two holes totalling 200 m, intersecting mineralization to a depth of 80 m below surface. During the 2017-2018 RC drilling program, drillhole KJRC010 intersected 9 m of 1.99 g/t Au (0.0 to 9.0 m) and 29.0 m grading 0.87 g/t Au (21 to 50 m) confirming continuity of the mineralized gold zone.

10.5 2020-2021 Drilling

The 2020-2021 Reverse Circulation (RC) drilling program undertaken by Newcore on the Project commenced on August 7, 2020 and drilling continues as of the effective date of this technical report. A total of 246 RC drillholes have been completed for a total of 38,641 m (Table 10-1) (Figure 10-7).

Table 10-1: 2020-2021 RC Summary

Deposit	Year	No. of Holes	Metre
Sewum	2020	25	4,269
	2021	6	1,194
Boin	2020	51	8,219
	2021	60	8,340
Nyam	2020	8	1,030
	2021	32	5,235
Kwakyekrom	2020	18	2,457
	2021	31	5,209
Kojina Hill	2020	9	1,918
	2021	5	670
Nkwanta	2020	1	100

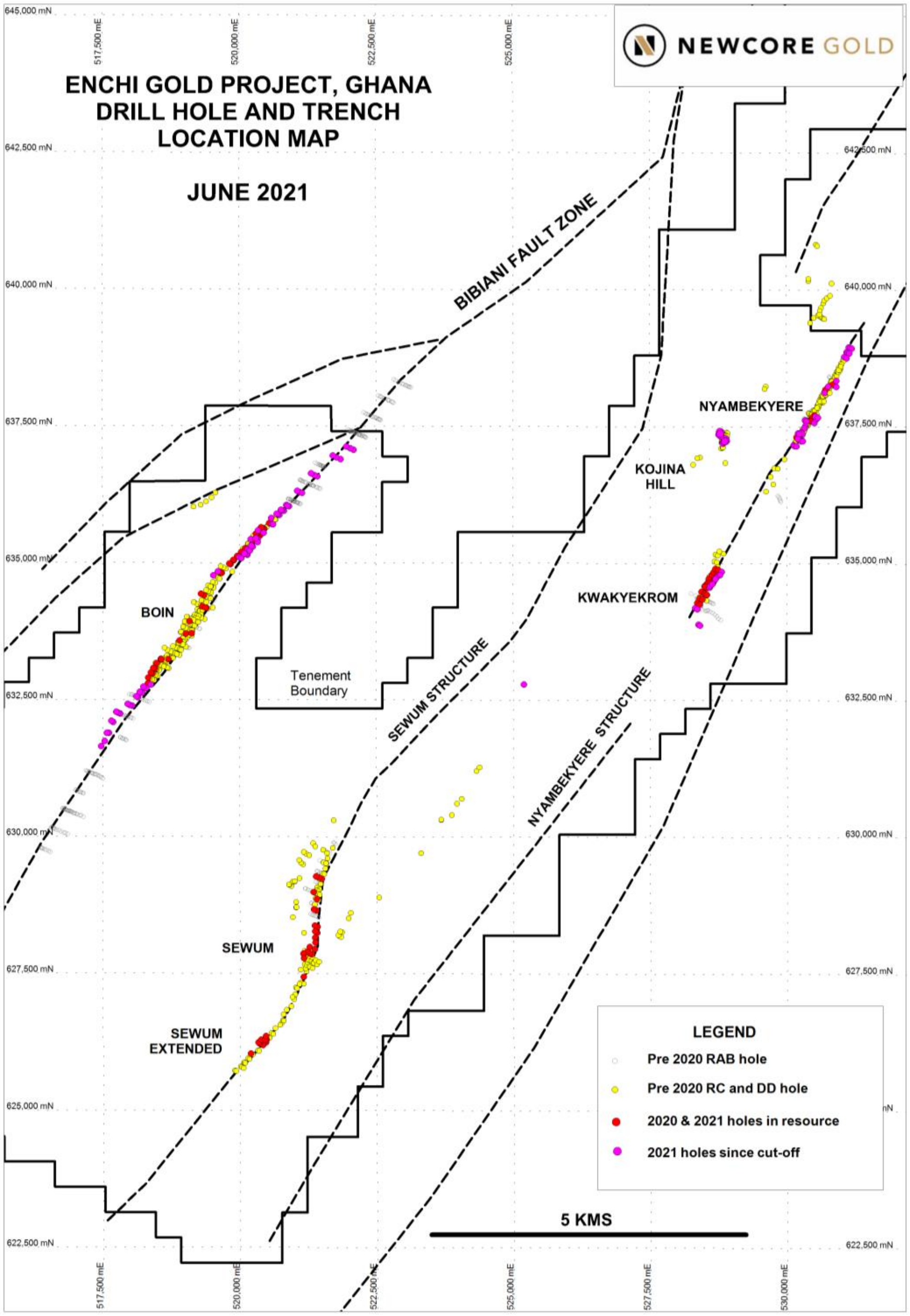


Figure 10-7: Drill Locations 2020-2021 (Newcore, 2021)

A total of 103 holes totalling 15,758 m were completed after the Mineral Resource Estimate cut-off dates in 2021. Table 10-2 lists the hole numbers drilled in the various deposit areas after the resource cut-off dates.

Table 10-2: RC Holes Drilled after Mineral Resource Cut-off Dates

Deposit	Series Start	Series Finish	No. of Holes	Metre
Sewum			0	0
Boin	KBRC185	KBRC244	60	8,340
Nyam	NBRC024	NBRC055	32	5,235
Kwakyekrom	KKRC055	KKRC065	11	2,183

A review of the locations of the holes drilled after the Mineral Resource cut-off dates shows all the holes are outside the immediate resource shells (Figure 10-7).

The QP is of the opinion that while the addition of these new holes would increase knowledge and confidence in the Project and may have a positive impact on future resource estimates, the holes do not have a material impact on the current study presented in this report.

10.5.1 Sewum

Sewum is located 15 km south of the town of Enchi and 5.0 km southeast of the other major gold resource identified at Boin. A local access road passes through the Sewum Zone with further access provided by drill roads extending an additional 300 m.

Gold mineralization at Sewum can be traced continuously for over 3.5 km and is contained within broad (up to 80 m thick) steep to moderate dipping, gold-bearing shear zones. The mineralized shears occur within a centrally located dolerite intrusion (Ridge Top shears) and at the contact zone of the intrusion with adjacent sedimentary rock units most notably at the Sewum West Contact Zone. Additional shears are interpreted to the east based on linear gold in soils anomalies and to the south where a strong gold in soil anomaly extends along the trend of the main Sewum shear for a further 3 km.

In 2020-2021, 31 RC holes totalling 5,463m were drilled at Sewum to test extensions of known mineralized zones aimed at expanding resources. Wide intervals of gold mineralization were intersected in several holes extending the Ridge Top shear mineralization to depth toward the West Contact Zone and along strike to the south towards Checker Board Hill (Figure 10-8).

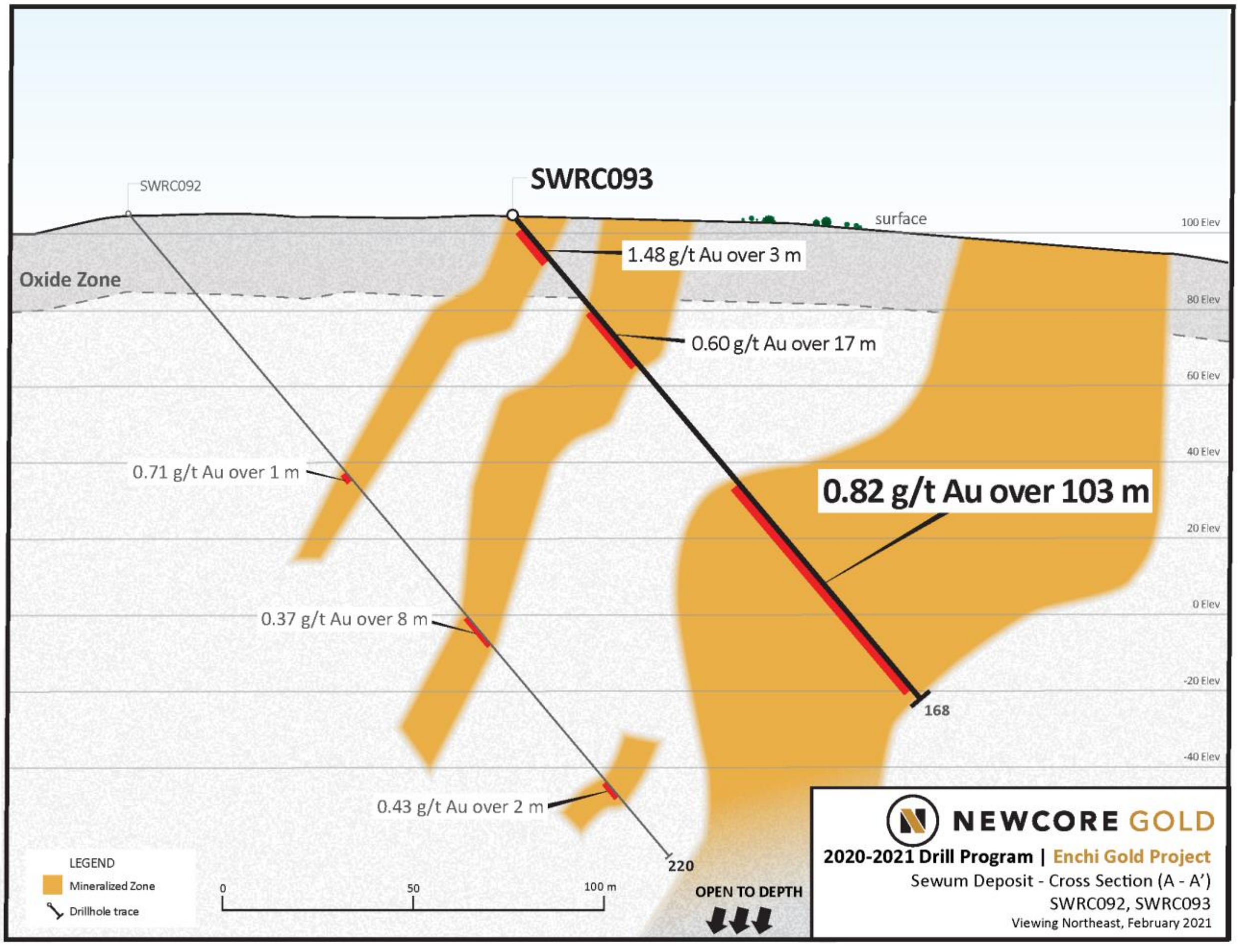


Figure 10-8: Sewum General Section (Newcore, 2021)

10.5.2 Boin

Boin is located with nearby roads and power and 10 km south of the town of Enchi. Further access is provided by a series of drill roads.

Boin is outlined on surface by a 5 km long and 0.5 to 1.0 km wide gold in soil anomaly. The response of the airborne electromagnetic along the structure is a highly conductive trend interpreted to be associated with the shallow dipping graphitic shear that occurs in the footwall to the gold mineralization. The geophysical anomaly extends for a further 2 km north and 5 km south beyond the currently drill tested section.

The 2020-2021 RC drill program was targeting infill and expansion drilling along the mineralized zone. The program consisted of 51 holes totalling 8,219 m that targeted large undrilled gaps with the resource areas and were included in the Mineral Resource Estimate. The program also included 60 holes totalling 8,340 m, which successfully extended the main zone of continuous gold mineralization to approximately 400 m to the north and 1 km to the south, with the drill tested strike extent at Boin now defined to over 4 km (Figure 10-9).

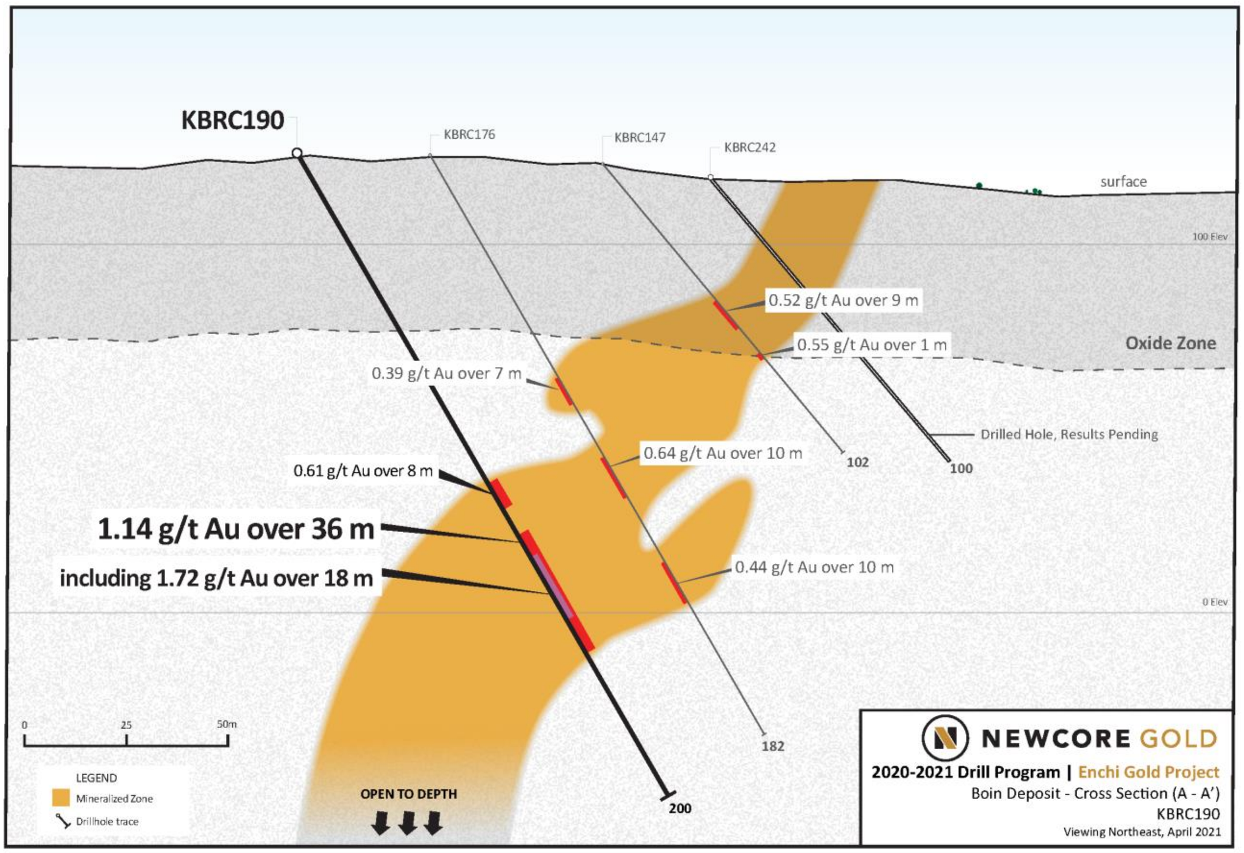


Figure 10-9: Boin General Section (Newcore, 2021)

10.5.3 Nyam

The 2020-2021 RC drill program was targeting down-dip, up-dip and expansion drilling along the mineralized zone. The program consisted of eight holes totalling 1,030 m that stepped out on the resource areas and were included in the Mineral Resource Estimate. The program also included 32 holes totalling 5,235 m, which were completed after the cut-off date and not included in the Mineral Resource Estimate. Drilling has successfully expanded the drill tested strike extent at Nyam to 1.8 km as well as below the existing resource model (Figure 10-10).

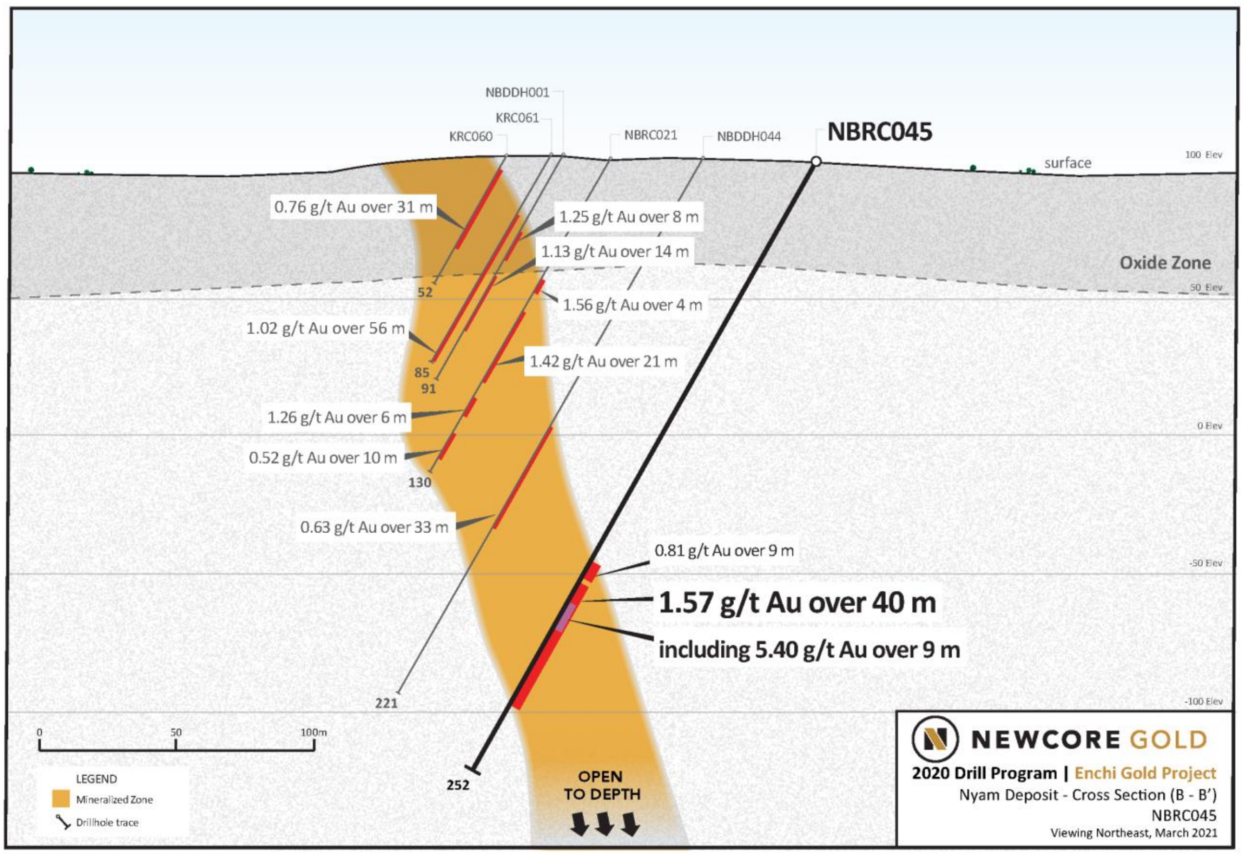


Figure 10-10: Nyam General Section (Newcore, 2021)

10.5.4 Kwakyekrom

The Kwakyekrom Zone is located 3 km south of the Nyam Zone and is interpreted to be related to the extension of the same structure that runs north-south through the 40-km property. Drilling has tested the Kwakyekrom Zone over a strike distance of 1.3 km and to a depth of approximately 150 m. The zone is hosted by altered phyllite, 700 to 800 m west of the interpreted position of the second order NS structure. The phyllite has been intersected by metre-scale dolerite dykes similar in composition to the larger intrusive bodies encountered at Sewum.

The 2020-2021 RC drill program was targeting down-dip, up-dip and expansion drilling along the mineralized zone. The program consisted of 38 holes totalling 5,483 m that stepped out on the mineralized areas and were included in the initial Mineral Resource Estimate. The program also included 11 holes totalling 2,183 m, which were completed after the cut-off date and not included in the Mineral Resource Estimate and successfully extended the zone of continuous gold mineralization (Figure 10-11).

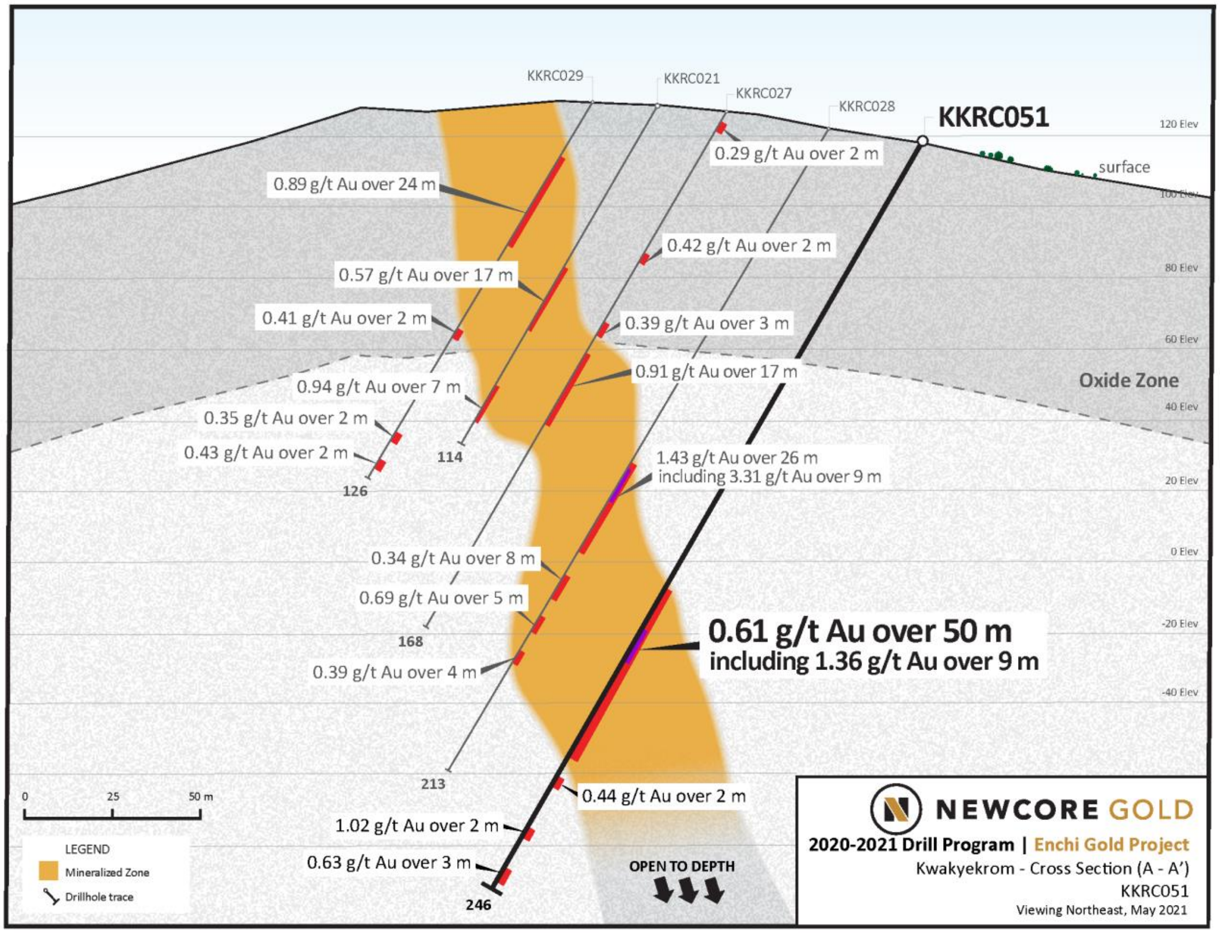


Figure 10-11: Kwakyekrom General Section (Newcore, 2021)

10.5.5 Kojina Hill

Kojina Hill is an advanced target on the Property that is outlined on surface by a two-kilometre-long by one-kilometre-wide gold-in-soil anomaly. It is related to a structure sub-parallel to the Nyam Shear Zone and is located approximately 1.5 northwest of the Nyam Zone. Kojina Hill is not included in the current Mineral Resource Estimate. Mineralization outlined to date is associated with a structurally complex zone with multiple sub-parallel structures.

Kojina Hill is accessed by drill roads leading off the main local access road situated 300 m to the east.

The 2020-2021 program consisted of 14 holes totalling 2,588 m, intersecting mineralization to a depth of approximately 100 m below surface (Figure 10-12).

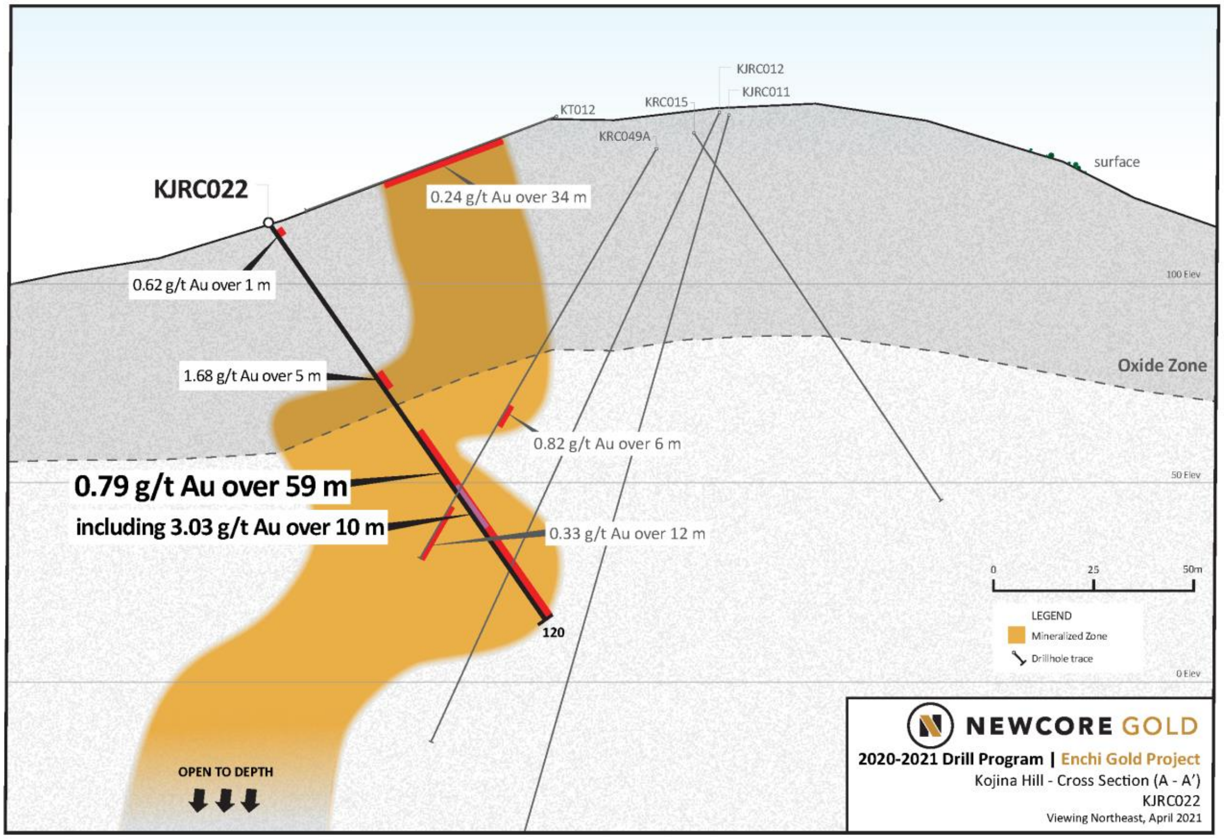


Figure 10-12: Kojina Hill General Section (Newcore, 2021)

10.5.6 Nkwanta

The 2020-2021 program consisted of one hole totalling 100 m testing a 1.5 km by 2.0 km soil anomaly. The hole intersected a sequence of variably altered intermediate volcanics, yet no anomalous gold values.

10.6 Drilling Procedures

10.6.1 Surveying

10.6.1.1 Collar Survey

Prior to drilling a hole, the proposed collar position is located by tape and compass survey from the nearest point whose coordinates are accurately known, or by handheld GPS. When there is a surveyor on site, the collar is located by electronic distance measurement (EDM) survey (Figure 10-13).

The inclination is set using a clinometer attached to the rod tracks while the mast is tilted and is checked and approved by the geologist prior to the start of drilling.

Comparison of the first downhole surveys with the nominal collar dip and azimuth should be checked by the geologist.

After drilling, all holes drilled were accurately surveyed for collar locations. The survey is by EDM, operated by qualified and experienced surveyors.

It was the responsibility of the geologist to enter all collar details from each day of drilling into the relevant computer file.

Figure 10-13: Collar Survey (McCracken, 2012)

10.6.1.2 Downhole Survey

A minimum of two surveys were completed on each hole. For holes less than 100 m, the survey was completed at half-depth and at the end of hole. Holes over 100 m were surveyed at 50 m intervals and at the end of the hole.

All surveys were completed during the drilling process.

10.6.2 Drilling

10.6.2.1 Rotary Air Blast

A downhole hammer was used to penetrate the ground with compressed air used to lubricate and cool the bit and carry drill cuttings to the surface. The drill cuttings were carried up the hole outside of the drill steel by compressed air. The drill cuttings were collected from the collar at 1 m intervals. The hole was flushed with compressed air after each 1 m interval to minimize downhole contamination.

10.6.2.2 Reverse Circulation

Only face-sampling hammers were used. A length of PVC casing was inserted into the top of the hole at a sufficient depth to create a secure seal at the top of the hole.

The hole was cleaned out at the end of each rod by blowing the hole in order to reduce any potential contamination (Figure 10-14).

The cyclone was cleaned after every hole to minimize contamination between holes.

Figure 10-14: Reverse Circulation Drill (Newcore, 2021)

10.6.2.3 Diamond Drill

Diamond drilling was completed using a wireline system, drilling NQ sized core. Holes were converted to HQ when poor ground was encountered. Core was retrieved at 3-m runs and the core was placed in the core box by the drillers. Geotechnicians monitored the drill rig operation 24 h/d. Forms were filled out during each shift recording the type of work completed and the time taken, such as rig shifts, pulling rods, changing the bit, drilling, breakdowns and downhole survey.

10.6.3 Logging Procedures

10.6.3.1 Chip Logging

RC and RAB drill logs were completed manually on standard logging forms. All necessary fields were completed, and a standard set of codes was documented.

The geological log recorded the percentage sample recovery for each 1 m interval estimated by visual comparison.

Samples were examined and logged on site and washed chips were glued to a chip board for future reference (Figure 10-15 and Figure 10-16). Chip boards are stored at the Newcore field office in Enchi.

Figure 10-15: RC Chip Logging (Newcore, 2017)

Figure 10-16: Chip Board Preparation (Newcore, 2017)

10.6.3.2 Diamond Drill Core Logging

Any full core boxes were collected at the end of each shift and taken to the Enchi site office. All drill core is systematically marked out, logged by geologists using geotechnical and geological logs, photographed, sawn with a core saw and sampled at 1 m intervals. Very soft, clayey core was halved using a broad bladed steel spatula (paint scraper). The following is a detailed description of the logging procedure that was carried out on the Project:

- Prior to logging, all drill core trays are laid out on logging shelves for geologists to check the mark ups made by the technicians and label the 1 m intervals on the core trays. All core is then photographed on a stand with a digital camera, a single box at a time together with a white board describing the date, borehole number, box number, and interval.
- After the entire core has been photographed, the core is laid out to be logged by geologists. Two logging forms are used: a descriptive geological form and a geotechnical form. The descriptive logs are used to record core recoveries, intensity of weathering, rock types, alteration styles and intensities, percentage and types of sulphides and other general information that cannot be recorded on the geotechnical logs. The geotechnical forms are mainly used to record detailed structural information (alpha – beta measurements) from the oriented drill core but also contain rock quality designation (RQD), fracture and joint data, core hardness, etc.
- Once completed, all the logging data is entered into a drillhole database.

10.6.4 Sampling Approach

10.6.4.1 Reverse Circulation Sampling

Sampling was done at the rig. The standard form and ticket books were completed by a technician and signed-off by the project geologist.

A 1-m sampling interval was used in all holes with the entire hole being sampled.

10.6.4.1.1 Dry Samples

Each sample was collected in a large plastic bag clamped tightly onto the base of the cyclone.

In 2012, each sample was weighed, then a split was taken for analysis using a 4-inch polyvinyl chloride (PVC) tube splitter (Figure 10-17). Care was taken to ensure the tube was speared down the centre of the bag to the base of the plastic. In 2017, samples were passed through the riffle and an approximate 3 kg split collected for submittal to the assay lab (Figure 10-18).

The sample split was placed in pre-numbered calico sample bags for dispatch to the geochemical laboratory. A record was made on the geological log and in the ticket books, at the drill site, of the sample identity numbers and corresponding intervals.

The splitter was thoroughly cleaned between samples.

Figure 10-17: Reverse Circulation Sampling Using Tube Splitter (Newcore, 2012)

Figure 10-18: Reverse Circulation Sampling Riffle Splitter (Newcore, 2017)

10.6.4.1.2 Wet Samples

Wet samples were collected in Fabrene bags and placed in the sun to allow the excess water to drain, and whenever possible, left to settle before subsequent sampling using the same procedure as with the dry samples (Figure 10-19).

The samples were transported each day to Newcore's core storage facility to await shipment to the analytical laboratory. The core storage facility maintained a night watchman on the Property to ensure samples and equipment were not tampered.

Figure 10-19: Drying Wet RC Samples (Newcore, 2017)

10.6.4.2 Diamond Drill Core Sampling

The following is the diamond drill core sampling procedure carried out on the Project:

- Once geologists have completed logging, the core is ready to be sampled and two aluminium tags are placed at each 1-m interval: one tag stating the depth down the hole and the other with sample number for that 1 m interval.
- All diamond drill core is sampled at 1-m intervals. When the core is too soft to be sawn using a diamond saw, the samplers use chisels or paint scrapers to halve the core. When cut using a saw, the core is cut in half following the markings made by the geotechnicians at the rig site.
- The half-core not sampled is retained in the core boxes and stored for future reference, petrological work, further geochemical sampling, specific gravity (SG), or other engineering tests.
- All sampling is monitored by geologists. The 1 m half-core samples are placed in a numbered clear plastic bag and the numbered aluminium tag for that 1-m interval placed in the bag with the sample. During sampling, forms are completed recording the hole number, sample interval, sample number, and core loss.

- Every 10th sample is a QA/QC sample. These samples are prepared prior to core sampling and are placed in the sample stream. Every 20th sample is a duplicate, and between the duplicates either a standard or a blank is used. Duplicate samples are prepared at the laboratory. The entire sample is crushed to -2 mm and two splits (more than 1.5 kg) are collected from the one sample using a Jones Splitter and the splits are then processed as separate samples.
- Once sampling of an entire drillhole is complete, the 1-m samples are placed into nylon rice sacks, 10 samples per sack. Each sack is tied and labelled with the company name and sample numbers the sack contains. All samples from a single drillhole are delivered to SGS laboratories as a single batch. If the samples are not sent the same day to the lab they are stored in a room inside the Enchi site office until ready to be transported.
- Each batch of samples is delivered using the company's vehicles and drivers directly from site to the SGS lab in Tarkwa, approximately a four-hour drive. Each batch of samples is submitted to SGS with a sample submission form outlining the method of preparation and analysis. Once the samples are delivered, the laboratory staff sign and date Edgewater's copy of the sample submission form acknowledging receipt of the samples.
- Each time a delivery is made to the laboratory, any pulps available are collected and brought back to the Enchi site office for storage. The pulps from selected drillholes are sent regularly for umpire assaying at different laboratories as a check on the results from SGS.
- Assay results are received both electronically and in hard copy form.

10.7 QP's Opinion

It is the QP's opinion that the drilling and logging procedures put in place by Newcore meet acceptable industry standards and that the information can be used for geological and resource modelling.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Rotary Air Blast

11.1.1 Sample Preparation

The following is summarized from the Red Back Geologist's Procedures Manual (Red Back, 2005).

Red Back drivers delivered the samples to the Intertek Tarkwa facility, which operates under the umbrella of Intertek/Genalysis Services Pty Ltd. The facility is certified with the following credentials: ISO 17025 and NATA certificate 3244.

All RAB chip samples were prepared at the Intertek laboratory in Tarkwa using preparation code PT01/SP02.

Below is a brief description of the sample preparations procedure:

- Samples are sorted and dried at 105°C;
- Once dried, the entire sample is crushed to a 75% passing at 2 mm;
- Sample is then split to get a sample up to 2 kg in weight for pulverizing;
- The entire split sample is then pulverized to allow a 95% passing of 75 µm;
- The pulp is split to 150 g for analysis.

At no time was an employee, officer, director, or associate of Newcore involved in the preparation of the samples.

11.1.2 Analytical Procedure

The following is summarized from the Red Back Geologist's Procedures Manual (Red Back, 2005).

A 30-g portion of pulverized sample is weighed, mixed with a fluxing reagent containing litharge (PbO) and then placed into a fusion furnace and fused at approximately 1,100°C. During this stage, the reduced lead collects the precious metals and forms a button. The sample is then removed from the furnace and cooled. The lead button is separated from the silicate slag.

The second stage of fire assay is called cupellation. During the cupellation process, at approximately 950°C the lead in the button oxidizes and is absorbed into the cupel leaving a precious metal bead known as a prill. The resultant prill is digested with Aqua Regia, first by adding nitric acid to dissolve the silver, and then hydrochloric acid. Gold content is determined by Atomic Absorption Spectrometer (AAS). The detection threshold limits are in the range of 0.01 ppm to 100 ppm.

At no time was an employee, officer, director, or associate of Newcore involved in the analysis of the samples.

11.2 Reverse Circulation

11.2.1 Sample Preparation

Each batch of samples is delivered using the Newcore vehicles and drivers directly from site to the Intertek laboratory in Tarkwa, approximately a 130 km from Enchi. Each batch of samples is submitted to Intertek with a sample submission form outlining the method of preparation and analysis. Once the samples are delivered, the laboratory staff sign and date Newcore's copy of the sample submission form acknowledging receipt of the samples.

The Intertek Tarkwa facility operates under the umbrella of Intertek/Genalysis Services Pty Ltd. and is independent of Newcore. The facility is certified with the following credentials: ISO 17025 and NATA certificate 3244.

Of the samples sent to the laboratories, 10% were either a duplicate sample, blank, or standard.

All RC chip samples were prepared at the Intertek laboratory in Tarkwa using preparation code PT01/SP02.

Below is a brief description of the sample preparations procedure.

- Samples are sorted and dried at 105°C;
- Once dried, the entire sample is crushed to a 75% passing at 2 mm;
- Sample is then split to get a sample up to 2 kg in weight for pulverizing;
- The entire split sample is then pulverized to allow a 95% passing of 75 µm;
- The pulp is split to 150 g for analysis.

At no time was an employee, officer, director, or associate of Newcore involved in the preparation of the samples.

11.2.2 Analytical Procedure

A 50-g portion of pulverized sample is weighed, mixed with a fluxing reagent containing litharge (PbO) and then placed into a fusion furnace and fused at approximately 1,100°C. During this stage, the reduced lead collects the precious metals and forms a button. The sample is then removed from the furnace and cooled. The lead button is separated from the silicate slag.

The second stage of fire assay is called cupellation. During the cupellation process, at approximately 950°C the lead in the button oxidizes and is absorbed into the cupel leaving a precious metal bead known as a prill. The resultant prill is digested with Aqua Regia, first by adding nitric acid to dissolve the silver, and then hydrochloric acid. Gold content is determined by AAS. The detection threshold limits are in the range of 0.01 ppm to 100 ppm.

At no time was an employee, officer, director, or associate of Newcore involved in the analysis of the samples.

11.3 Diamond Drill

11.3.1 Sample Preparation

Each batch of samples is delivered using company vehicles and drivers directly from site to the SGS laboratory in Tarkwa, approximately a 4-hour drive. Each batch of samples is submitted to SGS with a sample submission form outlining the method of preparation and analysis. Once the samples are delivered, the laboratory staff sign and date Edgewater's copy of the sample submission form acknowledging receipt of the samples. Of the samples sent to the laboratories, 10% were either a duplicate sample, blank, or standard.

All drill core samples were prepared at the SGS laboratory in Tarkwa using preparation code PRP89:

- Samples are sorted and dried;
- Once dried, less than 3.0 kg of the sample is crushed to a 75% passing at 2 mm;
- Sample is split to get a 250-g sample for pulverizing;
- 250 g of the crushed sample is then pulverized to allow an 85% passing of 75 µm.

Each time a delivery is made to the SGS laboratory, any pulps available are collected and brought back to the Enchi site office for storage. The pulps from selected drillholes are sent regularly for umpire assaying and are sent to Intertek laboratories to check for gold fire assay and ICP multi trace element analysis.

At no time was an employee, officer, director, or associate of Newcore involved in the preparation of the samples.

11.3.2 Analytical Procedure

Samples were assayed for gold using a 50-g charge fire assay code FAA505 using the following criterion:

- Gold 0.01 ppm – 100 ppm 50 g, fire assay, AAS finish.

A few selected holes were analyzed for trace elements using the ICP12B method, which is based on a two-acid digest (a combination consisting of nitric acid and hydrochloric acid). Once the material is digested, the solution is analyzed either by inductively coupled plasma-atomic emission spectroscopy (ICP-AES) or by inductively coupled plasma-mass spectrometry (ICP-MS) or by both. Two-acid digestion methods are the weakest of the digestions and silicate material is not affected, resulting in partial results for most elements (SGS, 2012).

The ICP12B method used is based on a combination of 2:1 nitric acid to hydrochloric acid and is recommended for samples with organic or high sulphide content.

SGS has geochemical accreditation that conforms with the requirements of CAN-P- 1579 and CAN-P-4E (International Organization for Standardization/International Electrotechnical Commission (ISO/IEC) 17025:2005).

At no time was an employee, officer, director, or associate of Newcore involved in the analysis of the samples.

11.4 Soil Sample Preparation and Analysis

Sample preparation and analyses were completed at the independent analytical facility of SGS in Tarkwa, Ghana.

Soil samples were dried and pulverized to 90% -75 micron.

The analysis was completed with a 50-g fire assay with aqua regia digest and di-isobutyl ketone (DIBK) extraction with AAS finish at a detection limit of 1 ppb.

11.5 Trench Sample Preparation and Analysis

Trench samples were dried and pulverized to 90% -75 micron.

The analysis was completed with a 50-g fire assay with aqua regia digest with AAS finish at a 10 ppb detection limit.

11.6 Auger Sample Preparation and Analysis

Sample preparation and analysis was completed at the independent analytical facility of SGS in Tarkwa, Ghana.

Auger samples were dried and pulverized to 90% -75 micron.

The analysis was completed with a 50-g fire assay with aqua regia digest and DIBK extraction with AAS finish at a detection limit of 10 ppb.

11.7 QA/QC

QA/QC programs were carried out during each drilling and trenching program. The QP generated and reviewed QA/QC charts for each program.

11.7.1 Soil

Blanks were inserted at a frequency of one every 50 samples with a minimum of one per batch. The material consisted of red-brown soils (2.5 kg) collected in Accra.

Commercial standards were inserted at a frequency of one every 50 samples with a minimum of one per batch.

The QP has not reviewed the QA/QC results for the soil survey program. The soil results are not material to the Mineral Resource Estimate.

11.7.2 Auger

Blanks were inserted at a frequency of one every 50 samples with a minimum of one per batch. The material consisted of red-brown soils (2.5 kg) collected in Accra.

Commercial standards were inserted at a frequency of one every 50 samples with a minimum of one per batch.

The QP has not reviewed the QA/QC results for the auger survey program. The auger results are not material to the Mineral Resource Estimate.

11.7.3 Trench

Blanks were inserted at a frequency of one every 50 samples with a minimum of one per batch. The material consisted of oxide rock fragments supplied from Accra.

Commercial standards were inserted at a frequency of one every 50 samples with a minimum of one per batch.

The results of the trench QA/QC samples were incorporated with the drill results and charted accordingly.

11.7.4 Pre-2011 Rotary Air Blast

Red Back inserted a blank and duplicate QA/QC samples into the sample stream. There was no set interval for insertion. Blanks were typically inserted approximately every 50th samples. Duplicates are inserted approximately every 20th sample. Reports indicate that standards are also inserted into the sample stream, yet there is no digital data available for the QP to review.

11.7.4.1 Blanks

A total of 69 blank samples were submitted to test for preparation contamination or carry over. A failure was considered to be ten times the detection limit. The high threshold for blanks is due to the drilling process, were the chips are transported to the outside of the drill steel and will likely result in downhole contamination. A single sample, or 1% of the samples failed. There is more variability in the results in the later part of the program.

11.7.4.2 Duplicates

A total of 180 duplicate samples were submitted. The control limit of ±20% is typically considered a failure by industry standards. A total of seven samples, or 4%, failed. If the samples below 0.1 g/t are not considered in the dataset, the failure rate increases to 8%.

This is a high failure rate. Due to the nature of gold mineralization, and the type of drilling, it is not uncommon to have a high failure rate.

11.7.5 Pre-2011 Reverse Circulation

Red Back inserted a blank and duplicate QA/QC samples into the sample stream. There was no set interval for insertion. Blanks were typically inserted approximately every 50th samples. Duplicates are inserted approximately every 20th sample. Reports indicate that standards are also inserted into the sample stream, yet there is no digital data available for the QP to review.

11.7.5.1 Blanks

A total of 198 blank samples were submitted to test for preparation contamination or carry over. A failure was considered to be ten times the detection limit. Twelve samples, or 2% of the samples failed. There are more variations in the results at the beginning of the program.

11.7.5.2 Duplicates

The control limit of ±20% is typically considered a failure by industry standards. A total of 21 samples, or 2%, failed. If the samples below 0.1 g/t are not considered in the dataset, the failure rate increases to 9%.

This is a high failure rate. Due to the nature of gold mineralization, and the type of drilling, it is not uncommon to have a high failure rate.

11.7.6 2012 Reverse Circulation

Every 10th sample submitted was a QA/QC sample. These samples were prepared prior to core sampling and were placed in the sample stream. There was a duplicate every 20th samples and between the duplicates was either a standard or a blank. Duplicate samples were prepared at the laboratory. The entire sample was crushed to -2 mm and two splits (less than 1.5 kg) were collected from the one sample using a Jones Splitter and were then processed as separate samples.

11.7.6.1 Blanks

A total of 108 blank samples were submitted to test for preparation contamination or carry over. A failure was considered to be three times the detection limit. A total of three samples, or 2% of the samples, failed. A single sample was removed from the blank dataset as it returned the same value as one of the standards.

11.7.6.2 Duplicates

A total of 211 course rejects of duplicate samples were submitted. The control limit of ±20% is typically considered a failure by industry standards. A total of 13 samples, or 6%, failed. If the samples below 0.1 g/t are not considered in the dataset, the failure rate increases to 50%.

This is a high failure rate. Due to the nature of gold mineralization, it is not uncommon to have a high failure rate. Efforts have been made to minimize the variation of the grades within the samples by using a larger sample size.

11.7.6.3 Standards

The charts generated for the Standard Reference Materials (SRM) have two components. The top portion of the chart displays the accuracy, which is how close the result comes to the expected value. The bottom portion of the chart displays the precision to the results, which is how repeatable the results are from one sample to the next.

Five separate SRM were used during the drilling program, with grades ranging from 0.36 g/t Au up to 6.75 g/t Au. The SRM GLG904-6, with an expected value of 0.36 g/t, had 20 samples submitted and returned an average of 0.361 g/t. The SRM G909-10, with an expected value of 0.52 g/t, had 20 samples submitted and returned an average of 0.508 g/t. There is a significant amount of variability in the results, specifically Samples 13 and 14 which are considerably lower than the rest of the dataset.

SRM G901-7, with an expected value of 1.52 g/t, had 22 samples submitted and returned an average of 1.507 g/t Au. The SRM G995-1, with an expected value of 2.75 g/t, had 22 samples submitted and returned an average grade of 2.736 g/t Au. The SRM G905-10, with an expected value of 6.75 g/t, had 16 samples submitted and returned an average grade of 6.89 g/t Au.

11.7.7 2017-2018 Reverse Circulation

Every 10th sample submitted was a QA/QC sample. These samples were prepared prior to core sampling and were placed in the sample stream. There was a duplicate every 20th sample and between the duplicates was either a standard or a blank. Duplicate samples were prepared at the laboratory. The entire sample was crushed to -2 mm and two splits (less than 1.5 kg) were collected from the one sample using a Jones Splitter and were then processed as separate samples.

