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Farim Phosphate Project

National Instrument 43-101 Technical Report

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GLOSSARY OF ACRONYMS AND ABBREVIATIONS

2D	two dimensional
3D	three dimensional
AAS	atomic absorption spectrometry
AFNOR	Association Français de Normalisation
aGL	original ground level
A.I.	Acid Insoluble
ALARP	as low as reasonably practicable
ALS	ALS Metallurgy and ALS Global
AOC	AOC Archaeology Group
APGB	Administration of Ports of Guinea-Bissau
AQ	air quality
ARD/ML	acid rock drainage/metal leaching
ARI	average return interval
ARO	after receipt of order
ASD	Azimuth Stern Drive
asl	above sea level
AUD	Australian Dollar
Baird	W.F. Baird & Associates Ltd.
bcm	bank cubic metres
bcm/t	bank cubic metres per tonne
bn	billion
BOD	biological oxygen demand
BOQ	bill of quantities
BRGM	Bureau de Recherche Geologiques et Minieres
°C	Degrees Celsisus
CAD	Canadian Dollar
CAIA	Cellule d’Evaluation des Impacts Environnementaux
CAPEX	capital expenditure
CBR	California bearing ratio
CEA	Commissariat à l'Energie Atomique
Champion	Champion Resources Inc.
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	clean in place
CM	Construction Management
CMP	construction management plan

COFRAC	Comite Francais D'Accreditation
CRPG	Centre de Recherches Pétrographiques et Géochimiques
CRU	CRU Group
CU	consolidated undrained triaxial tests with pore pressure measurements
d/y	days per year
DAP	diammonium phosphate
DES	discrete event simulation
DGM	Directorate of Geology and Mines
DGMGB	Directorate of Geology and Mines of Guinea-Bissau
DSO	direct shipping option
DTM	digital terrain model
DWT	dead weight tonne(s)
ECD	environmental control dam
EDA	exploratory data analysis
EIA	Environmental Impact Assessment
EGL	effective grinding length
EHS	Environmental, Health and Safety
EMP	environmental management plan
EMPA	electron microprobe analysis
EP	engineering and procurement
EPC	engineering, procurement and construction
EPCM	engineering, procurement and construction management
EPFI	Equator Principles Financial Institution
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Plan
ESMS	Environmental and Social Management System
ETP	effluent treatment plant
EUR	Euro
F_{80}	80% passing particle size in feed stream
FEL	front-end loader
FIP	fire indication panel
FOB	freight on board
FOS	factor of safety
FPA	decarbonised phosphate unit (geological unit)
FPB	calcareous phosphate member (geological unit)
FPC	fleet production and cost analysis
FPO	phosphatic interval (geological unit)

FS	feasibility study
g	gram(s)
GA	general arrangement
G&A	General and Administration
GB Minerals	GB Minerals Ltd.
GBMAG	GB Minerals AG
GB Phosphate	GB Phosphate Mining Ltd.
GBP	Pound Sterling
GEEEM	Géologie Exploitation Environnement Expertise Mine
GoGB	Government of Guinea-Bissau
Golder	Golder Associates Inc.
GPO	general purpose outlet(s)
GPS	global positioning system
GSW	gland seal water
Ha	hectare(s)
HAZOP	Hazard and Operability Study
HDPE	high-density polyethylene
HP	Horsepower
HSSE	health, safety, security and environment
HV	high voltage
HVAC	heating, ventilating and air conditioning
IBC	International Building Code
ICMM	International Council of Metals and Mining
ICP	inductively coupled plasma
IDB	influent design basis
IDW2	inverse distance weighted method
IFC	International Finance Corporation
IGN	Institut Géographique National
IHP	Instituto Hidrografico de Portugal
IOB	In-pit overburden backfill
IRR	internal rate of return
ISO	International Organisation for Standardisation
ISPS	International Ship and Port Facility Security Code
IUCN	International Union for Conservation of Nature
IWL	integrated waste landform
l	litre(s)
l/s	litres per second

lcm	Loose cubic meter(s)
LOM	life-of-mine
Kemworks	KEMWorks Technology Inc.
KFI	Kristal Font Incorporated
Knight-Piésold	Knight-Piésold Pty. Ltd and Knight-Piésold Ltd
kg	kilogram(s)
km	kilometre(s)
km^2	square kilometre(s)
km/h	kilometre(s) per hour
kN/m^3	kilonewton(s) per cubic meter
KP	Knight-Piésold
kPa	kilopascal(s)
KPI	key performance indicator
kV	kilovolt(s)
kW	kilowatt(s)
lcm	loose cubic meter(s)
LG	Lerchs & Grossman
LGP	low ground pressure
LiDAR	light detection and ranging
LOM	life of mine
LOMP	life of mine plan
LSA	local study area
LV	low voltage
LVA	landscape and visual amenity
Lycopodium	Lycopodium Minerals Canada Ltd.
M	million(s)
m	meter(s)
mhr	man-hour(s)
m/s	meters per second
mm	millimetre(s)
µm	micrometer(s) (microns)
m^3	cubic meter(s)
m^3/h	cubic meters per hour
mamsl	meters above mean sea level
MAP	monoammonium phosphate
MASW	multichannel analysis of surface waves
Mbcm	million bank cubic meter(s)

MCC	motor control centre
MER	minor element ratio
MER*	adjusted minor element ratio accounting for iron present as pyrite
MFIP	master fire detection panel
Mm^3	million cubic metre(s)
MRCP	mine reclamation and closure plan
Mt	million tonne(s)
Mtpa	million tonnes per annum
MTO	material take-off
MW	megawatt(s)
n/a	not applicable
NDE	non-destructive examination
NGL	natural ground level
NI 43-101	National Instrument for Standards of Disclosure for Mineral Projects in Canada
NIMA	National Imagery and Spatial Agency
nm	nautical mile(s)
NPC	net present cost
NPV	net present value
OBO	Ore Block Optimizer
OCP	Office Cherifien des Phosphates
OK	Ordinary Kriging
O&M	operation and maintenance
OPA	open pit area
OPEX	operating expenditure
OSD	overburden storage dump
OSF	overburden stockpile facility
P_{80}	80% passing particle size in product stream
P&ID	piping and instrumentation diagram
PCM	Plains Creek Mining
PCP	project controls plan
PDF	Portable Document Format
PEP	project execution plan
PF	productivity factor
PMP	project management plan
POB	point of observation
PPE	personnel protection equipment
Project	Farim Phosphate Project

PS	performance standard
PVC	polyvinyl chloride
QA	quality assurance
QC	quality control
QAQC	quality assurance quality control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
QP	Qualified Person
RAP	resettlement action plan
RAPF	resettlement action plan framework
RHC	Resource Hunter Capital Corporation
ROM	run-of-mine
rpm	revolutions per minute
RSA	Republic of South Africa
S/cm	siemens per centimetre (electrical conductivity)
S.I.E	selective ion electrode
SC&A	Scherman, Colloty and Associates
SCD	sediment control dam
SEDAR	System for Electronic Document Analysis and Retrieval
SG	specific gravity
SGS	SGS Mineral Services
SOP	standard operating procedure
SOS	surcharge overburden storage
SPT	standard penetration testing
SPM	single point mooring
SR	stripping ratio
SS	soft starter
TDS	total dissolved solids
TSF	tailings storage facility
t	tonne(s) (metric)
t/h	tonnes per hour
t/m^3	tonnes per cubic meter
TSS	total suspended solids
TWCALC	tidal assisted vessel departure model (Baird's)
UNDP	United Nations Development Programme
UNEP	United Nations Environment Programme
UNISDR	United Nations International Strategy for Disaster Reduction
USD	United States Dollar(s)

USD/t	United States Dollars per tonne
UTM	Universal Transverse Mercator coordinate system
UU	undrained unconsolidated triaxial tests
V	volt(s)
VA	variance adjustment
VSD	variable speed drive
WAP	wet acid process
WB	World Bank
WBS	work breakdown structure
WD	waste dump
WHIMS	wet high intensity magnetic separation
WHO	World Health Organization
WMF	waste management facility
wt%	weight percent
w/w%	weight in weight (weight percent)
XLPE	cross-linked polyethylene
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence

1.0 SUMMARY

1.1 Introduction

The following Technical Report was compiled by Lycopodium Minerals Canada Ltd. (Lycopodium) and presents the results of the Feasibility Study for the Farim Phosphate Project, located in north-central Guinea-Bissau, West Africa, approximately 25 km from the northern border with Senegal and 80 km north of the capital, Bissau. The Technical Report was prepared at the request of GB Minerals Ltd. (GB Minerals), a British Columbia corporation. GB Minerals is a Canadian mining and development company that is focused on developing the Farim Phosphate Project. GB Minerals is listed on the TSX Venture exchange (GBL).

Lycopodium was commissioned by GB Minerals in September 2014 to prepare the NI 43-101 compliant technical report on the project. The purpose of this Technical Report is to provide GB Minerals with sufficient information to determine the economic feasibility of developing the Farim Phosphate Project.

This NI 43-101 Technical Report was completed by:

- Lycopodium Minerals Canada Ltd. ("Lycopodium") for the process plant infrastructure, port facilities on land infrastructure, and process plant and port facilities operating costs;
- KEMWorks Technology Inc. ("Kemworks") for metallurgical test work and process design;
- Golder Associates Inc. ("Golder") for mining and geology;
- W.F. Baird & Associates Ltd. ("Baird") for marine infrastructure, marine vessels, Capital and Operating costs for marine operations and shipping;
- Knight-Piésold Pty. Ltd., Perth, Australia, ("Knight-Piésold") for the design of the integrated waste landform, geotechnical, hydrogeology, hydrology, site water management, geochemistry and infrastructure design support. Knight-Piésold's Canadian office in North Bay, Ontario, was responsible for environmental studies, and social/community impact.
- Alex Duggan (Kristal Font Incorporated) for capital costs and economic analysis.

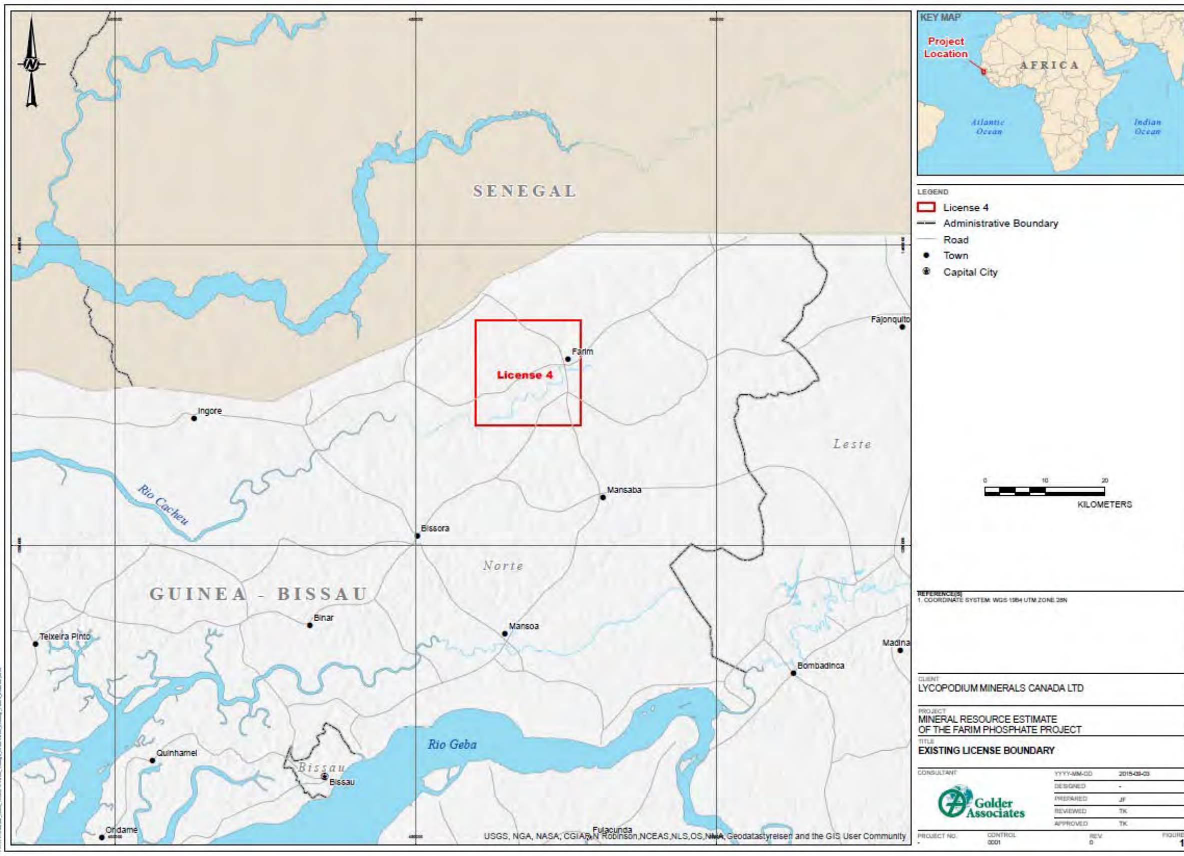
Unless otherwise denoted, all costs referred to in this Feasibility Study are quoted in current Q3 United States Dollars.

1.2 Property Description and Location

The Farim Project is located in the northern part of central Guinea-Bissau, West Africa, approximately 25 kilometres south of the Senegal border, approximately 5 kilometres west of the town of Farim and some 120 kilometres northeast of Bissau, the capital of Guinea-Bissau. The Farim Project lies within mining lease license No. 004/2009 ("**Mining Lease 004/2009**"), covering 30,625 hectares, granted by the Government of Guinea-Bissau to GB Minerals AG ("**GBMAG**"), a wholly owned subsidiary of GB Minerals registered in Switzerland, on May 28, 2009.

The area covered by Mining Lease 004/2009 is shown in the following map:

Figure 1-1 Location of the Farim Phosphate Project



Following a submission of an application for an authorization for mining and production to the Ministry of Energy and Natural Resources of Guinea-Bissau, GB Minerals AG was granted on May 28, 2009, a mining license, Mining License No. 001/2009 ("**Mining License 001/2009**"), for a period of 25 years, giving it the exclusive right to prospect, explore, extract, mine, treat, transport and sell any material mined within the license area of Mining Lease 004/2009. Mining License 001/2009 also provides all the permits required for both the construction and production phases of the Farim Project.

GB Minerals AG and the Government of Guinea-Bissau also agreed, on May 28, 2009, to enter into a mining agreement (the "Mining Agreement") that would govern the execution of Mining Lease 004/2009 and Mining License 001/2009 and clarify the framework applicable to certain ancillary rights granted to GB Minerals AG for the development of the Farim Project. The Mining Agreement is valid for 25 years and can be extended for an additional 25 years upon request. The Mining Agreement also allows GB Minerals to build roads, buildings, port or other infrastructures required in connection with the project without being subject to taxes, license fees or other costs both within and outside the concession area.

1.3 Geology and Mineralization

The Farim phosphate deposit is located within the Middle Eocene Lutetian Formation that forms part of the southern margin of the Mauritania-Senegal-Guinea Cenozoic sedimentary basin (Prian, 1987). The basin extends from Morocco in the north through Mauritania, Senegal, Guinea-Bissau and into Guinea to the south. The Mid-Eocene and particularly the Lutetian of the basin contains known phosphate horizons and hosts a number of important economic phosphate deposits including Bofal in Mauritania and Taïba, Thiès and Matam in Senegal. It accounts for almost 25% of current world rock phosphate production.

The Farim area forms part of the southern margin of the former Casamance Gulf and is located 60 km northwest of the southern edge of the Senegal-Mauritania-Guinea sedimentary basin in which the Maastrichtian strata unconformably overlies the Devonian pelite sequence (Prian, 1987).

The Farim phosphate deposit is a flat-lying sedimentary phosphatic bed, which underlies an area in excess of 60 km^2.

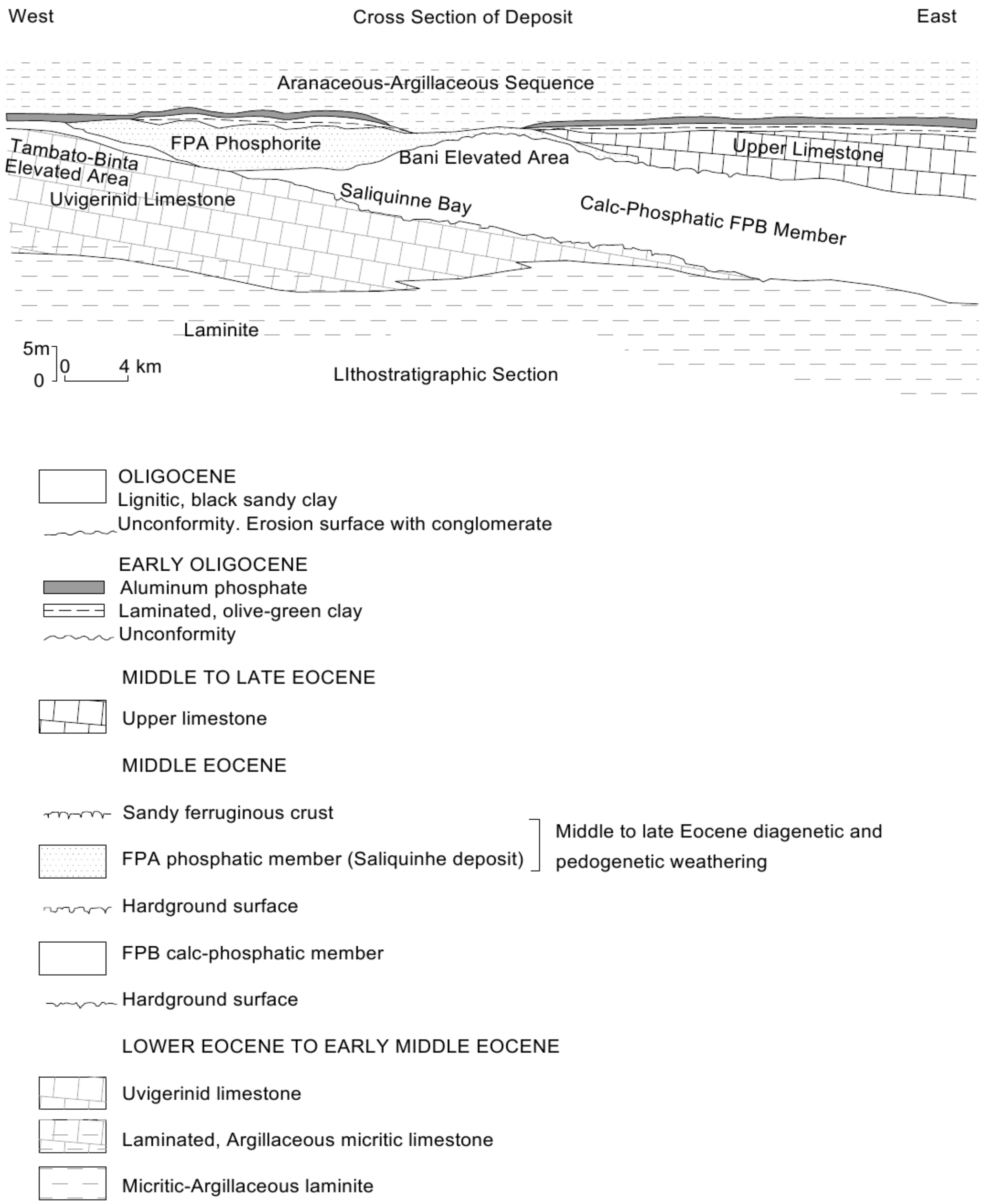
The geological sequence at Farim displays the following lithological units from top to bottom:

Sandy-argillaceous overburden with soft, alternating sandy, clayey and sandy-clayey layers;

- Phosphatic interval (FPO);
- Upper dolomitic limestone;
- Decarbonized phosphate unit (FPA) corresponding to the Saliquinhé phosphate deposit;
- Calcareous phosphate member (FPB); and
- Limestone at the footwall of the phosphate sequence, white, soft and porous.

Figure 1-2 shows a typical cross section of the Farim deposit together with a lithostratigraphic column (Prian, 1989).

Figure 1-2 A Typical Cross Section of the Farim Deposit with a Lithostratigraphic Column (Reproduced from Prian, 1989)



In the Farim phosphate property, two main types of phosphate have been identified, differentiated by their petrography and chemical composition:

- FPA layer, a de-carbonated phosphate matrix with very high P_2O_5 content of about 30% P_2O_5, formed exclusively in the shallow water of the Saliquinhé basin; and

- The lower grade FPB layer of highly carbonated phosphate, generally containing 5 to 15% P_2O_5 (average 13% P_2O_5) with some values up to 20%.

1.4 Mineral Resource Estimate

The Farim deposit has been delineated over an area of approximately 40 km^2 and is divided by the Cacheu River. The deposit consists of both FPA and FPB mineralised units. This Mineral Resource Estimate concerns FPA only, as the FPB unit was previously deemed to be uneconomic. No additional mineralisation outside the deposit modelled was considered in the Mineral Resource Estimate.

Golder modelled the Farim resource based on a 2D grid of 125 m by 125 m cells covering the extents of the FPA layer. The extents of the FPA layer were digitised based on the presence or absence of the FPA layer in the drill holes. P_2O_5 grade plus four deleterious elements; Al_2O_3, CaO, Fe_2O_3 and SiO_2, were estimated. The thickness of the overburden and FPA units were also estimated.

Golder considers the mineralization contained within the Farim deposit to fulfil the criteria of "potentially economic" to be reported as a resource. A phosphate cut-off grade and maximum strip ratio were not applied to report the Mineral Resource Estimate. Instead, a minimum FPA thickness of 1 m was used to define a mineral inventory which has reasonable expectation of eventual economic extraction.

Table 1-1 summarize the results of the 2 July 2015 Mineral Resource Estimate based on a minimum FPA thickness of 1.0 m and a constant density of 1.4 t/m^3. Estimated Resources within the extents of the 25 year pit design are provided in Table 1-2 which summarizes the Global Resource estimate. This assumes the resource would be exploitable using open pit mining methods.

The 25-Year Mineral Resource Estimate, dated 2 July 2015, defines a Measured Resource of 46.7 Mt at an average grade of 30.6% P_2O_5. The Global Mineral Resource Estimate, dated 2 July 2015, defines a Measured Resource of 105.6 Mt at an average grade of 28.4% P_2O_5 and an Inferred Resource of 37.6 Mt at an average grade of 27.7% P_2O_5. Tonnage and grade have been rounded to an appropriate decimal place after calculations. No recoveries or dilution factors have been considered in this estimate and the results should be considered strictly *in situ*, in accordance with NI 43-101 reporting guidelines for resources.

Table 1-1 25-Year Mineral Resource Statement, Farim Phosphate Deposit, 2 July 2015

Class	Block	Tonnage, Dry Basis (Mt)	FPA (m)	P_2O_5, Dry Basis (%)	Al_2O_3, Dry Basis (%)	CaO, Dry Basis (%)	Fe_2O_3, Dry Basis (%)	SiO_2, Dry Basis (%)	Overburden (Mbcm)	Stripping Ratio (bcm/t)
Measured	North Pit	32.2	3.77	30.31	2.66	41.17	5.15	10.36	318.0	9.87
	South Pit	14.4	3.77	31.23	2.34	40.51	3.77	11.21	102.9	7.13
	Subtotal	46.7	3.77	30.59	2.56	40.96	4.72	10.62	420.9	9.02
Indicated	North Pit	-	-	-	-	-	-	-	-	-
	South Pit	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-
Measured + Indicated	North Pit	32.2	3.77	30.31	2.66	41.17	5.15	10.36	318.0	9.87
	South Pit	14.4	3.77	31.23	2.34	40.51	3.77	11.21	102.9	7.13
	Subtotal	46.7	3.77	30.59	2.56	40.96	4.72	10.62	420.9	9.02
Inferred	North Pit	-	-	-	-	-	-	-	-	-
	South Pit	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-

Notes:

Assumes a minimum FPA seam thickness of 1 m
FPA within 50 m of Cacheu River has been assigned as "unclassified" due to the uncertainty attached to the extraction of material in this area.

Table 1-2 Global Mineral Resource Statement, Farim Phosphate Deposit, 2 July 2015

Class	Block	Tonnage, Dry Basis (Mt)	FPA (m)	P_2O_5, Dry Basis (%)	Al_2O_3, Dry Basis (%)	CaO, Dry Basis (%)	Fe_2O_3, Dry Basis (%)	SiO_2, Dry Basis (%)	Overburden (Mbcm)	Stripping Ratio (bcm/t)
Measured	North of River	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
Indicated	North of River	-	-	-	-	-	-	-	-	-
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-
Measured + Indicated	North of River	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
Inferred	North of River	11.4	1.71	24.88	2.84	39.63	4.42	10.52	210.9	18.44
	South of River	26.2	2.12	28.99	5.37	35.90	5.28	11.58	258.2	9.85
	Subtotal	37.6	1.98	27.74	4.60	37.03	5.02	11.26	469.0	12.46

Notes:

Assumes a minimum FPA seam thickness of 1 m.
FPA within 50 m of River Cacheu has been assigned as "unclassified" due to the uncertainty attached to the extraction of material in this area.

1.5 Mineral Reserve Estimate

This Mineral Reserve Estimate concerns FPA only, as the FPB unit was previously deemed to be uneconomic. No additional mineralisation outside the modelled deposit was considered in the Mineral Resource and Reserve Estimates.

The reserve estimation was undertaken in Ventyx®'s Minescape™ software (Version 5.8). The Mineral Reserve statement is effective 24 June 2015.

The assessment of surface mineable phosphate matrix reserves within the Project area was based on the 25-year mine plan and corresponding open pit design. The pit design was developed based on a pit optimization exercise that delineated the most economical 44 Mt of ROM material to feed a 25 year plan at a rate of 1.75 Mtpa on a dry basis.

As per the Mineral Resource Estimation methodology, a true phosphate cut-off grade was not applied to the Mineral Reserve Estimate. However, Golder applied a penalty to blocks with ROM grade values lower than 29% P_2O_5 and rewarded blocks with a ROM grade value greater than 29% P_2O_5 in the optimizations.

Estimated ROM phosphate matrix reserves and phosphate rock reserves for the proposed 25 year, 1.75 Mtpa pit are listed in Table 1-3 below. Golder considers the criteria used to define the 25 year mineral inventory to be reasonable for public reporting. However, adequate financing and permitting will be required prior to the commencement of the project.

Table 1-3 Proven and Probable Reserves

Category	Units	Phosphate Matrix Reserves		
		Proven	Probable	Total/Average
ROM FPA Tonnes (Dry Basis)	Mt	44.0	-	44.0
ROM %P_2O_5 (Dry Basis)	%	30.0	-	30.0
ROM %Al_2O_3 (Dry Basis)	%	2.6	-	2.6
ROM %CaO (Dry Basis)	%	41.0	-	41.0
ROM %Fe_2O_3 (Dry Basis)	%	4.7	-	4.7
ROM %SiO_2 (Dry Basis)	%	10.6	-	10.6

For the Farim Phosphate Deposit Beneficiation Option the total estimated Proven and Probable Reserves are 44.0 Mt (dry basis) with an average ROM P_2O_5 grade (dry basis) of 30.0%. The overall ROM strip ratio is estimated to be 10.26 bank cubic meters (bcm) per tonne of ROM phosphate matrix, requiring the removal of approximately 451.7 million bcm of overburden over the life of the mine.

1.6 Mining Methods

The FPA matrix is mined by a multiple bench open pit haul back mine using excavators and trucks. Golder selected the excavator/truck mining method based on lower initial capital, lower investment risk, increased grade control, limited power supply, and flexibility to adapt to a smaller scale Direct Shipping Option (DSO) operation if needed.

For the 1.75 million tonnes per annum (Mtpa) (dry basis) open pit, it is planned that overburden will be stripped and removed with 12 cubic metre (m^3) front end loaders (FEL) or other similar excavator matched with 97 tonne (t) capacity haul trucks. The matrix will be mined with 5 m^3 bucket class backhoes matched with 36 t capacity trucks to minimize mining dilution and maximize matrix recovery. The matrix will be hauled to a 175,000 t (dry basis) ROM stockpile adjacent to the plant, and segregated by quality. The matrix will be reclaimed and carefully blended into a ROM Bin by front-end wheel loaders with 12 m^3 buckets to achieve the desired product P_2O_5 grade. The plant feed hopper will be installed so that matrix haul trucks can directly feed matrix to the plant if possible.

Overburden excavation will advance ahead of the matrix extraction in maximum 10 m height production benches. Because the overburden thickness is greater than 30 m within the 25 year pit, multiple overburden stripping benches will be developed and maintained in advance of the matrix extraction.

The most critical design element of the proposed mining plan is water management. All mining areas must be fully dewatered in advance of mining activities. Dewatering of the overburden and phosphate matrix zone must be done approximately six months prior to mining activities to accommodate dry mining of the deposit. Dewatering pump test data indicates that dry open-pit mining will be feasible. Dry mining the deposit will allow 65° temporary dig face angles. The proximity of the mine site to the Cacheu River will require the construction of a protection bund to prevent in-pit flooding. Sufficient overburden material from pre-stripping operations (Year 0) will be diverted to construct a bund between the mine site and the tidal extents of the river. This bund will be constructed for flood control and will serve as the primary barrier between the river and mining areas. In addition to advance dewatering, in-pit water management is critical. Mine perimeter ditches and protection bunds with water storage ponds and pumps must be established and rigorously maintained to keep surface water from entering the mining areas. Roads must be well-graded and crowned with a thick layer of pervious crushed rock.

Because of the concentrated annual rainfall from July through September, the mine plan limits mining activities at full production to nine months out of the year; the other three months will be mined at reduced productivity. Operations must be vigilant with in-pit dewatering to prevent pit flooding and maintain pit stability.

The remote nature of the Farim operation, with limited power supply, precludes the use of electric mining equipment. All mining equipment selected for the plan is diesel mobile equipment.

Table 1-4 summarizes mine plan parameters and factors.

Table 1-4 Summary Table of Mine Plan parameters

Description	Value
Permanent wall angle	20°
Permanent wall operational FOS	>1.3
Bench Height	10 m
Short-Term Bench Face (Batter) Angle	65°
Short-Term Berm Width	14.9 m
Long-Term Bench Face (Batter) Angle (After Sloughing)	25°
Long-Term Berm Width (After Sloughing)	6.5 m
Overburden angle of repose WD/IOB/SOS	1V:4H / 1V:6H / 1V:6H
Overburden spoil swell factor	27%
Total Moisture (As-Received Basis), Overburden	20%
Overburden Density (As-Received Basis)	2.10 t/m^3
Overburden Density (Dry Basis)	1.68 t/m^3
Total Moisture (As-Received Basis), Matrix	20%
Matrix Density (As-Received Basis)	1.75 t/m^3
Matrix Density (Dry Basis)	1.40 t/m^3
Minimum mineable matrix thickness	1 m
Mining roof loss	100 mm
Mining floor dilution	75 mm
Geology and mining recovery factor	95%
Buffer between pit and river	100 m
Full production mining months per year	9 months
Reduced production mining months per year	3 months
Mine dewatering possible	Yes
Material to support truck traffic	Yes
Spoil Stackability	Yes

The overall 20° permanent slope angle is the controlling factor for the slope recommendations. The temporary dig face angle of 65 degrees is an assumed typical temporary slope angle cut by an excavator or loader that, over time, will slough and erode to a flatter slope angle. The benches in the higher cohesion clay soils will maintain steeper bench faces over the lifetime of the pit wall. Near surface soils may be expected to have additional cohesion from laterite formation and cementation by iron oxides. Cohesionless sand will reach flatter bench face angles over time. The intent of the slope design is to maintain an effective safety bench through the duration of the phased final pit walls. The 25° permanent bench face angle represents the minimum expected long term bench face angle and provides a 6.5-m wide safety bench.

The mine plan production scenario was targeted to produce approximately 2.19 Mtpa of ROM phosphate matrix on an as-received basis (at approximately 20% moisture), or 1.75 Mtpa ROM phosphate matrix on a dry basis. The mine production schedule was developed to achieve these targets and to optimize the plan to defer costs and maximize net present value (NPV) while also providing a reasonable lead-in time for pit dewatering and surface water management activities. Separate scheduling blocks 50 m by 50 m in size were developed for the FPA matrix and each 10 m overburden interval. This block size was chosen to provide a high degree of resolution while maintaining the ability to analyze an alternative scheduling option in a timely manner. The scheduling blocks were confined by the 25 year mine plan pit shell and topographic surfaces to exclude volumes or tonnages outside of the pit.

Four key factors drove the progression of the sequence. In decreasing order of importance, these were: annual ROM production, stripping ratio, dewatering and surface water management, and backfill opportunities. The mine was sequenced with stripping ratio increasing from low-to-high to the extent possible to defer capital and operating costs and to minimize investment risk. The mine sequence includes six months of pre-stripping in “Year 0” to allow for immediate matrix production in Year 1. Approximately three months of matrix inventory was pre-stripped in Year 0.

The yearly production statistics associated with the sequence are shown in Table 1-5. Note that a Year 26 was added to the production schedule to mine out the remaining 257,000 t of matrix in the designed pit shell.

Table 1-5 Annual Mine Plan Production Statistics

Category	Units	Production Year															
		0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
In Situ Overburden Volume	000s BCM	5,811	11,077	14,819	14,214	12,978	11,798	10,849	13,146	17,598	18,253	17,153	17,854	19,391	19,412	19,373	19,355
In Situ FPA Tonnes, Dry Basis	000s tonnes	-	1,855	1,857	1,858	1,856	1,856	1,856	1,857	1,860	1,875	1,855	1,857	1,856	1,854	1,854	1,855
In Situ FPA Thickness	m	-	4.04	3.75	3.54	3.88	3.95	4.15	3.80	3.65	3.69	4.27	3.62	3.69	4.23	4.24	3.89
In Situ P_2O_5, Dry Basis	%	-	31.56	31.85	30.94	30.87	31.10	31.78	31.23	29.55	29.27	28.85	28.71	30.96	30.90	31.05	31.93
In Situ Al_2O_3, Dry Basis	%	-	2.50	2.27	2.21	2.16	2.37	2.22	2.17	2.72	2.82	2.47	2.72	2.04	1.75	1.61	1.50
In Situ CaO, Dry Basis	%	-	40.67	41.37	41.60	40.87	39.90	39.44	39.89	38.97	39.41	40.47	39.85	42.83	43.49	43.33	42.22
In Situ Fe_2O_3, Dry Basis	%	-	3.96	3.67	3.70	4.03	4.32	3.68	3.28	5.36	7.31	5.53	4.81	3.80	3.63	3.98	5.61
In Situ SiO_2, Dry Basis	%	-	11.21	10.66	11.12	11.46	11.41	11.54	11.24	11.78	10.46	11.89	13.31	10.09	8.86	8.73	8.90
ROM Waste Volume	000s BCM	5,818	11,172	14,922	14,318	13,079	11,898	10,947	13,247	17,701	18,369	17,252	17,959	19,494	19,512	19,472	19,454
ROM (Plant Feed) Tonnes, Dry Basis	000s tonnes	-	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	-	6.38	8.53	8.18	7.47	6.80	6.26	7.57	10.11	10.50	9.86	10.26	11.14	11.15	11.13	11.12
Cumulative ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	-	9.71	9.12	8.81	8.47	8.14	7.82	7.79	8.08	8.35	8.50	8.66	8.87	9.04	9.19	9.32
ROM P_2O_5, Dry Basis	%	-	30.97	31.21	30.26	30.26	30.51	31.20	30.61	28.95	28.62	28.31	28.08	30.30	30.31	30.47	31.31
ROM Al_2O_3, Dry Basis	%	-	2.50	2.27	2.21	2.16	2.37	2.22	2.17	2.71	2.80	2.47	2.72	2.04	1.75	1.61	1.50
ROM CaO, Dry Basis	%	-	40.67	41.37	41.60	40.87	39.91	39.44	39.89	38.97	39.43	40.47	39.85	42.83	43.49	43.33	42.22
ROM Fe_2O_3, Dry Basis	%	-	3.96	3.67	3.70	4.03	4.32	3.68	3.28	5.35	7.33	5.53	4.81	3.80	3.63	3.98	5.61
ROM SiO_2, Dry Basis	%	-	11.21	10.66	11.13	11.46	11.41	11.54	11.24	11.78	10.45	11.88	13.31	10.09	8.86	8.72	8.90
Rock (Product) Tonnes, Dry Basis	000s tonnes	-	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321
Rock %P_2O_5 [1], Dry Basis	%	-	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00
Tailings Tonnes [1], Dry Basis	000s tonnes	-	429	429	429	429	429	429	429	429	429	429	429	429	429	429	429

Notes:
1 Expected product tonnages are based off of an average 75.5% plant mass yield.

Category	Units	Production Year											25 Year Total / Average	26 Year Total / Average
		16	17	18	19	20	21	22	23	24	25	26		
In Situ Overburden Volume	000s BCM	19,458	18,707	17,504	17,476	14,995	15,081	15,270	16,944	19,425	21,182	2,827	419,121	421,948
In Situ FPA Tonnes, Dry Basis	000s tonnes	1,855	1,854	1,856	1,856	1,857	1,857	1,855	1,854	1,855	1,856	273	46,417	46,689
In Situ FPA Thickness	m	4.31	4.55	3.56	3.66	3.42	3.45	3.95	4.53	4.19	3.91	3.80	3.92	3.92
In Situ P_2O_5, Dry Basis	%	31.12	31.34	29.96	29.39	29.28	30.01	31.27	31.68	30.17	29.89	30.83	30.59	30.59
In Situ Al_2O_3, Dry Basis	%	1.93	2.33	3.06	3.15	3.37	3.21	3.15	3.33	3.43	3.18	3.91	2.55	2.55
In Situ CaO, Dry Basis	%	41.30	42.31	41.68	40.80	39.27	39.90	41.32	41.72	40.48	40.83	41.26	40.96	40.96
In Situ Fe_2O_3, Dry Basis	%	6.14	4.62	4.67	5.26	5.54	5.66	5.08	4.39	4.66	5.25	4.87	4.72	4.72
In Situ SiO_2, Dry Basis	%	8.42	8.40	9.17	9.48	11.82	11.65	10.18	9.92	12.85	11.12	9.23	10.63	10.62
ROM Waste Volume	000s BCM	19,555	18,806	17,605	17,580	15,099	15,186	15,370	17,040	19,521	21,282	2,841	421,657	424,498
ROM (Plant Feed) Tonnes, Dry Basis	000s tonnes	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	257	43,750	44,007
ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	11.17	10.75	10.06	10.05	8.63	8.68	8.78	9.74	11.15	12.16	11.06	9.64	9.65
Cumulative ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	9.43	9.51	9.54	9.57	9.52	9.48	9.45	9.46	9.53	9.64	9.65	9.64	9.65
ROM P_2O_5, Dry Basis	%	30.58	30.81	29.32	28.78	28.63	29.35	30.66	31.15	29.63	29.32	30.24	29.98	29.99
ROM Al_2O_3, Dry Basis	%	1.93	2.33	3.06	3.15	3.37	3.21	3.15	3.33	3.43	3.18	3.91	2.55	2.55
ROM CaO, Dry Basis	%	41.30	42.31	41.68	40.80	39.28	39.91	41.32	41.72	40.48	40.83	41.26	40.96	40.96
ROM Fe_2O_3, Dry Basis	%	6.14	4.62	4.67	5.26	5.54	5.66	5.08	4.38	4.66	5.25	4.87	4.72	4.72
ROM SiO_2, Dry Basis	%	8.42	8.40	9.17	9.48	11.82	11.65	10.17	9.92	12.85	11.12	9.23	10.63	10.62
Rock (Product) Tonnes, Dry Basis	000s tonnes	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	194	33,031	33,225
Rock %P_2O_5 [1], Dry Basis	%	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00
Tailings Tonnes [1], Dry Basis	000s tonnes	429	429	429	429	429	429	429	429	429	429	63	10,719	10,782

Notes:

Expected product tonnages are based off of an average 75.5% plant mass yield

1.7 Mineral Processing and Metallurgical Testing

The objective of the test work was to quantify the metallurgical response of ore from the Farim Phosphate Deposit. The program was designed to develop the parameters for process design criteria for ore washing/scrubbing, desliming, flotation, and dewatering in the processing plant.

The metallurgical program was conducted by KEMWorks Technology Inc. (KEMWorks), SGS Mineral Services (SGS) and ALS Metallurgy Kamloops (ALS).

The samples used for this testwork were selected to represent the potential mining areas for the first seven years, ore grade, and mineralization types for the South Pit of the Farim deposit.

Five size fractions of the Farim Composite Sample were sent to SGS Lakefield for QEMSCAN analysis. This work confirmed the mineral distributions, mineral release curves, grain size distribution, and chemical analyses by size fractions that were performed by KEMWorks.

100 kg of core samples from the Farim Phosphate deposit were received at KEMWorks on December 26, 2014. This sample consisted of four subsamples, SB9, SC10, SC11, and SE10. These subsamples corresponded to the Block Model and Assay Model data for the deposit, representing the first seven years of production. The samples showed that the main contaminants were A.I. (Insol) and iron bearing minerals as indicated by Fe_2O_3, S_{total}, and $S_{pyritic}$ analyses followed by Al_2O_3 contaminants. These samples are confirmed representative of the deposit. A weighted composite was prepared for characterization studies, horizontal scrubbing (drum), attrition scrubbing, and reverse amine flotation tests.

A composite sample, called the Farim Composite, was prepared after the weighted subsamples were homogenized split. Care was taken to preserve the moisture content of these subsamples.

The Characterization studies, Head chemical analysis, screen analyses, screen assays, and mineralogical QEMSCAN showed that the Farim Composite was representative of this area of the deposit, presenting similar elements and compounds values. The results of the Head Sample chemical analysis showed that the composite P_2O_5 grade was 33.0% ± 0.7% with a 2.0% error, resulting in a P_2O_5 grade between 31.5% to 34.5% range.

The particle size distribution (PSD) reported a mean particle size (d_{50}) of 0.140 mm with a single mode in the distribution (unimodal), the mode located at 0.106 mm (150 mesh). Screen assays showed that aluminum silicates were present containing Al_2O_3 and MgO. The Fe_2O_3, S_{total}, and $S_{pyritic}$ are associated and part of the Fe_2O_3 seemed to constitute part of the aluminum silicates. The A.I. is evenly distributed throughout all size fractions coarser than 0.106 mm and decreasing for particles smaller than 0.106 mm. The A.I. is the most critical impurity to be rejected. QEMSCAN results confirmed this interpretation and conclusions of the screen assays.

To develop the beneficiation process required for the Farim Composite to reach the desired specifications, horizontal scrubbing (drum), attrition scrubbing and reverse amine flotation tests were carried out.

1.7.1 Horizontal Scrubbing

Tests were conducted under standard conditions as a baseline at six different conditions to evaluate two solids contents (35% and 50%) at three scrubbing times: 150 seconds, 300 seconds, and 600 seconds (2.5 minutes, 5 minutes, and 10 minutes, respectively). These tests showed that A.I., Al_2O_3, Fe_2O_3, S_{total}, $S_{pyritic}$, and MgO decreased in the product size range (1.18x0.020 mm). At 35% solids content and 300 seconds (5 minutes) of scrubbing time, the best yield (73.7%) P_2O_5 recovery (77.3%) and P_2O_5 grade (34.4%) was obtained. In addition, the lowest A.I. grade (5.97%) was obtained under these conditions with an A.I. rejection of 34.9%.

- Mass yield 73.7%
- P_2O_5 recovery 78.4%
- CaO/ P_2O_5 ratio 1.4
- MER 0.103
- MER* 0.034

Confirmation tests validated these results. All of these tests were analyzed based on the actual results and then normalized based on the feed grades of each test to eliminate the effect of small differences in feed grade of each test that could mislead the interpretation of results. These tests considered the +6.3 mm and 6.30x1.18 mm size fractions as rejects and the -0.020 mm material as slimes.

1.7.2 Attrition Scrubbing

Tests were designed to release significant amounts of quartz, clay, and iron bearing minerals attached to the francolite surfaces in the 6.30x0.075 mm size fraction obtained after horizontal scrubbing (drum). However, A.I. rejection was limited to the -0.020 mm size fraction due to the hardness of quartz and the small amounts of fine silica locked onto the surface of phosphate bearing minerals according to the QEMSCAN and mineralogical studies. Nine tests were carried at three solids contents (35%, 45%, and 55%) for three different scrubbing times, 150 seconds, 300 seconds, and 600 seconds. The best results were obtained at 55% solids content and scrubbing for 150 seconds (2.5 minutes):

- Mass yield 73.9%
- P_2O_5 recovery 77.2%

- CaO/ P_2O_5 ratio 1.5
- MER 0.075
- MER* 0.070
- P_2O_5 grade 33.8%

Again, normalized data were evaluated and confirmed the results.

1.7.3 Reverse Amine Flotation

Studies of the 1.18x0.106 mm size fraction were carried out. Seven flotation tests were conducted for the selection of the type of condensate amine to be used, and to obtain the best flotation results. ArrMaz CA-1208 amine was selected. The addition of 1.168 kg/ton of flotation feed resulted in a 1.18x0.075 mm concentrate of 36.7% P_2O_5 grade with 2.2% A.I. grade, and 1.48% Fe_2O_3 grade. The P_2O_5 recovery was 97.3% of the P_2O_5 content of the flotation feed with a rejection of 77.4% of A.I. and 17.0% of the Fe_2O_3 of the flotation feed.

The beneficiation process using flotation to further upgrade the ore by removing silica resulted in the following products:

- +6.3 mm rejection 5.2% ± 1.9%
- 6.3x1.18 mm rejection 2.2% ± 0.2%
- 1.18x0.106 mm flotation concentrate 49.3% ± 2.8%
- reverse flotation tailings 4.7% ± 1.7%
- 0.106x0.020 mm fine concentrate 16.6% ± 0.5%
- -0.020 mm slimes rejection 21.9% ± 0.3%

1.7.4 Pilot Plant Results

The results of the pilot plant testwork confirmed KEMWorks' circuit design using horizontal and attrition scrubbing to remove the impurities from the ore to achieve a concentrate product of 34% P_2O_5.

The pilot testing concluded that the following product specifications can be achieved using this process:

- Mass yield 75.5%

- P_2O_5 recovery 78.4%
- CaO/P_2O_5 ratio 1.4
- MER 0.093
- MER* 0.062
- P_2O_5 grade 34.0%

1.8 Recovery Methods

The design of the processing facility for this feasibility phase of the Project is based on the metallurgical test work conducted to date combined with industry best practises.

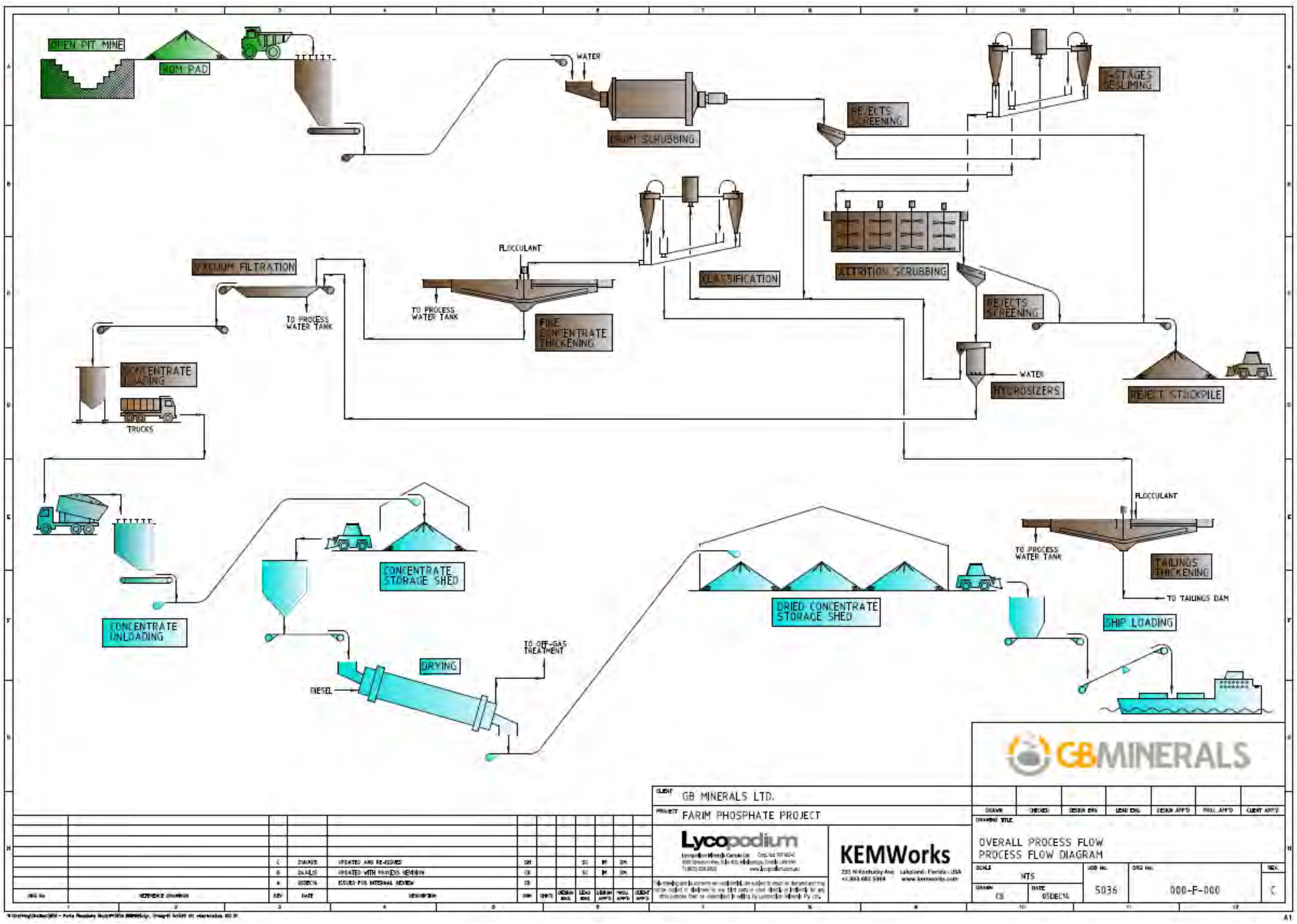
The flowsheet is developed based on the use of scrubbing and sizing technologies while avoiding the use of grinding and wet high intensity magnetic separation to reduce capital and operating costs for the project. The testwork results have successfully proven that the proposed flowsheet is able to achieve the required product specifications.

An overall schematic diagram of the process is shown in Figure 1-3 below.

The following steps are included in the selected flowsheet:

- Ore storage and reclaiming of Run-of-Mine (ROM) ore;
- Two stage scrubbing and screening to reject +1.18 mm material;
- Sizing with hydrosizers and cyclones to separate -1.18 x 0.106 mm material to a coarse concentrate, -0.106 x 0.020 mm material to a fine concentrate material, and to reject -20 µm material to tailings;
- Fine concentrate thickening;
- Concentrate filtration, storage and reclaim;
- Thickening and disposal of tailings (reject material) to the Integrated Waste Landform (IWL) and return of decant water to the beneficiation plant;
- Concentrate reclaim, drying and stockpile at Port site;
- Dried concentrate ship-loading at Port site.

Figure 1-3 Overall Process Flow Diagram



The process description details the 1.75 Mtpa beneficiation plant for the production of 1.32 Mtpa of phosphate concentrate.

ROM will be delivered by 36 tonne dump trucks from the open pit. ROM will either be dumped directly into the ROM bin or dumped onto the ROM stockpile. ROM ore (P_{80} 25 mm) will be dumped by haul trucks or loaded by front-end loaders directly into the ROM bin. A belt feeder will extract ROM rock from the bin to be conveyed to the horizontal scrubber.

The product from the horizontal scrubber will discharge onto a vibrating screen with 5 mm slotted openings to remove +5 mm material. The oversize material from the vibrating screen is considered reject and will be conveyed to the reject bin, to be transported off-site. The screen undersize will be deslimed with a two-stage cyclone cluster circuit using a cut point at 75 µm. The overflows of the cyclone clusters will combine with the overflow of the hydrosizers (-106 µm material). The underflow of the secondary cyclone cluster will flow into the attrition scrubber. The attrition scrubber will have four compartments – each 3.8 m^3 in volume to give a total retention time of 5 minutes.

Attrition scrubber products will discharge onto a vibrating screen with 1.18 mm slotted openings to remove +1.18 mm material. Oversize material from the vibrating screen is considered reject and will be combined with the +5 mm material on a rejects conveyor to be conveyed to the rejects bin. Vibrating screen undersize will be pumped to two hydrosizers for additional separation at 0.106 mm.

Hydrosizer underflow at 1.18 x 0.106 mm and 70% solids density will be diluted to 55% solids in an agitated tank prior to being pumped to the concentrate filter feed tank. Hydrosizer overflow at -0.106 mm will be sent to a pump feed tank to be combined with the desliming cyclones overflow from which the material will be pumped to a cyclone cluster for classification at 0.020 mm. Classification cyclone underflow at 45% solids will become the 0.106 x 0.020 mm fine concentrate and reports the fine concentrate pump tank for transfer to the fine concentrate thickener. The -0.020 mm cyclone overflow will be rejected as fines and will be sent to the coarse tailings tank.

The concentrate filtration and storage area will include a vacuum belt filter, a product transfer conveyor, and concentrate bin. The concentrate transfer conveyor crosses the Cacheu River and discharges into a 2,000 m^3 live concentrate bin. Concentrate dump trucks, with 31 tonne payload, will drive under the concentrate bin to be loaded for transport to the port facility.

Use of reagents is not required in the process, but space in the beneficiation plant is allocated for future addition if reverse flotation is required.

Filtered concentrate from the beneficiation process plant is trucked 75 km southwest, unloaded at the port of Ponta Chugue, sent through a rotary dryer to achieve 3% moisture, stored, and conveyed via shiploader onto 35,000 DWT ships to be transported to market.

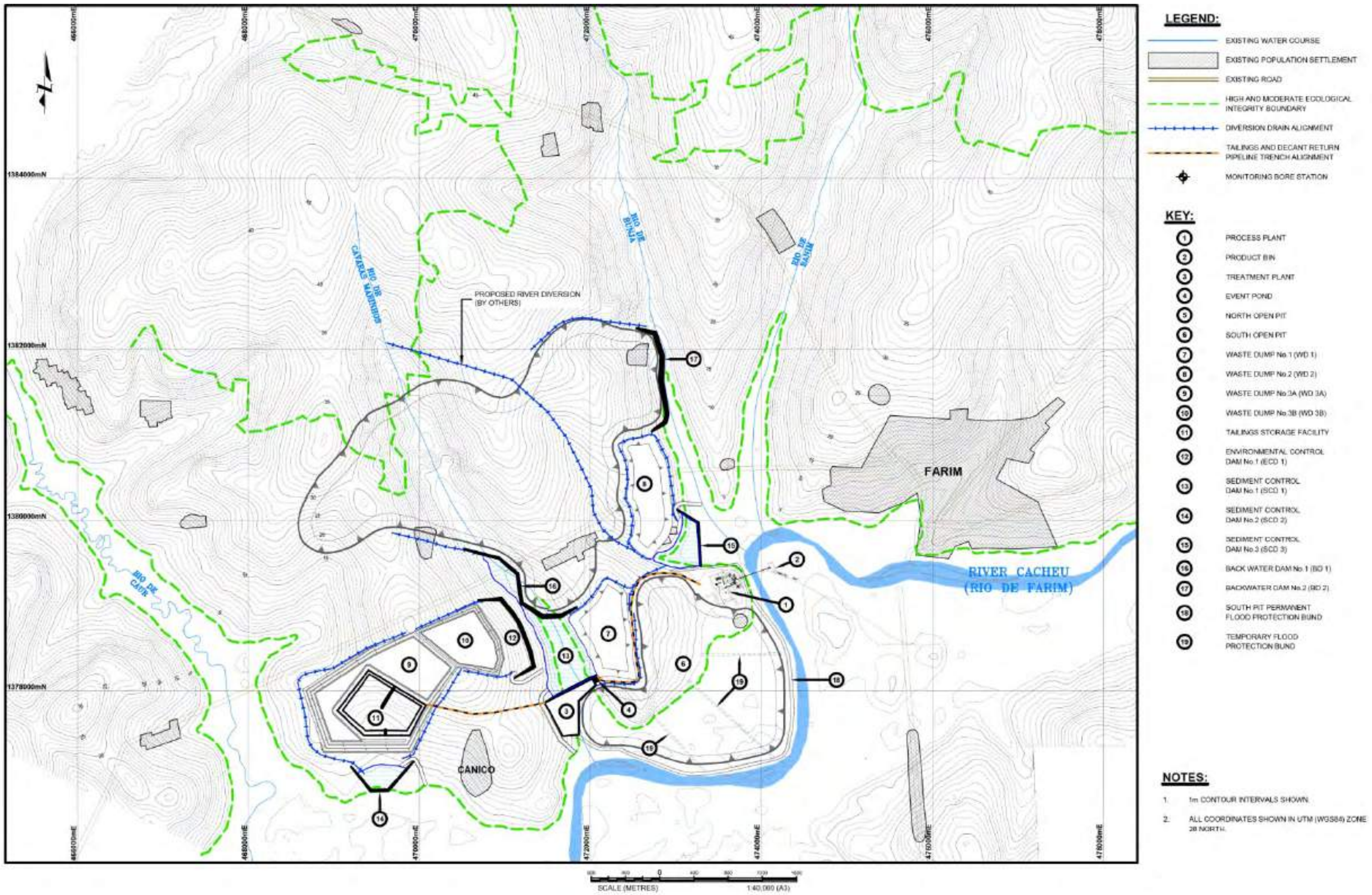
1.9 Project Infrastructure

The mine site is approximately 5 km west of the town of Farim. The mine site is bound by the Cacheu River to the east and south of the open pit. The beneficiation plant has been located between the southern and northern open pits, adjacent to the Cacheu River. The plant area, including site buildings, is approximately 200 m x 200 m. The beneficiation plant is located at the narrowest point of the Cacheu River, where it is approximately 150 m wide, to minimize the cost of the conveyor crossing. A conveyor is utilized to transfer dewatered phosphate rock into a storage bin on the east side of the river, where trucks are loaded.

Site buildings at Farim and Ponta Chugue sites will include a single-storey administration building, plant kitchen, change houses, ablutions, combined plant workshop/warehouse, and main security gatehouse. A laboratory used to test metallurgical samples will also be installed adjacent to the beneficiation plant at Farim. At Ponta Chugue a 109 m x 21 m wet concentrate shed and 150 m x 36 m dry concentrate shed will be installed for drying and storing of the product.

The Integrated Waste Landform (IWL), a tailings co-disposal facility, is located approximately 5 km west of the process plant. The project mining plan indicates that of the 539 Mm^3 (loose cubic metres) of waste “rock” produced during the mining operation, 78% (420 Mm^3) will be placed as in-pit backfill, 13% (71 Mm^3) placed as surcharge overburden stockpile, and 9% (48 Mm^3) deposited in ex-pit waste dumps. Four ex-pit waste dumps are proposed, three of which (WD1, WD2, and WD3a) are proposed to stockpile inert waste only. Two dumps will be located between the North and South pits (WD1 and WD2). A third (WD3a) will be formed around the perimeter of the proposed tailings storage facility, to form an integrated waste landform (Figure 1-4). A fourth waste dump, WD3b, is located directly to the northeast of WD3a but will contain potentially leachable waste in specially designed encapsulation cells contained within inert waste.

Figure 1-4 Tailings and Waste Rock Facilities



The Farim and Ponta Chugue onsite and offsite roads will be constructed of crushed waste rock from existing quarries in Guinea-Bissau and from any naturally available materials. At Farim, the offsite gravel road is approximately 2 km in length and connects the truck loadout facility to the new paved highway leading to Ponta Chugue. At the port in Ponta Chugue, the offsite gravel road is approximately 4 km in length and connects the port facilities to the paved highway.

Power supply to the Farim plant site and the Ponta Chugue Port Facilites will be from a Diesel Onsite Power Plant (OPP). At Farim, the power plant will supply a Main HV switchroom inside the processing plant from which power will be distributed at 11 kV.

The configuration of the Farim Plant OPP is:

- 4 x 1.2 MW prime rated 11 kV generators (3 duty, 1 standby).
- 11 kV switchroom.

The configuration for the Port OPP is:

- 3 x 0.5 MW prime rated 0.4 kV generators (2 duty, 1 standby).
- Direct feed to the LV switchroom.

The primary elements of infrastructure making up the direct load wharf at Ponta Chugue include steel pile bents to support the conveyor and truss system delivering phosphate to the wharf, two steel pile supported platforms are used to support a single telescoping radial shiploader, four steel pile supported mooring dolphins, four steel pile berthing dolphins with steel decks, access gangways, a floating wharf to moor 2 tug boats and the pilot boat, maintenance barge, guide piles, access ramp, and navigation aids along the anticipated navigation route.

1.10 Environmental Studies, Permitting, and Social or Community Impact

Below is a summary of the environmental and social considerations relevant to the Project, including:

- Environmental, cultural and socio-economic setting and studies conducted;
- Regulatory context; and
- Known environmental issues that could materially impact the issuer’s ability to proceed with the planned/proposed mining development.

In October 2014, Lycopodium contracted Knight Piésold to complete a gap analysis of the environmental baseline data and the previous ESIA as part of the feasibility study team (Knight Piésold, 2014). This gap analysis formed the basis of supplemental baseline studies undertaken by Knight Piésold in April and May 2015. A summary of previous (2011 to 2013) and 2015 supplemental baseline studies is presented in Table 1-6.

Table 1-6 Summary of Environmental, Socio-economic and Cultural Heritage Baseline Studies

Discipline	Previous Baseline Studies (2011-2013)	Supplemental Baseline Studies (2015)
Meteorology	A meteorology station has operated nearly continuously at Farim since 2011.	Updated analysis of additional meteorological data completed; port site meteorology was described from Bissau climate records.
Air quality	Baseline measurements of particulate matter (PM), sulphur dioxide (SO_2), nitrogen oxides (NO_x) and dustfall collected in 2012 at representative locations at the mine site, along the transportation route and near the port site.	No supplemental data collection deemed required.
Noise and vibration	Noise measurements collected at receptor locations near the site and along the transport route.	Noise measurements collected at the port site.
Geochemistry	50 overburden samples were collected from 2 boreholes in each of the north and south pit. Samples were composited and analyzed for acid rock drainage / metal leaching (ARD/ML) potential. Chemical analysis for metals completed on 50 ore samples.	The collection of additional overburden samples of overburden is currently underway for acid rock drainage/metal leaching (ARD/ML) potential. Tailings samples from bench scale testing (1 sample) completed and additional testing of tailings from a pilot plant testing underway. Ore and phosphate product undergoing testing including chromium (VI).
Soils	Comprehensive soil sampling program and land capability assessment within the mine site area.	Supplemental soil sampling program conducted at the mine site (metals only), and the port site (metals and soil fertility parameters).
Surface water	Surface water sampling conducted over multiple wet and dry seasons at the mine site.	Surface water sampling was conducted in the vicinity of the port site.
Groundwater	Comprehensive groundwater investigations completed, and one dry season and wet season sampling campaign completed.	Additional wells installed at the mine site and pump tests conducted. Revised groundwater model prepared. Supplemental groundwater quality sampling (dry season) conducted at select wells in the mine and port areas.
Aquatic ecology	Aquatic studies conducted in the River Cacheu and tributaries near the mine site.	Aquatic studies conducted in the River Geba at the port site, and supplemental aquatic studies at the mine site.

Discipline	Previous Baseline Studies (2011-2013)	Supplemental Baseline Studies (2015)
Terrestrial ecology	Terrestrial ecology studies conducted in the mine site area.	Terrestrial studies conducted at the port site, with supplemental terrestrial ecology studies at the mine site focusing on biodiversity.
Socio-economics	Preliminary socio-economic surveys and data collection.	Household surveys at the mine and port. Detailed land use mapping at the mine and port site areas.
Cultural heritage	The mine site area surveyed by a qualified international archaeologist.	The port site area surveyed by a qualified international archaeologist, and a follow-up survey was completed at the mine site.

A round of public meetings were held in May and June 2015 to present the Project plans and to solicit feedback from the Guinea-Bissau Government, local communities and other interested stakeholders. These meetings are in accordance with the company`s Stakeholder Engagement Plan (Knight Piésold, 2015a). The feedback from these engagement sessions will be incorporated into the ESIA that is currently under preparation.

The ESIA will be provided to Equator Principle Financial Institutions (EPFIs) potentially interested in financing part or the entire Project. The ESIA will be translated into Portuguese for submission to the Guinea-Bissau Government, as well as other stakeholders. Summary ESIA information will be prepared and presented to local stakeholders in Portuguese, or presented orally in the local languages of Creole and Mandinga.

1.11 Capital Costs

The capital cost for mining, process (beneficiation) plant facilities, port facilities, marine services, tailings waste management facilities and infrastructure required to treat the throughput capacity of 1.75 Mtpa, for “Farim Phosphate Project”, is USD $193.8 million excluding Owner’s cost in third quarter 2015 US dollars. The Owner’s cost is estimated at a cost of USD $11.9 million and includes items such as Owner’s construction team cost, USD $4.0 million Resettlement allowance, USD $2.0 million for insurance.

The capital cost is summarized in Table 1-7 and is inclusive of the costs to design, procure, construct up to and including plant commissioning and start up; sunk cost, sustaining capital cost, interest during construction, deferred capital costs, escalation, foreign exchange fluctuations and owner’s costs are excluded from these estimates.

Table 1-7 Capital Cost Summary

Area Description		Total USD
Contractor's Preliminary & General (P & G) Costs Including Mob & Demob Costs		10,855,245
Mining		66,097,679
Process (Beneficiation) Plant		38,265,761
Tailings and Water Management Facilities		8,039,851
Port Facilities		15,646,802
Marine Facilities and Services		23,836,865
Total Direct Cost		**162,742,202**
Indirect Cost		17,444,509
Subtotal for Contingency		**180,186,711**
Contingency (overall average)	7.6%	13,635,703
Total Direct & Indirect Costs		**193,823,000**
Owner's Cost		
Project Total		**193,823,000**

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs) were identified and analyzed. To each of these categories, a percentage of contingency was allocated based on the accuracy of the data and an overall contingency amount derived for the process plant and the port facilities. Other consultants provided their own contingencies.

Golder estimated the costs of matrix production and capital requirements associated with producing FPA matrix from the two Farim mining pits. Production cost and project capital estimates were developed on an annual basis to reflect the yearly matrix release, waste removal (or “stripping”) requirements, and matrix/waste haulage parameters dictated by the respective mine plan. The mining cost estimate assumes all mining functions are directly performed by GB Minerals using company-owned equipment and company employees.

The Farim beneficiation plant and associated facilities estimates have been prepared on a commodity basis (i.e. divided into earthworks, concrete, structural steel, architectural, etc.) and reported by area (i.e. Feed Preparation, Reclaim, Concentrate Stockpiling, etc.).The estimate is based on the purchase of new mechanical equipment and quantities have been assessed from first principles.

Knight-Piésold established the scope and quantities for the Integrated Waste Landform (IWL), surface water management, and dewatering infrastructure. Knight-Piésold estimated the earthwork costs based on West African contractor rates.

Port marine costs were based on the scope of work established by Baird and Associates and the capital estimate has been prepared by same.

1.12 Operating Costs

The direct cash operating cost for the Farim Phosphate Project have been estimated under three functional headings: mining, process plant and general and administration (G&A). The operating costs have been estimated by the following parties:

- Mining – Golder and GB Minerals
- Beneficiation Plant and Port Facilities – Lycopodium, Baird and GB Minerals
- G&A – Lycopodium and GB Minerals.

The operating cost estimates are expressed in US dollars (USD) in first quarter 2015 terms and are expected to be accurate within ±15%.

A summary of the life-of-mine (LOM) operating costs are summarized in Table 1-8.

Table 1-8 Operating Cost Summary

COST CENTRE	Total Cost		
	USD/year	USD/t conc.	USD/t ore
Process & Admin. Labour	$ 6,626,034	$ 5.01	$ 3.78
Operating Consumables	$ 11,269,791	$ 8.53	$ 6.44
Power	$ 6,995,841	$ 5.30	$ 4.00
Maintenance	$ 1,360,007	$ 1.03	$ 0.78
Shiploading	$ 3,127,351	$ 2.37	$ 1.79
G&A Expenses	$ 3,535,000	$ 2.68	$ 2.02
Corporate Costs	$ 2,912,500	$ 2.20	$ 1.66
Mining Total	$ 33,044,463	$ 25.01	$ 18.88
TOTAL	**$ 68,870,097**	**$ 52.13**	**$ 39.35**

1.13 Economic Analysis

This financial model is prepared to reflect the revenue stream and corresponding operating cost for GB Mineral's Farim Phosphate green-field project which contains measured and indicated resources of 105.6 million tonnes at 28.4% P_2O_5, and additional inferred resources of 37.6 million tonnes at 27.7%% P_2O_5. The reserves are estimated at 44.0 million tonnes at 30.0% P_2O_5 based on a 25 year mining plan. 1.75 Mtpa of ore are mined with 1.32 Mtpa of beneficiated phosphate rock product produced. The final beneficiated phosphate rock concentrate will have a grade of 34%.

The financial analysis model covers the time span from years -3 through +27 with pre-production years of year -3, -2 and -1. Detailed engineering, construction and pre-stripping is assumed to occur during the pre-production period, it is envisaged that all the necessary permits to commence construction and execute this project will be in place at this time. Production years are from +1 to +25.

Project closure is deemed to take place in years +26 and +27. Golder estimated the costs of matrix production and capital requirements associated with producing FPA matrix from the two Farim mining pits. Production cost and project capital estimates were developed on an annual basis to reflect the yearly matrix release, waste removal (or "stripping") requirements, and matrix/waste haulage parameters dictated by the respective mine plan.

It is assumed that the company will mine the south pit from the last quarter of -1 production year to production year +8 and the then mine the north pit for the rest of the mine life.

The bench scale tests have been performed on samples from the South pit only. For the South pit, a 9.7% premium over the CRU Group's (CRU) estimate for Morocco K10 FOB price has been assumed. Further bench scale tests on the North pit will be performed in the fourth quarter of 2015. Because of the modest differences in the ore in the South pit versus the North pit, a premium of 4.7% has been assumed for the North Pit until bench scale tests for the North pit can be completed. Product pricing was provided by CRU Group (CRU) in July 2015 for the period of 2015 to 2019, and include an average long term of USD $123/tonne for the K10 Morocco P_2O_5 from 2019 onward. Added to this price are premium percentages for the higher grade of the Farim phosphate ore. From 2020 onward, the model pricing has been computed using the current K10 Morocco P_2O_5 CRU price from 2019 and then escalated on a yearly basis at a rate of 2% per annum. The product is priced on an FOB basis, it therefore includes all operating costs up to loading on the ocean vessels.

Table 1-9 and Table 1-10 summarize the financial analysis modeled. NPV is calculated on an end basis.

Table 1-9 Financial Data

FINANCIAL DATA		
Revenue	USD$ X'000	5,476,899
Total Pre-Production Capital	USD$ X'000	205,279
Life of Mine Operating Cost	USD$ X'000	2,409,967
Total Sustaining Capital	USD$ X'000	366,597
Operating Margin Ratio (Op. Revenue / OpEx)		2.3
Royalties	USD$ X'000	109,538
Income Taxes	USD$ X'000	443,898
Pre-Tax Cumulative Cash flow	USD$ X'000	2,358,458
After-Tax Cumulative Cash flow	USD$ X'000	1,914,560

Table 1-10 Financial Statistics

FINANCIAL STATISTICS			
		After Tax	**Pre-tax**
Cumulative net cash flow			
Undiscounted (BASE YEAR 2015)	USD $000	1,914,560	2,358,458
Net present value			
Discounted at 5%	US$000	869,789	1,026,461
Discounted at 8%	US$000	570,224	657,860
Discounted at 10%	US$000	436,891	497,396
Discounted at 15%	US$000	231,384	256,679
Internal rate of return	US$000	34.5%	34.9%
Payback period	Years	4.3	4.3

1.14 Project Implementation

The overall schedule duration from the start of detailed engineering to the end of commissioning is 19 months. The engineering activities will take approximately 10 months, the site construction activities will be completed in 12 months followed by commissioning. This schedule is based on Lycopodium's understanding of the project scope, current lead times for the delivery of critical equipment, and typical duration of engineering and site activities based on similar size projects executed by Lycopodium. The major project milestones are summarized in Table 1-11 below.

Table 1-11 Major Project Milestones

Major Milestone	Month
Start of Detailed Engineering	Month 1
Award Bulk Earthworks Contract	Month 4
Start Construction (Bulk Earthworks)	Month 5
Start Concrete Works	Month 8
Start SMP Installation	Month 9
Start Field Erected Tankage	Month 10
Detailed Engineering Complete	Month 11
Start E&I Installation	Month 11
Complete Bulk Earthworks (Farim site)	Month 12
Field Erected Tankage Complete (Farim site)	Month 14
Concrete Works Complete (Farim site)	Month 15
SMP Installation Complete (Farim and Port sites)	Month 16
E&I Installation Complete (Farim and Port sites)	Month 17
Start Commissioning	Month 17
Commissioning Complete	Month 19

The major long lead delivery items have been considered in the schedule, which are:

- Shiploader - 11 Months ARO (After Receipt of Order);
- Diesel Power Plants – 9 Months ARO;
- Rotary Dryer – 9 Months ARO;
- Attrition Srubber – 9 Months ARO.

1.15 Interpretation and Conclusions

The following conclusions arise from the information provided in the previous sections:

- The scope of design is estimated to require an initial capital investment of USD $193.8M, and sustaining capital of USD $366.6M.

- Life of mine operating costs for the project are estimated to be USD $52.13/t, which falls into the lowest quartile of the phosphate rock industry business cost curve (source: CRU Group)

- Based on a P_2O_5 price of USD $123/t plus a 9.7% premium over the CRU estimate for Morocco K10 FOB price, the after-tax NPV_{10} for the Project is USD $436.9M, while the after tax IRR is 34.5% and the payback period is 4.3 years. The economic analysis demonstrates robust economics and confirms the overall viability of the project. There is consequently justification for advancing to the next phase of detailed engineering.

- The reserves outlined in the study are based on a targeted mine life of 25 years. Additional Measured and Indicated resources have been delineated on the property, which have the potential to add substantial additional reserves.

- The phosphate rock produced is a high grade, high quality, product that will attract a premium price.

- The samples used for this testwork were selected to represent the potential mining areas for the first seven years, ore grade, and mineralization types for the South Pit of the Farim deposit.

- The beneficiation plant is based on bench scale and pilot plant testwork designed for optimum recovery and minimum operating costs. The flowsheet is based upon unit operations that are proven in industry.

- Foundation design for the marine loading facility and navigation aids was based on geotechnical information from land-based boreholes. As such, the foundation design may require adjustment during final design to accommodate geotechnical conditions actually existing in the Geba River.

- While full bridge navigation simulations have not been conducted to verify the navigability of the Geba River to Ponta Chugue, some confidence in its navigability can be gained as deep draft vessels are currently visiting the Port of Bissau.

- The ground conditions at the TSF, processing plant (plant site west) and port site typically comprise overconsolidated clay interbedded with sand horizons and near surface laterite layers in places. These ground conditions are considered suitable for the proposed development.

- The ground conditions at the proposed product bin site (plant site east) are poor and similar to those identified along the southern side of the proposed South Pit. The ground is not considered suitable to support notable structures on spread footings, and therefore piling has been proposed and budgeted.

- A limited tailings testing program has been carried out to date. Consequently, the tailings physical behavior characteristics need further definition. This has implications for the TSF staging and water balance. Testing of a larger representative tailings sample at the nominated design tailings percent solids will be required to confirm the tailings properties for final design purposes. More definitive tailings testing should provide for optimization of the TSF staging and may provide embankment cost savings.

- It may be feasible to optimize the TSF embankment raise construction schedule by providing for increased inter-stage capacity, thereby reducing the total number of raises and hence the overall earthworks contractor mobilization costs.

- The political, location, environmental, social and permitting risks appear to be generally commensurate with other mining projects in West Africa. EIS work completed to date has not resulted in the identification of any fatal flaws or impacts that are expected to be of critical significance with mitigation measures applied.

- The Project is subject to a signed Mining Agreement, a mining lease (granted) and a mining licence (granted). The Project is also subject to an environmental review by the Government of Guinea-Bissau (GoGB). Successful completion of the Incentive Annex and the ESIA review both represent permitting risks that are judged to be low based on the priority the GoGB appears to place on seeing the Project be developed.

- An ESIA for the mine site area only was completed in December 2014, and a project-wide ESIA is near completion based on the project design presented in this technical report. The ESIA is being drafted to be compliant with the World Bank Equator Principles III (Equator Principles Association, 2013) and the IFC Performance Standards on Environmental and Social Sustainability (IFC, 2012).

1.16 Recommendations

Recommendations for future work are listed below:

- The Technical Report for the Farim Phosphate Project has been completed in sufficient detail to refine the economics to a +/-15% level of accuracy and outline the issues facing the project going forward. The project economics are sufficiently robust to warrant moving to the next phase of detailed engineering and construction.

- The results from the 10 recently completed beneficiation and metallurgical drill holes should be used to update the geologic resource model once the data and observations are available for these drill holes.

- Further investigation into the bearing capacity and wear characteristics of the material on site and proposed road construction methods to ensure adequate "trafficability" particularly in the rainy season.

- Consideration should be made to develop an onsite quarry to reduce the cost of road material.

- Determine the rheological characteristics of the products and tailings to determine the slurry and pumping characteristics, static and dynamic settling, and filtration characteristics.

- Evaluate the settling and filtration parameters in the presence of coagulants and/or flocculants for the design of the thickeners and filtration devices.

- Perform variability bench scale tests for different areas of the South Pit and of the North Pit of the deposit applying the beneficiation technology developed.

- Carry out extensive pilot plant tests for each the North and South Pit phosphate ore to obtain enough information on material balances, operating conditions, variability effects, products and their marketing, and to evaluate the use of column flotation cells for 0.106x0.020 mm size fraction when high iron bearing minerals are present.

- Implement a metallurgy testwork program to include:

  - Vacuum belt filter dewatering

  - Bulk material handling flowability tests for product bin design

  - Drying optimization tests

- Conduct continuous phosphoric acid plant tests to assess likely performance in an industrial plant. Conduct bench-scale phosphoric acid concentrations and clarification tests, and bench-scale fertilizer test work.

- Conduct initial geotechnical investigations for the bulk carrier loading facility at Ponta Chugue, aids to navigation in the Geba River, and the Cacheu River crossing structures at Farim

- Develop a marine operational readiness plan that details necessary training for vessel operators, logistics channels for sourcing spare parts, International Ship and Port Facility Security Code (ISPS) requirements, safety procedures, equipment and personnel required to maintain the marine facility.

- Conduct an analysis to estimate scour around the piles supporting the Cacheu River crossing structure.

- Conduct an investigation into the potential for sedimentation of the bulk carrier berth at Ponta Chugue.

- Conduct desktop and full bridge navigation simulations to better understand the navigability of the Geba River, the recommended vessel berthing procedures, and propulsion requirements of the assisting tugs.

- Gather additional hydrographic data between Banco do Alenquer and Ponta de Caio to validate the allowable vessel draft recommendation.

- Conduct further geotechnical investigations for all surface infrastructure, including the beneficiation plant site, and Ponta Chugue port facilities.

- Complete physical, geochemical and radiological testing programs on a representative sample of tailings in order to confirm the tailings characteristics for design.

- Complete a geochemical testing program on samples of specific geological lithologies in order to de-lineate and quantify mine waste in terms of material type and geochemical risk.

- Provide an updated orebody geological model to allow a refinement of the hydrogeological model domain. In this regard, the development of a block geological model, to be used as a basis for the hydrogeological and other mining design and development purposes is essential.

- Completion of additional pumping tests in the southern pit to improve understanding of the groundwater flow regime, the hydraulic connection with nearby creeks and surface water bodies, water impacts associated with mining the pit and the variation of aquifer hydraulic properties over the area of interest.

- Submit the ESIA (Environmental Plan under the Mining Agreement) and the Mining Operations Plan to the government of GoGB, and complete negotiation of the Incentive Annex with the GoGB, as soon as possible as intended.

- Initiate development of the Resettlement Action Plan and cultural heritage mitigation plans as soon as possible, to minimize the potential for this aspect to affect the development schedule.

- Complete the planned supplemental wet season biodiversity field program to identify any flowering plants or other species of conservation concern, and update the Biodiversity Management Plan accordingly.

- Stakeholder Engagement – The Project has conducted several rounds of public consultation meetings over the last several years, but intensification of this consultation in the near term will be necessary both before and after distribution of the ESIA.

2.0 INTRODUCTION

2.1 General

The following Technical Report was compiled by Lycopodium Minerals Canada Ltd. (Lycopodium) and presents the results of the Feasibility Study for the Farim Phosphate Project, located in north-central Guinea-Bissau, West Africa, approximately 25 km from the northern border with Senegal and 80 km north of the capital, Bissau. The Technical Report was prepared at the request of GB Minerals Ltd. (GB Minerals), a British Columbia corporation. GB Minerals is a Canadian mining and development company that is focused on developing the Farim Phosphate Project. GB Minerals is listed on the TSX Venture exchange (GBL).

Lycopodium was commissioned by GB Minerals in September 2014 to prepare the NI 43-101 compliant technical report on the project. The purpose of this Technical Report is to provide GB Minerals with sufficient information to determine the economic feasibility of developing the Farim Phosphate Project. This Technical Report, the resource and reserve estimate, and the Feasibility Study have been prepared in compliance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014.

This NI 43-101 Technical Report was completed by:

- Lycopodium Minerals Canada Ltd. ("Lycopodium") for the process plant infrastructure, port facilities on land infrastructure, and process plant and port facilities operating costs;
- KEMWorks Technology Inc. ("Kemworks") for metallurgical test work and process design;
- Golder Associates Inc. ("Golder") for mining and geology;
- W.F. Baird & Associates Ltd. ("Baird") for marine infrastructure, marine vessels, Capital and Operating costs for marine operations and shipping;
- Knight-Piésold Pty. Ltd., Perth, Australia, ("Knight-Piésold") for the design of the integrated waste landform, geotechnical, hydrogeology, hydrology, site water management, geochemistry and infrastructure design support. Knight-Piésold's Canadian office in North Bay, Ontario, was responsible for environmental studies, and social/community impact.
- Alex Duggan (Kristal Font Incorporated) for capital costs and economic analysis.

Unless otherwise denoted, all costs referred to in this Feasibility Study are quoted in current Q3 2015 United States Dollars.

2.2 Sources of Information

Lycopodium has based its interpretation on the data and information provided by GB Minerals for the completion of this report. The information provided by GB Minerals and other references are listed in Section 27.

2.3 Qualified Persons

The responsibilities of each author are provided in Table 2-1.

Table 2-1 Technical Report Section List of Responsibility

Section Number	Section Title	Responsible QP	Co-Author	Other Experts
1	**Summary**	Lycopodium (Dan Markovic)/ Golder (Ted Minnes) /Knight-Piésold (David Morgan, Richard Cook)/Kemworks (Dr. Francisco Sotillo)/Baird (Ed Liegel)		GB Minerals
2	**Introduction**	Lycopodium (Dan Markovic)		GB Minerals
3	**Reliance on Other experts**	Lycopodium (Dan Markovic)		
4	**Property Description and location**	Lycopodium (Dan Markovic)	GB Minerals	
5	**Accessibility, Climate, Local Resources, Infrastructure and Physiography**	Lycopodium (Dan Markovic)	GB Minerals	
6	**History**	Lycopodium (Dan Markovic)	GB Minerals	
7	**Geological Setting and Mineralization**	Golder (Jerry DeWolfe)		GB Minerals
8	**Deposit Types**	Golder (Jerry DeWolfe)		GB Minerals
9	**Exploration**	Golder (Jerry DeWolfe)		GB Minerals
10	**Drilling**	Golder (Jerry DeWolfe)		GB Minerals
11	**Sample Preparation, Analyses and Security**	Golder (Jerry DeWolfe)		GB Minerals
12	**Data Verification**	Golder (Jerry DeWolfe)		GB Minerals
13	**Mineral Processing and Metallurgical Testing**	KEMWorks (Dr. Francisco Sotillo)	Lycopodium	
14	**Mineral Resource Estimates**	Golder (Jerry DeWolfe)		GB Minerals
15	**Mineral Reserve Estimates**	Golder (Ted Minnes)		
16	**Mining Methods**	Golder (Ted Minnes, George Lightwood)		

Section Number	Section Title	Responsible QP	Co-Author	Other Experts
17	**Recovery Methods**	KEMWorks (Dr. Francisco Sotillo), Lycopodium (Dan Markovic)	Lycopodium	
18	**Project Infrastructure**	Lycopodium (Dan Markovic), Knight-Piésold (David Morgan), Baird (Ed Liegel)		GB Minerals
19	**Market Studies and Contracts**	Lycopodium (Dan Markovic)	GB Minerals	
20	**Environmental Studies, Permitting and Social or Community Impact**	Knight-Piésold (Richard Cook)		GB Minerals
21	**Capital and Operating Costs**	Kristal Font Inc. (Alex Duggan), Lycopodium (Dan Markovic)/ Golder (Ted Minnes)	Knight-Piesold (David Morgan)/Baird (Ed Liegel)	GB Minerals
22	**Economic Analysis**	Kristal Font Inc. (Alex Duggan)		GB Minerals
23	**Adjacent Properties**	Lycopodium (Dan Markovic)	GB Minerals	
24	**Other Relevant Data and Information**	Lycopodium (Dan Markovic)		GB Minerals
25	**Interpretation and Conclusions**	Lycopodium (Dan Markovic)/ Golder (Ted Minnes) /Knight-Piésold (David Morgan, Richard Cook)/Kemworks (Dr. Francisco Sotillo)/Baird (Ed Liegel),/Kristal Font Inc. (Alex Duggan)		GB Minerals
26	**Recommendations**	Lycopodium (Dan Markovic)/ Golder (Ted Minnes) /Knight-Piésold (David Morgan, Richard Cook)/Kemworks (Dr. Francisco Sotillo)/Baird (Ed Liegel)/Kristal Font Inc. (Alex Duggan)		GB Minerals
27	**References**	Lycopodium (Dan Markovic)		GB Minerals

The Qualified Persons listed below have contributed to the Technical Report as specified.

- Dan Markovic of Lycopodium for process plant and port infrastructure, operating costs and study coordination. Dan visited the property on October 5 through 8, 2014 and July 11, 2015.

- Dr. Francisco Sotillo of KEMWorks for metallurgical test work and process design. Francisco did not visit the site (not required).

- Jerry DeWolfe of Golder completed the mineral resource estimation and data verification, and is responsible for the geology and exploration contribution. Jerry visited the site on April 5 through 8, 2015.

- Ted Minnes of Golder for reserve estimation, mining, mine capital and operating costs. Ted visited the site on April 5 through 8, 2015.

- George Lightwood of Golder for pit slope design. George did not visit the site (not required).
- David Morgan of Knight-Piésold for soil geotechnical, integrated waste landform design (tailings and overburden facility), hydrology, hydrogeology, environmental geochemistry, and site water management. David visited the site on February 21 and 22, 2015.
- Ed Liegel of Baird for marine infrastructure, marine vessels, marine Capital and Operating Costs. Ed did not visit the site,however a Baird representative visited the site in 2012.
- Alex Duggan (Kristal Font Incorporated) for Capital Cost Estimate and Economic Analysis. Alex did not visit the site (not required).
- Richard Cook of Knight-Piésold for Environmental Studies, Permitting and Social/Community Impact. Richard visited the site on March 25 and 26, 2015.

3.0 RELIANCE ON OTHER EXPERTS

In preparing this report, Lycopodium has relied on input from GB Minerals and a number of well-qualified independent consulting groups.

Lycopodium is not an expert in legal matters, such as the assessment of the legal validity of mining concessions, private lands, mineral rights, and property agreements. Although Lycopodium has reviewed available data and visited the site, these activities serve to validate only a portion of the entire set of data. Therefore, Lycopodium has made judgements about the general reliability of the underlying data; where deemed either inadequate or unreliable, the data were either not used or procedures were modified to account for the lack of confidence with that specific information.

The various agreements under which GB Minerals holds title to the mineral claims for this project have not been reviewed by Lycopodium, and Lycopodium offers no legal opinion as to the validity of the mineral title claimed. A description of the property, and ownership thereof, is provided for general information purposes only.

The information provided to GB Minerals was supplied by reputable companies or government agencies and Lycopodium has no reason to doubt its validity. Lycopodium has relied on GB Minerals for current legal title of the concessions. Section 4 is based on information provided by GB Minerals, and the authors offer no professional opinions regarding the provided information.

Lycopodium has relied on GB Minerals and their financial staff and advisors to determine appropriate tax implications in the financial analysis for this Technical Report. Lycopodium is not an expert on Guinea-Bissau tax issues.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

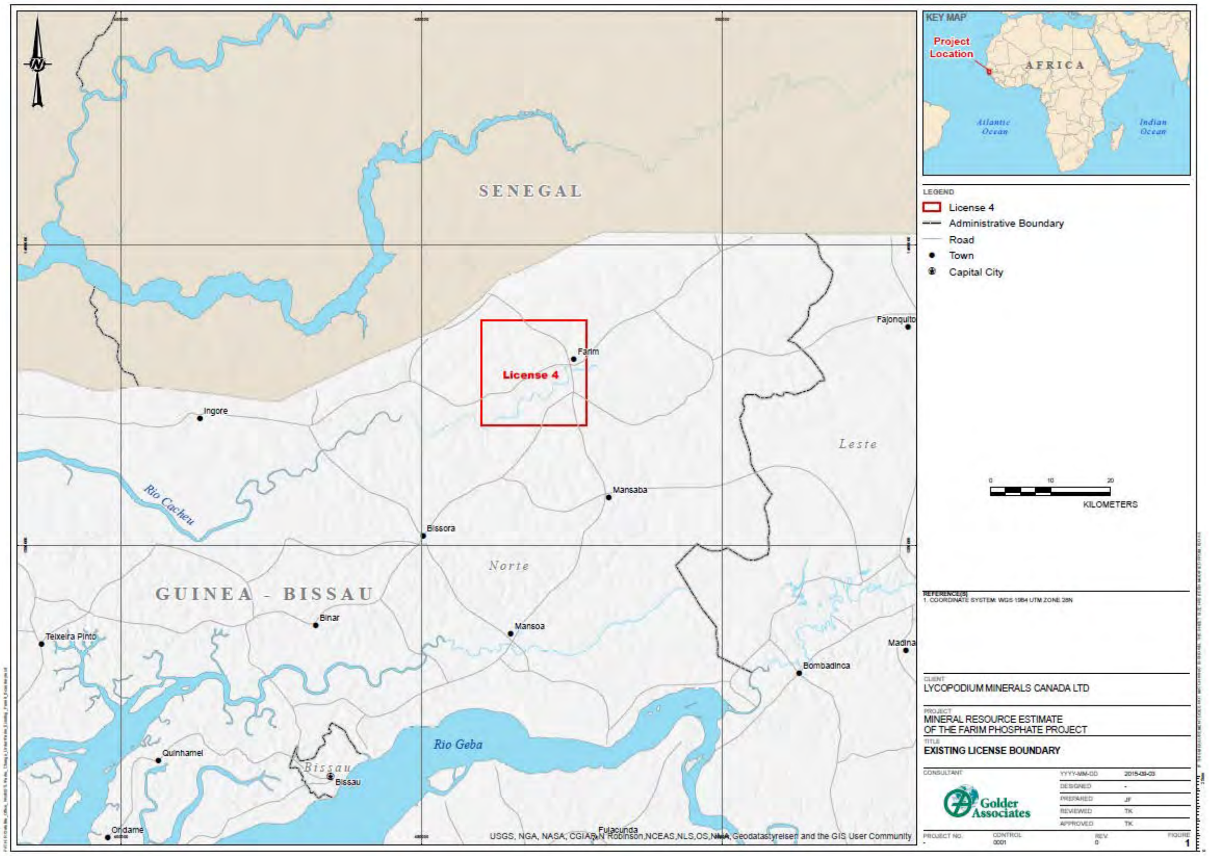
The Farim Project is located in the northern part of central Guinea-Bissau, West Africa, approximately 25 kilometres south of the Senegal border, approximately 5 kilometres west of the town of Farim and some 120 kilometres northeast of Bissau, the capital of Guinea-Bissau. The Farim Project lies within mining lease license No. 004/2009 ("**Mining Lease 004/2009**"), covering 30,625 hectares, granted by the Government of Guinea-Bissau to GB Minerals AG ("**GBMAG**"), a wholly owned subsidiary of GB Minerals registered in Switzerland, on May 28, 2009. The following are the co-ordinates of Mining Lease 004/2,009 ("License Area"):

Table 4-1 Border limits of Mining Lease License 004/2009 (UTM Coordinates)

	Northing	Easting
Point 1	1,387,500	460,000
Point 2	1,387,500	477,500
Point 3	1,370,000	477,500
Point 4	1,370,000	460,000

The License Area is shown in the following map:

Figure 4-1 Location of the Farim Phosphate Project



4.2 Ownership, Title, Licensing and Permitting

Mining Lease 004/2009 was granted by the Government of Guinea-Bissau to GBMAG for the exploration and extraction of mining substances within the License Area with the objective of commercializing them. The exclusive right of GBMAG to perform mining operations within the License Area is subject to the payment of an annual license fee to the Government of Guinea-Bissau and to reporting requirements.