11.7.7.1 Blanks

A total of 87 blank samples were submitted to test for preparation contamination or carry over. A failure was considered to be three times the detection limit. No failures were recorded.

11.7.7.2 Duplicates

A total of 211 course rejects of duplicate samples were submitted; 148 of the duplicates were above detection limit. The control limit of ±20% is typically considered a failure by industry standards. A total of six samples, or 3%, failed.

Efforts have been made to minimize the variation of the grades within the samples by using a larger sample size. Newcore should work with the laboratory to determine what preparation and analytical methodology should be used to minimize the variation of the assays.

11.7.7.3 Standards

Two separate SRM were used during the drilling program. Table 11-1 summarized the results of the SRMs.

Accuracy is measured by the difference between the average of all laboratory results (after the out-of-control results have been excluded) and the assigned value, as provided in the Certificate of Analysis that accompanies the SRM. The difference is expressed as a percentage of the assigned value.

Precision is a measure of how variable the laboratory analytical procedure is. This is expressed as a median moving range standard deviation (RSD) in percentage.

The laboratory aim should be to produce results that are both accurate and precise.

Table 11-1: 2017 – 2018 SRM Samples and Results

Standard	No. of Samples	Expected Value (g/t)	Accuracy (%)	Precision (%)	Outlier	Comments
OREAS 403	41	1.99	3.05	3.30	0	Industry standard
OREAS 452	42	1.03	3.35	3.47	0	Industry standard

The SRMs was prepared by Ore Research & Exploration (OREAS) of Australia - ISO 9001:2015 certified for Quality Management System including development, manufacturing, certification and supply of standards.

11.7.8 2020-2021 Reverse Circulation

All RC drill samples from the Enchi 2020-2021 RC Drilling Program were analysed at the independent analytical facility of Intertek laboratory located in Tarkwa, Ghana. Samples were analyzed with a 50-g fire assay for gold with an AAS finish.

Analytical QA/QC procedures include the systematic insertion of blanks, standards and duplicates into the sample stream.

11.7.8.1 Blanks

A total of 560 blanks were inserted into the Enchi 2020-2021 RC Drilling Program and were analyzed in each batch submitted to the independent analytical facility of Intertek laboratory located in Tarkwa, Ghana. Samples were analyzed with a 50-g fire assay for gold with an AAS finish. 76% of the results are below detection limit with the highest assay being 0.04 g/t.

11.7.8.2 Duplicates

A total of 1,112 pairs of duplicates were inserted into the Enchi 2020-2021 RC Drilling Program and were analyzed in each batch submitted to the independent analytical facility of Intertek laboratory located in Tarkwa, Ghana. Samples were analyzed with a 50-g fire assay for gold with an AAS finish. Of the duplicates, 658 were above detection limit. The control limit of ±20% is typically considered a failure by industry standards.

Three outliers in the dataset results in a R^2 of 0.81 for the data set. Removal of the three outliers from the data set resulted in a R^2 of 0.95, indicating a good correlation between original and duplicate.

11.7.8.3 Standards

A total of 560 SRM were inserted into the Enchi 2020-2021 RC drilling program and were analyzed in each batch submitted to the independent analytical facility of Intertek laboratory located in Tarkwa, Ghana. Samples were analyzed with a 50-g fire assay for gold with an AAS finish.

Accuracy is measured by the difference between the average of all laboratory results (after the out-of-control results have been excluded) and the assigned value, as provided in the Certificate of Analysis that accompanies the SRM. The difference is expressed as a percentage of the assigned value.

Precision is a measure of how variable the laboratory analytical procedure is. This is expressed as a median moving range standard deviation (RSD) in percentage.

The laboratory aim should be to produce results that are both accurate and precise.

Several SRM were used for the 2020-2021 program as summarized in Table 11-2.

Table 11-2: 2020-2021 SRM Samples and Results

Standard	No. of Samples	Expected Value (g/t)	Accuracy (%)	Precision (%)	Outlier	Comments
OREAS 152	24	0,016	-	-	-	Too few samples to chart
OREAS 251	102	0.504	3.97	4.77	0	Industry typical
OREAS 528	64	0.51	-2.27	5.62		Industry typical
OREAS 452	38	1.03	-1.36	4.04	1	Industry typical
OREAS 253	103	1.22	5.05	2.90	0	Good / Probable shift
OREAS 403	77	1.99	0.93	4.12	0	Industry typical
OREAS 224	80	2.15	3.50	3.90	0	Industry typical
OREAS 434	9	3.84	-	-	-	Too few samples to chart
OREAS 398	1	4.87	-	-	-	Too few samples to chart
OREAS 552	62	4.93	1.62	2.46	0	Good

The SRMs was prepared by Ore Research & Exploration (OREAS) of Australia - ISO 9001:2015 certified for Quality Management System including development, manufacturing, certification and supply of standards.

11.8 QP's Opinion

It is the QP's opinion that the sample preparation, analytical procedures, and security measures put in place for the trenches, reverse circulation, and diamond drill programs met acceptable industry standards at the time and that the information can be used for geological and mineral resource modelling.

The 2021 RC drillholes disclosed in Section 10.6 were not used in the 2021 mineral resource estimation and are not assessed as part of the QA\QC review.

12. DATA VERIFICATION

12.1 Site Inspection

12.1.1 2010

Mr. Todd McCracken, P. Geo., is a qualified person (QP) and co-author of this report. Mr. McCracken is a professional geologist with 30 years of experience in exploration, mine operations and consulting including several years working in shear hosted lode gold deposits and 20 years completing resource estimation and block models. Mr. McCracken visited the Property licence area and the Accra offices of Red Back between March 18 and 22, 2010 inclusive. Mr. McCracken inspected drill collar locations, trench locations, property geology, drill core and chip boards.

The QP is not treating the 2010 site inspection as a current inspection.

12.1.2 2011

Mr. McCracken visited the Property for four days from December 11 to 16, 2011. Mr. McCracken inspected drill collar locations, property geology and chip boards.

The QP is not treating the 2011 site inspection as a current inspection.

12.1.3 2014

Mr. McCracken visited the Property for three days from April 28 to May 1, 2014. Mr. McCracken inspected drill collar locations, property geology and chip boards.

The QP is not treating the 2014 site inspection as a current inspection.

12.1.4 2017

Mr. Joe Amanor, MAusIMM(CP), is a qualified person (QP) and co-author of this report. Mr. Amanor, is a professional geologist with over 40 years of experience in exploration and operations, including several years working in shear hosted lode gold deposits. Mr. Amanor visited the Property for two days from June 6 to 7, 2017, and two days from September 5 to 6, 2017. Drilling was confirmed through a site inspection, which included review of chip trays of representative material and original logs from the 2017-18 RC drilling, as well as field inspections of the locations for the drillholes which are clearly marked by concrete monuments.

The QP is not treating the 2017 site inspection as current.

12.1.5 2020 and 2021

Mr. Amanor visited the property from November 5 to 8, 2020 and most recently from June 2 to 6, 2021 (inclusive). For the 2021 visit, Mr. Amanor was accompanied by Daniel Adusei, the SEMS Exploration Services project geologist on site and Dan Wilson, Newcore's Country Manager. Drilling was confirmed through a site inspection, which included review of chip trays of representative material and original logs from the 2020-2021 RC drilling, as well as field inspections of the locations for the drillholes, which are clearly marked by concrete monuments. Locations were confirmed through a verification with adjacent drillholes and with GPS checks. Additionally, the field inspection did not reveal any active or recent artisanal mining affecting the areas of the mineral resources. Mr. Amanor was able to determine that there were no additional interferences, risks or cultural effects on the project through discussions with local hereditary chiefs and community leaders.

12.2 Drill Collar

12.2.1 Pre-2011 Drill Collar

A validation of the Red Back reverse circulation drill collars was conducted during the 2010 site visit. Seventeen collars representing 11% of the reverse circulation drilling completed by Red Back were surveyed using a handheld Garmin GPSMAP 60CSx. GPS readings were collected in UTM WGS 84 coordinate system. Table 12-1contains the results of the collar checks. Two of the Boin Zone collars have substantial errors, which are likely due to the collar number being incorrectly recorded in the field. The accepted error for the handheld GPS is typically 3 to 5 m of which all but one collar passed (SWRC005).

Table 12-1: Red Back Collar Validation

BHID	QP GPS			Red Back Log			Site	Delta Distance (m)	Comment
	Northing (m)	Easting (m)	Elevation (m)	Northing (m)	Easting (m)	Elevation (m)			
KBRC-003	633,907	519,032	194	633,905	519,034	194	Boin	2.6	-
KBRC-038	633,844	519,178	193	633,902	519,021	193	Boin	167.4	Incorrect GPS reading
KBRC-040	633,936	519,183	187	633,937	519,183	187	Boin	0.7	-
KBRC-044	634,032	519,218	163	634,033	519,217	163	Boin	1.1	-
KBRC-046	634,133	519,321	156	634,131	519,322	156	Boin	2.0	-
KBRC-054	634,748	519,566	176	634,749	519,567	176	Boin	1.5	-
KBRC-083	634,177	519,388	157	636,784	521,474	135	Boin	3,338.3	Incorrectly identified BH ID
KBRC-085	633,993	519,299	163	633,992	519,300	163	Boin	1.6	-
KBRC-086	634,021	519,248	167	634,021	519,248	167	Boin	0.4	-
KBRC-106	633,953	519,156	180	633,954	519,155	180	Boin	0.8	-
SWRC-001	629,009	520,941	138	629,097	520,937	138	Sewum	2.0	-
SWRC-005	629,175	521,003	143	629,182	520,997	143	Sewum	9.5	-
SWRC-007	627,932	521,296	128	627,931	521,298	128	Sewum	2.3	-
SWRC-009	627,669	521,179	154	627,668	521,179	154	Sewum	1.0	-
SWRC-014	627,707	521,237	158	627,705	521,232	158	Sewum	5.0	-
SWRC-015	627,735	521,261	159	627,735	521,260	159	Sewum	1.0	-
SWRC-021	627,809	521,294	160	627,806	521,298	160	Sewum	4.9	-

Validation of the RAB holes could not be completed during the site visit as there were no monuments marking the location of the RAB holes.

12.2.2 2011 Drill Collar

A validation of the Edgewater diamond and reverse circulation drill collars was conducted during the 2011 site visit. Twenty-one collars representing 2% of the drilling completed on the Project were surveyed using a handheld Garmin GPSMAP 60CSx. GPS readings were collected in Universal Transverse Mercator (UTM) World Geodetic System (WGS) 84 coordinate system (Figure 12-1). Table 12-1 contains the results of the collar checks.

The accepted error for the handheld GPS is typically 3 to 5 m in the X and Y coordinates. There appears to still be issues with the Z coordinates in the database relative to the GPS.

Figure 12-1: 2011 Collar Validation (McCracken, 2012)

Table 12-2: 2011 Collar Validation

BHID	QP GPS			Edgewater Exploration Log			Site
	Northing (m)	Easting (m)	Elevation (m)	Northing (m)	Easting (m)	Elevation (m)	
KBDDH013	633,636	519,045	159	633,636	519,045	158	Boin
KBDDH022	633,156	518,612	146	633,159	518,620	137	Boin
KBDDH024	633,164	518,598	152	633,167	518,597	139	Boin
KBDDH025	633,080	518,548	145	633,078	518,547	137	Boin
KBDDH026	633,093	518,527	156	633,090	518,522	139	Boin
KBRC114	634,542	519,523	130	634,541	519,522	140	Boin
KBRC115	634,563	519,477	136	634,561	519,476	140	Boin
NBDDH001	637,689	530,453	98	637,686	530,454	116	Nyam
NBDDH018	638,187	530,732	118	638,171	530,733	115	Nyam
NBDDH040	637,346	530,290	84	637,344	530,290	102	Nyam
NBDDH042	637,361	530,259	86	637,364	530,255	101	Nyam
NBDDH044	637,667	530,506	106	637,665	530,503	122	Nyam
NBDDH046	638,132	530,790	102	638,126	530,785	110	Nyam
SWDDH003	626,262	520,424	128	626,260	520,428	119	Sewum South
SWDDH004	626,248	520,450	128	626,252	520,455	122	Sewum South
SWDDH038	627,711	521,310	157	627,712	521,307	124	Sewum Checker
SWDDH039	627,756	521,217	130	627,757	521,213	100	Sewum Checker
SWDDH050	629,174	520,958	116	629,175	520,963	103	Sewum Road
SWDDH053	629,698	521,583	212	629,697	521,588	194	Sewum Ridge
SWDDH065	629,118	520,892	108	629,116	520,899	98	Sewum Road
SWDDH067	628,978	521,386	204	628,971	521,379	194	Sewum Ridge

12.2.3 2012 Drill Collar

A validation of the Newcore 2012 reverse circulation drill collars was conducted during the 2014 site visit. Twenty-one collars, representing 2% of the drilling completed on the Project, were surveyed using a handheld Garmin GPSMAP 62. GPS readings were collected in Universal Transverse Mercator (UTM) World Geodetic System (WGS) 84 coordinate system (Figure 12-2). Table 12-3 contains the results of the collar checks.

The accepted error for the handheld GPS is typically 3 to 5 m in the X and Y coordinates. Three collars are outside the customary error range. There appears to be issues with the Z coordinates in the database relative to the GPS. Although the Z coordinates from a handheld GPS tend to have a large error, the elevation of the drill collars did not match the topographic file provided.

Figure 12-2: 2012 Collar Validation (McCracken 2012)

Table 12-3: 2012 Collar Validation

BHID	QP GPS			Edgewater Exploration Log			X-Y Delta
	Northing (m)	Easting (m)	Elevation (m)	Northing (m)	Easting (m)	Elevation (m)	
SWRC041	521734	629877	228	521735	629875	196	2.46
SWRC042	521768	629966	182	521767	629966	190	1.19
SWRC043	521772	629965	181	521771	629960	190	4.56
SWRC044	521288	629011	231	521289	629015	193	3.88
SWRC045	521356	629091	192	521358	629089	194	2.60
SWRC046	521383	629176	231	521384	629179	196	3.52
SWRC047	521533	629530	235	521535	629527	198	3.75
SWRC048	521601	629724	234	521599	629723	197	2.32
SWRC049	521692	629788	194	521691	629786	195	2.23
SWRC050	521733	629877	228	521733	629880	195	2.90
SWRC051	521294	628589	226	521290	628592	200	4.79
SWRC052	521169	628244	129	521166	628242	138	3.67
SWRC053	521001	628916	99	521000	628918	114	2.83

BHID	QP GPS			Edgewater Exploration Log			X-Y Delta
	Northing (m)	Easting (m)	Elevation (m)	Northing (m)	Easting (m)	Elevation (m)	
SWRC055	522064	630264	120	522059	630268	131	6.56
SWRC056	521894	630258	121	521887	630262	128	8.12
SWRC057	521784	630273	121	521777	630279	127	9.23
NBRC009	530424	637523	91	530425	637524	100	1.27
NBRC012	530746	638054	133	530746	638054	145	0.45
NBRC013	530706	637956	101	530701	637956	113	5.39
NBRC014	530651	637904	89	530650	637904	101	1.18
NBRC015	530657	637946	90	530656	637947	103	1.26

12.2.4 2017 Drill Collar

A validation of the Newcore 2017 reverse circulation drill collars was conducted during the 2017 site visits. Locations were confirmed through a verification with adjacent drillholes. While no GPS readings were conducted, 24 of the 28 collars are in close proximity to previous drill collars, which remained clearly marked in the field. Likewise, the 2017 drill collars are clearly marked by concrete monuments. A total of 26 of the 28 drill collars were inspected during the site visits.

12.2.5 2020-2021 Drill Collar

A validation of the Newcore 2018 and 2020-2021 reverse circulation drill collars was conducted during the 2020 and 2021 site visits. Forty-two collars were surveyed using a handheld Garmin etrex-10. GPS readings were collected in Universal Transverse Mercator (UTM) World Geodetic System (WGS) 84 coordinate system (Figure 12-4). Table 12-4 contains the results of the collar checks.

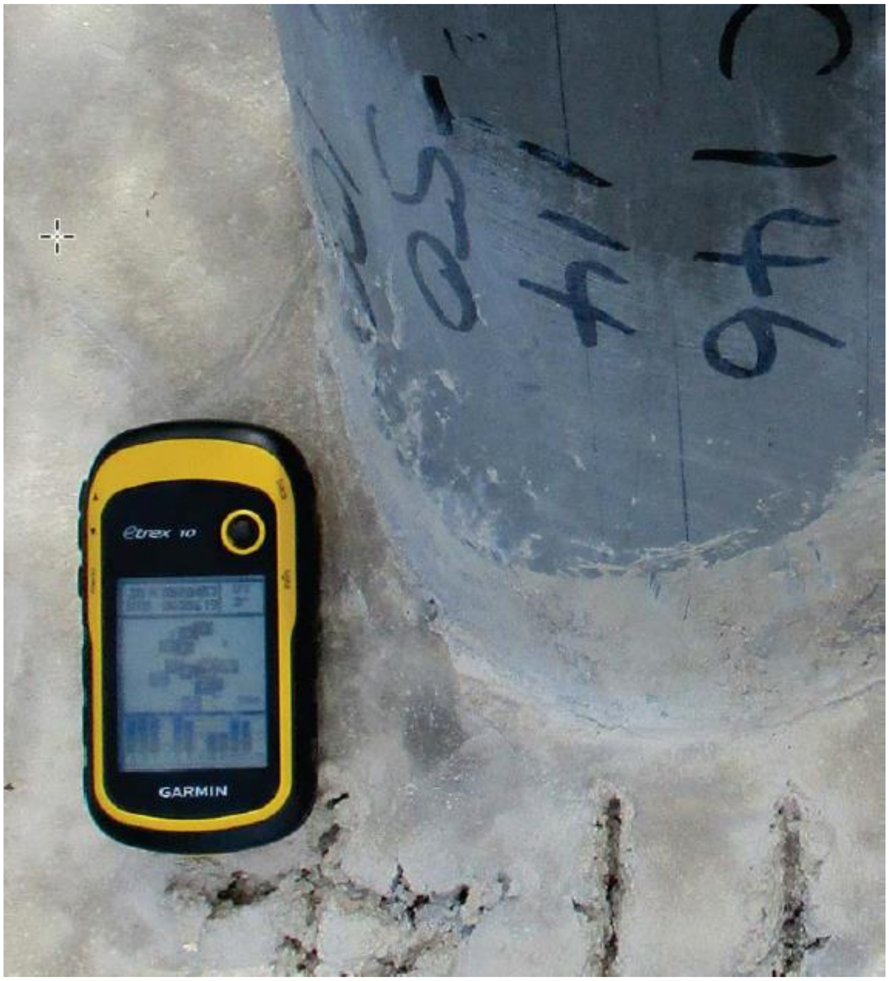


Figure 12-3: 2020-201 Collar Validation (Amanor 2021)

Table 12-4: 2020-2021 Collar Validation

BHID	QP GPS			Newcore Exploration Log			X-Y Delta
	Northing (m)	Easting (m)	Elevation (m)	Northing (m)	Easting (m)	Elevation (m)	
SWRC060	521498	629228	197	521496	629229	185	2.29
SWRC061	521444	629253	197	521442	629254	194	2.24
SWRC062	521408	629068	195	521407	629066	192	2.24
SWRC064	521382	628870	191	521381	628872	190	2.24
SWRC069	521442	629254	197	521442	629252	198	2.00
SWRC072	521409	628860	196	521408	628860	199	1.00
SWRC073	520470	626241	116	520472	626241	119	2.00

BHID	QP GPS			Newcore Exploration Log			X-Y Delta
	Northing (m)	Easting (m)	Elevation (m)	Northing (m)	Easting (m)	Elevation (m)	
SWRC074	520419	626190	99	520416	626193	105	4.24
SWRC082	520482	626289	116	520483	626291	115	2.24
SWRC083	520457	626302	119	520459	626301	124	2.24
SWRC084	520474	626356	114	520472	626357	114	2.24
KBRC128	633189	518547	136	633188	518545	137	2.24
KBRC130	633662	518982	157	633662	518981	155	1.00
KBRC131A	633664	518980	157	633662	518982	153	2.83
KBRC132	633747	519042	154	633748	519044	156	2.24
KBRC133	633748	519042	154	633747	519040	162	2.24
KBRC137	634187	519350	96	634187	519350	121	0.00
KBRC138	632974	518410	131	632972	518408	147	2.83
KBRC139	633237	518702	135	633234	518700	130	3.61
KBRC141	633706	519018	157	633704	519018	156	2.00
KBRC142	633242	518558	133	633239	518558	129	3.00
KBRC146	635618	520457	126	635619	520456	124	1.41
KBRC150A	635513	520335	124	635514	520338	122	3.16
KBRC151	634402	519349	131	634402	519349	131	0.00
KBRC155	633585	518898	160	633582	518896	155	3.61
KBRC156A	633165	518481	133	633164	518480	130	1.41
KBRC158	632991	518377	127	632992	518379	126	2.24
KBRC162	632790	518322	127	632793	518321	120	3.16
KBRC163	634810	519679	137	634811	519678	141	1.41
KBRC172	635250	520131	126	635252	520131	125	2.00
KKRC017A	528698	634829	129	528698	634833	127	4.00
KKRC018	528659	634785	132	528658	634784	129	1.41
KKRC028	528633	634667	122	528632	634668	126	1.41
KKRC036	528624	634797	128	528625	634796	130	1.41
KKRC048	528496	634456	101	528499	634456	106	3.00
KKRC049	528538	634433	109	528539	634435	109	2.24
KKRC053	528543	634592	117	528542	634595	117	3.16
KKRC054	528602	634630	124	528602	634632	128	2.00
NBRC017	530242	637277	75	530241	637278	69	1.41
NBRC019	530846	638272	81	530845	638277	81	5.10
NBRC020	530497	637713	83	530506	637714	86	9.06
NBRC021	530472	637678	93	530474	637679	93	2.24

12.3 Assay

12.3.1 Pre-2012 Assay

The QP, while employed by Tetra Tech, collected 21 pulps from the Boin reverse circulation drilling program and re-submitted the samples to the SGS laboratories in Ghana for check analysis. The small number of samples collected does not statistically represent the entire Boin Zone drilling data set yet should indicate the reproducibility of the material. Overall, the data collected has a strong correlation to the original samples submitted by Red Back (R=0.84) (Figure 12-4). The correlation does appear to break down with samples with assay values above the 2.5 g/t range. This is likely due to the analytical process of flame atomic absorption (FAA), which is fire-assay with an atomic absorption finish. FAA is accurate at the lower detection limits yet tends to have poor precision range at the higher detection limits.

The QP did not collect any check samples from the Redback RAB or RC drilling.

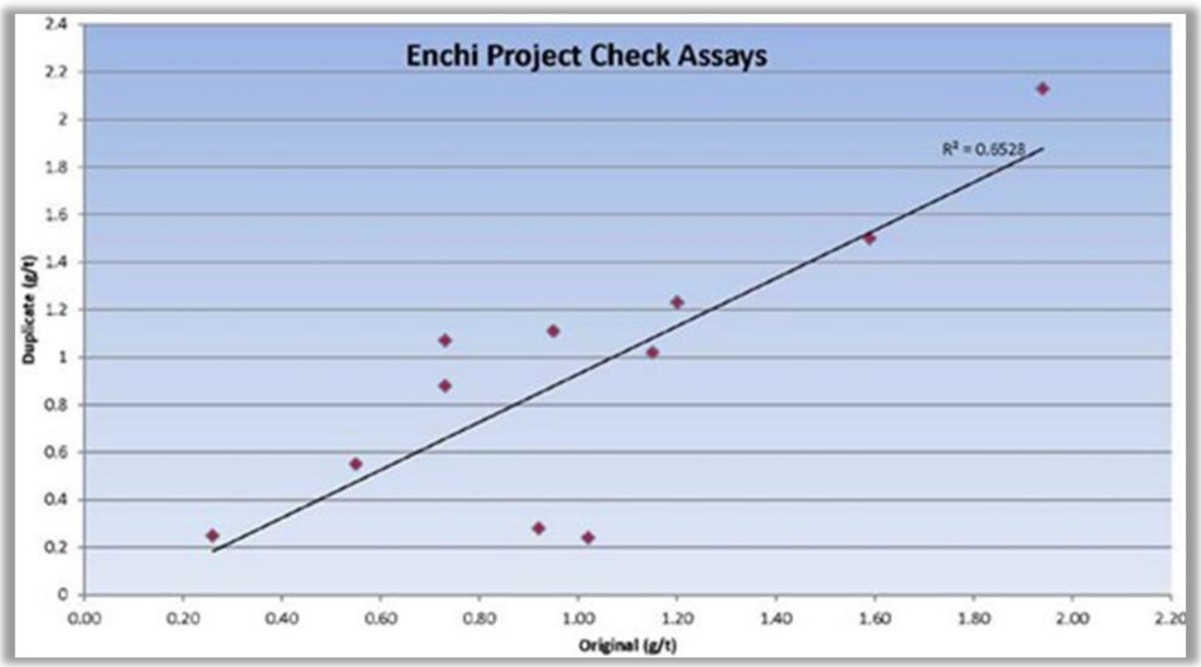


Figure 12-4: 2010 Check Samples (McCracken, 2011)

12.3.2 2012 Assay

The QP, while employed by Tetra Tech, collected 38 pulps from the drilling program and re-submitted the samples to ALS laboratories in Sudbury for check analysis. The QP used the same analytical procedure as Newcore in order to minimize the potential variance from different analytical methods.

Overall, the data collected had a strong correlation to the original samples submitted by Newcore (Table 12-5) with a R2 of 0.92 (Figure 12-5).

Table 12-5: 2012 Check Analysis

Borehole ID	Original Sample ID	Original Gold (g/t)	Check Sample D	Check Gold (g/t)	Log-QC (% passing 75 µm)
NBRC009	1046734	0.51	120087500920	0.005	75.7
	1046763	0.41	120087501210	0.51	88.9
	1046778	1.23	120087501360	1.22	94.9
NBRC010	1046910	0.50	120087600680	0.64	-
	1046935	1.35	120087600930	1.27	56.3
	1046940	0.37	120087600980	0.35	80.6
NBRC011	1047110	7.38	120088001020	6.75	-
	1047115	0.33	120088001070	0.33	92.2
	1047135	0.42	120088001270	0.48	94.9
NBRC012	1047297	1.39	120088501220	1.19	87.6
	1047310	0.48	120088501350	0.48	-
	1047352	0.15	120088501770	0.20	82.8
NBRC013	1047538	0.22	120089401180	0.32	-
	1047604	0.44	120089401840	0.35	56.0
NBRC014	1047782	1.81	120098401400	1.74	92.0
	1047786	0.06	120098401440	0.04	90.0
	1047837	0.80	120098401950	0.78	84.2
NBRC015	1047978	0.78	120101101140	0.98	95.1
	1047992	4.36	120101101280	4.11	87.8
SWRC041	1042704	1.72	120061400330	1.43	89.9
	1042828	0.65	120061401570	0.29	75.5
	1042836	0.46	120061401650	0.51	92.8
SWRC043	1043104	0.48	120065001000	0.56	82.4
	1043108	0.87	120065001040	0.80	88.9
	1043114	0.32	120065001100	0.22	83.6

Borehole ID	Original Sample ID	Original Gold (g/t)	Check Sample D	Check Gold (g/t)	Log-QC (% passing 75 µm)
SWRC044	1043337	0.52	120065102220	0.58	90.1
	1043345	0.71	120065102300	1.28	80.6
	1043348	0.82	120065102330	0.87	80.4
SWRC047	1043888	0.71	120068300710	2.70	-
SWRC048	1044108	1.15	120068600680	1.10	94.9
	1044223	0.64	120068601830	0.81	88.4
	1044265	0.41	120068602250	0.23	89.2
SWRC049	1044408	0.58	120069401420	0.80	94.6
	1044411	2.00	120069401450	2.36	78.0
	1044415	1.57	120069401490	1.75	84.7
SWRC050	1044515	0.33	120069500690	0.38	93.1
	1044520	4.38	120069500740	4.71	-
	1045046	1.86	120089401650	1.90	50.5

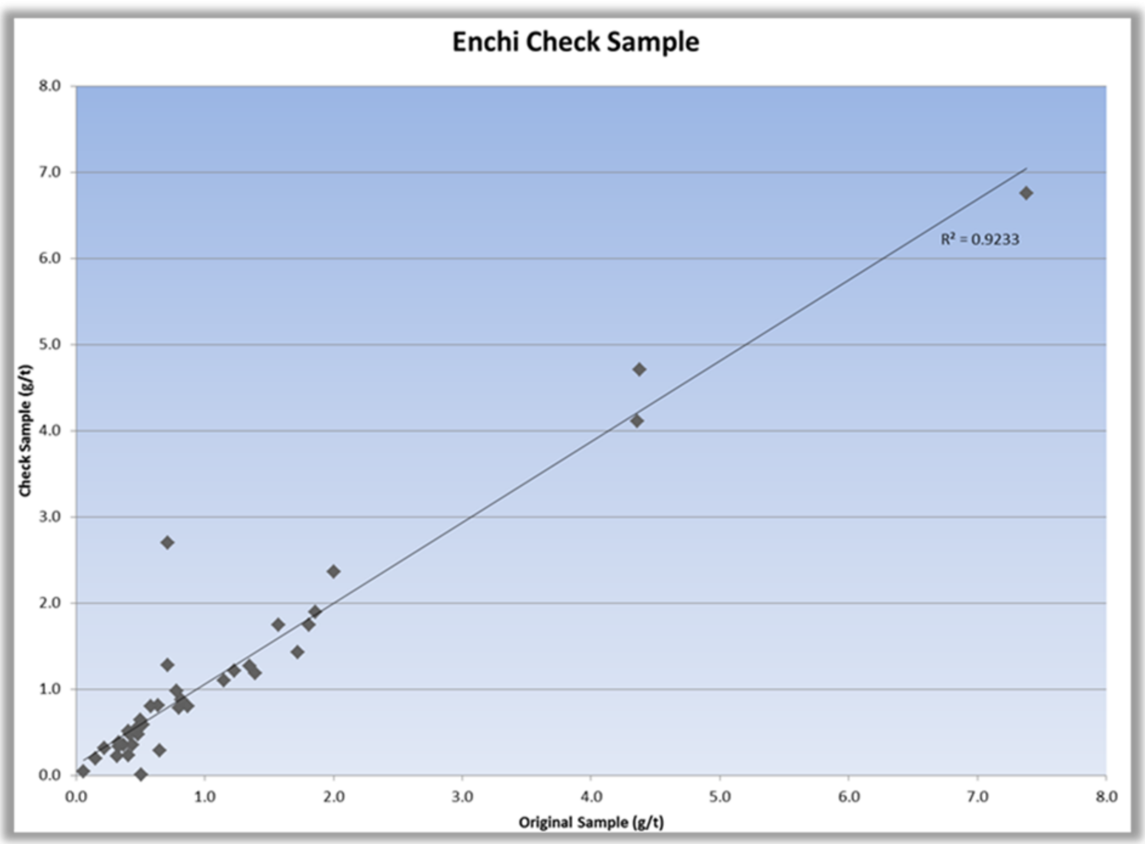


Figure 12-5: 2012 Check Samples

12.3.3 2017-2018 Assay

A series of representative samples from the 2017-2018 RC drill program were selected by the QP for validation assays. The QP, while employed by WSP, directed onsite Newcore personnel to collect 15 pulps from the 2017-2018 drilling program and the samples in their original sample bags were-submitted to SGS laboratories in Ghana for check analysis. Photographs of the original pulps as received from the primary assay laboratory were provided. The QP used the same analytical procedure as Newcore, at the second umpire laboratory, to minimize the potential variance from different analytical methods. The pulps were also subjected to a sieve test to investigate the preparation procedure.

Overall, the data collected has a good correlation to the original samples submitted by Newcore (Table 12-6) with a R2 of 0.89 (Figure 12-6). One sample failed the sieve test, yet the duplicate assay matched the original sample.

The chain of custody was not maintained during this process. The results of the check samples are similar to the check samples the QP collected on previous site visits.

Table 12-6: 2017-2018 Check Analysis

Borehole ID	Original Sample ID	Original Gold (g/t)	Check Sample D	Check Gold (g/t)	Log-QC (% passing 75 µm)
KBRC126	1064867	2.14	1076701	1.8	96.28
	1064899	1.56	1076702	1.97	89.56
	1064937	1.37	1076703	1.26	97.00
KBRC128	1065214	1.28	1076704	1.32	96.36
KBRC133	1065852	2.47	1076705	2.46	64.88
KBRC137	1066354	2.07	1076706	2.6	81.52
	1066365	1.38	1076707	1.63	95.36
KBRC129	1065232	<0.01	1076708	<0.01	98.72
SWRC058	1063403	1.52	1076709	1.46	97.60
SWRC059	1063503	<0.01	1076710	0.06	92.88
SWRC060	1063603	0.54	1076711	1.41	90.36
SWRC061	1063704	0.25	1076712	0.05	99.04
SWRC062	1063943	0.67	1076713	0.66	92.12
SWRC065	1064425	0.18	1076714	0.22	99.00
SWRC067	1066664	0.2	1076715	0.28	83.96

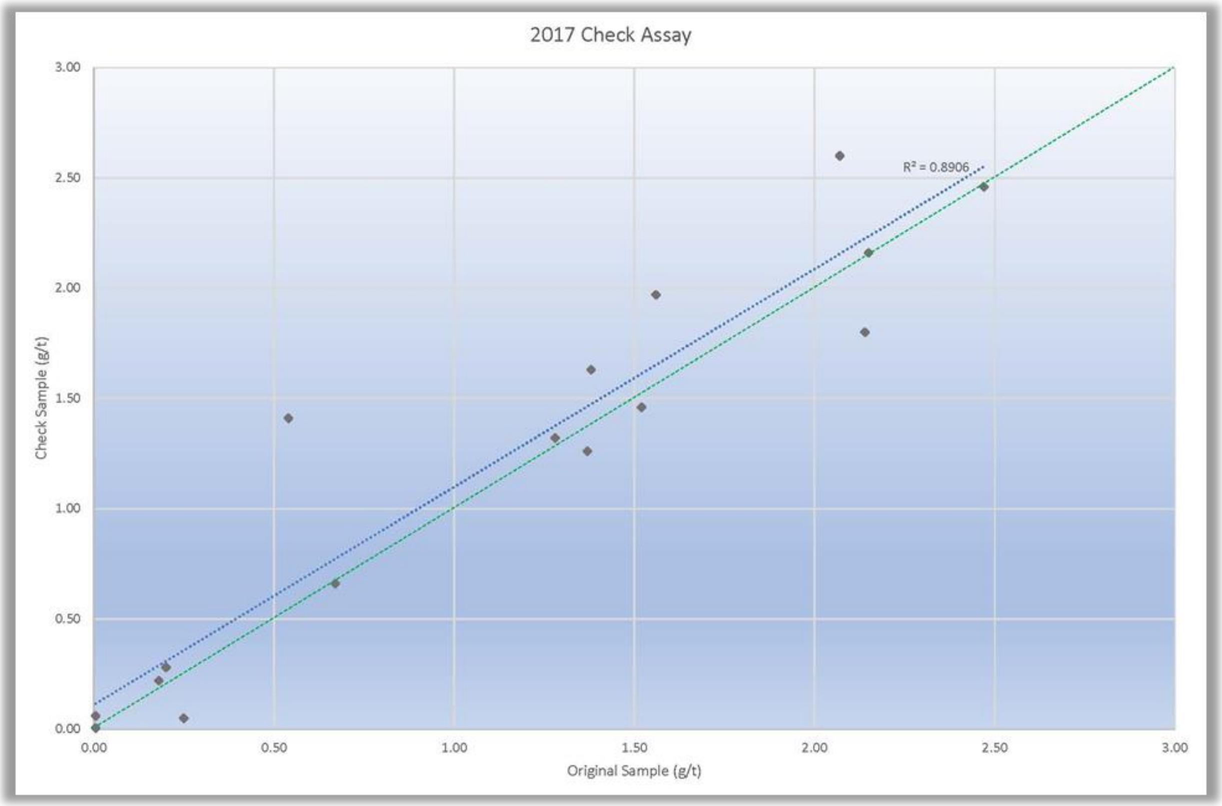


Figure 12-6: 2017-2018 Check Assay

12.3.4 2020-2021 Assay

A series of representative samples from the 2020-2021 RC drill program were selected by the QP for validation assays. The QP, while employed by SEMS Exploration Services Limited, collected 60 pulps from the 2020-2021 drilling program and the samples were-submitted to SGS laboratories in Ghana for check analysis. The QP used the same analytical procedure as Newcore, at the second umpire laboratory, to minimize the potential variance from different analytical methods.

Overall, the data collected confirms the presence of gold mineralization with an acceptable correlation to the original samples submitted by Newcore (Table 12-7) (Figure 12-9).

Table 12-7: 2020-2021 Check Analysis

Borehole ID	From (m)	To (m)	Original Sample ID (Intertek)	Original Gold (g/t)	Check Sample ID (SGS)	Check Gold (g/t)
KBRC157B	160	161	1070358	0.52	1110901	0.50
KBRC158	86	87	1070716	0.73	1110902	0.67
KBRC159	195	196	1071126	0.59	1110903	0.42
KBRC169	64	65	1080655	0.65	1110904	0.58
KBRC172	133	134	1081225	1.09	1110905	0.96
KBRC176	135	136	1081945	0.80	1110906	0.74
KBRC178	92	93	1082403	0.53	1110907	0.46
KBRC179	136	137	1082601	0.81	1110908	0.58
KBRC181	81	82	1093205	0.77	1110909	0.73
KBRC184	154	155	1093745	1.03	1110911	1.06
KBRC185	169	170	1088931	0.75	1110912	0.55
KBRC188	12	13	1089352	0.79	1110913	0.99
KBRC189	102	103	1089618	0.92	1110914	1.01
KBRC190	145	146	1089888	0.78	1110915	0.49
KBRC198	31	32	1091059	0.53	1110916	0.01
KKRC033	70	71	1095168	0.89	1110917	2.04
KKRC034	27	28	1095261	0.59	1110918	0.64
KKRC035	20	21	1095366	0.71	1110919	0.63
KKRC036	109	110	1095585	0.72	1110921	0.71
KKRC037	89	90	1095689	0.60	1110922	0.47
KKRC038	162	163	1095893	0.88	1110923	1.00
KKRC040	27	28	1096108	0.62	1110924	0.64
KKRC041	46	47	1096309	0.62	1110925	0.73
KKRC043	89	90	1096602	0.75	1110926	0.81
KKRC046	66	67	1103853	0.87	1110927	1.38
KKRC048	69	70	1104116	1.09	1110928	1.15
KKRC049	198	199	1104426	0.61	1110929	0.38
KKRC050	107	108	1104562	1.00	1110931	0.29
KKRC051	181	182	1104791	0.66	1110932	0.51
KKRC052	130	131	1105007	0.51	1110933	0.77
NBRC022	85	86	1075932	0.62	1110934	0.80
NBRC024	77	78	1096748	0.72	1110935	0.66
NBRC027	44	45	1097305	0.64	1110936	0.60
NBRC030	70	71	1097769	0.91	1110937	0.98
NBRC031	182	183	1098094	0.78	1110938	0.50
NBRC032	203	204	1098351	0.95	1110939	0.51

Borehole ID	From (m)	To (m)	Original Sample ID (Intertek)	Original Gold (g/t)	Check Sample ID (SGS)	Check Gold (g/t)
NBRC033	204	205	1098638	1.04	1110941	0.67
NBRC034	5	6	1098711	0.86	1110942	1.05
NBRC035	136	137	1099029	0.65	1110943	0.69
NBRC037	75	76	1099248	0.67	1110944	1.27
NBRC039	12	13	1099578	0.93	1110945	1.14
NBRC040	77	78	1099777	0.55	1110946	0.60
NBRC041	121	122	1099999	0.66	1110947	0.76
NBRC042A	34	35	1100126	0.77	1110948	0.74
NBRC043	116	117	1100331	0.68	1110949	1.23
SWRC068	51	52	1072024	0.51	1110951	0.60
SWRC069	106	107	1072196	0.95	1110952	0.53
SWRC070	146	147	1072379	0.76	1110953	0.77
SWRC071	39	40	1072461	1.08	1110954	1.45
SWRC072	68	69	1072604	0.52	1110955	0.60
SWRC073	25	26	1072723	0.76	1110956	0.61
SWRC074	13	14	1072821	0.96	1110957	1.05
SWRC075	172	173	1073108	0.87	1110958	0.61
SWRC076	41	42	1073207	1.03	1110959	0.95
SWRC077	33	34	1073332	0.63	1110961	0.64
SWRC079	279	280	1083411	1.13	1110962	0.13
SWRC080	158	159	1083603	1.86	1110963	0.96
SWRC082	97	98	1084039	0.70	1110964	0.73
SWRC084	120	121	1084618	0.59	1110965	0.60
SWRC085	10	11	1084774	0.63	1110966	0.49

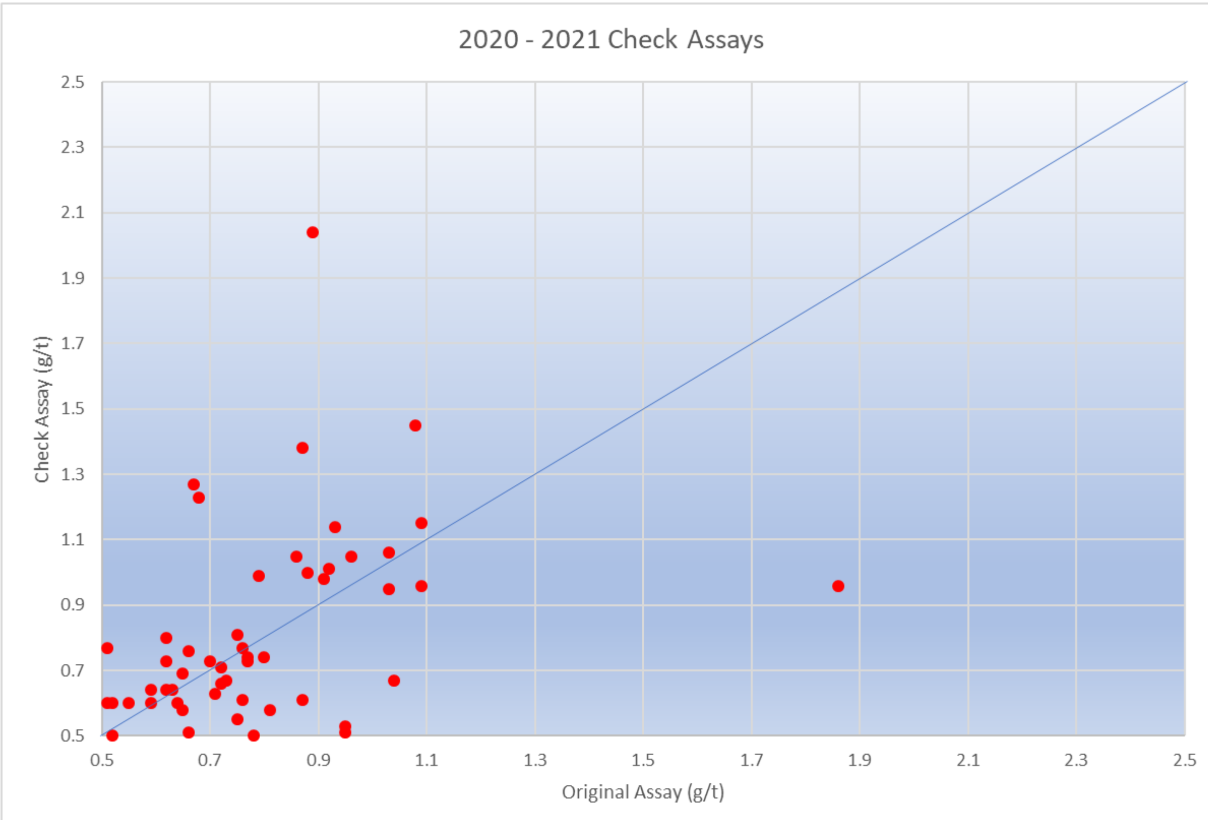


Figure 12-7: 2020-2021 Check Assay

12.4 Database

12.4.1 Red Back Data Validation

The trench, rotary air blast, and reverse circulation drilling log formats, procedures and dictionaries are found in the Exploration Procedures Manual provided by Red Back during the 2010 site visit conducted by Tetra Tech. The QP for the Tetra Tech report is the current QP and therefore accepts the results of the Tetra Tech validation.

All Red Back geological logging and sampling was conducted by geologists at the face, or on the rig, manually using standardized paper copy logging forms and dictionaries. All hard-copy field logs were manually transcribed by a data entry clerk into a Red Back designed Microsoft® Access® front-end database. The front-end database was designed with a set of data entry forms, the digital equivalent of the Exploration Manual, to capture all drillhole and trench collar, survey, geology, and sampling data. Each alphanumeric field, or attribute, has a linked look-up table, which controls the entry of the specifically defined dictionary codes for each of the defined database attributes. This prevents the entry of incorrect codes. The numeric fields included in the survey, sampling, and geology forms were also manually transcribed to complete the database for each project.

Following data entry, each drillhole or trench log is printed out for the logging geologist to validate and approve, or sign-off. The file is then transferred to the senior geologist to compile into the drillhole database. Micromine™ software has been used by the senior geologists to validate and compile the 3D drillhole databases at both the Boin and Sewum zones. Micromine™ includes a variety of techniques to validate the drillhole data. Senior geological staff completes the validation of the Project drillhole databases and all reverse circulation and diamond drill geological, geotechnical and structural logs were reviewed during the 2005 and 2006 programs, prior to compiling the final mineral resource and project sections.

The numeric assay data produced by Transworld and Analabs laboratories were merged and validated into the Access™ database through Datashed™ in Accra's head office. The senior geologist at site is responsible for the routine analysis and reporting on the QA/QC standards, blanks, and duplicates submitted during the programs.

Red Back routinely submitted a combined 12% quality control component with project sampling, comprising 8% blind field duplicates, 2% in-house blanks, and 2% Rocklabs certified reference material (CRM) standards.

The sampling programs conducted by Red Back were not observed or audited by the QP. At no time was an employee, officer, director or associate of Newcore involved in the sampling programs conducted by Red Back. A review of the Red Back geologist's procedure manual suggests a robust program that meets industry standards for sampling and quality assurance/quality control (QA/QC) measures.

Red Back constructed RC chip boards for the 2005 and 2006 RC drill programs. Boards were clearly marked with borehole numbers and depth intervals. The boards are stored in numerical order, in boxes at the Red Back office in Accra (Figure 12-8 and Figure 12-9).

Figure 12-8: Red Back Chip Boards (McCracken, 2010)

Limited diamond drill coring has been completed on the concession by Red Back. Diamond drillholes are extensions of RC holes. The core was sampled repeatedly and as a result very little core remains. Core was stored on site and was transported to the Red Back office in Accra. Core was placed in wooden boxes with wooden run markers placed at 1 m intervals. The boxes were marked with a marker on the end with the borehole number and the box number (Figure 12-10).