In addition to Mining Lease 004/2009, GB Minerals AG was granted on May 28, 2009, a mining license, Mining License No. 001/2009 ("**Mining License 001/2009**"), for a period of 25 years, giving it the exclusive right to; (i) execute its mining operations within the License Area; (ii) erect the equipment, installations and buildings necessary for the extraction, transportation and treatment of minerals; (iii) commercialize the minerals, inside or outside the national territory; (iv) undertake prospecting activities; and (v) store or discharge any mining product or waste.

GB Minerals AG and the Government of Guinea-Bissau also signed, on May 28, 2009, a mining agreement ("**Mining Agreement**") that governs the execution of Mining Lease 004/2009 and Mining License 001/2009 and clarifies the framework applicable to the rights granted to GB Minerals AG for the development of the Farim Project. The Mining Agreement is valid for 25 years and is automatically renewed upon the renewal of Mining Lease 004/2009 and Mining License 001/2009. The Mining Agreement also allows GB Minerals to build roads, buildings, port or other infrastructures required in connection with the project without being subject to taxes, license fees or other costs both within and outside the License Area.

4.3 Incentives Annex, Royalties and Other Financial Agreements

Pursuant to the Mining Agreement, the Government of Guinea-Bissau will be entitled to, for the duration of the commercial mining operations at the Farim Project, a 2% royalty that will be tax deductible.

The Mining Agreement comprises an “Incentives Annex” defining the financial terms associated with the Farim Project, and providing to GBMAG certain guarantees and financial incentives. The terms of the Incentives Annex to the Mining Agreement have been fully negotiated with, and as of the date of this Report remain subject to final approval of, the Government of Guinea-Bissau.

4.4 Environmental Regulatory Framework

The Farim Project’s concession and port areas consist of both virgin land and farmland. All mining activities will be conducted in accordance with local legislation and internationally recognized standards.

Guinea-Bissau has developed a legal framework on the environment which lays the foundation for environmental policy and environmental assessments as better described herein. Law No. 1/2011 of March 2, 2011 established the Basic Legislation on the Environment by defining the basic concepts, norms and principles related to the protection, preservation and conservation of the environment, and aims to improve quality of life through the management and rational use of natural resources, to achieve the sustainable use of such resources.

Law No 10/2010 of September 24, 2010 - or the “Environmental Assessment Law” - regulates environmental and social impact assessment (“ESIA”) in Guinea-Bissau, which has been identified as the fundamental preventative tool of environmental policy and is an important component of the Government’s overarching policy of sustainable development. The Environmental Assessment Law sets out the types of projects for which an ESIA is required and the categorisation of these projects, in line with the categories established by the International Finance Corporation (“IFC”). Due to the size of the operation, the Farim Project has been classified as a Category A project and, as such, requires the delivery of a full ESIA.

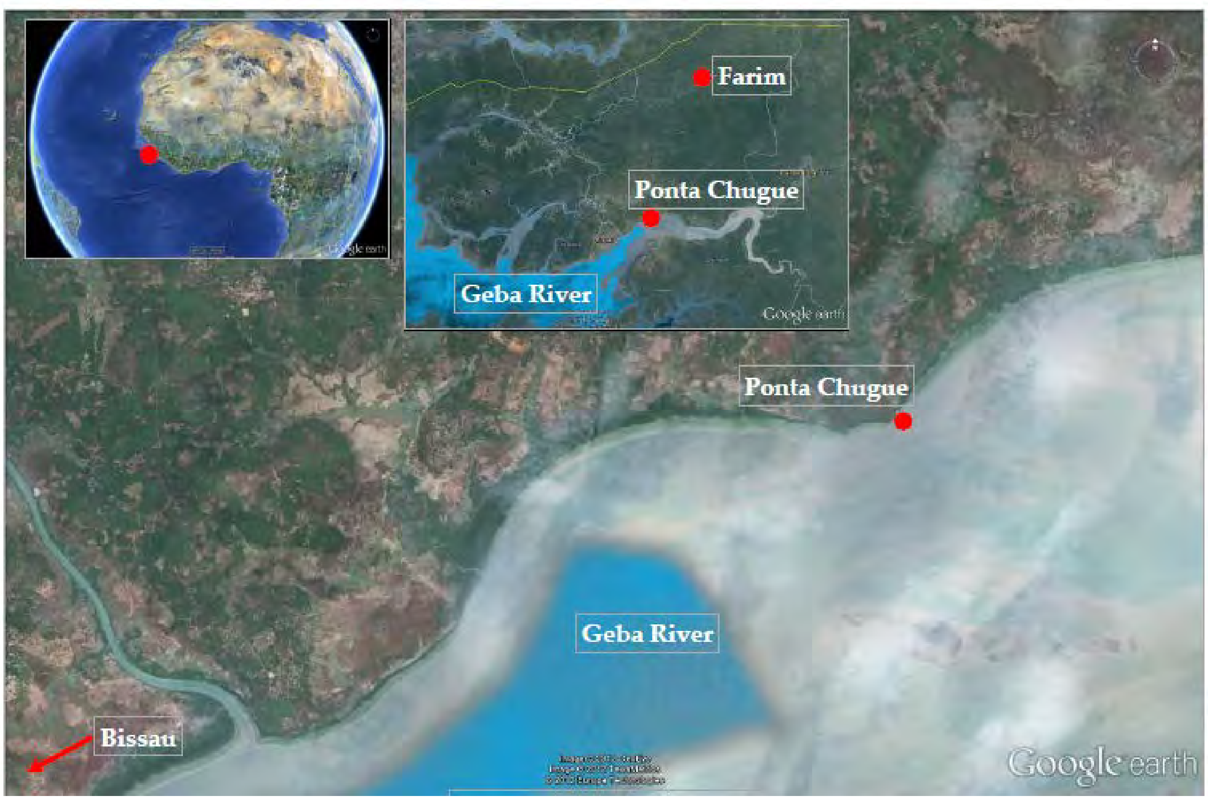
5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access to Property

The Farim property is located in the northern part of central Guinea-Bissau, West Africa, approximately 25 km south of the Senegal border, approximately 5 km west of the town of Farim and 120 km NE of Bissau, the capital of Guinea-Bissau. The property is accessible via 120 km of paved highway northeast of Bissau. A ferry provides access to the town of Farim, located on the north bank of the Cacheu River. The Cacheu River at the ferry crossing is approximately 300 m wide. From the town of Farim, the property can be accessed via a 5 km unpaved dirt road.

The port location, Ponta Chugue, in the Geba River estuary, is approximately 18 km east of Bissau, and approximately 75 km south of Farim. Beneficiated phosphate rock will be trucked from Farim to Ponta Chugue via a newly constructed highway in excellent condition. The phosphate rock will be dried, stored, and direct loaded onto 35,000 DWT ships. See Figure 5-1 for Farim and Ponta Chugue locations.

Figure 5-1 Farim and Ponta Chugue locations



5.2 Physiography

The Project area is flat, and the Farim mine site, where phosphate is mined, varies from approximately 5 to 10 mamsl (meters above mean sea level). The project area drains into the Cacheu River. The area where the process plant is located is approximately 4 to 5 mamsl. See Figure 5-2 for a typical view of the plant site.

Figure 5-2 Process Plant Site at Farim

The region is open, semi-arid savannah woodland.

A conveyor will cross the Cacheu River to transport phosphate rock from the process plant on the western side to the loading bin on the eastern side. At this crossing, the river is at its narrowest point, approximately 150 m wide. The river crossing is shown in Figure 5-3.

Figure 5-3 Conveyor Crossing Location at Cacheu River

The drying and storage facility at Ponta Chugue is approximately 2 mamsl. See Figure 5-4 and Figure 5-5 for a typical view of the port site at Ponta Chugue.

Figure 5-4 Typical View of Port Site at Ponta Chugue

Figure 5-5 Bay at Ponta Chugue

5.3 Local Infrastructure and Resources

The local economy is based on agriculture, cashew nuts, and fishing. The sustainable nature of these industries has contributed to a stable population. The local infrastructure is primitive. The largest town in the vicinity, Farim, has an approximate population of 7,000 people. The area surrounding the port site at Ponta Chugue is agricultural and sparsely populated the nearest village is Chugue and hosts approximately 100 people. The capital city, Bissau, has a population of 407,000 people, is approximately 18km from Ponta Chugue and accessible by paved road.

There are no operating mines in Guinea-Bissau and very little heavy industry. Labour will be sourced from local communities where possible, at both Farim and Ponta Chugue, and trained in the skills required. Since these local communities are focused on agriculture, it is anticipated that a portion of the labour force will need to be sourced from expatriate personnel from neighbouring countries.

Water is available from wells. There is no local power supply for both Farim and Ponta Chugue. Power required for the Farim Phosphate Project will be provided by diesel generating sets at both locations. All working areas of the Project will be accessible by well maintained dual lane gravel roads.

The town of Farim has limited infrastructure that is suitable to a mining operation of the scale proposed for the Project. It will be necessary to construct housing, medical and associated infrastructure to accommodate the impact on the town of Farim. At Ponta Chugue, the staffing requirements are significantly smaller and workers will most likely reside the capital of Bissau or nearby villages.

5.4 Climate

The climate is tropical with a mean annual temperature of 25°C. At the Farim climate station, the maximum temperature recorded from December 2011 to March 2015 was 42.8^{o}C. The minimum temperature recorded during the same time period was 8.1^{o}C. The rainy season occurs from June to October and is most intense in July, August and September. Average annual rainfall is 1,950 mm in Bissau and about 1,143 mm in the area of the deposit.

The average monthly relative humidity ranges from 92% in August to 49% in February.

5.5 Regional Seismicity

A literature review of the seismicity of Guinea-Bissau and West Africa, and probabilistic and deterministic seismic hazard assessments have been carried out for the Project. Available information and historical data, including earthquake catalogues and technical publications on the tectonics and seismicity have been reviewed.

In accordance with the International Building Code (IBC) for structural design, the maximum considered earthquake ground motion has been defined as the ground motion with a 2% probability of exceedance in 50 years. Specifically, seismic parameters for use with IBC are provided below for the site:

- Seismic coefficient, SS = 0.15g
- Seismic coefficient, S1 = 0.04g
- Peak ground acceleration = 0.06g

6.0 HISTORY

Phosphate was first discovered in one geotechnical drill hole as part of a water survey in 1950 and noted again in one oil drill hole drilled by Esso in 1965.

The French Bureau of Geological and Mining Research (BRGM) conducted an extensive exploration and delineation drilling program from 1981 to 1983, during which time they drilled 5 672 m of large diameter core in 101 holes. This enabled them to carry out a detailed geological study of the deposit and provided a comprehensive database for the French agency Sofremines to conduct a prefeasibility study in 1986. The prefeasibility study was positive but market conditions and political considerations precluded development at that time and the French agencies withdrew from the Project.

The "geological resource" of the area was estimated using polygonal methods to be 113 million tonnes of phosphate matrix at 30 % P_2O_5. The "geological resource" was calculated from the geological resource by application of loss and dilution factors. The extraction of the phosphate matrix was estimated to result in a loss of 10 % and dilution of 10 % from the barren hanging wall and lower grade of phosphate in the footwall. The tonnage of run-of-mine phosphate was taken to remain the same but its grade reduced to 29.75 % P_2O_5. Economic estimation yielded an historic resource of 68 Mt at a cut-off stripping ratio of 15:1 and 37 Mt at a cut-off stripping ratio of 10:1, both at an average grade to beneficiation of 29.75 % P_2O_5. This Historic Resource estimate by BRGM has not been reviewed by the Qualified Person, and should not be considered to represent a current resource and as such should not be relied upon.

In 1997 a Canadian exploration company, Champion Resources Inc., acquired ownership of the Farim phosphate deposit and carried out diamond drilling campaigns in 1998 and 1999 totalling 1810 m in 34 holes. The main purposes of the Champion drilling were to confirm the BRGM results or due diligence purposes, to provide fresh phosphate samples for metallurgical test work, and to expand the phosphate resource to the west. The programmes were successful in that all three goals were achieved. Champion Resources Inc carried out a prefeasibility study which again demonstrated that a phosphate mining and beneficiation project was economically viable. However, no further progress was made due to external market conditions and phosphate rock prices.

The Champion historic resource was modelled and estimated using Medsystem software with the Inverse Distance to the power of 2 (ID2) algorithm. The historic resource was 166 Mt at 29 % P_2O_5 , 10.5 % Fe+Al and 10 % SiO_2 of which the Measured Resource was 53 Mt at 29.8 % P_2O_5 and the Indicated Resource was 113 Mt at 28.7 % P_2O_5 . The resource calculations were conducted according to USGS Circular 882 but with a modified projection radius of 500 m for a Measured Resource. Again this Historic Resource Estimate by Champion has not been reviewed by the Qualified Person, and again should not be considered to represent a current resource and should not be relied upon. A summary of historical Resources is provided in Table 6-1.

Table 6-1 Comparison of Historical Resources for the Farim Deposit

Company	Stripping ratio cut-off	Deposit area (km^2)	Resource (10^6 tonnes)	Average thickness (metres)	P_2O_5 (%)
BRGM	None	24.5	113	3.29	30
CHAMPION	None	38	166	3.15	29

In 2006, GB Phosphate Mining Ltd. was granted by the Government of Guinea-Bissau (GoGB) mineral rights over the Farim phosphate deposit and evaluated its potential. They undertook several comprehensive studies including excavating a box cut drilling 30 holes to confirm and validate the work of previous explorers, a hydrological study, an environmental impact study and an economic evaluation of the Project. In 2009, GoGB granted to GB Minerals AG (GBMAG), now a wholly-owned subsidiary of GB Minerals Ltd., Mining Lease 004/2009 and Production License 001/2009, both covering 30,625 ha (Concession Area). GoGB and GBMAG also entered into a mining agreement to govern the execution of Mining Lease 004/2009 and Production License 001/2009 and clarify the framework applicable to the development of the Project.

GB Minerals Ltd. was originally incorporated under the British Columbia Business Corporations Act in 2007 under the name of Resource Hunter Capital Corp. (RHC). In 2011, RHC was acquired by Plains Creek Mining (PCM) in a reverse take-over and changed its name to Plains Creek Phosphate Corp. (PCP). Concurrent with closing of the reverse take-over, PCM changed its name to GB Minerals Holdings Ltd. (GBM Holdings), and completed a transaction leaving it with 50.1% ownership of GBMAG, which held 100% of the ownership of the Farim project. In 2013, PCP changed its name to GB Minerals Ltd., trading under the symbol “GBL” and GBM Holdings acquired the remaining 49.9% of the ownership of GBMAG. GB Minerals Ltd. currently owns 100% of GBMAG, which owns 100% of the Project.

On February 22, 2011, RHC filed a NI 43-101 technical report entitled “Technical Report on the Preliminary Economic Assessment of the Farim Phosphate Project, Guinea-Bissau” prepared by IMC Group Consulting Limited with an effective date of February 10, 2011.

On September 13, 2012, GB Minerals Ltd. filed a NI 43-101 technical report entitled “Technical Report on the Preliminary Economic Assessment of the Direct Shipping Option of the Farim Phosphate Project, Guinea-Bissau” prepared by GBM Minerals Engineering Consultants Limited (GBMMEC) and Golder Associates (U.K.) Ltd. (Golder) and dated effective September 5, 2012 on the System for Electronic Document Analysis and Retrieval (SEDAR).

On January 17, 2013 GB Minerals Ltd. filed a NI 43-101 technical report for the feasibility study on the Farim Project entitled “Feasibility of the Beneficiated Phosphate Rock Concentrate of the Farim Phosphate Project, Guinea-Bissau”, dated effective December 19, 2012.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

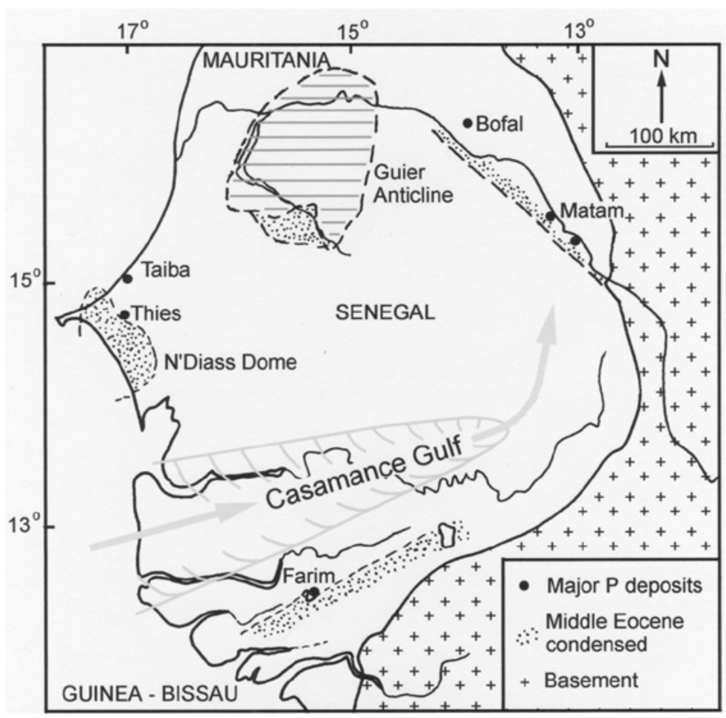
7.1 Regional Geology

The Farim phosphate deposit is located within the Middle Eocene Lutetian Formation that forms part of the southern margin of the Mauritania-Senegal-Guinea Cenozoic sedimentary basin (Prian, 1987). The basin extends from Morocco in the north through Mauritania, Senegal, Guinea-Bissau and into Guinea to the south. The Mid-Eocene and particularly the Lutetian of the basin contains known phosphate horizons and hosts a number of important economic phosphate deposits including Bofal in Mauritania and Taïba, Thiès and Matam in Senegal. It accounts for almost 25% of current world rock phosphate production.

The sediments of this basin were formed in the palaeo-gulf of Casamance, which extended from the southeast of Mauritania in a generally southwesterly direction into what is now the Atlantic Ocean.

The regional geology and setting of Farim is shown on Figure 7-1.

Figure 7-1 Regional Geology and Setting of Farim



7.2 Local Geology

The Farim area forms part of the southern margin of the former Casamance Gulf and is located 60 km northwest of the southern edge of the Senegal-Mauritania-Guinea sedimentary basin in which the Maastrichtian strata unconformably overlies the Devonian pelite sequence (Prian, 1987). The various

Mesozoic and Cenozoic formations become thinner and wedge out progressively from northwest to southeast towards the Devonian bedrock. Abrupt condensing and wedging out of the Eocene sedimentary units occurs in the Farim area around an elevated structure known as the Rio Jumbembem ridge, which gives way south-westwards to the Binta high. The high, rectilinear Rio Jumbembem ridge strikes 050 to 060° and is positioned over a basement flexure. Immediately to the southwest of Farim, between the high points of Rio Jumbembem and Binta, is the smaller Saliquinhé bay, 3 km wide from northwest to southeast and 5 km long from southwest so northeast, open to the northeast and closed to the southwest. A subsidence zone at the southeast edge of the Casamance Gulf lies to the northwest of this zone of highs, which is marked by sequential condensing and frequent wedging out of the various Palaeocene and Eocene sedimentary units.

The late Palaeocene occupies an elevated position and forms the greater part of the Rio Jumbembem ridge, in which it is composed of nummulitic limestone, becoming argillaceous and marly towards the Palaeocene subsidence zone to the northwest.

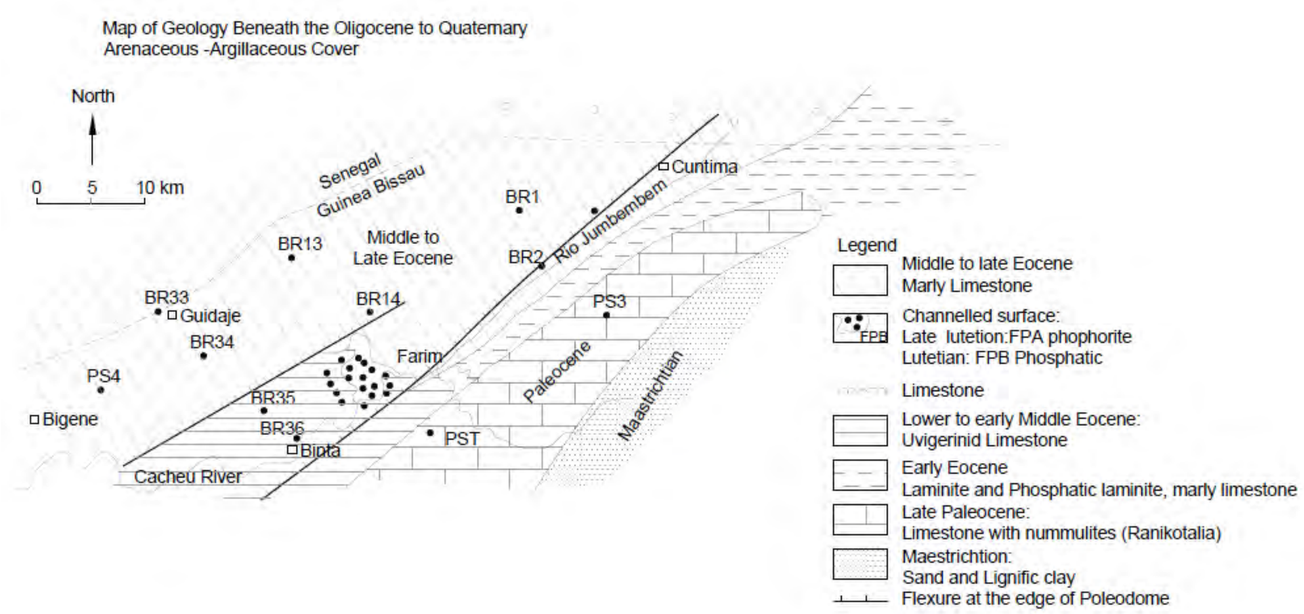
The Eocene is condensed and/or reduced over elevated zones. Boreholes located on the Rio Jumbembem high have all the lithologic units of the lower to upper Eocene present, but extremely condensed (39 m). The thickness of these units in the subsidence zone is over 70 m.

Abrupt, sequential condensing occurs in the Farim area near the phosphate deposit. This is particularly evident in the calcareous and phosphatic sequence. Only the lower to basal middle Eocene, composed of argillaceous and micritic laminite, is present in the elevated zone. The calcareous-phosphatic middle Eocene and the calcareous-dolomitic upper Eocene are notably absent the Binta high. The middle and upper Eocene are, however, well developed to the north of the high.

Throughout this area of the Senegal-Guinea sedimentary basin, the Eocene, Palaeocene and Maastrichtian are respectively unconformably overlain southeastwards by an Oligo-Mio-Pliocene and Quaternary sandy argillaceous sequence displaying black lignitic clay at the base. This is locally overlain by a greensand sequence, probably Miocene in age, containing thin limestone beds. These units underlie a sandy-argillaceous sequence assigned to the late Continental. The thickness of post Eocene sandy-argillaceous cover ranges from 15 m to 35 m in the Farim area and from 50 m to 64 m in the basin subsidence zone.

The local geology beneath the overburden is shown in Figure 7-2.

Figure 7-2 Local Geology Beneath the Overburden (Reproduced by Golder from Prian, 1987)



7.3 Property Geology

7.3.1 Stratigraphy

The Farim phosphate deposit is a flat-lying sedimentary phosphatic bed, which underlies an area in excess of 60 km^2.

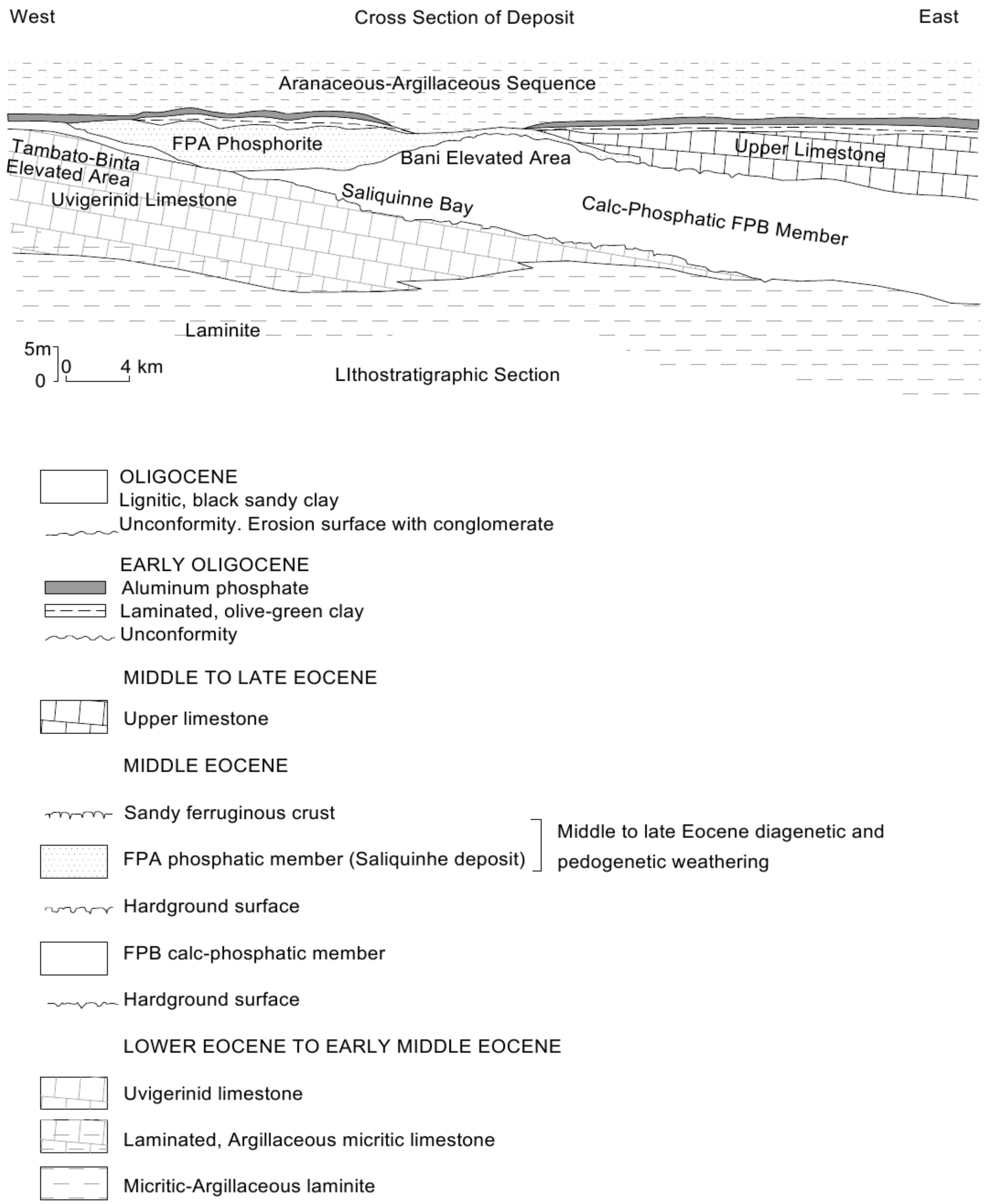
The geological sequence at Farim displays the following lithological units from top to bottom:

Sandy-argillaceous overburden with soft, alternating sandy, clayey and sandy-clayey layers;

- Phosphatic interval (FPO);
- Upper dolomitic limestone;
- Decarbonised phosphate unit (FPA) corresponding to the Saliquinhé phosphate deposit;
- Calcareous phosphate member (FPB); and
- Limestone at the footwall of the phosphate sequence, white, soft and porous.

Figure 7-3 shows a typical cross section of the Farim deposit together with a lithostratigraphic column (Prian, 1989).

Figure 7-3 A Typical Cross Section of the Farim Deposit with a Lithostratigraphic Column (Reproduced from Prian, 1989)



7.4 Deposit Geology and Mineralization

The three phosphate bearing horizons referred to as FPO, FPB and FPA are described below and are located below a variable thickness of overburden.

7.5 Overburden

The overburden waste at Farim typically consists of a layer of reddish brown laterite gravel, followed by cream coloured clay with occasional cobbles and boulders of cemented orange sand and brown clay. This is followed by a layer of stiff brown to orange sandy clay and a layer of firm light grey, moist, high plasticity clay of a similar thickness. No laboratory test results are currently available for these materials.

The thickness of the overburden layers range from less than 20 m to over 40 m in the mining areas, whereas the phosphate matrix layer which is also a sedimentary deposit ranges from less than 2 m in thickness to over 5 m thick in places. Below these two layers is a soft rock limestone layer which increases quickly with depth to medium and hard bedrock.

7.5.1 FPO

The FPO is a clayey dolomitic limestone that is weakly phosphatic and has limited economic potential. It comprises laminated green clays and aluminophosphate and is 0.5 m to 1 m thick. At the surface in the higher zones, laterite with a ferruginous cover in places may be found.

7.5.2 FPA

The FPA phosphate matrix is homogenous and has a grainstone texture, with grains less than 800 µm in size. It is a soft, poorly cemented unit of phosphatic sand, which includes phosphatised shell and bone material, teeth, faecal pellets and crustacean coprolites. There is no calcareous cement and it contains little silica and clay. It is mildly indurated and includes siliceous or pyritised layers 5 to 20 cm thick which comprise an average of 6% of the unit. The FPA layer has a P_2O_5 content of approximately 30% (consistently higher than 25%). The FPA unit is currently considered the potentially economic phosphate horizon. Grades of sedimentary phosphate deposits of worldwide distribution as compiled by IMC (2011) are in the range of 15 to 32%. The Farim deposit is at the higher end of that range (Champion, 2000).

The FPA is localised within the Saliquinhé bay sub-basin and is the potentially economic phosphate bed. The sub-basin is bounded to the south and east by carbonate platform rocks against which the FPA wedges out. The north-western limit of the FPA has not yet been defined. To the north, the Tambato submarine bar, which formed a barrier between the Saliquinhé bay and the deeper Casamance basin, will likely form the northern limited of the FPA unit but this has not been demonstrated by drilling.

The limits of the FPA unit, the hanging and the foot walls, are clearly defined. A mixture of saprolitic fine sand and clays, which are generally unconsolidated, overlies the FPA. The immediate hanging wall to the FPA is 20 to 60 cm thick unconsolidated sand. The hanging wall rocks are oxidised reddish brown to an elevation of about 10 m below sea level. The FPA is grey to beige and brown and lies in a generally reducing environment below the oxidised interval. This is important because iron oxide, which is soluble in sulphuric acid, is a contaminant in phosphate deposits whereas iron sulphide, which is insoluble in sulphuric acid, is not (Champion, 2000).

The FPA unit has an average width of about 3 m (in the resource area) and underlies an area of about 60 km^2. In the northern part of the basin, north of the village of Saliquinhé, a northeasterly trending area about 5.5 km long and 1.5 km wide has FPA thickness typically greater than 3.0 m and up to 6.0 m. A smaller area to the south of Saliquinhé, near the Cacheu River, also exceeds 3.0 m in thickness.

The FPA is very regular, sub-horizontal and continuous. Given this continuity, there is no requirement for geological control by % P_2O_5 cut-off. Both SR and FPA thickness have been used to define the *in situ* resources.

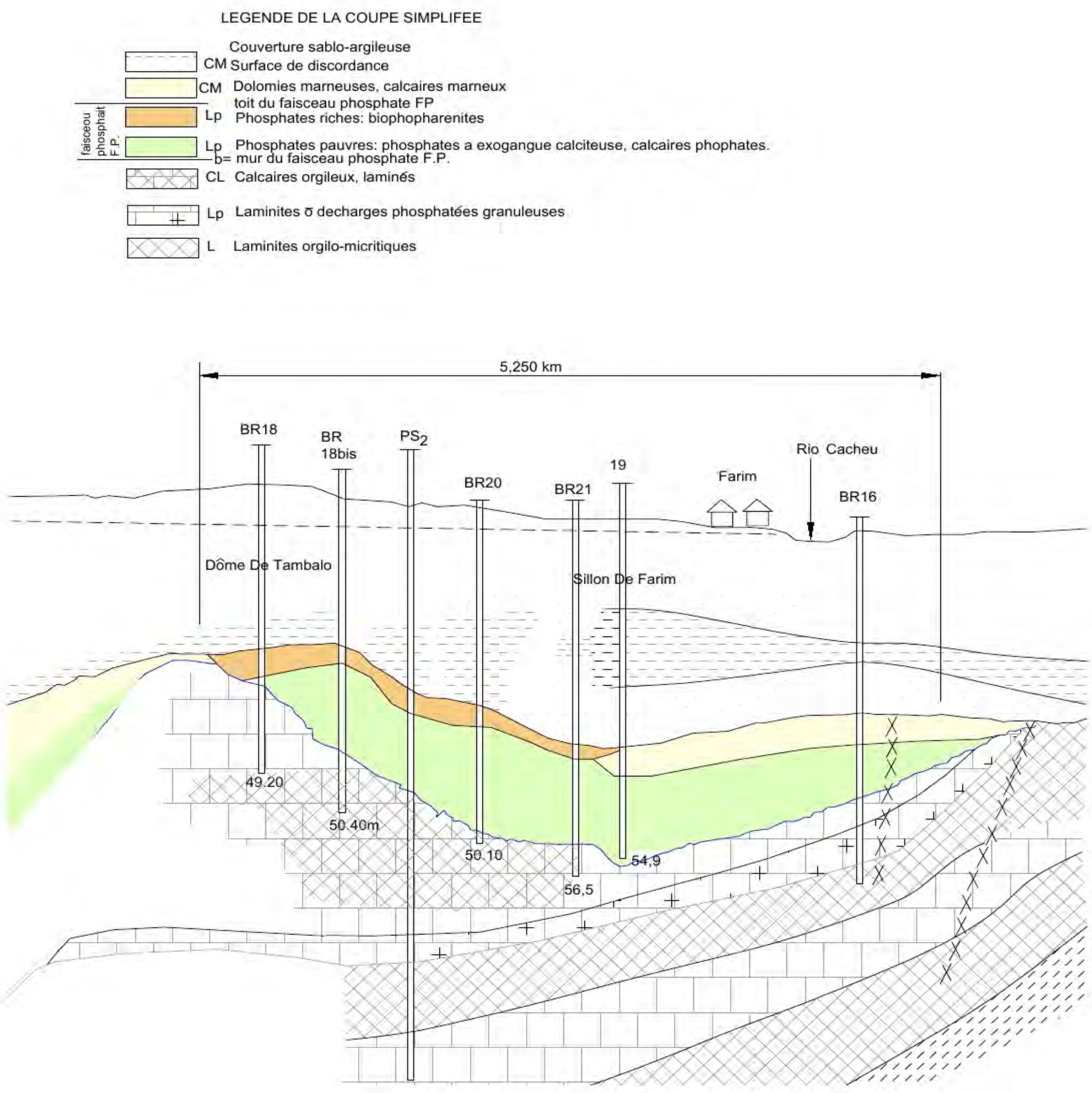
7.5.3 FPB

The FPB is a calcareous phosphate unit consisting of alternating soft phosphate strata with carbonaceous gangue and thinner, hard strata of slightly phosphatic bioclastic limestone. The lower grade FPB layer consists of highly carbonated phosphate, generally containing 5% to 20% P_2O_5 with an average of 13% P_2O_5. The FPB phosphatic limestone is indurated and much harder than FPA.

FPB is located immediately below FPA, but exists under only 50% of the area of FPA. FPB also has a large extent outside of FPA. This horizon is known to extend 20 km north to south and 50 km east to west with thickness variable from 1 to 15 m with an average thickness of approximately 5.3 m (Figure 7-4).

FPB is considered to be of marginal economic potential and is not included in this resource estimate.

Figure 7-4 Representative Cross Section through Farim Deposit



7.5.4 Hydrogeology

The hydrogeological conditions of the Farim area can be summarized as:

- An upper aquifer in the overburden formations (that predominantly comprises gravels, sands and clays);
- An intermediate aquitard, comprising the blue grey clay at the base of the overburden and where present potentially the FPB layer; and

- A lower aquifer, which corresponds to the micritic limestones and the FPA phosphate-bearing layer.

The groundwater elevations recorded in both the overburden and the underlying geology indicate that groundwater flows from the northwest, where groundwater elevations are highest, towards the Cacheu River in the southeast. The groundwater elevations recorded between August 2009 and February 2012 ranged between -1.13 mamsl and 4.01 mamsl in the lower aquifer, and between -0.81 mamsl and 4.46 mamsl in the overburden aquifer.

Groundwater elevations increase in the wet season in comparison to the dry season, with the groundwater elevations observed within the overburden showing a larger rise in water levels than those in the underlying geology. Comparison of the groundwater elevations recorded in the paired boreholes installed in 2011, indicates that there is an element of vertical flow downwards from the overburden to the limestone to the northwest of the proposed Open Pit Area, while nearer the Cacheu River the overburden receives upward flow from the limestone. The vertical flow direction is indicated to change seasonally, in the one borehole that was installed during the dry season (MW04). The lateral variation of this seasonal change is not known at this stage.

The field data collected indicates that the groundwater in the lower aquifer is slightly less acidic and has a higher electrical conductivity than the groundwater in the shallower overburden boreholes. The deep boreholes located closest to the River Cacheu (MW01 and MW02), have a higher electrical conductivity and pH than those located away from the river. From the laboratory analysis of fifteen groundwater samples a good groundwater quality is indicated. Only the iron content (total and dissolved) and manganese content (dissolved) of the groundwater was reported above the WHO guideline value. The chloride and sodium concentrations reported are low, indicating a freshwater source.

Several pumping tests have been carried out historically to determine the transmissivity and storativity of the aquifers. For the overburden aquifer a transmissivity range of between 1.6×10^{-4} and 2.5×10^{-3} m^2/s and a storativity range of between 1×10^{-5} and 1×10^{-3} are reported. For the limestone aquifer a transmissivity range of between 4×10^{-5} and 7×10^{-4} m^2/s and a storativity range of between 2×10^{-4} and 4×10^{-4} are reported.

Two long-term pumping tests were undertaken in 2011 and 2012 (one in the northern part of the Open Pit Area and one in the southern part of the Open Pit Area). The analysis of the pumping test data indicates that:

- The lower limestone and phosphate aquifer generally has a slightly higher transmissivity (ranging between 4.0 x 10-4 and 3.3 x 10-3 m^2/s) than the upper overburden aquifer (ranging between 3.1 x 10-5 and 1.8 x 10-3 m^2/s); and

- The results indicate that both the lower and upper aquifers are slightly less permeable in the southern part of the proposed Open Pit Area (close to the Cacheu River), compared to the northern part of the proposed Open Pit Area.

8.0 DEPOSIT TYPES

In the Farim phosphate property, two main types of phosphate have been identified, differentiated by their petrography and chemical composition:

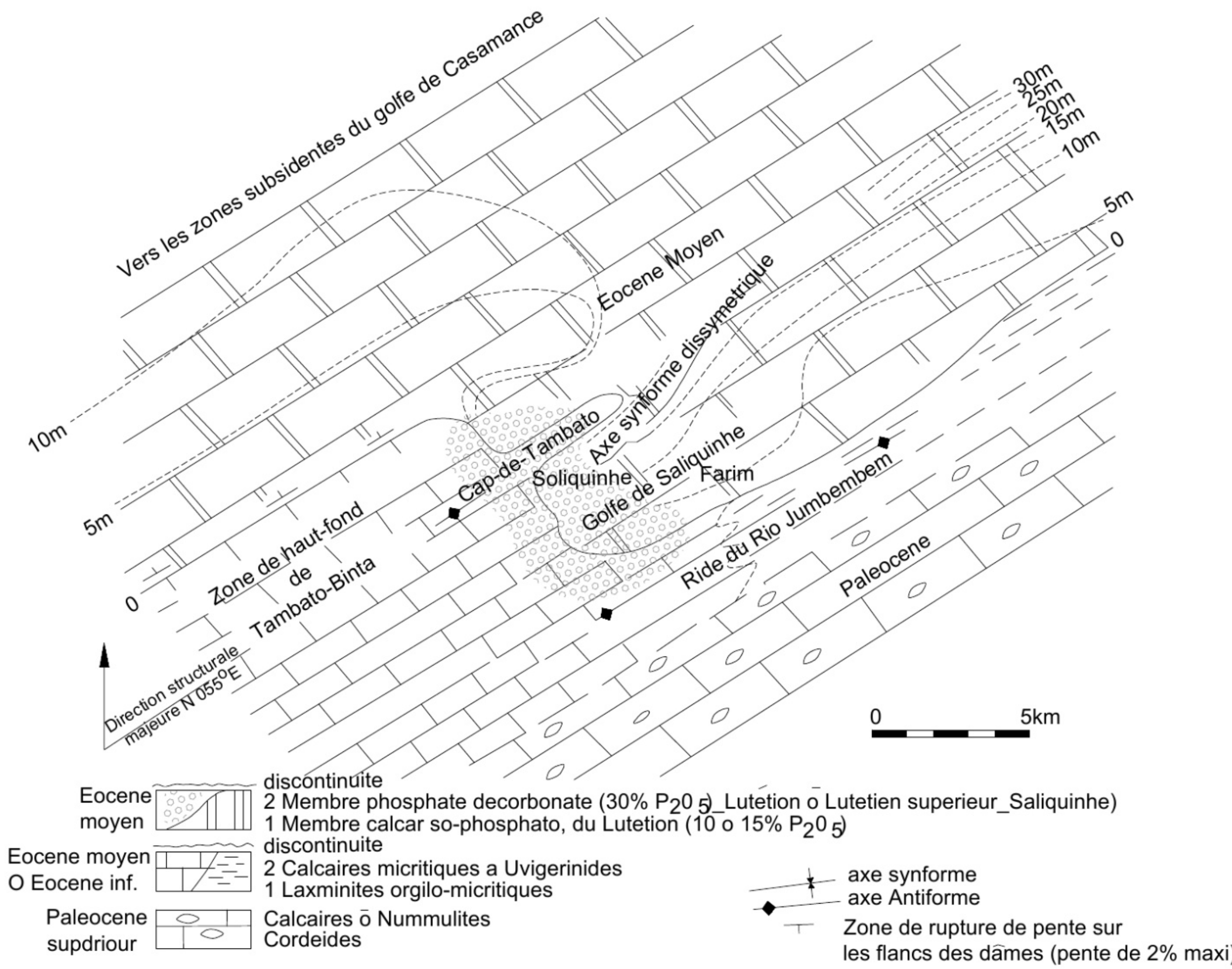
- FPA layer, a de-carbonated phosphate matrix with very high P_2O_5 content of about 30% P_2O_5, formed exclusively in the shallow water of the Saliquinhé basin; and

- The lower grade FPB layer of highly carbonated phosphate, generally containing 5 to 15% P_2O_5 (average 13% P_2O_5) with some values up to 20%.

The phosphate of Farim was formed in an infra-littoral maritime environment, in the gulf of Saliquinhé which opens on to the ocean. The first phosphate deposit, FPB, was thick at the entry of the gulf and formed a bar (the “bar of Bani”) which slowed down the water exchange with the ocean. The phosphate deposited in the shallow water of Saliquinhé was thus trapped. The interaction between the two bodies of water supported the de-carbonation and enrichment of phosphate in the upper layers of FPB, thus differentiating the high grade FPA deposit.

The isobaths of the micritic limestone hanging wall shows a palaeostructure in the bottom of the gulf that is open to the northeast and encircled to the southwest by low water level areas. The phosphate horizons are transgressive on the micritic limestone. FPA lies just above FPB or above the limestone when FPB is absent (suggesting early erosion of FPB). For FPA, the “bar of Bani” at least partly prevented this phenomenon. However, agitation by shallow marine water altered the deposit and formed the phosphate grains, destroying the carbonates (cement and crystals) and leaving the FPA with a structure consisting almost exclusively of phosphate with only minor detrital quartz and a little clay binder remaining. The upper part of FPA is a level of aluminophosphate (crandallite) with strong indurations that has a thickness of 100 to 500 mm.

Figure 8-1 Palaeogeography of the Regional Farim Area at the End of the Eocene



9.0 EXPLORATION

9.1 Historical Exploration

Phosphate was first discovered in one geotechnical drill hole as part of a water survey in 1950 and noted again in one oil drill hole drilled by Esso in 1965. The Directorate of Geology and Mines of Guinea-Bissau (DGMGB) commenced initial exploration of the Farim area in 1973, funded by the United Nations Development Program. They drilled seven holes between 1977 and 1979. These findings were reported in 1980 and showed the presence of the Eocene phosphate similar to the sedimentary deposits of Bofal in Mauritania and Taïba and Matam in Senegal under Miocene-Pliocene cover. One drill hole intersected 4.9 m of phosphate at 25% P_2O_5 under 40 m of sand-clay overburden.

9.1.1 BRGM

Exploration, geological investigation and reserve assessments of the Farim phosphate deposit were conducted during the following three Bureau de Recherche Géologiques et Minières (BRGM) campaigns, which provided extensive information including respective sample evaluations, data assimilation and investigative reports.

In 1981 BRGM carried out regional strategic exploration (Phase 1) between the Cacheu River and the Senegal border on a 40 km x 25 km area lying northwest to southeast and including Farim. A total of 32 holes of 35 m to 95 m depth were drilled for a total of 2100 m of drilling, of which 1384 m were cored. This demonstrated the presence of a layer of phosphate (FPA) over an area of 40 km^2 centred on the village of Saliquinhé. This layer, 2 to 5 m thick, had a high phosphate content of 30% P_2O_5 under 30 to 60 m of cover.

An 18 kg composite sample of FPA was taken from four drill holes and used for laboratory scale metallurgical testing. These tests yielded concentrates of 35% P_2O_5 from a sample containing 30.9% P_2O_5. A 14 kg composite sample of FPB was taken from four drill holes but did not produce good results.

In 1982 and 1983, BRGM conducted a campaign (Phase 2) to evaluate the mineralisation of this zone of Saliquinhé to determine potential mining parameters and to carry out a detailed geological investigation. There were 69 holes drilled on a grid of 500 m x 500 m (opening up to 1000 x 1000 m on the edge of the deposit) for a total of 3,527 m of drilling. This comprised 2,145 m of percussion drilling in the overburden and 1,472 m of 108 mm diameter core drilling in FPA and FPB where it existed.

The historic geological resource of FPA was estimated to be 113 Mt at 30% P_2O_5 over an area of 24.5 km^2, with an average thickness 3.27 m (minimum 1.5 m) under an average overburden thickness of about 40 m (28 m to 69 m). Gamma ray logging was carried out on all the drilling. This Historic Resource Estimate by BRGM has not been reviewed by the Qualified Person, should not be considered to represent a current resource and should not be relied upon.

Representative samples totalling 470 kg were taken from 30 holes for beneficiation tests carried out in BRGM's Orleans facilities, France, and in the laboratory of the Taïba Phosphates Company.

Concentrates containing 32% P_2O_5 and 3.5% FeAl were produced by simple magnetic separation. This was improved to 37% P_2O_5 and 2.5% FeAl by using flotation plus wet high intensity magnetic separation (WHIMS) and 1.5% FeAl with dry magnetic separation.

Gamma ray logging was carried out in some holes, the number of which is unclear. The logs obtained were of excellent quality and allowed identification (to the nearest 100 mm) of the contact between the overburden and the phosphate rich and phosphate poor with limestone or phosphate and limestone footwall.

The following examinations were also carried out on drill-cores:

- 400 thin sections for petrography and 400 washings for micropalaeontology. These were examined by BRGM specialists;
- 90 samples of micropalaeontology of vertebrates, teeth of Selacians and Betides. These were examined by the Faculty of Science of Montpellier;
- Examination of invertebrates (ostracised) in seven surveys, by the Faculty of Science of Lyon; and
- X-ray diffraction of 47 samples. These were used to determine the argillaceous minerals of the phosphate series and the ferrous minerals of the FPA hanging wall.

The gangue is minor in quantity. Detrital quartz represents 5% to 10% of the mass. The pyrite and marcasite are present in variable amounts in FPA, occurring as fine particles, coatings of phosphate grains or as cement in the narrow secondary silicified and pyritised levels associated with iron carbonates (ankerite). In certain thin sections a ferruginous epigenesis of the organic structures is present. There is also a very small amount of clay present as a discrete matrix between the phosphate grains.

This was followed in 1985 (Phase 3) when BRGM drilled eight or nine drill holes for geotechnical and hydrogeological information. Concurrent with this third phase of work, Sofremines carried out a Prefeasibility Study that was reported in 1986. This detailed mining a 29.8% P_2O_5 resource to produce 500,000 t of 35.5% P_2O_5 concentrate annually for 20 years.

The BRGM staff on-site were:

- Mining Geologist/Project Manager;
- Exploration Geologist; and
- Drilling Technical Manager.

9.1.2 Champion

After acquiring the project in 1997, Champion engaged consultants to review the previous work. No fatal flaws in the project were found and recommendations for further work leading to a full feasibility study of the project were presented.

During 1998 and 1999, Champion drilled 34 core holes totalling 1,810 m, mainly in the west and northwest of the zone explored by BRGM, to check the extension of the deposit in those two directions.

Champion's geological resources calculation included these 34 drill holes and the 69 of the BRGM program and estimated the resource to contain 166 Mt at 29.06% P_2O_5. This resource covered an area of 38 km^2 and was in a layer of 3.15 m average thickness under about 40 m of overburden. This Historic Resource Estimate by Champion has not been reviewed by the Qualified Person, should not be considered to represent a current resource and should not be relied upon.

The Mineral Corporation (TMC) audited the exploration work of 1998 and directed that of 1999. The resources were estimated by MRDI of Canada on the basis of data supplied by Champion and audited by John Zbeetnoff.

9.2 Recent Exploration

GBMAG carried out two phases of exploration, one between 2008 and 2009 and the second in 2011.

During 2008 and 2009 GBMAG drilled 10 resource holes in the northern area of the deposit and 20 resource holes at short spacing, centred on hole SE 5 in order to calculate a variogram at short distance. Figure 9-1 shows the layout of the 20 holes. This phase totalled 1564 m of drilling including 423 m of core drilling. Gamma ray logging was carried out on 26 drill holes. The rationale of GBMAG drilling was to extend the evaluation of the resources in the prospective northwest direction and to evaluate the short distance variability of the grade and thicknesses parameters, which would affect the possible exploitation.

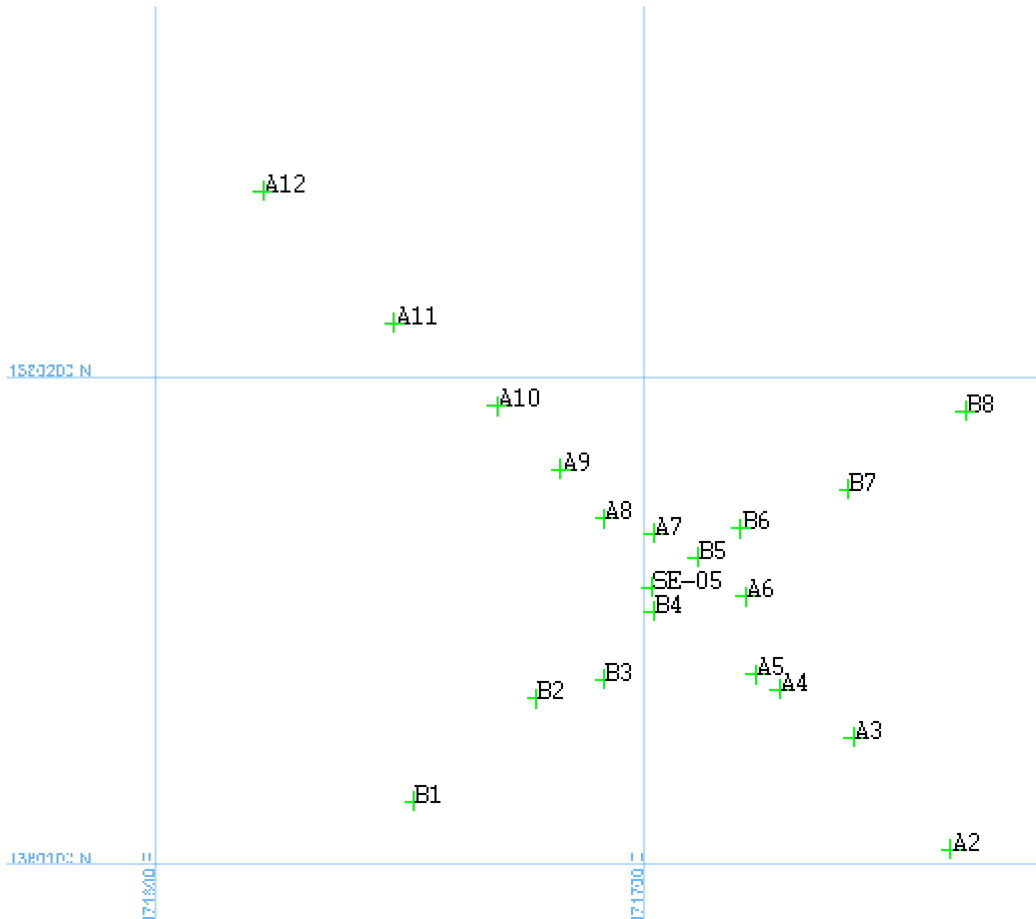
During 2011, GBMAG drilled 25 resource holes in the north and western areas of the deposit. These holes totalled 1,280 m including 180 m of core drilling. The aim of the second drilling program was to further extend the resources to the northwest and also to a grid of 500 m. Drilling was supervised by two senior geologists and the company's Chief Geologist.

GBMAG has also embarked upon a number of other studies including:

- Environmental and social impact assessment including baseline monitoring;
- Ground characterization program that includes geotechnical investigation and hydrogeological testing; and
- Preliminary mining engineering studies.

Global Geomatics was contracted in 2011 to re-survey all drill hole collars. The surveys were carried out using GPS with absolute horizontal accuracy of 0.03 m and vertical accuracy of 0.05 m. In addition, AOC (AOC Archaeology Group) conducted an airborne LiDAR survey, with horizontal accuracy of 0.5 m and vertical accuracy of 0.2 m.

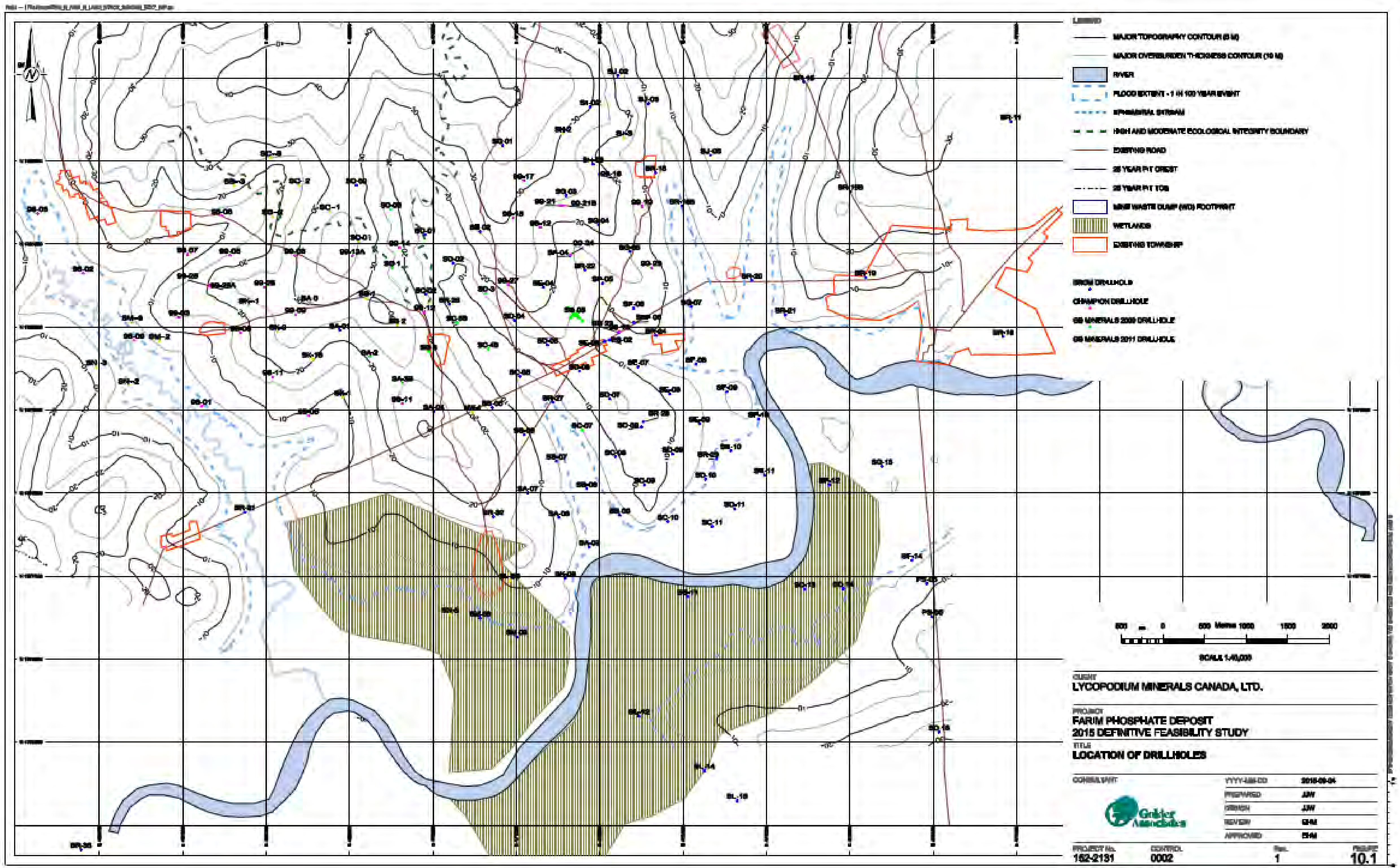
Figure 9-1 Location of GBMAG 20 Drill Holes for Short Distance Variability (GBMAG Variogram SD 5)



10.0 DRILLING

Drilling in and around the Farim project area has been carried out by several companies since discovery of the deposit. The current database contains 190 holes comprising 10,327 m of drilling using a combination of percussion and core drilling techniques. The drilling coverage and phosphate thickness is shown in Figure 10-1. Since the layers of phosphate are horizontal, all the holes were drilled vertically and therefore thicknesses shown are believed to be true thicknesses. The average depth of drill holes at Farim is 54 m.

Figure 10-1 Location of Drill Holes and Phosphate Thickness



10.1 BRGM Drilling

The BRGM drilling program was carried out in three phases between 1981 and 1985. This consisted of 101 drill holes totalling 5,672 m of which 2,861 m was core drilling.

Generally the upper formations were drilled with a destructive rotary bit. The bit was removed some 2 m above the estimated roof of the FPA and diamond core drilling used for the phosphatic horizons down to 1 m below the FPB. If the roof was missed, the hole was generally re-drilled but it may happen that the FPA roof was above the cored interval.

Drilling a soft formation containing hard nodules and lenses like the FPA is challenging as the hard nodules and fragments present tend to destroy the sand below, which is disaggregated and washed away. Even with a triple barrel and an expert driller the recovery may be expected to vary and low recoveries reported in the BRGM and following reports should not be attributed to bad practice or to negligence. The phosphatic clasts have a porous texture and a very low density. Although the crystallised apatite is denser than quartz, the phosphatic clasts are lighter than quartz and may be washed away more easily. An increase in the phosphate grade is not expected in this process.

The thickness of the phosphatic layer was systematically double checked with a gamma probe, in close correlation with the phosphorous content.

Phase 1 of the BRGM drilling program was a regional exploration program carried out in 1981 covering a 40 km x 25 km area lying northwest to southeast and including Farim. A total of 32 holes of 35 m to 95 m depth were drilled, representing 2,100 m, of which 1,384 were cored.

Phase 2 was a local exploration campaign carried out in 1982 to 1983 to define the resources at Farim. A total of 69 holes were drilled over an area of approximately 40 km^2 (5 km by 8 km) on a 500 m grid (1,000 m on the northern part of the deposit). This comprised 3,572 m including 2,145 m percussion drilling in the overburden and 1,472 m of core drilling in FPA and FPB.

The setting up of the 500 m x 500 m exploration grid was implemented by a team of topographers from the Ministry for the Natural Resources of Guinea-Bissau, directed by a Peruvian specialist. The ground survey was completed in April 1983, with the general survey of Guinea-Bissau by the IGN (French National Geographical Institute).

The following equipment was used:

- A Longyear 34 drill on a truck and a tanker of 7,000 litres;
- Trepans tri-cone of 160 mm for overburden drilling, casing of diameter 135 to 145 mm;
- A Craelius 131T6 drill equipped with high-carbon or diamond core barrels for continuous core;

- Sampling of the FPA and FPB layers, extracting cores 108 mm in diameter and of maximum length 3.05 m; and

- Casing before introduction of the probe gamma ray (Probe Mount Sopris).

Phase 3 consisted of gathering geotechnical and hydrogeological information from eight or nine drill holes. It is unknown if these were new or existing drill holes.

Most of the hole collars are marked in the field with strong concrete beacons as shown in Figure 10-2 and were relocated effectively by GBMAG geologists. It is unknown whether the location was surveyed.

Figure 10-2 BRGM Collar Marked in the Field with Concrete Beacon

10.2 Champion Drilling

During 1998 and 1999, Champion carried out 34 core drill holes totalling 1,810 m, mainly in the north and northwest of the zone explored by BRGM, to check the extension of the deposit in these two directions. No information is available about the type of drill rig used, the diameter of core drilled or whether the collar locations were surveyed or the holes were gamma ray logged. The drill hole collars were marked by smaller, flat, concrete plugs which were more difficult to locate.

10.3 GBMAG/GEEEM Drilling

GBMAG generally drilled the upper formations, following BRGM's protocol, with a destructive rotary bit until approximately 2 m above the estimated roof of the FPA. The remainder of the hole was drilled using diamond core drilling. The geologist stopped the hole once it passed through the floor of the FPA layer and into the footwall (FPB or limestone).

The collar location of the holes was surveyed using a handheld GPS, except for the set of holes used for the variogram. These holes were surveyed and levelled locally by a consulting surveyor. The holes are currently open and visible but not marked.

The core was placed in wooden core boxes in the field and, while still wet, was manually cut using a steel bladed knife longitudinally to recover the complete half core intervals. GBMAG geologists collected the core and transported it back to the core shed, in the GBMAG office in Farim.

Between 2008 and 2009 GBMAG drilled 30 holes totalling 1,564 m, of which 423 m was core drilling. In 2011, 25 holes were drilled totalling 1,280.5 m of which 180.5 m was core. The balance of the drilling represents the open hole drilling undertaken with a destructive rotary bit. Gamma ray logging was carried out on 29 holes. As the mineralisation is horizontal, the vertical hole intersections are representative of the true thickness of the mineralisation.

The first phase of drilling was located to provide better coverage of the north and west part of the deposit, validate the range of grades and thicknesses observed in the previous drill holes and give better definition of the variability of the mineralisation. The second phase of drilling was planned to further extend the known mineralisation towards the north and west and also to infill to an approximate 500 m grid spacing.

This work was managed and supervised by GEEEM (Geologie Exploration Environment Expertise Mine), an independent geological consulting company that was contracted by GBMAG to manage and supervise exploration activities and conduct exploration work programs including the drilling at the Project. The principals of GEEEM have extensive geological experience in phosphate deposits, phosphate exploration and mining.

10.4 Drill Core Recovery

The rate of recovery of the FPA cores is fair. 80% of the cores from the BRGM holes have a rate higher than 50% and for GBMAG the average rate of recovery is of 83%. These results are related to the granular nature of FPA, with low cohesion due to the absence of argillaceous matrix and by particular constraints:

- The silica-alumina-iron level at the top of FPA is hard and a piece of core can remain stuck and break the phosphate, preventing it forming a core; and
- The large amount of water in the drilled phosphate matrix makes it difficult to core a semi-liquid product.

BRGM states that the P_2O_5 content of the drill core with weak recovery is lower than the average. This is explained by the fact that the finer phosphate sand, the most easily lost, is of high grade. The use of the P_2O_5 contents of the core with weak recovery leads to under estimation of the P_2O_5 content.

A statistical study carried out by a consultant of Champion concluded that: “There is no relationship between thickness of FPA and core recovery and the uses of lower core recovery drill holes in the geological model would tend to make the P_2O_5 grade estimate slightly conservative and would not affect the Fe_2O_3 grade estimate”.

10.5 Drilling Factors Impacting Accuracy and Reliability of Results

The exploration programs performed on the Project area were generally carried out according to appropriate professional methodologies and procedures. Exploration procedures for the early phases of exploration on the Farim Phosphate Project were developed in accordance with BRGM protocol. All exploration drill program work appears to have been performed by experienced and qualified personnel, including GB Minerals personnel as well as reputable third-party contractors.

There are no identified significant factors or concerns regarding the accuracy and reliability of the results from the exploration programs on the Project area.

11.0 SAMPLING PREPARATION, ANALYSES AND SECURITY

11.1 BRGM Program

BRGM paper records and descriptions are detailed. Copies of all original geological logs are kept in a data room at the UBS bank in Zurich, Switzerland. No assay certificates are available, but the assay results are written on the log for each hole.

11.1.1 Lithological Logging

BRGM and Champion cores were stored in the shed of the Ministry of Mines in Bissau. Sometime after the beginning of the civil war, in 1998, the sheds were bombed and the cores destroyed. For this reason, Golder was unable to view the historic cores and validate any of the geological logging.

11.1.2 Density

Dry density measurements were made by BRGM in 1983. BRGM took 31 samples from 14 drill holes and sent them to the BRGM laboratory in Orleans for density determination using a "membrane densitometer". Only samples with 100% recovery were selected. The mean density value is 1.43 t/m^3 with a lowest value of 1.18 t/m^3 and a highest value of 1.82 t/m^3. The lower density values correspond to a clear colour phosphate and the high density values relate to a dark colour phosphate. Use of only those samples for which there was 100% recovery may bias the results as density may differ between solid core and friable core.

11.1.3 Sample Preparation Procedures

BRGM documents the following procedure for core sampling and sample preparation at the BRGM facilities at that time:

- Splitting drill core increments as received along core length. One half was kept as a reference, the other half was split into two parts longitudinally to obtain quarter core samples for analyses and constitution of composites samples for treatment tests;

- Initial chemical determinations were made on one quarter, representing about 2 kg of dry material per metre length. The remaining quarter core was retained as a control sample. Drying was carried out in an oven or by natural drying, weighing and stage crushing of quarter core samples down to about 8 mm using jaw crushers;

- Grinding jaw crusher product down to about 0.5 mm to 2 mm using either a roll crusher or a disc mill;

- Splitting less than 2 mm ground materials using chute splitters with 25 mm and 10 mm channel widths to produce two representative subsamples of 100 g to 150 g which were kept in plastic bags. The remaining material was bagged and kept as a spare sample. When applicable, basic Gy's equations were used to estimate sampling errors made in primary sample splitting. Typically, drawing a 100 g sub-sample of 2 mm top size would give rise to a theoretical sampling error of 0.05% P_2O_5 at an average P_2O_5 content of 29%, which is considered negligible. Regarding the sampling error, the 95% confidence limits on grade are 29% P_2O_5 ± 0.1% P_2O_5; and

- Drying in an oven at 105° C, weighing and milling one of the subsamples down to 80 µm (100% passing the 80 µm screen, corresponding to about 95% passing 200 mesh), using a vibrating cup mill with tungsten carbide or agate grinding chamber and rings. The pulverised material was split, sub-sampled and spare samples kept in sealed plastic tubes to be dispatched to laboratories in charge of analysis and check analysis.

11.1.4 Analytical Procedures

Based on reports, it has been determined that BRGM carried out chemical analyses at the laboratory of the DGMGB (Directorate of Geology and Mines of Guinea-Bissau) in Bissau. A total of 838 intervals were selected from 101 cores.

From the Phase 1 BRGM drilling, 470 samples were assayed by the laboratory at the DGMGB for P_2O_5 using colorimetry. Of these samples, 178 samples containing more than 10% P_2O_5 were analysed for a further 10 elements.

For the 69 holes drilled during the second BRGM campaign, 368 intervals were assayed at the laboratory at the DGMGB. Of these, 288 recorded greater than 10% P_2O_5 and were analysed for a further 10 elements.

Forty two analyses for 26 elements were performed in Orleans. The uranium analyses were carried out by Cogema (Areva).

No information is available on the size of these samples.

Core samples collected from 60 drill holes of the 1982 to 1983 campaign were analysed in the DGMGB laboratory for the purpose of resource calculation.

In 1986, check analyses were done at BRGM laboratories in Dakar and Orleans, France on finely ground samples prepared by DGMGB as part of a Prefeasibility Study by Sofremines.

It was recorded that the BRGM subsidiary, DGM (Directorate of Geology and Mines) used the following analytical methods:

- P_2O_5: spectrophotometry (no original data or samples were available for review);
- CaO: volumetric titration;
- SiO_2: either AAS or gravimetric determination;

- Al_2O_3, Fe_2O_3, TiO_2, MgO: AAS;
- F: spectrophotometry using Eriochrome Cyanine R as a colour development reagent;
- CO_2 from carbonates and phosphate particles: CO_2 volume measurement following acid dissolution; and
- U: uranium content of selected finely ground samples were determined by the CEA (Commissariat à l'Energie Atomique) in France, which specialises in uranium analyses.
- At BRGM in Orleans P_2O_5 contents of samples were determined from solutions obtained after acid dissolution with sulphuric and nitric acids, using:
- A spectrophotometric method based upon the yellow colour of the ammonium phosphor-vanadomolybdate complex; and
- A gravimetric method based upon weight of the precipitate of the phosphomolybdate of quinoline, (Perrin-Wilson-Dahlgren method).

Both methods followed analytical procedures given in the French Association Français de Normalisation (AFNOR) standards NF U42-201 and NF U42-245 relevant to the control of phosphate fertilisers. Spectrophotometric determinations were validated against gravimetric determinations at BRGM since at that time the gravimetric method was the reference method in the phosphate fertiliser industry in France.

Routine analyzes for P_2O_5, CaO, SiO_2, Fe_2O_3, Al_2O_3, MgO, Na_2O, K_2O, TiO_2, MnO, were made by X-Ray Fluorescence (XRF) on fused glass beads using lithium tetraborate as a fluxing reagent. Loss on Ignition (LOI) was also determined.

11.1.5 Sample Storage and Dispatch

There is no information on how BRGM stored the core or dispatched samples.

11.1.6 QAQC

Pulp Duplicates

Check analyzes for P_2O_5 by spectrophotometry were made by the BRGM laboratories in Dakar and Orleans on 11 samples as finely ground powders (less than 80 µm) and compared with the corresponding determinations at Directorate of Geology and Mines (DGM). Results are shown in Table 11-1, Figure 11-1 and Figure 11-2. As they are pulp duplicates, the sampling error will be comparable and therefore any differences observed can be accounted for by analytical errors. Comparison of results of phosphate analysis provided by laboratories of the DGM in Guinea-Bissau, BRGM in Dakar and BRGM in France was carried out (IMC, 2011). Regression equations indicate a significant overestimation, mainly in the low grades, of the P_2O_5 content by the DGM laboratory compared to the BRGM laboratories in Dakar and Orleans:

- % P_2O_5 Dakar = 1.0519% P_2O_5 DGM - 2.28314 Correlation coefficient, R = 0.9973; mean absolute error SD = 0.759 % P2O5 with 95% confidence limits on parameters of 1.0519 ± 0.052 and -2.28314 ± 1.3356; and

- % P_2O_5 Orleans = 1.03238% P_2O_5 DGM - 2.35389 Correlation coefficient, R = 0.9959; mean absolute error SD = 0.913% P_2O_5, with 95% confidence limits on parameters of 1.03238 ± 0.0626 and -2.35389 ± 1.608.

It should be noted that the gradients of the regression lines are not significantly different from one.

Overestimation is almost constant over the controlled interval ranging from 13.4 to 37.6% P_2O_5. However, the low number of samples used in these comparisons results in broad confidence intervals on the intercepts with the ordinate axis, thereby rendering quantitative assessment less conclusive.

Table 11-1 BRGM-DGM Drilling Campaign (1981 to 1983)

Drill Hole	Drill Core Distance from surface (m)	Drill Core Length (m)	P_2O_5 Content (%) by DGM Dakar	P_2O_5 Content (%) by BRGM Dakar	P_2O_5 Content (%) by BRGM Orléans
BR20	34.50 to 35.50	1.00	13.4	12.51	11.9
	35.50 to 37.10	1.60	14.6	13.89	12.2
	45.50 to 47.30	1.80	15.5	13.20	13.0
BR21	37.50 to 38.70	1.20	26.6	24.69	24.1
	41.00 to 42.00	1.00	17.5	16.01	16.2
	50.00 to 50.65	0.65	15.4	13.52	14.2
BR23	35.15 to 35.70	0.55	31.8	31.95	31.0
	35.70 to 37.60	1.90	33.4	33.46	33.0
	37.60 to 38.80	1.20	24.6	23.29	22.5
BR28	45.40 to 46.40	1.00	35.0	35.13	35.0
	46.40 to 46.75	0.38	37.6	36.41	35.0

Figure 11-1 Phosphate Analysis by BRGM Dakar and BRGM France (Orleans)

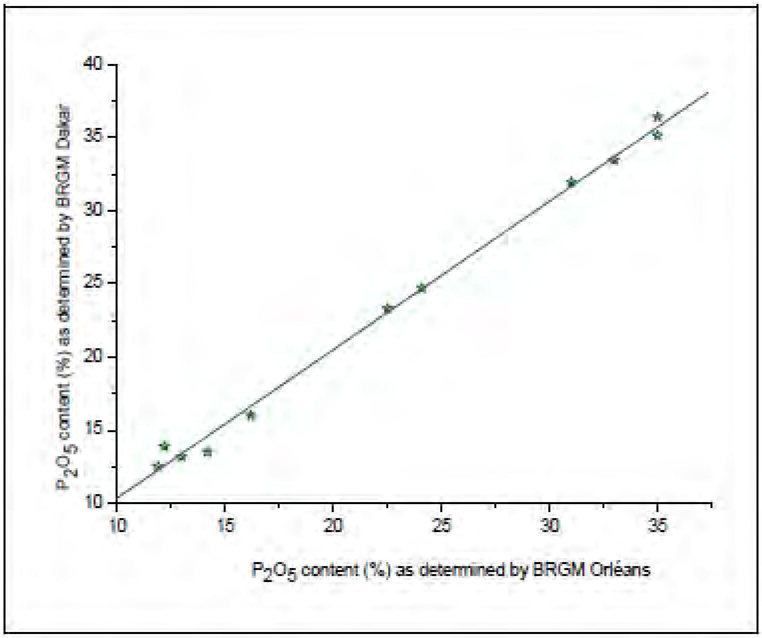
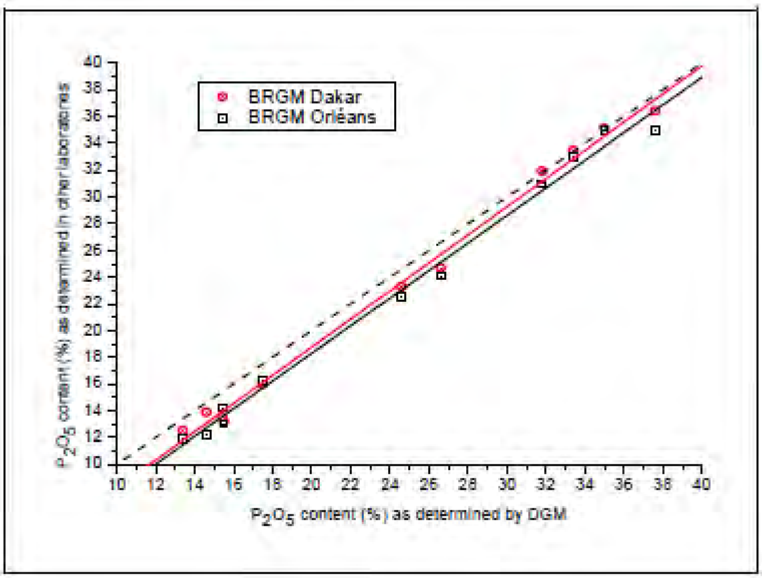


Figure 11-2 Comparisons of Phosphate Analysis by DGM, BRGM Dakar and BRGM France



As most of the phosphate matrix samples have P_2O_5 contents ranging from 28% to 32% P_2O_5, the possible error is acceptable since the relative difference in grade does not exceed 4.7% for a P_2O_5 content of 30% determined by the DGM laboratory.

Good agreement is observed between determinations provided by BRGM laboratories:

- % P_2O_5 Dakar = 1.01513% P2O5 Orléans + 0.20055.

Correlation coefficient, R = 0.9977, mean absolute error SD = 0.700% P_2O_5 with 95% confidence limits on parameters of 1.01513 ± 0.046 and +0.20055 ± 1.127.

Internal QAQC

It is reported that BRGM is well qualified and has significant experience in P_2O_5 analyzes. BRGM was in charge of periodic check analysis made of phosphate concentrate exports from the Office Togolais des Phosphates. The laboratory performed extensive work to estimate analytical errors made on P_2O_5, SiO_2, Al_2O_3, Fe_2O_3, H_2O and Cl contents on ores and concentrates. Round robin checks comprising comparisons between many laboratories in France, Togo and Senegal were conducted to assess the accuracy and reliability of the BRGM analysis. As an example, 95% confidence limits on a reference sample assaying 37% P_2O_5 were 37.09% ± 0.15%. Periodic analyses were also made on international phosphate rock reference samples prepared by the CRPG (Research Centre for Petrography and Geochemistry) in Nancy, France. These would be analogous to external standards.

Field Duplicates

Selected drill core samples, as quarters of initial core samples, were taken by Sofremines in December 1985 for check analysis and production of a composite sample for beneficiation tests. The principal purpose of these tests was to validate the BRGM sampling and analyses.

Twenty one Sofremines samples analysed for P_2O_5 by XRF on glass beads are listed in Table 11-2 together with comparisons between P_2O_5 contents in samples as analysed and P_2O_5 contents as determined by DGM (length weighted averages). Statistical treatment of the distribution of differences between P_2O_5 contents in Sofremines and DGM samples results in rejection of two analyses (Figure 11-3). Differences with DGM results account for both sampling and analytical errors.

Within 95% confidence limits, it can be shown that no significant bias exists for the 19 observations remaining even though Figure 11-3 displays scattered data points. The gradient of the regression line passing through the origin is not significantly different from 1. High dispersion around the regression line, (standard deviation of 1.90% P_2O_5), accounts for both analytical and sampling errors. The latter are expected to be high as a consequence of physical reconstitution of core intervals for Sofremines samples and reconstitution of phosphate grades of DGM samples by calculation.

Seven Sofremines samples were analysed for Fe_2O_3 by XRF on glass pellets and six of them were also analysed by wet chemical methods for S as sulphides with the objective to estimate proportions of total iron occurring as iron sulphides (FeS_2 as pyrite and marcasite in accordance with BRGM mineralogical data). Data including comparisons of Fe_2O_3 contents in Sofremines samples and Fe_2O_3 contents calculated from DGM analysis are shown in Table 11.2. From the limited data available, the proportion of iron occurring as sulphides in FPA samples assaying 28.85% to 36.20% P_2O_5 appears to be highly variable, ranging from 43% to 95%.

Figure 11-4 shows correlation between Fe_2O_3 contents as analysed in Sofremines' samples and Fe_2O_3 contents as calculated from DGM analysis. As mentioned for phosphate analysis, differences with DGM results account for both sampling and analytical errors.

Similar to comparisons for P_2O_5, the slope of the regression line passing through the origin is not significantly different from one, indicating that significant bias does not exist. However, too few data points give rise to poor accuracy in statistical analysis of the data.

Figure 11-3 Comparison of Sofremines Check Analysis on Spare Drill Core Samples with Phosphate Analysis at DGM Laboratory on Initial Core Samples

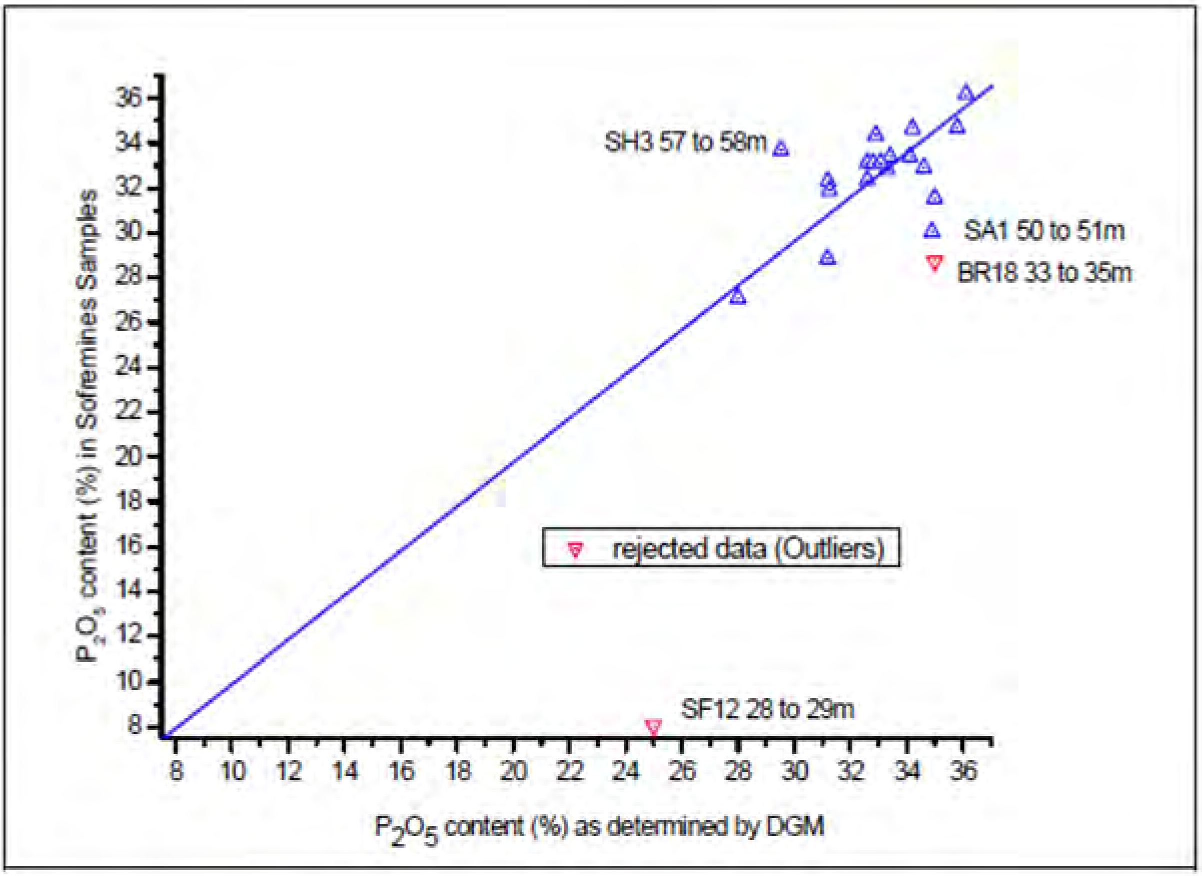
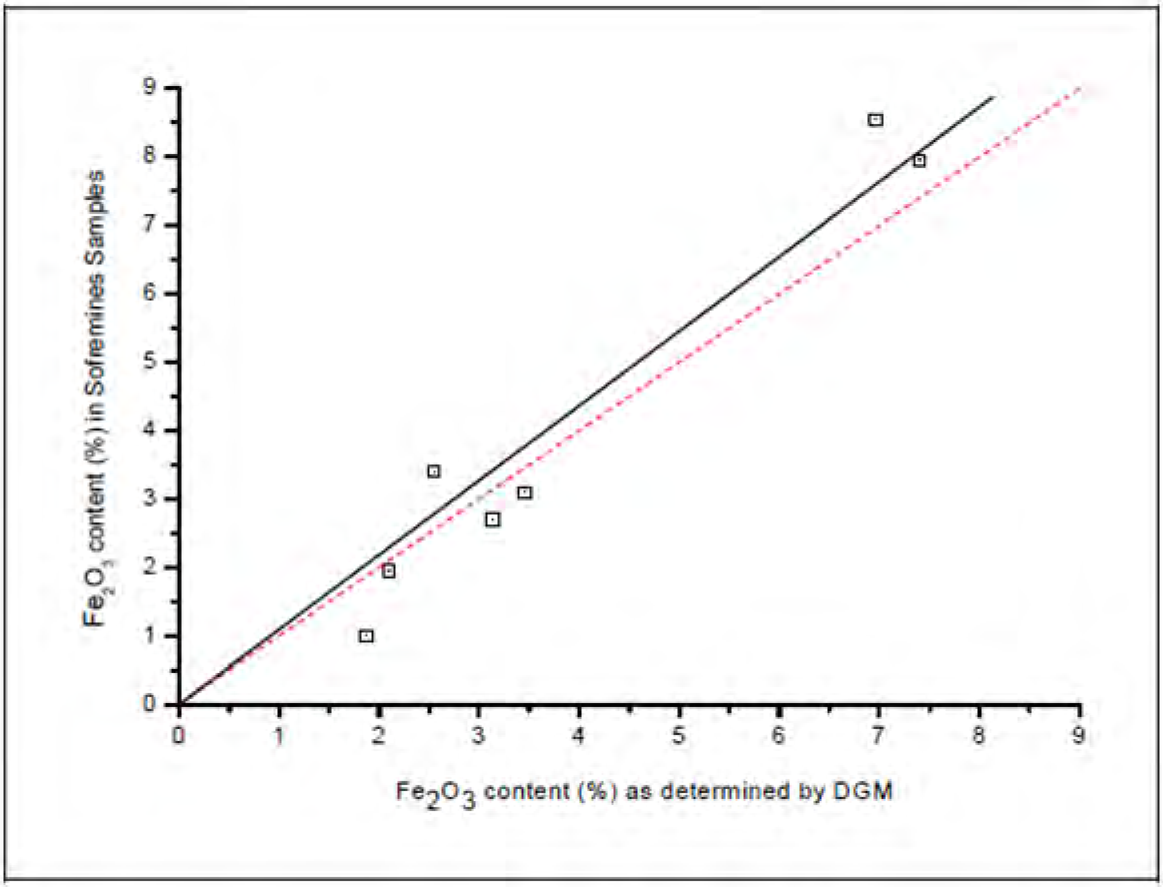


Table 11-2 Results of Check Analysis for Phosphate Conducted by Sofremines in 1985 to 1986 on Drill Core Samples (25% of Initial Drill Cores) (Reformatted from AMC, 2005)

Sofremines Check Analysis					DGM Initial Data					
Drill Hole	Drill Core Distance From Surface (m)		Drill Core Length (m)	P_2O_5 (%)	P_2O_5 (%) as Analysed or Reconstituted	Drill Hole	Drill Core Length (m)	Drill Core Distance from Surface (m)		P_2O_5 Content (%) in Core Length
SC 2	41.25	42.30	1.05	32.30	31.20	SC 2	1.00	41.40	42.40	31.2
	42.30	43.30	1.00	33.15	32.60		1.00	42.40	43.40	32.6
	43.30	44.40	1.10	34.65	34.20		1.50	43.40	44.90	34.2
	44.40	46.00	1.10	32.35	32.60		0.60	44.90	46.00	32.6
	46.00	47.30	0.90	33.40	34.10		0.90	46.00	47.20	34.1
SB11	28.50	30.50	2.00	31.90	31.26	SB11	0.30	28.50	28.80	294.0
	30.50	32.10	1.60	28.85	31.18		0.70	28.80	29.50	32.4
							0.90	29.50	30.40	31.0
							0.80	30.40	31.20	28.0
							0.90	31.20	32.10	34.0
SE2	56.30	58.00	<1.00	27.10	28.00	SE2	1.00	56.30	58.50	28.0
	58.00	60.60	About 1.50	33.70	29.53		0.50	58.50	59.30	32.6
							1.00	59.30	60.60	28.0
BR23	36.00	38.00	2.00	32.85	33.30	BR23	1.90	35.70	37.60	35.2
							1.20	37.60	38.80	25.7
BR28	46.00	47.00	1.00	36.20	36.11	BR28	0.90	45.50	46.40	36.3
							0.35	46.40	46.75	37.9
							1.15	46.75	47.90	33.3
BR29	32.00	34.00	2.00	33.10	32.80	BR29	0.70	31.70	32.40	28.0
							1.00	32.40	33.40	35.5
							1.00	33.40	34.40	31.5
SH3	54.00	55.00	1.00	34.35	32.90	SH3	1.00	53.50	54.50	33.2
	55.00	56.00	1.00	33.40	33.40		1.00	54.50	55.50	32.6
	56.00	57.00	1.00	32.90	34.60		1.00	55.50	56.50	34.2
	57.00	58.00	1.00	31.55	35.00		1.00	56.50	57.50	35.0
	54.00	58.00	4.00	33.05	33.98		1.00	57.50	58.50	35.0
SH3	54.00	55.00	1.00	34.35	32.90	SH3	1.00	53.50	54.50	33.2
	55.00	56.00	1.00	33.40	33.40		1.00	54.50	55.50	32.6
	56.00	57.00	1.00	32.90	34.60		1.00	55.50	56.50	34.2
	57.00	58.00	1.00	31.55	35.00		1.00	56.50	57.50	35.0
	54.00	58.00	4.00	33.05	33.98		1.00	57.50	58.50	35.0

Sofremines Check Analysis					DGM Initial Data					
Drill Hole	**Drill Core Distance From Surface (m)**		**Drill Core Length (m)**	**P_2O_5 (%)**	**P_2O_5 (%) as Analysed or Reconstituted**	**Drill Hole**	**Drill Core Length (m)**	**Drill Core Distance from Surface (m)**		**P_2O_5 Content (%) in Core Length**
SF12	28.00	29.00	chert pebbles inter waste	8.05	25.00	SF12	1.80	27.70	29.50	25.0
SA1	48.00	49.00	1.00	35.15	33.07	SA1	0.60	47.70	48.30	30.2
	49.00	50.00	1.00	34.70	35.80		0.70	48.30	49.00	31.8
	50.00	51.00	1.00	30.05	34.90		0.05	49.00	49.05	35.0
							0.95	49.05	50.00	35.8
							0.50	50.00	50.50	33.2
							1.20	50.50	51.70	36.6
BR18	33.00	35.00	2.00	28.75	35.00	BR18	1.00	33.00	34.00	35.8
							1.20	34.00	35.20	34.2

Figure 11-4 Comparison of Sofremines Check Analysis on Spare Drill Core Samples with Fe_2O_3 analysis at DGM Laboratory on Initial Core Samples



The Qualified Person is satisfied that the adequacy of the sample preparation, analytical and security procedures described and recorded is satisfactory for the time the analysis was undertaken and that the results meet the prevailing international standards.

11.2 Champion Program

The Champion paper record available is incomplete, no original geological logs or assay certificates are available. There is no information on how Champion processed the core, e.g. logging and sample preparation.

11.2.1 Lithological Logging

Refer to Section 11.1.1.

11.2.2 Sample Preparation Procedures

There is no information on how Champion processed the core.

11.2.3 Density

Champion conducted bulk density measurements on 37 samples derived from drill core samples with 100% recovery.

The program included the measurements of core diameter and length. The ‘theoretical volume’ for each sample was determined by simple computation using the average measured core diameter for competent core and the drilled length. Each sample also had the percentage of core recovered calculated from the ratio of the volume of core recovered and the ‘theoretical volume’. Three mass measurements were recorded at the Bateman laboratory. The mass of the sample was measured in an ‘as received state’, referred to as ‘wet’, a mass in an ‘air dried state’ and a mass in an ‘oven dried state, at 105°C. The bulk density values were computed by dividing the ‘theoretical volume’ by the ‘oven dried mass’. Samples that had less than full core recovery had the ‘oven-dried mass’ adjusted upwards by an amount based on the percent of core loss. This mass adjustment assumes that the material with poor recovery has the same bulk density as the material with good recovery.

The mean of the values derived by Champion is 1.45 t/m^3 after excluding the highest abnormal values. The lowest value is 1.18 t/m^3 and a highest value of 1.98 t/m^3. There is no apparent evidence of a relationship between density and phosphate grade.

11.2.4 Analytical Procedures

The following discussion and observations result only from review of reports. No original data or samples are available for review.

The 1998 Champion assaying was carried out by Mineral Resources Associates in Florida. This included P_2O_5 for all data and an additional seven elements for selected intervals. The 1999 Champion assaying was carried out by Bateman Projects Limited in South Africa. The assaying included P_2O_5 for all sample intervals and an additional seven elements for selected intervals. The number of samples or the sizes of sample intervals were not detailed in the Champion report (Champion, 2000).

According to the resource audit by the consultant to Champion (Zbeetnoff, 2000), drill hole core samples collected during the Champion exploration campaigns were analysed for:

- P_2O_5 for all sampled intervals and CaO, Fe_2O_3, Al_2O_3, MgO, F, As and Cl for selected intervals by Minerals Resources Associates of Florida (USA) in 1998; and

- P_2O_5 for all sampled intervals and CaO, Fe_2O_3, Al_2O_3, MgO, F, Cd and TiO_2 for select intervals by Bateman Projects Limited and by Performance laboratories, (for Al_2O_3), in RSA in 1999.

It should be noted that sampling procedures were not given and the analytical methods used are not fully described in reports made available.

11.2.5 Sample Storage and Dispatch

There is no information on how Champion processed the core.