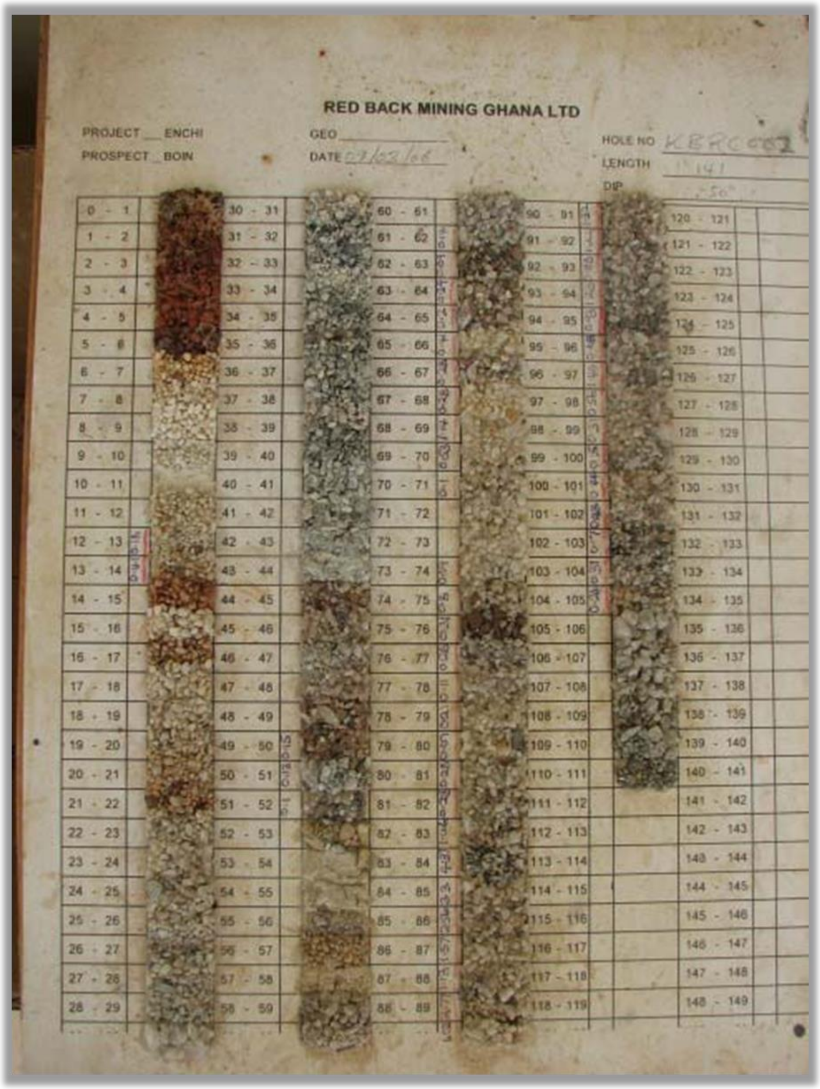


Figure 12-9: Red Back Chip Board (McCracken, 2010)

Figure 12-10: Red Back Drill Core (KBRC059) (McCracken, 2010)

12.4.2 Edgewater Pre-2012 Data Validation

The QP for the 2011 Tetra Tech report is the current QP and therefore accepts the results of the Tetra Tech validation. The QP validated 18 of the 180 drillholes completed in 2011. The QP carried out an internal validation of the drillhole data files against the original drillhole logs and assay certificates.

12.4.3 Newcore 2012 Data Validation

The QP carried out an internal validation of the diamond drillhole data files against the original drillhole logs and assay certificates. The validation of the data files was completed on the 23 drillholes finished in 2012.

Data verification was completed on collar coordinates, end-of-hole depth, downhole survey measurements, from and to intervals, assay sample intervals, and analytical results. Assay intervals in the database listed as less than 0.01 were converted to a value of 0.01 and were not considered an error. Sample intervals that were not assayed were input as absent data. It is the QP's opinion that material not sampled should not be assigned a zero value.

A significant error rate in the collar coordinates was indicated in the validation (Table 12-8). The collars in the digital database match reasonably well with the GPS coordinates collected during the site visit. It would appear that the drill logs were not updated with the correct coordinates after the final survey was completed.

There is a 14.9% error rate in the survey data in the digital data compared to the drill logs. The electronic survey files were reviewed and indicated that the digital database results are correct. The errors are always the last reading in the hole.

The drillhole data was imported into the Datamine™ program, which has a routine that checks for duplicate intervals, overlapping intervals, and intervals beyond the end-of-hole. The errors identified in the routine were checked against the original logs and corrected.

It is the QP's opinion that the data is of sufficient quality to support the mineral resource estimation.

Table 12-8: 2012 Data Validation Summary

Field	Number of Records	Number of Errors	Error Rate
Collar			
BHID	23	0	0%
East	23	19	83%
North	23	19	83%
Elevation	23	0	0%
Azimuth	23	0	0%
Dip	23	0	0%
Depth	23	0	0%
Survey			
BHID	87	0	0%
Depth	87	0	0%
Azimuth	87	13	15%
Dip	87	13	15%
Lithology			
BHID	3,806	0	0%
From	3,806	0	0%
To	3,806	0	0%
Litho	3,806	37	1%

Field	Number of Records	Number of Errors	Error Rate
Assay			
BHID	3,800	0	0%
Sample #	3,800	0	0%
From	3,800	0	0%
To	3,800	0	0%
Au	3,800	4	0%

12.4.4 Newcore 2017-2018 Data Validation

The QP carried out an internal validation of the diamond drillhole data files against the original drillhole logs and assay certificates. The validation of the data files was completed on the 28 drillholes finished in 2017-2018.

Data verification was completed on collar coordinates, end-of-hole depth, downhole survey measurements, from and to intervals, assay sample intervals, and analytical results. Assay intervals in the database listed as less than 0.01 were converted to a value of 0.01 and were not considered an error.

Several holes had elevation issues relative to the available topography. Elevations in the digital file were adjusted to closely match the topography.

The drillhole data was imported into the Datamine™ program, which has a routine that checks for duplicate intervals, overlapping intervals, and intervals beyond the end-of-hole. The errors identified in the routine were checked against the original logs and corrected

12.4.5 Newcore 2020 – 2021 Data Validation

The QP carried out an internal validation of the RC drillhole data files against the original drillhole logs and assay certificates. The validation of the data files was completed on the 128 drillholes completed in 2020-2021. Only drillholes completed before the cut-off dates were validated and used in the Mineral Resource Estimate holes listed in Table 12-2: RC Holes Drilled after the Mineral Resource Cut-off Dates were not validated.

Data verification was completed on collar coordinates, end-of-hole depth, downhole survey measurements, from and to intervals, assay sample intervals, and analytical results. Assay intervals in the database listed as <0.01 were converted to a value of 0.009 and were not considered an error.

Several holes had elevation issues relative to the available topography. Elevations in the digital file were adjusted to closely match the topography.

The drillhole data was imported into the Datamine™ program, which has a routine that checks for duplicate intervals, overlapping intervals, and intervals beyond the end-of-hole. The errors identified in the routine were checked against the original logs and corrected.

12.5 QP's Opinion

It is the QPs' opinion that the database has been adequately validated and is suitable to be used for geological and mineral resource modelling.

For RC drilling completed in 2021, drilling post hole number KBRC184 at Boin, NBRC023 at Nyam, and KKRC054 at Kwakyekrom were not used in the 2021 Mineral Resource Estimation.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineralogy

The four main zones (Sewum, Boin, Nyam and Kwakyekrom) are generally considered to be mesothermal quartz vein style gold deposits. The mineralization is found in structurally controlled zones of quartz veining or silicified volcanics with pyrite. With quartz-vein style mineralization, the gold occurs mainly as liberated gold particles but may have some disseminated gold. Gold is present in saprolite outcrops and chlorite and epidote clots and has very low levels of sulphides, less than 1% S; other metal contents are low such as less than 2 ppm silver, and 100 ppm copper. The levels of gangue minerals such as quartz, chlorite, carbonates and other carbonaceous matter are not known. They may have a negative impact on the extraction of gold, as they tend to re-adsorb the gold after it has been leached. Kaolin content should also be analyzed since its presence will result in reducing percolation in the heaps, increasing leach time and reducing the overall gold recovery.

13.2 Metallurgical Test Work

Preliminary metallurgical test work has been performed on reverse circulation (RC) drill chip samples from the four zones at Sewum, Boin, Nyam and Kwakyekrom during the past 10 years in order to assess amenability for gold recovery by cyanidation.

The initial test work was done by SGS in 2012 for Edgewater. The 2012 test work involved 24-hour bottle roll leach tests on samples from the Sewum, Boin and Nyam deposits. The following is a brief summary of the 2012 SGS leach test work performed and results as described in Edgewater's internal report 'Summary of Metallurgical Tests Completed on Samples from the Enchi Gold Project' (SGS, 2012).

- First stage of 2012 bottle roll tests were performed on a series of 23 highly oxidized samples from one drillhole from the Boin deposit.
- Second stage of 2012 bottle roll tests were performed as two sets of samples from drillholes from Boin (4 highly oxidized samples), Sewum (11 vaguely oxidized samples) and Nyam (5 slightly oxidized samples).
- Good metallurgical recoveries were achieved from the Boin and Nyam 2012 bottle roll tests, averaging 87% and 70% respectively. In contrast, only one of the 11 Sewum samples tested in 2012 had a reasonable recovery (67%) with the Sewum recovery averaging only 18.7%.

Newcore has continued a reverse circulation (RC) drilling campaign on the Enchi deposit to expand mineral resources. More recent metallurgical test work was performed by Intertek in 2020 and 2021 for Newcore on chip samples from the 2020 RC drill program in support of the Project. The recent metallurgical test work has been performed on a variety of samples from the Sewum, Boin and Kwakyekrom deposits covering a range of gold grades, weathering intensities and different portions of each deposit. The recent metallurgical test work performed by Intertek was done as three series of tests, which have been reported by Newcore sequentially as follows:

1. Newcore First Set Metallurgical Results (Boin/Sewum) January 3rd, 2021;
2. Newcore Second Set Metallurgical Results (Kwakyekrom) April 27th, 2021;
3. Newcore Column Tests of Composite Samples and Metallurgical Results (Boin & Sewum) May 7th, 2021.

A summary description of the scope and results for each of these recent metallurgical test work campaigns are as follows:

NCAU First Set Metallurgical Results (Boin/Sewum) January 3rd, 2021

A total of 50 RC drill samples from the Sewum (30 samples) and Boin (20 samples) deposits were analyzed and tested as follows:

- Head grade fire assay (50 gram fire assay with AAS finish);
- Bottle roll cyanide assay (1 kg material for 24 hours);
- Fire assay of bottle roll tails.

The bottle roll leach test results averaged 89.4% gold extraction for 49 of the samples. The 20 Boin samples and the 29 Sewum samples averaged 86.4% and 91.4% extraction, respectively. One of the Sewum samples was excluded from the bottle roll testing due to erratic head assays ranging between 17.3 and 1.23 g/t Au.

Independent of the bottle roll tests, 10 of the 50 samples were randomly selected for leach rate dissolution tests under the following conditions:

- 1 kg samples;
- 40% solid: liquid ratio;
- Leach time 24 hours;
- Cyanide addition: 1,200 ppm cyanide solution;
- pH 9.5-10.5 adjusted with lime as required;
- Grind of ≥ 85% passing 75 µm;
- Aliquots were taking at 6, 12 and 24 hours during leaching.

The 10 samples averaged 82.2% extraction after 24 hours, 79.4% after 12 hours and 73.2% after 6 hours. There was a significant range, greater than 50%, in the high and low extraction values for each time period.

Other observations from the First Set Metallurgical Results are listed as follows:

- The majority of the 50 samples were from the W3 and W4 categories, which represent the largest component of the weathered profile and gave the highest extraction values averaging 91.5%;
- Both the W2 and W5 samples gave more erratic and slightly lower extraction values at 80.0% and 84.6%, respectively.
- No significant relationship exists between gold head grade and gold extraction percentage;
- No significant relationship exists between sample depth and extraction percentage;
- Bottle roll gold extraction results for the Sewum samples from the 2020 RC drill program were very good, averaging 91.4%, compared to the Sewum samples from the SGS (2012) testing, which averaged only 18.7%.
- There was a wide range in lime consumption between 2.5 and 7.5 kg/t required to maintain the desired pH in the rate dissolution tests with more weathered samples requiring more lime;
- There is some evidence of coarse and possibly encapsulated gold in the test results as indicated by some erratic head assay and low extraction values.

Newcore Second Set Metallurgical Results (Kwakyekrom) April 27th, 2021

A total of 25 oxide samples from the Kwakyekrom Zone 2020-2021 RC drilling program were submitted to the Intertek Lab in Tarkwa, Ghana with all samples analyzed and tested as follows:

- Head grade fire assay (50 gram fire assay with AAS finish);
- Bottle roll cyanide assay (1 kg material for 24 hours);
- Fire assay of bottle roll tails.

The 25 samples were selected from six drillholes distributed across 350 m of strike length within the main area of the drill tested portion of the Kwakyekrom structure. The bottle roll leach test results averaged 79.8% gold extraction with a range from 66.1% to 90.3%.

Other noteworthy remarks and observations from the Second Set Metallurgical Results are listed as follows:

- All 25 samples were from the W3 and W4 categories, which are representative of the oxidized portion of the weathered profile;
- Samples from the moderately weathered W3 profile gave an average gold extraction of 81.0% compared to the more highly weathered W4 samples that averaged 74.9% extraction;
- No significant relationship exists between gold head grade and gold extraction percentage;
- No significant relationship exists between sample depth and extraction percentage.

Newcore Column Tests of Composite Samples and Metallurgical Results (Boin & Sewum) May 7th, 2021

Samples remaining from the 2020 RC drill program from the Boin and Sewum deposits were selected for the preparation of four different composite samples for preliminary column leach tests. These tests were performed at the Intertek Lab in Tarkwa, Ghana.

Each of the four composites was prepared by combining six different reject samples left over from the First Set Metallurgical tests from various zones of the Sewum and Boin deposits. The deposit/zone represented by each composite and the average bottle roll gold extraction for each of the six samples that went into each composite sample is shown in Table 13-1.

Table 13-1: Summary of Composite Samples Prepared for Initial Small Column Leach Tests

Composite	Deposit / Zone	Average Bottle Roll Au Ext'n % from First Set Tests
# 1	Boin Central	98.1%
# 2	Boin North and South	92.7%
# 3	Sewum Ridge	98.1%
# 4	Sewum CH and Ext	97.8%
Average		**96.7%**

A size analysis was performed on each of the four composites and the average size distribution was found to be approximately 80 weight percent passing 2 mm (P_{80} = 2 mm). Based on the very fine size of the composite sample materials, the composite samples were agglomerated with 20 kg/t of Portland cement before being placed into the small test columns. The average composite weight available for the four column tests was 16.3 kg.

In addition to the column tests, a coarse subsample was prepared from each of the four composite samples and subjected to a 5-day Coarse Bottle Roll. Extraction results from the Coarse Bottle Roll tests and the column leach tests after 15, 30 and 60 days of leaching are summarized in Table 13-2.

Table 13-2: Au Extraction Results for Coarse Bottle and Column Leach Tests on Composite Samples

Composite	Coarse Bottle Roll Au Ext'n %	Column Test 15 Day Au Ext'n %	Column Test 30 Day Au Ext'n %	Column Test 60 Day Au Ext'n %
# 1	88.0	55.0	59.1	81.9
# 2	80.6	64.0	77.8	97.5
# 3	84.7	69.7	75.1	97.7
# 4	87.9	66.4	73.8	91.7
Average	**85.3**	**63.8**	**71.4**	**92.2**

Observations from the initial column leach tests are listed as follows:

- The average extraction achieved from 60 day column leaching of the four composite samples was 92.2% with a range from 81.9% to 97.7%;
- The kinetic leach profile of the four column tests showed rapid leaching for the first 15 days to an average extraction of 63.8%, followed by slower leaching from day 15 to 30 (to an average extraction of 71.4%), followed by continued gradual leaching to 60 days. Based on these preliminary small column tests, 60 days of leaching was assumed for the Project design criteria.
- The average P_{80} size of 2 mm for the RC drill cuttings used to make the four composite samples is much finer than the P_{80} crush size of 12.5 mm assumed in the Project design criteria.
- The average gold extraction achieved in these initial column tests is expected to be significantly higher than the extraction that would be achieved if the composite samples had been a more typical heap leach feed size material. Based on this, and for the purpose of the Project evaluation, the expected Sewum and Boin heap leach gold extraction is estimated at 82%.
- Expected reagent consumption rates for cement, lime and cyanide cannot be estimated based on the preliminary column leach tests. Reasonable consumption estimates of 15 kg/t cement, 2.0 kg/t lime and 0.3 kg/t cyanide have been assumed for the process design criteria and used in the process operating cost estimates.

13.3 Conclusions

The results of the recent preliminary bottle roll tests, kinetic leach tests and initial small column tests confirm that gold extraction via cyanidation is generally good for the oxide domains of the Boin, Sewum and Kwakyekrom deposits. It is noted that no samples from the Nyam deposit have been evaluated in the recent metallurgical tests. Further sample collection, preparation and specific heap leach metallurgical test work is required to definitively select heap leaching as the best technical process option for the Project. For the purpose of this study, heap leaching with a permanent pad HLF has been selected as the process option.

Based on all the reported preliminary leach tests performed to date, an average overall gold recovery of 79% has been estimated for the Project design criterion. The overall gold recovery estimate is based on the updated project mineral resource model tonnage and grade as summarized in Table 13-3. Leach solution gold recovery of 98.5% has been applied to the HLF gold extraction percent for each deposit to estimate the overall gold recovery. This recovery estimate is consistent with other successful gold heap leach operations for similar deposits.

Table 13-3: Overall Project Gold Recovery Estimate

Enchi Deposit	Leach Feed (Mt)	Leach Feed Grade (Au g/t)	Overall (Au Rec. %)
Sewum	39.75	0.505	80
Boin	22.65	0.649	80
Nyam	4.52	0.735	60
Kwakyekrom	1.65	0.519	72
Average		**0.568**	**79**

Gold extraction and recovery from cyanide leaching is generally sensitive to particle size. Typically, gold recovery will increase with finer particle size but with associated higher operating costs, and potential permeability issues in the case of heap leaching. The particle size distribution was not reported for most of the bottle roll tests; however, the size distribution of the composite samples used for the preliminary column leach test was measured and found to be much finer than a typical crushed mineralized material heap leach feed size.

All indications are that the product sample material obtained by RC drilling of the four Project deposits and used in the recent preliminary metallurgical tests is too fine for heap leach design purposes. The average P_{80} size distribution of the composite sample materials used for the four preliminary column leach tests was about 2 mm, compared to the more conventional heap leach feed size P_{80} of 12.5 mm that was assumed in the Project design criteria. As a result, all bottle roll and column leach test work extraction values have been discounted by about 10% to estimate the heap leach gold extraction for the four Project deposits as shown in Table 13-3.

Heap leach process design, gold extraction and overall recovery is determined and affected by:

- Mineralized material grade, mineralogy and variability;
- Crush product size;
- Reagent concentration and addition rates;
- Agglomeration;
- Irrigation system distribution and spacing;
- Solution recovery design;
- Stacking method and operation;
- Heap height;
- Heap edge effects;
- Operations.

Based on the preliminary leaching test work that has been done to date, many of the key design variables have not yet been determined. It is recommended that diamond drilling and/or bulk sampling be used to obtain samples for subsequent and confident metallurgical and HLF design purposes.

14. MINERAL RESOURCE ESTIMATES

BBA completed a mineral resource estimation of the Enchi Project under the supervision of Mr. Todd McCracken, the QP for this chapter. Ms. Charlotte Athurion, P. Geo., assisted with the completion of the Mineral Resource Estimate. The mineral resource update was completed on the Sewum, Boin, Nyam and Kwakyekrom Zones using data up to various cut-off periods for each deposit.

Drillholes completed after the various cut-off periods were not used in the Mineral Resource Estimate. Chapter 10 discloses the list of holes not used in the Mineral Resource Estimate.

14.1 Database

Newcore maintains all borehole data in a Microsoft® Access® database. Header, survey, assays, and lithology tables are saved on individual tabs in the database. Individual Excel® files exported from the database for each of the deposits was provided to the QP by Newcore on various cut-off dates:

- Sewum: February 8, 2021
- Boin: January 7, 2021
- Nyam: December 22, 2020
- Kwakyekrom: April 7, 2021

The Project database used to determine the mineral resource contains a total of 1,174 boreholes (diamond drill, reverse circulation, and reverse air blast) and trenches. Of this database, 98% of the drilling has occurred within the Sewum, Boin, Nyam and Kwakyekrom Zones. Table 14-1 summarizes the borehole database.

Table 14-1: Enchi Borehole Summary

Hole type	No. of Holes	Total Metres	% Holes	% Metres
Sewum				
Diamond Drill	71	9,925	6	9
RC	99	14,003	8	13
RAB	44	3,086	4	3
Trench	88	8,546	7	8
Boin				
Diamond Drill	62	7,570	5	7
RC	197	24,824	17	22
RAB	275	9,338	23	8
Trench	64	5,761	5	5

Hole type	No. of Holes	Total Metres	% Holes	% Metres
Nyam				
Diamond Drill	47	5,131	4	5
RC	102	8,355	9	8
Trench	18	1,303	2	1
Kwakyekrom				
Diamond Drill	1	100	0	0
RC	67	7,821	6	7
Trench	18	2,684	2	2
Other Zones				
RC & Trench	21	2,535	2	2
Total	**1,174**	**110,982**	**100%**	**100**

The non-assayed intervals within the database were assigned a value equal to the detection limit. The QP believes that non-assayed material should not be assigned a zero value, as this does not reflect the true value of the material. Sample intervals with values below detection limit (<) in the database were assigned the detection limit.

The mineral resource database was validated before proceeding to the mineral resource estimation. The validation steps are detailed in Chapter 12 of this Report. Minor variations have been noted during the validation process but have no material impact on the 2021 MRE.

Since the various cut-off dates listed above, 103 holes totalling 15,758 m have been drilled on the Project. All these holes are outside the mineral resource shells.

The QP is of the opinion that while the addition of these new holes would increase knowledge and confidence on the Project and may have a positive impact on future mineral resource estimates, the holes do not have a material impact on the current study presented in this Report.

14.2 Specific Gravity

The QP used a Specific Gravity (SG) of 2.20 for oxide, 2.45 for transition and 2.70 for fresh material in the Mineral Resource Estimate. These values are based on the values for SG in the various material at the Chirano Mine and Asanko Gold's Esaase Project. Chirano and Asanko were selected as suitable analogies because these projects occur in the similar rock types, along the same regional structure, in the same country. The Chirano Mine is located 50 km north along strike of the Enchi project, with Esaase found further along strike past Chirano. The SG dataset from Asanko is approximately 15,000 records (DRA Projects Pty Limited, 2017).

The use of assumed SG from the region is acceptable practice in the absence of data. The absence of SG data is a contributing factor in the determination of the mineral resource classification.

The drill logs recoded a weathering profile from W1 (fresh) to W6 (strong). Digital terrain models (DTM) where generated based on the logs for oxidized (W6, W5 and some W4) and fresh (W1, W2 and some W3) for all the deposits. Blocks below the W1 DTM were assigned a SG of 2.70, blocks above the W5 DTM were assign a SG of 2.20 and blocks between the W1 and W5 blocks were assigned a SG of 2.45.

The QP recommends that Newcore collect SG measurements based on the weathering profile (weathered, transition, and fresh) in order to build up the dataset. At a minimum, 5% of the dataset should have SG measurements before an acceptable value can be determined.

14.3 Topographic Area

The topographic surface used in the Mineral Resource Estimate is digital terrain model (DTM) based on a larger digital topographic file and adjusted on the collar coordinates of the drillhole and trenches at each zone. Additional data points were inserted to smooth out some of the major discrepancies in the DTM.

14.4 Geological Interpretation

The original three-dimensional wireframe models of mineralization were developed in Datamine™ by the QP with approval of all shapes by Newcore. The basic wireframe designs for each of the zones were based on design criteria that included a minimum downhole width of 2 m and a minimum grade of 0.2 g/t gold.

The zones of mineralization interpreted for each area were generally contiguous; however, due to the nature of the mineralization there are portions of the wireframe that have grades less than 0.2 g/t gold yet are still within the mineralized trend.

14.4.1 Sewum

Sewum has five zones; Sewum South (Z01), Checkerboard Hill (Z02), Sewum Shear (Z03), Ridge Top (Z04) and Gap (Z05). Ridge Top is further sub-divided into eight domains.

14.4.2 Boin

Boin has six zones. Zones 1, 2, 3, 5, and 6 are sub-parallel veining systems. Zone 4 is located to the northwest of the main Boin system.

14.4.3 Nyam

Nyam has five zones. The zones are sub-parallel veining systems.

14.4.4 Kwakyekrom

Kwakyekrom has seven zones. The zones are sub-parallel veining systems.

14.5 Exploratory Data Analysis

14.5.1 Assays

The portion of the deposit included in the mineral resource was sampled by 25,093 gold assays. The assay intervals within each zone were flagged within the database. These borehole files were reviewed to ensure that all the proper assay intervals were captured. Table 14-2 summarizes the basic statistics for the assays at Enchi and for each zone within the four deposit areas individually.

Table 14-2: Enchi Drill Statistics by Zone

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
Sewum							
z01	Length (m)	709	0	0.20	2.20	1.07	0.113
	Au (ppm)	707	2	0.01	9.43	0.75	1.432
z02	Length (m)	3,679	0	0.10	22.5	1.18	0.954
	Au (ppm)	3,591	88	0.01	30.60	0.37	0.903
z03	Length (m)	339	0	0.40	2.60	1.01	0.033
	Au (ppm)	323	16	0.01	5.58	0.24	0.152
z4_1	Length (m)	1,405	0	0.20	3.00	1.26	0.311
	Au (ppm)	1,299	106	0.01	24.75	0.41	0.930
z4_2	Length (m)	451	0	0.50	3.00	1.42	0.638
	Au (ppm)	450	1	0.01	2.84	0.15	0.088
z4_3	Length (m)	333	0	0.50	3.00	1.36	0.578
	Au (ppm)	328	5	0.01	12.50	0.21	0.581
z4_4	Length (m)	577	0	0.50	3.00	1.25	0.424
	Au (ppm)	575	2	0.01	18.97	0.42	0.965
z4_5	Length (m)	747	0	0.50	3.00	1.22	0.352
	Au (ppm)	746	1	0.01	14.40	0.55	1.149
z4_6	Length (m)	218	0	0.30	3.00	1.09	0.176
	Au (ppm)	217	1	0.01	4.00	0.37	0.395
z4_7	Length (m)	119	0	0.50	2.00	1.04	0.065
	Au (ppm)	119	0	0.01	2.47	0.31	0.186
z4_8	Length (m)	67	0	1.00	1.00	1.00	-
	Au (ppm)	67	0	0.01	2.67	0.57	0.296

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
z05	Length (m)	83	0	1.00	1.00	1.00	-
	Au (ppm)	83	0	0.01	3.02	0.30	0.200
z99	Length (m)	21,405	0	0.10	84.00	1.20	0.906
	Au (ppm)	21,096	309	0.01	12.50	0.05	0.0244
Boin							
z1	Length (m)	1,088	0	0.20	3.00	1.03	0.040
	Au (ppm)	1,088	0	0.01	11.98	0.49	0.855
Z2	Length (m)	392	0	1.00	3.40	1.07	0.116
	Au (ppm)	392	0	0.01	3.25	0.23	0.215
Z3	Length (m)	206	0	1.00	3.00	1.03	0.044
	Au (ppm)	206	0	0.01	7.81	0.43	0.776
Z4	Length (m)	229	0	1.00	2.00	1.15	0.129
	Au (ppm)	229	0	0.01	4.73	0.41	0.589
Z5	Length (m)	8,625	0	0.20	3.00	1.07	0.114
	Au (ppm)	8,623	2	0.01	17.61	0.40	1.120
Z6	Length (m)	46	0	1.00	1.00	1.00	-
	Au (ppm)	46	0	0.01	1.84	0.24	0.150
Z99	Length (m)	27,922	0	0.10	33.00	1.29	0.473
	Au (ppm)	27,839	83	0.01	13.09	0.04	0.019
Nyam							
Z1	Length (m)	2,516	0	0.30	2.00	1.14	0.119
	Au (ppm)	2,500	16	0.01	9.54	0.70	0.985
Z2	Length (m)	33	0	1.00	2.00	1.12	0.107
	Au (ppm)	33	0	0.04	3.37	0.48	0.313
Z3	Length (m)	703	0	1.00	2.00	1.20	0.160
	Au (ppm)	701	2	0.01	44.20	0.59	3.734
Z4	Length (m)	571	0	1.00	2.00	1.21	0.164
	Au (ppm)	571	1	0.01	16.10	0.61	1.953
Z5	Length (m)	117	0	1.00	2.00	1.06	0.056
	Au (ppm)	115	2	0.01	16.70	1.19	6.504
Z99	Length (m)	7,331	0	0.20	6.00	1.21	0.170
	Au (ppm)	7,309	22	0.01	83.87	0.07	0.983
Kwakyekrom							
Z1	Length (m)	346	0	1.00	4.00	1.19	0.467
	Au (ppm)	345	1	0.01	6.44	0.39	0.553
Z2	Length (m)	375	0	1.00	4.00	1.15	0.285
	Au (ppm)	375	0	0.00	8.95	0.55	1.102

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
Z3	Length (m)	583	0	1.00	4.00	1.10	0.223
	Au (ppm)	583	0	0.00	13.18	0.68	1.036
Z4	Length (m)	360	0	1.00	4.00	1.08	0.205
	Au (ppm)	360	0	0.01	7.49	0.40	0.605
Z5	Length (m)	295	0	1.00	4.00	1.02	0.061
	Au (ppm)	295	0	0.01	7.44	0.28	0.361
Z6	Length (m)	109	0	1.00	1.00	1.00	-
	Au (ppm)	109	0	0.01	1.40	0.17	0.055
Z7	Length (m)	17	0	1.00	4.00	1.18	0.498
	Au (ppm)	17	0	0.01	0.61	0.17	0.037
Z99	Length (m)	6,313	0	0.50	150.00	1.32	4.671
	Au (ppm)	6,291	22	0.00	3.71	0.04	0.011

14.5.2 Composite

Sewum, Boin, Nyam and Kwakyekrom assay data was composited on 2 m intervals honoring the geological interpretation.

The process was used in the compositing routine to ensure all captured sample material was included. The routine adjusts the composite lengths for each individual borehole in order to compensate for the last sample interval. Table 14-3 summarizes the statistics for the boreholes after compositing.

Table 14-3: Enchi Composite Statistics by Zone

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
Sewum							
z01	Length (m)	389	0	1.5	2.00	1.95	0.00754585
	Au (ppm)	387	2	0.01	8.43	0.72	1.08195798
z02	Length (m)	2,188	0	1.94	2.40	1.99	0.00045382
	Au (ppm)	2,077	111	0.01	30.60	0.39	0.9099088
z03	Length (m)	174	0	1.67	2.00	1.96	0.00654957
	Au (ppm)	167	7	0.01	3.10	0.24	0.09934673
z4_1	Length (m)	893	0	1.50	2.01	1.98	0.00345773
	Au (ppm)	794	99	0.01	24.75	0.41	1.09116977
z4_2	Length (m)	335	0	1.67	2.00	1.92	0.00791822
	Au (ppm)	335	0	0.01	1.88	0.18	0.07135826
z4_3	Length (m)	239	0	1.00	2.00	1.90	0.01557546
	Au (ppm)	237	2	0.01	12.50	0.23	0.71580396

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
z4_4	Length (m)	378	0	1.50	2.00	1.91	0.011
	Au (ppm)	378	0	0.01	10.88	0.41	0.516
z4_5	Length (m)	471	0	1.50	2.10	1.94	0.007
	Au (ppm)	471	0	0.01	7.03	0.52	0.626
z4_6	Length (m)	123	0	1.67	2.00	1.93	0.010
	Au (ppm)	123	0	0.01	3.44	0.35	0.291
z4_7	Length (m)	65	0	1.67	2.00	1.91	0.014
	Au (ppm)	65	0	0.01	1.51	0.31	0.128
z4_8	Length (m)	35	0	1.83	2.00	1.91	0.004
	Au (ppm)	35	0	0.02	2.03	0.57	0.202
z05	Length (m)	44	0	1.67	2.00	1.89	0.019
	Au (ppm)	44	0	0.01	1.54	0.30	0.099
z99	Length (m)	12,950	0	1.00	2.07	1.98	0.002
	Au (ppm)	12,664	286	0.01	10.15	0.05	0.020
Boin							
z1	Length (m)	572	0	1.83	2.20	1.97	0.002
	Au (ppm)	572	0	0.01	7.17	0.48	0.608
Z2	Length (m)	214	0	1.50	2.00	1.95	0.005
	Au (ppm)	214	0	0.01	2.68	0.27	0.211
Z3	Length (m)	111	0	1.50	2.08	1.91	0.015
	Au (ppm)	111	0	0.01	4.63	0.42	0.458
Z4	Length (m)	134	0	1.91	2.00	1.97	0.001
	Au (ppm)	134	0	0.01	4.73	0.46	0.598
Z5	Length (m)	4,672	0	1.00	2.035	1.98	0.002
	Au (ppm)	4,671	1	0.01	13.19	0.40	0.783
Z6	Length (m)	25	0	1.50	2.00	1.84	0.023
	Au (ppm)	25	0	0.01	1.01	0.23	0.078
Z99	Length (m)	18,239	0	1.00	2.50	1.98	0.002
	Au (ppm)	18,187	52	0.01	7.42	0.04	0.011
Nyam							
Z1	Length (m)	1,454	0	1.75	2.15	1.97	0.002
	Au (ppm)	1,449	5	0.01	8.23	0.66	0.680
Z2	Length (m)	21	0	1.50	2.00	1.76	0.032
	Au (ppm)	21	0	0.09	1.99	0.47	0.149
Z3	Length (m)	443	0	1.00	2.00	1.91	0.022
	Au (ppm)	443	0	0.01	12.72	0.53	1.226
Z4	Length (m)	363	0	1	2.00	1.90	0.021
	Au (ppm)	363	0	0.01	8.73	0.54	1.162

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
Z5	Length (m)	65	0	1.50	2.00	1.91	0.009
	Au (ppm)	64	1	0.01	13.72	1.17	4.816
Z99	Length (m)	4,499	0	1.00	2.15	1.97	0.006
	Au (ppm)	4,485	14	0.01	41.94	0.07	0.406
Kwakyekrom							
Z1	Length (m)	215	0	1.50	2.00	1.91	0.013
	Au (ppm)	214	1	0.01	5.09	0.36	0.391
Z2	Length (m)	226	0	1.50	2.00	1.90	0.013
	Au (ppm)	226	0	0.00	5.10	0.50	0.670
Z3	Length (m)	329	0	1.50	2.00	1.96	0.006
	Au (ppm)	329	0	0.00	6.35	0.64	0.676
Z4	Length (m)	202	0	1.67	2.00	1.92	0.010
	Au (ppm)	202	0	0.01	5.89	0.38	0.467
Z5	Length (m)	157	0	1.67	2.00	1.92	0.009
	Au (ppm)	157	0	0.01	4.78	0.28	0.242
Z6	Length (m)	58	0	1.67	2.00	1.88	0.006
	Au (ppm)	58	0	0.01	0.97	0.17	0.046
Z7	Length (m)	10	0	2.00	2.00	2.00	-
	Au (ppm)	10	0	0.03	0.53	0.15	0.030
Z99	Length (m)	4,196	0	1.00	2.17	1.98	0.003
	Au (ppm)	4,041	155	0.00	2.28	0.05	0.011

14.5.3 Grade Capping

Composite gold assays were examined individually to assess the amount of metal that is at risk from high grade assays. Decile analysis (Parrish, 1997) was used to assist in the determination if grade capping was required. A review of the 3D spatial distribution of the capped samples was completed to determine if the samples are spatially close and potentially a higher-grade sub-domain. This was not observed in any of the zones on the Project. Capping was set for individual zones within each deposit (Table 14-4).

Table 14-5 is a summary of the drillhole composites after grade capping has been applied.

Table 14-4: Enchi Grade Capping Value by Zone

Zone	Grade Cap (g/t)
Sewum	
Z01, Z03, Z05	no cap
Z02	8
Z04	10
Boin	
All Zones	8
Nyam	
All Zones	10
Kwakyekrom	
Z1, Z2, Z3, Z4	5
Z5, Z6, Z7	no cap

Table 14-5: Enchi Capped Composite Statistics by Zone

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
Sewum							
z01	Length (m)	389	0	1.50	2.00	1.95	0.008
	Au (ppm)	387	2	0.01	8.44	0.72	1.082
z02	Length (m)	2,188	0	1.94	2.40	1.99	0.000
	Au (ppm)	2,077	111	0.01	8.00	0.38	0.456
z03	Length (m)	174	0	1.67	2.00	1.96	0.007
	Au (ppm)	167	7	0.01	3.10	0.24	0.099
z4_1	Length (m)	893	0	1.50	2.01	1.98	0.003
	Au (ppm)	794	99	0.01	10.00	0.39	0.431
z4_2	Length (m)	335	0	1.67	2.00	1.92	0.008
	Au (ppm)	335	0	0.01	1.88	0.18	0.071
z4_3	Length (m)	239	0	1.00	2.00	1.90	0.016
	Au (ppm)	237	2	0.01	10.00	0.22	0.483
z4_4	Length (m)	378	0	1.50	2.00	1.91	0.011
	Au (ppm)	378	0	0.01	10.00	0.41	0.470
z4_5	Length (m)	471	0	1.50	2.10	1.94	0.007
	Au (ppm)	471	0	0.01	7.04	0.52	0.626
z4_6	Length (m)	123	0	1.67	2.00	1.93	0.010
	Au (ppm)	123	0	0.01	3.44	0.35	0.291

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
z4_7	Length (m)	65	0	1.67	2.00	1.91	0.014
	Au (ppm)	65	0	0.01	1.51	0.31	0.127
z4_8	Length (m)	35	0	1.83	2.00	1.91	0.005
	Au (ppm)	35	0	0.02	2.03	0.57	0.202
z05	Length (m)	44	0	1.67	2.00	1.89	0.019
	Au (ppm)	44	0	0.01	1.54	0.30	0.099
z99	Length (m)	12,950	0	1.00	2.07	1.98	0.002
	Au (ppm)	12,664	286	0.01	10.15	0.05	0.020
Boin							
z1	Length (m)	572	0	1.83	2.20	1.97	0.002
	Au (ppm)	572	0	0.01	7.17	0.48	0.608
Z2	Length (m)	214	0	1.50	2.00	1.95	0.005
	Au (ppm)	214	0	0.01	2.68	0.27	0.211
Z3	Length (m)	111	0	1.50	2.08	1.91	0.015
	Au (ppm)	111	0	0.01	4.63	0.42	0.458
Z4	Length (m)	134	0	1.91	2.00	1.97	0.001
	Au (ppm)	134	0	0.01	4.73	0.46	0.598
Z5	Length (m)	4,672	0	1.00	2.04	1.98	0.002
	Au (ppm)	4,671	1	0.01	8.00	0.40	0.732
Z6	Length (m)	25	0	1.50	2.00	1.84	0.023
	Au (ppm)	25	0	0.01	1.01	0.23	0.078
Z99	Length (m)	18,239	0	1.00	2.50	1.98	0.002
	Au (ppm)	18,187	52	0.01	7.42	0.04	0.011
Nyam							
Z1	Length (m)	1,454	0	1.75	2.15	1.97	0.002
	Au (ppm)	1,449	5	0.01	8.23	0.66	0.680
Z2	Length (m)	21	0	1.50	2.00	1.76	0.032
	Au (ppm)	21	0	0.09	1.99	0.46	0.149
Z3	Length (m)	443	0	1.00	2.00	1.91	0.022
	Au (ppm)	443	0	0.01	10.00	0.51	0.962
Z4	Length (m)	363	0	1.00	2.00	1.90	0.021
	Au (ppm)	363	0	0.01	8.73	0.54	1.162
Z5	Length (m)	65	0	1.50	2.00	1.91	0.010
	Au (ppm)	64	1	0.01	10.00	1.12	3.570
Z99	Length (m)	4,499	0	1.00	2.15	1.97	0.006
	Au (ppm)	4,485	14	0.01	10.00	0.06	0.037

Zone	Field	No. Samples	No. Miss	Minimum	Maximum	Mean	Variance
Kwakyekrom							
Z1	Length (m)	215	0	1.50	2.00	1.92	0.013
	Au (ppm)	214	1	0.01	5.00	0.36	0.387
Z2	Length (m)	226	0	1.50	2.00	1.91	0.013
	Au (ppm)	226	0	0.00	5.00	0.50	0.667
Z3	Length (m)	329	0	1.50	2.00	1.95	0.006
	Au (ppm)	329	0	0.00	5.00	0.63	0.595
Z4	Length (m)	202	0	1.67	2.00	1.92	0.010
	Au (ppm)	202	0	0.01	5.00	0.38	0.422
Z5	Length (m)	157	0	1.67	2.00	1.92	0.009
	Au (ppm)	157	0	0.01	4.78	0.28	0.242
Z6	Length (m)	58	0	1.67	2.00	1.88	0.006
	Au (ppm)	58	0	0.01	0.97	0.17	0.046
Z7	Length (m)	10	0	2.00	2.00	2.00	-
	Au (ppm)	10	0	0.03	0.53	0.15	0.030
Z99	Length (m)	4,196	0	1.00	2.17	1.98	0.003
	Au (ppm)	4,041	155	0.00	2.28	0.05	0.011

14.6 Spatial Analysis

Variography, using Snowden Supervisor™ (v.2020) software, was completed for gold within Sewum, Boin, Nyam, and Kwakyekrom. Downhole variograms were used to determine nugget effect and then correlograms were modelled with two structures to determine spatial continuity in each of the zones. The variograms are not normalized. Table 14-6 summarizes results of the variography.

Zones with similar variograms within a deposit were assigned the same parameters. No viable variograms were generated at Kwakyekrom.

Table 14-6: Enchi Variogram Parameters

Zone	Angle 1	Angle 2	Angle 3	Nugget	ST1 PAR1	ST1 PAR2	ST1 PAR3	ST1 PAR4	ST2 PAR1	ST2 PAR2	ST2 PAR3	ST2 PAR4
Sewum												
Z1, Z3, Z5	130	90	180	0.15	50	35	10	0.15	78	50	15	0.7
Z2	135	80	175	0.35	40	35	10	0.15	60	45	30	0.5
Z4_1	-70	20	0	0.24	40	35	10	0.15	45	64	30	0.61
Z4_2, Z4_3, Z4_4, Z4_5, Z4_6, Z4_7, Z4_8	-70	40	-10	0.30	70	35	6	0.22	100	40	25	0.48
Boin												
All Zones	-60	60	20	0.20	30	25	10	0.20	50	30	15	0.6
Nyam												
Z1, Z2, Z4, Z5	120	80	180	0.32	41	12	7	0.12	52	27	16	0.56
Z3	120	80	180	0.10	40	30	10	0.45	55	45	15	0.45
Kwakyekrom												
No variogram	-	-	-	-	-	-	-	-	-	-	-	-

14.7 Mineral Resource Block Model

Individual block models were established in Datamine Studio RM for each of the zones using a separate parent model as the origin. The models were rotated to improve the efficiency of filling the solids with blocks. Drillhole spacing is variable with most of the surface drilling spaced at 25 m sections, and 25 to 75 m on sections. A block size of 10 m x 10 m x 10 m was selected to accommodate the nature of the mineralization and to be amenable for potential open pit mining.

The block model was sub-celled on a 2.5 m x 2.5 m x 2.5 m pattern allowing the parent block to be split in each direction to more accurately fill the volume of the wireframes, thus more accurately estimating the tonnes in the mineral resource.

Table 14-7 summarizes details of the parent block model.

Table 14-7: Enchi Parent Block Model

	Sewum	Boin	Nyam	Kwakyekrom
Origin X	518100	516500	527500	527500
Origin Y	625200	633500	634000	634000
Origin Z	-300	-150	-200	-200
Rotation (Z axis)	31	36	29	29
Block Size (m)	10 x 10 x 10	10 x 10 x 10	10 x 10 x 10	10 x 10 x 10
Sub-Cell (m)	2.5	2.5	2.5	2.5
No. Block (X)	300	300	300	300
No. Block (Y)	680	500	700	700
No. Block (Z)	80	50	45	45

Table 14-8 compares the difference between the volume of the solids and the volume of the block model. All the model volumes have less than 1% variance from the solid volumes with the exception of Kwakyekrom Zone 7, which is has a model volume 2.4% less than the wireframe solid.

Table 14-8: Enchi Solid vs. Model Comparison

Zone	Model Volume (m^3)	Wireframe Volume (m^3)	Difference (%)
Sewum			
Z1	1,779,875	1,777,928	-0.11
Z2	9,568,968	9,591,228	0.23
Z3	3,320,718	3,318,652	-0.06
Z4_1	10,708,843	10,717,769	0.08
Z4_2	3,030,187	30,30,717	0.02
Z4_3	3,274,156	3,274,157	0.00
Z4_4	2,828,281	2,832,215	0.14
Z4_5	5,195,281	5,196,696	0.03
Z4_6	1,990,062	1,990,181	0.01
Z4_7	1,290,093	1,289,349	-0.06
Z4_8	807,906	808,084	0.02
Z5	527,031	529,249	0.42
Boin			
Z01	3,064,343	3,065,939	0.05
Z02	2,498,593	2,493,423	-0.21
Z03	700,375	702,103	0.25
Z04	1,414,156	1,416,493	0.16
Z05	30,175,562	30,201,152	0.08
Z06	134,937	135,259	0.24
Nyam			
Z1	3,708,531	3,710,049	0.04
Z2	109,406	109,180	-0.21
Z3	1,212,843	1,210,539	-0.19
Z4	852,718	851,989	-0.09
Z5	158,468	157,500	-0.61
Kwakyekrom			
Z1	708,625	710,982	0.33
Z2	752,281	751,894	-0.05
Z3	1,099,156	1,098,877	-0.03
Z4	594,218	595,448	0.21
Z5	695,687	694,808	-13
Z6	430,375	430,592	0.05
Z7	98,593	101,105	2.38

14.7.1 Estimation Parameters

The interpolations of the zones were completed using the estimation methods: NN, ID^2, and OK. The estimations were designed for two or three passes. In each pass a minimum and maximum number of samples were required as well as a maximum number of samples from a borehole to satisfy the estimation criteria. The OK methodology is the method used to report the mineral estimate statement, except for Kwakyekrom, which reported estimated grades using ID^2. Table 14-9 summarizes the interpolation criteria for the zones.

Table 14-9: Enchi Estimation Parameters

Estimation No.	Search Ellipse Size	Min No. of Comp	Max No. of Comp	Max No. of Comp per Hole
Sewum Z1, Z5				
1	1	4	15	3
2	2	4	15	3
Sewum Z2				
1	1	4	15	3
2	2.5	4	15	3
Sewum Z3_1				
1	1	4	15	2
2	2.5	3	15	2
Sewum Z3_2				
1	1	3	15	2
2	2.5	3	15	2
Sewum Z4_2, Z4_3, Z4_4, Z4_5, Z4_6, Z4_7, Z4_8				
1	1	4	15	2
2	2	3	15	2
Sewum Z4_1				
1	1	3	15	2
2	2.5	3	15	2
Boin All Zones				
1	1	4	12	2
2	2	3	12	2
3	3	3	12	2
Nyam All Zones				
1	1	4	12	3
2	1.75	3	12	3
Kwakyekrom All Zones				
1	1	4	15	3
2	2	4	15	3

Table 14-10 summarizes the search ellipse size and rotations for each of the zones.

Table 14-10: Enchi Search Ellipse Parameters

Zone	SDIST 1 (m)	SDIST 2 (m)	SDIST 3 (m)	Axis 1 Rotation Dip Direction	Axis 2 Rotation Dip	Axis 3 Rotation Plunge
Sewum Z1						
	80	50	15	130	90	180
Sewum Z2						
	60	45	30	135	80	175
Sewum Z 3_1						
	80	50	15	125	90	180
Sewum Z3_2						
	80	50	15	75	90	480
Sewum Z4_2, Z4_3, Z4_4, Z4_5, Z4_6, Z4_7, Z4_8						
	100	40	25	-70	40	-10
Sewum Z4_1						
	100	40	25	-60	25	-10
Sewum Z5						
	80	50	15	-80	90	180
Boin Z01, Z02, Z03, Z05, Z06						
	50	30	10	129	-75	0
Boin Z04						
	50	30	10	145	90	0
Nyam All Zones						
	55	45	15	120	80	180
Kwakyekrom All Zones						
	80	50	15	120	75	180

14.8 Mineral Resource Classification

Several factors are considered in the definition of a mineral resource classification:

- NI 43-101 requirements;
- Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines, 2019;
- Author's experience with shear-hosted gold deposits and in particular the Enchi Project;
- Spatial continuity based on variography of the assays within the drillholes;
- Drillhole spacing and estimation runs required to estimate the grades in a block;
- The uncertainty in the drillhole collar elevations; and
- Lack of Specific Gravity reading on the Project.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction (CIM, 2094). An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

Currently, based on the factors listed above and the definitions provided by CIM, the mineral resources at Sewum, Boin, Nyam, and Kwakyekrom are classified as Inferred.