11.2.6 QAQC

Internal quality control at Bateman laboratories that undertook testing for Champion consisted of:

- Duplicate analysis on two sets of samples: the first included 45 duplicate analyses covering P_2O_5 assays ranging from about 2% to 36%, the second included 23 duplicate analyses of P_2O_5 contents ranging from about 28% to 36%. Correlation coefficients, R2, between pairs of determinations were about 1 for the first set and 0.998 for the second set. These high R2 values clearly indicate high repeatability of P_2O_5 determinations at the Bateman laboratory;

- Repeated analysis of two phosphate rock standards assaying 26.7% and 26% P_2O_5 obtained from the Israel phosphate industry. Some 41 analyses of the 26.7% standard provided assays ranging from about 26% to 26.7% P_2O_5 with an average value slightly lower than the reference assay. The 90 analyses of the 26% standard did not reveal significant bias with regard to the reference assay; and

- Repeated analysis of calibration standards assaying 10%, 20%, 30% and 35% P_2O_5. Results of more than 60 analyses on the 10% standard indicate a slight overestimation (results ranging from about 10.05% to 10.25% with an average of about 10.15%) whereas no significant bias was detected for the other calibration standards.

It should be noted that information from the original Bateman and Champion reports was incomplete. Accordingly, certain information corresponding to those reports is also incomplete in this report, such as the absence of confidence limits on the gradients and intercepts of the regression equations used to relate results of initial and check analysis.

Check Analysis for P_2O_5

Check analyses on 17 composite samples were undertaken at Setpoint Laboratories using a gravimetric method.

The linear regression equation that expresses the relationship between Setpoint and Bateman determinations was found to be:

- % P_2O_5 Setpoint = 0.9044% P_2O_5 Bateman + 2.87; and

- Correlation Coefficient, R= 0.9743.

Bateman provides slightly higher grades when compared with Setpoint for P_2O_5 grades less than about 30% and slightly higher grades for P_2O_5 grades exceeding 30%. As most of the Farim samples have P_2O_5 contents in the 28 to 32% range, possible analytical errors are not expected to strongly alter estimations of phosphate matrix grade and resources.

Check Analysis for Al_2O_3

Alumina grades were determined by Atomic Absorption Spectrometers (AAS) on solutions from standard acid attack at Performance Laboratories. Check analyses were performed by Setpoint using ICP (Inductively Coupled Plasma) spectrometry on 47 samples. The relationship between the 47 couples of alumina determinations is given by the following regression equation:

- % Al_2O_3 Setpoint = 1.077% Al_2O_3 Performance + 0.02; and
- Correlation Coefficient, R= 0.9963.

Alumina grades obtained by Performance Laboratories using AAS are slightly lower than alumina grades determined by ICP at Setpoint. For Al_2O_3 grades lower than 5%, corresponding to most of the francolite-containing ores encountered in the deposit, the difference in grade given by the two methods is acceptable less than 8.1% relative difference for a 5% Al_2O_3 grade determined by the method used by Performance Laboratories. The AAS method used at Performance Laboratories is recommended by the Association of Florida Phosphate Chemists, an organisation specialised in chemical characterisation of phosphate substances. For this reason, results from AAS determinations should be preferred.

Check Analysis for Fe_2O_3

Bateman used titration with dichromate to determine total Fe as Fe_2O_3 in solutions from modified acid attack of phosphate samples, a method recommended by the Association of Florida Phosphate Chemists. Check analyses were carried out at Setpoint on 121 samples using ICP spectrometry on solutions from standard acid attack. Results given by the two methods are related by the following regression equation:

- Fe_2O_3 Setpoint = 1.1841% Fe_2O_3 Bateman + 0.66; and
- Correlation Coefficient, R= 0.9807.

It appears that Bateman provides lower Fe_2O_3 grade in comparison to Setpoint. The relative difference in grade reaches 22.8% for a Fe_2O_3 content of 15% determined by the method adopted by Bateman. As the latter method is recommended by the Association of Florida Phosphate Chemists, it can be considered as the reference method for Fe_2O_3 analysis within the appraisal of the deposit.

11.3 GBMAG/GEEEM Program

11.3.1 Lithological Logging

The GBMAG cores are currently stored in a core shed located within the main office compound in the village of Farim. The area used for core logging and storage has a concrete base with the cover of a corrugated steel roof and is open at the sides. The cores were inspected by the Qualified Person, assisted by Geologist Guy Voglet who was in charge of GBMAG drilling program. The Qualified Person has verified that no aspect of sample preparation was conducted by any employee, officer, director or associate of the Issuer or Vendor.

At the time of Golder's site visit, a number of recommendations were made to improve the housekeeping within the core shed. Golder recommended the use of a logging table, rather than core boxes being placed on the floor for logging and sampling. Golder recommended processed core boxes should be covered to help preserve the remaining core and be stacked in a neat and ordered manner for ease of retrieval. These recommendations were implemented shortly after the site visit and for the remainder of the drilling program.

The lithological log of each hole was compiled by the geologist (BRGM and GBMAG) after an examination of materials and a simple identification test. Homogeneous intervals were differentiated by petrography, colour, hardness and friability, sometimes after examination with a binocular magnifying glass.

The log was recorded on a section showing the lithology, the rate of recovery, the gamma ray log (where taken), the intervals selected the P_2O_5 content and a photograph. The gamma ray logging defined the hanging and foot wall of FPA to within 100 mm where core recovery was poor. There is also a good correlation between P_2O_5 content and the amplitude of the recorded gamma log.

11.3.2 Density

GBMAG carried out no density measurements.

11.3.3 Sample Preparation Procedures

The procedure currently in use for core sampling and sample preparation is as follows:

- Splitting drill core along the core length. One half is kept as a reference, the other half used for sampling and analysis;
- After natural drying in core boxes, the half core material is crushed by hand to approximately 15 mm. Half of this material is selected and crushed by hand to about 2 mm. This is followed by homogenisation and splitting to obtain approximately 400 g followed by further crushing to less than 1 mm; and
- One quarter (approximately 100 g) of the crushed sample is placed in a heavy duty plastic bag marked with the sample unique number; the remaining crushed samples are stored for reference.

11.3.4 Sample Storage and Dispatch

The collection and processing of all samples prior to dispatch to laboratory is conducted by GEEEM and GBMAG employees. All sampling is sent as a single batch once drilling is complete. Samples are stored in a locked room at the GBMAG office, to which only GBMAG and GEEEM employees have access. Samples are despatched in wooden crates, submitted using a standardised laboratory submission form which lists the sample numbers, type of material and analysis required and batch number.

There were no reports of security problems as the commercial value of the samples is low, and standard courier services were used. There was a security service present around the rigs, as well as in the offices and storage areas.

11.3.5 Analytical Procedures

From the holes drilled by GBMAG, 156 intervals from 55 holes were sampled and analysed. The samples were prepared by ALS Valencia and then sent on to ALS Vancouver for analysis by a standard ALS "Phosphate package".

GBMAG attempted to setup an on-site laboratory, but this was never implemented fully. The 2011 samples were assayed locally by colorimetry prior to dispatch to the ALS Chemex laboratory in Spain. These results are not included in the resource database used for the 2011 estimate.

The samples are despatched to ALS Chemex in Seville, Spain for further sample preparation as described below:

- They are pulverized using disc mills with steel bowls until 90% of the sample passes a 75 micron (µm) screen;
- A subsample is taken and placed in a Kraft envelope for dispatch to the analytic lab. The amount of material in the envelope is weighed and recorded in the system;
- The preparatory laboratory stores the pulps for GBMAG; and
- Samples are sent to ALS Chemex in Vancouver, Canada for analysis.
- The analytical procedures were:
    - Samples are processed in batches of 40 including one blank, two standards and a duplicate inserted by the laboratory;
    - Fusion with a lithium metaborate flux into a glass disc, followed by XRF for P_2O_5 (including major oxides SiO_2, Al_2O_3, K_2O, Na_2O, MgO, MnO, CaO, TiO_2, P_2O_5, Fe_2O_3 (Phosphate Package));
    - F by alkali fusion and fluorine S.I.E (selective ion electrode);
    - Total carbon and total sulphur using the Leco method;
    - The laboratory stores the pulps rejects for GBMAG; and

- GBMAG receives the assay results from the laboratory via email as Microsoft spreadsheets and PDF scan of the original certificate.

Golder staff visited both of the ALS Chemex laboratories used by GBMAG and carried out an audit of the standards and procedures used. Both ALS labs are ISO 9001 accredited; the Vancouver laboratory also holds ISO/IEC 17025: 2005 accreditation for some procedures.

The Qualified Person is satisfied that the adequacy of the sample preparation, analytical and security procedures described and recorded is satisfactory and that the results meet NI 43-101 standards.

11.3.6 QAQC

The GBMAG QAQC program consisted of field duplicates and standards; no blanks were submitted. In 2009, 103 samples were presented for assaying. Duplicates were conducted on six samples, two international standards were assayed three times. There were no blanks for phosphorus but one of the two standards was low in P_2O_5. In 2011, 53 samples were presented for assaying. Two duplicates were submitted; no standards or blanks were submitted.

The QAQC data is of insufficient quantity to assess the performance statistically.

The Qualified Person is satisfied that the adequacy of the sample preparation, analytical and security procedures described and recorded are satisfactory for the time the analysis was undertaken and that the results meet the prevailing international standards.

11.4 Laboratory Accreditation

Formal accreditations for the various analytical laboratories used throughout the Farim Phosphate Project history include:

- ALS Chemex, Seville, Spain – ISO 9001 accreditation;
- ALS Chemex, Vancouver, Canada – ISO 9001 and ISO/IEC 17025:2005 accreditation;
- BRGM Orleans – COFRAC (Comite Francais D’accreditation) accreditation;
- BRGM Dakar Laboratories – COFRAC accreditation;
- Directorate of Geology and Mines of Guinea-Bissau Laboratory – unable to confirm accreditation in place at time of analytical programs.

12.0 DATA VERIFICATION

12.1 Independent Sampling

Golder did not collect any samples during the May 2015 site visit. During the May 2011 site visit, Golder collected a total of six samples from coarse rejects prepared by GEEEM as part of their sample preparation procedure. Two of the six samples were collected from the same sample, constituting a blind coarse duplicate. These samples were sent to OMAC (now part of ALS Global), an independent laboratory in Ireland, for assaying. Sample preparation involved drying, milling until 85% is less than 75 microns and analysis using XRF for a suite of 12 compounds. Any differences between the assay results of the original and coarse duplicate will result from both sampling error from the drying and grinding stage of sample preparation and analytical error. It will also highlight the effectiveness of the homogenization process carried out by GEEEM during sample preparation.

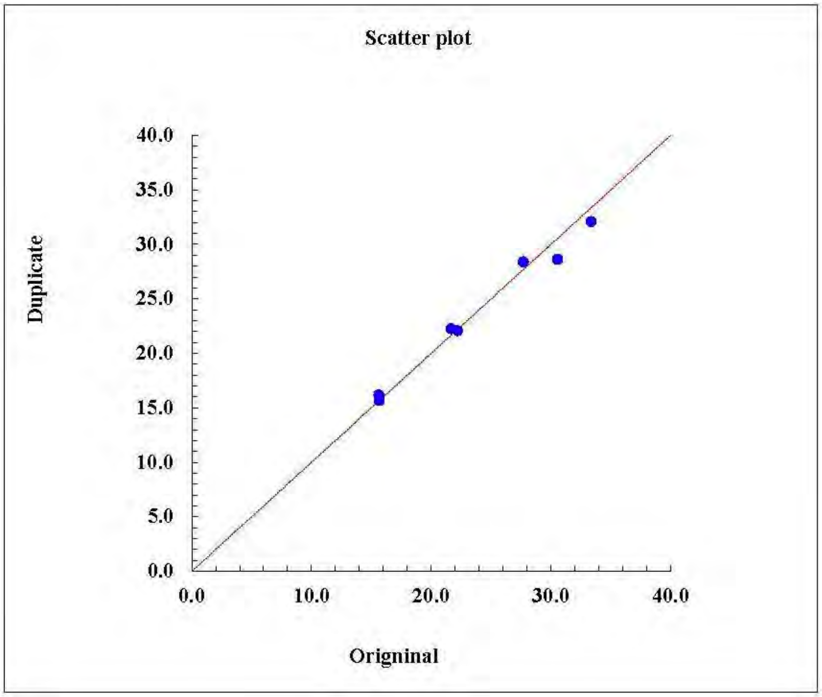
The comparison between the original assay values and the values obtained by the independent assaying carried out by Golder is presented in Table 12-1 and the scatter plot in Figure 12-1. The largest differences in P_2O_5 grade are observed in the two high grade samples, 070 (1.97% difference in P_2O_5 grade) and 086 1.31% difference in P_2O_5 grade with both showing higher grades in the original samples. However, the differences are acceptable and the dataset is small. The lab replicate of 034 shows very little difference, indicating good analytical procedures; however the blind duplicate shows a small difference, possibly indicating some error inherent in the sample preparation.

Overall, the results of the check samples indicate an acceptable level of error. In addition, grades were generally confirmed within acceptable ranges, for example, the P_2O_5 grades all lie between 15% and 34%.

Table 12-1 2011 Independent Sampling - Original vs Coarse Duplicate

Sample No.	Compound (%)									
	Al_2O_3	CaO	Cr_2O_3	Fe_2O_3	K_2O	MgO	Mn_3O_4	P_2O_5	SiO_2	TiO_2
034 original	6.58	27.17	0.05	4.28	0.06	0.19	0.08	21.69	30.51	0.21
034 duplicate	6.28	27.73	0.05	4.01	0.12	0.12	0.05	22.21	30.17	0.28
34 (lab replicate)	6.28	27.56	0.05	3.94	0.11	0.11	0.05	22.05	29.84	0.26
068 original	11.69	31.42	0.05	3.8	0.03	0.18	0.04	27.72	13.11	0.07
068 duplicate	9.89	34.07	0.05	2.77	0.09	0.07	0.01	28.36	8.59	0.09
070 original	1.69	43.2	0.06	5.92	0.06	0.33	0.08	30.56	7.18	0.05
070 duplicate	1.34	43.35	0.06	6.76	0.13	0.25	0.09	28.59	5.99	0.07
086 original	0.92	45.92	0.05	5.38	0.02	0.19	0.06	33.37	4.94	0.02
086 duplicate	0.65	46.39	0.04	4.89	0.10	0.09	0.05	32.06	4.18	0.03
101 original	6.35	20.53	0.04	2.84	0.05	0.16	0.03	15.68	45.83	0.30
101 duplicate	6.56	19.34	0.04	4.57	0.09	0.07	0.02	15.64	41.50	0.34
blind duplicate of 101	6.00	20.81	0.04	2.79	0.11	0.11	0.03	16.15	44.50	0.36

Figure 12-1 Scatter Plot – 2011 Independent Samples, P_2O_5



12.2 Drilling Supervision and Core Logging Check

Historic cores from BRGM and Champion phases of exploration were not available due to being destroyed in the civil war in Guinea-Bissau. During the May 2015 and May 2011 site visits, the QPs viewed a random selection of cores from the GBMAG phases of exploration and compared original logs with the core.

Drilling activity was not in progress during the 2015 Golder site visit and Golders review of drilling, logging and sampling procedures focused on a review of the procedures and methodologies that were used as described by GB Minerals project personnel. During the 2011 site visit Golder supervised drilling of both resource and metallurgical drill holes. Golder is satisfied that the procedures being used are adequate for the style of mineralization (Figure 12-2).

Figure 12-2 Drilling at Farim

During the 2011 core logging review, Golder confirmed that the remaining core matched the information that was recorded in the geological logs and mineralization was observed in each of the drill holes in quantities that were consistent with the logging and general mineralization. No material discrepancies were noted.

12.3 Drill Hole Collar Survey Check

It is not known whether BRGM and Champion drill hole collars were surveyed. The GBMAG drill hole collars were initially surveyed using a hand held GPS, except for the set for the variogram, which was surveyed and levelled locally by a consulting surveyor. In 2011, all drill hole collars were re-surveyed using a GPS system which was accurate to within 0.03 m horizontally and 0.05 m vertically. The surveys were recorded in UTM WGS84, Zone 29N.

During both the 2011 and 2015 site visits Golder inspected a random subset of collars and took measurements of the locations using a hand held GPS (GARMIN GPSmap76CSx and GARMIN GPSmap60CSx). Golder visited a total of 12 drill sites for the purpose of verifying collar coordinate; five (5) drill sites during the 2011 site visit and 12 drill sites during the 2015 site visit. With the exception of one drill hole (KP-SGW-BH01) where differences were in excess of 20 m, no material differences were found between the original and Golder GPS coordinates; all collars plotted within 12 m of the recorded position, which is within the expected accuracy of such equipment.

A table summarizing the validation undertaken by Golder in the respective areas is included as Table 12-2.

Table 12-2 Drill Hole Collar Survey Check

Drill Hole	Golder GPS (m)		Original Survey (m)		Difference (m)		Site Visit
	Easting	Northing	Easting	Northing	Easting	Northing	
SE5	471,708.0	1,380,159.0	471,701.7	1,380,156.8	6.3	2.3	2011
PS2	472,118.0	1,379,843.0	472,118.2	1,379,842.0	-0.2	1.0	2011
SE6	472,055.0	1,379,829.0	472,058.4	1,379,834.0	-3.4	-5.0	2011
BR20	473,829.0	-	473,826.2	-	2.8	-	2011
BR23	472,223.0	1,379,943.0	472,228.6	1,379,935.4	-5.6	7.6	2011
KP-PS-BH05	474,109.0	1,379,444.0	474,101.0	1,379,446.0	8.0	-2.0	2015
KP-PS-BH02	473,605.0	1,379,100.0	473,607.0	1,379,088.0	-2.0	12.0	2015
KP-PS-BH01	473,597.0	1,379,214.0	473,592.0	1,379,219.0	5.0	-5.0	2015
SE10	473,571.0	1,378,511.0	473,572.4	1,378,511.5	-1.4	-0.5	2015
KP-SGW-BH01	474,228.0	1,378,173.0	474,250.0	1,378,160.0	-22.0	13.0	2015
KP-DGW-BH02	472,854.0	1,377,642.0	472,854.0	1,377,635.0	0.0	7.0	2015
SB09	472,243.0	1,377,723.0	472,240.3	1,377,734.0	2.7	-11.0	2015
KP-TMF/OB-BH01	468,019.0	1,378,652.0	468,018.0	1,378,654.0	1.0	-2.0	2015
KP-TMF/OB-BH03	468,405.0	1,377,748.0	468,410.0	1,377,752.0	-5.0	-4.0	2015
SD2	470,241.0	1,380,758.0	470,242.9	1,380,770.0	-1.9	-12.0	2015
SD5	471,377.0	1,379,782.0	471,374.9	1,379,786.0	2.1	-4.0	2015
SG4	471,980.0	1,381,234.0	471,983.6	1,381,234.1	-3.6	-0.1	2015

12.4 Database Integrity Checks

The digital database compiled by GEEEM and supplied by GBMAG consisted of a Microsoft Excel spreadsheet with a single worksheet, detailing for each hole:

- X co-ordinate;
- Y co-ordinate;
- Z co-ordinate;
- FPA from depth, m;
- FPA to depth, m;
- FPA thickness, m;

- Recovery,%;
- SR; and
- P_2O_5, %.

The P_2O_5 grades reported in the database were only the length weighted average per drill hole. The individual assay results for each sample were not detailed. No separate lithology, collar or survey files were supplied.

Golder manipulated the data supplied to produce a Microsoft Access database with four separate tables for assay, collar, lithology and survey information. As the holes are very short, no survey information was provided and the drill holes were assumed to be vertical and not to deviate. A basic lithology file was reconstructed for overburden and FPA only (FPB was not consistently sampled through to the footwall), taking the top of the FPA to be the depth of overburden.

As part of the database validation, photocopies of original geological logs were compared to the digital database. The following checks were carried out:

- Presence or absence of FPA layer;
- “From” and “To”, depths of overburden and FPA layer;
- Overburden and FPA thickness;
- Recovery; and
- P_2O_5 drill hole composite grade.

During the 2011 project Golder visited the GBMAG data room located in the UBS bank in Zurich, Switzerland. Geological logs were only available for the BRGM (1981 and 1983 campaigns) and GEEEM (2009 campaign) drill holes. No logs were supplied for any of the Champion (1998 to1999) holes. The BRGM logs have not only the original lithological log, but also a transcription of the original assay results. No original assay certificates were available for BRGM samples. Digital copies of the assay certificates of the GBMAG holes were provided on-site.

For each BRGM drill hole, Golder digitized the individual sample assay values as written on the geological logging sheet and recalculated a length weighted average per drill hole. This value was compared to the value in the GEEEM database. Numerous discrepancies were noted. In instances where there were differences between the original geological log and the GEEEM database, Golder adopted values calculated from the sample values on the original geological logs. Where discrepancies in “from” or “to” depths or the thickness of the FPA layer were noted, values were retained from the original geological log.

In a few drill holes, the FPA interval was logged, but due to poor drilling recovery (or in some cases other unknown reasons) samples were not assayed for the length of the FPA interval. In these cases, Golder took the conservative approach of reducing the FPA thickness to the sampled interval. This is due to a lack of confidence in the logged thickness (due to the poor recovery) and the unknown phosphate grade. This conservative approach may result in a decrease in the total FPA tonnage.

Length weighted averages were added to the database for other variables (AL_2O_3, CaO, Cr_2O_3, Fe_2O_3, K_2O, MgO, MnO, SiO_2, TiO_2, LOI, F and C).

Golder could not validate or check any of the Champion data.

The assay certificates and geological logs of the GBMAG holes were checked against the digital database for 100% of the samples. Similar checks as listed above were carried out. No material discrepancies were found.

12.5 Limitations to Data Verification

Golder did not actively participate in the implementation of the exploration drilling and sampling programs and site visits were performed outside of the implementation phases of the various drilling programs; therefore, Golder cannot speak to the implementation of exploration drilling, logging, sampling and analytical procedures and methodologies implemented during all phases of exploration work on the Farim Phosphate Project. However, it is Golder's opinion that the verified data and observations are consistent with data and observations collected using the exploration procedures and methodologies provided by GB Minerals and it is reasonable to infer that these processes were in place throughout the various exploration campaigns conducted on the Farim Phosphate Project property.

12.6 Qualified Person Statement on Data Verification

It is Golder's opinion that the exploration data and observations collected from the drill holes and analytical samples that comprise the Farim Phosphate Project geological database have been appropriately verified for the purpose of completing a geological model, estimating Mineral Resources and preparing and NI 43-101 compliant Mineral Resource estimate technical report.

13.0 MINERALS PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The objective of the test work was to quantify the metallurgical response of ore from the Farim Phosphate Deposit. The program was designed to develop the parameters for process design criteria for ore washing/scrubbing, desliming, flotation, and dewatering in the processing plant.

The metallurgical program was conducted by KEMWorks Technology Inc. (KEMWorks), SGS Mineral Services (SGS) and ALS Metallurgy Kamloops (ALS).

The samples used for this testwork were selected to represent the potential mining areas for the first seven years, ore grade, and mineralization types for the South Pit of the Farim deposit.

Five size fractions of the Farim Composite Sample were sent to SGS Lakefield for QEMSCAN analysis. This work confirmed the mineral distributions, mineral release curves, grain size distribution, and chemical analyses by size fractions that were performed by KEMWorks.

Exploratory flotation and scrubbing studies were performed by KEMWorks during 2013 and 2014. This work generated the preliminary test procedure which was the basis to develop a new process flowsheet which eliminates flotation and drastically reduces reagent consumption for the first seven years of mining of the Farim Phosphate Deposit.

13.2 Sample Preparation

The Farim Composite sample consisted of four subsamples, or drill holes, SB9, SC10, SC11, and SE10 with each subsample further subdivided into several cuts corresponding to sequential drilling depths. The subsample composition was based on the block model and assay model data of the deposit and it was considered representative of at least the first seven years of production of the deposit. After discussion and clarification on the handling and analyses of these subsamples, it was decided to select three cuts of each drill hole (top, middle, and bottom) to be sent for chemical analysis. The selected cuts are shaded in

Table 13-1 which shows the drill hole subsample depth and the proportional weight used for sample blending. These cuts were analyzed to confirm the block model assay data of the deposit and to determine the main contaminants in the ore for the first seven years of mining.

The sample preparation procedure was designed to obtain blended composites of each drill hole: SB9, SC10, SC11, and SE10 proportional to the weight of each cut of the corresponding hole. Initially, each cut of subsample was blended and then split in half. One half of each blended subsample cut was then placed in a plastic bag, sealed and stored as a reserve sample. Approximately 50 kg of reserve samples was preserved, while the remaining half of each cut was used to prepare the composites.

In addition to the individual hole composites, a composite of all the subsamples was blended to represent the Farim Phosphate ore for the first seven years. It was prepared based on using the proportional weights of each subsample in Table 13-1. Thus, five samples were obtained: SB9 Composite, SC10 Composite, SC11

Composite, SE10 Composite, and a general composite, called the Farim Composite. Care was taken during this process to maintain the moisture content of each cut by keeping it in sealed containers after blending and splitting. The prepared samples were also stored in sealed containers.

Table 13-1 summarizes the information received and the weights of each cut received along with the proportional weight used for each drill hole.

Table 13-1 Sample Reception and Composite Recipe

SB 9				
Section	**kg**	**Percent of Hole**	**Hole Composite, g**	**Reserve, g**
32,15-32,35	3.0	8.3%	1500	1287
32,35-32,65	4.2	11.7%	2100	2006
32,86-33,08	3.3	9.2%	1650	1653
33,08-33,46	5.4	15.0%	2700	2546
33,46-33,79	4.8	13.3%	2400	2394
33,79-34,09	4.2	11.7%	2100	1990
34,09-34,27	3.0	8.3%	1500	1540
34,50-34,82	4.2	11.7%	2100	2126
34,82-35,12	3.9	10.8%	1950	1924
TOTAL	**36.0**		**18000**	**17466**

SC 10				
Section	**kg**	**Percent of Hole Total**	**Hole Composite, g**	**Reserve, g**
32,24-32,56	3.8	10.6%	1900	1809
32,56-32,86	2.7	7.5%	1350	1406
32,86-33,26	3.8	10.6%	1900	1936
33,26-33,51	2.9	8.1%	1450	1436
33,51-33,73	2.7	7.5%	1350	1280
34,00-34,31	3.6	10.0%	1800	1741
34,31-34,61	2.9	8.1%	1450	1472
34,61-34,91	2.9	8.1%	1450	1511
34,91-35,17	2.7	7.5%	1350	1396
35,17-35,45	2.9	8.1%	1450	1528
35,45-35,73	2.7	7.5%	1350	1366
35,73-35,95	2.3	6.4%	1150	1111
TOTAL	**35.9**		**17950**	**17992**

SC 11				
Section	**kg**	**Percent of Hole Total**	**Hole Composite, g**	**Reserve, g**
30,47-30,82	0.42	9.4%	210	**202**
30,82-31,17	0.47	10.5%	235	235
31,17-31,52	0.47	10.5%	235	252
31,52-31,64	0.16	3.6%	80	82
31,93-32,28	0.39	8.7%	195	240
32,28-32,58	0.37	8.3%	185	89
32,58-32,93	0.39	8.7%	195	245
32,93-33,20	0.34	7.6%	170	154
33,20-33,60	0.47	10.5%	235	219
33,60-34,00	0.53	11.8%	265	259
34,00-34,55	0.47	10.5%	235	110
TOTAL	**4.48**		**2240**	**2085**

SE10				
Section	**kg**	**Percent of Hole Total**	**Hole Composite, g**	**Reserve, g**
30,63-31,11	3.0	16.5%	1500	1360
31,11-31,41	3.0	16.5%	1500	1535
31,42-31,87	2.3	12.7%	1150	1180
31,87-32,20	2.2	11.8%	1075	1106
32,10-32,56	2.6	14.3%	1300	1244
33,46-33,60	0.7	3.9%	350	376
34,33-34,61	1.9	10.5%	950	986
34,61-34,92	1.9	10.5%	950	949
34,90-35,30	0.6	3.3%	300	243
TOTAL	**18.2**		**9075**	**8980**

Shading represents the top, middle, and bottom cuts selected for preliminary analysis before the hole subsamples were blended.

Characterization subsamples and test samples were obtained from each of the prepared hole composites after blending and splitting according to the following scheme:

- Head samples for chemical analysis, 50 g each (wet weight)
- Screen analyses and screen assay, two-500 g (wet weight)
- Test samples of the Farim Composite, each split of 610 g (wet weight)

Head sample chemical analyses were conducted on all four hole composites, but only the Farim Composite will be discussed in this report. The individual hole composite data can be found in Appendix B.

13.3 Ore Characterization

The characterization studies included head sample chemical analysis, screen analysis, screen assays, and mineralogical studies (QEMSCAN) by SGS. QEMSCAN tests were carried out on selected size fractions obtained from the screen analysis which included +1.18 mm, 1.18x0.425 mm, 0.425x0.106 mm, 0.106x0.020 mm and -0.020 mm size fractions. The QEMSCAN results are in agreement with the interpretation and conclusions of the screen assays results.

In order to demonstrate that the Farim Composite sample was representative of the first seven years of mined phosphate ore, the subsamples of each drill hole were submitted to chemical analysis that included three selected cuts of each subsample. Table 13-2 shows the chemical analyses of the selected cuts from the drill holes. The individual drill hole composite samples and the Farim composite were also submitted for chemical analysis as seen in Table 13-3.

The chemical analyses of the SB9, SC10, SC11, and SE10 composites do correspond to the selected cuts as well as the Farim Composite.

Table 13-2 Chemical Analysis of Selected Cuts

	SB 9			SC 10			SC 11			SE 10		
Sample Identification	**Top**	**Middle**	**Bottom**	**Top**	**Middle**	**Bottom**	**Top**	**Middle**	**Bottom**	**Top**	**Middle**	**Bottom**
Phosphorus - P_2O_5	35.28	30.83	26.82	31.26	33.35	31.11	30.30	33.26	31.13	29.90	35.54	30.72
Aluminum - Al_2O_3	0.78	1.46	0.59	2.26	1.21	0.68	2.96	0.72	1.79	1.06	0.50	0.81
Iron - Fe2O3	1.85	2.00	1.58	3.30	3.11	2.60	2.21	1.16	2.58	3.57	1.17	4.45
Sulfur (S), Total	0.95	1.12	0.99	2.11	1.67	1.78	1.39	0.91	1.66	1.01	0.80	0.91
Pyritic Sulfur (S)	0.73	0.92	0.55	1.63	1.41	1.28	1.06	0.73	1.24	0.68	0.50	0.38
Pyritic Iron	1.18	1.39	1.23	2.63	2.08	2.22	1.73	1.13	2.07	1.26	1.00	1.13
Calcium - CaO	49.57	43.86	46.74	43.75	47.21	46.00	41.81	47.85	44.73	40.68	51.90	46.27
Magnesium - MgO	0.02	0.32	3.70	0.02	0.19	0.26	0.08	0.03	0.41	0.17	0.03	0.53
Acid Insolubles	4.46	11.27	0.88	9.69	4.30	0.94	10.99	9.86	5.72	11.59	3.92	2.31
MER	0.075	0.123	0.219	0.179	0.135	0.114	0.173	0.057	0.154	0.161	0.048	0.188
Adjusted MER *	0.042	0.077	0.173	0.094	0.073	0.043	0.116	0.023	0.087	0.118	0.020	0.152

MER* is the adjusted MER (minor element ratio) to account for iron present as pyrite which is insoluble and does not contribute to MER. It is calculated by removing the pyritic iron from the total iron present in the sample. The pyritic iron value is calculated from the amount of pyritic sulfur in the sample:

% Fe_2O_3 pyritic = % S pyritic x (160 / 128).

Then the MER* is calculated by:

(% Al_2O_3 + (% Fe_2O_3 - % Fe_2O_3 pyritic) + % MgO) / % P_2O_5

Table 13-3 Hole Composite Sample Analysis

Sample Description	SB9	SC10	SC11	SE10	Composite
Phosphorus - ICP - P_2O_5	30.99	35.03	34.51	32.44	33.42
Aluminum - Al_2O_3	0.87	0.92	1.15	1.01	1.17
Iron - Fe_2O_3	2.26	1.88	1.95	3.44	2.53
Sulfur (S), Total	1.32	1.43	1.56	1.12	1.36
Pyritic Sulfur (S)	0.95	1.03	1.09	0.71	0.95
$S_{pyritic}/S_{total}$ %	71.97	72.03	69.87	63.39	69.85
Pyritic Iron*	1.18	1.28	1.36	0.88	1.18
Calcium - CaO	46.13	49.52	48.44	46.04	47.57
Magnesium - MgO	0.85	0.13	0.09	0.24	0.32
Acid Insolubles	2.15	1.85	3.88	4.22	4.29
CaO/P_2O_5	1.49	1.41	1.40	1.42	1.42
MER	0.128	0.084	0.092	0.145	0.120
Adjusted MER *	0.090	0.047	0.053	0.117	0.085
Grade Potential, % P_2O_5	33.2	37.3	37.7	36.0	36.9

13.3.1 Head Sample Chemical Analysis

Three different Farim Composite samples were prepared and sent for chemical analysis throughout the test work process.

Table 13-4 presents the results and the parameters of interest, such as CaO/P_2O_5 ratio, MER, adjusted MER (MER*), and grade potential. It is clear that the analyses are within experimental and analytical error considering that some of the elements and compounds analyzed were calculated from elemental analysis. These results show that the Composite P_2O_5 grade was 33.0% ± 0.7% for a 2.0% error. Since a 5% error for analysis is considered reasonable, it is expected that ±1.7% P_2O_5 results could be obtained on any given sample. Thus, P_2O_5 grade can be expected to range between 31.5% and 34.5%.

Examining the main impurities, A.I., Fe_2O_3, and Al_2O_3; the error is higher. But considering the analytical techniques used for analysis, the sample preparation procedure, and the absolute value range, these results are acceptable.

Table 13-4 Head Sample Chemical Analysis

Composite Sample	P_2O_5 %	CaO %	Acid Insol %	Al_2O_3 %	Fe_2O_3 %	MgO %	S_{total} %	$S_{pyritic}$ %	Moisture
1	32.27	43.51	5.47	1.02	2.51	1.49	1.30	0.90	24.49
2	33.44	45.42	4.95	1.01	3.73	0.19	2.30	1.90	22.86
3	33.42	47.57	4.29	1.17	2.53	0.32	1.36	0.95	--
Average	33.04	45.50	4.90	1.07	2.92	0.67	1.65	1.25	23.68
Std. Dev.	0.67	2.03	0.59	0.09	0.70	0.72	0.56	0.56	1.15
Error, %	2.03	4.46	12.06	8.40	23.90	107.40	33.92	45.08	4.87

Composite Sample	CaO/P_2O_5	MER	Grade Potential P_2O_5, %	$S_{py.}/S_{total}$ %	Fe_2O_3 Pyritic, %	$Fe_2O_3^*$ Pyritic, %	Adjusted MER
1	1.35	0.16	36.05	69.23	1.12	1.62	0.12
2	1.36	0.15	37.11	82.61	2.37	2.86	0.08
3	1.42	0.12	36.45	69.85	1.18	1.69	0.08
Average	1.38	0.14	36.54	73.90	1.56	2.06	0.09
Std. Dev.	0.04	0.02	0.53	7.55	0.70	0.70	0.02
Error, %			1.46				

13.3.2 Screen Analysis

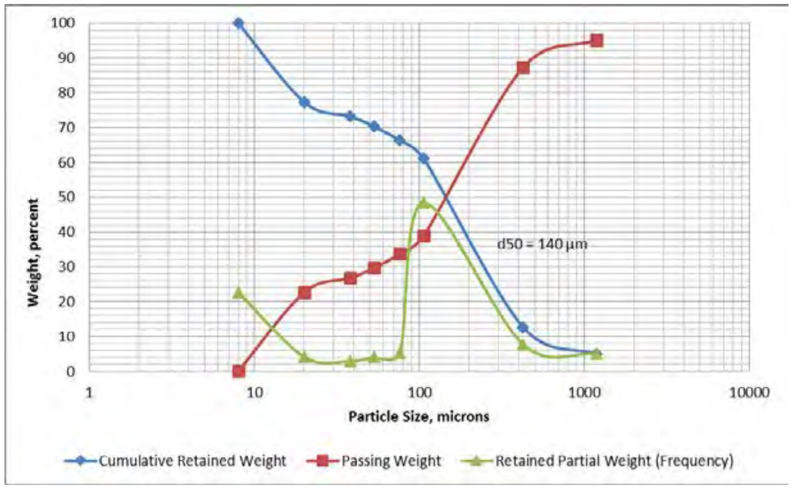
Table 13-5 and Figure 13-1 show the Frequency and Cumulative Retained and Passing distributions as a function of particle size, the particle size distribution (PSD) of the Farim Composite. The results show that the Mean Particle Size, d_{50}, is 140 µm which shows a single mode distribution (unimodal), the mode being at 106 µm (150 mesh), retaining 48.4% of the weight.

Thus, it is expected that the weight distribution dominates the system differences in the Frequency and Cumulative distributions of the different values analyzed (Screen Assays). Significant changes in the phosphate ore composition as a function of particle size due to accumulation of certain impurities may be difficult to observe.

Table 13-5 Particle Size Distribution

US Mesh	Opening, µm	Retained Weight, g	Retained Weight, %	Cumulative Retained Weight, %	Passing Weight, %
16	1180	19.0	4.91	4.91	95.09
16x40	425	30.1	7.77	12.68	87.32
40x140	106	187.6	48.44	61.12	38.88
140x200	76	20.1	5.19	66.31	33.69
200x270	53	15.5	4.00	70.31	29.69
200x400	38	11.3	2.92	73.22	26.78
400x635	20	16.0	4.13	77.36	22.64
-635	8	87.7	22.64	100.00	0.00
Total		387.3	100.0		

Figure 13-1 Cumulative Retained and Passing Particle Size Distribution



13.3.3 Screen Assays

The results of the screen assays are shown in Figure 13-2 to Figure 13-5. Figure 13-2 presents the grades as a function of particle size for P_2O_5, A.I., Al_2O_3, Fe_2O_3, MgO, S_{total}, and $S_{pyritic}$. The loci of the curves indicate that aluminum silicates are present since the loci of the curves for Al_2O_3 and MgO are virtually identical. Fe_2O_3, S_{total}, and $S_{pyritic}$ showed similar curves; the difference in the Fe_2O_3 may indicate that these aluminum silicates may contain some Fe. The locus of the A.I. curve shows a different shape, whereas Al_2O_3, MgO, Fe_2O_3, S_{total}, and $S_{pyritic}$ increase at particle sizes greater than 0.425 mm and finer than 0.053 mm. A.I. is almost flat for particles larger than 0.106 mm and decreasing for particles smaller than 0.106 mm. The cumulative grades are presented in Figure 13-3 that shows this trend.

Figure 13-4 shows the Frequency Distribution for P_2O_5, A.I., Al_2O_3, Fe_2O_3, MgO, S_{total}, and $S_{pyritic}$ as a function of particle size. This figure indicates that the Cumulative Weight Retained Distribution dominates this system; the variation in grades of the different compounds not being enough to modify the Weight Frequency Distribution (Figure 13-1) significantly. Figure 13-5 shows the Cumulative Distribution for all compounds studied as a function of particle size. The Cumulative Distribution of Fe_2O_3, S_{total}, and $S_{pyritic}$ shows that they accumulated in the +0.106 mm size fraction, whereas Al_2O_3 and MgO steadily increase over the whole range of particle sizes studied. The loci of the curves for P_2O_5 and A.I. follow a similar trend, indicating that A.I. is the most critical impurity and may be associated with francolite requiring liberation and separation by scrubbing, desliming, and sizing. It also indicates that more selective methods of separation, such as flotation, may also be required.

Figure 13-2 Grades as a Function of Particle Size

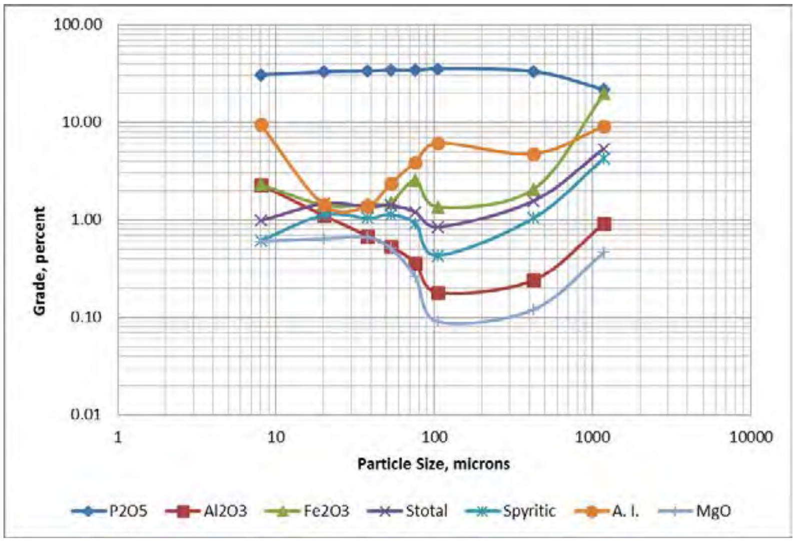


Figure 13-3 Cumulative Grades as a Function of Particle Size

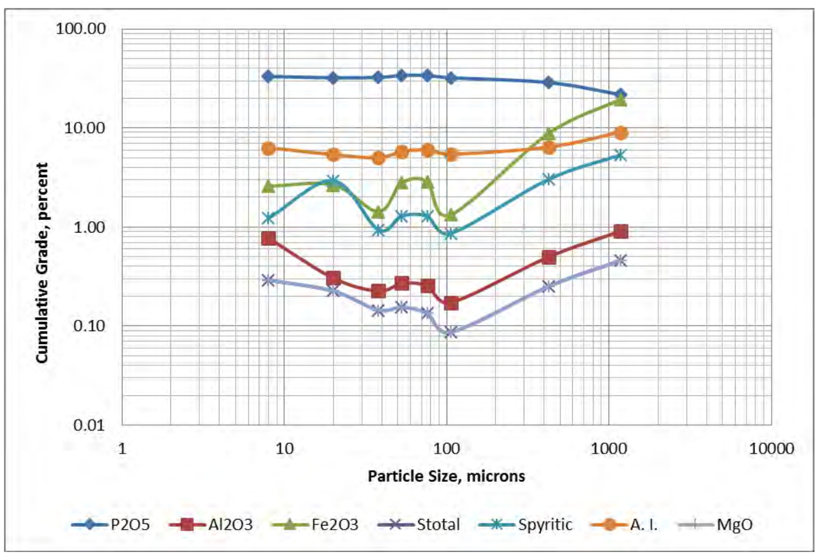


Figure 13-4 Frequency Distribution as a Function of Particle Size

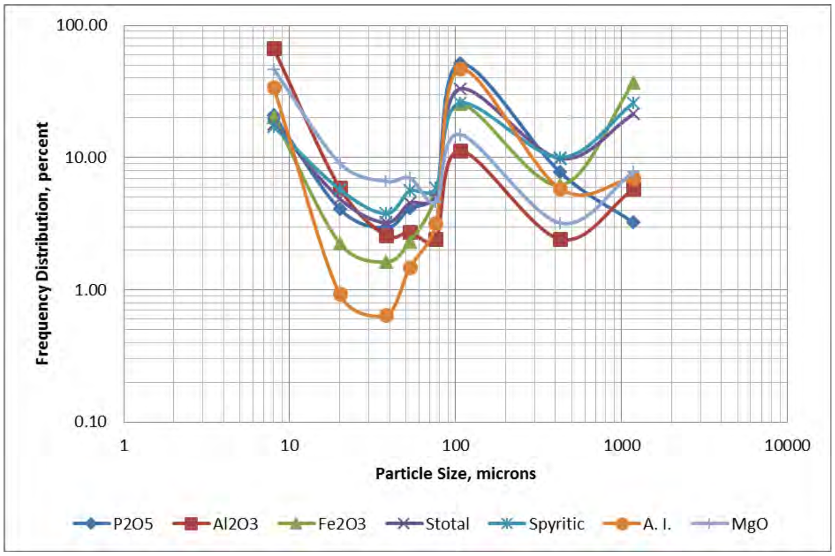
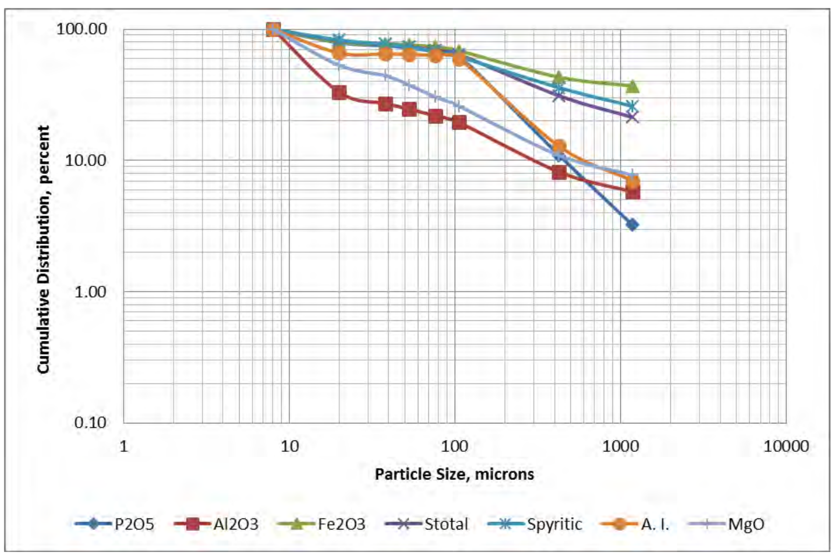


Figure 13-5 Cumulative Distribution of as a Function of Particle Size



13.3.4 QEMSCAN Analysis Report Executive Summary from SGS Report

One feed composite sample labelled Farim Comp was submitted to the Mineral Services group within SGS for mineralogical characterization using QEMSCAN technology, chemical analysis, electron microprobe analysis (EMPA), and X-Ray Diffraction (XRD). This mineralogical characterization was originally requested by Marten Walters, from KEMWorks Technology, on behalf of Lycopodium Minerals Canada Ltd. The objective of this investigation was to determine the mineral assemblage of each sample, the liberation characteristics of the apatite, silicates, carbonates, oxides, and sulphides.

To aid with this objective, the deliverables from this size by size mineralogical study include:

- the mineral abundance of the sample (by size fraction),
- the liberation and association information of total apatite, silicates, oxides, sulphides, and
- carbonate minerals,
- determinative mineralogical parameters such as:
    - mineral release curves,
    - mineralogically limiting grade recovery curves, and
- grain size data.

The sample preparation and the details of the results are discussed in the main body of the report. Some points of interest are discussed in this summary.

Mass Distributions and Elemental Chemical Data

The mass distributions and elemental chemical data by size fraction are summarized in Table 13-6. Note the higher abundance of aluminum and silicate in the -20 µm fraction and the much higher concentration of iron in the +1,180 µm fraction.