No environmental, permitting, legal, title, taxation, socio-economic, marketing, or other relevant issues are known to the QP that may affect the estimate of mineral resources.

14.9 Mineral Resource Statement

The Mineral Resource Statement, effective as of June 8, 2021, has been tabulated in terms of a gold cut-off grade.

Each mineral resource is constrained within a pit using Deswik software (2020.01), which runs the pseudoflow algorithm to determine the potential economic pit limits. Table 14-11 summarizes the input parameters for the pit shells.

Table 14-11: Enchi Pit Parameters by Deposit

Item	Unit	Value
Mining Cost (contractor)		
Variable Mining Cost:		
Mining cost oxide	$/t mined	1.40
Mining cost transition	$/t mined	2.10
Mining cost fresh rock	$/t mined	2.60
Incremental haulage cost	$/t mined/bench	0.05
Reference level for incremental haulage cost:		
Sewum	m	110
Boin	m	70
Nyam	m	30
Kwakyekrom	m	30
Fixed Mining Cost:		
Allowance for contractor and owner fixed costs	$/t milled	1.00
Processing Cost		
Processing cost	$/t milled	5.18
General and administration cost	$/t milled	0.65

Item	Unit	Value		
Gold Price Model				
Gold price	$/oz	1,650		
Royalties (Ghana and Maverix)	% of metal price	7		
Refining Charges, Doré Transport and Insurance	$/oz	4		
Discount rate	%	8		
Overall Pit Slopes Angle		**Oxide**	**Transition**	**Fresh rock**
Sewum	degree	33	35	48
Boin	degree	28 - 40	35 - 43	50
Nyam	degree	30	32	48
Kwakyekrom	degree	30	35	46
Recovery		**Oxide**	**Transition**	**Fresh rock**
Sewum				
Sewum Extension.	%	72	82	75
Sewum Checkerboard Hill.	%	70	80	70
Sewum Ridge Top	%	75	85	75
Boin				
Boin South	%	67	72	62
Boin Central	%	80	85	75
Boin North	%	75	85	65
Nyam				
Nyam	%	60	65	55
Kwakyekrom				
Kwakyekrom	%	70	80	65

The pit constrained mineral resource statement for each of the zones at Enchi is tabulated in Table 14-12 for the Inferred mineral resources.

Table 14-12: Pit Constrained Mineral Resource Cut-off Table

Cut-off (g/t)	Tonnes	Gold (g/t)	Gold (ounces)
Sewum			
0.1	46,695,000	0.5	750,700
0.2	**41,009,000**	**0.55**	**725,200**
0.3	31,491,000	0.64	648,000
0.4	22,013,000	0.76	537,900
0.5	16,729,000	0.9	484,100

Cut-off (g/t)	Tonnes	Gold (g/t)	Gold (ounces)
Boin			
0.1	27,058,000	0.61	530,700
0.2	**21,807,000**	**0.72**	**504,800**
0.3	17,300,000	0.84	467,200
0.4	13,715,000	0.97	427,700
0.5	11,067,000	1.09	387,800
Nyam			
0.1	5,245,000	0.78	131,500
0.2	**4,892,000**	**0.82**	**129,000**
0.3	4,524,000	0.87	126,500
0.4	4,080,000	0.93	122,000
0.5	3,519,000	1.00	113,000
Kwakyekrom			
0.1	3,212,000	0.56	57,800
0.2	**2,703,000**	**0.64**	**55,600**
0.3	2,235,000	0.72	51,700
0.4	1,811,000	0.8	46,600
0.5	1,374,000	0.92	40,600

Based on the assumptions in previous Table 14-11, a 0.2 g/t gold cut-off is deemed suitable for this Enchi mineral resource statement. Table 14-13 is a summary of the pit constrained mineral resource statement.

Table 14-13: Enchi Mineral Resource Statement

Deposit	Tonnes	Gold (g/t)	Gold (ounces
Sewum	41,009,000	0.55	725,200
Boin	21,807,000	0.72	504,800
Nyam	4,892,000	0.82	129,000
Kwakyekrom	2,703,000	0.64	55,600
Total	**70,411,000**	**0.62**	**1,414,600**

14.10 Validation

14.10.1 Visual Inspection

The visual comparison of the block model grades against composite grades for each of the zones in each deposit show a reasonable correlation between the values. No significant discrepancies were apparent from the sections reviewed, yet grade smoothing is apparent in some locations due to the distance between drill samples being broader in some regions.

14.10.2 Swath Plots

Swath plots of easting, northing and elevations were generated for each mineralized zone respectively. These plots are comparing the OK estimates with the NN and ID^2 estimates and the drillhole composites.

14.10.3 Global Comparison

The overall block model statistics for the OK model were compared to the overall ID^2 and NN model values as well as the composite capped drillhole data. Table 14-14 shows this comparison of the global estimates for the estimation method calculations. In general, there is agreement between the OK model, ID^2 model, and NN model. Larger discrepancies are reflected as a result of lower drill density in some portions of the model. There is a degree of smoothing apparent when compared to the diamond drill statistics. Comparisons were made using all blocks at a 0 g/t cut-off.

Table 14-14: Enchi Global Statistics Comparison

Zone	Composite (g/t)	OK (g/t)	ID^2 (g/t)	NN (g/t)
Sewum				
z01	0.720	0.582	0.555	0.545
z02	0.380	0.417	0.428	0.426
z03	0.240	0.327	0.338	0.436
z4_1	0.390	0.384	0.389	0.440
z4_2	0.180	0.202	0.206	0.191
z4_3	0.220	0.231	0.225	0.230
z4_4	0.410	0.371	0.369	0.401
z4_5	0.520	0.485	0.484	0.451
z4_6	0.350	0.246	0.244	0.274
z4_7	0.310	0.268	0.264	0.277
z4_8	0.570	0.479	0.489	0.563
z05	0.300	0.297	0.279	0.324

Zone	Composite (g/t)	OK (g/t)	ID^2 (g/t)	NN (g/t)
Boin				
z1	0.480	0.481	0.485	0.512
Z2	0.270	0.257	0.252	0.235
Z3	0.420	0.446	0.464	0.509
Z4	0.460	0.513	0.524	0.519
Z5	0.400	0.420	0.418	0.443
Z6	0.230	0.234	0.231	0.278
Nyam				
Z1	0.660	0.540	0.529	0.558
Z2	0.460	0.452	0.448	0.421
Z3	0.510	0.549	0.535	0.569
Z4	0.540	0.654	0.617	0.815
Z5	1.120	0.988	1.022	0.926
Kwakyekrom				
Z1	0.360	-	0.435	0.456
Z2	0.500	-	0.498	0.562
Z3	0.630	-	0.645	0.555
Z4	0.380	-	0.421	0.392
Z5	0.280	-	0.325	0.357
Z6	0.170	-	0.153	0.107
Z7	0.150	-	0.176	0.196

14.11 Previous Estimates

Newcore commissioned WSP to generate a mineral resource estimate in 2020 (McCracken, 2020). The 2020 estimate was based on the premise that the mineral resource could potentially be extracted using traditional open pit methods.

Table 14-15 illustrates the differences in the 2020 mineral resource statement with the current mineral resource statement from 2021.

Table 14-15: Comparison with Previous Mineral Resource Statement

	BBA 2021 Mineral Resource Statement			WSP 2020 Mineral Resource Statement		
Deposit	**Tonnes**	**Gold (g/t)**	**Gold (ounces)**	**Tonnes**	**Gold (g/t)**	**Gold (ounces)**
Sewum	41,009,000	0.55	725,200	27,600,000	0.60	532,400
Boin	21,807,000	0.72	504,800	19,837,000	0.84	535,700
Nyam	4,892,000	0.82	129,000	5,489,000	0.88	155,300
Kwakyekrom	2,703,000	0.64	55,600	-	-	-
Total	**70,411,000**	**0.62**	**1,414,600**	**52,926,000**	**0.72**	**1,223,400**

The difference between the 2020 mineral resource model and the 2021 mineral resource model is largely due to the following items:

- A gold price of $1,500 was used in 2020 to determine cut-off grade versus $1,650 in 2021;
- A gold cut-off grade of 0.3 g/t was used in 2020 versus 0.2 g/t gold cut-off used in 2021;
- Addition of the Kwakyekrom mineral resource estimate;
- 186 additional drillholes;
- Re-interpretation of mineral solids; and
- Specific gravity values based on oxidation profile.

15. MINERAL RESERVE ESTIMATES

Since this report summarizes the results of a Preliminary Economic Assessment (PEA), no Mineral Reserves have been estimated for the Enchi gold deposits as per NI 43-101 guidelines. The subset of Mineral Resources with the open pit designs are presented in Chapter 16.

16. MINING METHODS

The Project contains multiple deposits that will be mined using conventional open pit mining methods (drill, blast, load and haul) with the mining operations being outsourced to a mining contractor. Newcore will provide supporting technical services and mine management.

The Project considers the mineral resources that have been estimated for the Sewum, Boin, Nyam and Kwakyekrom deposits. Mining will be carried out in a series of 10 open pits across the four deposits. Sewum and Boin were split into four mining areas each. Drilling and blasting will be carried out on 10-m benches. The oxide formation will be free-dug, thus not requiring drilling and blasting. For the transition formation, it has been assumed that 50% of the material will be free-dug, while 50% will require drilling and blasting. All of the fresh rock will be drilled and blasted. Mining will be carried out in 5-m flitches using a hydraulic excavator in a backhoe configuration. The excavator will sit on top of the production bench and the 64-tonne haul trucks will be loaded on the level below.

The mining sequence will begin with the clearing, removal and stockpiling of any usable soil, and the construction of pit protection berms and diversion drains. Non-economic oxide material will be hauled to oxide waste rock facilities, while the non-economic transition and fresh rock will be hauled to additional waste rock facilities.

At Sewum and Boin, the mineralized material mined will be hauled either directly to the primary crusher at the heap leach facility (HLF) or to one of the low grade stockpiles that will be located close to each pit. Low grade stockpile rehandling will be done with a fleet of wheel loaders and haul trucks. The majority of the mineralized mined will be directly dumped into the primary crusher and the rest will be placed in stockpiles located near the run-of-mine (ROM) pad. For safety reasons, mineralized material mined from Nyam and Kwakyekrom will only be hauled on day shift.

The open pits will operate at a rate of 56,000 t/d for both mineralization and waste rock and have an overall strip ratio for the Project of 2.1:1. The mine production schedule is based on two 12-hour shifts, seven days a week for a total of 360 days per year (five days of weather delays have been considered per year). Production drilling will be done on the day shift only.

Standard auxiliary equipment will be used to maintain the pits, roads and rock piles, and to prepare blast patterns for drilling.

16.1 Resource Block Model

The mine design for the PEA is based on the four mineral resource block models that were prepared by BBA for each deposit, as presented in Chapter 14 of this report. The 3-dimensional sub-blocked models are composed of parent blocks that are 10 m x 10 m x 10 m high, sub-celled on a 2.5 m x 2.5 m x 2.5 m pattern and contain only Inferred Mineral Resources. The block model coordinates are in the Universal Transverse Mercator 84 World Geodetic Coordinate System (UTM WGS 84) for Ghana. The block model extents, dimensions, and rotation angles are presented in Chapter 14 of this report.

Table 16-1 presents the items in the block model that were used for the mine design.

Table 16-1: Block Models Items

Item	Description
General	
VOLUME	Fill Volume in m^3
DENSITY	In situ dry density in t/m^3 (2.20 = Oxide, 2.45 = Transitions, 2.70 = Fresh Rock)
WEATHER	Material type (3 = Oxide, 2 = Transitions, 1 = Fresh Rock)
TONORE	Mineralized Material Tonnes in Block
TONWST	Waste Tonnes in Block
ORE	Leach Feed Code (1 = Leach Feed, 0 = Waste)
COG	Cut-off grade (g/t)
METREC	Metallurgical Recovery (%)
Sewum	
ZONE	Mineralized Zones (1, 2, 3, 32, 41, 42, 43, 44, 45, 46, 47, 48, 5 and 99)
AU_OK	Gold grade (g/t)
METZONE	Metallurgical Recovery Zone
Boin	
ZONE	Mineralized Zones (1, 2, 3, 4, 5, 6 and 99)
AU_UNCAP	Gold grade (g/t)
METZONE	Metallurgical Recovery Zone
Nyam	
ZONE	Mineralized Zones (1, 2, 3, 4, 5 and 99)
AU_OK	Gold grade (g/t)
Kwakyekrom	
ZONE	Mineralized Zones (1, 2, 3, 4, 5, 6, 7 and 99)
AU_ID	Gold grade (g/t)

Even though Rock Type 99 has estimated grades, it was deemed a host rock and was not considered as potential heap leach feed for this PEA.

16.2 Mine Dilution and Mining Recovery

Considering the mining equipment that will be used for the Project, mining dilution and mining recovery were evaluated by converting the sub-blocked models into regularized mining block models with block sizes of 5 m x 5 m x 5 m. For Sewum and Boin, the re-blocking process introduced an average mining dilution of 4% with a gold grade of 0.07 g/t and a mining recovery of

97%. For Nyam and Kwakyekrom, mining dilution was estimated at 12% with a gold grade of 0.10 g/t and a mining recovery of 94%, as a result of having narrower mineralized veins compared with Sewum and Boin. These global factors were applied as flat rates for each deposit in the PEA.

No geological modelling was carried out to complete this process.

16.3 Pit Optimization

A pit optimization analysis was conducted to determine the cut-off grades and the extent to which each deposit can be mined profitably. The pit optimization was carried out using the Deswik software (version 2020.01), which runs the pseudoflow algorithm to determine the economic pit limits based on inputs of mining and processing costs, revenue per block and operational parameters such as the gold recovery, pit slopes and other imposed physical constraints.

Since the Project is at a PEA level, NI 43-101 guidelines and the Canadian Institute of Mining, Metallurgy and Petroleum's Definition Standards for Mineral Resources and Mineral Reserves allow Inferred Mineral Resources to be used in the pit optimization analysis and mine plan. Table 16-2 presents the parameters that were used for the pit optimization analysis.

The cost inputs were based on budgetary quotations provided by contractors and BBA's experience with similar studies. The costs and operating parameters that were used are preliminary estimates for developing the economic pit and are different from the operating costs subsequently developed in this PEA and presented in Chapter 21.

A geotechnical study for pit slopes has yet to be completed for the Project. Pit geometry configuration inputs were therefore based on engineering experience with similar studies. BBA adjusted the slope to account for the haul ramps that will be added during the pit design process.

Mining dilution and mining recovery were considered in the analysis as flat rates, as discussed in Section 16.2. No constraining boundaries were used for the pit optimization analysis.

Table 16-2: Pit Optimization Inputs

Item	Unit	Value
Mining Cost (contractor)		
Variable Mining Cost:		
Mining cost oxide	$/t mined	1.40
Mining cost transition	$/t mined	2.10
Mining cost fresh rock	$/t mined	2.60
Incremental haulage cost	$/t mined per 10 m bench	0.05

Item	Unit	Value		
Reference level for incremental haulage cost:				
Sewum	m	110		
Boin	m	70		
Nyam	m	30		
Kwakyekrom	m	30		
Fixed Mining Cost:				
Allowance for contractor and owner fixed costs	$/t milled	1.00		
Processing Cost				
Processing cost	$/t milled	5.18		
General and administration cost	$/t milled	0.65		
Gold Price Model				
Gold price	$/oz	1,650		
Royalties (Ghana and Maverix)	% of metal price	7		
Refining Charges, Doré Transport and Insurance	$/oz	4		
Discount rate	%	8		
Overall Pit Slopes Angle		**Oxide**	**Transition**	**Fresh rock**
Sewum	degree	33	35	48
Boin	degree	28 - 40	35 - 43	50
Nyam	degree	30	32	48
Kwakyekrom	degree	30	35	46

The gold extraction was determined from the preliminary metallurgical leach test work completed to date on the Sewum, Boin, Nyam and Kwakyekrom deposits, which consisted of a series of bottle roll tests completed on representative samples. The calculation of the metallurgical gold recoveries is presented in more detail in Chapter 13. Table 16-3 presents the gold extractions that were used for the pit optimization by weathering and metallurgical zone. This analysis was completed prior to the receipt of the preliminary results from the column test work and updated figures were used for the final economic model.

Table 16-3: Gold Extraction

Item	Unit	Value		
Weathering Zone		**Oxide**	**Transition**	**Fresh rock**
Sewum				
Sewum Ext.	%	72	82	75
Sewum CH.	%	70	80	70
Sewum Ridge	%	75	85	75

Item	Unit	Value		
Boin				
Boin South	%	67	72	62
Boin Central	%	80	85	75
Boin North	%	75	85	65
Nyam				
Nyam	%	60	65	55
Kwakyekrom				
Kwakyekrom	%	70	80	65

Using the input parameters presented above and varying the gold price from $495/oz to $2,145/oz (Revenue Factor of 0.3 to 1.3), the algorithm outputs a series of nested pit shells for each deposit. The Net Present Value (NPV) for each pit shell was then calculated from the discounted cash flows (DCF) and analyzed using the industry standard pit-by-pit graph.

The NPV values allow for relative comparison of the different pit shells. It should be noted that these values are hypothetical in nature, as they are not derived from a detailed mining plan nor do they include any capital or sustaining costs. Figure 16-1 to Figure 16-4 present the results of the analysis for Sewum, Boin, Nyam and Kwakyekrom, which include the selected shells that were used for the pit designs.

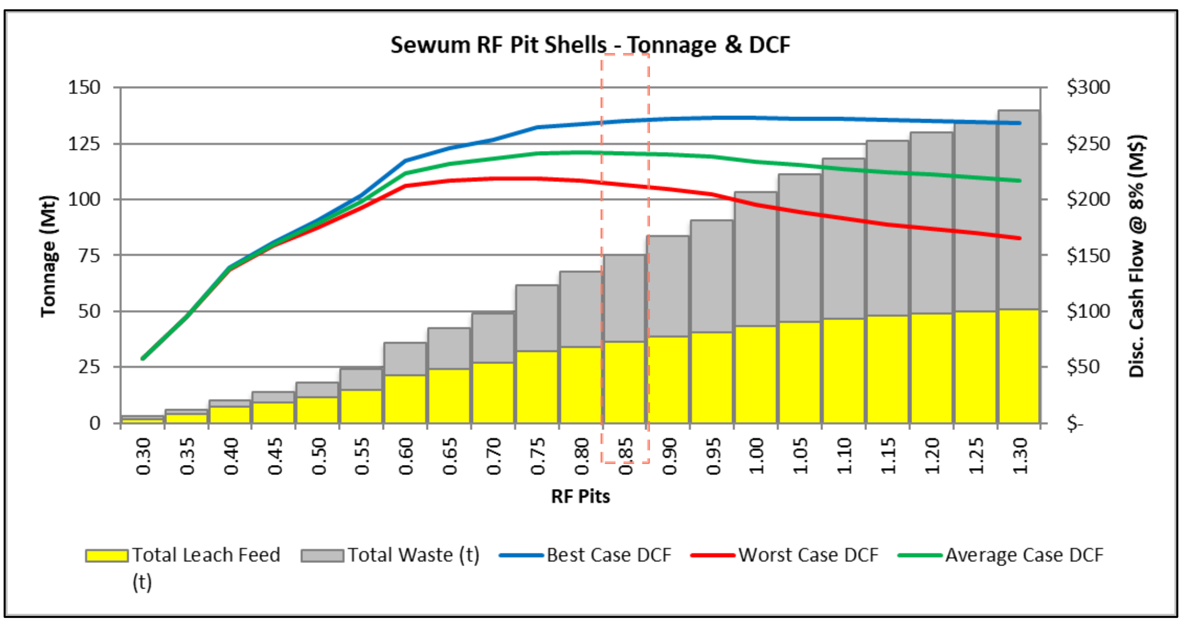


Figure 16-1: Sewum Pit Optimization Results

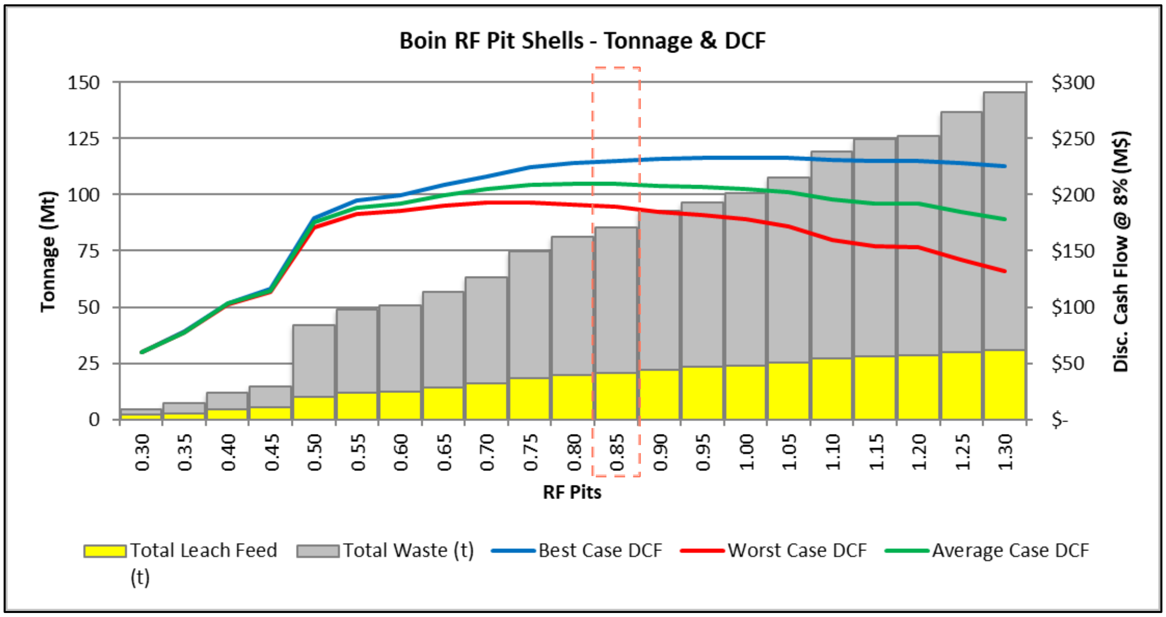


Figure 16-2: Boin Pit Optimization Results

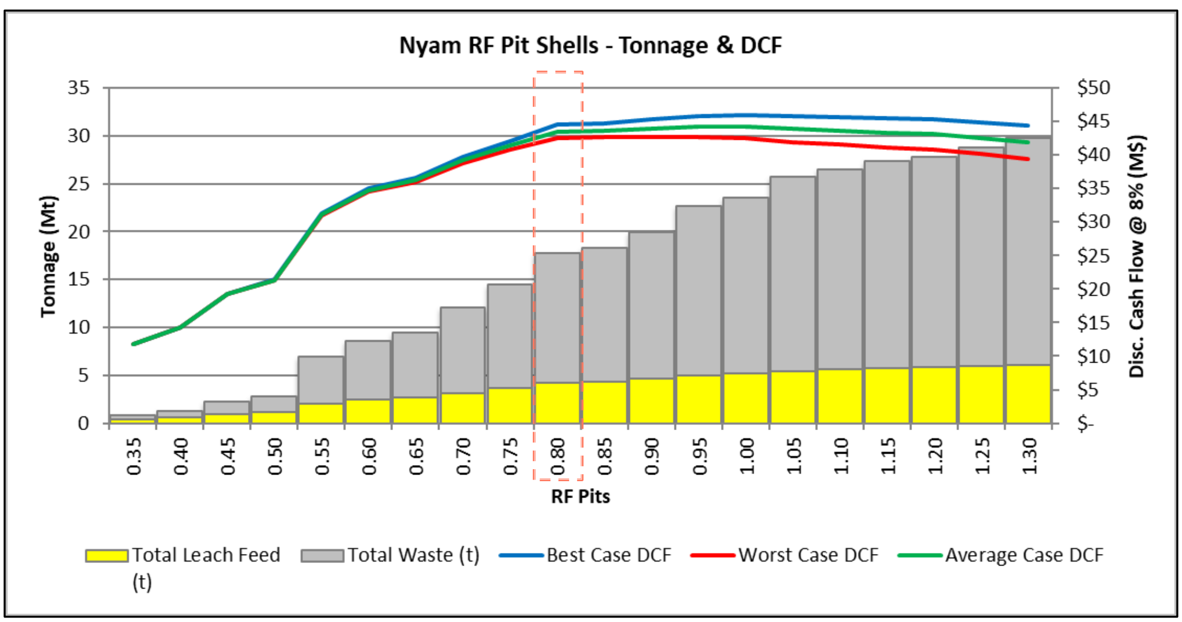


Figure 16-3: Nyam Pit Optimization Results

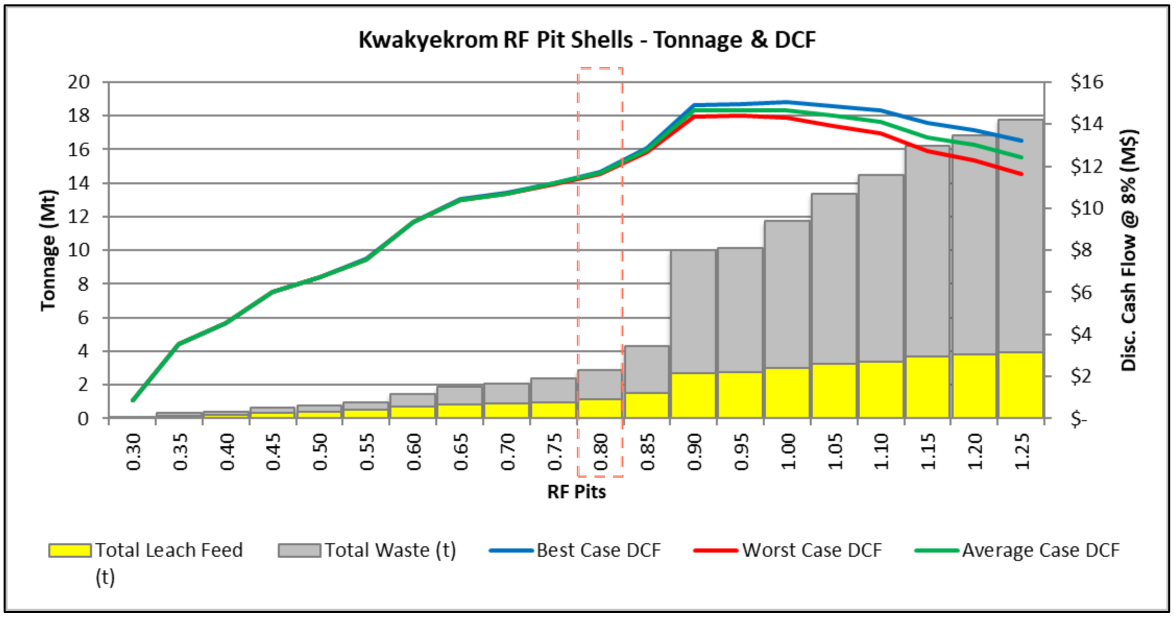


Figure 16-4: Kwakyekrom Pit Optimization Results

Table 16-4 presents the mineral resources within the selected pit shells for each deposit, which consider mining dilution and mining recovery. Pit shells were chosen based on the following parameters:

- Undiscounted cash flows;
- NPV's at the 8% discount rate; and
- Incremental stripping ratio.

For Nyam and Kwakyekrom, it has been decided to select smaller shells since the longer transportation to the HLF will incur additional costs.

Table 16-4: Pit Optimization Results

Description	Units	Sewum	Boin	Nyam	Kwakyekrom
Selected pit shell	#	13	13	11	12
Associated RF	-	0.90	0.90	0.80	0.85
Gold price according to RF	$/oz	1,485	1,485	1,320	1,402
Inferred	Mt	38.5	22.2	4.3	1.5
Au	g/t	0.53	0.68	0.76	0.55
Waste	Mt	45.0	70.5	13.4	2.7
Strip Ratio	-	1.2	3.2	3.15	1.8

16.4 Cut-off Grade

A cut-off grade has been calculated for each deposit and weathering zone using the economic parameters presented in Table 16-2 and the gold recovery presented in Table 16-3. The cut-off grades range from 0.14 to 0.24 g/t Au.

16.5 Open Pit Design

The following section presents the design criteria and resulting subset of mineral resources within the open pit designs that were used as a basis for the mine production plan. The pit designs use the optimized pit shells as a guideline and include smoothing the pit walls, adding ramps to access the pit bottom and ensure that the pits can be mined safely and efficiently using the selected equipment.

16.5.1 Geotechnical Pit Slope Parameters

There were no geotechnical investigations available at the time the Project was conducted. Table 16-5 and Figure 16-5 present the geotechnical pit slope parameters that were used for the pit designs based on engineering experience with similar studies.

Table 16-5: Geotechnical Pit Slope Parameters

Item	Unit	Value
Bench Height		
Oxide	m	10
Transition	m	10
Fresh	m	10
Benching Arrangement		
Oxide	m	Single
Transition	m	Single
Fresh	m	Double
Berm Width		
Oxide	m	7
Transition	m	7
Fresh	m	9
Bench Face Angle (BFA)		
Oxide	degrees	65
Transition	degrees	70
Fresh	degrees	75

Item	Unit	Value
Inter-Ramp Angle (IRA)		
Oxide	degrees	40.6
Transition	degrees	43.2
Fresh	degrees	54.3

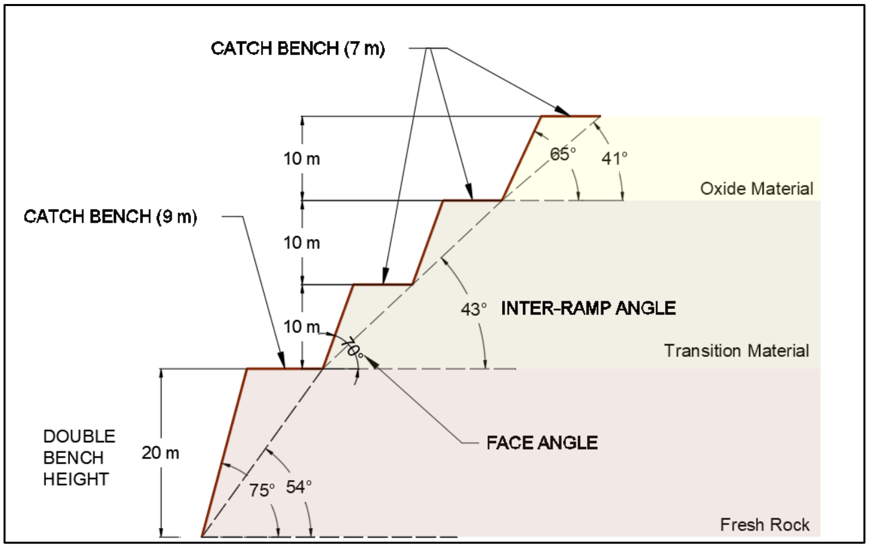


Figure 16-5: Pit Wall Configuration

16.5.2 Hydrogeological and Hydrological Assessment

A hydrogeological study has yet to be complete for the Project. An allowance was provided for mine dewatering in the contractor budgetary quotes.

16.5.3 Haul Road Design

The ramps and haul roads were designed for haulage with 64-tonne rigid frame mining trucks, with an overall width of 21 m. For double lane traffic, industry practice indicates the minimum running surface width to be three times the width of the largest truck. The overall width of a 64-tonne haul truck is 5.7 m, which results in a running surface of 17.1 m. The allowance for berms and ditches increases the overall haul road width to 21 m. A maximum ramp grade of 10% was used. Figure 16-6 presents a typical section of the in-pit ramp design.

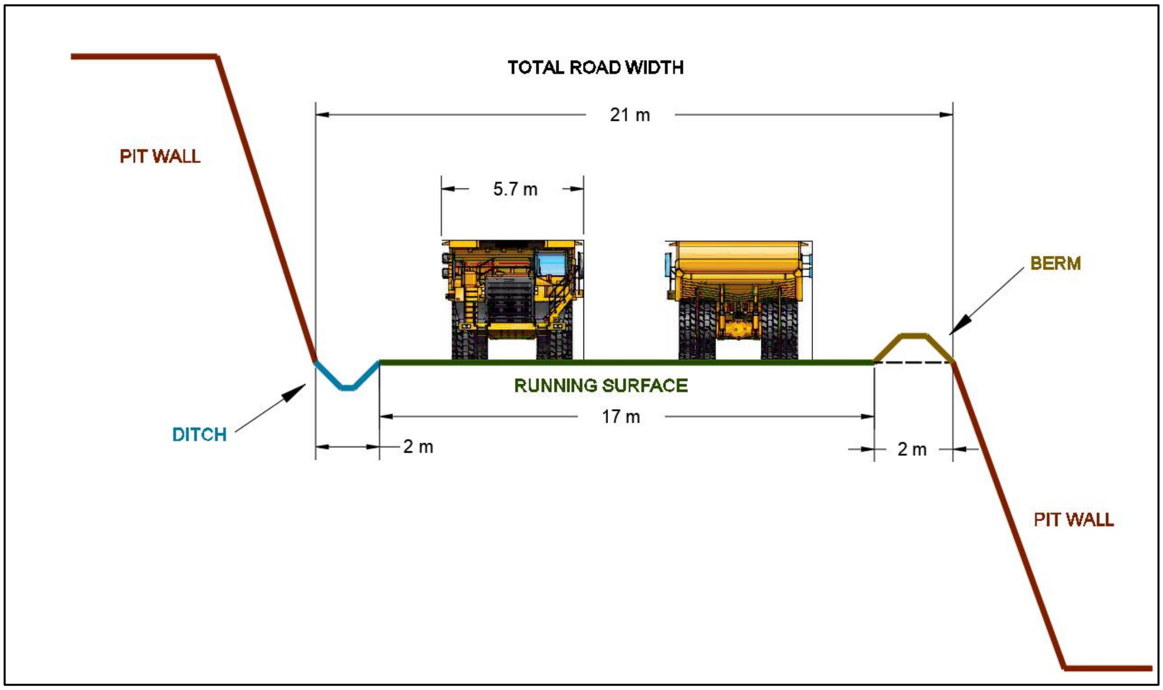


Figure 16-6: Ramp Design

16.5.4 Subset of Mineral Resources within the Open Pit Designs

Mining be carried out in a series of 10 open pits across the four deposits with depths ranging from approximately 20 to 180 m and with a total surface area of 2.15 Mm^2. The open pits contain 68.6 Mt of mineralized material at an average gold grade of 0.57 g/t with a stripping ratio of 2.1:1. Table 16-6 presents the subset of mineral resources within the pit designs, which considers cut-off grades ranging from 0.14 to 0.24 g/t gold, depending on the rock type and deposit. It is important to note that this PEA includes Inferred Mineral Resources. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 16-7 presents the approximate dimensions of the pits including the surface area. The pits are completely contained within the mining allotments. Figure 16-7 to Figure 16-10 present plan views of the 10 open pits including four for Sewum, four for Boin, one for Nyam and one for Kwakyekrom.

Table 16-6: Subset of Mineral Resources Within the Pit Designs (1)(2)

Item	Tonnage (k tonnes)	Gold Grade (g/t)	Contained Ounces (k ounces)	Strip Ratio
Sewum	39,750	0.50	645	1.3:1
Boin	22,646	0.65	472	3.3:1
Nyam	4,520	0.73	107	3.2:1
Kwakyekrom	1,650	0.52	28	2.0:1
Total	**68,566**	**0.57**	**1,252**	**2.1:1**

(1) Including mining dilution and mining recovery.
(2) Numbers may not add due to rounding.

Table 16-7: Approximate Pit Dimensions

Item	Length (m)	Width (m)	Depth (m)	Surface Area ('000 m^2)
Sewum	500 – 1,500	70 – 700	20 – 140	976
Boin	450 – 2,000	120 – 450	50 – 180	842
Nyam	1,700	80-250	50-120	261
Kwakyekrom	550	130	50-70	72

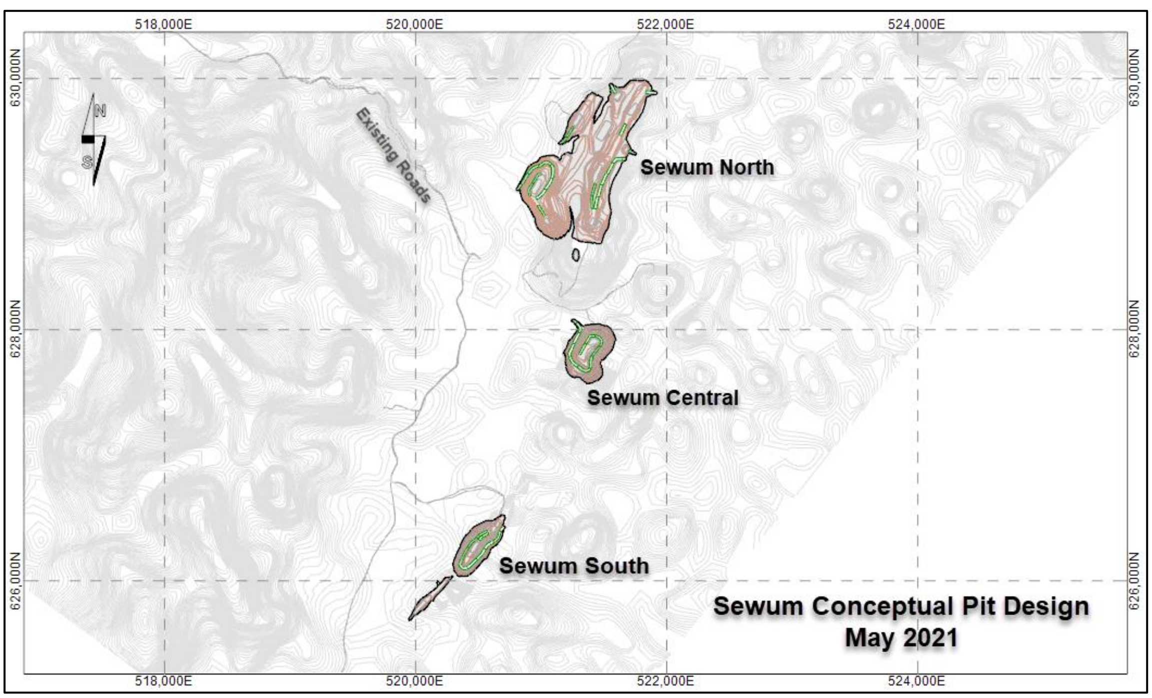


Figure 16-7: Sewum Pit Design

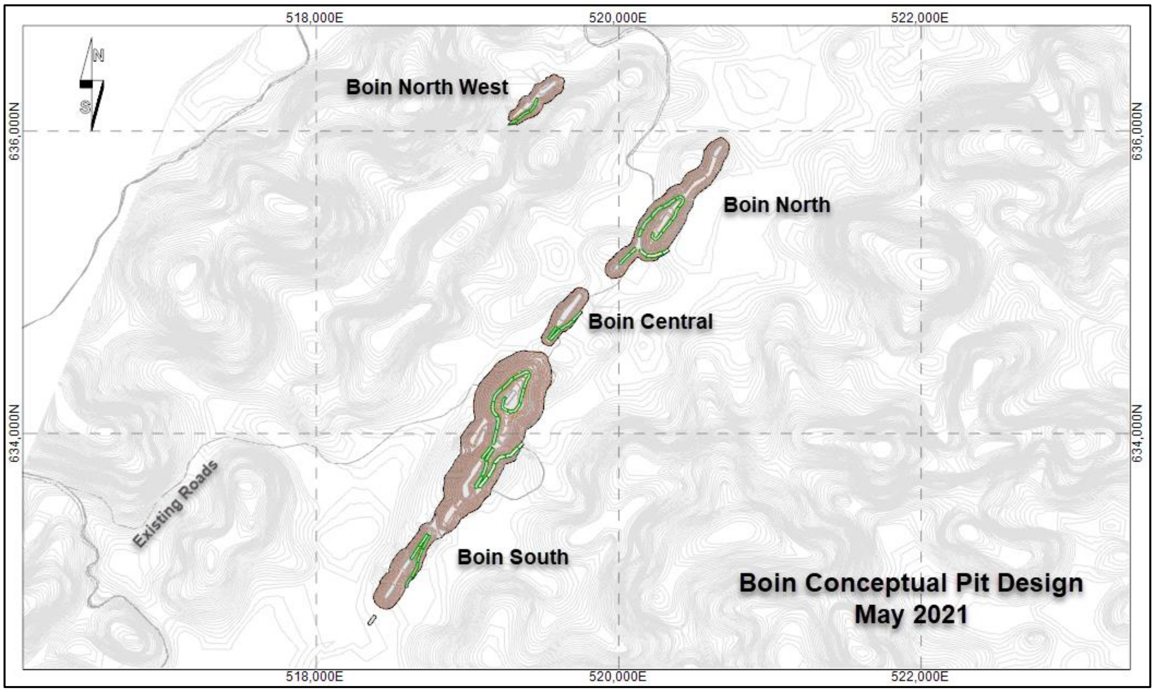


Figure 16-8: Boin Pit Design

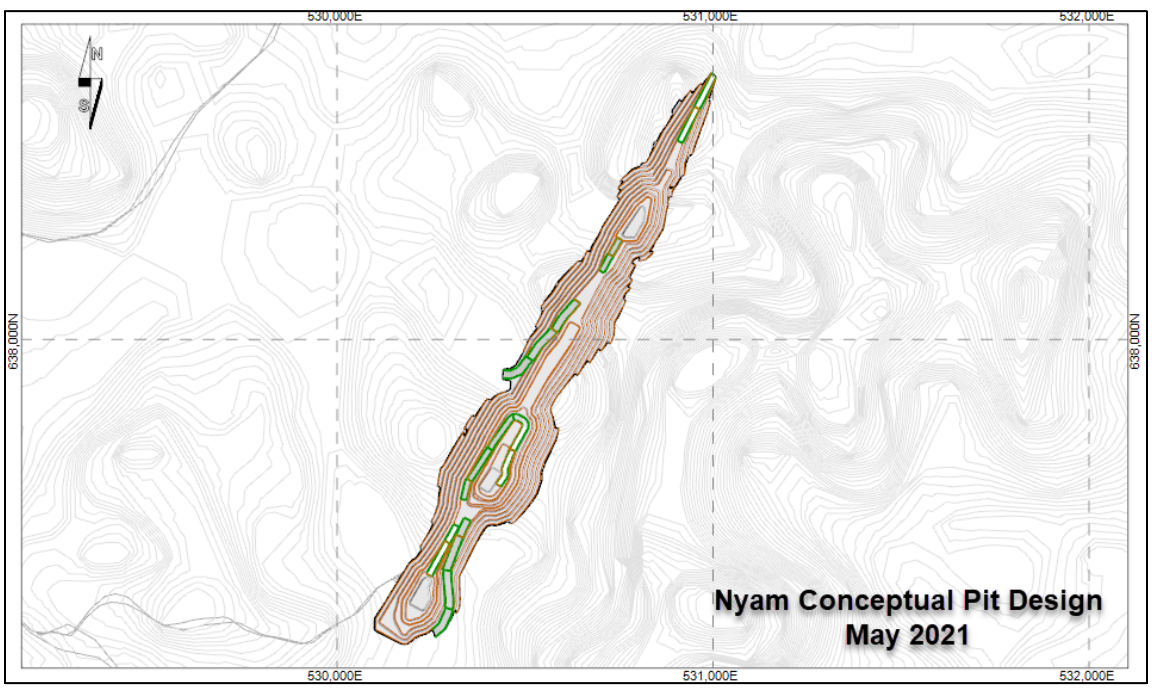


Figure 16-9: Nyam Pit Design

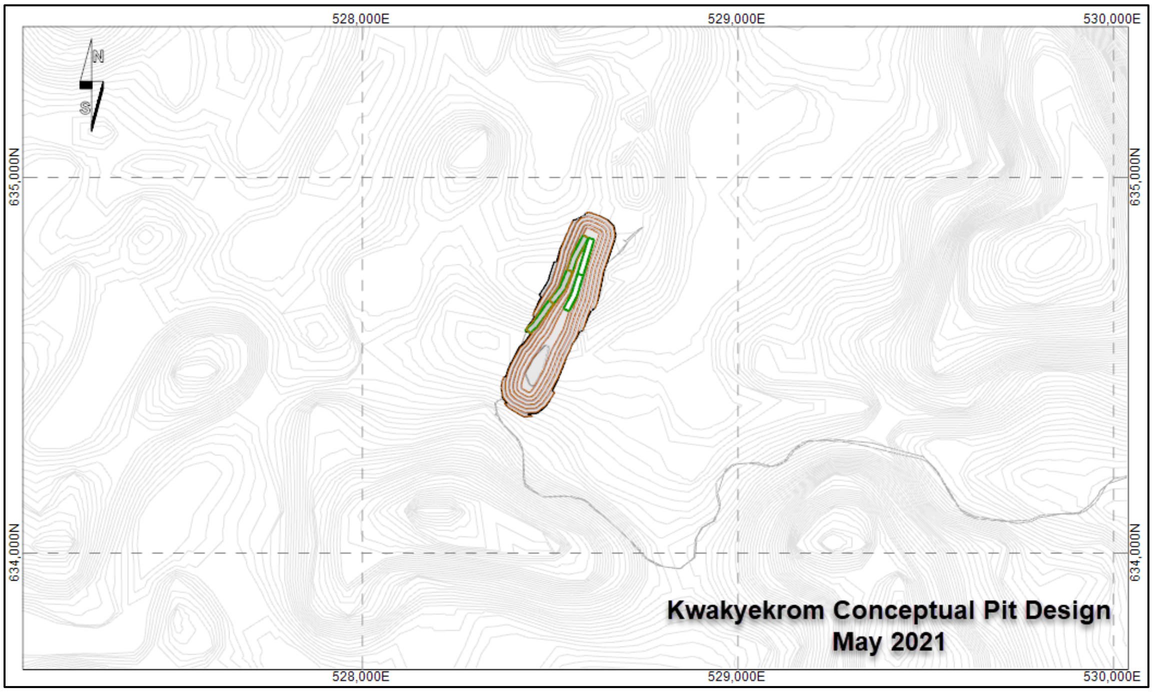


Figure 16-10: Kwakyekrom Pit Design

16.6 Mine Production Schedule

The following section discusses the mine plan that was prepared for the PEA and used as the basis for the mine capital and operating cost estimate presented in Chapter 21. The mine plan was completed using Hexagon's MinePlan Schedule Optimizer (MPSO) tool with the objective of maximizing NPV.

The mine plan focuses on achieving consistent production rates, mining of the larger deposits that are located in closer proximity to the HLF early in the schedule, and balancing gold grades and strip ratios. The mine plan considers a ramp up of 75% of capacity for the HLF in the first year of production. Since the mineralization is close to surface, very little pre-production waste stripping is required.

The mine plan considers a vertical advance rate (bench sink rate) of no more than nine benches (90 m) per year for each pit. Mining at Nyam was intentionally delayed until Year 6 at the earliest due to road construction requirements and the longer haul distance to the HLF.

Mine development at Nyam will begin in Year 6 and operations at that deposit will run for three years. Mining at Kwakyekrom will start in Year 8 and last for a year. This deposit will provide a source of production in case there are operational issues in other active pits. Year 8 is the only year where all four deposits will be actively mined.

The mine plan results in an approximate 11-year mine life, which delivers a total of 69 Mt of mineralized material with an average gold grade of 0.57 g/t to the HLF and approximately 143 Mt of waste rock to the storage facilities located near each mine. The maximum annual mining capacity reaches 22 Mtpy between Years 5 and 8.

Table 16-8 presents the pit sequencing of the life of mine schedule per deposit. Table 16-9 presents the mine production schedule and Table 16-10 and Table 16-11 present the material that will be sent to the HLF.

Table 16-8: Pit/Phase Sequence for the Life of Mine Plan

Deposit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Sewum	North	North	-	-	North	North	North	North	North	-	North
	Central	Central	Central	Central	Central	-	-	-	-	-	-
	South	South	-	South	South	-	-	-	-	-	-
Boin	North	-	-	North	North	North	-	North	North	North	North
	Central	-	-	Central			Central	Central			
	South	South	South	South	South	-	South	South	South	South	South
	-	-	-	-	-	North West	North West	-	-	-	-
Nyam	-	-	-	-	-	Nyam	Nyam	Nyam	-	-	-
Kwakyekrom	-	-	-	-	-	-	-	Kwak	-	-	-

Figure 16-11 presents a chart showing the gold production each year as well as the gold grade. The average gold production between Year 2 and Year 5 remains fairly consistent at 104,000 ounces per year. Gold production averages of 83,000 ounces between Years 6 and 9, reaching a maximum of 121,000 ounces in Year 10. The average annual gold grade remains fairly close to the average of the subset of mineral resources within the open pit designs of 0.57 g/t, ranging from 0.45 g/t to 0.73 g/t.