Table 13-6 Size Fractions for Analysis and Mass Distribution (%) of the Farim Comp

Fraction	Combined	+1180 µm	- 1180/ +425µm	-425/ +106 µm	-106/ +20 µm	-20 µm
Mass Size Distribution (%)	**100.0**	**15.5**	**19.3**	**26.0**	**15.4**	**23.8**
Mg (Chemical)	0.25	0.56	0.07	0.05	0.27	0.39
Al (Chemical)	0.70	0.39	0.13	0.12	0.40	2.20
Si (Chemical)	3.26	3.13	2.21	3.73	1.77	4.67
P (Chemical)	13.0	6.59	14.6	14.8	14.7	12.8
S (Chemical)	1.20	2.17	1.46	0.76	1.21	0.82
K (Chemical)	0.04	0.02	0.01	0.01	0.02	0.11
Ca (Chemical)	31.1	16.9	34.6	34.9	35.3	30.4
Fe (Chemical)	4.88	22.0	2.88	0.93	1.87	1.57

Mineral Abundances

A summary of the mineral abundances is discussed below.

- Calculated Head
    - The apatite content is 74.4%.
    - The "Apatite Impure" category accounts for 12.8% and predominately occurs in the -20 µm size fraction.
    - The gangue minerals are mainly:
        - quartz (3.13 wt%)
        - Fe-oxides (5.58 wt%)
        - dolomite (0.50 wt%)
        - pyrite (2.83 wt%).
- Size by Size Mineral Distributions
    - Apatite abundance is highest in the +106 µm size fraction (91.2%) and the least in the -20 µm size fraction (48.3%).
    - The Fe-oxide content is much higher in the +1,180 µm fraction and accounts for ~28% by mass. This correlates well with the higher iron assay in this fraction.
    - Pyrite content is also highest in the +1,180 µm fraction and also correlated well with the sulphur assay.
    - The apatite impure phase is mainly composed of Ca-phosphate but it can have high levels of impurities. Aluminum and silica are the main ones bit it can also contain low levels of potassium & magnesium. This phase mainly occurs in the -20 µm fraction accounting for 48.9%.

EMPA

The data from the electron microprobe analysis (EMPA) indicates that the average P_2O_5 content of the apatite is 37.21%. If a perfect concentrate of apatite was produced, this would be close to the maximum P_2O_5 grade that could be achieved. The EMPA also reveals that apatite contains significant SO_2 and Fluorine at ~0.65% and 4.72%, respectively.

Liberation and Grain Size

The liberation of the "Apatite Total" (which combines the apatite and apatite impure as one mineral group) is good, accounting for 96% (both "free" and "liberated" combined) of the calculated head. With the exception of the +1,180 µm size fraction, apatite liberation is very good in each of the other fractions. The non-liberated apatite particles are generally associated with the complex mineral class.

The calculated head for the carbonate liberation is poor, at 28%. The size by size liberation profiles of the carbonates shows poor liberation at the coarser sizes. Liberation generally increases with decreasing particle size. The non-liberated carbonate grains are commonly associated with the complex grains.

The liberation of the silicates for the comp is good, accounting for 77% (both “free” and “liberated” combined) of the calculated head. The liberation is poor in the +1,180 µm size fraction (13%) but is good in the remaining size fractions.

By mass, the oxide and sulphide are most abundant in the +1,180 µm size fraction and show poor liberation.

Grade-Recovery

Grade-recoveries are calculated based on the liberation and chemistry (EMPA) of apatite. The mineralogical limiting grade recovery curves indicate that an 80% apatite recovery for a theoretical maximum P_2O_5 concentrate grade of 36%, respectively, would be possible at this grind target.

13.4 Horizontal scrubbing tests

Based on the interpretation of the Characterization Studies results for the Farim Composite sample Horizontal Scrubbing (Drum) tests were conducted to determine if the major impurities could be rejected. For this purpose, the Farim Composite sample was first submitted to the standard scrubbing procedure developed by KEMWorks in the exploratory testing phase which included horizontal and attrition scrubbing as a baseline. Then, the horizontal scrubbing tests were performed at varying conditions to determine the optimum operating conditions for the Farim Composite sample.

13.4.1 Standard Scrubbing – Baseline

The standard baseline scrubbing test consists of a horizontal scrubbing step at 50% solids content for 5 minutes. Then, the +6.3 mm size fraction is screened out (reject), dried and weighed; and the -6.3 mm material is dewatered before being submitted to attrition scrubbing. It was observed during this stage that the Farim Composite sample contains heavy clays that do not allow for an increase in the solids content of the slurry beyond 41% by weight.

The dewatered Farim Composite sample was then attrition scrubbed for 10 minutes at 560 rpm and 41% solids. The product was screened at 1.18 mm, 0.425 mm, 0.106 mm and 0.020 mm to obtain the 6.3x1.18 mm, 1.18x0.425 mm, 0.425x0.106 mm, 0.106x0.020 mm, and -0.020 mm size fractions.

Even though this test proved the operating conditions of both horizontal scrubbing and attrition scrubbing were not ideal for the Farim Composite, the results were encouraging. Using the screen assay of the new scrubbed product, it was clear that by rejecting the +1.18 mm material and the -0.020 mm size fraction, the highest P_2O_5 and CaO grades were obtained with the lowest level of impurities with the exception of the A.I. (see Figure 13-6 and Figure 13-7).

Figure 13-8 and Figure 13-9 present the Frequency and Cumulative Distributions of P_2O_5, Al_2O_3, Fe_2O_3, S_{total}, and $S_{pyritic}$, CaO, A.I., and MgO as a function of particle size. Clearly, the Weight Frequency Distribution dominates the system, but it also shows that P_2O_5, CaO, and A.I. values are lower above the 1.18 mm and below the -0.020 mm size fractions. Figure 13-9 shows lower cumulative recoveries of Al_2O_3, P_2O_5, and A.I. above 1.18 mm (reject), but higher recoveries between 1.18 mm and 0.020 mm.

In summary, it was possible to increase the P_2O_5 grade to 33.4% (an increase of 1.3% P_2O_5 in grade) with a mass yield of 68.5%, and P_2O_5 recovery of 73.3%. The parameters obtained were:

- CaO/P_2O_5 ratio 1.4
- MER 0.102
- MER* 0.035
- P_2O_5 grade potential 36.8%

The presence of large amount of clay material in the ore results in a cushioning effect and a high viscosity of slurry in the scrubbing stages. It was cautiously inferred that by horizontal scrubbing under the right conditions, then desliming at 75 µm followed by attrition scrubbing, the 1.180x75 µm size fraction would result in a higher P_2O_5 grade and recovery. However, the presence of high A.I. in the 425x106 µm size fraction was also taken into account and would require special treatment to achieve the target 36% P_2O_5 grade. As a result reverse flotation was considered to remove the A.I.

Figure 13-6 Grades as a Function of Particle Size after Baseline Horizontal Scrubbing

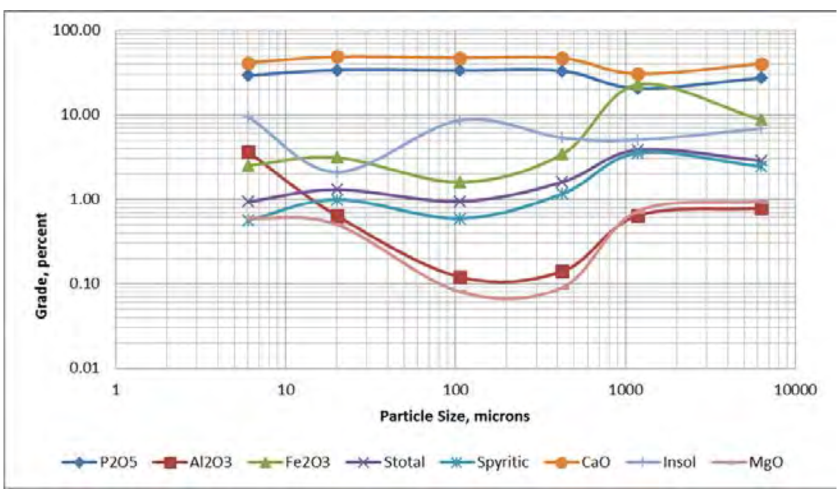


Figure 13-7 Cumulative Grades as a Function of Particle Size after Baseline Horizontal Scrubbing

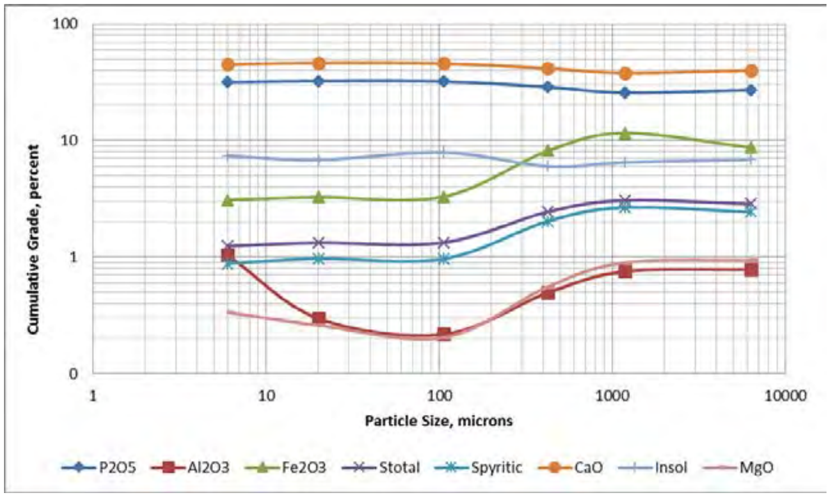


Figure 13-8 Frequency Distribution as a Function of Particle Size after Baseline Horizontal Scrubbing

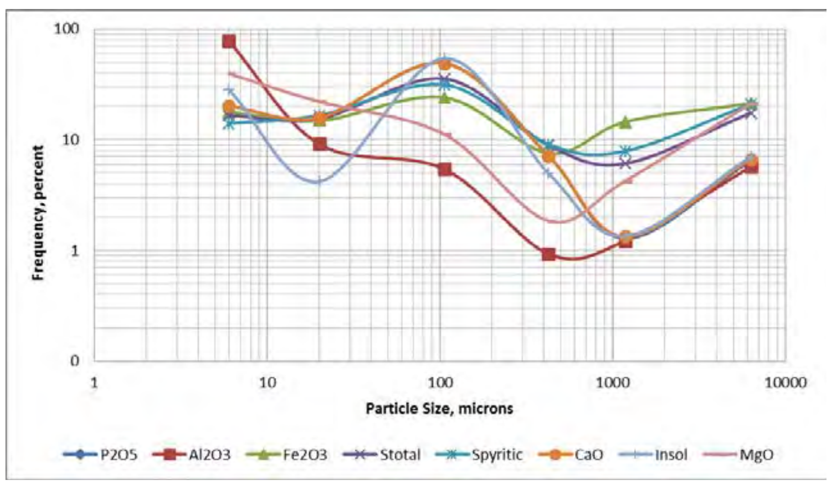
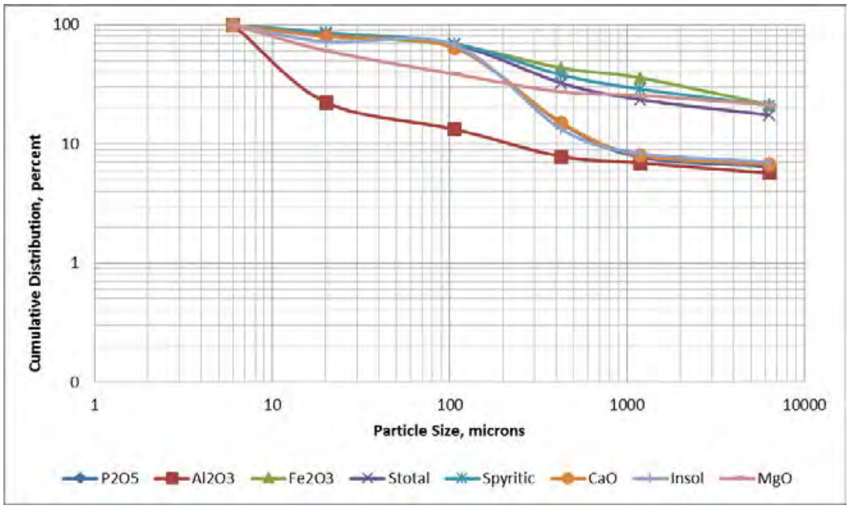


Figure 13-9 Cumulative Distribution as a Function of Particle Size after Baseline Horizontal Scrubbing



13.4.2 Effect of Horizontal Scrubbing Time at 35% and 50% solids Content

For these tests, the samples were submitted to horizontal scrubbing for 150 seconds (2.5 minutes), 300 seconds (5 minutes), and 600 seconds (10 minutes) at 35% and 50% solids content. After each test, a screen assay was carried out on selected size fractions to observe the behavior of the P_2O_5, CaO, A.I., and impurities (Al_2O_3, Fe_2O_3, S_{total}, $S_{pyritic}$, and MgO) contents. In general, A.I., Al_2O_3, Fe_2O_3, S_{total}, $S_{pyritic}$, and MgO decreased in the product size range of 1.18x0.020 mm as the scrubbing time was increased.

At 50% solids content, the horizontal scrubbing resulted in a higher mass yield (72.6%), P_2O_5 recovery (75.9%), and P_2O_5 grade (33.7%) after 10 minutes of scrubbing than at lower scrubbing times. However, at 35% solids content and 5 minutes of scrubbing time the highest mass yield (73.7%), P_2O_5 recovery (77.3%), and P_2O_5 grade (34.4%) were obtained. Apparently, the kinetics of scrubbing increased at 35% solids content which resulted in a better product. These results also showed that at short scrubbing time (2.5 minutes) the yield and P_2O_5 recovery are the lowest due to P_2O_5 losses in the +6 mm and +1.18 mm size fractions. At 10 minutes of scrubbing time, the P_2O_5 losses occurred due to the abrasion of the P_2O_5 particles into the -0.020 mm size fraction.

At 50% solids content, a cushioning effect by the slimes prevented the abrasion of the P_2O_5 particle surfaces. As a result, the yield, P_2O_5 recovery and grade were still increasing after 10 minutes of scrubbing time. At 35% solids content, the abrasive effect on the P_2O_5 particles was observed in the mass yield, P_2O_5 recovery and P_2O_5 grade. A maximum of these values was observed after 5 minutes scrubbing and decreased at 10 minutes of scrubbing time. The results were normalized based on the feed grades of each test to eliminate the effect of small differences in feed grade that could be misleading in the interpretation of the results. Using the normalized feed grades, the Horizontal Scrubbing tests were analyzed and the results are presented in Appendix B.

Figure 13-10 presents the mass yield and P_2O_5 recovery as a function of scrubbing time at 35% and 50% solids content. These results show that the loci of the yield and P_2O_5 recovery curves for 35% solids content were higher, indicating a more efficient process. This figure also shows that at 300 seconds (5 minutes) the mass yield and P_2O_5 recovery at 35% solids content levels off, whereas at 50% solids content, both the yield and P_2O_5 recovery is still increasing. The results of 50% solids content are still considered inferior to those obtained at 35% solids content and 300 seconds (5 minutes).

The P_2O_5 grade, grade potential and the A.I. grade as a function of scrubbing time is presented in Figure 13-11 for 35% and 50% solids content. Again, the results show that high P_2O_5 grade and grade potential are obtained at 35% solids content and 300 seconds (5 minutes) of horizontal scrubbing time. The P_2O_5 grade and grade potential slightly decrease at higher scrubbing times at both solids content studied. As expected, the lowest A.I. grade is obtained by horizontal scrubbing at 35% solids content for 300 seconds. Figure 13-12 presents the CaO/P_2O_5 ratio and MER* as a function of scrubbing time at 35% and 50% solids content. The results show that the CaO/P_2O_5 ratio did not change for all the tests carried out at both 35% and 50% solids content. This was expected since no significant amounts of carbonates are present in the ore. The MER* showed a continuous decrease for both 35% and 50% solids content as scrubbing time increased. This may be due to the liberation of fine pyrite and aluminum silicates at a faster rate than the increase in P_2O_5 grade.

The normalized P_2O_5 grade, grade potential, A.I. grade, the normalized CaO/P_2O_5 ratio and MER* parameters as a function of horizontal scrubbing time are presented in Figure 13-13 and Figure 13-14.

When the P_2O_5 grade and grade potential are normalized with respect to their corresponding feed grades, the results are marginally better at 50% solids content than those obtained at 35% solids content while the normalized A.I. grade is lower at 50% solids content (see Figure 13-13). However, Figure 13-14 shows that the CaO/P_2O_5 ratio did not change for all the tests carried out, but the MER* was significantly better at 35% solids content.

In summary, the results show that horizontal scrubbing at 35% solids content for 300 seconds (5 minutes) renders the highest mass yield of 73.7%, the highest P_2O_5 grade of 34.4% and the highest P_2O_5 recovery of 77.3%. As a result, the operating conditions for the horizontal scrubbing stage in the bench scale tests were set for 300 seconds (5 minutes) at 35% solids content.

Figure 13-10 Yield and P_2O_5 Recovery as a Function of Horizontal Scrubbing Time

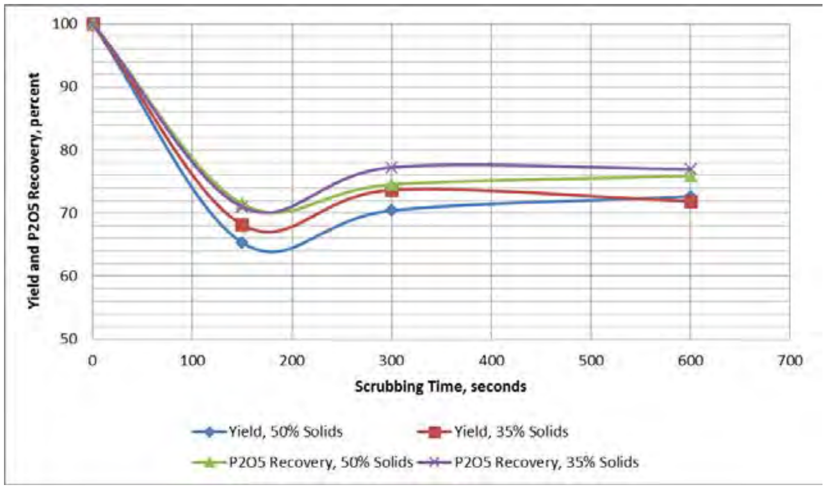


Figure 13-11 Grades as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content

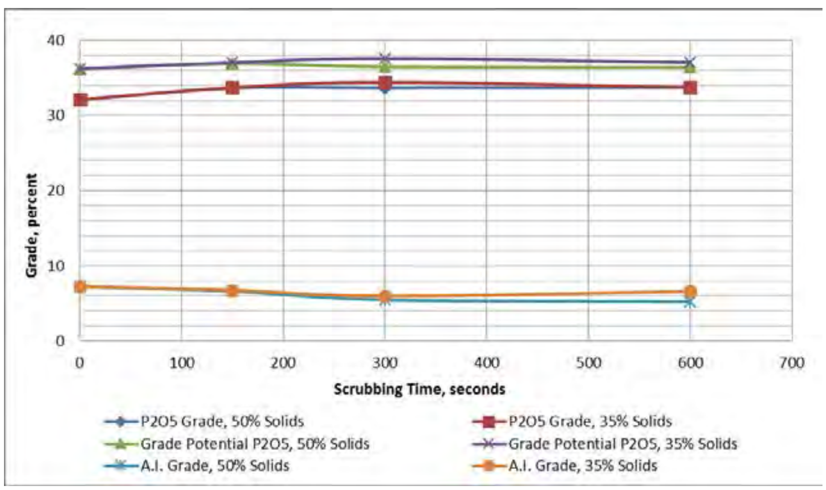


Figure 13-12 CaO/ P_2O_5 Ratio and MER* as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content

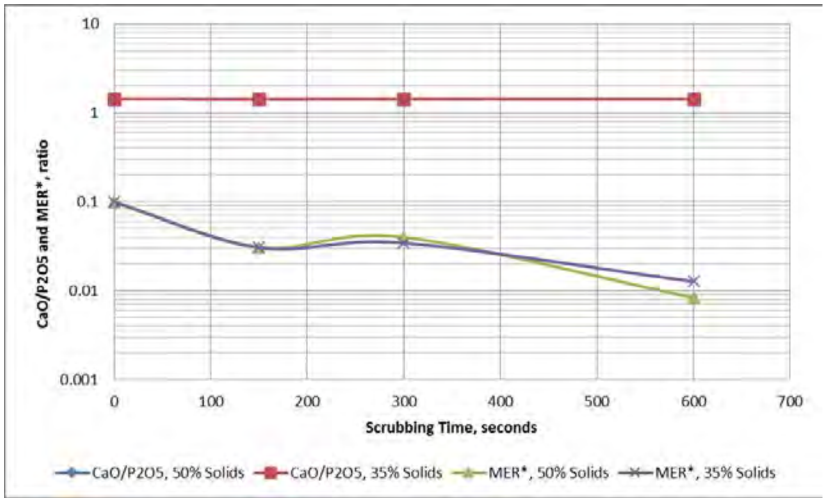


Figure 13-13 Normalized Grades as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content

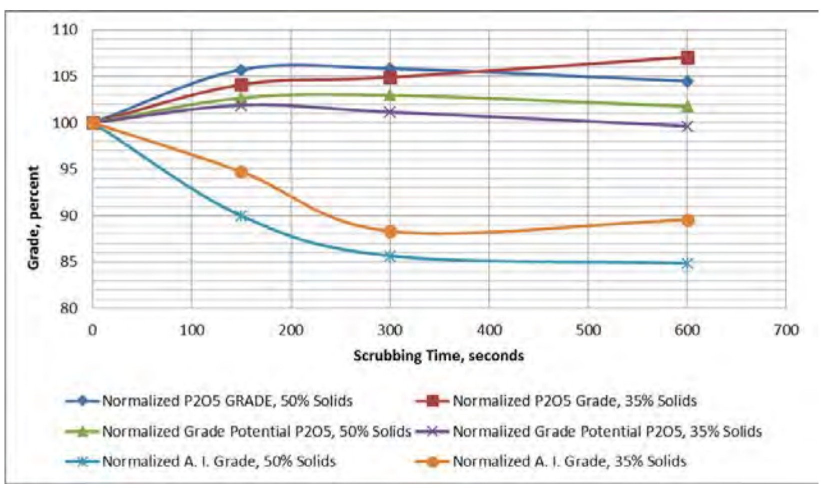
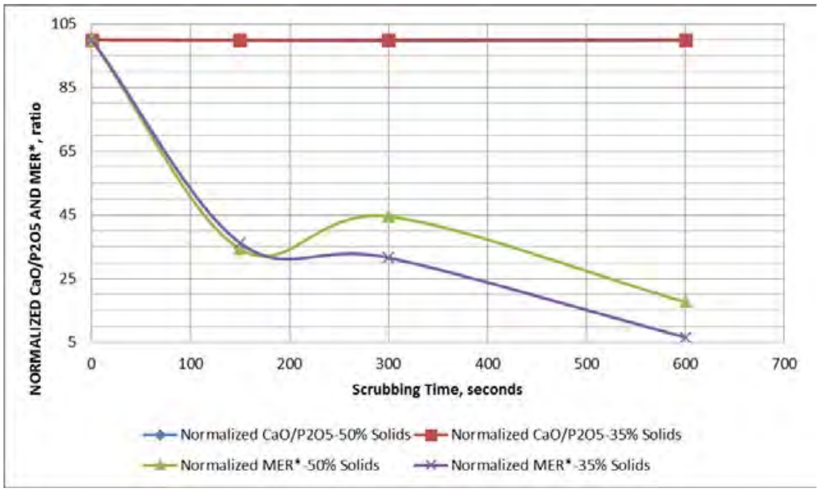


Figure 13-14 Normalized CaO/ P_2O_5 Ratio and MER* as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content



13.4.3 Confirmation Test

The initial horizontal scrubbing of the Farim Phosphate ore is of utmost importance to successfully achieve the maximum P_2O_5 grade in the beneficiated product with the lowest MER* possible. A confirmation test was conducted using these conditions: 35% solids content for 300 seconds (5 minutes) using the same drum as in the previous tests at 50% of the critical speed (36.8 rpm).

Appendix B contains the complete information from the Horizontal Scrubbing tests and Confirmation Test.

While there were small differences in the feed grades of P_2O_5, CaO, MgO, Al_2O_3, Fe_2O_3, S_{total}, $S_{pyritic}$, and A.I. for Test HS #5 and Test HS #7, the screen assays for these tests produced similar grades, Cumulative grades, Frequency Distributions and Cumulative Distributions as a function of particle size for the different compounds considered. This indicated that the horizontal scrubbing design produced for this sample resulted in reproducible results.

For the comparison of results, it was considered at this stage that the 1.18x0.020 mm size fraction was product, the 6.3x1.18 mm was considered reject, the 0.075x0.020 mm size fraction part of the fine product, and the material finer than 0.020 mm was considered slimes (tailings). The results obtained from Test HS #7 Confirmation Test for the 1.18x0.020 mm size fraction is summarized in Table 13-6.

Test HS #5 is also included in this table for comparison. The data in Table 13-7 and Table 13-8 show that the results of Test HS #7 were virtually identical to those obtained at the selected conditions (Test HS #5) with the error being within the acceptable 1% margin.

Comparing the mass yields, the difference in the results was -0.3% with a difference in P_2O_5 grade of 0.3% P_2O_5, resulting in a difference in the A.I. grade of -0.4% in the 1.18x0.020 mm product.

The P_2O_5 recovery difference was -0.4%, whereas the A.I. rejection increased by 1.3%. The beneficiation parameters were also similar: the CaO/P_2O_5 ratio was 1.421 for the HS #7 tests and 1.430 for the HS #5 test, the MER was 0.100 and 0.103, the MER* was 0.027 and 0.034 for Test HS #7 and HS #5, respectively. The difference in P_2O_5 grade potential for these tests was 0.1% P_2O_5.

Table 13-7 Wet Horizontal Scrubbing Results at 35% Solids Content for 5 Minutes

Time (seconds)	Test Number	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Cum. Grades							
							P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
300	HS #5	1180x20	348.00	73.36	73.36	26.64	34.72	49.34	0.16	0.28	1.99	1.19	0.79	5.53
300	HS #7	1180x20	349.20	73.66	73.66	26.34	34.40	49.19	0.18	0.25	2.18	1.14	0.54	5.97

Cum. Distribution								CaO/P_2O_5	MER	MER*	Grade Pot. P_2O_5, %
P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %				
76.87	77.33	43.53	20.90	52.78	64.53	61.37	63.80	1.421	0.100	0.027	37.65
77.26	77.24	44.12	19.31	40.57	64.67	53.81	65.05	1.430	0.103	0.034	37.55

Table 13-8 Normalized Wet Horizontal Scrubbing Results at 35% Solids Content for 5 Minutes

Time (seconds)	Test Number	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Cum. Grades							
							P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
300	HS #5	1180x20	348.00	73.36	73.36	26.64	104.79	105.41	59.35	28.49	71.95	87.97	83.66	86.97
300	HS #7	1180x20	349.20	73.66	73.66	26.34	104.89	104.87	59.90	26.22	55.08	87.80	73.05	88.31

Cum. Distribution								CaO/P_2O_5	MER	MER*	Grade Pot. P_2O_5, %
P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %				
76.87	77.33	43.53	20.90	52.78	64.53	61.37	63.80	100.596	82.249	38.668	102.17
77.26	77.24	44.12	17.62	51.14	73.66	73.66	65.05	99.976	65.397	31.561	101.16

The Normalized data with respect to the corresponding feed grades of Tests HS #5 and HS #7 confirms that the results are similar and independent of the small difference in feed. Thus, the results are reproducible and the horizontal scrubbing process is robust and applicable to the Farim deposit.

13.5 Attrition Scrubbing Studies

After setting the operating conditions for the horizontal scrubbing stage to reject clay balls and iron bearing coarse particles, and releasing fine aluminum silicates particles into the fine size fractions (minus 75 µm size fraction), it was found that significant amounts of quartz, clay, and iron bearing minerals remained in the 6.3x0.075 mm size fraction. It was apparent that most of these impurities were attached to the surface of the phosphate particles. Therefore, it became necessary to further scrub the surfaces of the phosphate particles to release the quartz attached to the francolite, the coarse iron bearing minerals, and to clean the surfaces of the phosphate bearing minerals of any remaining clays. This discovery required a more intensive energy scrubbing process. Thus, the 6.3x0.075 mm size fraction was submitted to attrition scrubbing. The objectives of this unit operation were to:

- Reject coarse iron bearing minerals with minimum phosphate losses;
- Release the remaining clay material into the -0.020 mm size fraction;
- Selectively release the ultra-fine quartz particles into the -0.020 mm size fraction; and

- Reduce the quartz content (A.I.) in the 1.18x0.106 mm and 0.106x0.020 mm size fractions.

According to the QEMSCAN and mineralogical analyses, the quartz rejection into the -0.020 mm size fraction may be limited due to the low levels of fine silica present in this phosphate ore. Under these conditions, coarse quartz may remain in the 1.18x0.106 mm and 106x0.020 mm size fractions since the P_2O_5 grade of these products is only marginally upgraded due to the rejection of iron bearing minerals and clays into the -0.020 mm size fraction (slimes). However, after attrition scrubbing, the phosphate bearing minerals and quartz particles had clean surfaces and were free of slimes. This prepares the ore for a surface chemistry based separation process: flotation.

13.5.1 Effect of Attrition Scrubbing Time for Three Different Solids Contents

Nine attrition scrubbing tests were carried out to investigate the effect of on the 6.3x0.075 mm size fraction obtained after the phosphate feed material was submitted to the previously selected horizontal scrubbing process conditions at 35% solids content for 300 seconds (5 minutes). The conditions varied during these attrition scrubbing tests were:

Scrubbing time:

- 150 seconds (2.5 minutes)
- 300 seconds (5.0 minutes)
- 600 seconds (10 minutes)
- solids content:
    - 45% solids
    - 55% solids
    - 60% solids.

Using the same screening procedure after attrition scrubbing that was used after horizontal scrubbing, the material was submitted to screen assays to trace the course of impurities through the size fractions corresponding to the different products:

- +1.18 mm is rejected as oversize;
- 1.18x0.106 mm becomes flotation feed;
- 0.106x0.020 mm becomes fine concentrate;
- -0.020 mm is rejected slimes.

The results are presented in Appendix B and show that depending on the attrition scrubbing conditions, Al_2O_3, Fe_2O_3, S_{total}, $S_{pyritic}$, and MgO decreased in the 1.18x0.020 mm size range, but the A.I. increased in the 1.18x0.020 mm range and decreased in the 0.106x0.020 mm size range. Ultimately, the selective rejection of impurities requires that the P_2O_5 recovery be the highest for the lowest corresponding mass yield. This parameter is the most important to avoid P_2O_5 losses.

At 45% solids content, a trend was observed of increasing P_2O_5 recovery as the scrubbing time increased. It is possible that the attrition scrubbing at low solids content reduced the surfaces' particle-particle interaction which required a longer scrubbing time to allow the release of impurities (with the exception of A.I.) without significantly increasing the viscosity of the slurry. Under these conditions, the longer the attrition scrubbing time led to a higher P_2O_5 recovery with the lowest increase in yield. Thus, at 600 seconds (10 minutes) of scrubbing time and 45% solids content, the higher yield and P_2O_5 recovery with adequate parameters was obtained (see Table 13-9 below)

Small differences in the P_2O_5 feed grade were observed, therefore these results were normalized with respect to feed grade and the data contained in Appendix B confirm these conclusions.

Tests carried out at 55% solids content demonstrate the same trend in impurities and a similar recovery of P_2O_5 and yield were observed. However, the best results were obtained at 150 seconds (2.5 minutes) of scrubbing time:

The increase of the surfaces' particle-particle interaction in this system without observing an increase in the viscosity of the slurry led to the conclusion that cushion effects are not present for the 150 seconds (2.5 minutes) of scrubbing time. This absence of cushioning effect is responsible for obtaining the best results using a low scrubbing time. An increase in scrubbing time resulted in lower P_2O_5 recoveries, lower P_2O_5 grade, similar mass yields, and inferior results for the CaO/P_2O_5 ratio, MER, MER*, and P_2O_5 grade potential. However, the normalized results did not show the same effect of scrubbing time for the same parameters. The normalized results actually showed slightly more desirable values for CaO/P_2O_5 ratio, MER, MER*, and P_2O_5 grade potential as the scrubbing time was increased. This effect was not sufficient to overcome the P_2O_5 recovery benefit of scrubbing at 150 seconds (2.5 minutes).

In the case of using 60% solids content during attrition scrubbing, the results showed lower yield and P_2O_5 recovery. The best results at 60% solids content were obtained after 300 seconds (5 minutes) of scrubbing time.

Table 13-9 Effect of % Solids in Attrition Scrubbing

	45% Solids	55% Solids	60% Solids
Mass Yield	72.70%	73.90%	71.80%
P_2O_5 Recovery	76.30%	77.20%	75.60%
CaO/P_2O_5 Ratio	1.454	1.454	1.454
MER	0.104	0.1	0.105
MER*	0.033	0.033	0.034
P_2O_5 Grade Potential	37.10%	36.50%	37.20%

It was clear that the effect of a viscous media activated at 60% solids content resulted in a cushioning effect reducing the attrition scrubbing efficiency even though surfaces' particle-particle interactions increased. In this case, the Normalized data showed that variations in the P_2O_5 feed grade were not significant and the parameters obtained after attrition scrubbing were undesirable.

Several plots were generated to compare the results of the nine attrition scrubbing tests and are included in this report.

Figure 13-15 presents the mass yield and P_2O_5 recovery as a function of scrubbing time for the three solids contents evaluated: 45%, 55%, and 60%. This plot clearly shows that the highest yield and P_2O_5 recovery is obtained after scrubbing for only 150 seconds (2.5 minutes) at 55% solids content. The mass yield and P_2O_5 recovery levels off as the scrubbing time is increase for all solids content studied.

Figure 13-16 presents the P_2O_5 grade and grade potential along with the A.I. grade as a function of scrubbing time. Again, at 150 seconds (2.5 minutes) and 55% solids content the highest P_2O_5 grade and grade potential was observed with the lowest A.I. grade reported. The P_2O_5 grade trend decreases as the scrubbing time increases for 45% and 55% solids content, whereas for 60% solids content the P_2O_5 grade increased up to 300 seconds (5 minutes) then decreases at 600 seconds (10 minutes). The P_2O_5 grade potential for 55% solids content is higher than that for 45% and 60% solids content for all scrubbing times studied. In the case of the A.I. grade, the lowest values are obtained at 150 seconds (2.5 minutes) at 55% solids content while all other scrubbing times and solids content studied report higher A.I. grades.

The CaO/P_2O_5 ratio and MER* parameters as a function of scrubbing time for the three solids content studied are presented in Figure 13-17. This figure shows that the CaO/P_2O_5 ratio is virtually constant for all scrubbing times and solids content studied. However, the MER* parameter shows a minimum at 600 seconds (10 minutes) of scrubbing time for 55% solids content. However, this MER* improvement alone does not justify the long scrubbing time due to lower yield, P_2O_5 recovery, P_2O_5 grade and grade potential, and a higher A.I. at 600 seconds (10 minutes) of scrubbing time.

The normalized data as a function of scrubbing time for 45%, 55%, and 60% solids content are presented in Figure 13-18 and Figure 13-19. These figures further show the same trends observed for the actual results, indicating that the P_2O_5 feed grade variations are not significant for these tests.

Figure 13-15 Yield and P_2O_5 Recovery as a Function of Attrition Scrubbing Time

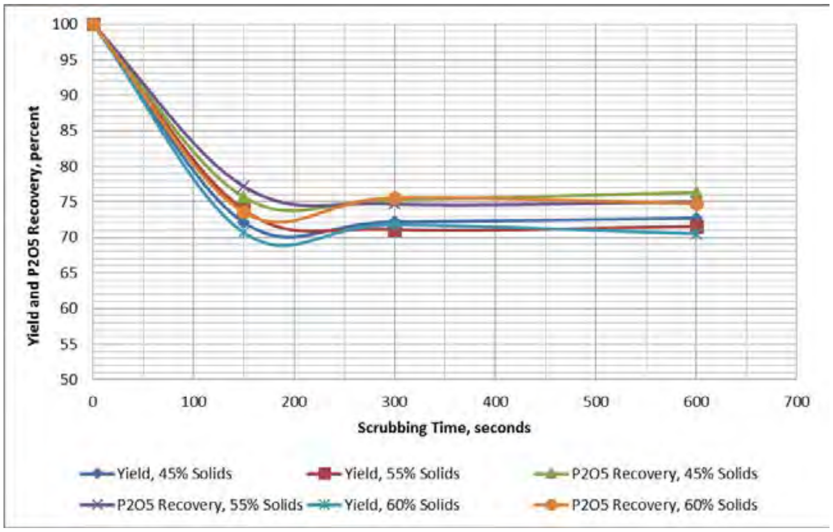


Figure 13-16 P_2O_5 Grade and Potential Grade, and A.I. Grade as a Function of Attrition Scrubbing Time

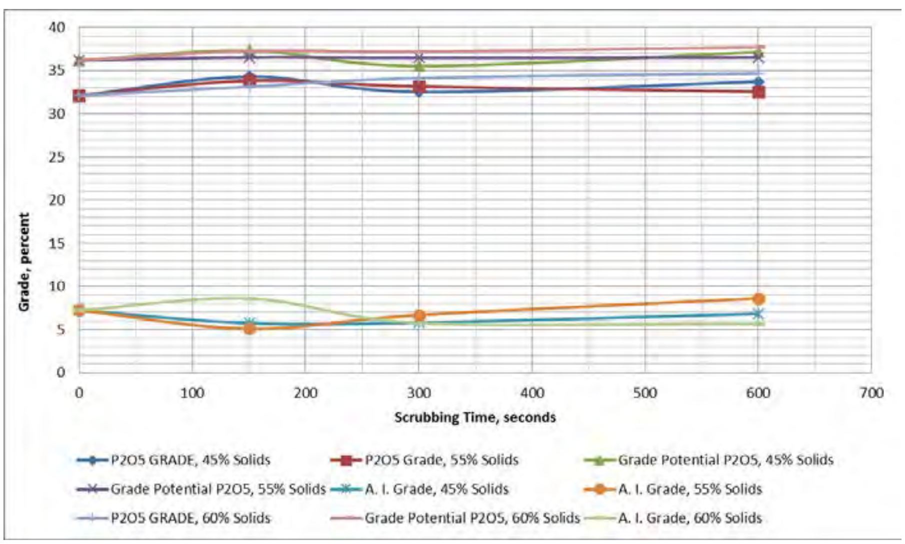


Figure 13-17 CaO/ P_2O_5 Ratio and MER* Parameters as a Function of Attrition Scrubbing Time

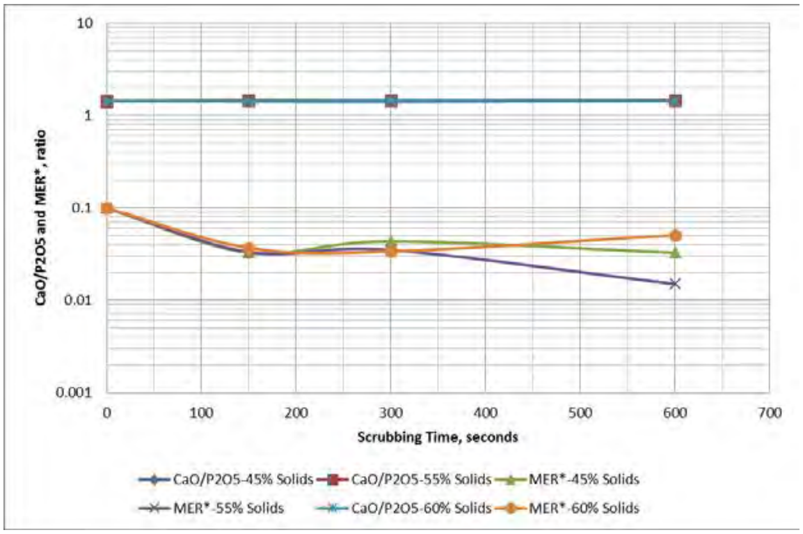


Figure 13-18 Normalized Grade as a Function of Attrition Scrubbing Time

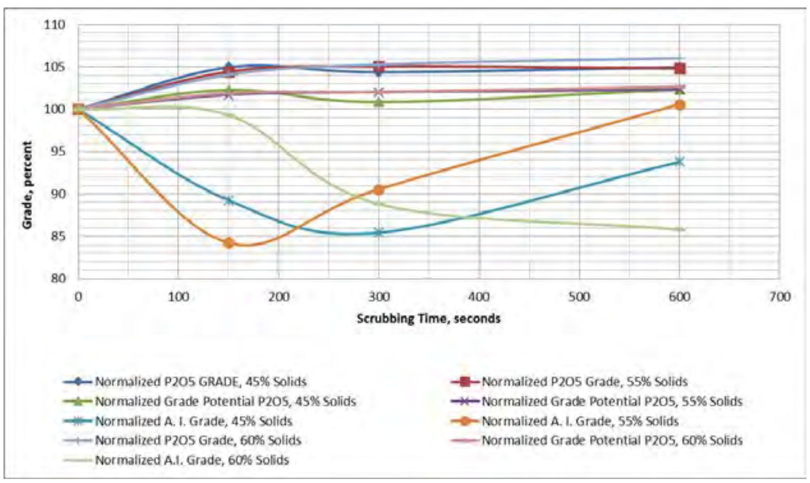
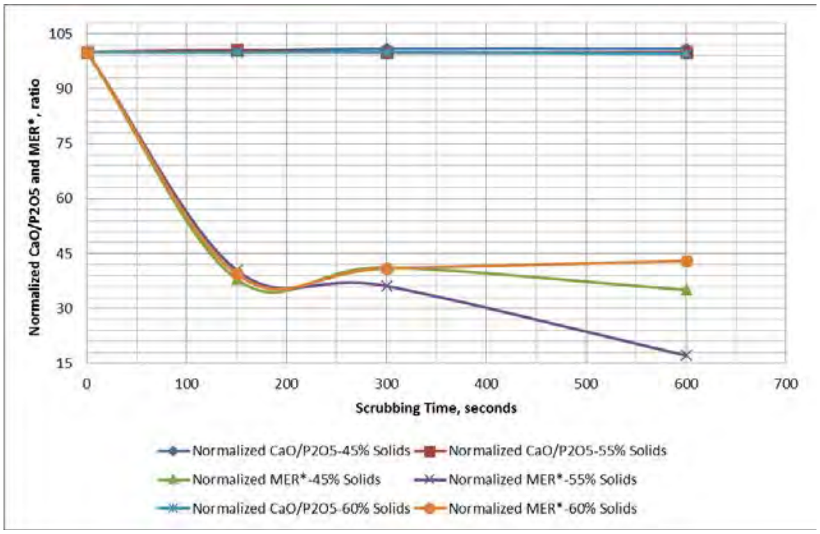


Figure 13-19 Normalized CaO/ P_2O_5 Ratio and MER* as a Function of Attrition Scrubbing Time



13.6 Reverse Amine Flotation Studies on the 1.18x0.106 mm Size Fraction

Seven tests were carried out to determine the flotation operating conditions for the 1.18x0.075 mm size fraction. For the flotation tests, the Farim Composite samples were first submitted to horizontal scrubbing and attrition scrubbing under the previously selected conditions. This section presents the results of the flotation tests performed. The overall metallurgical balance of the best test is presented in the next section of this chapter.

13.6.1 Experimental Procedure

The Farim Composite sample was horizontally scrubbed at 35% solids content for 300 seconds (5 minutes), followed by the screening of the +6.3 mm size fraction that was considered reject. The remaining material was screened at 0.075 mm to remove the clays before attrition scrubbing. Then, the 6.3x75 mm size fraction was submitted to attrition scrubbing at 55% solids content for 150 seconds (2.5 minutes). This scrubbed material was then screened at 1.18 mm where the 6.3x1.18 mm size fraction was considered reject. The remaining 1.18x0.075 mm size fraction was then screened at 0.106 mm. Two products were obtained here, the 1.18x0.106 mm size fraction, which constitutes the amine reverse flotation feed, and the -0.106 mm size fraction. This -0.106 mm size fraction and the -0.075 mm size fraction removed after horizontal scrubbing were combined and deslimed again at 0.020 mm to produce the 0.106x0.020 mm concentrate product which considered the fine concentrate. The -0.020 mm size fraction was rejected as slimes.

13.6.2 Flotation Results

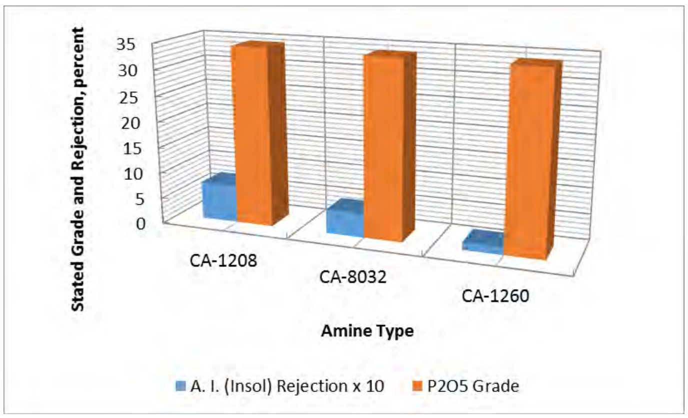
The individual flotation test data and the metallurgical balance of the process developed for the Farim Composite phosphate ore are presented in Appendix B. Flotation tests of the prepared feed were carried out to select the most efficient of three condensate amines provided by ArrMaz and to determine the required dosage of the selected amine to obtain the maximum P_2O_5 recovery, maximum A.I. rejection, and the highest P_2O_5 grade in the 1.18x0.106 mm concentrate.

13.6.3 Amine Selection

In order to determine which of the three condensate amines was best suitable for the 1.18x0.106 mm flotation feed, an arbitrary but common dosage was selected (0.23 kg/ton). Each flotation was carried out under the same flotation conditions at 20 seconds conditioning time and 1 minute of flotation time. These flotation conditions were not optimized with respect to any parameter as they are common procedure in bench flotation laboratories.

Figure 13-20 presents the P_2O_5 grade and A.I. rejection as a function of amine type. This figure shows that Amine CA-1208 produced the highest A.I. rejection (0.73%) without affecting the P_2O_5 grade of the concentrate (34.4% P_2O_5). Thus, Amine CA-1208 was more selective and stronger than the other two amines tested.

Figure 13-20 Effect of Amine Type at 0.23 kg/ton of Amine Addition



13.6.4 Effect of CA-1208 Addition

Once the amine was selected, the effect of dosage was studied to determine the maximum A.I. rejection with minimal reduction in the P_2O_5 recovery along with the maximum P_2O_5 grade in the concentrate. The effect of this reverse flotation on the Fe_2O_3 grade and rejection was also included in this report for completion since iron is a secondary contaminant.

The P_2O_5 grade, A.I. grade, and Fe_2O_3 grade in the concentrate as a function of CA-1208 amine addition are presented in Figure 13-21. The best results are obtained with the addition of 1.168 kg CA-1208 amine per ton of flotation feed. The concentrate reports 36.7% P_2O_5 with 2.2% A.I. and 1.5% Fe_2O_3. Figure 13-22 shows the P_2O_5 recovery, A.I. and Fe_2O_3 Rejection as a function of amine addition. At 1.168 kg/ton dosage of CA-1208 amine, it was possible to recover 97.3% of the P_2O_5 content in the flotation feed concentrate while rejecting 73.4% of the A.I. and 17.0% of the Fe_2O_3.

Figure 13-21 Grades as a Function of CA-1208 Amine Addition

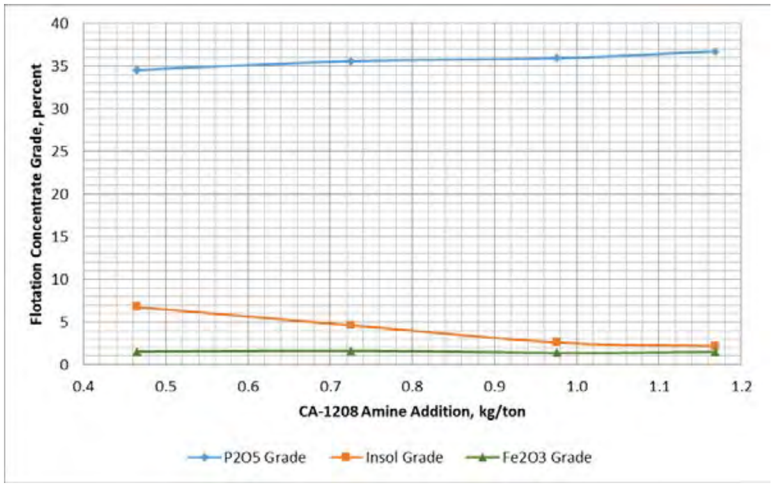
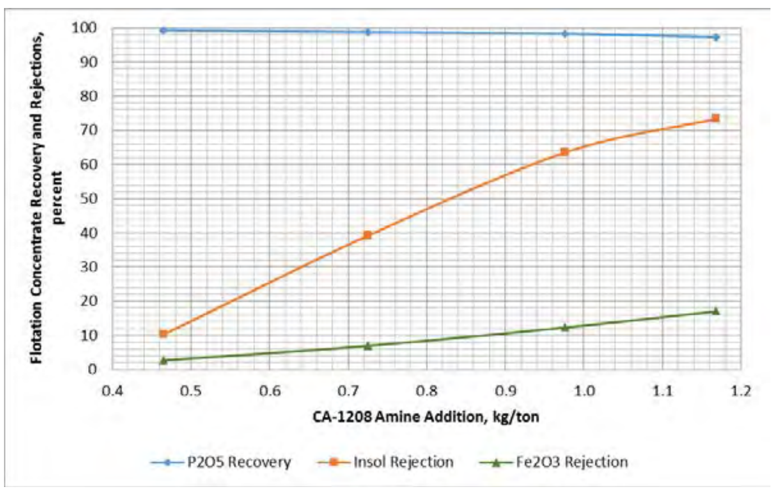


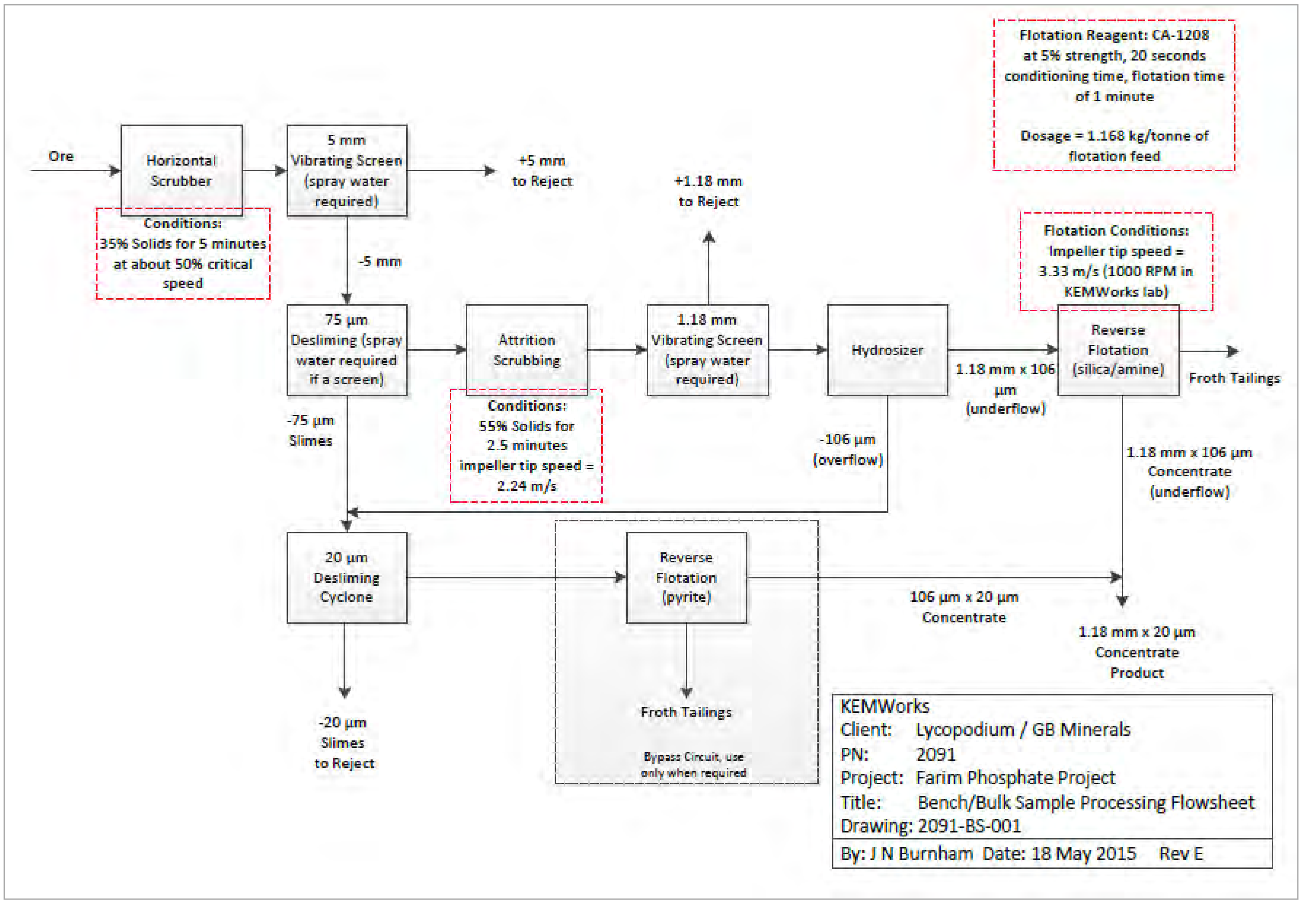
Figure 13-22 P_2O_5 Recovery, A.I. and Fe_2O_3 Rejections as a Function of CA-1208 Amine Addition



13.7 Metallurgical Balance from KEMWorks Bench Scale Testwork Using Scrubbing and Flotation

Using the results for the flotation feed preparation procedure and the reverse flotation tests the standard bench scale procedure was developed as shown in Figure 13-23. This diagram summarizes the experimental procedure delineated above and required process conditions and is the basis for the process flowsheet.

Figure 13-23 Process Block Flow Diagram for the Farim Composite Sample



Following this block flow diagram for the bench scale processing of the composite sample of Farim Phosphate Ore, it is possible to obtain the metallurgical balance presented in Table 13-10. This table shows that 5.8% of the feed is rejected in the +6.3 mm size fraction and 2.1% of the feed is rejected in the 6.3x1.18 mm size fraction. The total slimes (-0.020 mm material) reported were 21.6% of the feed. Table 13-10 also shows that the reverse flotation concentrate makes up 49.3% of the feed and the fine concentrate is 16.5% of the feed for a total mass yield of 65.8% for the concentrate blend. The flotation tailings constitute 4.7% of the ore feed.

The achieved P_2O_5 grade of the flotation concentrate is 36.7% P_2O_5 and the fine concentrate grade is 33.5% P_2O_5 resulting in a concentrate blend of 35.9% P_2O_5. The P_2O_5 recovery of the flotation concentrate is 55.9% and that of the fine concentrate is 17.1% for a total product blend P_2O_5 recovery of 73.0%. The total rejection of A.I. is 85.5% and 91.7% for the flotation concentrate and fine concentrate, respectively. The blend reports 77.2% of A.I. rejection. The MER obtained from the flotation concentrate is 0.047. The MER of the fine concentrate is 0.131 and the concentrate blend MER is 0.067. The P_2O_5 grade potential obtained are 38.2%, 36.2%, and 37.7% for the flotation, fine and concentrate blends, respectively.

Table 13-10 Metallurgical Balance for the Farim Composite Sample

Products	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
Rejects	6300	27.6	5.81	5.81	94.19
Rejects	1180	9.8	2.06	7.87	92.13
Flot Con	106	234.3	49.32	57.19	42.81
Flot Tails	106	22.5	4.74	61.92	38.08
Fine Con	20	78.4	16.50	78.43	21.57
Slimes	6	102.5	21.57	100.00	0.00
Total		475.10	100.00		

		Grades					Cum. Grades		
P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %	P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %
26.82	0.42	1.13	9.87	7.73	26.82	0.42	1.13	9.87	7.73
18.85	0.49	0.66	20.52	5.37	24.73	0.44	1.01	12.66	7.11
36.7	0.076	0.164	1.48	2.2	35.05	0.13	0.28	3.02	2.88
10.6	0.097	0.199	3.16	63.1	33.18	0.12	0.27	3.03	7.48
33.45	0.43	0.83	3.11	3.76	33.24	0.19	0.39	3.05	6.70
29.14	0.6	3.95	2.42	10.28	32.35	0.28	1.16	2.91	7.47
32.35	0.28	1.16	2.91	7.47					

		Distribution					Cum. Distribution-Products			Products
P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %	P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %	
4.82	8.81	5.67	19.69	6.01	4.82	8.81	5.67	19.69	6.01	
1.20	3.65	1.17	14.54	1.48	6.02	12.46	6.84	34.23	7.49	Rejects
55.94	13.53	6.98	25.07	14.52	55.94	13.53	6.98	25.07	14.52	Flot Con
1.55	1.66	0.81	5.14	40.00	1.55	1.66	0.81	5.14	40.00	Flot Tails
17.06	25.62	11.82	17.63	8.30	17.06	25.62	11.82	17.63	8.30	Fine Con
19.43	46.73	73.55	17.93	29.68	19.43	46.73	73.55	17.93	29.68	Slimes
100.00	100.00	100.00	100.00	100.00						

Figure 13-24 presents the yield, P_2O_5 recovery and A.I. rejection. The P_2O_5 grade and grade potential, and the A.I. grade as a function of CA-1208 amine addition is presented in Figure 13-25. These figures show that the bench scale process is successful in producing the required product specifications.

The tests performed following the beneficiation process delineated in Figure 13-23 results in an average feed mass distribution of:

- +6.3 mm rejection 5.2% ± 1.9%

- 6.3x1.18 mm rejection 2.2% ± 0.2%
- 1.18x0.106 mm flotation concentrate 49.3% ± 2.8%
- reverse flotation tailings 4.7% ± 1.7%
- 0.106x0.020 mm fine concentrate 16.6% ± 0.5%
- -0.020 mm slimes rejection 21.9% ± 0.3%

Figure 13-24 Yield, P_2O_5 Recovery, and A.I. Rejection as a Function of CA-1208 Amine Addition

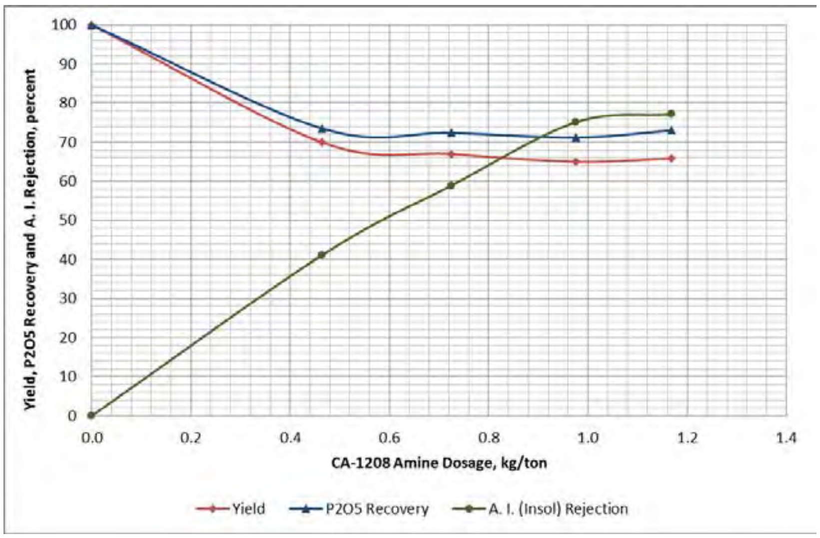
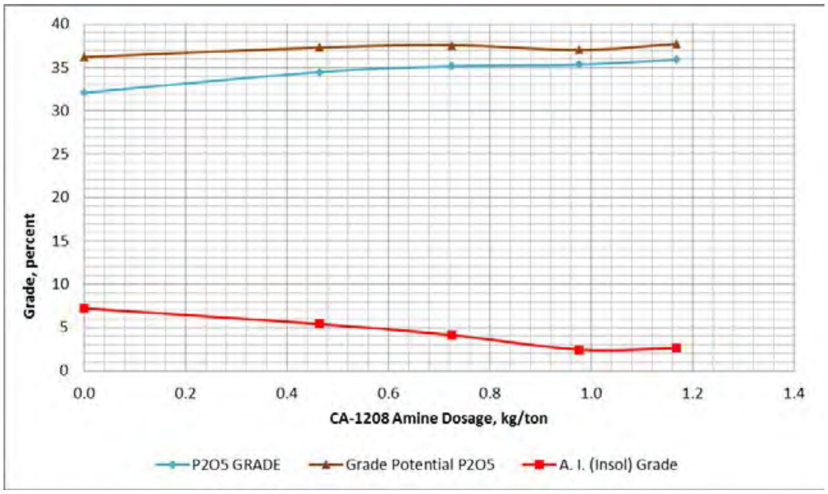


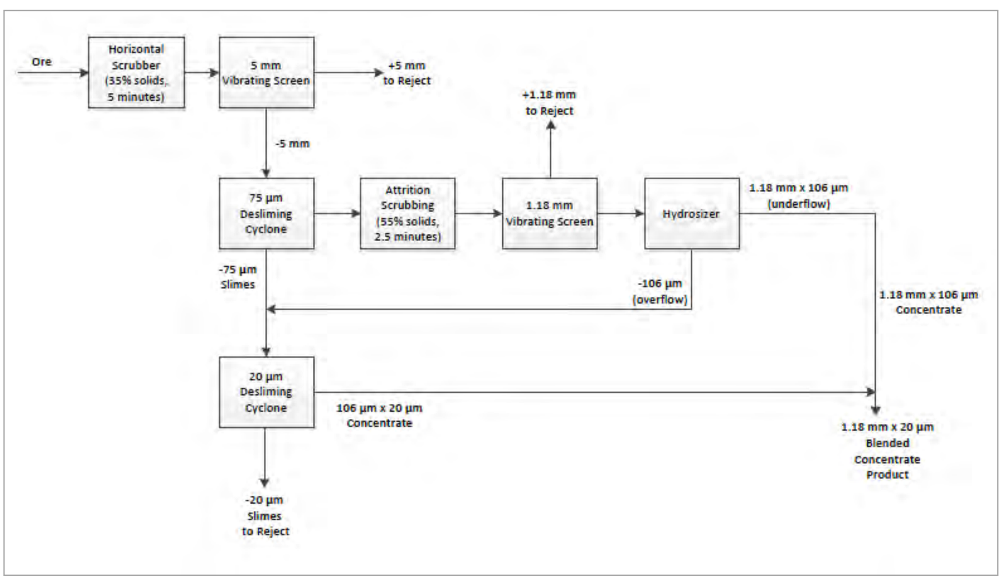
Figure 13-25 P_2O_5 Grade and Grade Potential, and A.I. Grade as a Function of CA-1208 Amine Addition



13.8 Metallurgical Balance from KEMWorks Bench Scale Testwork Using Only Scrubbing Techniques

Combining the most successful tests and procedures from the horizontal and attrition scrubbing tests, the standard bench scale procedure was developed as shown in Figure 13-26. This diagram summarizes the experimental procedure delineated above and required process conditions and is the basis for the process flowsheet.

Figure 13-26 Process Block Flow Diagram for the Farim Composite Sample



Following this block flow diagram for the bench scale processing of the composite sample of Farim Phosphate Ore, it is possible to obtain the metallurgical balance presented in Table 13-11. This table shows that 2.0% of the feed is rejected in the +6.3 mm size fraction and 2.2% of the feed is rejected in the 6.3x1.18 mm size fraction. The total slimes (-0.020 mm material) reported were 21.9% of the feed. Table 13-11also shows that the coarse concentrate makes up 56.2% of the feed and the fine concentrate is 17.7% of the feed for a total mass yield of 73.9% for the concentrate blend.

The achieved P_2O_5 grade of the coarse concentrate is 34.2% P_2O_5 and the fine concentrate grade is 32.6% P_2O_5 resulting in a concentrate blend of 33.8% P_2O_5. The P_2O_5 recovery is 77.2%. The concentrate product blend MER is 0.07.

Table 13-11 Bench Scale Metallurgical Balance for the Farim Composite Sample

Product Designation	Opening	Weight %	P_2O_5, %	Insol, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Pyrtic Sulfur	Pyritic Iron	MER of Fraction	MER* of Fraction
Reject	6300	1.98	27.47	6.53	41.36	1.83	0.94	4.93	1.71	1.489	0.280	0.222
Reject	1180	2.24	20.95	6.13	23.22	0.70	0.42	22.14	3.65	3.179	1.110	1.099
Concentrate	425	7.88	33.28	5.29	47.40	0.10	0.17	3.88	0.91	0.793	0.125	0.125
Concentrate	106	48.27	34.40	6.13	50.33	0.09	0.15	1.42	0.41	0.357	0.048	0.047
Concentrate	20	17.75	32.57	2.18	47.00	0.60	0.80	2.66	0.71	0.618	0.125	0.109
Slimes	-20	21.88	29.17	9.21	41.97	0.64	4.12	2.47	0.84	0.732	0.248	0.230

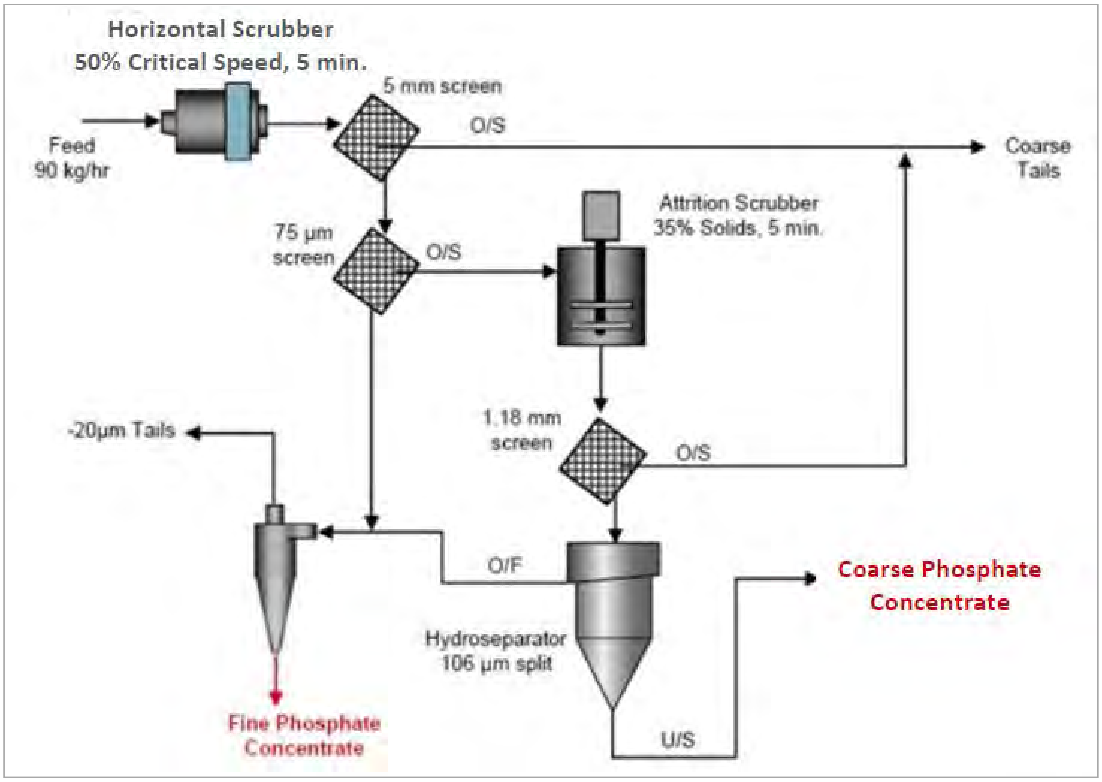
Feed P_2O_5, %	Combined Product MER	Combined Product MER*	Combined Product P_2O_5	Combined Product CaO	CaO/P_2O_5 Product Ratio	Combined Tailings P_2O_5	Ratio of Concentration	P_2O_5 Recovery	Mass Recovery
32.4	0.075	0.070	33.8	49.2	1.45	28.3	1.35	77.2	73.9

13.9 Metallurgical Balance from ALS Pilot Plant Testwork Using Scrubbing Only

Pilot plant testing was conducted at ALS Metallurgy Kamloops. The objectives of the test program were to demonstrate the metallurgical performance of the scrubbing flowsheet in Figure 13-26 in a continuous pilot circuit and to produce concentrate and tailings samples for downstream testing.

Approximately 620 kg of bulk sample, on a dry basis, was processed through a small pilot circuit shown in Figure 13-27.

Figure 13-27 Pilot Flowsheet Developed by ALS



The results from the pilot testing at ALS presented slightly better P_2O_5 recoveries and mass recoveries from the ore. These results are shown in Table 13-12.

The pilot circuit recovered more phosphate to the fine concentrate via the cyclone underflow than in the laboratory tests. This is attributed to the use of screens in bench scale testing versus using actual cyclones and a hydroseparator unit in the pilot laboratory. The pilot testing better represents the behavior of the Farim ore in the proposed process plant.

Table 13-12 Pilot Scale Metallurgical Balance for the Farim Composite Sample

Product Designation	Opening	Weight %	P_2O_5, %	Insol, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Pyrtic Sulfur, %	Pyritic Iron, %	MER of Fraction	MER* of Fraction
Reject	6300	6.5	25.9	7.9	36.8	0.31	1.59	12.9	3.22	2.804	0.571	0.463
Reject	1180	3.1	31.4	4.1	42.5	0.13	0.63	8.8	2.83	2.465	0.304	0.226
Concentrate	425	48	35.5	5	48.8	0.08	0.26	1.5	0.66	0.575	0.052	0.036
Concentrate	106	5.8	23.1	30.3	30.5	0.21	2.08	4.2	2.67	2.325	0.281	0.180
Concentrate	20	21.7	33.7	3.2	47.2	0.18	1.25	3.1	1.71	1.489	0.134	0.090
Slimes	-20	14.9	29.6	9.7	41.2	0.46	5.44	2.2	0.73	0.636	0.274	0.252

Feed P_2O_5, %	Combined Product MER	Combined Product MER*	Combined Product P_2O_5	Combined Product CaO	CaO/P_2O_5 Product Ratio	Combined Tailings P_2O_5	Ratio of Concentration	P_2O_5 Recovery	Mass Recovery
32.8	0.093	0.062	34.0	46.9	1.38	28.8	1.32	78.4	75.5

ALS Kamloops generated 425 kg of concentrate product using this process to be used for the WAP (wet acid process) by KEMWorks for phosphoric acid production.

13.10 Summary and Conclusions

13.10.1 Ore Characterization

100 kg of core samples from the Farim Phosphate deposit were received at KEMWorks on December 26, 2014. This sample consisted of four subsamples, SB9, SC10, SC11, and SE10. These subsamples corresponded to the Block Model and Assay Model data for the deposit, representing the first seven years of production. The samples showed that the main contaminants were A.I. (Insol) and iron bearing minerals as indicated by Fe_2O_3, S_{total}, and $S_{pyritic}$ analyses followed by Al_2O_3 contaminants. These samples are confirmed representative of the deposit. A weighted composite was prepared for characterization studies, horizontal scrubbing (drum), attrition scrubbing, and reverse amine flotation tests.

A composite sample, called the Farim Composite, was prepared after the weighted subsamples were homogenized split. Care was taken to preserve the moisture content of these subsamples. From this Farim Composite sample, the following subsamples were prepared:

- Head Sample for chemical analyses, 50 g each (wet).
- Screen analysis and screen assay, two samples of 500 g each (wet).
- Test samples of each subsample, each split of 610 g (wet).

The Characterization studies, Head chemical analysis, screen analyses, screen assays, and mineralogical QEMSCAN showed that the Farim Composite was representative of this area of the deposit, presenting similar elements and compounds values. The results of the Head Sample chemical analysis showed that the composite P_2O_5 grade was 33.0% ± 0.7% with a 2.0% error, resulting in a P_2O_5 grade between 31.5% to 34.5% range. The complete Head chemical analysis was shown in Table 13-2. The metallurgical parameters were:

- CaO/ P_2O_5 ratio 1.4
- MER 0.141
- MER* 0.079
- P_2O_5 grade potential 36.5%.

The particle size distribution (PSD) reported a mean particle size (d_{50}) of 0.140 mm with a single mode in the distribution (unimodal), the mode located at 0.106 mm (150 mesh). Screen assays showed that aluminum silicates were present containing Al_2O_3 and MgO. The Fe_2O_3, S_{total}, and $S_{pyritic}$ are associated and part of the Fe_2O_3 seemed to constitute part of the aluminum silicates. The A.I. is evenly distributed throughout all size fractions coarser than 0.106 mm and decreasing for particles smaller than 0.106 mm. The A.I. is the most critical impurity to be rejected. QEMSCAN results confirmed this interpretation and conclusions of the screen assays.

To develop the beneficiation process required for the Farim Composite to reach the desired specifications, horizontal scrubbing (drum), attrition scrubbing and reverse amine flotation tests were carried out.

13.10.2 Horizontal Scrubbing

Tests were conducted under standard conditions as a baseline at six different conditions to evaluate two solids contents (35% and 50%) at three scrubbing times: 150 seconds, 300 seconds, and 600 seconds (2.5 minutes, 5 minutes, and 10 minutes, respectively). These tests showed that A.I., Al_2O_3, Fe_2O_3, S_{total}, $S_{pyritic}$, and MgO decreased in the product size range (1.18x0.020 mm). At 35% solids content and 300 seconds (5 minutes) of scrubbing time, the best yield (73.7%) P_2O_5 recovery (77.3%) and P_2O_5 grade (34.4%) was obtained. In addition, the lowest A.I. grade (5.97%) was obtained under these conditions with an A.I. rejection of 34.9%.

- Mass yield 73.7%
- P_2O_5 recovery 78.4%
- CaO/ P_2O_5 ratio 1.4
- MER 0.103
- MER* 0.034

Confirmation tests validated these results. All of these tests were analyzed based on the actual results and then normalized based on the feed grades of each test to eliminate the effect of small differences in feed grade of each test that could mislead the interpretation of results. These tests considered the +6.3 mm and 6.30x1.18 mm size fractions as rejects and the -0.020 mm material as slimes.

13.10.3 Attrition Scrubbing

Tests were designed to release significant amounts of quartz, clay, and iron bearing minerals attached to the francolite surfaces in the 6.30x0.075 mm size fraction obtained after horizontal scrubbing (drum). However, A.I. rejection was limited to the -0.020 mm size fraction due to the hardness of quartz and the small amounts of fine silica locked onto the surface of phosphate bearing minerals according to the QEMSCAN and mineralogical studies. Nine tests were carried at three solids contents (35%, 45%, and 55%) for three different scrubbing times, 150 seconds, 300 seconds, and 600 seconds. The best results were obtained at 55% solids content and scrubbing for 150 seconds (2.5 minutes):

- Mass yield 73.9%
- P_2O_5 recovery 77.2%
- CaO/ P_2O_5 ratio 1.5
- MER 0.075
- MER* 0.070
- P_2O_5 grade 33.8%

Again, normalized data were evaluated and confirmed the results.

13.10.4 Reverse Amine Flotation

Studies of the 1.18x0.106 mm size fraction were carried out. Seven flotation tests were conducted for the selection of the type of condensate amine to be used, and to obtain the best flotation results. ArrMaz CA-1208 amine was selected. The addition of 1.168 kg/ton of flotation feed resulted in a 1.18x0.075 mm concentrate of 36.7% P_2O_5 grade with 2.2% A.I. grade, and 1.48% Fe_2O_3 grade. The P_2O_5 recovery was 97.3% of the P_2O_5 content of the flotation feed with a rejection of 77.4% of A.I. and 17.0% of the Fe_2O_3 of the flotation feed.

The beneficiation process using flotation to further upgrade the ore by removing silica was presented in Figure 13-23 which resulted in the following products:

- +6.3 mm rejection 5.2% ± 1.9%
- 6.3x1.18 mm rejection 2.2% ± 0.2%
- 1.18x0.106 mm flotation concentrate 49.3% ± 2.8%

- reverse flotation tailings 4.7% ± 1.7%
- 0.106x0.020 mm fine concentrate 16.6% ± 0.5%
- -0.020 mm slimes rejection 21.9% ± 0.3%

13.10.5 Pilot Plant Results

The results of the pilot plant testwork confirmed KEMWorks' circuit design using horizontal and attrition scrubbing to remove the impurities from the ore to achieve a concentrate product of 34% P_2O_5. 425 kg of concentrate products were generated and shipped to KEMWorks for future WAP testing for phosphoric acid production.

The pilot testing concluded that the following product specifications can be achieved using this process:

- Mass yield 75.5%
- P_2O_5 recovery 78.4%
- CaO/ P_2O_5 ratio 1.4
- MER 0.093
- MER* 0.062
- P_2O_5 grade 34.0%

14.0 MINERAL RESOURCE ESTIMATE

14.1 Mineral Resource Definition

In accordance with NI 43-101, for estimating Mineral Resources of the Farim Phosphate Project, Golder has applied the definition of "Mineral Resource" as set forth in the updated CIM Definition Standards adopted 10 May 2014 (CIMDS) by the Canadian Institute of Mining, Metallurgy, and Petroleum Council.

Under CIMDS, a Mineral Resource is defined as:

> "...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.
>
> The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling..."

Mineral Resources are subdivided into classes of Measured, Indicated, and Inferred, with the level of confidence reducing with each class, respectively. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource. Mineral Resources are always reported as in situ tonnage and are not adjusted for mining losses or mining recovery.

14.2 Introduction

The Farim deposit has been delineated over an area of approximately 40 km^2 and is divided by the Cacheu River. The deposit consists of both FPA and FPB mineralized units. This Mineral Resource Estimate concerns FPA only, as the FPB unit was previously deemed to be uneconomic. No additional mineralization outside the deposit modelled was considered in the Mineral Resource Estimate.

Golder modelled the Farim resource based on a 2D grid of 125 m by 125 m cells covering the extents of the FPA layer. The extents of the FPA layer were digitized based on the presence or absence of the FPA layer in the drill holes. P_2O_5 grade plus four deleterious elements; Al_2O_3, CaO, Fe_2O_3 and SiO_2, were estimated. The thickness of the overburden and FPA units were also estimated.

The initial Golder Mineral Resources Estimate for the Farim deposit was estimated performed in 2012 by Faye Jones (MSc, FGS, MAusIMM) of Golder under the supervision of QP, Marcelo Godoy (PhD, AusIMM CP). The Mineral Resource Estimate was subsequently updated by Jonathan Winne of Golder under the supervision of QP, Jerry DeWolfe (M.Sc. P.Geo.) The QP is independent of the Issuer as defined by Section 1.5 of the National Instrument. The Mineral Resource statement is effective 2 July 2015.

The initial 2012 Golder estimation was undertaken in Isatis™ (Version 2011.3) and Vulcan™ (Version 8.1.3) while the updated 2015 Golder estimate was performed in MineScape™ (Version 5.8) and Vulcan™ (Version 9.1.3).

14.3 Data Provided

14.3.1 Drill Hole Data

The Mineral Resource Estimate is based on diamond drill hole data. A total 10,326 m were drilled in 190 diamond core holes on the Farim deposit between 1981 and 2011.

BRGM 1981: 2,100 m from 32 diamond core holes were drilled over a 25 km^2 grid. Complete and detailed logs, assay analysis and other data are available, but core and samples were not available for inspection.

BRGM 1983: 3,572 m from a further 69 diamond core holes were drilled by BRGM over a 25 km^2 grid. Complete and detailed logs, assay analysis and other data are available, but core and samples were not available for inspection.

Champion 1999: 1,810 m from 34 infill diamond core holes were drilled over a 38 km^2 grid. Assay data is available but detailed logs, drill core and samples were not available for inspection. However the upper and lower position of the phosphate bed is recorded.

GBMAG 2008 to 2009: 1,564 m from 30 diamond core holes were drilled by GEEEM. Complete and detailed logs, assay analysis and certificates, half core, samples and other data are available for inspection.

GBMAG 2011: 1,280 m from 25 diamond core holes were drilled by GEEEM. Complete and detailed logs, assay analysis and certificates, half core, samples and other data are available for inspection.

All drill holes are drilled vertically and are assumed not to deviate significantly due to the short length of the holes (maximum 90 m) and the hardness of the rocks. The Farim resource is intersected by 148 drill holes with the majority on 500 m grid spacing. A number of holes either had very low or no recovery and were therefore excluded from the database or fall outside the Farim deposit. Holes which are close to the Farim deposit and did not intersect FPA are assigned a thickness of zero and used to define the limits of mineralization and control the estimation.

The sources of data have been reviewed by Golder through thorough validation checks against digital data. Observations from the site visit and data validation procedures completed indicate that the data used in the estimate follows industry standard practices for their drilling and QAQC program and the compiled drill hole database used in the estimation is sufficiently free of errors to be used in the Mineral Resource Estimate.

14.3.2 Other Data

A topographic survey was carried out during 2011 by AOC using airborne LiDAR, which had a horizontal accuracy of 0.5 m and a vertical accuracy of 0.2 m. The DTM (digital terrain model) used in the estimate was derived from that survey.

14.4 Geological Modelling

The FPA unit is a sub-horizontal, laterally extensive unit that is relatively thin. The footwall of the FPA undulates causing variations in the FPA thickness from less than 1 m at the edge of the resource area up to 6.2 m in the centre. In addition, the overburden thickness is known to increase towards the north of the deposit due to the higher elevation of the surface and this will be a defining factor of what can be economically extracted. The thickness of the overburden and the FPA units were therefore estimated in the resource model, so that the stratigraphic sequence could be rebuilt from the topographic surface. No geological wireframe modelling was carried out of the individual stratigraphic units.

A set of roof and floor regularized grid surfaces were generated in MineScape defining the extent of the FPA unit using the logged FPA thicknesses in the drill holes as a guide to where the unit thins out. This outline and the resource drill hole database used is shown in Figure 14-1. In addition to the MineScape grid surfaces a solid wireframe using the same data was also created in Vulcan for comparison purposes.

14.5 Exploratory Data Analysis

Exploratory data analysis (EDA) helps to identify the basic statistical and spatial behaviour of the elements before estimation is carried out and generally involves looking at histograms, base maps of sample location, univariate and multivariate statistics and log-probability plots. This helps to guide some decisions such as:

- Domaining;
- Declustering;
- Top-cutting or treatment of high grades or outliers;
- Compositing;
- Parameters to be used during variography such as lag distance;
- Block size for the resource model; and
- The results of these analyses are described in the following chapters where appropriate.

14.5.1 Data Capture

Domains are used to separate statistically different populations for estimation. One domain was used to constrain composites and the block models during estimation. This domain is represented by the solid wireframe created which define the extent of the FPA. The wireframes were used to select all the composites lying inside, which were flagged with a numeric code.

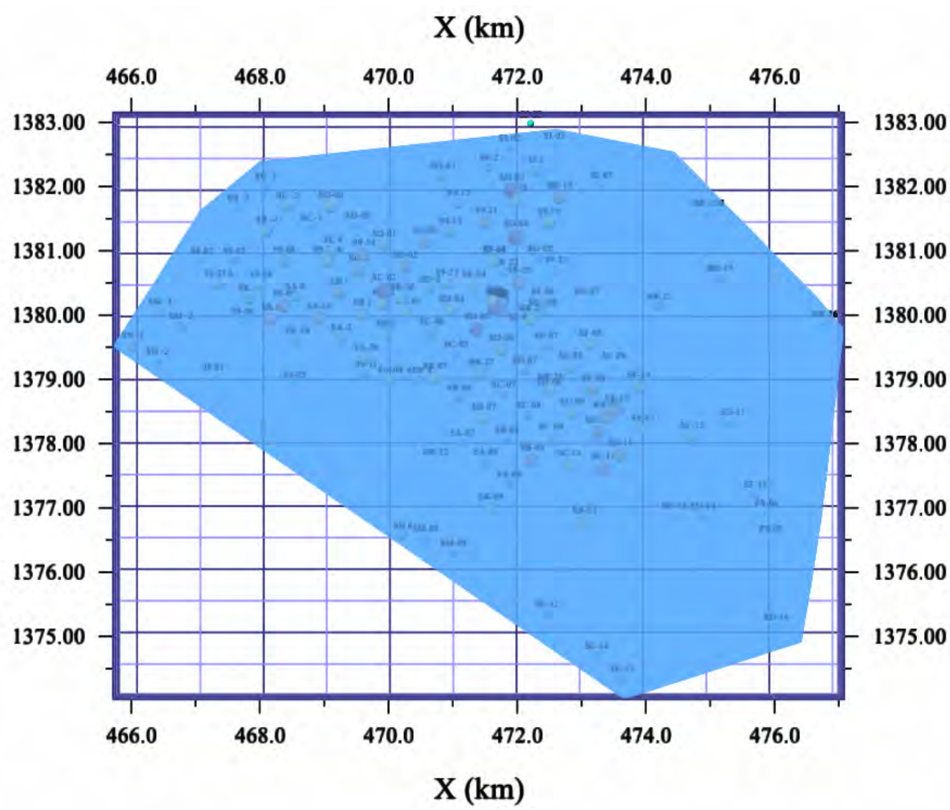
14.5.2 Composites

Often samples are not taken at regular intervals, which presents a problem during estimation as the samples do not have the same statistical support (volume representation), which may introduce a bias. All sampling at Farim has been done on irregular length intervals according to changes in the visual and physical properties of the core. As described in Section 12.4, individual assay results were not entered into the digital database, instead length weighted averages per drill hole were entered by GBMAG. This is in effect lithological compositing, where drill holes are composited to a single value per lithological unit. Considering the morphology of the deposit and the proposed mining method, this is appropriate for use in the resource estimate.

14.5.3 Statistical Analysis

Figure 14-1 shows the location of the drill holes contained within the current resource database for Farim. The majority of the drill holes are located in the north and central parts of the deposit. Here spacing is approximately 500 m. On the periphery, especially to the south of the River Cacheu, the drill holes are more sparse.

Figure 14-1 Farim, Drill Hole Location Map



Univariate statistics and histograms of grade and thickness variables were generated and are summarised in Table 14-1 and Figure 14-2. Histograms for all variables are stored in Appendix A of Golder report 11514950043.508/B.3.

Table 14-1 Farim, Univariate Statistics

Variable	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	Weight
AL_2O_3	104	0.51	29.86	3.05	2.91	8.48	length
CaO	104	7.15	50.13	39.62	6.05	36.64	length
Fe_2O_3	104	0.49	40.98	5.33	3.69	13.65	length
P_2O_5	129	0.73	36	28.69	5.24	27.5	length
SiO_2	104	4.36	35.5	11.56	4.97	24.69	length
REC	134	0	100	75.43	22.56	509.03	length
Overburden thickness	141	26.9	69.8	43.14	10.36	107.3	none
FPA thickness	148	0	6.2	2.7	1.53	2.35	none

Figure 14-2 Farim, Histograms

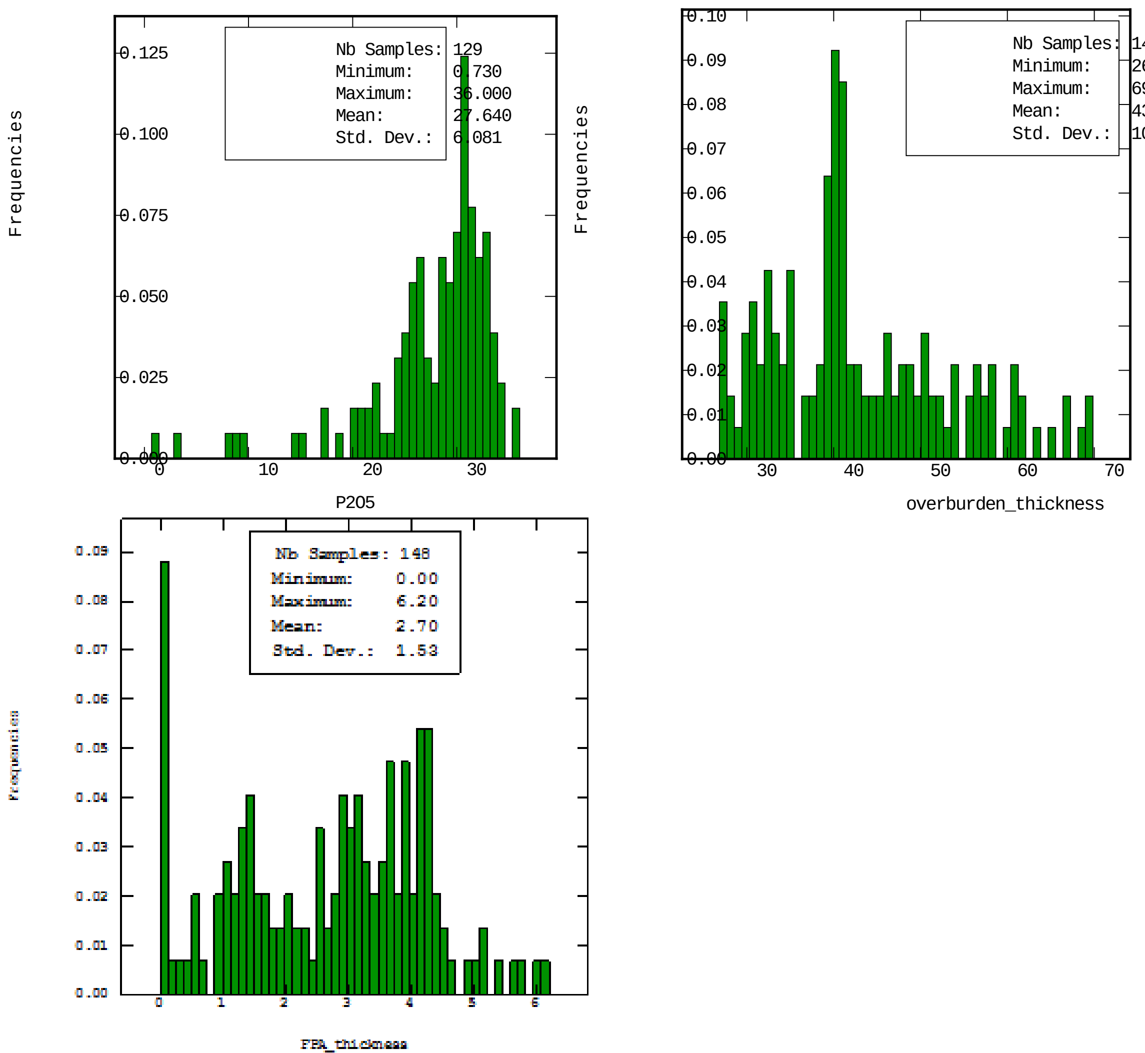
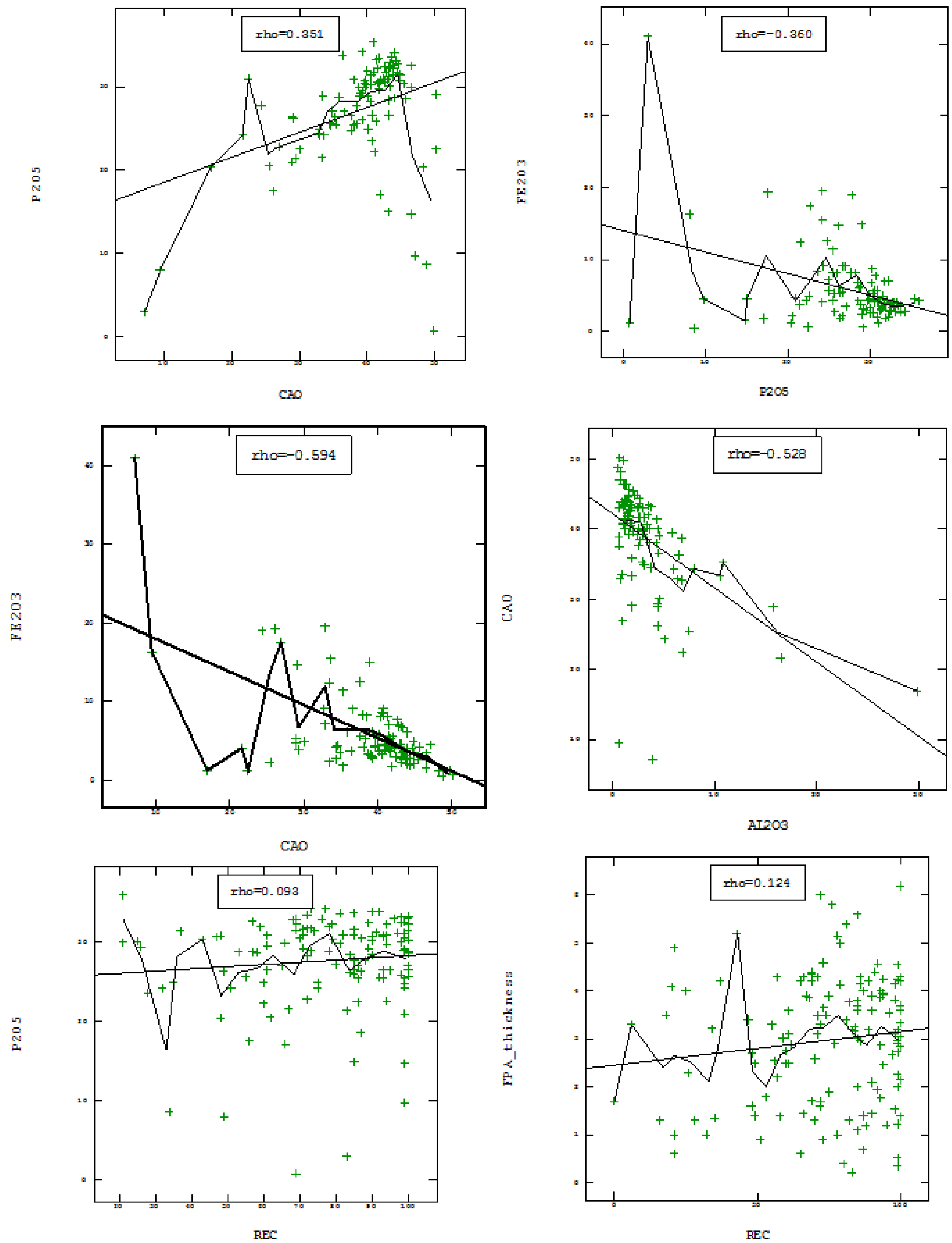


Figure 14-3 Farim, Scatter Plot



The P_2O_5 and CaO distributions are similar, with a negative skew. Al_2O_3, Fe_2O_3 and SiO_2 all show some positive skew. The recovery histogram shows some very low values. Overburden thickness shows a fairly random distribution, while the FPA thickness is normally distributed.