Figure 16-12 presents a chart showing the leach feed tonnages each year as well as the overall gold recovery. The average annual leach feed remains fairly consistent at 6.6 Mtpy. It should be noted that the gold recoveries for Sewum and Boin were adjusted based on recent column test results that were completed after the pit optimization analysis. The adjustment resulted in an increased gold recovery of 3.3% for Sewum and 1.8% for Boin deposits, resulting in an 80.4% overall gold recovery for both deposits.

Figure 16-13 presents the material movement by destination and Figure 16-14 presents the material movement by deposit.

Table 16-9: Enchi Mine Production Schedule (1)(2)

Description	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
ROM to Leach Pad	**k tonnes**	**4,866**	**6,253**	**6,576**	**6,487**	**6,576**	**6,630**	**6,668**	**6,770**	**6,576**	**6,576**	**3,333**	**67,311**
Boin	k tonnes	518	475	1,996	1,382	2,054	955	1,541	1,371	3,209	6,576	2,390	22,468
Sewum	k tonnes	4,348	5,778	4,580	5,105	4,522	4,771	2,925	2,335	3,367	-	943	38,673
Nyam	k tonnes	-	-	-	-	-	904	2,202	1,414	-	-	-	4,520
Kwakyekrom	k tonnes	-	-	-	-	-	-	-	1,650	-	-	-	1,650
ROM to Stockpile	**k tonnes**	**411**	**-**	**-**	**156**	**688**	**-**	**-**	**-**	**-**	**-**	**-**	**1,255**
Boin	k tonnes	89	-	-	46	43	-	-	-	-	-	-	178
Sewum	k tonnes	323	-	-	110	645	-	-	-	-	-	-	1,078
Nyam	k tonnes	-	-	-	-	-	-	-	-	-	-	-	-
Kwakyekrom	k tonnes	-	-	-	-	-	-	-	-	-	-	-	-
Waste Rock	**k tonnes**	**9,059**	**13,225**	**13,300**	**12,949**	**14,741**	**14,915**	**15,284**	**14,988**	**13,328**	**11,733**	**9,966**	**143,490**
Boin	k tonnes	2,570	2,556	11,325	6,680	6,061	5,537	8,410	5,525	8,996	11,733	6,210	75,603
Sewum	k tonnes	6,489	10,669	1,975	6,269	8,680	4,919	2,261	697	4,332	-	3,756	50,048
Nyam	k tonnes	-	-	-	-	-	4,459	4,613	5,458	-	-	-	14,531
Kwakyekrom	k tonnes	-	-	-	-	-	-	-	3,308	-	-	-	3,308
Total Material Moved	**k tonnes**	**14,337**	**19,801**	**19,876**	**19,681**	**22,005**	**21,545**	**21,992**	**21,757**	**19,904**	**18,309**	**14,103**	**213,311**
Boin	k tonnes	3,177	3,031	13,321	8,197	8,158	6,492	9,991	6,896	12,205	18,309	8,650	98,427
Sewum	k tonnes	11,159	16,770	6,555	11,484	13,848	9,691	5,186	3,032	7,699	-	5,453	90,876
Nyam	k tonnes	-	-	-	-	-	5,363	6,815	6,872	-	-	-	19,050
Kwakyekrom	k tonnes	-	-	-	-	-	-	-	4,958	-	-	-	4,958
Total ROM	**k tonnes**	**14,337**	**19,478**	**19,876**	**19,593**	**22,005**	**21,545**	**21,952**	**21,757**	**19,904**	**18,309**	**13,299**	**212,055**
Boin	k tonnes	3,177	3,031	13,321	8,109	8,158	6,492	9,951	6,896	12,205	18,309	8,601	98,249
Sewum	k tonnes	11,159	16,447	6,555	11,484	13,848	9,691	5,186	3,032	7,699	-	4,698	89,799
Nyam	k tonnes	-	-	-	-	-	5,363	6,815	6,872	-	-	-	19,050
Kwakyekrom	k tonnes	-	-	-	-	-	-	-	4,958	-	-	-	4,958
Stripping Ratio		**1.7**	**2.1**	**2.0**	**1.9**	**2.0**	**2.2**	**2.3**	**2.2**	**2.0**	**1.8**	**3.0**	**2.1**

(1) Including mining dilution and mining recovery.
(2) Numbers may not add due to rounding.

Table 16-10: Enchi Leach Feed Production Schedule [1][2]

Description	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Leach Feed	**k tonnes**	**4,866**	**6,576**	**6,576**	**6,576**	**6,576**	**6,630**	**6,708**	**6,770**	**6,576**	**6,576**	**4,137**	**68,566**
Boin	k tonnes	518	475	1,996	1,470	2,054	955	1,581	1,371	3,209	6,576	2,439	22,646
Sewum	k tonnes	4,348	6,101	4,580	5,105	4,522	4,771	2,925	2,335	3,367	-	1,698	39,750
Nyam	k tonnes	-	-	-	-	-	904	2,202	1,414	-	-	-	4,520
Kwakyekrom	k tonnes	-	-	-	-	-	-	-	1,650	-	-	-	1,650
Grade	**g/t**	**0.41**	**0.63**	**0.63**	**0.63**	**0.58**	**0.48**	**0.55**	**0.52**	**0.45**	**0.73**	**0.63**	**0.57**
Boin	g/t	0.39	0.58	0.78	0.63	0.72	0.67	0.49	0.49	0.46	0.73	0.79	0.65
Sewum	g/t	0.41	0.63	0.57	0.62	0.51	0.39	0.42	0.44	0.45	-	0.39	0.50
Nyam	g/t	-	-	-	-	-	0.75	0.77	0.68	-	-	-	0.73
Kwakyekrom	g/t	-	-	-	-	-	-	-	0.52	-	-	-	0.52
Ounces Produced	**oz**	**50,660**	**104,638**	**104,034**	**104,286**	**103,728**	**79,964**	**87,639**	**83,384**	**79,647**	**121,387**	**63,929**	**983,296**
Boin	oz	5,304	7,263	41,067	24,108	40,350	15,923	20,822	18,131	39,746	121,387	45,914	380,015
Sewum	oz	45,356	97,374	62,967	80,179	63,377	50,951	34,044	26,703	39,901	-	18,015	518,866
Nyam	oz	-	-	-	-	-	13,090	32,774	18,587	-	-	-	64,451
Kwakyekrom	oz	-	-	-	-	-	-	-	19,963	-	-	-	19,963
Overall Gold Recovery	**%**	**80%**	**78%**	**78%**	**79%**	**85%**	**78%**	**74%**	**74%**	**83%**	**79%**	**77%**	**79%**
Boin	%	82%	82%	82%	81%	84%	78%	84%	85%	84%	79%	74%	80%
Sewum	%	80%	78%	75%	78%	85%	85%	86%	81%	82%	0%	84%	80%
Nyam	%	-	-	-	-	-	60%	60%	60%	-	-	-	60%
Kwakyekrom	%	-	-	-	-	-	-	-	72%	-	-	-	72%

(1) Including mining dilution and mining recovery.
(2) Numbers may not add due to rounding.

Table 16-11: Enchi Leach Feed Production Schedule – Weathering Zone (1)(2)

Description	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Leach Feed	**k tonnes**	**4,866**	**6,576**	**6,576**	**6,576**	**6,576**	**6,630**	**6,708**	**6,770**	**6,576**	**6,576**	**4,137**	**68,566**
Oxide	k tonnes	518	475	1,996	1,470	2,054	955	1,581	1,371	3,209	6,576	2,439	22,646
Transition	k tonnes	4,348	6,101	4,580	5,105	4,522	4,771	2,925	2,335	3,367	-	1,698	39,750
Fresh Rock	k tonnes	-	-	-	-	-	904	2,202	1,414	-	-	-	4,520

(1) Including mining dilution and mining recovery.
(2) Numbers may not add due to rounding.

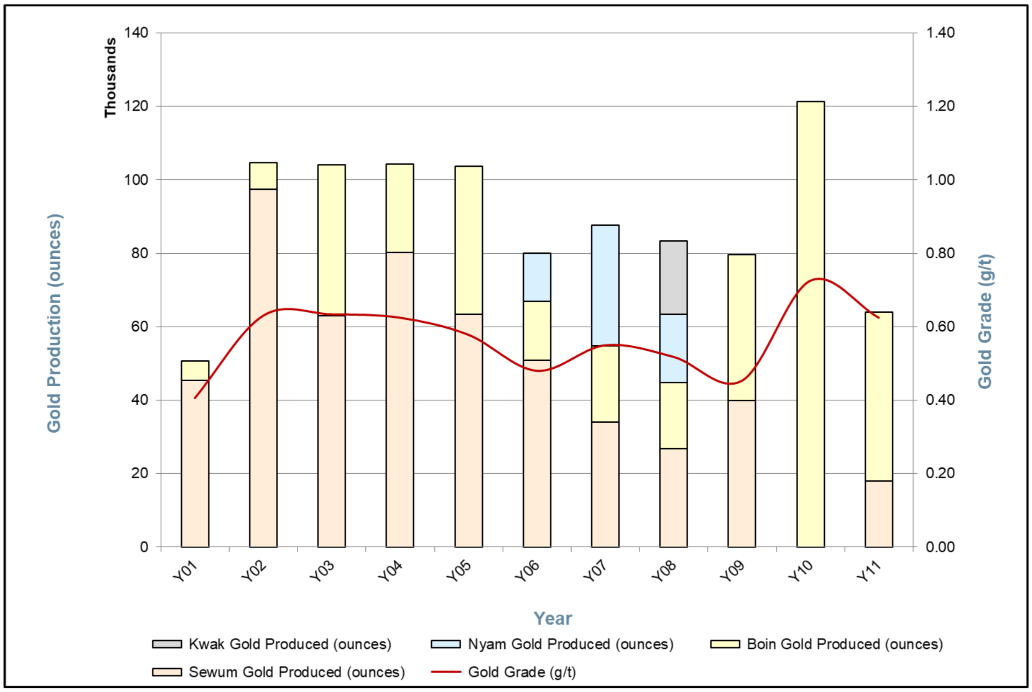


Figure 16-11: Gold Production

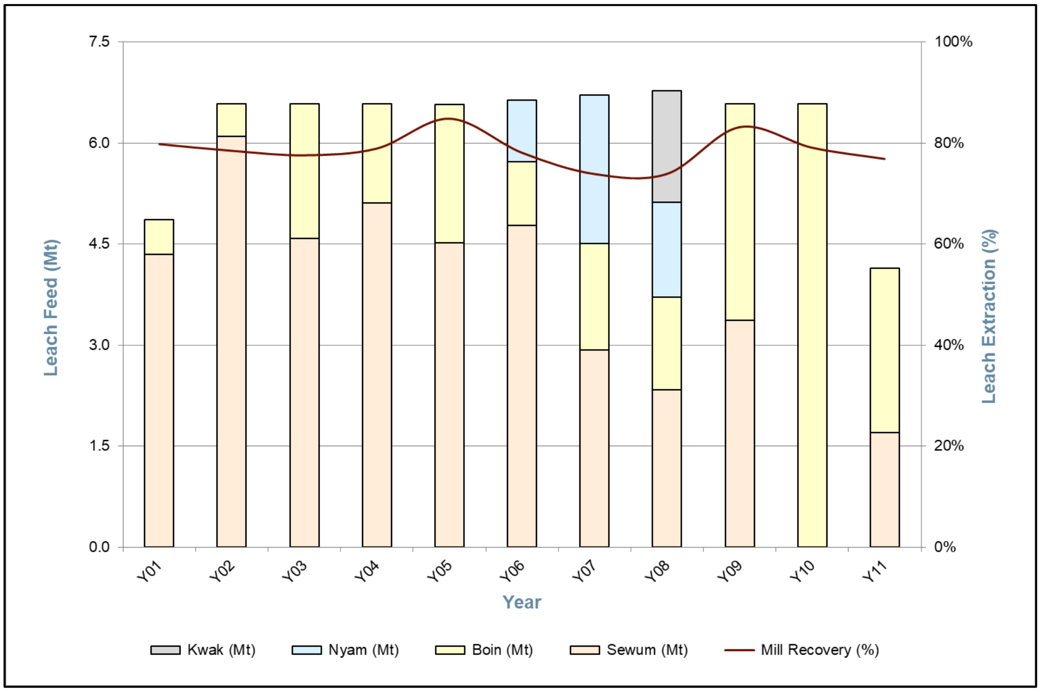


Figure 16-12: Heap Leach Feed

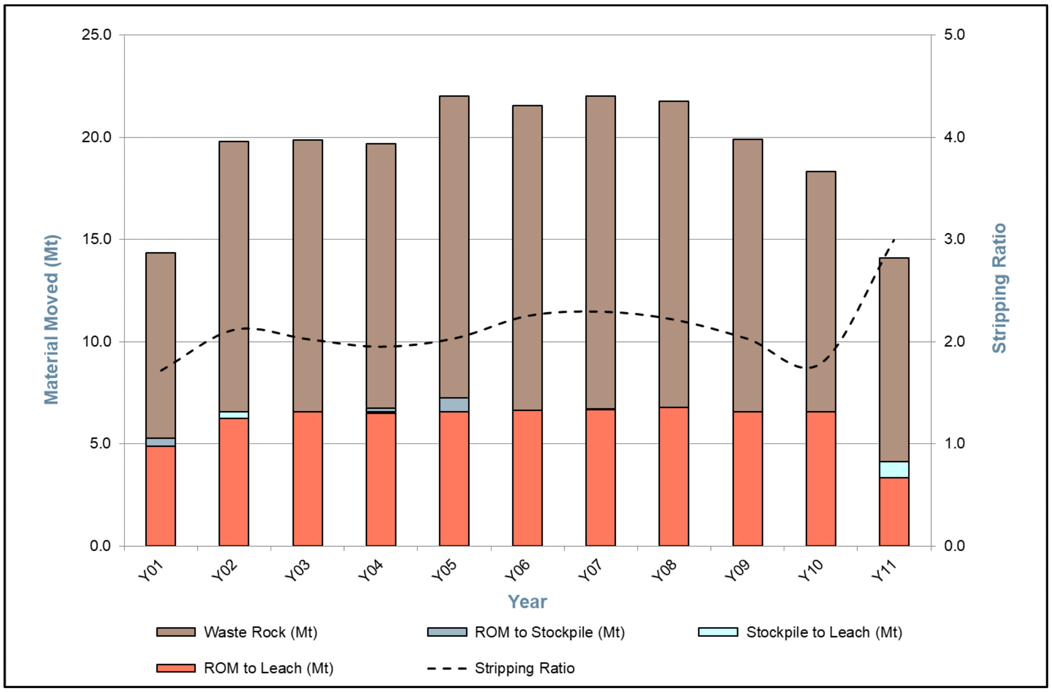


Figure 16-13: Total Material Moved

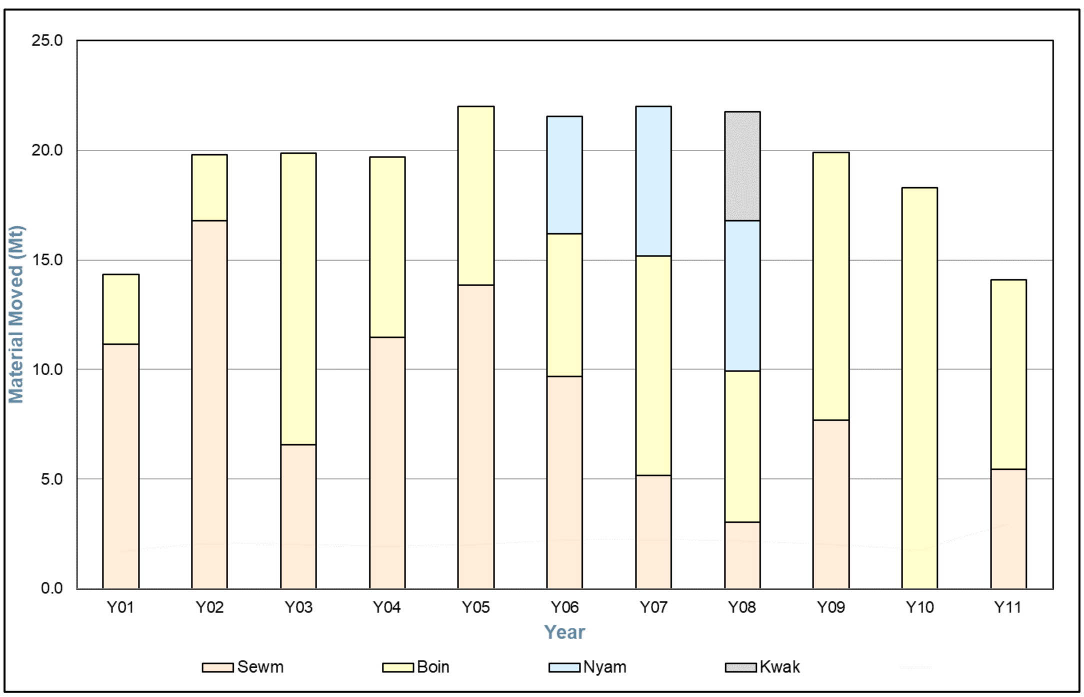


Figure 16-14: Total Material Moved by Pit

16.7 Waste Rock Facilities and Low Grade Stockpiles

Waste rock generated from the open pits will be stockpiled in several waste rock facilities. To ensure the long term stability of the waste rock facilities, each deposit will have two waste rock facilities, one for oxide and a second one for transition and fresh rock.

No geotechnical study for the stability of the waste rock facilities has yet been completed; therefore, BBA has considered typical geometric configurations for the design and separated the oxide, transition and fresh rock, which are presented in Table 16-12.

Table 16-12: Waste Rock Facilities Design Basis

Item	Unit	Value
Swell Factor:		
Oxide/Transition	LCM/BCM	1.2
Fresh Rock	LCM/BCM	1.3
Individual Lift Height	m	20
Bench Face Angle	degree	37.5
Bench Width	m	15
Road Width (dual lane)	m	21
Maximum Ramp Grade	%	10

A swell factor of 1.2 was used for oxide and transition material and 1.3 for fresh rock, which are typical values used in open pit mines for the waste rock facilities as well as equipment fleet requirements. The volume requirements for the various areas of the waste rock facilities are presented in Table 16-13. Note that in-pit backfilling was considered to reduce haulage distances and to minimize the overall environmental footprint of the Project. A total of 30% of the waste rock is planned to be placed as in-pit backfilling.

Table 16-13: Waste Rock Disposal Requirements

Item	Waste Material Tonnage (M tonnes)	Volume Required (Mm^3)
Sewum		
Oxide	44.0	6.5
Transition/ Fresh Rock	6.0	6.7
In-Pit Backfilling	-	12.4

Item	Waste Material Tonnage (M tonnes)	Volume Required (Mm^3)
Boin		
Oxide	69.3	27.7
Transition/ Fresh Rock	6.2	10.5
In-Pit Backfilling	69.4	1.5
Nyam		
Oxide	13.7	3.7
Transition/ Fresh Rock	0.8	1.1
In-Pit Backfilling	13.7	2.9
Kwakyekrom		
Oxide	2.7	0.3
Transition/ Fresh Rock	0.6	1.3
In-Pit Backfilling	2.7	-

Figure 16-15 to Figure 16-18 present layouts showing the location of the Sewum, Boin, Nyam and Kwakyekrom waste rock facilities.

Leach feed will be delivered directly to the HLF; however, two small low grade stockpiles for Sewum and Boin will be used for the purposes of mine planning to maximize the NPV of the Project. The low grade material will be re-handled to the HLF later in the mine life. A maximum of approximately 1,100 kt and 180 kt will be stockpiled from Sewum and Boin respectively. Refer to the general site layout (Figure 18-1) for the location of these stockpiles.

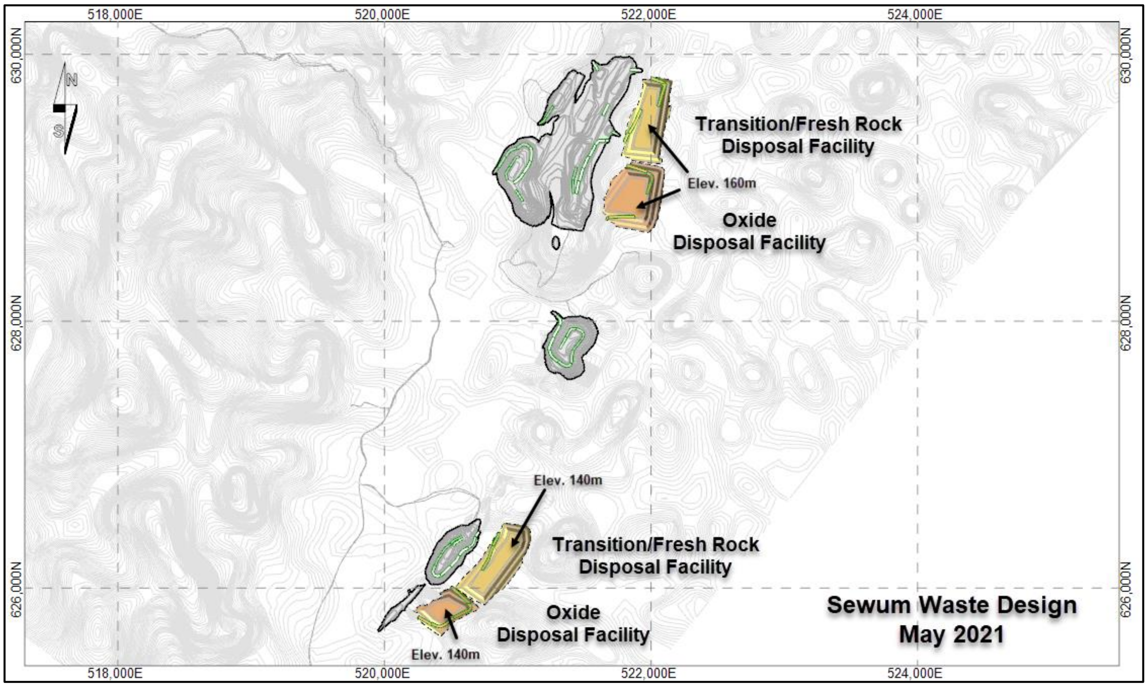


Figure 16-15: Sewum Waste Rock Facilities

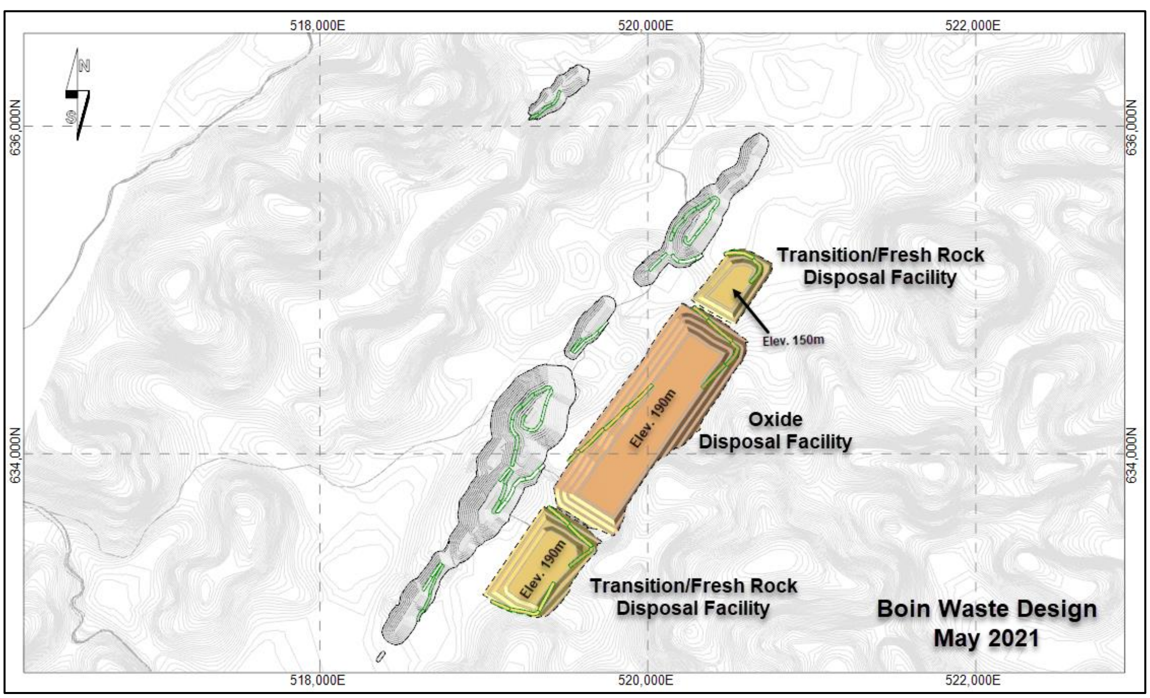


Figure 16-16: Boin Waste Rock Facilities

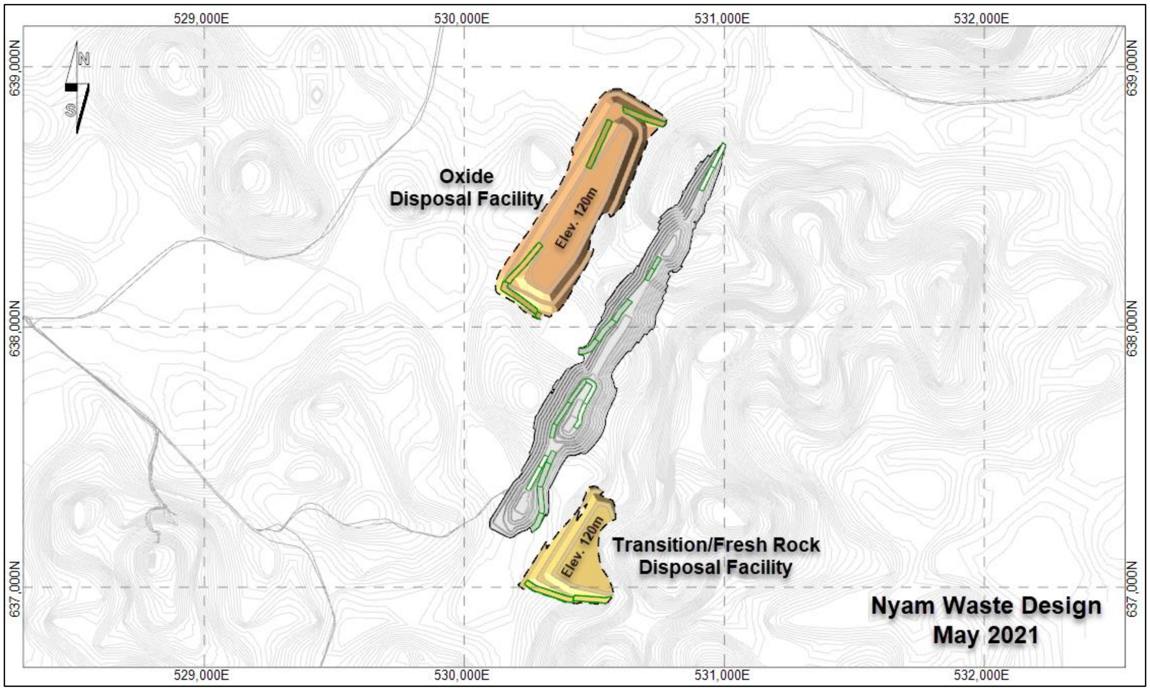


Figure 16-17: Nyam Waste Rock Facilities

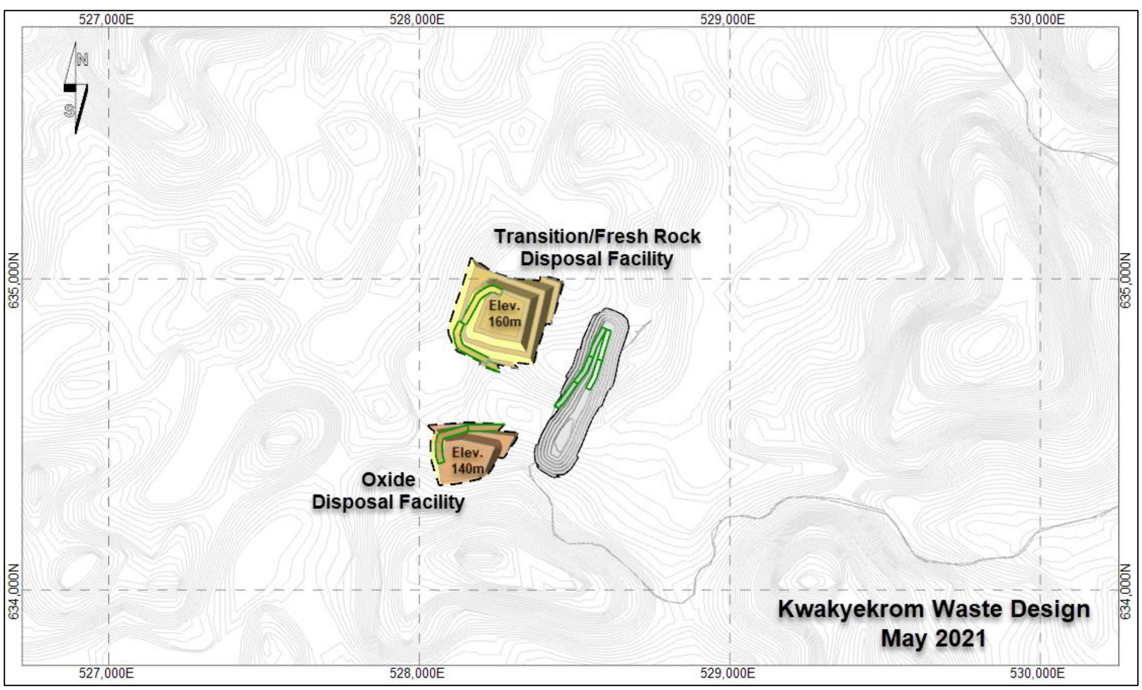


Figure 16-18: Kwakyekrom Waste Rock Facilities

16.8 Mining Methodology

Mining will be carried out with a local contract miner with technical services and mine management being done by Newcore personnel. BBA has nonetheless completed equipment and workforce calculations in order to give the reader an appreciation for the scale of the mining operation. The following sections of the report describe the main mining production activities, the parameters considered and the equipment requirements.

The mine production schedule is based on two 12-hour shifts, seven days a week for a total of 360 days per year (five days of weather delays have been considered per year). Production drilling will be done on the day shift only.

Considering the longer haul distances from the Nyam and Kwakyekrom deposits to the HLF, loading and hauling of the mineralized material is planned to be conducted only during the day shift; however, mining of the waste rock will be done both day and night. Table 16-14 presents the main key performance indicators (KPI) metrics for the production equipment, which are based on experience from similar mining operations.

Table 16-14: Production Equipment KPI Metrics (h/y)

KPI	Haul Truck	Excavator	Wheel Loader	Drill
Calendar Time	8,760	8,760	8,760	4,380
Available Time	7,446	7,446	6,570	3,285
Utilized Time	6,410	6,258	5,509	2,822
Operating Time	5,633	4,696	4,961	2,074

16.8.1 Drilling and Blasting

The production drill that has been selected is a top hammer rig designed for mid-sized open pits that can drill 150 mm (6”) holes and is also well suited for wall control and development work.

It is assumed that 50% of the transition material and 100% of the fresh rock will need to be drilled and blasted and that the remainder of the transition material and all the oxide material will be mined by ripping and dozing or by free digging.

Table 16-15 presents the drilling and blasting parameters that have been designed for the PEA.

Table 16-15: Drilling and Blasting Parameters

Description	Unit	Transition	Fresh Rock
Bench Height	M	10	10
Blast hole Diameter	Mm	150	150
Burden	M	5	4
Spacing	M	5	5
Subdrilling	M	1.0	1.2
Stemming	M	2.0	2.0
Powder Factor	kg/t	0.22	0.26

Considering a penetration rate of 21 m/h, according to the production plan, a maximum of six production drills will be required in Year 10, with an average of four during the majority of the mine life.

16.8.2 Loading

Material will be loaded into rigid frame trucks with hydraulic excavators equipped as backhoes. A 6.6-m^3 bucket capacity was assumed when estimating the loading fleet requirements based on the operating hours required to achieve the production schedule.

Material re-handled from stockpiles will be loaded by a 6.4-m^3 bucket capacity loader, which will act as an alternate loading unit when the excavators are unavailable. Table 16-16 presents the productivities for each equipment based on the material type.

Table 16-16: Loading Equipment Productivities

Description	Unit	Excavator	Loader
Oxide	t/GOH*	1,053	938
Transition	t/GOH*	1,308	1,166
Fresh rock	t/GOH*	1,321	1,178

* GOH = Utilized time

A total of four excavators and one wheel loader are required during the life of mine.

16.8.3 Hauling

Material will be transported by rigid frame haul trucks with a payload of 64 tonnes.

In order to calculate the fleet size, haul routes were generated for each period of the mine plan. Haul routes were imported in Talpac©, a commercially available truck simulation software package that BBA has validated with mining operations. Talpac© calculated the travel time required for a 64-tonne haul truck to complete each route. Table 16-17 shows the various components of a truck's cycle time. The load time is calculated using an excavator with a 6.6-m^3 (13.4-tonne) bucket as the loading unit. This size excavator, which is discussed in Section 16.8.2, loads oxide material in a 64-tonne haul truck in six passes, and in five for transition and fresh rock.

Table 16-17: Truck Cycle Time

Description	Second
Spot @ Excavator	42
Load Time (1)	165
Travel Time	Calculated by Talpac©
Spot at Dump	30
Dump Time	42

(1) Five passes at 33 sec/pass.

A moisture content of 5% was used for oxide and transition material and 2% for fresh rock for haul truck payload calculations. Haul productivities were calculated for each haul route using the truck payload and cycle time. Table 16-18 shows the cycle time and productivity for the mineralized material and waste haul routes for Boin in Year 3 as an example.

Table 16-18: Truck Productivities (Boin - Year 3)

Material	Cycle Time (min)					Productivity	
	Travel	Spot	Load	Dump	Total	Loads/h	t/h
Mineralized Material / Oxide	16.40	1.20	3.42	0.70	21.72	2.76	166
Waste / Transition	6.01	1.20	2.75	0.70	10.66	5.63	337

Truck hour requirements were then calculated by applying the tonnages hauled to the productivity for each haul route.

Table 16-19 represents the average one-way haul distance for the life of mine.

Table 16-19: Average One-Way Haulage Distance for Life of Mine

Description	Haulage Distance (km)
Sewum	
ROM to HLF	4.0
ROM to Stockpile	3.8
Waste Rock	1.9
Stockpile Rehandled	2.9
Boin	
ROM to HLF	5.7
ROM to Stockpile	2.4
Waste Rock	1.9
Stockpile Rehandled	2.9
Nyam	
ROM to HLF	18.9
Waste Rock	2.4
Kwakyekrom	
ROM to HLF	14.2
Waste Rock	1.3

A fleet of 12 trucks is required during Year 1, ramping up to a maximum of 26 during Year 8.

16.9 Auxiliary Equipment

A fleet of support equipment including track dozers, wheel loaders, road graders, and water trucks have been included in the fleet. The fleet of mining equipment also includes fuel and lube trucks, mechanic service trucks, mobile cranes, a tire handler, transport busses, light plants and pick up trucks.

Table 16-20 presents the list of mining equipment required for the operation. The table identifies the Caterpillar equivalent to give the reader an appreciation for the size of each machine, although the specific equipment selection will be done during the procurement phase of the Project.

Table 16-20: Mining Equipment Fleet

Equipment	Description	Max Units
Major Equipment		
Haul Truck	Payload – 64 t	26
Excavator	Bucket Payload – 13 t	4
Production Drill	150 mm hole (6")	6
Wheel Loader	Operating Weight – 55 t	1
Support Equipment		
Track Dozer	Operating Weight – 50 t	6
Road Grader	Operating Weight – 35 t	3
Water Truck	50,000 Litre Capacity	2
Big Utility Excavator	Operating Weight – 90 t	1
Small Utility Excavator	Operating Weight – 50 t	2
Utility Loader	Operating Weight – 30 t	1
Service Equipment		
Fuel and Lube Truck	-	2
Mechanical service truck	-	2
Welding truck	-	1
Lowboy	-	1
Tire Handler	-	1
Mobile Crane	-	1
Boom Truck	-	1
Dewatering Pump	-	5

16.10 Pit Operation Personnel

The mine workforce will reach a maximum of 173 employees during peak production. The workforce is composed of 123 employees in Mine Operations, 25 employees in Mine Maintenance, and 25 employees in Mine Technical Services. The contractor will have 144 employees and the owner personnel will be 29. Owner personnel will provide supporting technical services and mine management. The mine operations will be composed of four crews in order to provide a 24 h/d continuous operation.

It was assumed that the workforce will largely consist of local personnel except for five supervisory roles that would likely be filled by expatriate personnel.

17. RECOVERY METHODS

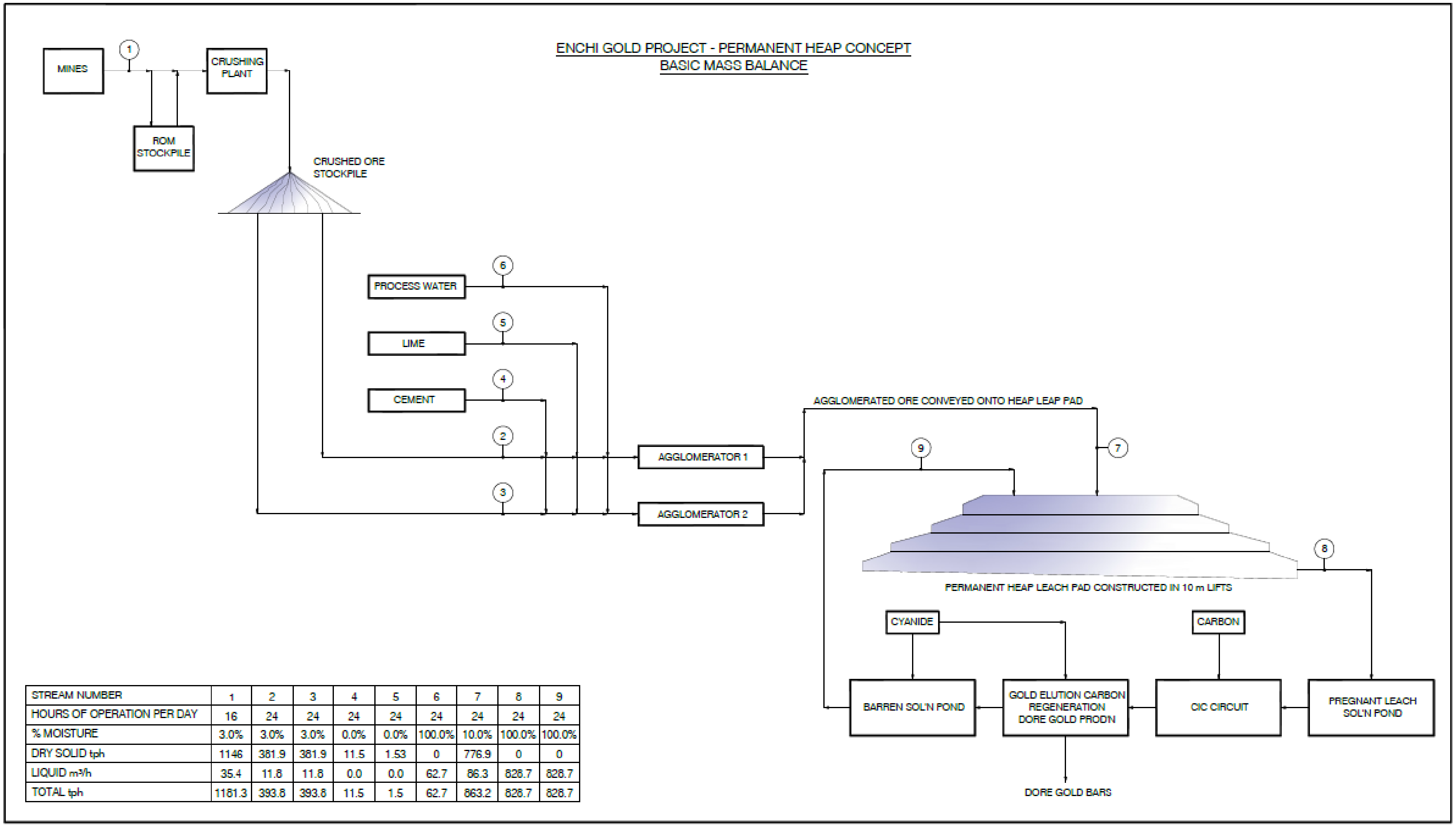
17.1 Overview

The process facility for the Project has been designed to process oxide, transition and fresh mineralization from the Sewum, Boin, Nyam and Kwakyekrom deposits. The process route selected is a permanent heap leaching facility. The process facilities including crushing, agglomeration, heap leach pad and the recovery plant will operate year-round. The design is like that of the existing heap leach operations that process similar material under comparable conditions.

The following is a summary of the recovery methods:

1. The run-of-mine (ROM) material will be delivered by haul trucks from the open pits to a truck dump pocket and/or the ROM mineralized material stockpile adjacent to the primary crusher.
2. During regular operations (360 d/a), mine haul trucks will discharge the mineralized material into a dual truck dump hopper/apron feeder that will feed material into the primary sizer.
3. The product from the primary sizer will be fed into a secondary sizer to achieve a product size of -50 mm.
4. The -50 mm product from the secondary sizer will be conveyed to feed four parallel vibrating double deck screens that will separate undersize product material, with a P_{80} of 12.5 mm, from oversize material that will be fed into four parallel cone crushers.
5. The product from the cone crushers will be conveyed back to the vibrating screens in order to achieve the final crushed product P_{80} size of 12.5 mm.
6. The crushed mineralized material product will be conveyed to a conical stockpile with a live storage capacity of 20,000 tonnes. Four apron feeders installed beneath the stockpile will feed two parallel conveyors that will transfer the crushed mineralized material to the two parallel rotating drum agglomerators.
7. Hydrated lime for pH control and cement as a binding agent will be added onto both agglomerator feed conveyors at a controlled rate. Process water will be added into the feed chute of each agglomerator for wetting the product to 10% moisture.
8. The product from both agglomerators will be combined, transported and then stacked onto the heap leach pad through a series of moveable conveyors.
9. A dilute barren process solution containing cyanide will be pumped onto the surface of the leach pad through a drip or wobbler irrigation system.
10. The resultant pregnant leach solution (PLS) will flow through the collection system and into the PLS pond.
11. The PLS will be pumped to a carbon adsorption-desorption-regeneration (ADR) plant for gold recovery. The gold will be adsorbed onto activated carbon, then periodically stripped from the carbon using a Zadra type desorption process.
12. The gold will then be plated onto steel wool cathodes in an electrowinning cell, removed by pressure washing, filtered, dried and then smelted to produce doré bullion.

Based on a design throughput rate of 6.6 Mt/a and an overall gold recovery of 79%, approximately 90,000 troy ounces of gold doré will be produced on an annual basis. Each process step is described in more detail under the following process headings. Figure 17-1 illustrates a preliminary block flow diagram of the heap leach facility (HLF) and process facilities.



STREAM NUMBER	1	2	3	4	5	6	7	8	9
HOURS OF OPERATION PER DAY	16	24	24	24	24	24	24	24	24
% MOISTURE	3.0%	3.0%	3.0%	0.0%	0.0%	100.0%	10.0%	100.0%	100.0%
DRY SOLID tph	1146	381.9	381.9	11.5	1.53	0	776.9	0	0
LIQUID m^3/h	35.4	11.8	11.8	0.0	0.0	62.7	86.3	828.7	828.7
TOTAL tph	1181.3	393.8	393.8	11.5	1.5	62.7	863.2	828.7	828.7

Figure 17-1: Preliminary Flow Diagram of the HLF and Process Facilities

17.2 Crushing

The ROM material will be delivered to the crusher dual truck dump hopper or the ROM mineralized material stockpile adjacent to the primary crusher by mine haul trucks from the various pits. The crusher operating schedule will typically be 16 hours per day with maintenance performed during off shift hours. The schedule will be arranged to minimize ROM stockpiling and reclaim at the primary crusher by front end loader. The mineralized material haulage from the Sewum and Boin deposits to the crusher may occur during the daytime or at night due to transport over mine property roads without public access. The mineralized material haulage from the Nyam and Kwakyekrom deposits will be limited to 12 hours per day during daylight hours due to the shared use of public roads.

During periods when the mine haul trucks are not able to feed directly into the truck dump pocket, they will discharge onto the ROM stockpile of mineralized material. The ROM stockpile area will be sized for a capacity of 20,000 tonnes, which is equivalent to about one day of leach feed. An apron feeder located beneath the dual truck dump hopper will feed the material into a series of two twin shaft mineral sizers at a feed rate of approximately 1,146 dry t/h. The product from the two mineral sizers will have an average particle P_{80} size of 50 mm. The crushed product from the two sizers will be transferred to the secondary crushing circuit by a single conveyor equipped with a rotating belt magnet to remove tramp steel material that could damage the downstream equipment.

Secondary crushing to achieve a final target product P_{80} size of 12.5 mm will be accomplished by screening and cone crushing of the screen oversize material. A preliminary Bruno crushing simulation has been performed to model the secondary crushing circuit configuration. The proposed circuit includes four parallel double deck vibrating screen and cone crusher lines. The four vibrating screens will each be 2.44 m x 6.12 m (8 feet x 20 feet) with upper and lower deck aperture sizes of 38 mm and 20 mm respectively. The four cone crushers will each be 1,320 mm (4.25 feet) head diameter and 400 hp (315 kW) with a closed side setting of 20 mm.

The product from the cone crushers will be conveyed back to the vibrating screens in a closed circuit to achieve the desired crushing plant product P_{80} size of 12.5 mm. The availability of the crushing plant will average about 75% during regular operations (360 d/a).

The crushing plant product will be conveyed onto a conical stockpile with a live capacity of about 20,000 t. This stockpile will provide surge capacity between the 16 h/d crushing plant operation and the more continuous agglomeration circuit operation.

17.3 Agglomeration

Four variable speed apron feeders located beneath the crushed stockpile of mineralized material will be arranged so that two apron feeders will each feed onto one of the two parallel agglomerator feed conveyors. A conveyor belt scale will be installed on each agglomerator feed conveyor so that the crushed mineralized material feed rate can be controlled at an average of about 382 dry t/h into each of the two parallel rotating agglomeration drums. The inclined agglomeration drums will each be 3.7 m diameter x 10 m long with a retention time of about three minutes. Cement and lime will be added onto the feed conveyor of each agglomeration drum at a controlled rate of approximately 15 kg/t of cement and 2 kg/t of lime. Water will also be added into the agglomeration feed chute to wet the product to a moisture content of about 10%. Based on an assumed average mineralized material moisture content of 3%, approximately 31.4 m^3/h of water will be added into each agglomerator feed chute to produce the desired product moisture of 10%.

The agglomeration circuit will be staffed to operate 360 d/y and 24 h/d. The 20,000-tonne crushed mineralized material stockpile between the crushing circuit and the agglomeration circuit will provide adequate surge capacity to enable the more continuous operation of the agglomeration circuit and the downstream heap stacking operation. As there is no surge capacity between the agglomeration circuit and the downstream heap stacking operation, the agglomeration circuit will operate in unison with the heap stacking operation. Any shutdowns of the heap stacking operation for equipment repositioning, grasshopper conveyor removal and/or maintenance shutdowns of the stacking equipment will require shutdown of the agglomeration circuit. Well coordinated operation and maintenance of the agglomeration circuit equipment and the heap stacking equipment will be important to achieve the desired production continuity.

17.4 Heap Leach Facility

A permanent heap leach facility has been developed for the Project. The site plan illustrated in Figure 17-2 shows the proposed HLF located in a topographically favourable basin approximately 2.5 km north of the Sewum Mine and 4 km south-east of the Boin Mine. The surface topography of the HLF area allows for future expansion. The overall dimensions of the HLF are approximately 1,170 m long by 675 m wide. The HLF is sized for 10 years at the proposed annual production rate of 6.6 Mt/a. The ROM mineralized material stockpile, crushing plant and agglomeration circuit are located on the south-west side of the HLF as shown on Figure 17-3.

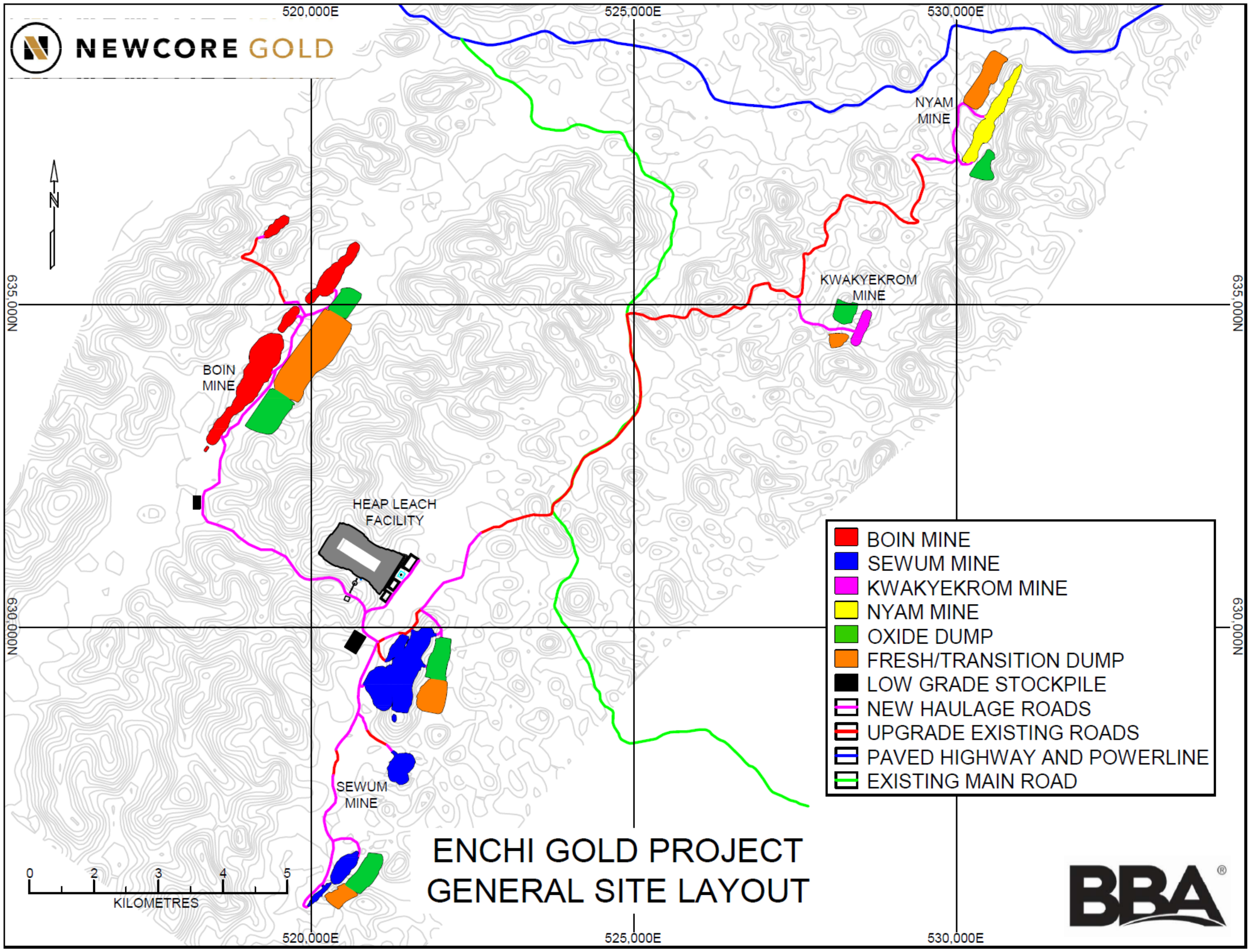


Figure 17-2: Site Plan

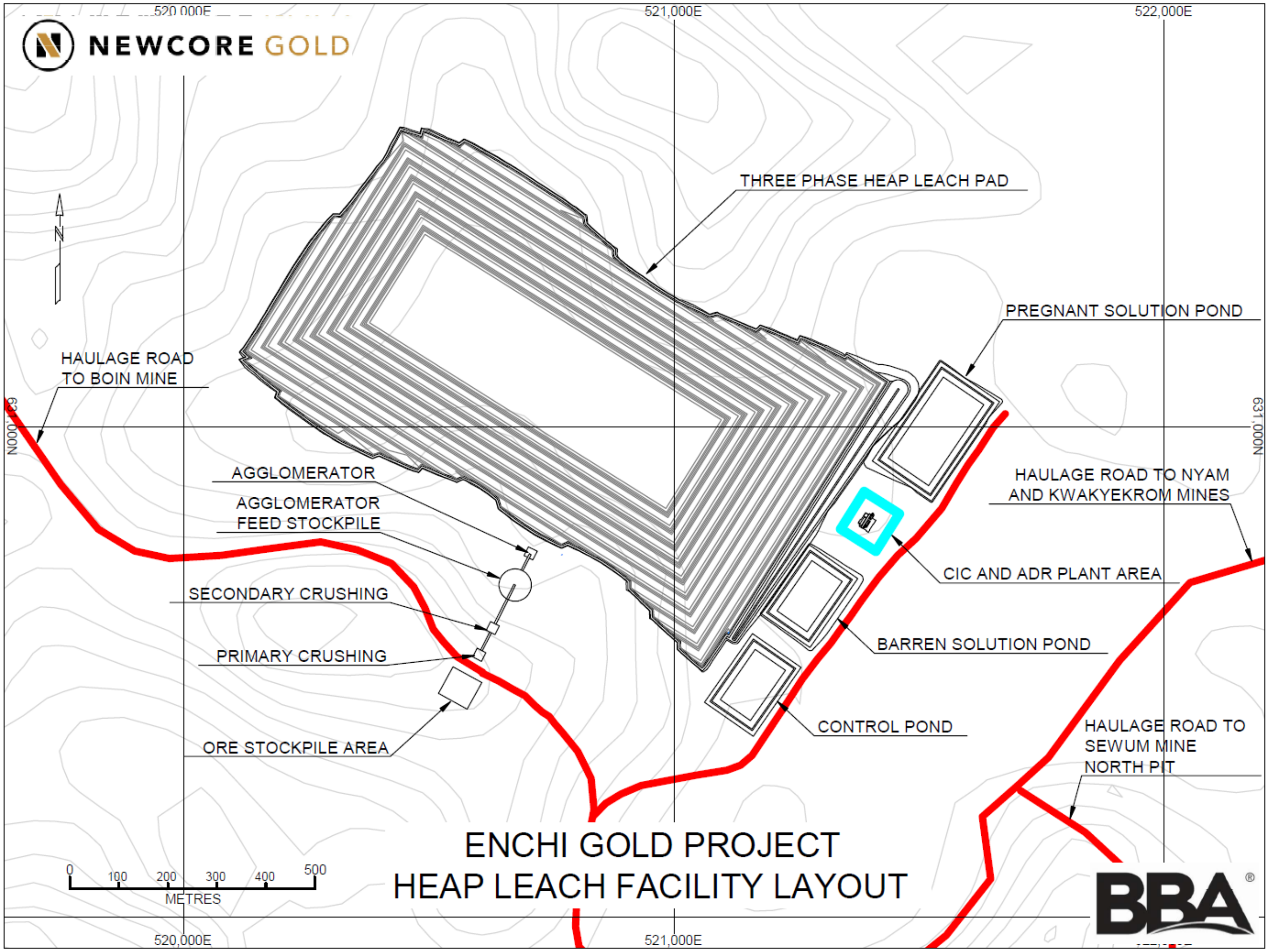


Figure 17-3: Location of the ROM Mineralized Material Stockpile, Crushing Plant and Agglomeration Circuit

A total of ten lifts of 10 m each will be required to process the estimated PEA Mineral Resource. The HLF can accommodate additional lifts. The HLF is planned to be constructed in three phases starting from the south-east and progressing upslope towards the north-west. Three production ponds including the pregnant solution pond (PLS), barren solution pond and the control pond will be constructed south-east of the HLF below the bottom elevation of the HLF. The approximate plan view dimensions and live volumes of the three ponds are summarized as follows:

- PLS: 250 m x 150 m x 5 m deep with a volume of 130,000 m^3;
- Barren Pond: 170 m x 120 m x 5 m with a volume of 57,000 m^3;
- Control Pond: 170 m x 120 m x 5 m with a volume of 57,000 m^3.

The gold recovery plant facilities including the carbon-in-column (CIC) circuit and the adsorption, desorption and regeneration (ADR) circuits will be located between the PLS and barren solution ponds as illustrated in Figure 17-3.

17.5 Heap Leach Stacking

Agglomerated material will be transported onto the heap leach pad by a series of grasshopper-type portable conveyors that will feed a portable high angle conveyor, a moveable indexing conveyor and moveable telescopic radial stacker conveyor with a 46-m stacking width. A vertical height of 10 m has been assumed as the design lift height.

Based on the average annual mineralized material production rate of 6.6 Mt, approximately 13,800 m^3 of agglomerated material will be placed onto the HLF each operating day. At any given time, there will be four different 60-day production areas active on the surface of the HLF. The summary material balance shown on the block flow diagram for the HLF (Figure 17-1) is based on average continuous 24 h/d operations. During actual operations, the placement of agglomerated mineralized material onto the HLF will not be a continuous operation as stacking shutdowns will be required for agglomeration equipment maintenance and for repositioning of conveyor components and stacking equipment. The design capacity of the agglomeration circuit and the conveyor and stacking equipment will be suitably higher than the average tonnage and flow values shown on the block flow diagram so that longer term production targets can be achieved. A detailed stacking plan and irrigation plan will have to be developed for the next level of study.

17.6 Leaching and Solution Irrigation

The recently stacked material will be irrigated by drip emitters with a dilute solution of sodium cyanide, which percolates through the section of the HLF under active irrigation and leaching. The period of active leaching is assumed to be 60 days. The solution is delivered through a network of irrigation piping including a 24-inch primary central feed pipe, a 12-inch secondary pipe, and 4 inch delivery pipes along each irrigation section to which the drip emitter lines are attached.

Based on each irrigation section being equivalent to 60 days of production, the area under active irrigation at any given time will be approximately 83,000 m^2. With a proposed radial stacker width capability of 46 m, the heap surface area under active irrigation could be 230 m x 360 m. Based on a solution irrigation unit flow rate of 10 L/h/m^2, an average flow of approximately 830 m^3/h of barren solution would be pumped from the barren solution pond onto the active leach area. Concentrated cyanide solution would be added into the barren solution pipeline that flows to the HLF at a rate to achieve the desired cyanide concentration for leaching. Injecting the concentrated cyanide solution directly into the barren pipeline will help to minimize cyanide consumption via natural degradation in the barren solution pond.

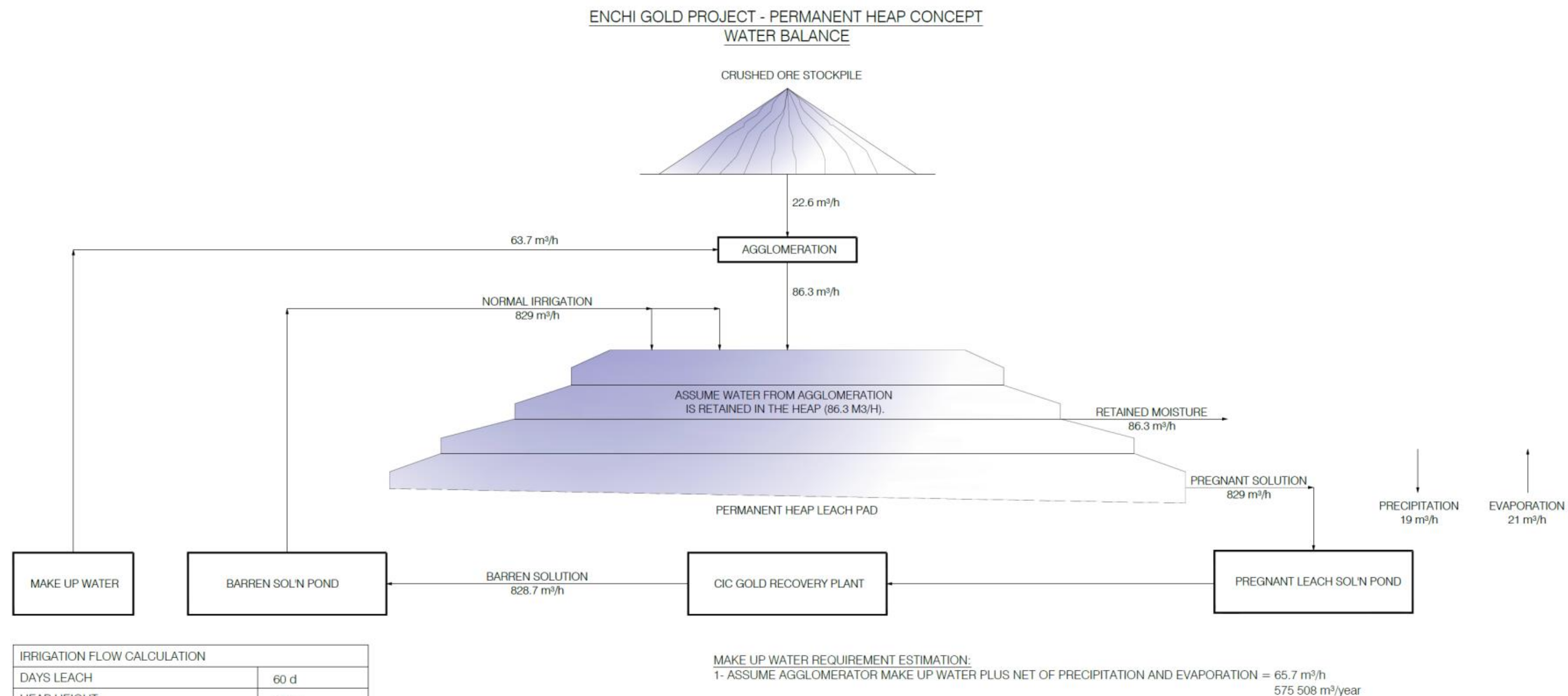
17.7 Solution Collection and Ponds

After passing through the HLF, the leachate solution and storm water will flow through a network of pipes at the base of the HLF and flow by gravity into the PLS pond. Located underneath the heap and embedded within the drain rock, the collection pipe network will consist of a series of perforated pipes appropriately spaced and connected to larger secondary pipes. The secondary pipes are connected to the larger outside main pipes that transport the gold bearing solution to the PLS pond. The solution drainpipe system will be sized to accommodate the leachate flow and storm water flows.

As indicated previously, the PLS pond will have an operational storage volume of approximately 130,000 m^3. Vertical or submersible pumps located in the PLS pond will be used to pump the PLS at a rate of approximately 830 m^3/h to the gold recovery plant or ADR plant.

Barren solution from the ADR plant will flow into the barren solution pond with an operational storage volume of approximately 57,000 m^3. Reagents and process water will be added into the barren solution pond and/or the barren solution pipeline as required to maintain the desired cyanide, pH and pond level. Vertical pumps will be used to transfer the barren solution to the active irrigation/leach section area via the irrigation piping.

A control pond will be used to store process water that will be collected from diversion ditches around the perimeter of the HLF and site run off that is cyanide free. The control pond will be the source of make up water for the process to off set water retained in the HLF. The proposed operational storage volume of the control pond is approximately 57,000 m^3 to contain peak precipitation events. A simple HLF site water balance schematic is shown in Figure 17-4. All solution systems are designed for 100% availability at 365 days per year since solution flow will continue when stacking operations are shutdown. A more detailed water balance for the site will be developed at the next level of study.



IRRIGATION FLOW CALCULATION	
DAYS LEACH	60 d
HEAP HEIGHT	10 m
IRRIGATION RATE	10 l/hr./m2
AGGLOMERATE BULK DENSITY	1.5 m3/t
AGGLOMERATE STACKED PER DAY	20 717 t/d
AGGLOMERATE VOLUME STACKED	13 811 m3/d
AGGLOMERATE AREA STACKED	1381 m2/d
AREA UNDER IRRIGATION PER CYCLE	82 867 m2/cycle
IRRIGATION FLOW RATE	829 m3/h

*NOTE: ALL FLOWRATES ADJUSTED TO 24 HOUR PER DAY, 365 DAYS PER YEAR AVERAGE VALUES.

Figure 17-4: HLF Site Water Balance Schematic

17.8 Gold Recovery Plant

The ADR plant will receive pregnant leach solution from the PLS pond by vertical or submersible pumps at a rate of approximately 830 m^3/h. The plant will use an efficient carbon-in-column (CIC) recovery process with industry proven techniques as described below. The recovery of gold to carbon using this process is typically 98%. Final selection of process flow is to be determined at later stages of study.

17.8.1 Carbon Absorption

The PLS solution will be fed to the CIC circuit where the dissolved gold in solution is absorbed by the activated carbon granules in a train of five cascade type carbon columns. Each column will contain approximately four tonnes of activated carbon granules. The PLS will be fed to the first column and will flow by gravity from column to column. In each column, the PLS will flow upwards creating a rising current suitable for suspending the activated carbon granules, allowing the gold to be adsorbed onto the carbon granules. The carbon will be transferred in a counter current manner to the flow of PLS.

The barren solution leaving the last column will have a weak gold concentration and is transferred back to the barren pond for reuse at the heap leach facility. The carbon in the first column with high gold content, termed 'loaded', is transferred to the acid wash. By exposing the freshest carbon to the weakest gold concentration PLS, the counter-current CIC circuit has been proven to maximize gold recoveries to the loaded carbon and minimize residual gold concentration in the barren solution.

17.8.2 Acid Washing

The 4 tonnes of loaded carbon from the first column of the CIC circuit will be transferred to an acid wash column and washed by a circulating dilute hydrochloric acid solution at approximately 3%. The acid wash cycle time will be approximately 6 hours and is designed to control scaling and other inorganic contaminants on the carbon granules so as to maintain its ability to absorb gold. After the acid wash cycle is complete, the carbon batch will be transferred to the elution column.

17.8.3 Elution / Stripping

Once acid washed, the loaded carbon will be transferred to the elution column or strip vessel designed to treat a 4-tonne batch of loaded carbon. At high temperature and pressure, the Zadra-type desorption process uses a hot caustic and sodium cyanide solution that is recycled through the bed of carbon to remove the gold from the carbon. The elution solution will be heated using a heater skid consisting of stainless steel plate and frame heat exchangers and a fuel-fired water heater. The hot elution solution will flow upward through the elution column/strip vessel; 1.1 strips per day will be required. The total elution cycle time, including carbon transfer, will be about

20 hours producing a gold bearing strip solution or 'eluate' at a flow rate of 8 m^3/h. The eluate will then flow by gravity through cooling heat exchangers to the electrowinning circuit, which operates in closed circuit with the elution column.

17.8.4 Electrowinning and Smelting

Gold will be recovered from the eluate by electrowinning where it is plated onto steel wool cathodes in two 100 ft^3 electrowinning cells. The barren solution 'electrolyte' will exit the cells by gravity into the electrowinning solution return tank, from which it will be recycled back to the elution column. Caustic and cyanide will be added as needed to maintain the required solution strength. The gold concentrate will be washed from the cathodes periodically with a high-pressure water sprayer, and then dewatered through a plate and frame pneumatic/hydraulic filter press. The gold filter cake will be dried in an oven and mixed with fluxing agents then smelted in a furnace and poured into doré molds. The doré bullion will be shipped off-site for further refining. The plant is designed to produce more than 100,000 oz Au/year.

17.8.5 Carbon Handling and Regeneration

After gold removal in the elution/stripping column, the eluted/stripped carbon or 'barren' carbon is screened with a portion transferred to a fuel-fired regeneration kiln where it is thermally regenerated at a temperature of about 750°C. Before returning to the circuit, the regenerated carbon is sized in a carbon sizing screen to remove any fines generated from handling and transfer of carbon between the various unit operations. The carbon fines are recovered by means of a filter press, collected and sent off-site to a 3rd party precious metals refinery for trace gold recovery. Lost activated carbon is replenished by new carbon after attrition and fines removal. Fines from fresh attrited carbon are discarded as they contain no gold. Fresh and regenerated carbon is returned to the CIC circuit in the last column where the PLS gold concentration is weakest.

17.9 Process Water Balance

The process design takes into consideration the circulation of solutions through the HLF, collection ponds, recovery plant, as well as precipitation and evaporation on an annual basis. Using data collected from Newcore with respect to average annual precipitation and evaporation in the area, the results of the water balance estimates the make-up water requirements to maintain zero discharge of solutions. This gives an indication of the amount of water needed per year, but by no means does it indicate the peak water consumption when there is no rain and evaporation is high, or when precipitation is high and all water cannot be retained and some discharge of solution will need treatment.

The water balance as shown schematically on Figure 17-4 uses annual average flow rates and volumes and is dependent on the moisture content in the ROM material, the amount of water used to wet the product during agglomeration, as well as the moisture retained in the heap leach pads. Changes in the moisture content in these areas will affect the make-up water requirements. The design solution irrigation rate to the heap leach pads will be 829 m^3/h based on a nominal rate of 10 $L/h/m^2$. Taking into consideration the moisture retained in the heap leach pads, evaporation losses, and precipitation gains, the average annual PLS solution flow to the carbon columns will be 829 m^3/h. Make-up water is assumed to come from the control pond (sized based on 1 in 100-year precipitation) when excess water is available, and rain is assumed to be used as wash water. While annual evaporation in the area is only slightly higher than precipitation, they have little effect on the water balance. The make-up water requirements based on seasonal climate conditions will need to be evaluated.

18. PROJECT INFRASTRUCTURE

18.1 Existing Regional Infrastructure

The heap leach facility (HLF) and central processing area for the Project are located approximately 20 kilometres south of the town of Enchi, the capital of the Aowin Municipality. The nearest seaport, Takoradi, is 205 km southeast of the Property by way of the N1 and N12 Highways. Accra, the capital of Ghana and the main point of entry by sea or air is 427 km east of the Property by road.

Ghana currently has an installed electrical generating capacity of 4,399 MW consisting of 36% hydroelectric, 64% thermal (diesel and natural gas) and less than 1% renewable. Available power tends to fluctuate due to numerous factors, including fuel interruptions, and as a result the total available electrical power tends not to exceed 2,400 MW. The substation nearest to the Project site is located 50 km to the south in the town of Elubo and is serviced by a 225 kV line from Prestea to the east. A 33 kV powerline runs parallel to the Asankragua-Enchi Road north of the Property, which is 600 m from the Nyam Mine pit and 8 km from the HLF. There are numerous options for connection routes for the Project.

Residents of most localities in the Aowin-Suaman District do not have access to fixed telephone line systems. Enchi is one of the few locations in the region to have a fixed telephone line and mobile service. Mobile cell service exists over much of the Project.

Fuel, accommodations, food, and most supplies can be obtained in the town of Enchi.

18.2 Project Site Layout

With the Project being located 20 kilometres south of the town of Enchi, between the villages of Sewum and Achimfo, most of the infrastructure will be greenfield. Other than a low capacity electrical power line and sand/gravel/earth roads, the Project site has no existing infrastructure.

The overall site layout is shown in Figure 18.1. Figure 18.2 to Figure 18.5 illustrate a close-up view of the four areas, Heap Leach Facility, Sewum Mine, Boin Mine, Nyam Mine, and Kwakyekrom Mine.

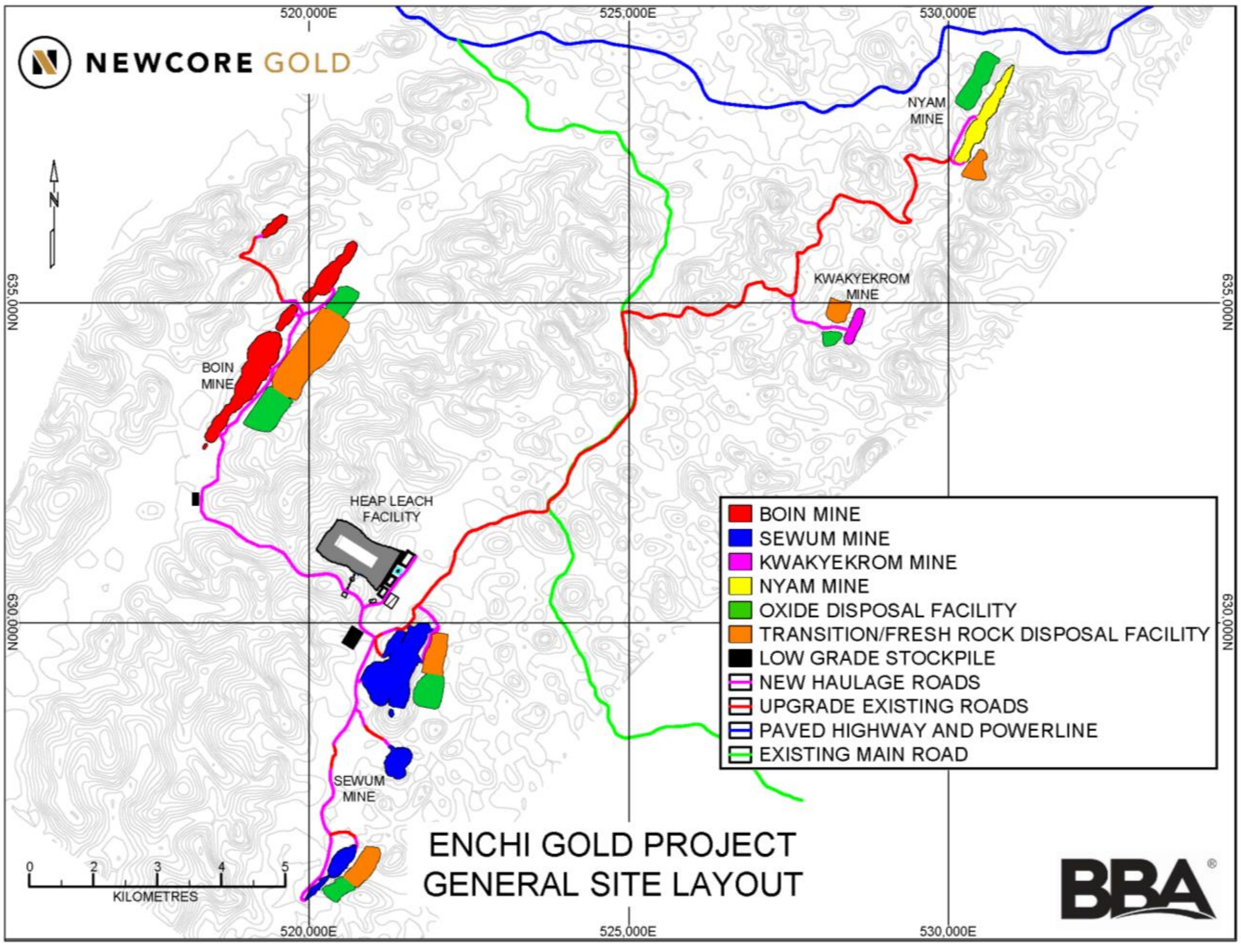


Figure 18.1: General Site Layout

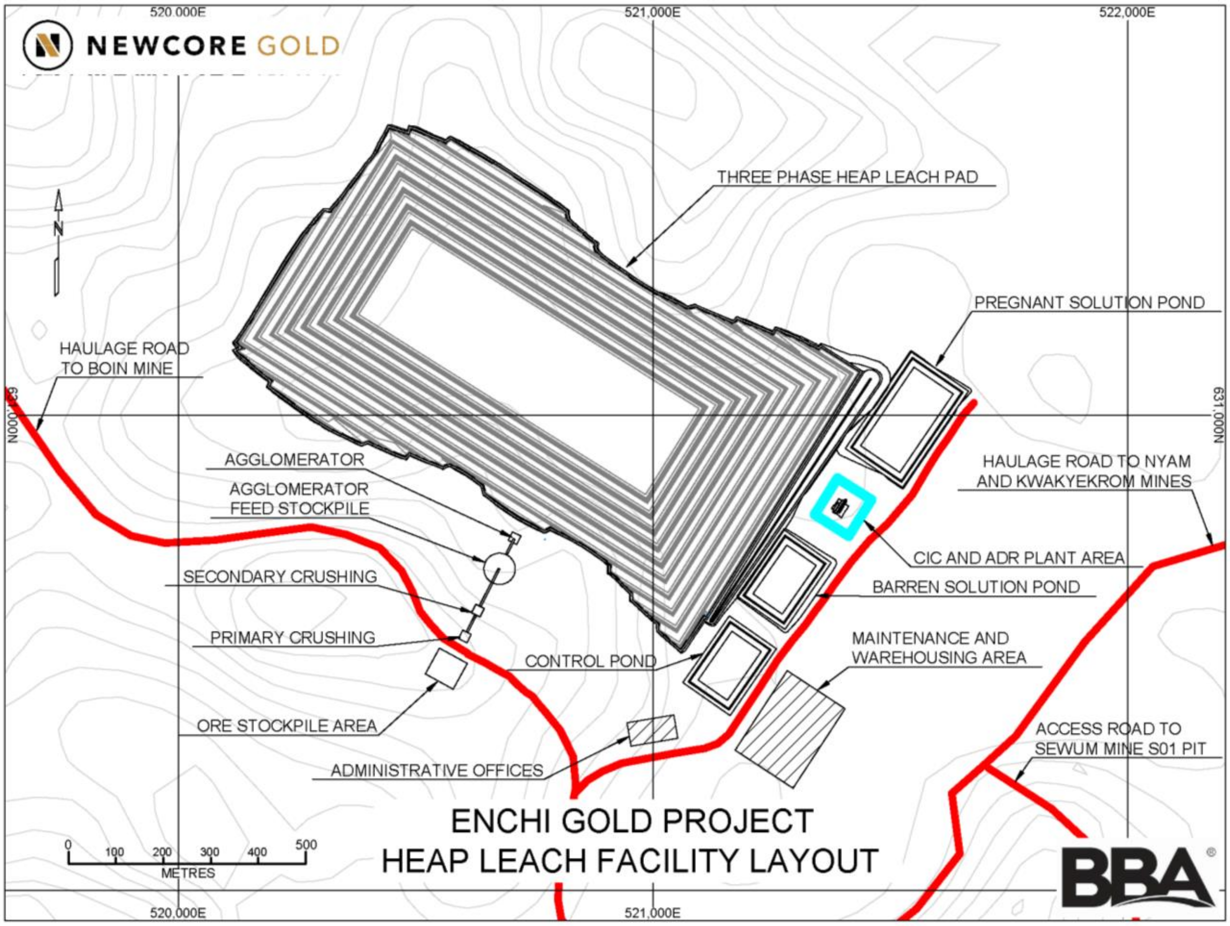


Figure 18.2: Heap Leach Facility Layout

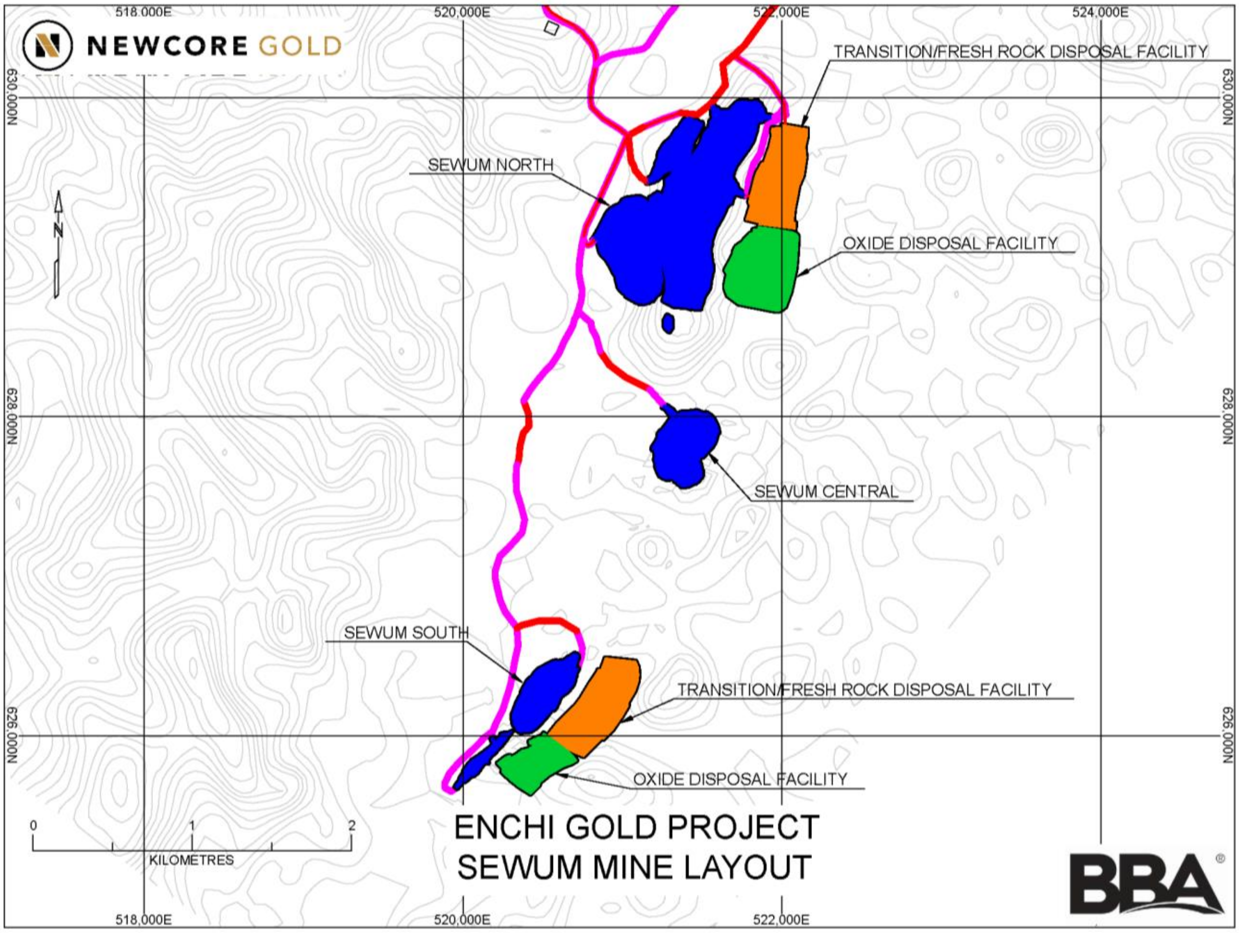


Figure 18.3: Sewum Mine Layout

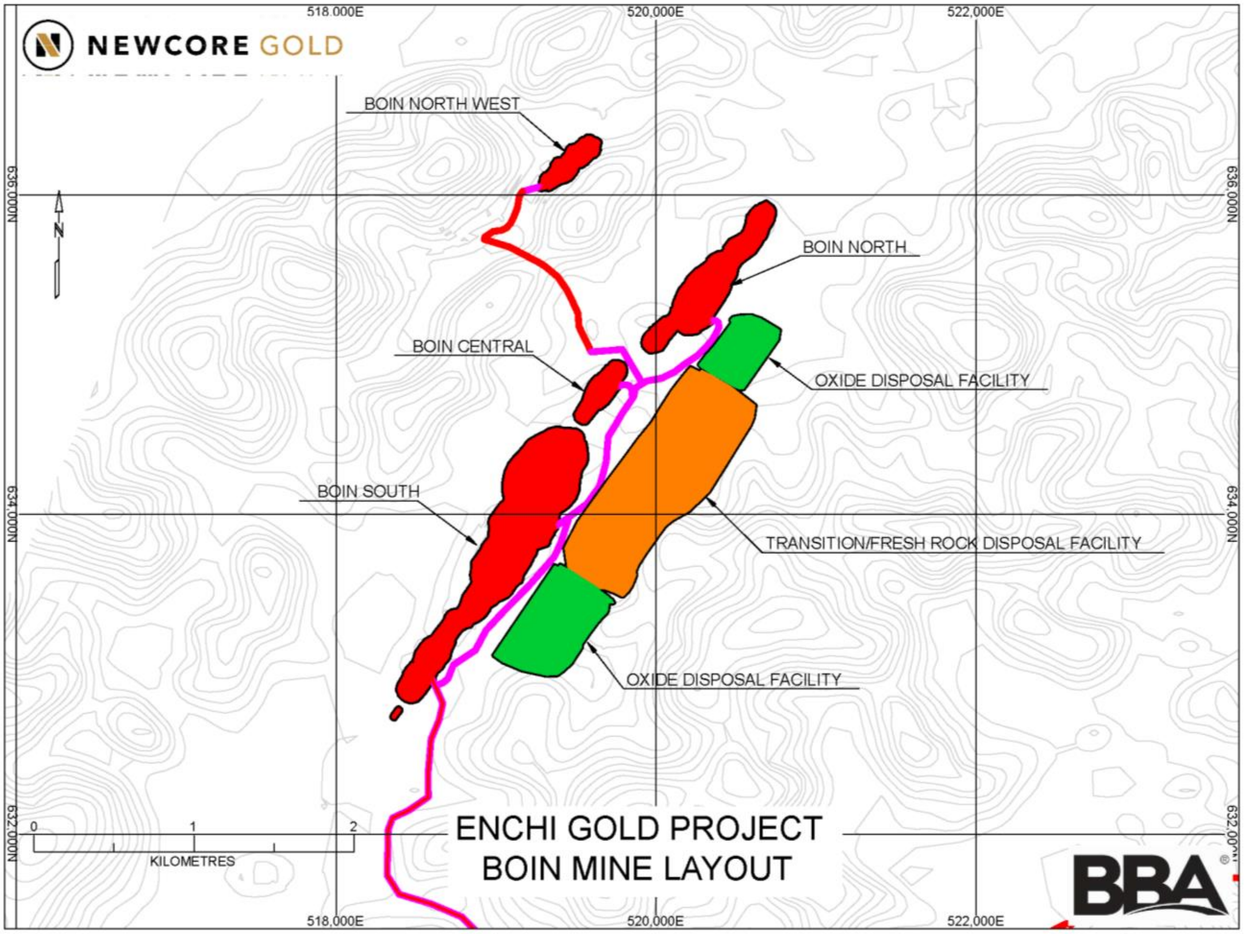


Figure 18.4: Boin Mine Layout

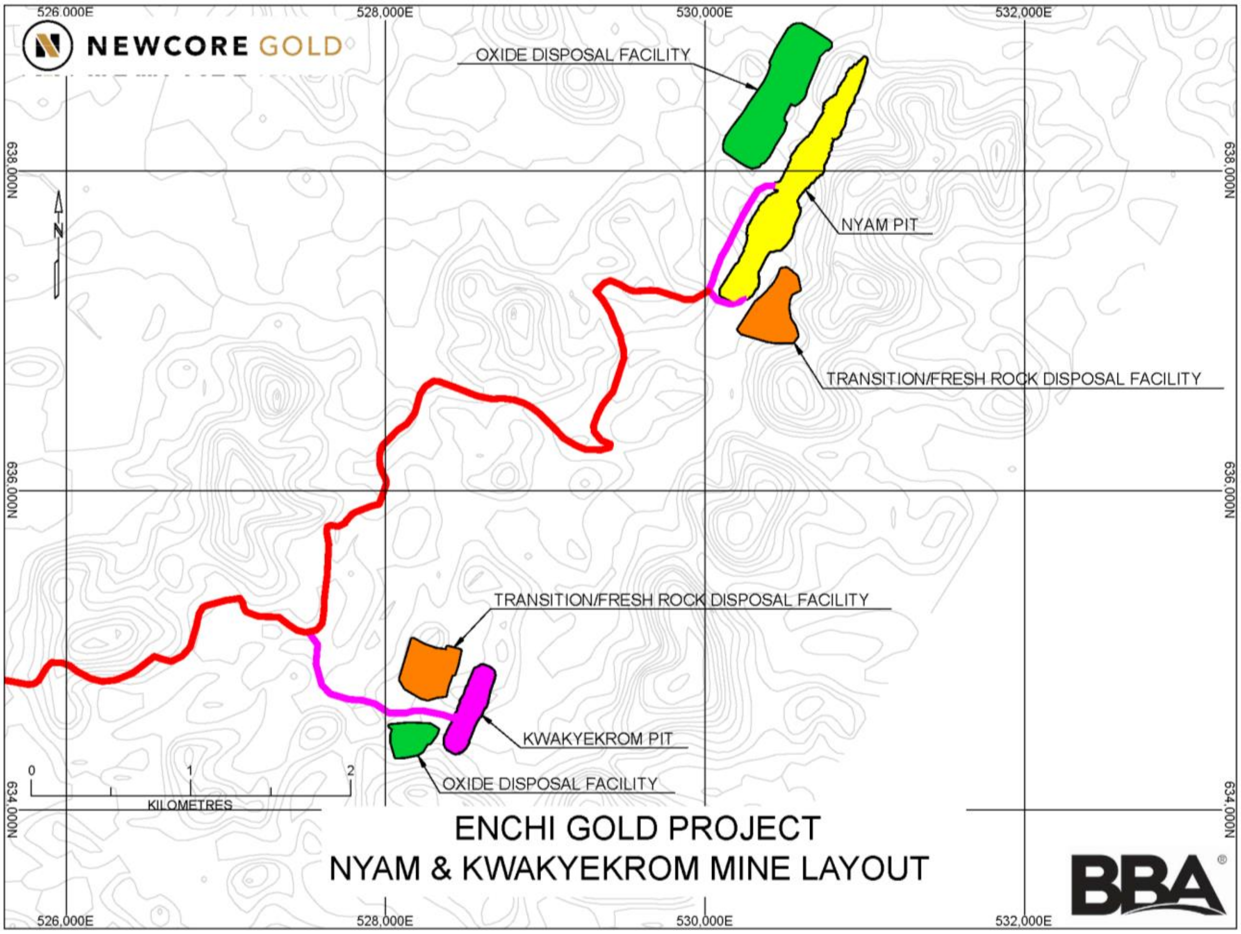


Figure 18.5: Nyam and Kwakyekrom Mine Layout

18.3 Site Development

For the purposes of this PEA, site infrastructure locations were selected to minimize capital requirements, optimize material transportation between the mine sites and the HLF, and minimize the impact on local communities and the environment.

Site preparation will consist of the removal of trees, clearing and grubbing of areas that will accommodate the main processing area, administration buildings, and the maintenance complex. Sites will be levelled and graded only in areas where construction will take place. Cut-and-fill will be utilized where large, level areas are required.

An allowance has been made for surface water diversion and fencing around all of the open pits as well as the central processing area.

A total area of approximately 120 ha is estimated to be developed for the HLF, main processing area, administration buildings and the maintenance complex.

The plant site areas will require a geotechnical investigation prior to the next phase of the Project to determine the suitability of the proposed locations and types of material encountered.

18.4 Project Infrastructure

The infrastructure will support the mining, processing plant, and construction operations. The main site will contain all the maintenance workshops, processing plant operations, and administration buildings, and is located centrally between the pits at Sewum and Boin Mines (see Figure 18.1).

The open pit mine sites of Sewum, Boin, Nyam and Kwakyekrom are located outside of the central processing area and are controlled as separate operations. Each will be connected to the central mine site by haul roads and have a mine access road with satellite offices.

Infrastructure and service requirements to support the mining and processing operations are summarized below.

- Central site:
  - Crushing, grinding and agglomeration;
  - Heap Leach Facility;
  - Processing/Extraction plant;
  - Administrative offices;
  - Mine dry complex;
  - Assay lab;
  - Explosive magazine;
  - Fuel and lubricant storage;
  - Maintenance workshops (mechanical and electrical);
  - ROM stockpiling area.
- Power supply and distribution;
- Water supply and distribution;
- Surface water management;
- Sewage and waste management;
- Mine pit access and haul roads;
- Satellite office facilities at the Boin, Nyam and Kwakyekrom mine sites.

Where possible, offices and amenities will be housed in portable/prefabricated structures. Maintenance shops and warehousing will be structural steel buildings on concrete slabs.

Numerous water sources are available in the area, in addition to water collected on-site through the surface water management infrastructure. It is recommended that a site water balance be done to determine the consumption and required storage capacity. This PEA allows for the provision of a containerized water treatment system to provide potable water as required for the site.

An elevated water storage and distribution system is recommended for the main offices and the process plant to avoid interruptions.

Electrical power requirements for site preparation and administration buildings will be covered by the local distribution lines that already pass through the site. It is assumed that power for the crushing, agglomeration and extraction plants will be provided by the 33 kV power lines north of the Project site, with diesel powered generators for backup.

This PEA allows for a wastewater storage and treatment facility for the central site.

All solid wastes, other than the domestic waste, will be collected and transported off-site by a licensed contractor for appropriate recycle and disposal. Domestic wastes generated on-site within the administration areas will be incinerated on-site.

To facilitate the collection of solid wastes for off-site disposal, collection and sorting areas will be provided within a dedicated solid waste management area. No long-term, on-site storage areas were included as part of the Project.

18.5 Main Site

In support of the mining operations, the main site will include a mine dry building, equipment maintenance workshops, refuelling facilities and an explosives magazine.

Infrastructure related to the processing plant will include internal access roads, ROM stockpile area, office and administration building, assay laboratory and warehousing.

To support these facilities there will also be storm water handling facilities, a water supply and power supply network, backup diesel generators, sewage treatment plant, waste management facility, and site access road.

18.6 Mine Haul and Pit Access Roads

A number of gravel roads will be necessary to transport the ROM material from the four mines to the heap leach facility. Due to the distance between the individual mine sites and the heap leach facility, the length and quality of haulage roads will have a significant impact on the efficiency and consistency of production.

Wherever possible, the haulage roads will be constructed by upgrading existing roadways. This reduces up-front capital requirements and the time required for construction.

Finished roads will have a gravel surface and a width of at least 10 metres. A total of just over 37 km of haulage and pit access roads will be constructed over the life of mine. The breakdown of new road construction and upgrades to existing roads is presented in Table 18-1

Table 18-1: Haulage and Access Road Construction

Complex	New Road Construction [m]	Upgrade to Existing Road [m]	Total Road Requirement [m]
Sewum Mine	6,715	4,759	11,474
Boin Mine	6,878	1,465	8,343
Nyam Mine	1,253	0	1,253
Kwakyekrom Mine	1,616	14,707	16,323
Total	**16,462**	**20,931**	**37,393**

18.7 Accommodations

No on-site accommodations have been accounted for in this study. Senior staff and expatriate employees will be accommodated in rental units in the town of Enchi.

19. MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Markets for doré are readily available and the doré bars produced from the Project could be sold on the spot market. Gold markets are considered mature.

The gold price used in this study is similar to other technical studies recently published in 2020 and 2021 and is above the 3-year trailing average of approximately $1,544/oz. As of June 8, 2021, the date the PEA was announced by Newcore, the spot price for gold was $1,893.20/oz (www.gold.org).

19.2 Contracts

There are no sales contracts on the Project.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

BBA is summarizing the environmental studies, permitting and social or community impact findings from the 2016 Enchi PEA (McCracken, et. al, 2016). As no site specific environmental studies have been undertaken to date, the information presented in this chapter is based on a desktop assessment of existing information relating to the Project and site, maps, and site photographs that have provided insight into current conditions.

20.2 Environmental and Social Context

20.2.1 Physical Environment

20.2.1.1 Air Quality

The climatic setting is described in Section 5.3 of this report. As the Project is located in a rural environment, the local air quality is anticipated to be good. Current emissions are likely to be limited to localized domestic sources such as outdoor fires used for cooking and vehicle movements (exhaust and dust emissions).

20.2.1.2 Ground and Surface Water

The Project is underlain by crystalline igneous and metamorphic formations. These have little primary porosity, and therefore the presence of groundwater results from secondary porosity, i.e., jointing, shearing, fracturing and weathering, and give rise to weathered zone aquifers and fractured zone aquifers. The depths of aquifers are normally between 10 m and 60 m, and rarely yield more than 6 m^3 per year. Groundwater is likely to be the predominant source of water supply given the rural context of the Project area. The town of Enchi, the district capital, obtains a reliable supply of potable water from groundwater pumped to homes and businesses in the town.

The Project is characterized by moderate topography, incised by river tributaries, and scattered flat plateaus. The Project is primarily drained by the Tano River and its tributaries, most notably the Disue. Unlike most of the major river systems in Ghana, this river drains towards the south-west of the country and does not form part of the Volta River system, which covers nearly 70% of the country. Local communities and homestead are likely to rely on local rivers and streams for potable water for domestic and farming activities.

20.2.1.3 Soils and Land Capability

The major soil types within the district are Forest Ochrosol and Forest Oxysol, which have developed from weathering of the metamorphosed rocks. A description of these soils is adapted from Centre for Policy Analysis (CEPA) Ghana Selected Economic Issues No. 3 2000 Soil Classification in Ghana.

- Forest Ochrosol soils are red, brown and yellow-brown in colour, and occur on the summits, upper, middle and lower slopes of the terrain. The upper horizons generally have a strong organic content, and strongly leached lower horizons. The first subgroup of the upland Forest Ochrosols (Wenchi series) is minor in extent, of low agricultural significance, and requires to be kept permanently under forest to maintain its structure and prevent erosion. The second subgroup (Bekwai series) (the most extensive subgroup within the semi-deciduous forest belt) is moderately shallow to moderately deep, with red and brown colouring. These soils occur over moderately undulating and to sloping topography, and consist of concretionary and/or gravelly, moderately heavy to medium textured soils overlying mostly highly weathered phyllite. The third subgroup (Akumadan series) is deep to very deep and red and brown in colour. They are well to moderately drained soils, which are devoid of concretions and gravel at least to 60 cm and are moderately heavy to medium textured, and occur of the flatter areas of the uplands.
- Forest Oxysols are characterized by deeply weathered, yellow, moderately well drained, acid soils. They are generally medium to moderately heavy textured upland soils. These soils occur within the high rainfall areas of the extreme south-west of Ghana, and thus subject to high rainfall, and are therefore characterized by strongly leached soil profiles with poor nutrient retention. Forest Oxysols are divided into three subgroups. The Boin series (overlying phyllite), Abenia series (overlying granite), and Tikobo series, develops in tertiary sands and are mainly reddish brown to brown, well drained and medium in texture.

Any soils removed during the Project are stockpiled separately from the waste rock dumps and would be used during the reclamation at the end of the Project.

20.2.2 Biophysical Environment

While the Project is located within the moist tropical rainforest area of the Western Region of Ghana, the majority of the Project is disturbed through farming practices. Cocoa plantations are most prevalent, in additional to other locally grown food crops such as plantain, maize, cocoyam, cassava and rice. With the original vegetation cover in the majority of the Project having been removed, it is likely that secondary forest may only occur on steep slopes unsuitable for farming. Wood from these forested areas will be used as a source of domestic fuel and for use as building materials.

The northern part of the Project lies adjacent to forest reserves, with the majority of the area reserved for commercial timber plantation.

The Boin National Park, characterized by wet evergreen forest, is located beyond the north-western and western boundary of the Project.

20.2.3 Social Environment

Enchi, the Aowin District capital, is the nearest town to the Project and has an estimated population of 11,737 people. The district as a whole is rural in nature, infrastructure is limited, and settlement limited to scattered towns and hamlets (single family compounds). There are educational and healthcare facilities in Enchi and a number of other small villages in the district.

Current farming land use within the Project is likely to be characterized by subsistence and cash-crop agricultural practices. The key stakeholders, with respect to the Project, will be communities that use the Project for farming activities and other resources such as firewood. Based on aerial photography, settlement appears to be sparse within the proposed mining operational areas, as such resettlement is considered to be unlikely and, if required, will be limited to small scale resettlement of individual hamlets.

Cultural and heritage aspects within the Project are likely to include local grave sites and cultural areas of importance.

Skilled and unskilled labour can be drawn from the local population, particularly as a result of the regions mining history.

20.3 Summary of Potential Environmental and Social Issues

This section highlights the potential environmental and social issues that may be associated with the proposed mine operations. As the Project is in the PEA stage, the ESIA for the Project is pending. The potential issues identified are based on a desktop understanding of the environmental and social context as well as previous experience of mining operations. The area is well developed through subsistence farming and any impact from mining are anticipated to be minimal. The following is a list of potential environmental and social issues:

- High storm water runoff, as a result of high rainfall and steep topography, are likely to increase the potential for soil erosion in operational areas, which may result in contamination of water courses.
- Increased turbidity is likely to impact the functioning of the aquatic systems and water quality.
- Natural water systems may be impacted by increased heavy metal concentrations, acidification and changes in chemical characteristics from operational activities.
- Alterations in stream flow regimes and dynamics are likely to affect riparian and aquatic environments.
- Changes in water quality may have some impact on downstream users, in particular heavy metal accumulation in agricultural crops, and changes in soil nutrient availability due to alteration of the water pH.

- Endemic aquatic species are likely to be particularly vulnerable to changes in water quality.
- Acid mine drainage may present risk of impact to groundwater reserves as well as surface water systems.
- Dewatering of mine areas may impact the groundwater availability of adjacent areas.
- Cyanide pregnant solution may permeate into groundwater through a failure in the heap leach liner. Additionally, cyanide pregnant solutions transmitted through pipelines and from stations from the heap leach facilities may impact the surface water quality.
- Water from industrial support services such as workshops, is likely to contain hydrocarbons and heavy metals, which are likely to impact surface and groundwater resources.
- Treated effluent from waste water treatment and water treatment works may have impact to water systems in terms of water quality and biological standards.
- Clearance of vegetation will expose soils to erosion with the operational area as a result of climatic elements (winds, rainfall).
- Any introduction of pesticides and herbicides may have a negative impact on flora and fauna.
- Reduction in connectivity of habitats may affect movements of wildlife species.
- Decrease in local air quality and a result of dust from mining and heavy vehicular movements on untarred roads. Impact of dust is likely to be localized to areas immediately adjacent to roadways and within the mine lease area, depending on the chemical composition of the mine dust health risks to the local population via inhalation and dermal exposure may increase.
- Noise of mining machinery and equipment may impact any surrounding communities as well as local fauna.
- Vibrations from blasting may affect the structural stability of localized dwellings, other buildings and infrastructure.
- Development of artisanal and illegal mining has the potential to introduce mercury into the region. Mercury may have long term impacts for the health of people and the environment.
- Development of the mine, its associated infrastructure and operations, residential areas and roadways may result in loss of biodiversity.
- Any communities and/or land use within the mine lease area is likely to require resettlement. As such, there is likely to be an impact on the feeling of placement and livelihood for communities as intensification of land outside the mine lease area.
- Cultural heritage resources that may be situated within the mine lease area may be negatively affected.
- Immigration of people into the area is likely to have significant impact on the local communities in terms of increased competition for employment opportunities, social/cultural complications, changes in security and potential importation of diseases.
- Post mine closure is likely to have a negative impact on the community, due to their dependency on the mining activities in the region.

The most significant potential environmental and social issues are likely to be related to water management, social-economic impacts and post mine closure expectations. These issues are likely to either be of major concern to local communities and/or likely to have cost implications with respect to impact management during the operational and closure phases.

20.4 Approach to Environmental and Social Management

Environmental management requirements for the mining operations will be incorporated into an Environmental Management Plan (EMP), which is submitted to the Government of Ghana for review and approval prior to commencement and every three years during the operational phase. The EMP is the mechanism for environmental approvals and seeks to address environmental and socio-economic component of the Project and includes impact identification, impact management and monitoring, environmental action plans, and a rehabilitation and closure plan.

Community management will need to be carried out through a range of initiatives, which aim to engage the community and understand their concerns and needs, while providing an approach to manage stakeholder's expectations. Effective and early community engagement to address issues such as social investment, local hiring, land access, and resource loss and associated compensation and livelihood restoration considerations, will be a key determinant for the Project's success.

An Environmental and Social Management Plan/System should be developed in accordance with the requirements and expectations of the International Finance Corporations Performance Standards (IFC PS; January 2012). Based on a desktop review of the Project, the following Performance Standards are determined to be applicable:

- Performance Standard 1: Assessment and Management of Environmental and Social Risks and Impacts;
- Performance Standard 2: Labour and Working Conditions;
- Performance Standard 3: Resource Efficiency and Pollution Prevention;
- Performance Standard 4: Community Health, Safety, and Security;
- Performance Standard 5: Land Acquisition and Involuntary Resettlement;
- Performance Standard 6: Biodiversity Conservation and Sustainable Management of Living Natural Resources;
- Performance Standard 8: Cultural Heritage.

Performance Standard 7: *Indigenous Peoples* may not apply because there may not be indigenous people in the Project's area of influence. Under the Performance Standards the terms 'indigenous peoples,' 'indigenous ethnic minorities,' 'tribal groups,' and 'scheduled tribes' describe social groups with a social and cultural identity distinct from the dominant society that makes them vulnerable to being disadvantaged in the development process.

Any ESIA, as well as the resultant Environmental and Social Management Plan/System should be prepared in accordance with applicable IFC Performance Standards and expectations; the IFC Environmental, Health and Safety Guidelines for Mining (December 2007); the International Cyanide Management Code; and any other relevant guideline or standard.

20.5 Permitting Requirements

The Project will trigger a range of regulatory requirements and processes, which will require the application for, receipt of, and compliance with a variety of environmental permits and approvals from the relevant Ghanaian authorities. WSP provided a high-level overview of the permitting and compliance requirements associated with the proposed Project in the 2016 PEA (McCracken et. al., 2016).

20.5.1 Environmental Approvals

Environmental legislation in Ghana is well developed and enforced by the Environmental Protection Agency (EPA). The overarching environmental legislation in Ghana is the Environmental Protection Agency Act (490 of 1994) (EPAA). Environmental Assessment Regulations (EAR) regulations are provided for by Legal Instrument 1652 of 1999 (LI 1652), which facilitates the regulation and monitoring of mineral operations within the country.

In line with the Mining Act (703 of 2006), environmental approvals are required to be obtained from relevant environmental agencies. Table 20-1 provides an overview of the primary environmental approvals needed for mining operations.

Table 20-1: Primary Environmental Approvals for Mining Operations

Legislation and Regulatory Institution	Approval Required
Environmental Assessment Regulations, 1999 (LI 1652) The Environmental Protection Agency is responsible for the enforcement of environmental regulations.	**Environmental Permit:** ▪ In accordance with the Mining Act (Act 703 of 2006) and the Environmental Assessment Regulations, 1999 (LI 1652) of the EPA, a holder of a mineral right requires an Environmental Permit from the Environmental Protection Agency (EPA) in order to undertake any mineral operations **Environmental Management Plan Approval:** ▪ An EMP must be submitted within 18 months of commencement of operations and updated every three years (Regulation 24 of LI 1652). **Environmental Certificate:** ▪ This must be obtained from the EPA within 24 months of commencement of an approved undertaking (Regulation 22 of LI 1652) a. **Approved Reclamation Plan:** ▪ Mine closure and decommissioning plans have to be prepared and approved by the EPA (Regulation 14 of LI 1652). **Reclamation bond:** ▪ Mines must post a reclamation bond based on an approved reclamation plan (Regulation 22 of LI 1652).
Water Resources Commission Act, 1996 (Act 522). The Water Resources Commission (WRC) is responsible for the regulation and management of the use of water.	**Approvals for water usage:** ▪ Under Section 17 of the Mining Act (Act 703 of 2006), a holder of a mineral right may obtain, divert, impound, convey and use water from a watercourse or underground reservoir on the land of the subject of the mineral right, subject to obtaining the requisite approvals under Act 522. ▪ This application must be undertaken in terms of Section 2 of the Water Use Regulations (LI 1692, 2001), and should be included in the environmental assessment and permit.
The Forest Protection Act, 2002 (Act 624) Regulated by the Forestry Commission	**Authorization:** ▪ In accordance with Section 18 of the Mining Act (Act 703 of 2006), a holder of a mining right must obtain necessary approvals from the Forestry Commission. ▪ These are generally for the destruction of trees within National Forest Reserves, but the Forestry Commission should be consulted with to confirm.

Note: To obtain the Environmental Certificate, the applicant will have to submit evidence of: (a) actual commencement of operations; (b) acquisition of other permits and approvals where applicable; and (c) compliance with mitigation commitments indicated in the environmental impact statement or preliminary environmental report;

The ESIA is the basis for the Environmental Permit (authorized by the EPA). The Environmental Impact Statement (EIS) must be submitted to the EPA for consideration, and should consider all aspects listed under Approval Requirements in Table 20-1.

Fees are required to be paid for the EPA process, which are dependent on the total Project value.

20.5.1.1 Mining Regulations

In 2012, new regulations relating to the implementation of mining activities in Ghana were promulgated, including three relevant to the management of socio-economic aspects. These regulations are regulated by the Minerals Commission of Ghana. An overview is provided in Table 20-2.

Table 20-2: Mining Regulations Specific to Socio-Economic Aspects

Legislation	Relevance
Mining General Regulations 2012 (LI2173)	Promotes preferential employment of Ghanaians and preferential procurement of goods and services from Ghanaian service providers. Mines are required to develop the following localization plans: **Recruitment and training plan** - including measures to be implemented to achieve the targets given in the regulations (no expatriates in unskilled positions and an only very small percentage of staff in other positions may be expatriates); and **Procurement plan** - including proposed targets for local procurement of goods and services.
Mines (Support Services) Regulations, 2012 (LI 2174)	Support services for mining operations should be sourced preferentially from Ghanaians
Mines (Compensation & Resettlement) Regulations, 2012 (LI 2175)	Pertains to planning and implementation of resettlement programmes. Includes the principles for compensation and requires that displaced people are resettled to suitable alternative land and that their livelihoods and living standards are improved. Details activities that should be undertaken to develop a resettlement plan, including stakeholder engagement. The resettlement plan must be approved by the District Assembly (Planning Authority), first and then by the Minister responsible for Mines. Affected people are resettled from the area before activity is undertaken in the area.

20.5.1.2 Monitoring Requirements

Environmental Protection Agency

- Annual reports: Mines must submit annual environmental reports to the EPA.
- Inspections: The EPA undertakes regular inspections to ensure that mineral right holders are compliant with permit conditions and the environmental laws generally.
- Enforcement: The EPA is empowered to suspend, cancel or revoke an environmental permit or certificate and/or even prosecute offenders when there is a breach.

Water Resources Commission (WRC)

The monitoring and regulation of water resource use is provided for in the Water Use regulations (LI 1692, 2001). Full records must be kept by the permit holder of all water that is abstracted, diverted or stored. The WRC has the authority to inspect works and ascertain the amount of water abstracted.

Both the Water Resources Commission Act (Act 552 of 1996), and the Water Use regulations (LI 1692, 2001), provide for the enforcement of sanctions for breaches.

- Inspections: The WRC has the authority to inspect works and ascertain the amount of water abstracted.
- Enforcement: Both Act 522 and LI 1692 prescribe sanctions for breaches.

Forestry Commission

There are no generic requirements identified in the legislation; however, the conditions of the Environmental Permit (obtained from EPA) may stipulate specific management and mitigation measures relating to Forestry resources.

Mining Commission

- Reporting:
  - Monthly, six-monthly and annual reports on Ghanaian and expatriate staff numbers as well as information on payments of salaries and wages, royalty and corporate tax;
  - Mines are also required to submit audited annual financial reports for review.

20.6 Closure Planning

20.6.1 Closure Objectives

Closure objectives should be defined early in the mine planning process and integrated into all activities throughout the life-of-mine. The aim of closure it to ensure that current and future impacts of mining are firstly avoided, then reduced and managed appropriately. High-level closure objectives are outlined below:

- Achieve legal compliance as a minimum requirement.
- All areas are to be made stable and safe to humans and animals.
- Where possible, the land is to be returned to its pre-mining land use.
- The land is to be developed in accordance with the socio-economic demands of the local area and district.
- Impacts on the local community are to be minimized as far as possible.
- Closure must be undertaken in consultation with local communities, authorities, and employees.
- Current best practices are to guide the closure plan, and latest developments in mine rehabilitation sciences are to be considered.

20.6.2 Closure Options and End Uses

The post-closure land use(s) should represent the Best Practicable Environmental Options and should balance the needs of the mining company, local communities and authorities. Defining the end use(s) requires consultation with all key stakeholders. Land rehabilitated and returned to its pre-mining conditions will have significant positive impacts on the sustainable livelihoods of the affected communities.

The pre-mining land use is predominantly cocoa plantations in addition to other locally grown food crops. The land is significantly disturbed, and ecological habitats already altered. The mine closure strategy should seek to restore the current land use activities and develop opportunities to provide protective ecological habitats thereby balancing local community needs and biodiversity enhancement.

Anticipated closure options in respect of the mine related infrastructure are presented in Table 20-3.

Table 20-3: High-Level Anticipated Closure Options

Component	End of Life Closure Scenarios (options listed in decreasing cost)
Open Mine Pits: ▪ Boin ▪ Sewum ▪ Nyam ▪ Kwakyekrom	Infilling of the open pits with the mine dump material (waste rock). The end use would be to return the land to a natural shape in line with the local topography, resulting in the restoration of the pre-mining land use (farming).
Mine Dumps ▪ Boin ▪ Sewum ▪ Nyam ▪ Kwakyekrom	Mine dump material to be used to infill the open mine pits, resulting in the restoration of the mine pits' areas to the pre-mining land use (farming). Any remaining material to be shaped to blend in with natural surroundings. Progressive and targeted revegetation to be implemented to stabilize the land form.
Mining-related infrastructure: ▪ Centralized heap leach facility (primary and secondary crusher; agglomerator; carbon plant; heap leach pads and ponds)	Decontaminate (where required) dismantle and demolish all mining related infrastructure and rehabilitate the footprints to pre-mining conditions (farming of similar crops variants such as cocoa).
Non-mining related infrastructure: ▪ Operations camp – limited (offices, stores, workshops) ▪ Additional infrastructure - fuel station, explosive magazine, laboratory ▪ Camp services – limited (water purification system and storage; localized sewage treatment and storage) ▪ Fencing	Dismantle and demolish infrastructure (unless provision for handover to local communities is agreed to / planned for). Footprints are to be restored to pre-mining conditions.
Landfill (mine specific)	Utilize existing Enchi landfill, as such no closure or rehabilitation implications.
Roads: ▪ Haul roads ▪ Operational roads (gravel / unpaved)	Haul roads, which will be well constructed, will be retained for use by local communities after mine closure. Operational roads (pit access, etc.) will be rehabilitated.
Services: ▪ Power lines, water and sewer infrastructure	Dismantle and demolish infrastructure not retained by the local communities.
Human Resources	Development and implementation of: ▪ Employee strategy to manage the communication and consultation process ▪ Stakeholder strategy and liaison programme ▪ Engage with key stakeholders prior to and during closure.

Ideally, the rehabilitation of mine dump facilities should occur progressively during the operational phase, especially as the Life of Mine is relatively short (11 years).

Rigorous physical monitoring and maintenance is to be undertaken typically for a three- to five-year period after closure. This must include water quantity and quality monitoring (covering surface water, groundwater and open pit water), and monitoring the success of the rehabilitation and revegetation of the tailings facilities, dumps and general areas.

21. CAPITAL AND OPERATIONS COSTS

Capital and operating costs have been estimated for the Project. These costs were developed in support of a projected cash flow for the operation, which would assess the financial viability of the project. The capital cost estimates are based on the PEA and address the engineering, procurement, construction, and start-up of the mine and processing facilities, as well as the ongoing sustaining capital costs. The operating cost estimate includes the cost of mining, processing, waste management, and related G&A services.

The capital and operating cost estimates were developed for a conventional open pit mine, heap leach process plant and supporting infrastructure for an operation capable of treating 6.6 Mt of material per annum.

The capital and operating cost estimates presented in this Study are based on the contract mining services mainly for the mining related activities. Some operation areas, such as processing and environmental (monitoring and compliance) will remain under the Owner's responsibilities. The following sections present the responsibilities of the Owner and the Contractor assumed for the economic evaluation.

Contractor costs were estimated based on budgetary quotes and unit rates provided by three specialized contractors experienced in Ghana.

All capital and operating cost estimates cited in this Report are referenced in United States dollars as of Q1 2021 and, unless otherwise stated, are referred as "$".

21.1 Cost Estimate Accuracy

The potential variance of the actual costs compared to the cost estimates developed in this analysis (the Cost Estimate Accuracy) is dependent upon the level of engineering, the estimating methodology, and the degree to which the project implementation activities have been estimated.

The Cost Estimate Accuracy of the Project capital cost estimate is within a +/-35% range of the overall project costs, as of Q1 2021.

21.2 Exclusions

The following were not included in this estimate:

Costs associated with scope changes.

- Escalation beyond 2021 Q1;
- Financing costs;
- Cost associated with schedule delays such as those caused by:
  - Scope changes;

  - Unidentified ground conditions;
  - Labour disputes.
- Accommodations for local labour;
- Environmental permitting activities;
- Permits; and
- Sunk costs.

21.3 Owner's and Contractor responsibilities

21.3.1 Capital Costs

21.3.1.1 Owner's Responsibility

Direct Costs:

- Mining: Four light vehicles purchase and replacement, software, communications.
- Processing: Crushing, agglomeration, heap stacking, heap leach pads, ponds and gold recovery plant.
- Infrastructure: Office blocks, assay lab, mine dry for process MP, water supply, water ponds (drainage), septic tank/field for non-mining infrastructure, electrical distribution, communication, IT.
- Environmental: Costs unchanged from Owner Operated scenario.
- Mine closure and rehabilitation:
  - Only moveable equipment and infrastructure are removed by Contractor;
  - Contractor assumes nominal site clean-up;
  - Costs unchanged from Owner Operated scenario.

Indirect Costs:

- EPCM:
  - EPCM services were also factored from the direct costs and include project management and project controls, engineering services, and procurement services.
  - Mining: A fixed cost for initial EPCM (Year 0) was estimated and 10% factor applied from Year 1 onward.
  - Processing: A 10% factor applied from Year 0 onward.
  - Infrastructure: most capital cost remains as a capital item, either within the "Site Establishment" for Contractor or "Site Infrastructure" for Owner. A 10% factor applied from Year 0 onward.
  - Environmental: A 10% factor applied from Year 0 onward.

- Construction Indirect:
  - Construction indirect costs were factored from the total direct costs at 8% and are applied on initial capital and sustaining capital.
  - Construction indirect costs typically include first fill (mine, process, others), security, health and safety, environmental compliance, room and board, construction management staff (owner's team), training, recruiting, lodging of employees, mobile and communication, internet, fuel cost for owners' vehicles, repairs, ancillary equipment rental for construction (light towers, pumps, generators, etc.), utilities (water and waste).
- Owner's costs:
  - Owner's costs include expenses for the project to advance the Project during the early construction stage. Calculated as 2.75% of direct capital costs, prior to contingency in Years 0 and 1.
- Contingencies:
  - Contingency is defined as additional capital costs allowed for over and above the base estimate, to account for unexpected items and unforeseen activities and requirements not anticipated in the cost estimate. Contingencies were factored from the total direct costs at 30% and are applied on initial capital and sustaining capital..

21.3.1.2 Contractor's Responsibility

The mining contractor would provide all the administration and maintenance facilities to support mining activities. They would also remove all the equipment and movable buildings at the end of the project. The mobilization and demobilization costs are thus included as a Capital expenditure.

21.3.2 Operating Costs

21.3.2.1 Owner's Responsibility

- Engineering, management;
- Fuel, tires, maintenance for Owner's equipment and vehicles (ex.: light vehicles);
- Processing costs;
- Process related infrastructure costs;
- Environmental (monitoring, compliance);
- General & Administrative: nursing, non-contractor employees, site maintenance (electricity and plumbing), HR for non-contractor activities, IT, social and corporate responsibilities (SCR);
- Water supply;
- Power for Contractor's buildings: allowance of $50 K/year; and
- Mineral tenure and royalties.

21.3.2.2 Contractor's Responsibility

- All mining related activities including maintenance;
- Cost related to Contactor's infrastructure except power and water supply;
- Road maintenance (excluding Enchi existing road); and
- Monthly management fees.

21.4 Capital Costs

21.4.1 Summary

An initial capital expenditure of \$97 M (including 30% contingency on direct costs) has been estimated to construct the Project, with a further \$23 M in sustaining capital during operations, \$23 M for closure (including reclamation) and \$14 M of salvage value. The capital cost estimate is based on an open pit mining and heap leach operation processing 6.6 Mt/a utilizing contract mining. Capital costs are detailed in Table 21-1.

Table 21-1: Capital Cost Estimate Details

Description	Initial (K \$)	Sustaining (K \$)	Closure (K \$)	LOM (K \$)
Direct Costs				
Mining	2,576	270	796	3,642
Processing	55,264	13,405	0	68,669
Environmental (1)	0	0	15,053	15,053
Infrastructure	5,726	1,719	0	7,445
Salvage Value (2)	0	0	0	-14,106
Total Direct Costs	**\$63,566**	**\$15,394**	**\$15,849**	**\$80,703**
Indirect Costs				
Engineering and Procurement	7,371	1,539	1,545	10,456
Construction Indirect	4,879	1,210	1,204	7,293
Owner’s Cost	1,748	0	0	1,748
Total Indirect Costs	**\$13,998**	**\$2,749**	**\$2,750**	**\$19,497**
Capital Costs Pre-Contingency	**\$77,564**	**\$18,144**	**\$18,599**	**\$100,200**
Contingency: 30% of Direct Costs (3)	19,070	4,618	4,755	28,443
Total Capital Costs	**\$96,634**	**\$22,762**	**\$23,353**	**\$128,643**

(1) Environmental includes closure and remediation works in Years 11 and 12, as well as post closure maintenance for three years.

(2) Salvage value recovered in Year 12, assumes 20% of processing costs and 5% of Owner’s infrastructure costs recouped.

(3) Contingency not applied to salvage value

This capital cost estimate is based on industry standard estimates. Capital cost estimates were developed using budgetary quotes provided by contractors experienced in Ghana, as well as updated estimates from historical reports.

Construction is estimated to be 15 months. The Project benefits from relatively flat terrain (rolling hills) and simple infrastructure, limiting the amount of earthworks required. The initial capital costs reflect an estimate for the design and development of the plant and mine infrastructure that includes crushing, agglomeration, heap leaching, processing ponds and a gold recovery plant. The heap leach pads will be built in three phases, with a third of the cost upfront and the remainder included in sustaining capital in years three and six.

Reclamation and closure costs have been estimated based on the preliminary infrastructure plans and are inclusive of an allowance for rehabilitation monitoring and care and maintenance for three years post completion of mining.

21.4.2 Direct Capital Cost Estimates

21.4.2.1 Mining

Table 21-2 summarizes the capital cost requirements for the mine, which are split into Owner and Contractor. The costs shown are direct costs presented without any contingency allowance.

Table 21-2: Summary of Mining Capital Cost Estimate, Contract Mining Services

Description	LOM (K $)
Mining - by Owner	**$990**
Vehicles & Equipment	810
Engineering & Miscellaneous	180
Mining - by Contractor	**$2,652**
Mobilization	1,856
Demobilization	796
Total Direct Costs	**$3,642**

Note: Numbers may not add exactly due to rounding.

21.4.2.2 Mineral Processing

A mechanical equipment list was developed using a 6.6 Mt/y design basis for the process flow diagram described in Chapter 17. Table 21-3 summarizes the process capital requirements. The capital cost estimates are based on budget pricing from suppliers, quantity take-offs from design criteria basic engineering, in-house data, and current market prices for smaller equipment component costs. The costs shown are direct costs presented without any contingency allowance and are based on equipment costs multiplied by factors to include installation costs.

Table 21-3: Summary of Process Capital Cost Estimate

Description	LOM (K $)
Primary Crushing	9,676
Secondary Crushing	4,925
Agglomeration	10,784
Heap Stacking	10,000
Heap Leach Pads	20,108
Ponds	2,152
Gold Extraction	6,054
Reagents	1,462
Services	3,509
Total Direct Costs	**$68,669**

Note: Numbers may not add exactly due to rounding.

21.4.2.3 Infrastructure

The capital cost estimate for Operations Management, Facilities for site management, and Power Water, Sewer, Telecom and Security Installations is based on historical reports costs escalated at 2.5% per year.

Capital costs for Site Preparation, Access and Water Management were calculated as follows:

- Unit costs for clearing and grubbing, new haulage road construction, and upgrading of existing roads are the average of three contractor quotations obtained by BBA;
- Haulage and access roads were laid out to follow existing roads where possible;
- Where existing roads were not available or their relocation was necessary, routes were selected to minimize the overall length and consider topography to avoid excessive grades;
- Clearing and grubbing costs for all waste dumps are based on the design area of each pile; and
- Ditching and fencing costs for all open pits and waste dumps are based on the design perimeter length for each installation.

Table 21-4 summarizes the infrastructure capital requirements. The costs shown are direct costs presented without any contingency allowance.

Table 21-4: Summary of Infrastructure Capital Cost Estimate

Description	LOM (K $)
Operations Management	618
Facilities for Site Management	598
Power, Water, Sewer, Telecom and Security Installations	661
Access Roads for Pits and Processing:	3,210
Sewum roads	1,187
Boin Roads	1,106
Nyam Roads	724
Kwakyekrom Roads	193
Site Preparation, Access and Water Management	2,357
Waste Dump Grubbing:	1,737
Sewum	484
Boin	898
Kwakyekrom	104
Nyam	250
Ditching & Fencing:	620
Sewum	216
Boin	220
Kwakyekrom	67
Nyam	118
Total Direct Costs	**$7,445**

21.4.2.4 Closure Cost

The closure cost estimates for the Project were based on the 2015 PEA values (McCracken et. al., 2016) escalated 2.5% annually for 6 years. The estimate used the following approach.

- Gathering and collating the available engineering design information to develop the Project closure context;
- Obtaining the required additional information (where required), on infrastructural layouts and mine planning;
- Compiling a high-level bill of quantities (BoQ), or inventory, based on the available information for the projected closure scenarios on site;
- Determining applicable unit rates for the costing; and
- Compiling costing spread sheets, structured in terms of the following categories:
  - Infrastructural areas;
  - Mining areas;
  - General surface rehabilitation;
  - Water management;
  - Post-closure aspects.

The cost estimate quantities were taken from available plans and maps available. Unit rates were obtained from escalated rates from the previous PEA.

The high-level closure costs have been estimated based on preliminary infrastructure plan and the potential environmental issues likely to be associated with the proposed mining operations. An estimate of $15.1 million (direct costs) has been assumed for the study. This is inclusive of an allowance for rehabilitation monitoring and care and maintenance, as well as surface water quality and groundwater quality monitoring, to be conducted for a minimum period of two to five years post closure. This is undertaken to ensure and assess the relative success of the implemented rehabilitation and closure measures.

21.5 Operating Costs

21.5.1 Summary

Table 21-5 presents the Life-of-Mine (LOM) operating costs for the Project, which have been estimated to be $908 M. Cash costs are also presented in the table below as a separate item, and include operation costs, royalties and refining charges. Cash costs and All-in Sustaining Costs (AISC) are estimated to $1,025 M and $1,048 M, respectively for LOM.

Table 21-6 summarizes the operating costs for Year 2 to Year 5 period.

The Operating costs and Cash costs shown in this section are presented without any contingency allowance.

Table 21-5: Summary of Operating Cost Estimate (LOM)

Description	Operating Cost		
	LOM (K $)	$/tonne milled	$/oz Au
Mining	422,363	6.16	430
Processing	361,817	5.28	368
Environment & Infrastructure	5,241	0.08	5
On-Site G&A	118,284	1.73	120
Total Operating Costs	**$907,705**	**$13.24**	**$923**
Treatment & Refining Charges	3,933	0.06	4
Royalties	113,492	1.66	115
Total Cash Costs	**$1,025,130**	**$14.95**	**$1,043**
Sustaining Capital (1)	22,762	0.33	23
All-in Sustaining Costs (AISC) (2)	**$1,047,891**	**$15.28**	**$1,066**

(1) Sustaining capital excludes closure costs and salvage value

(2) AISC consists of cash costs plus sustaining capital (excluding closure costs and salvage value).

Table 21-6: Summary of Operating Cost Estimate for Years 2 to 5

Description	Operating Cost		
	Y2 to Y5 (K $)	$/tonne milled	$/oz Au
Mining	158,234	6.02	380
Processing	138,799	5.28	333
Environment & Infrastructure	1,551	0.06	4
On-Site G&A	43,409	1.65	104
Total Operating Costs	**$341,994**	**13.00**	**821**
Treatment & Refining Charges	1,667	0.06	4
Royalties	48,094	1.83	115
Total Cash Costs	**$391,754**	**14.89**	**940**
Sustaining Capital (1)	11,850	0.45	28
All-in Sustaining Costs (AISC) (2)	**$403,604**	**15.34**	**969**

(1) Sustaining capital excludes closure costs and salvage value

(2) AISC consists of cash costs plus sustaining capital (excluding closure costs and salvage value).

21.5.2 Mining

The mine operating costs were development using budgetary quotations that were provided by three mining contractors with experience in Ghana. The following average costs were used for the study, which include drilling, blasting, loading and hauling; $1.40/t mined for oxide, $2.10/t mined for transition, and $2.60/t mined for fresh rock. These unit rates consider a maximum transportation distance of 2 km.

An additional haulage cost of $0.09/t per kilometre was used for hauls greater than 2 km. This unit rate is based on experience from similar sized open pit heap leach operations and local conditions. An incremental bench haulage cost of $0.05/t per 10 m drop in elevation within the pit. Drilling and blasting considers a cost of $0.60/t blasted for transition and $0.90/t for fresh rock.

Transportation to heap leach considers a rehandle costs of $0.35/t for loading and $0.09/t per kilometre hauled.

Table 21-7 and Table 21-8 summarize the mine operating cost for the LOM and Year 2 to Year 5 respectively.

Table 21-7: Summary of Mining Operating Cost Estimate (LOM)

Description	Mining Operating Cost			
	LOM (K $)	$/tonne milled	$/tonne mined	$/oz Au
Drilling & Blasting	75,119	1.10	0.35	76
Loading & Hauling	321,326	4.69	1.52	327
Transportation to HLF	25,237	0.37	0.12	26
Stockpile Rehandle	681	0.01	0.00	1
Total	**$422,363**	**$6.16**	**$1.99**	**$430**

Note: Numbers may not add exactly due to rounding.

The following items related to the mine have been included in general and administration costs for the Project:

- Contractor monthly management fee ($44.5 million for the LOM, which resulted $0.66 per tonne milled or $0.21 per tonne mined)
- Owner's cost related to fuel, tires, and maintenance for light vehicles; and
- Owner's cost related to mine technical services and mine management salaries;

Table 21-8: Summary of Mining Operating Cost Estimate (Year 2 to Year 5)

Description	Mining Operating Cost			
	Y2 to Y5(K $)	$/tonne milled	$/tonne mined	$/oz Au
Drilling & Blasting	28,294	1.08	0.35	68
Loading & Hauling	122,969	4.68	1.53	295
Transportation to HLF	6,746	0.26	0.08	16
Stockpile Rehandle	226	0.01	0.00	1
Total	**$158,234**	**$6.02**	**$1.97**	**$380**

Note: Numbers may not add exactly due to rounding.

21.5.3 Mineral Processing

For the purposes of the PEA, processing operating costs, reagents, consumables, fuel, and power are based on similar heap leach operations. Workforce is based on the estimated personnel for a 6.6 Mt/a processing operation with 2,340 hours per year for staff and hourly plant crew. Rates from other similar projects were considered for the purposes of this study. An additional 40% to the salaries was incorporated for health benefits, pension, overtime, training, travel, etc. For the hourly plant crew, an additional 15% to the salaries was incorporated for overtime. Table 21-9 and Table 21-10 summarize the process operating costs for the LOM and Year 2 to Year 5, respectively.

Table 21-9: Summary of Process Operating Cost Estimate (LOM)

Description	Process Operating Cost		
	LOM (K $)	$/tonne milled	$/oz Au
Primary Crushing	23,388	0.34	24
Secondary Crushing	27,426	0.40	28
Agglomeration / Stacking	6,857	0.10	7
Leach Operation (Incl. Sprinklers)	20,570	0.30	21
Recovery Plant	34,248	0.50	35
General Site Maintenance	20,570	0.30	21
Cement for Agglomeration	102,849	1.50	105
Cyanide, Lime, Other Reagents	99,712	1.45	101
Process Operation Workforce	26,197	0.38	27
Total	**$361,817**	**$5.28**	**$368**

Note: Numbers may not add exactly due to rounding.

Table 21-10: Summary of Process Operating Cost Estimate (Year 2 to Year 5)

Description	Process Operating Cost		
	Y2 to Y5 (K $)	$/tonne milled	$/oz Au
Primary Crushing	8,972	0.34	22
Secondary Crushing	10,521	0.40	25
Agglomeration / Stacking	2,630	0.10	6
Leach Operation (Incl. Sprinklers)	7,891	0.30	19
Recovery Plant	13,138	0.50	32
General Site Maintenance	7,891	0.30	19
Cement for Agglomeration	39,455	1.50	95
Cyanide, Lime, Other Reagents	38,251	1.45	92
Process Operation Workforce	10,050	0.38	24
Total	**$138,799**	**$5.28**	**$333**

Note: Numbers may not add exactly due to rounding.

The following is the summary of the basis of operating cost estimate for process:

- Operating costs for crushing has been determined based budgetary quotations for wearing parts and experience from similar crushing operations and local conditions;
- Power unit cost of $0.3 per kW/h was based on similar operations; and
- Table 21-11 summarizes the reagent consumption cost.

Table 21-11: Summary Reagent Consumption Cost

Description	Reagent		
	Unit Consumption rate (kg/tonne milled)	Cost ($/Kg)	Annual cost (K $)
Cement	15.0	0.10	9,900
Sodium Cyanide	0.3	2.15	4,276
Lime	2.0	0.37	4,884
Carbon	0.01	3.21	557
Caustic Soda	0.03	1.11	200

Note: Numbers may not add exactly due to rounding.

21.5.4 Infrastructure

The infrastructure section includes an allowance for power for Contractor's buildings and the gravel road maintenance. Table 21-12 and Table 21-13 summarize the cost estimate for the LOM and Year 2 to Year 5 respectively.

Table 21-12: Summary of Infrastructure Operating Cost Estimate (LOM)

Description	Infrastructure Operating Cost		
	LOM (K $)	$/tonne milled	$/oz Au
Power; Allowance for Contractor's Building	535	0.01	0.5
Gravel Road Maintenance	3,298	0.05	3.4
Total	**$3,833**	**$0.06**	**$3.9**

Note: Numbers may not add exactly due to rounding.

Table 21-13: Summary of Infrastructure Operating Cost Estimate (Year 2 to Year 5)

Description	Infrastructure Operating Cost		
	Y2 to Y5 (K $)	$/tonne milled	$/oz Au
Power; Allowance for Contractor's Building	200	0.01	0.5
Gravel Road Maintenance	825	0.03	2.0
Total	**$1,025**	**$0.04**	**$2.5**

Note: Numbers may not add exactly due to rounding.

Following is the summary of the basis of operating cost estimate for Infrastructure:

- Road maintenance costs are based on the unit rates established for historic projects;
- Unit rates are applied to the haulage and access roads only during the periods that they are in use to support mining activities in each mining area; and
- Annual allowance for power to the Contractor's building was taken from historic projects.

21.5.5 Environment

The environment section includes staff salaries and environmental disbursements for monitoring equipment, assays and geotechnical inspection. Table 21-14 and Table 21-15 summarize the cost estimate for the LOM and Year 2 to Year 5 respectively.

Table 21-14: Summary of Environmental Operating Cost Estimate (LOM)

Description	Environmental Operating Cost		
	LOM (K $)	$/tonne milled	$/oz Au
Salaries and Wages	1,124	0.02	1.1
Monitoring and Assays	285	0.00	0.3
Total	**$1,408**	**$0.02**	**$1.4**

Note: Numbers may not add exactly due to rounding.

Table 21-15: Summary of Environmental Operating Cost Estimate (Year 2 to Year 5)

Description	Environmental Operating Cost		
	Y2 to Y5 (K $)	$/tonne milled	$/oz Au
Salaries and Wages	420	0.02	1.0
Monitoring and Assays	106	0.00	0.3
Total	**$526**	**$0.02**	**$1.3**

Note: Numbers may not add exactly due to rounding.

21.5.6 General and Administrative

The general services include general management (not included within mining and processing), accounting, human resources, purchasing, health and safety, environment, and security. Table 21-16 and Table 21-17 summarize the cost estimate for the LOM and Year 2 to Year 5, respectively.

Table 21-16: Summary of General and Administrative Operating Cost Estimate (LOM)

Description	G&A Operating Cost		
	LOM (K $)	$/tonne milled	$/oz Au
General and Administration - By Owner	**$68,430**	**$1.00**	**$70**
Salaries and Wages	51,775	0.76	53
Expenses	14,675	0.21	15
Corporate and Social Responsibility	1,980	0.03	2
General and Administration - By Contractor	**$44,940**	**$0.66**	**$46**
Contractor Monthly Management Fee	44,940	0.66	46
Total	**$113,370**	**$1.65**	**$115**

Note: Numbers may not add exactly due to rounding.

Table 21-17: Summary of General and Administrative Operating Cost Estimate (Year 2 to Year 5)

Description	G&A Operating Cost		
	Y2 to Y5 (K $)	$/tonne milled	$/oz Au
General and Administration - By Owner	**$24,884**	**$0.95**	**$60**
Salaries and Wages	18,827	0.72	45
Expenses	5,336	0.20	13
Corporate and Social Responsibility	720	0.03	2
General and Administration - By Contractor	**$16,800**	**$0.64**	**$40**
Contractor Monthly Management Fee	16,800	0.64	40
Total	**$41,684**	**$1.58**	**$100**

Note: Numbers may not add exactly due to rounding.

The study has assumed a staff level of 70 associate with owner. The expenses include a living cost allowance for expatriate and some senior level staff by renting several houses in the nearby town of Enchi. Workforce cost associate with mining contractor is included in the mining cost and contractor monthly management fee.

21.5.7 Corporate Social Responsibility

Newcore acknowledges the importance of strong corporate social responsibility. The exact terms of Newcore's contribution to social development and initiatives focused on betterment of the local communities will be negotiated with government and local proponents. For the purpose of the PEA, an annual total of $180,000 has been allocated for corporate and social responsibility (~$2/oz of gold produced LOM).

22. ECONOMIC ANALYSIS

22.1 Introduction

This PEA is preliminary in nature. The economic analysis presented in this chapter is based entirely on Inferred Mineral Resources. There is no certainty that this PEA, which is based on these Inferred Mineral Resources, will be realized.

A preliminary economic analysis has been carried out for the Project by Todd McCracken, P. Geo., using a cash flow model. The model is constructed using annual cash flows by taking into account annual processed tonnages and grades for the heap leach feed. The associated process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining) were also taken into account. The price forecast of gold is given in US$. The financial model was completed on a 100% project basis and includes a 5% gross royalty to the Ghanaian Government and a 2% NSR royalty to Maverix. As a general rule, the financial assessment of projects of this nature is carried out on a “100% equity” basis, i.e. the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Results are given before and after taxation. Current Ghana tax regulations are applied to assess the corporate and mining tax liabilities. The Government of Ghana has the right to a 10% free carried interest in the Project. All amounts in this chapter are presented in United States Dollars (US$) and unless otherwise stated, are referred to “$”.

Discounting has been applied from the first year of mine construction (Year 0 onward).

The model reflects the base case and technical assumptions shown in the preceding chapters of this report.

This PEA presents the economic analysis for a Contract Mining scenario. The cost centres attributed to the mining contractor and those remaining under the Owner’s responsibility are presented in Chapter 21.

22.2 Assumptions

The assumptions used in the economic analysis are summarized in Table 22-1. The price forecast of gold has been discussed in Chapter 19. The heap leach throughput for Year 1 is reduced to 4.87 Mt/a to account for a typical operating ramp up period.

Table 22-1:Technical Assumption

Description	Unit	LOM
Plant throughput - Year 1	Mt/a	4.87
Plant throughput - Year 2, onward (average)	Mt/a	6.62
Gold Price	$/oz	1,650
Discount Rate	%	5
Corporate Income Tax	%	35
Depreciation	%	20
Government Participation on the Project	$/oz	4.00

22.2.1 Capital Expenditures

Capital expenditures for the Project have been scheduled according to the execution schedule with the majority of the expenditure happening within 15 months, and mine production starting three months before the initial capital construction is completed. Sustaining capital for the mining fleet, leach pad expansion, and future haul roads have also been included in the capital expenditures.

22.2.2 Project Exploitation Permits

Upon the grant of the exploitation permit, and in accordance with Mining Regulation, the Government of Ghana is entitled to retain a 10% free carried equity interest in the Project, which interest may not be diluted even if there is an increase in the share capital, under Section 8 of the Ghanaian Mining Act. The 10% equity interest is paid to the government when a dividend is declared.

The Government of Ghana is also entitled to collect a 5% royalty (calculated based on the international market value of gold) on the revenues from gold production on the Property covered by the exploitation permit. The Government also collects various taxes and duties on the importation of fuels, supplies, equipment, and outside services as specified in the Mining Code.

22.2.3 Royalties

The present economic analysis incorporates a royalty agreement with Maverix Metals Inc. This agreement consists of a 2% NSR royalty of which 1% is subject to a buy-back option for a lump sum of $3.5 M held by Sandstorm Gold Ltd. The economic analysis considers royalties in full for the entire mine life.

22.2.4 Depreciation

Depreciation allowance is considered at a rate of 20% over five years and it accumulates in the capital expenditures tax pool if it is not utilized within the qualifying year of assessment. The remaining capital asset tax pool is fully deductible in the final year of production.

22.2.5 Inflation

In line with the practice in the mineral industry, no inflation was applied to the cash flow analysis.

22.2.6 Currency

Cash flow analysis is reported in United States dollars ($) and where applicable, the exchange rates shown in Table 22-2 were used.

Table 22-2: Exchange Rate

Description	Unit	Value
Ghana	US$/Cedi	5.7593
South Africa	US$/Rand	15.7303

22.2.7 Salvage Value

For the Owner Operated Scenario, the Project equipment salvage value has been assumed at 20% on the following costs:

- Processing mechanical equipment;
- Electrical and communications;
- Fuel storage facility;
- Site vehicles and mobile equipment.

For other infrastructure such as buildings with structural steel and sheeting, plate works, and modular buildings, the salvage value is assumed at 5%.

There is no salvage value for earth works or remediation works.

22.2.8 Power Rates

A power rate of $0.30 per kW/h is used in the economic evaluation. The rate is based on similar operations.

22.2.9 Operating Costs

During the last year of production, a reduction of all constant annual operating costs (such as infrastructure cost and G&A) was applied for the last year to reflect the declining production.

22.2.10 Mineral Tenure Fees

Ghana requires the payment of annual fees for mining rights and ground rents (occupation of land). The mining rights apply on the entire claims with mining activity at $33.33 per hectares for the first two years, then $47.62 per hectares from the third year onward. The ground rent applies to the total surface of disturbed areas for a fee of 9,019 Cedi per square kilometre (approximately $1,566/km^2).

22.3 Financial Model and Results

The technical parameters, production forecasts and estimates described elsewhere in this report are reflected in the LOM Project cash flow model in Table 22-3.

Figure 22-1 shows the pre-tax cash flows as well as the cumulative cash flow over the Project's life. Figure 22-2 shows the post-tax cash flows as well as the cumulative cash flow over the Project's life. The payback period corresponds to the time at which the cumulative cash flow becomes positive: during Year 2 both the pre-tax model and post-tax model.

The cash flows occurring in Years 11 and 12 are related to the environmental closures costs. The cash flows occurring in Years 13 and 14 are expenses for environmental monitoring ($133,000/year) following final closure of the mine.

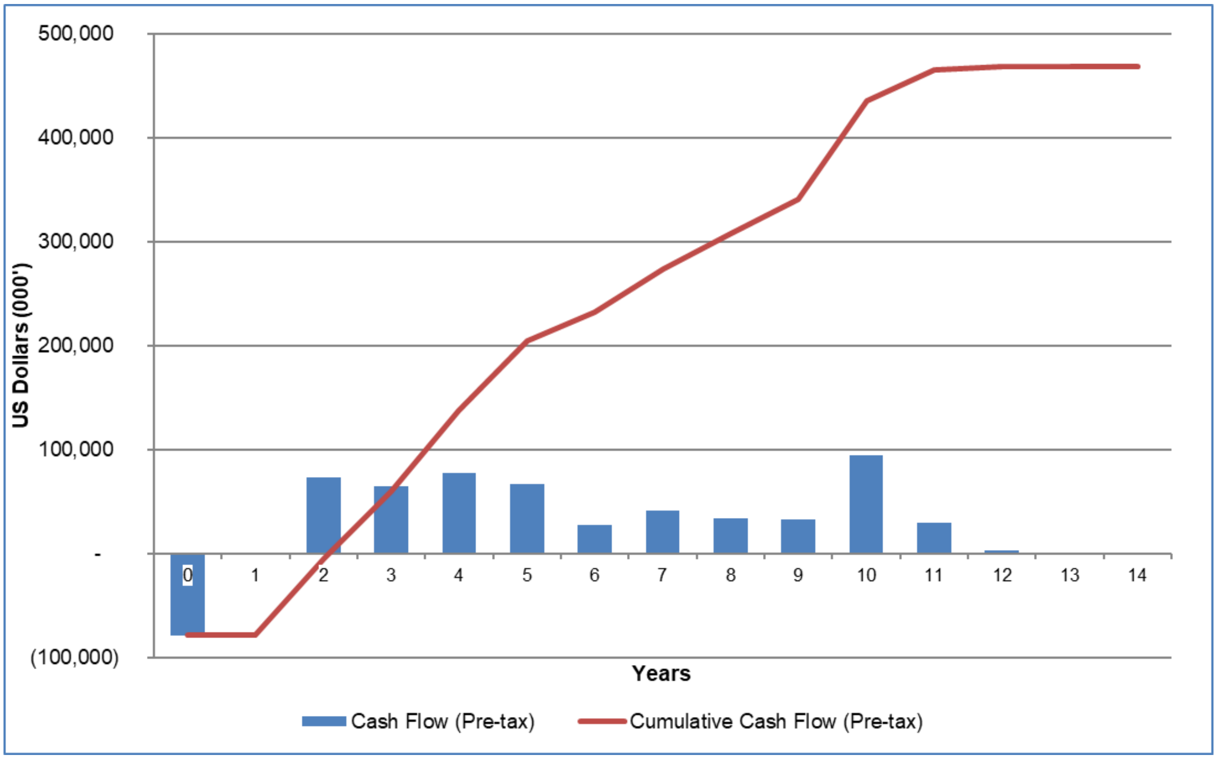


Figure 22-1: Pre-tax Cash Flow and Cumulative Cash Flow

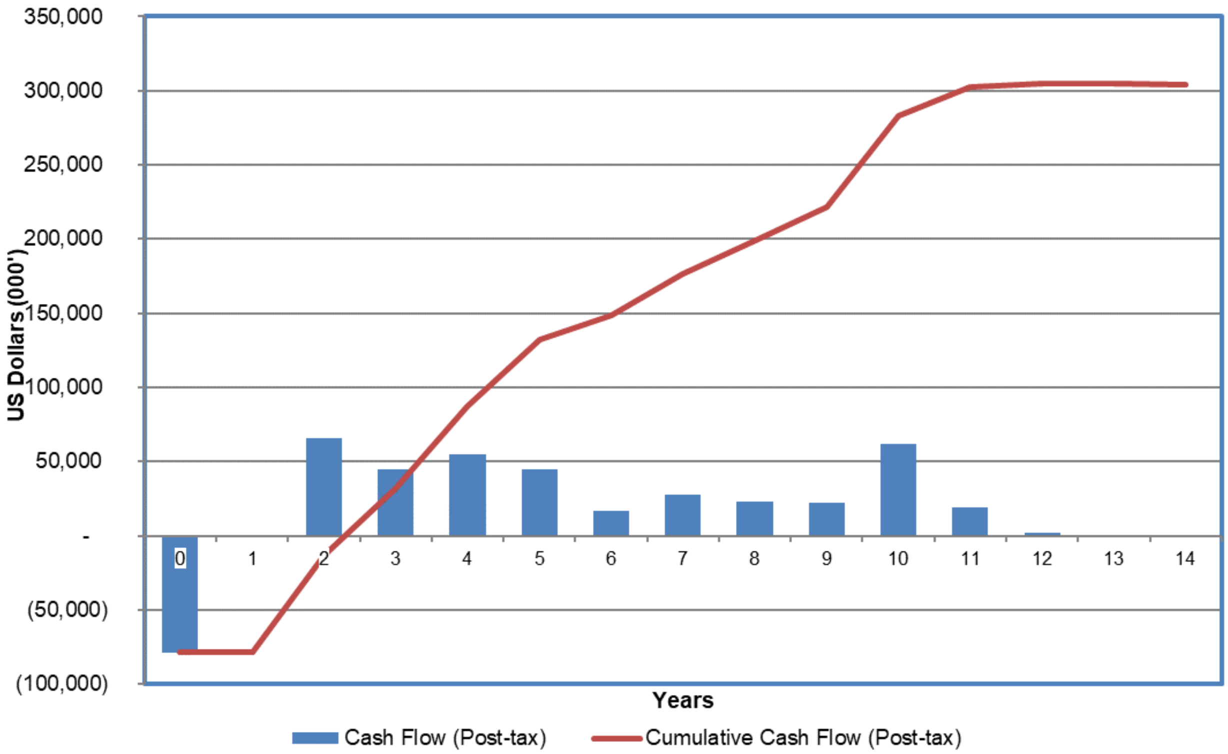


Figure 22-2: Post tax Cash Flow and Cumulative Cash Flow

Table 22-3: Cash Flow Statement

	Unit	Total/Average	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
REVENUES ($1,650/oz Gold Price)																	
Gold Production	oz	983,296	-	50,660	104,638	104,034	104,286	103,728	79,964	87,639	83,384	79,647	121,387	63,929	-	-	-
Gold Dore (Gross Revenues)	K $	1,622,438	-	83,588	172,652	171,656	172,072	171,151	131,941	144,605	137,584	131,418	200,288	105,482	-	-	-
Refining Charges, Dore Transport, Insurances	K $	3,933	-	203	419	416	417	415	320	351	334	319	486	256	-	-	-
Royalties	K $	113,492	-	5,847	12,077	12,008	12,037	11,972	9,229	10,115	9,624	9,193	14,010	7,379	-	-	-
Net Revenues	K $	1,505,013	-	77,539	160,156	159,232	159,619	158,763	122,392	134,139	127,627	121,907	185,792	97,848	-	-	-
OPERATING COSTS																	
Mining	K $	422,363	-	22,790	40,591	37,880	34,466	45,298	37,677	45,248	45,977	42,859	45,245	24,332	-	-	-
Processing	K $	361,817	-	25,679	34,700	34,700	34,700	34,699	34,987	35,398	35,723	34,700	34,700	21,831	-	-	-
Site Infrastructure & Environmental	K $	5,241	-	390	388	381	393	389	786	750	789	373	317	286	-	-	-
On-Site G&A (incl. Mineral Tenure Fees)	K $	116,304	-	10,571	10,571	10,706	10,706	10,706	10,708	10,708	10,709	10,709	10,709	9,449	18	18	18
Corporate & Social Responsibility	K $	1,980	-	180	180	180	180	180	180	180	180	180	180	180	-	-	-
Total Operating Costs	K $	907,705	-	59,609	86,430	83,847	80,445	91,271	84,337	92,284	93,379	88,820	91,150	56,077	18	18	18
Total Operating Costs	$/oz	923	-	1,177	826	806	771	880	1,055	1,053	1,120	1,115	751	877	-	-	-
Total Cash Cost (Operating + Refining + Royalties)	$/oz	1,043	-	1,296	945	925	891	999	1,174	1,172	1,239	1,235	870	997	-	-	-
CAPITAL COSTS																	
Direct Costs																	
Mining	K $	3,642	1,856	720	-	-	-	-	270	-	-	-	-	796	-	-	-
Processing	K $	68,669	44,211	11,053	-	6,703	-	-	6,703	-	-	-	-	-	-	-	-
Site Infrastructure & Environmental	K $	22,498	5,508	218	-	130	1,041	133	-	415	-	-	-	7,329	7,461	132	132
Salvage Value	K $	(14,106)	-	-	-	-	-	-	-	-	-	-	-	-	(14,106)	-	-
Total Direct Costs	K $	80,703	51,575	11,991	-	6,832	1,041	133	6,973	415	-	-	-	8,125	(6,645)	132	132
Indirect Costs																	
EPCM	K $	10,456	6,172	1,199	-	683	104	13	697	41	-	-	-	813	733	-	-
Construction Indirect	K $	7,293	3,978	902	-	547	83	11	536	33	-	-	-	586	597	11	11
Owner's Cost	K $	1,748	1,418	330	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Indirect Costs	K $	19,497	11,568	2,430	-	1,230	187	24	1,233	75	-	-	-	1,399	1,330	11	11
Total Capital Costs Pre-Contingency	K $	100,200	63,143	14,421	-	8,062	1,228	157	8,206	490	-	-	-	9,524	(5,316)	142	142
Contingencies on Direct Costs (30%) (1)	K $	28,443	15,473	3,597	-	2,050	312	40	2,092	124	-	-	-	2,438	2,238	39	39
Total Capital Costs	K $	128,643	78,615	18,018	-	10,112	1,541	197	10,298	614	-	-	-	11,962	(3,078)	182	182
Total AISC (Cash Cost + Sustaining Capital) (2)	$/oz	1,066	-	1,296	945	1,023	906	1,001	1,303	1,179	1,239	1,235	870	997	-	-	-
CASH FLOW																	
Pre-Tax Cash Flow	K $	468,665	(78,615)	(89)	73,726	65,274	77,633	67,295	27,756	41,240	34,248	33,086	94,642	29,809	3,060	(199)	(199)
Corporate Income Tax	K $	164,340	-	-	7,996	20,309	22,535	22,200	11,107	13,502	11,387	11,054	32,608	10,627	1,015	-	-
Post-Tax Cash Flow	K $	304,326	(78,615)	(89)	65,730	44,965	55,098	45,095	16,650	27,738	22,860	22,033	62,034	19,182	2,045	(199)	(199)

(1) Contingency not applied to salvage value.
(2) AISC consists of cash costs plus sustaining capital (excluding closure cost and salvage value).

Note: Numbers may not add exactly due to rounding.

The financial results of the Project are summarized in Table 22-4 to Table 22-6.

On a pre-tax basis, the Project has a Net Present Value (NPV) of $332.7M at a discount rate of 5%, an Internal Rate of Return (IRR) of 54%, and a payback period of 2.1 years. On a post-tax basis, the NPV is $212.5M at a discount rate of 5%, the IRR is 42%, and the payback period is 2.3 years.

Table 22-4: Summary of Financial Analysis

Description	Unit	LOM	Y2 to Y5
Tonnage Ore Feed	k tonnes	68,566	26,303
Feed Grade Processed	g/t Au	0.57	0.62
Gold Recovery (average)	%	79%	80%
Production Period	year	10.6	4.0
Tonnage Waste Rock	k tonnes	143,490	54,216
Stripping Ratio	-	2.09	2.06
Gold Production	ounce	983,296	416,685
Annual Gold Production (LOM) (average)	oz/y	92,530	104,171
Gold Production (Gross Revenues)	K $	1,622	687,531
Net Revenues (1)	K $	1,505	637,770
Total Operating Cost (2)	K $	907,705	341,994
Total Cash Costs (Operating + Refining Charges and Royalties)	K $	1,025,130	391,754
Total Capital Costs with contingencies	K $	128,643	11,850
Initial Capital Costs (3)	K $	96,634	-
Sustaining Capital Costs	K $	32,009	11,850
All-in Cost (Cash Costs + Capital Costs)	k tonnes	1,153,773	403,604

Notes:
(1) Includes refining charges and royalties
(2) Includes fees for mineral tenure
(3) Initial Capital Costs are for a period of 15 months in Year 0 and Year 1
Numbers may not add exactly due to rounding.

Table 22-5: Pre-tax Financial Results

Description	Unit	LOM
Total Cash Flow	K $	468,665
NPV @ 5%	K $	332,710
Pre-Tax IRR	%	54
Payback Period (from start of construction)	year	2.1

Note: Numbers may not add exactly due to rounding.

Table 22-6: Post tax Financial Results

Description	Unit	LOM
Total Cash Flow	K $	304,326
NPV @ 5%	K $	212,466
Pre-Tax IRR	%	42
Payback Period (from start of construction)	year	2.3

Note: Numbers may not add exactly due to rounding.

Table 22-7 summarizes other relevant average costs.

Table 22-7: Average Costs Summary

Description	$/t Feed (LOM)	$/oz (LOM)	$/t Feed (Y2 to Y5)	$/oz (Y2 to Y5)
Total Operating Cost	13.24	923	13.00	821
Total Cash Cost (Operating + Refining Charges and Royalties)	14.95	1,043	14.89	940
Total Capital Cost	1.88	131	0.45	28
Initial Capital Costs	*1.41*	*98*	-	-
Sustaining Capital Costs (1)	*0.47*	*33*	*0.45*	*28*
All-in Cost (Cash Costs + Capital Costs)	16.83	1,173	15.34	969

Notes:
(1) Includes salvage and closure costs.
Numbers may not add exactly due to rounding.

22.4 Sensitivity Analysis

A sensitivity analysis was performed on the pre-tax profits by varying the major key variables to a range of a percentage of the base case cash flow and each sensitivity analysis was performed independent of the other. Cash flows were discounted using the base case rate of 5% but also a 0% rate to reflect the total cash flow. The cash flow was also discounted using a rate of 10% as a measure of sensitivity of that economic parameter as well.

The results of the sensitivities on the pre-tax model are summarized in Table 22-8 to Table 22-11. In each table, the base case is shown using bold font.

Table 22-8: Gold Price Sensitivity (Percent Change from Base Case)

Description	Unit	-20%	-10%	0%	+10%	+20%
Gold Price Modelled	$/ounce	$1,320	$1,485	**$1,650**	$1,815	$1,980
Pre-Tax NPV 5%	M $	$105	$219	**$333**	$447	$560
Pre-Tax IRR	%	24%	40%	**54%**	67%	79%
Pre-Tax Payback	year	3.4	2.6	**2.1**	1.8	1.6
After-Tax NPV 5%	M $	$61	$138	**$212**	$287	$361
After-Tax IRR	%	18%	31%	**42%**	52%	62%
After-Tax Payback	year	3.7	2.8	**2.3**	2.0	1.8
After-Tax Cash Flow	M $	$105	$206	**$304**	$402	$500

Note: Numbers may not add exactly due to rounding.

Table 22-9: Gold Price Sensitivity ($100 Incremental Gold Price Change from Base Case)

Description	Unit	$1,450	$1,550	$1,650	$1,750	$1,850	$1,950
Pre-Tax NPV 5%	M $	$195	$264	**$333**	$402	$471	$540
Pre-Tax IRR	%	36%	45%	**54%**	62%	69%	77%
Pre-Tax Payback	year	2.7	2.3	**2.1**	1.9	1.7	1.6
After-Tax NPV 5%	M $	$123	$168	**$212**	$257	$302	$347
After-Tax IRR	%	29%	36%	**42%**	48%	54%	60%
After-Tax Payback	year	3.0	2.6	**2.3**	2.1	1.9	1.8

Note: Numbers may not add exactly due to rounding.

Table 22-10: Capital Cost Sensitivity

Description	Unit	+20%	+10%	0%	-10%	-20%
Pre-Tax NPV 5%	M $	$311	$322	**$333**	$344	$354
Pre-Tax IRR	%	45%	49%	**54%**	59%	65%
Pre-Tax Payback	year	2.4	2.2	**2.1**	1.9	1.8
After-Tax NPV 5%	M $	$198	$205	**$212**	$220	$227
After-Tax IRR	%	35%	39%	**42%**	46%	51%
After-Tax Payback	year	2.6	2.4	**2.3**	2.1	2.0
After-Tax Cash Flow	M $	$289	$297	**$304**	$312	$320

Note: Numbers may not add exactly due to rounding.

Table 22-11: Operating Cost Sensitivity

Description	Unit	+20%	+10%	0%	-10%	-20%
Pre-Tax NPV 5%	M $	$197	$265	**$333**	$401	$469
Pre-Tax IRR	%	37%	45%	**54%**	61%	69%
Pre-Tax Payback	year	2.7	2.3	**2.1**	1.9	1.7
After-Tax NPV 5%	M $	$124	$168	**$212**	$257	$301
After-Tax IRR	%	29%	36%	**42%**	48%	54%
After-Tax Payback	year	2.9	2.6	**2.3**	2.1	1.9
After-Tax Cash Flow	M $	$187	$246	**$304**	$363	$422

Note: Numbers may not add exactly due to rounding.

Figure 22-3 shows the detailed sensitivity analysis of changing the key variables to the above-mentioned percentage of variation.

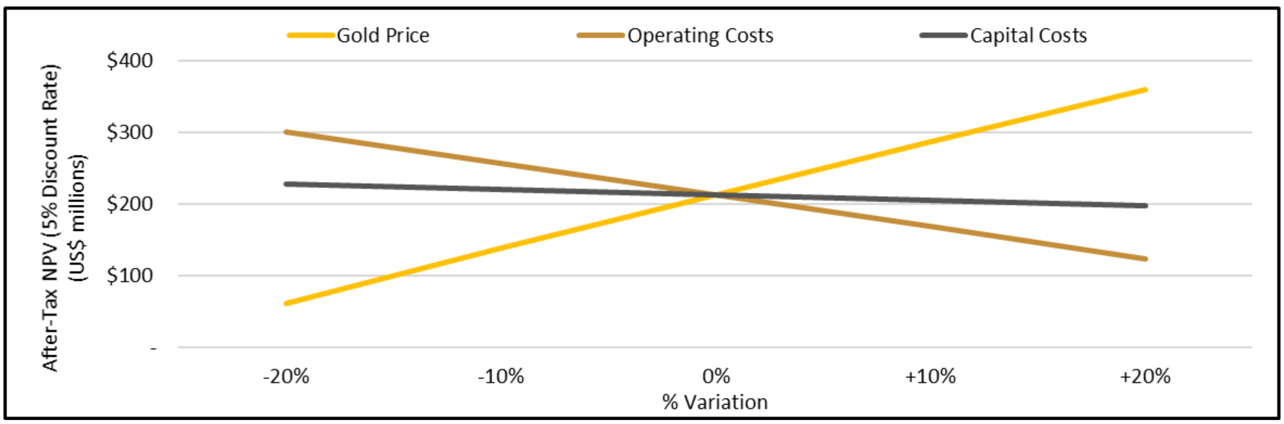


Figure 22-3: After Tax NPV Sensitivity at 5% Discount Rate

22.5 Discussion

When ranked, the sensitivity analysis indicates that the Project is most sensitive to gold price.

From a cost perspective, the Project is more sensitive to operating expenditure than capital costs.

23. ADJACENT PROPERTIES

Several exploration licenses are active or in the application phase immediately adjacent to the Project (Figure 23-1). These exploration licences are all held by individuals and there is no public disclosure on the activities related to the licences.

Endeavour Mining holds over 1,400 km^2 of mine licence and exploration permit as part of a joint venture with Sodim Limit in Côte d'Ivoire. The Property is the southern extension of the Enchi shear system. Afema includes a historical near-surface oxide and sulphide resource and lies within an area hosting several gold mineralized structures on extensions from prolific gold belts in Ghana. The Woulo Woulo prospect is the most recent discovery at Afema. Currently, multi-drill exploration campaign is underway at the Woulo Woulo prospect, the Niamienlessa structural trend and the Afema Shear. (www.endeavourmining.com).

Newcore's Enchi Gold Project is located 50 km south of Kinross' Chirano Gold Mine property. The Chirano Mine area lies within the Proterozoic terrain of southwest Ghana, along a major structure separating the Sefwi Belt to the west from the Kumasi Basin to the east known as the Bibiani Shear Zone. The Project covers a 40-km segment of the Bibiani Shear Zone where known gold mineralization is associated with major structures and subsidiary splays. The Chirano Gold Mine was commissioned in September 2005, and in 2020 it produced 165,000 oz of gold (www.kinross.com). Gold mineralization at Chirano is hosted within fractured and intensely hydrothermally altered granite where gold is associated with 1% to 5% pyrite, and the distribution of gold appears to be closely associated with the presence of pyrite (Red Back AIF March 30, 2010).

The Chirano Mine has 14 known gold deposits over a 9-km strike length range in individual lengths from 150 to 700 m, and range in thickness from a few metres to over 70 m (Stuart, 2009). Individual deposits may extend to over 700 m in depth. Mining at Chirano is done by both open pit and underground extraction. As of December 31, 2020, Proven and Probable reserves were 15.7 Mt grading 1.9 g/t gold for 0.974 Moz. The Measured and Indicated resources totalled 11.8.0 Mt grading 1.7 g/t gold for 0.628 Moz, and the Inferred resource totalled 5.7 Mt grading 2.1 g/t gold for 0.376 Moz (www.kinross.com).

The QP has not verified the technical data on the Chirano Mine and the gold mineralization at Chirano is not necessarily indicative of the mineralization on the Enchi Gold Project.

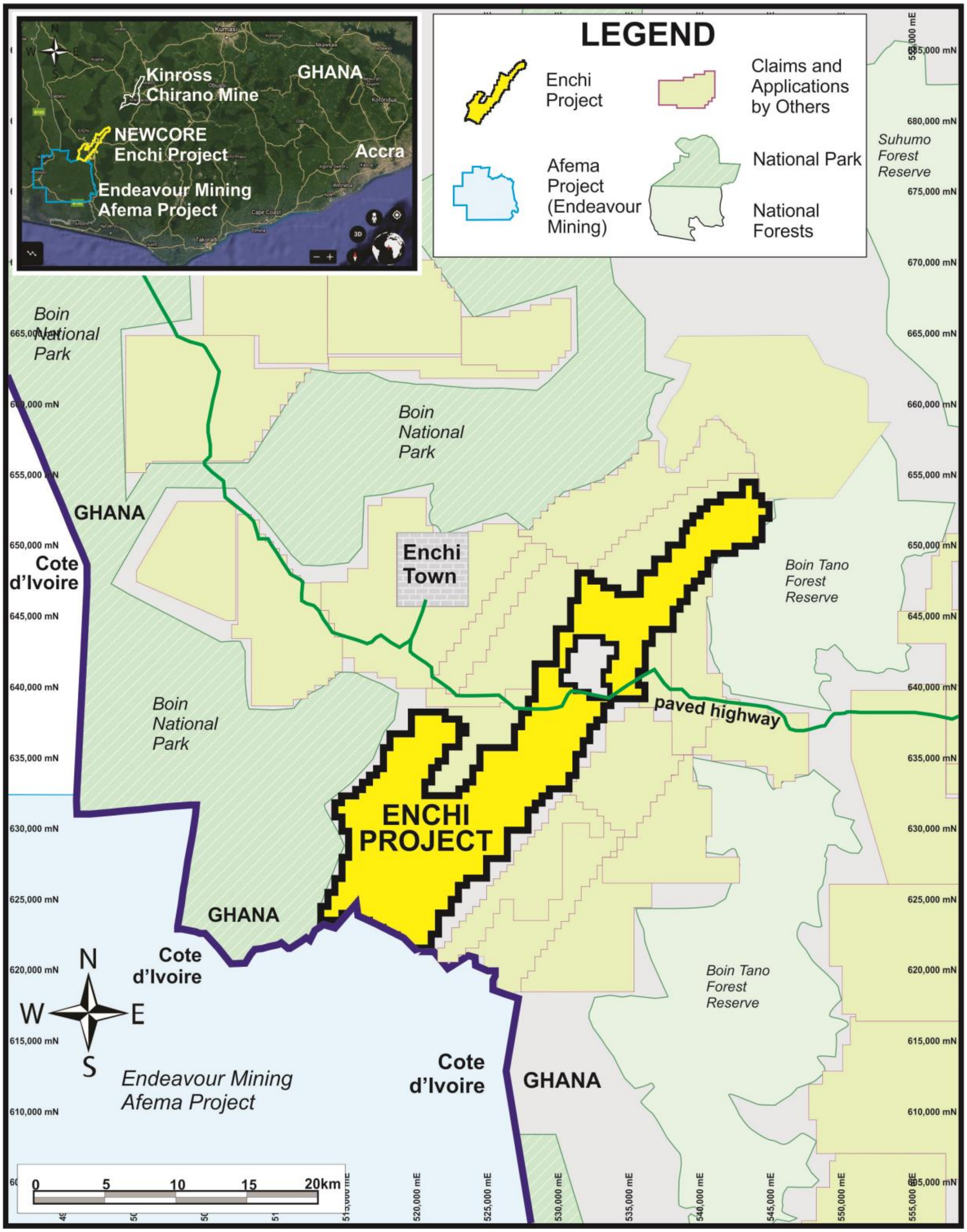


Figure 23-1: Adjacent Properties (Newcore Gold Ltd.)

24. OTHER RELEVANT DATA AND INFORMATION

As disclosed in Chapter 10, Newcore has continued to drill on the Project during the mineral resource estimation and PEA study. A total of 103 RC holes totalling 15,758 m have been completed on the Project since the data cut-off. The holes near the resources have been reviewed in terms of potential impact on the mineral resource. No negative impact on the mineral resource was visually identified.

25. INTERPRETATIONS AND CONCLUSIONS

This PEA Study is based on the mining and mineral processing methods developed for the Enchi mineral deposit. The NI 43-101 guidelines require that interpretations and conclusions related to the study, including an outline of key Project risks identified, be discussed.

The economic analysis contained in this report is based entirely on Inferred Resources and is preliminary in nature. Inferred Resources are considered too speculative geologically to have mining and economic considerations applied to them to be categorized as Mineral Reserves. There is no certainty that the economic results of this PEA will be realized.

25.1 Title and Geology

Based on the review of the available information, the QP concludes the following, in no particular order of perceived importance.

- The Property is currently held 100% by Newcore. The Government of Ghana is entitled to a 10% free carried interest in the Project.
- The approval process to grant licenses in the country can be slow. Although work can start on a license once an application is submitted, this does not guarantee the license will be granted.
- The Property is analogous to shear-hosted gold mineralization in quartz veining or quartz flooding. This style of mineralization is present in other mineral resources in the region.
- The Property is associated with mineralization related to the Bibiani Shear Zone that is known to host significantly large lode-gold deposits.
- The mineralization on the Property is associated to secondary and tertiary order shears that splays off the Bibiani Shear Zone.
- Newcore has a strong understanding of the regional and local geology to support the interpretation of the mineralized zones on the Property.
- Mineralization is currently defined in 15 individual zones at various stages of exploration. Four of the zones have drill defined mineral resources.
- Drilling and sampling procedures, sample preparation, and assay protocols are conducted in agreement with industry best practices.
- Verification of the drillhole collars, surveys, assays and drillhole logs indicates the Redback and Edgewater data is reliable to support the resource estimation.
- Discrepancy in the collar elevations relative to the topography continues to be an issue. Until this issue is resolved, the resources are likely to remain in the Inferred category.
- Based on the QA/QC program, the data is sufficiently reliable to support the resource estimate generated for the four zones on the Property.
- The mineral model has been constructed in conformance to industry standard practices.
- The geological understanding is sufficient to support the resource estimation.

- The presence of an oxide domain, a transition domain, and a fresh domain has been identified in the drill logs.
- At a gold cut-off grade of 0.2 g/t, the Inferred Resource totals 70.4 Mt with an average grade of 0.62 g/t Au, based on the parameters for a large tonnage open pit heap leach operation.
- The current mineral resource is pit constrained using parameters suitable for large open pit operations in Africa.
- The specific gravity (SG) value used to determine that tonnage was derived from data used by operating mines in the region, which may reflect a lack of precision with respect to the resource tonnages.
- The mineral resources at Sewum, Boin, Nyam and Kwakyekrom remain open along strike and in the down-dip directions.
- The remaining 11 mineral zones on the Property do not have enough data to support resource estimations. Additional exploration on these zones will not guarantee that the zones will support potentially economic material.

25.2 Mining

The Project's mine production schedule is based only on Mineral Resources in the Inferred category. There are no Mineral Reserves identified for the Project at this time.

The pit optimization, mine design, and mine plan were based on a gold price of $1,650/oz. A total of 10 discrete open pits were designed for the Sewum, Boin, Nyam and Kwakyekrom deposits. The open pits contain 68.6 Mt of mineralized material at an average gold grade of 0.57 g/t with a stripping ratio of 2.1:1. Table 25-1 presents the subset of mineral resources within the pit designs.

Table 25-2 presents the mining and processing schedule over an approximate 11-year mine life. Figure 25-1 summarizes the composition of the total heap leach feed tonnes by mining area.

Table 25-1: Subset of Mineral Resources Within the Pit Designs (1)(2)

Item	Tonnage (k tonnes)	Gold Grade (g/t)	Contained Ounces (k ounces)	Strip Ratio
Sewum	39,750	0.50	645	1.3:1
Boin	22,646	0.65	472	3.3:1
Nyam	4,520	0.73	107	3.2:1
Kwakyekrom	1,650	0.52	28	2.0:1
Total	**68,566**	**0.57**	**1,252**	**2.1:1**

(1) Including mining dilution and mining recovery.

(2) Numbers may not add due to rounding.

Table 25-2: Mining and Processing Production Schedule [(1)(2)]

Description	Unit	Total / Average	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Mining													
Mineralized Material Mined	k tonnes	**68,566**	4,866	6,576	6,576	6,576	6,576	6,630	6,708	6,770	6,576	6,576	4,137
Waste Mined	k tonnes	**143,490**	9,059	13,225	13,300	12,949	14,741	14,915	15,284	14,988	13,328	11,733	9,966
Total Material Mined	k tonnes	**212,055**	13,925	19,801	19,876	19,525	21,317	21,545	21,992	21,757	19,904	18,309	14,103
Strip Ratio	-	**2.09**	1.72	2.11	2.02	1.95	2.03	2.25	2.29	2.21	2.03	1.78	2.99
Processing													
Heap Leach Feed	k tonnes	**68,566**	4,866	6,576	6,576	6,576	6,576	6,630	6,708	6,770	6,576	6,576	4,137
Heap Leach Feed Grade	g/t Au	**0.57**	0.41	0.63	0.63	0.63	0.58	0.48	0.55	0.52	0.45	0.73	0.63
Gold Recovery	%	**79%**	80%	78%	78%	79%	85%	78%	74%	74%	83%	79%	77%
Gold Production	k ounces	**983.3**	50.7	104.6	104.0	104.3	103.7	80.0	87.6	83.4	79.6	121.4	63.9

(1) Including mining dilution and mining recovery.

(2) Numbers may not add due to rounding.

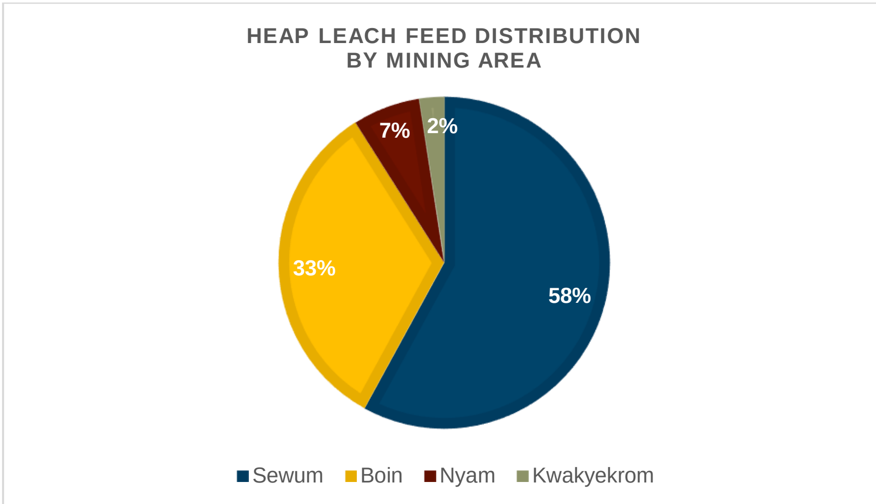


Figure 25-1: Heap Leach Feed by Mining Area

Figure 25-1 shows that the Sewum and Boin deposits contribute to the majority of the heap leach tonnages.

The PEA considers using a conventional truck and shovel operation and mining will be done using contract services under the supervision of Newcore. The mine production schedule is based on two 12-hour shifts, seven days a week for a total of 360 days per year.

Waste rock will be stored in waste rock facilities, which have been conceptually designed for the PEA at locations that minimized haul cycle times. Where possible, waste material will be placed in mined out pits to reduce closure costs and limit the Project’s overall footprint.

An operating cost model was developed to estimate the mining costs. The mine operating costs have been determined based on local contractor budgetary quotations and experience from similar sized open pit operations and local conditions. The mining costs used in the financial analysis consider variations in haulage profiles throughout the LOM.

25.3 Mineral Processing and Recovery Methods

BBA concludes the following with respect to mineral processing and recovery methods.

Preliminary metallurgical test work was carried out by Newcore at Intertek in 2020 and 2021 on rock chip samples from the recent RC drill campaign. Results from the preliminary bottle roll and small column leach tests indicate that cyanide leaching may be a viable option for the extraction of gold from the oxide domains. While indicative of cyanide solubility, the bottle roll and small column tests are preliminary and are not sufficient to make any meaningful conclusions on the metallurgy of the four zones involved in the Project.

For the purposes of this study, heap leaching has been selected as the preferred process option, although further work on particle size for optimal leach performance and physical constraints associated with heap leaching is still required to definitively select this as the most suitable option.

Good gold extraction results were achieved from the recent bottle roll tests performed on Sewum, Boin and Kwakyekrom samples, averaging 91%, 86% and 80% respectively. No samples from the Nyam Zone were tested in the recent bottle roll tests.

A conservative overall gold recovery of 79% was estimated and used to develop the process design criteria for the oxide zones. Although consistent with typical heap leaching operations of similar oxide mineralization, this recovery estimate will have to be confirmed with more comprehensive metallurgical test work.

A conventional heap leaching facility was sized to process 6.6 Mt/a of ROM material, including: primary and secondary crushing; agglomeration; permanent heap stacking and leaching; solution collection ponds, and a gold recovery plant. The process facilities will operate year-round on a 360 d/a basis.

Assumptions were made with regards to heap leach pad and back-end recoveries for the overall gold recovery of 79%. A secondary crusher product particle size P_{80} of 12.5 mm, and a heap leach time of 60 days were assumed, as well as pad dimensions and reagents consumptions in line with typical gold heap leach operations.

Gold extraction and recovery from cyanide leaching is generally sensitive to particle size. Typically, gold recovery will increase with finer particle size but with associated higher operating costs, and potential permeability issues in the case of heap leaching. The particle size distribution was not reported for most of the bottle roll tests; however, the size distribution of the composite samples used for the preliminary column leach test was measured and found to be much finer than a typical crushed mineralized material heap leach feed size.

All indications are that the product sample material obtained by RC drilling of the four deposits on the Project and used in the recent preliminary metallurgical tests is too fine for heap leach design purposes. The average P_{80} size distribution of the composite sample materials used for the four preliminary column leach tests was about 2 mm, compared to the more conventional heap leach feed size P_{80} of 12.5 mm that was assumed in the Project design criteria. As a result, all bottle roll and column leach test work extraction values have been discounted by about 10% to estimate the heap leach gold extraction for the Project.

Heap leach process design, gold extraction and overall recovery is determined and affected by:

- Mineralized material grade, mineralogy and variability;
- Crush product size;
- Reagent concentration and addition rates;

- Agglomeration;
- Irrigation system distribution and spacing;
- Solution recovery design;
- Stacking method and operation;
- Heap height;
- Heap edge effects; and
- Operations.

Based on the preliminary leaching test work that has been done to date, many of the key design variables have not yet been determined. It is recommended that diamond drilling and/or bulk sampling be used to obtain samples for subsequent metallurgical and HLF design purposes.

25.4 Infrastructure

Supporting infrastructure for the Project has been designed to a sufficient level of detail to arrive at cost estimates appropriate for a PEA level study. All designs, calculations, and assumptions are based on the available data and internal experience from working on similar projects.

Important items that still need to be finalized are the negotiations for a connection to the national HV power grid and the condition and availability of facilities for the mining staff in Enchi village.

25.5 Environmental

The most significant potential environmental and social issues are likely to be related to water management, social-economic impacts, and post mine closure expectations. These considerations are likely to be of key concern to local communities and/or likely to have cost implications in respect of impact management during the operational and closure phases.

The Project will trigger a range of regulatory requirements and processes, which will require the application for, receipt of, and compliance with a variety of environmental permits and approvals from the relevant Ghanaian authorities.

Closure objectives should be defined early in the mine planning process and integrated into all activities throughout the LOM.

25.6 Economic Analysis

The PEA is based on the proposed successive open-pit mining of the Sewum, Boin, Nyam and Kwakyekrom deposits. The target production rate during the LOM is 6.6 million tonnes per year using a contractor mining scenario.

The proposed Project process plant design will be based on gold heap leach technology, which will consist of crushing, agglomeration, stacking, and heap leaching with cyanide solution, adsorption of the pregnant solution, elution and gold smelting.

The results of the PEA comprise the following.

- Based on a gold price of $1,650 per troy ounce, the Project has a pre-tax internal rate of return (IRR) of 54% and pre-tax net present value (NPV) of $332.7 million, at a 5% discount rate;
- Based on a gold price of $1,650 per troy ounce, the Project has a post-tax IRR of 42% and a post-tax NPV of $212.5 million, at a 5% discount rate;
- Pre-tax payback period is 2.1 years and post-tax payback period is 2.3 years; and
- Gold production will average 92,765 troy ounces per annum and 983,300 troy ounces over the 10.6-year mine life.

25.7 Risk and Opportunities

Opportunities to improve the Project economics include the following:

- The mineral resource for all four zones remains open along strike and in the down-dip direction;
- Several auriferous zones in the Project need to be drill tested to a level sufficient to support a mineral resource estimate;
- The average overall gold recovery of 79% has been estimated for heap leaching based on the preliminary metallurgical test work that has been done to date. Increased gold recovery may be possible with a more conventional gold milling process; and
- A connection to reliable electrical power source for processing equipment will have a significant impact on operating costs as well as the ability to consistently achieve production targets. Early discussions with the utility companies (Ghana Grid Company and/or Electricity Company of Ghana) will be necessary to avoid reliance on diesel generated electricity.

Risks requiring mitigation strategies include the following:

- The current uncertainty in collar elevation is one of the reasons that the resources are classified as Inferred;
- Density test associate with oxide, transition and fresh rock;
- The mine plan assumed that the oxide material and 50% of the transition material will be mined without drilling and blasting. No test work has been done to validate this assumption;
- Assumptions were made for the pit slope angles. No geotechnical or hydrogeological studies has been conducted to date;
- The mining contractor will need to monitor tire temperatures for the long hauls for Nyam and Kwakyekrom if 64-tonne haul trucks are used;

- The average overall gold recovery of 79% has been estimated for heap leaching based on the preliminary metallurgical test work that has been done to date. Additional heap leach metallurgical test work is required to confirm that this average recovery can be achieved;
- The capital cost of the process facilities will be sensitive to the geotechnical conditions in the specific areas of facility construction. Geotechnical investigations will be necessary to confirm and optimize the location of process facilities;
- Newcore's future financial success depends on the ability to raise additional capital from the issuance of shares or the discovery of property that could be economically justifiable to develop. Such development could take years to complete and resulting income, if any, is difficult to determine. The sales value of any mineralization potentially discovered by Newcore is largely dependent upon factors beyond the Company's control, such as the market value of the products produced;
- The resource exploration industry is an inherently risky business with significant capital expenditures and volatile metals markets. The marketability of any minerals discovered may be affected by numerous factors that are beyond Newcore's control and which cannot be predicted, such as market fluctuations, mineral markets and processing equipment, and changes to government regulations, including those relating to royalties, allowable production, importing and exporting of minerals, and environmental protection;
- This industry is intensely competitive and there is no guarantee that, even if commercial quantities are discovered, a profitable market will exist for their sale. Newcore competes with other junior exploration companies for the acquisition of mineral claims as well as for the engagement of qualified contractors. Metal prices have fluctuated widely in recent years, and they are determined in international markets over which Newcore has no influence;
- Exploration and development on Newcore's Property are affected by government regulations relating to such matters as environmental protection, health, safety and labour, mining law reform, restrictions on production, price control, tax increases, maintenance of claims, and tenure. There is no assurance that future changes in such regulations would not result in additional expenses and capital expenditures, decreasing availability of capital, increased competition, title risks, and delays in operations;
- Availability of electric power on time for the construction of the Project. Contracts will have to be in place between Ghanaian electric companies (e.g.: Ghana Grid Company) and Newcore;
- Management of construction/engineering and procurement schedules, costs, and cost containment;
- Operating risks related to recruitment and training of mine workforce;
- Permitting risks;
- Pit slope design through geo-mechanics characterization and stability analysis; and
- Possibilities that the communities are not supportive of the Project.

These risks are common for this stage of gold projects and are similar risk factors to other gold projects of this stage and nature.

26. RECOMMENDATIONS

It is the QP's opinion that additional exploration and engineering test work expenditures are warranted to improve the understanding of the Project and delineate additional resources. The following recommendations and budgets have been determined based on advancing the Project.

26.1 Geology and Mineral Resources

The QP recommends a two-phase exploration program for Enchi. Phase 1 would be to further delineate the existing mineral resources through drilling and metallurgical test work. An estimated budget of $2.5 million is summarized in Table 26-1.

Table 26-1: Phase 1 Exploration Budget

Program	Cost ($)
Reverse Circulation Drilling 5,000 m @ $100/m	500,000
Diamond Drilling 3,000 m @ $250/m	750,000
Sample Assay @ $25/sample	200,000
Metallurgical Test Program	240,000
Labour & Accommodations	280,000
Access & Compensation	250,000
Topographic Survey/LiDAR	130,000
Community and Stakeholder Engagement	150,000
Total	**2,500,000**

The Phase 2 budget is designed to identify additional mineral resources on the Project and to collect additional data to support future engineering studies. Phase 2 is independent of Phase 1 and can be completed at anytime regardless of the results of Phase 1. The Phase 2 budget is estimated at $4.8 million and is summarized in Table 26-2.

Table 26-2: Phase 1 Exploration Budget

Program	Cost ($)
Reverse Circulation Drilling 10,000 m @ $100/m	1,000,000
Diamond Drilling 8,000 m @ $250/m	2,000,000
Sample Assay @ $25/sample	450,000
Metallurgical Test Program	240,000
Environmental Baseline Study	150,000
Labour & Accommodations	280,000
Access & Compensation	250,000
Geotechnical Assessment (open pit & site)	175,000
Community and Stakeholder Engagement	255,000
Total	**4,800,000**

26.2 Open Pit Mining and Project Infrastructure

- Initiate geotechnical studies for pit slope stability assessment;
- Initiate geotechnical studies for major surface infrastructure including heap leach pad and facilities, waste material storage areas, and roads;
- Initiate hydrogeological program to quantify the pit dewatering requirements;
- Initiate geochemical studies to determine potential Acid Rock Drainage (ARD) and Metal Leaching (ML) potential of the various rock types;
- Conduct condemnation drilling to confirm the locations for the waste rock piles and other infrastructure;
- Investigate potential sources for borrow pits including geotechnical laboratory testing for construction materials;
- Review and refine the site layout and central processing facility location in order to optimize haulage plans with facility layouts;
- Prior to the commencement of any further design of the road network, additional testing will be required to evaluate the in situ material conditions, foundation conditions for minor structures, and availability of road building materials. These tests will include a centreline investigation of the road alignments and the sourcing of road building material;
- Begin negotiations with mining contractors to obtain more accurate pricing and conduct a detailed trade-off study between an owner operated fleet and contract mining;
- Carry out a trade-off study for material transportation to heap leach facility;
- Conduct a trade-off study between single and multiple leach pads; and
- Re-evaluate the heap leach facility location.

26.3 Mineral Processing and Recovery Methods

BBA recommends the following with respect to mineral processing and recovery methods:

- An understanding of the morphology of the gold particles in their host mineral is essential in the development of the most effective process route to extract the metal values. In addition to multi-element analysis, microscopic gold scanning should be carried out to verify the dimensions of gold grains and their occurrence (e.g., liberated, attached or locked). If the gold is encapsulated, attached or locked, the association with its host mineral must be described. Depending on the particle size of the gold in the oxide domain and its occurrence, it may be predicted whether the material would have to be milled to a finer particle or grind size to expose a sufficient amount of gold particle surfaces to the cyanide solution in order to achieve acceptable recoveries. A geometallurgy program should be initiated to plan, collect and prepare appropriate samples from the various oxide, transition, and sulphide zones for analysis by a qualified laboratory to determine the required mineralogical, chemical and physical characteristics;

- Rock chip cuttings from the recent 2020/2021 reverse circulation (RC) drill program have been the source of all sample materials used in recent metallurgical testing by Newcore and Intertek. Although useful for preliminary bottle roll leach tests to indicate gold extraction by cyanidation, these RC samples are too fine for definitive heap leach metallurgical process design purposes. A diamond drilling program will be required to obtain representative core samples from the various zones for a comprehensive metallurgical and heap leach design test program. The diamond drilling (DD) program should be cooperatively designed to satisfy both the geological and metallurgical test requirements in terms of core size and quantity;
- Important aspects for a successful heap leach design and operation such as particle size, heap height, solution application rate, agglomeration parameters and similar variables are required to confidently determine heap stability, permeability, porosity, and recovery. The physical characteristics of the mineralized material should be analyzed during column leach tests by the laboratory. It will also be necessary to look at the gold morphology on the tailings material after heap leach to understand the gold losses and improve overall treatment;
- Once the optimum particle crush size for heap leach gold recovery has been determined, column leach tests must be conducted on composite samples representing each zone to simulate heap leach conditions. The tests will determine the gold extraction at different leach times, producing an extraction curve. The consumption requirements for sodium cyanide, lime, and cement for agglomeration should also be optimized;
- Coarse and fine bottle roll leach tests should also be conducted on the same composite samples to simulate conventional gold extraction operations. The tests will determine the gold extraction at different particle sizes and leach times until extraction stabilizes. Reagent consumption optimization should also be analyzed. These tests will confirm whether there is a progressive increase in gold extraction as the material is milled to finer particles sizes and will also suggest the degree of liberation of gold particles. The effect of grind is therefore an important parameter in optimizing the process flow sheet. If gold particles are coarse, a front-end gravity recovery option should also be considered;
- Comminution test work including Bond Crushability Work Index (CWi) and Abrasion Index (Ai) should be conducted on the composite samples to determine design conditions for particle reduction of the potential ore;
- Agglomeration and percolation test work should be conducted to determine optimum binder reagent requirements and irrigation flow rate;
- Other characteristics that will affect the process flow sheet and equipment design should be evaluated by the laboratory; these include:
  - Dissolved gold from the leach solution thereby reducing gold recovery. Porosity of the material that will affect the efficiency of the heap leach operation;

  - Clay content in the material that can increase the probability of 'rat-holing' or short circuiting of fluid through the bed of material and blocking the solution flow, thus reducing the overall gold recovery or will increase leach times and solution losses. Typically high in saprolites, the clay fines may need to be removed through screening prior to stacking on the heap. Equipment selection and design will have to be evaluated as the fine clay will blind vibrating screens during wet weather conditions; and
  - Carbonaceous material content will have a ''preg-robbing'' effect as it tends to adsorb.
- A review of the Project area for suitable materials to construct the heap leach pads is recommended. This includes gravel, sand, and clay materials that can be utilized for the construction of the overliners. The availability and proximity of suitable borrow materials can have a significant impact on the construction schedule and capital cost;
- Geotechnical testing of the ground conditions in proposed locations for the heap leach pad, infrastructure and plant facilities will be required to confirm suitable site selections and capital cost estimation; and
- For the latter stages of the mine life, the zones transition to an un-oxidized rock type that typically requires milling to less than 75 microns for optimum recovery of gold with conventional carbon in leach. To study the metallurgical behaviour of the oxide, transition and fresh zones, a complete test program should be anticipated including: comminution, head analysis, gravity amenability, dissolution appraisal of gravity tails, flotation of gravity tails, and dissolution appraisal of flotation concentrate.

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