14.6 Variography

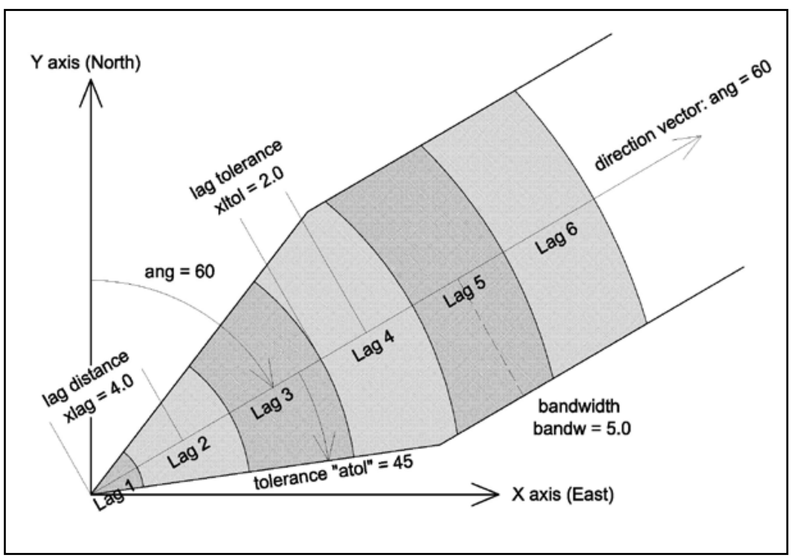
Variography is used to model the continuity of spatial phenomena such as the distribution of grade in a mineralised body. The objectives of the variography were to establish the directions of major and semi-major continuity for both P_2O_5 grade and thickness of the FPA phosphate horizon.

Directional variography requires search tolerances to be used for calculation of variograms to address the fact that the drillhole samples are not perfectly aligned in a given direction in 3D space and are not equally spaced along that direction. This requires the use of angular and distance tolerances. The tolerances used for directional variogram calculation are provided in Table 14-2. Figure 14-4 illustrates the relationship between the angular and distance tolerances with respect to the direction in which the variogram is required to be calculated.

Table 14-2 Farim, Experimental Variogram Search Parameters

Parameter	P_2O_5 (%)	Sample Thickness (m)
Horizontal Angle Tolerance	22.5	22.5
Vertical Angle Tolerance	22.5°	22.5°
Horizontal Distance Band width	1,200 m	1,200 m
Vertical Distance Bandwidth	30 m	30 m
Lag Distance	600 m	600 m
Lag Tolerance	300 m	300 m

Figure 14-4 Conventions for Variogram Search Parameters



The general variography approach used is as follows:

- Variogram parameters were selected with the aim of providing optimum directional coverage and taking into consideration the spatial distribution of both the thickness and P_2O_5 data sets;

- Absolute variograms were used for spatial continuity analysis as these generally produced the clearest variogram structure for all variables compared to other spatial continuity measures, e.g. correlograms;

- Selection and modelling of variogram orientations is based on visual evaluation of all variograms generated for stepwise azimuth and dip increments (5° increments between 0° and 180° azimuth and 1° increments between 5° and -5° plunges for thickness);

- Variogram plan maps are used as an indicator of the orientation of the major axes continuity in directing the evaluation of the variograms generated;

- Following visual inspection of the stepwise generated variograms, the modeller selects the major axes of continuity variogram and its orthogonal counterpart for modelling;

- Variograms were modelled using a two-structure spherical model. Modelling is an iterative process with the modeller starting with a nugget and single sill structure model, and then adding a second sill structure to produce the best fit between the variogram model and the variogram data; and

- Thickness and P_2O_5 variograms were generated using non-standardized variogram models.

14.6.1 Phosphate

Directional variography shows a direction of greatest continuity in the major direction of N95 in Figure 14-5 and in the semi-major direction of N05 in Figure 14-6. Maximum continuities in the order of 3,000 and 2,500 m respectively are observed.

Figure 14-5 Directional variogram in the major direction (N95) for P_2O_5 showing approximately 3,000 m maximum continuity

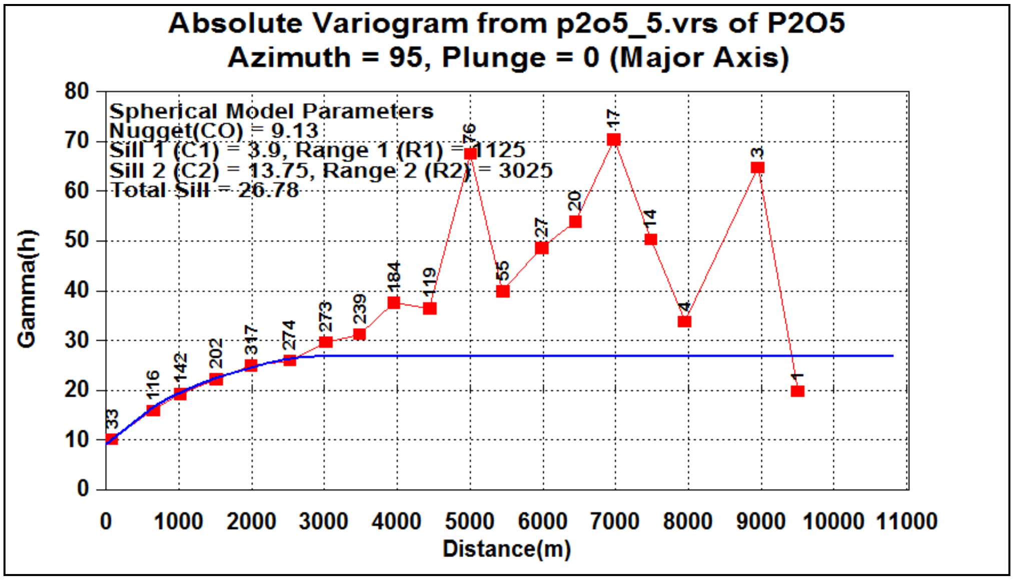
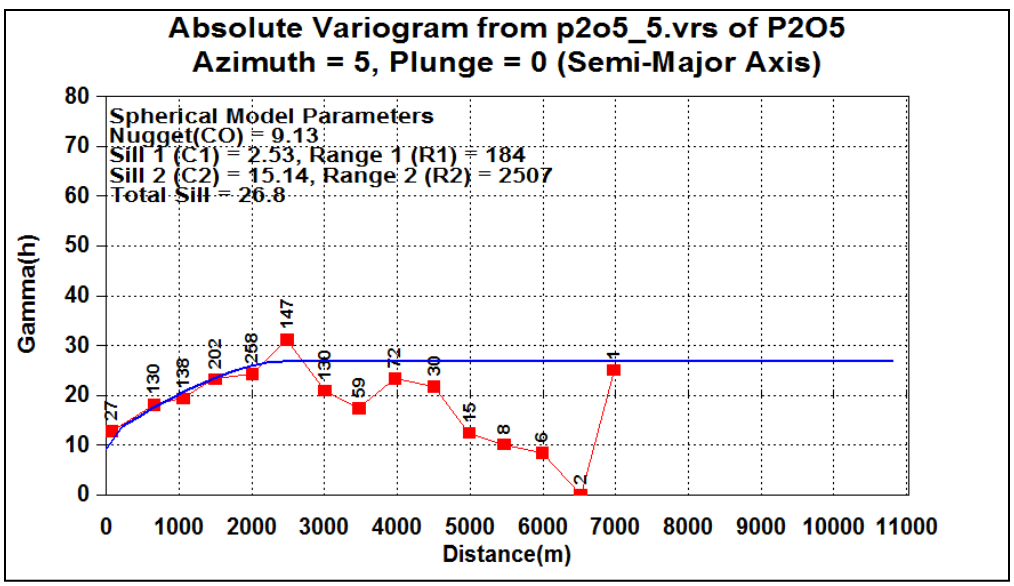


Figure 14-6 Directional variogram in the semi-major direction (N05) for P_2O_5 showing approximately 2,500 m maximum continuity



14.6.2 Thickness

Directional variography shows a direction of greatest continuity in the major direction of N10 in Figure 14-7 and in the semi-major direction of N01 in Figure 14-8. Maximum continuities in the order of 3,000 and 2,000 m respectively are observed. No cut-off was used for thickness.

Figure 14-7 Directional variogram in the major direction (N10) for thickness showing approximately 3,000 m maximum continuity

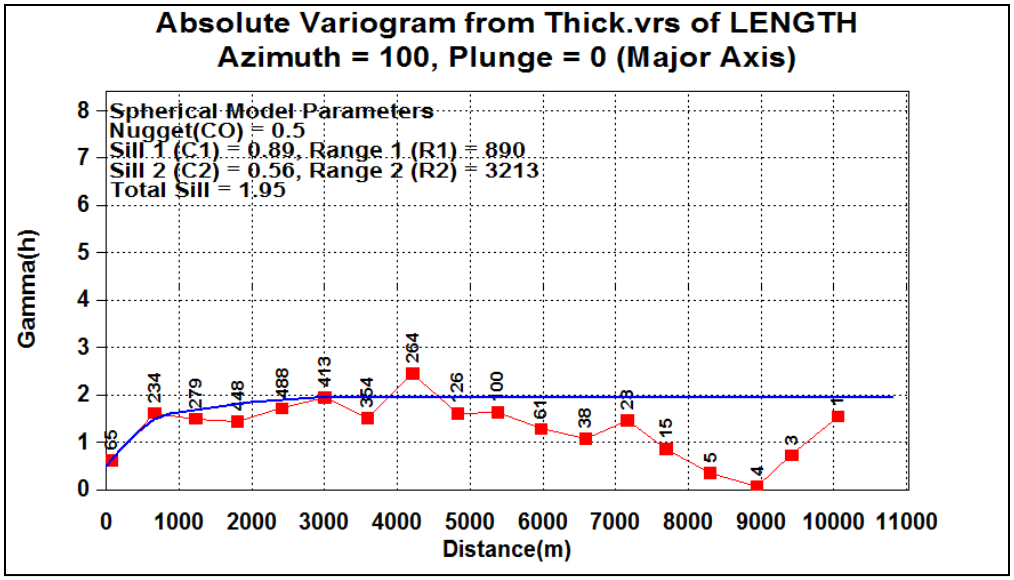
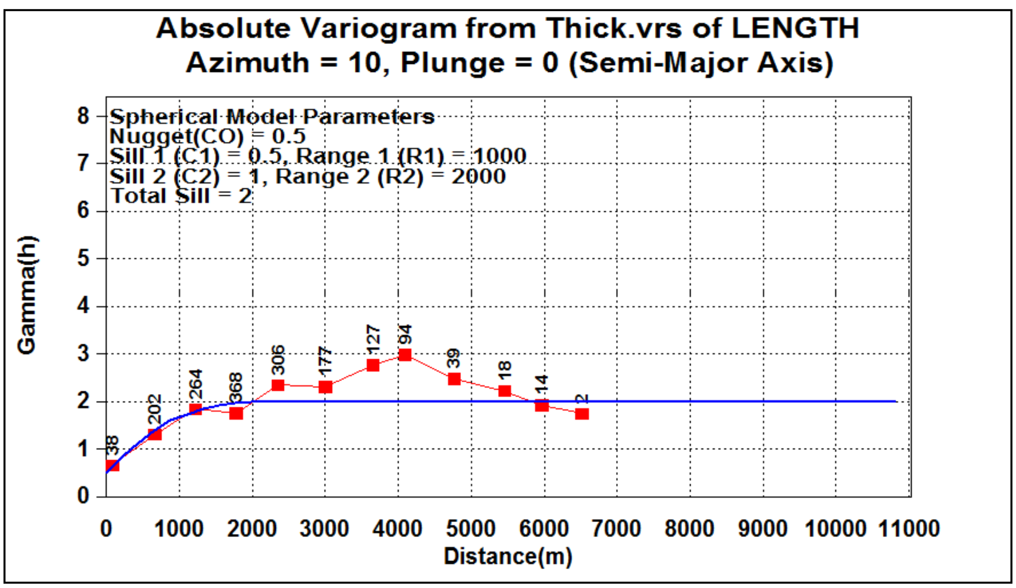


Figure 14-8 Directional variogram in the semi-major direction (N01) for thickness showing approximately 2,000 m maximum continuity



14.7 Summary of statistical and geostatistical assessment

The following are conclusions based on the statistical and geostatistical assessment of Al_2O_3, CaO, Fe_2O_3, P_2O_5, SiO_2 and sample thickness for the FPA horizon of the Farim phosphate project:

14.7.1 Phosphate

- The project possesses robust directional variograms for P_2O_5 displaying continuity in various directions. The direction of greatest continuity i.e. the major direction was in the E-W direction with the semi-major being in the N-S direction. Therefore, variography supports the geological observations that the FPA is very regular, sub-horizontal and continuous;

- The exploration drill pattern utilised has had a marked effect on variography results. The direction of greatest continuity (major direction) of mineralization appears to be different than the NE-SW and NW-SE orientated exploration drill pattern. This has resulted in some issues in terms of developing a good short range in the direction of greatest continuity (major direction), indicated by the variography. Average drill spacing in this direction is more like 700 m than 500 m;

- The nugget is approximately 33%. The nugget was picked using best fit from the variography. Due to the lack of data in the first 700 m of the variogram in the E-W direction, the true nugget may in fact be different from the modelled nugget in this study;

- The variography was sensitive to the bottom-cut and mostly likely the domaining of the P_2O_5 in the FPA horizon. EDA suggests that some of the lower FPA results appeared to be markedly different from the majority of the population. Some of these highlighted samples are proximal to the margin of the deposit. This may be a result of the FPA displaying different characteristics on the edge of the deposit or for example, these may include material from the underlying FPB material; and

- With this uncertainty in these samples in combination with often very poor sample recovery and limited knowledge of the drilling technique utilized for each drillhole, it is difficult to be confident in these samples which is turn has influenced the variography.

14.7.2 Thickness

- The deposit possesses robust directional variograms showing continuity in similar directions and ranges to those seen for P_2O_5; and

- The nugget is approximately 25%. The nugget was picked using best fit from the variography.

In some drillholes possessing poor recoveries, the thickness of the FPA has previously been reduced to the sample length recovered, in order to be conservative. This has had an effect on the geostatistics. It is recommended to consider exclusion of these uncertain samples for any future thickness geostatistical or resource estimation work.

14.8 Resource Estimation

14.8.1 Block Model Definition

A block model is used in resource estimation to calculate the unknown grade at uniform volumes across a deposit. It is a regular grid of blocks covering the area of the deposit. The size of the blocks within the model is decided according to the spacing of sample data and mining parameters. A guideline for block size is an optimum distance equal to half of the widest data spacing and a minimum of a quarter of data spacing, as well as consideration of the likely mining selectivity. Using a block that is too small presents a risk of over smoothing grade and providing apparent selectivity in mining which may not be achievable. This may result in local inaccuracies of grade and tonnage estimates and a lower (block model) variance than would be expected at the level of selectivity. This can impact the representativeness of the global grade-tonnage curve.

Table 14-3 shows the chosen block model parameters. The blocks are 125 m by 125 m reflecting a quarter of the average drill hole spacing of 500 m. This is appropriate considering the grade continuity.

Table 14-3 Block Model Parameters

Deposit		Origin (m)	Block size (m)	No. Blocks
Farim	X – N090	465,625	125	92
	Y – N000	1,373,875	125	76

14.8.2 Estimation Methodology

In order to ensure that the correct search (neighbourhood) parameters are used, the search ellipse which best reflects the continuity of the geology and the variogram ranges must be used. By determining the neighbourhood correctly, the most appropriate data for estimating a particular block can be determined.

Neighbourhood analysis was carried out to test the search distances, minimum number of composites and number of sectors required. A quadrant based search was adopted for the neighbourhood analysis and estimation. This is where the search ellipse is divided into four sectors. This helps to ensure that composites from more than one hole were used.

Variables were estimated using a three pass strategy, whereby each successive pass had an increased search radius and more relaxed sample selection criteria. This was to ensure all blocks received a value for each variable. Values were assigned using a combination of Ordinary Kriging and Inverse Distance Weighted methods for the following variables:

- P_2O_5 (OK);
- Al_2O_3 (IDW2);
- CaO (OK);
- Fe_2O_3 (OK);
- SiO_2 (IDW2);
- FPA Thickness, m (OK); and

- Overburden Thickness, m (IDW2).

Table 14-4 summarizes the final estimation parameters chosen following neighbourhood analysis.

Table 14-4 Farim Estimation Parameters

Criteria	Pass 1	Pass 2	Pass 3
Search distance, U	400	750	2000
Search distance, V	400	750	2000
Min samples total	3	2	1
No. sectors	4	4	4
Min samples per sector	2	2	2
Discretisation	5 x 5 x 1	5 x 5 x 1	5 x 5 x 1
Min sectors filled	3	2	1

14.9 Density

Dry density determinations were made by BRGM and Champion and are described in detail in Section 11. Density value estimates produced by BRGM and Bateman are considered valid after careful review of the density data. GeoImpact used a value of 1.43 t/m^3 for the FPA and 1.50 t/m^3 for the FPB in its resource estimation. A value of 1.40 t/m^3 for the FPA was used in the current Resource Estimate. In future resource estimations, further density measurements should be taken to increase the sample count and to allow for further evaluation of the data used to establish the default density values.

14.10 Block Model Validation

Validation against the raw input data is essential to ensure that the reproduction of drill hole grades is realistic and representative in the model. Both statistical and spatial aspects of validation are important on a global and local scale.

14.11 Statistics

Reproduction of the global statistical characteristics and the degree of smoothing in the model were assessed using comparisons of histograms, statistics and grade-tonnage curves.

Table 14-5 shows block model reproduction of composite values and global smoothing. Block average grades are within 10% of the equivalent composite grades for all variables. The degree of smoothing was only calculated for those variables for which it was possible to model a variogram. The degree of smoothing varies from -9% to +25%, which is an acceptable level of smoothing for this level of estimate.

14.11.1 Grade-Tonnage Curves

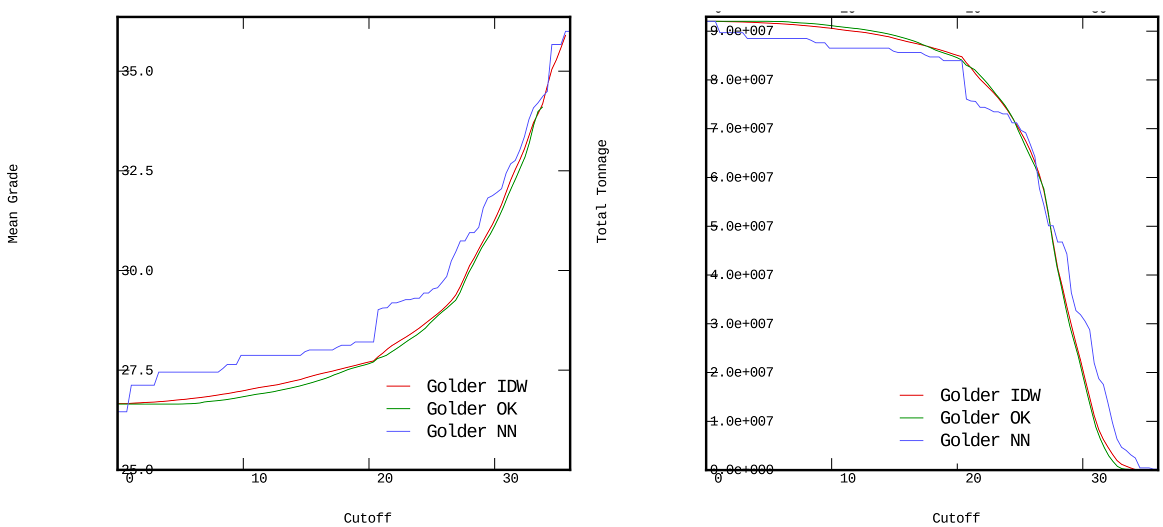
Graphs showing grade, tonnage and metal values versus cut off grades were plotted and attached in Appendix C of previous Golder report 11514950043.508/B.3. These compare the OK and IDW estimated block model curves, Figure 14-9 shows both grade and tonnage curves for P_2O_5. The IDW and OK models are similar, indicating the variogram model is not having a detrimental effect on the quality of the estimation.

Table 14-5 Block Model Validation, Statistical Comparison

Univariate Statistics of Composite Values and Block Estimates - First + Second Passes Only									
Variable	Drill Hole Composites		Block Estimates		EST/ CMP[1]	f [2]	f [3]	f diff[4]	Smoothing
	Mean	Variance	Mean	Variance	(%)				(%)
P_2O_5	28.69	27.50	27.32	20.14	95.22	0.732	0.923	0.190	19.0
Thickness	2.70	2.35	2.52	1.50	93.26	0.639	0.939	0.301	30.1
Thickness	43.10	107.30	43.70	119.30	101.34	1.111			
Al_2O_3	3.05	8.48	3.25	10.21	106.46	1.204			
CaO	39.62	36.64	38.61	25.41	97.46	0.693	0.901	0.207	20.7
Fe_2O_3	5.33	13.65	5.96	13.62	111.79	0.998	0.891	-0.107	-10.7
SIO_2	11.60	24.70	11.30	11.60	97.83	0.471			

Notes:
[1] Between composites and estimates mean values; [2] actual variance adjustment (VA); [3] theoretical VA; [4] between real and theoretical f factors

Figure 14-9 Farim, Grade-tonnage Curves, P_2O_5

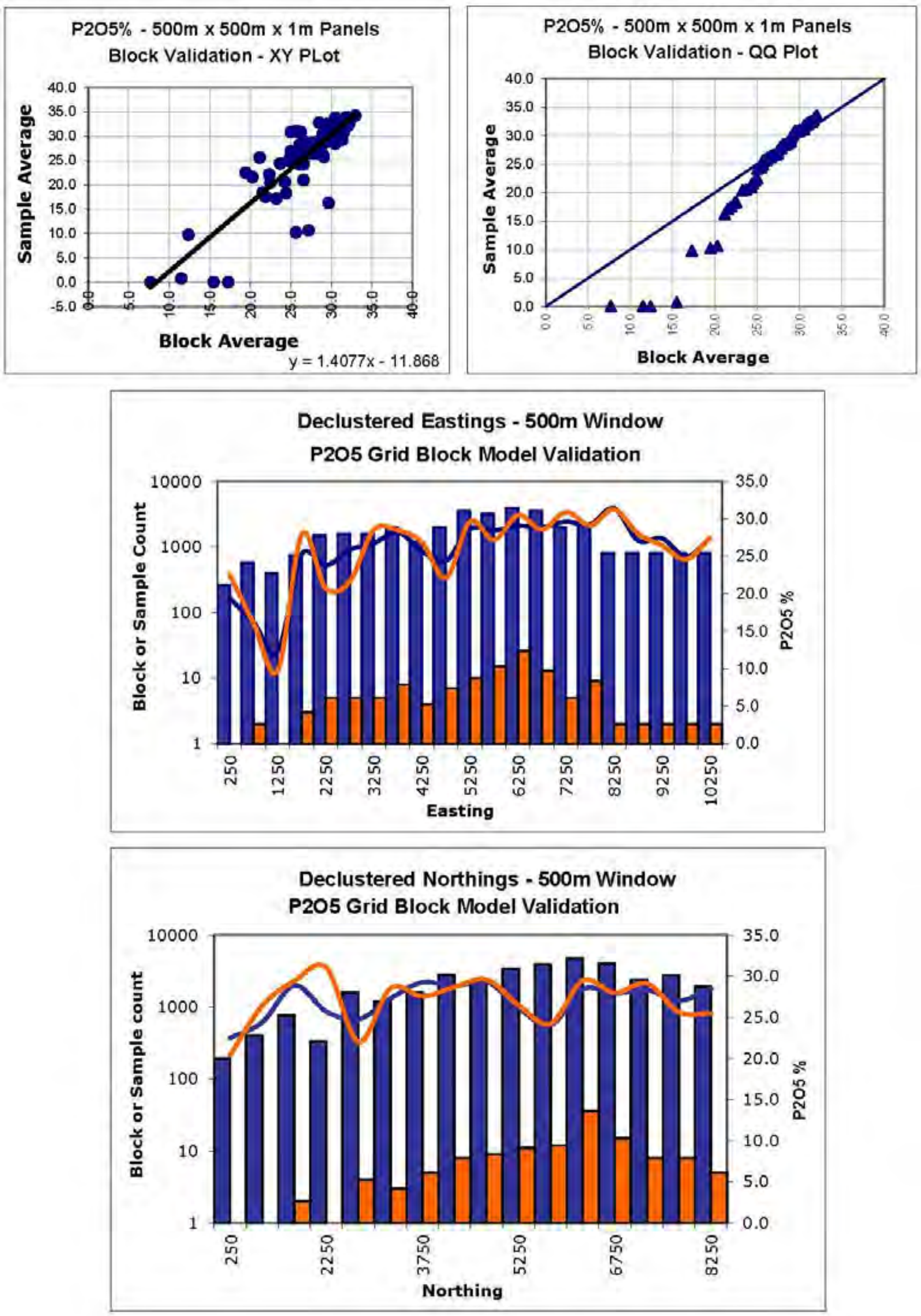


14.11.2 Swath Plots

Swath plots comparing local mean grades in broad "swaths" of the block model and corresponding composites were generated and are stored in Appendix C of previous Golder report 11514950043.508/B.3. This allows an analysis of local reproduction of composite grades by the block model.

Figure 14-10 shows an example of a P_2O_5 swath plot for the Farim deposit and corresponding Q-Q plot of the composites and blocks. The block model shows good global reproduction of composite grades of 25% P_2O_5 and above, but over estimation at low grades. This is not a significant issue are there are very few low grades within the FPA layer. The block model shows good local reproduction of composite P_2O_5 grades. Similar plots for FPA thickness showed very good reproduction of composite values on a global and local scale.

Figure 14-10 Farim, Swath Plots P_2O_5



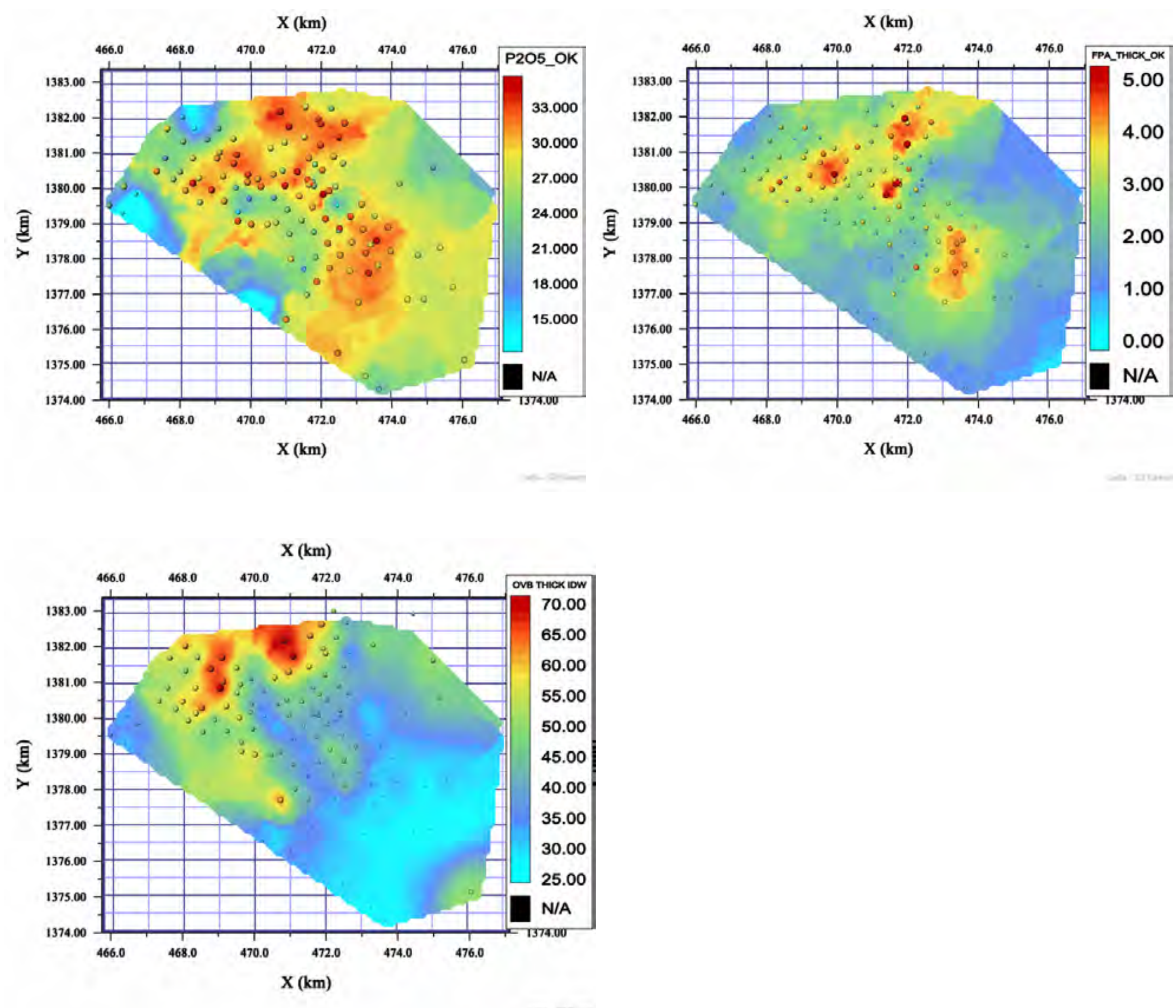
14.11.3 Visual Validation

Local and global grade patterns and variations were assessed visually by looking at a horizontal view of the model with the drill hole information in that slice displayed. This was done in Vulcan to ensure the local grade patterns of the composites are reproduced in the block model.

Figure 14-11 shows these sections for P_2O_5, FPA thickness and overburden thickness. Generally the block model shows good representation of the composite grades. There are clear areas with sparse data where sample grades can be seen spread over large distances.

Similar Plots for the other variable are shown in Appendix C of previous Golder report reference 11514950043.508/B.3.

Figure 14-11 Farim, Visual Validation - P_2O_5, FPA Thickness and Overburden Thickness



14.12 Mineral Resource Classification

Classification should be based on the confidence in the sampling data, geological knowledge, and geostatistical estimation.

Golder performed an updated statistical and geostatistical assessment of the FPA horizon for this Study using Golder's proprietary software, Ore Block Optimizer (OBO). A Technical Memorandum (TM) outlining the assessment's findings was provided as *Statistical and Geostatistical Assessment of the FPA Horizon – Farim Phosphate Project* (Golder, 2015). The following criteria have been applied to define Resources for the Project:

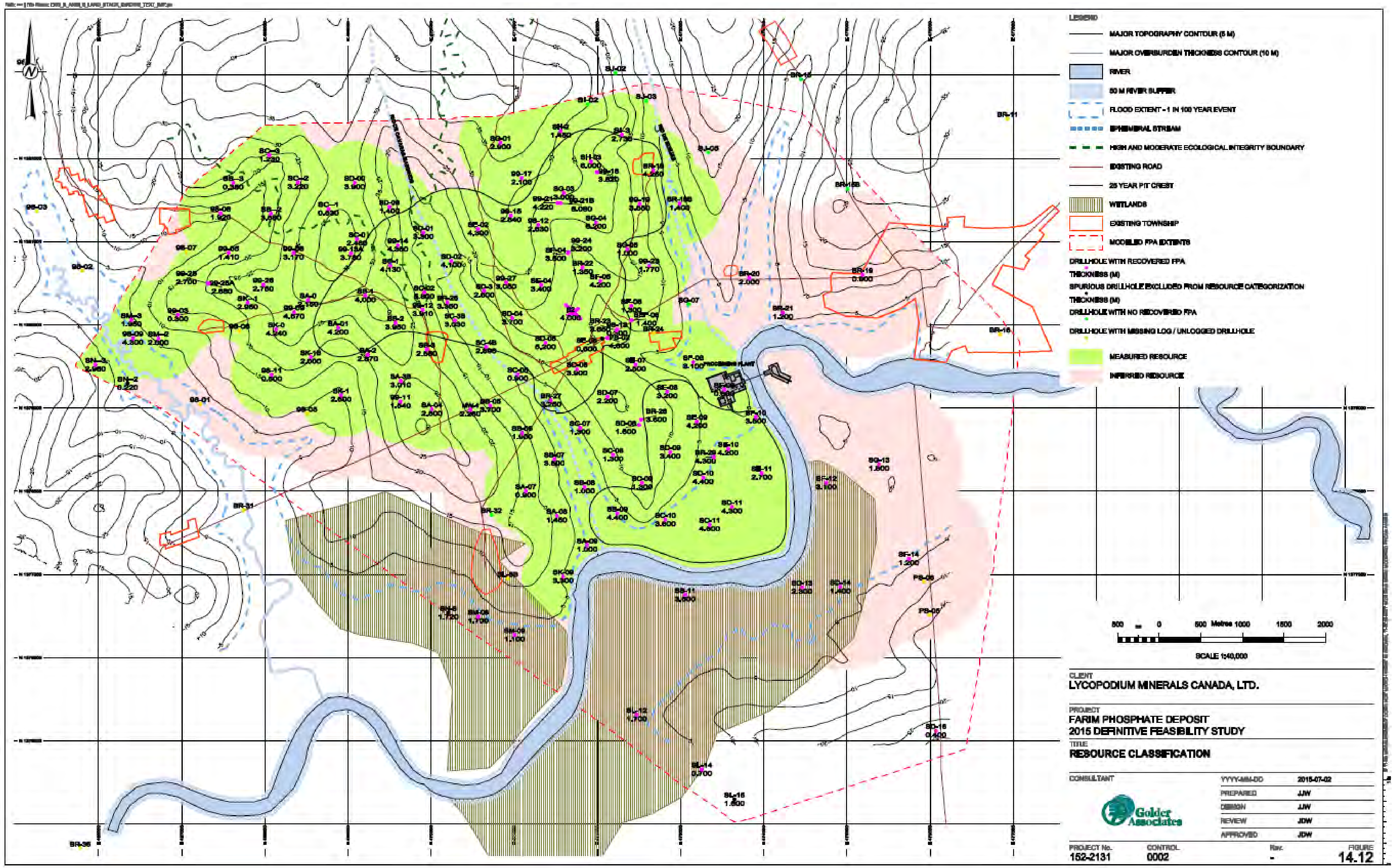
- Measured = Areas with samples within a 500 m radius (approximately 1/3 of the maximum continuity of 3,000 m) from drillholes classified as a Point of Observation (POB); and
- Inferred = Areas with samples within a 1,000 m radius (approximately 2/3 of the maximum continuity of 3,000 m) from drillholes classified as a POB.

A radius for Indicated Resources was not generated as it is the QP's opinion that the number of drillholes that could potentially be used as POB are too few. The density of drillholes quickly diminishes between resource classified as Measured and Inferred, so spacing between POB that would typically be used to classify Indicated Resources have instead been used to define Inferred Resources.

A nominal corridor of 50 m on either side of Cacheu River was also defined. FPA within this boundary was set to "unclassified" due to the uncertainty attached to the extraction of material in this area. A total of 28 drillholes missing lithology data and 8 drillholes with no observed FPA in the lithology were excluded as POB from the Resource classification. Drillholes SN-2, SN 8, and SL-15 were excluded as POB as they appear to possess spurious analytical data for P_2O_5 grade based on Exploratory Data Analysis (EDA). Additionally, drillhole SE-06 was excluded as POB due to spurious FPA thickness data in the EDA (Golder, 2015). In total, 144 of the 184 drillholes in the drillhole database were used as POB for Resource classification.

The resulting Resource classification is shown in Figure 14-12. The resource estimate has also been divided into FPA corresponding to location relative to Cacheu River: "North" or "South".

Figure 14-12 Farim Resource Classification



14.13 Mineral Resource Statement

Golder considers the mineralization contained within the Farim deposit to fulfil the criteria of “potentially economic” to be reported as a resource. A phosphate cut-off grade and maximum strip ratio were not applied to report the Mineral Resource Estimate. Instead, a minimum FPA thickness of 1 m was used to define a mineral inventory which has reasonable expectation of eventual economic extraction. This differs from Golder’s previous resource estimate in 2012, which applied a minimum FPA thickness of 1.5 m and a maximum strip ratio of 20 bcm/t. The minimum thickness has been reduced from 1.5 m as Golder's experience with similar mines indicates small backhoes can recover the FPA as thin as 1 m with minimal dilution and loss. No strip ratio cut-off has been applied as the Lerchs-Grossman (LG) optimizations used to define the most economical 25-year resources demonstrated potential for economic extraction of areas with a strip ratio greater than a 20 bcm/t. Further information regarding this LG optimization exercise is provided in Chapter 16: Mining Methods.

Table 14-6 and Table 14-7 summarize the results of the 2 July 2015 Mineral Resource Estimate based on a minimum FPA thickness of 1.0 m and a constant density of 1.4 t/m^3; estimated Resources within the extents of the 25-year pit design are provided in Table 14-6, and Table 14-7 summarizes the global Resource estimate. Additional information regarding the 25-year pit design is provided in Chapters 15 and 16. Golder considers the criteria used to define the mineral inventory to be reasonable for public reporting. This assumes the resource would be exploitable using open pit mining methods.

The 25-Year Mineral Resource Estimate, dated 2 July 2015, defines a Measured Resource of 46.7 Mt at an average grade of 30.6% P_2O_5. The Global Mineral Resource Estimate, dated 2 July 2015, defines a Measured Resource of 105.6 Mt at an average grade of 28.4% P_2O_5 and an Inferred Resource of 37.6 Mt at an average grade of 27.7% P_2O_5. Tonnage and grade have been rounded to an appropriate decimal place after calculations. No recoveries or dilution factors have been considered in this estimate and the results should be considered strictly *in situ*, in accordance with NI 43-101 reporting guidelines for resources.

Table 14-6 25-Year Mineral Resource Statement, Farim Phosphate Deposit, 2 July 2015

Class	Block	Tonnage, Dry Basis (Mt)	FPA (m)	P_2O_5, Dry Basis (%)	Al_2O_3, Dry Basis (%)	CaO, Dry Basis (%)	Fe_2O_3, Dry Basis (%)	SiO_2, Dry Basis (%)	Overburden (Mbcm)	Stripping Ratio (bcm/t)
Measured	North Pit	32.2	3.77	30.31	2.66	41.17	5.15	10.36	318.0	9.87
	South Pit	14.4	3.77	31.23	2.34	40.51	3.77	11.21	102.9	7.13
	Subtotal	46.7	3.77	30.59	2.56	40.96	4.72	10.62	420.9	9.02
Indicated	North Pit	-	-	-	-	-	-	-	-	-
	South Pit	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-
Measured + Indicated	North Pit	32.2	3.77	30.31	2.66	41.17	5.15	10.36	318.0	9.87
	South Pit	14.4	3.77	31.23	2.34	40.51	3.77	11.21	102.9	7.13
	Subtotal	46.7	3.77	30.59	2.56	40.96	4.72	10.62	420.9	9.02
Inferred	North Pit	-	-	-	-	-	-	-	-	-
	South Pit	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-

Notes:

Assumes a minimum FPA seam thickness of 1 m

FPA within 50 m of River Cacheu has been assigned as "unclassified" due to the uncertainty attached to the extraction of material in this area.

Table 14-7 Global Mineral Resource Statement, Farim Phosphate Deposit, 2 July 2015

Class	Block	Tonnage, Dry Basis (Mt)	FPA (m)	P_2O_5, Dry Basis (%)	Al_2O_3, Dry Basis (%)	CaO, Dry Basis (%)	Fe_2O_3, Dry Basis (%)	SiO_2, Dry Basis (%)	Overburden (Mbcm)	Stripping Ratio (bcm/t)
Measured	North of River	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
Indicated	North of River	-	-	-	-	-	-	-	-	-
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-
Measured + Indicated	North of River	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	105.6	2.87	28.41	2.68	39.74	5.66	11.24	1,193.0	11.30
Inferred	North of River	11.4	1.71	24.88	2.84	39.63	4.42	10.52	210.9	18.44
	South of River	26.2	2.12	28.99	5.37	35.90	5.28	11.58	258.2	9.85
	Subtotal	37.6	1.98	27.74	4.60	37.03	5.02	11.26	469.0	12.46

Notes:

Assumes a minimum FPA seam thickness of 1 m.
FPA within 50 m of River Cacheu has been assigned as "unclassified" due to the uncertainty attached to the extraction of material in this area.

15.0 MINERAL RESERVE ESTIMATES

In accordance with NI 43-101, for estimating resources and reserves of the Farim Phosphate Project, Golder has applied the definitions of “Mineral Resource” and “Mineral Reserve” as set forth in the updated CIM Definition Standards adopted 10 May 2014 (CIMDS) by the Canadian Institute of Mining, Metallurgy, and Petroleum Council.

A Mineral Reserve is defined as “… the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.” A Mineral Reserve is subdivided into two classes, Proven and Probable, with the level of confidence reducing with each class respectively. The CIMDS provides for a direct relationship between Indicated Mineral Resources and Probable Mineral Reserves, and between Measured Mineral Resources and Proven Mineral Reserves. In certain situations, Measured Mineral Resources could convert to Probable Mineral Reserves because of uncertainties associated with the modifying factors that are taken into account in the conversion from Mineral Resources to Mineral Reserves. Inferred Mineral Resources cannot be combined or reported with other categories.

Except as stated herein, Golder is not aware of any modifying factors exogenous to mining engineering considerations (i.e., competing interests, environmental concerns, socio-economic issues, legal issues, etc.) that would be of sufficient magnitude to warrant excluding reserve tonnage below design limitations or reducing reserve classification (confidence) levels from Proven to Probable or otherwise.

15.1 Introduction

As detailed in Section 14, the Farim deposit has been delineated over an area of approximately 40 km^2 and is divided by the Cacheu River. The deposit consists of both FPA and FPB mineralised units. This Mineral Reserve Estimate concerns FPA only, as the FPB unit was previously deemed to be uneconomic. No additional mineralisation outside the modelled deposit was considered in the Mineral Resource and Reserve Estimates.

The reserve estimation was undertaken in Ventyx®’s Minescape™ software (Version 5.8). The Mineral Reserve statement is effective 24 June 2015.

15.2 Geological Model Development

15.2.1 Block Model Conversion

The model quality and tonnage are on a dry basis using a density of 1.4 t/m^3 (dry basis). The 125 m by 125 m two-dimensional (2D) in situ block model discussed in Section 14 was used to develop three-dimensional (3D) grid-based geological surfaces of overburden and matrix in Minescape to aid in the planning work and reserves estimation through a multi-step process. These surfaces were re-blocked into a 25 m by 25 m by 5 m 3D block model for pit optimization purposes in Vulcan.

The 2D block model was re-blocked into 25 m by 25 m blocks using nearest neighbour linear interpolation to estimate in situ grades, volumes and dry tonnages. The resultant 25 m by 25 m 2D block model was checked against the original 125 m by 125 m 2D block model to confirm total tonnages, volumes, and average grades were not compromised due to the linear interpolation. This 25 m by 25 m 2D block model was then output to an Excel spreadsheet for import into Minescape to construct a 3D block model for pit optimization and to develop geological surfaces of overburden and matrix to aid in the planning work. Exported data included the Project area topographic surface from LiDAR survey data, block centroid easting and northing coordinates, overburden thickness, matrix thickness and assayed quality data. Assayed qualities for the matrix include P_2O_5 grade and the contaminants Al_2O_3, CaO, Fe_2O_3, and SiO_2. Triangulation surfaces for the in situ FPA roof and floor were also provided.

Geological grid-based surfaces of overburden and in situ matrix were created in Minescape using the LiDAR survey data and in situ FPA roof and floor triangulations obtained from the 2D block model. In situ FPA matrix quality data were imported into Minescape from the 2D block model as 25 m by 25 m grid-based surfaces using the block centroids as the grid nodes; assayed qualities were assumed to be evenly distributed from the FPA roof to the FPA floor. All geological model data imported into Minescape were checked to confirm the original data integrity was maintained and that the conversion of the 2D block model to 3D geological surfaces was successful.

15.2.2 Criteria for Determination of ROM Phosphate Matrix

Run-of-mine (ROM) mining surfaces were created in Minescape to account for anticipated 100 mm roof mining loss and 75 mm floor dilution gain where the FPA seam was greater than the minimum mineable thickness of 1 m. These anticipated dilution and mining loss factors are based on extracting the matrix with small excavators. An additional geology and mining recovery factor of 95% was applied when calculating ROM tonnages. ROM quality surfaces were also developed to account for the mining losses and dilution gains. Dilution material was assumed to have 0% P_2O_5 concentration and identical contaminant concentrations as the FPA matrix directly above it. The FPA was considered as a single unit with no plies or splits modelled.

15.2.3 Beneficiation Plant Yield and Product Quality Model

The effects of beneficiation on run-of-mine (ROM) material and P_2O_5, Al_2O_3, CaO, Fe_2O_3, and SiO_2 grades have been confirmed by both bench scale testing and pilot plant testing, and are detailed in Chapter 13 of this report. The results show a mass recovery of 75.5% and P_2O_5 product grade of 34%.

15.2.4 Development of the 3D Block Model for Pit Optimization

After developing the ROM surfaces, the grid-based Minescape model was blocked into 3D blocks 25 m by 25 m by 1 m in the X, Y, and Z, respectively, for the purposes of pit optimization. Using the same limits as the original 2D Vulcan block model, approximately 4.6 million blocks were created. The relevant geological and quality assay data for each block was populated using Minescape’s resource estimation functions; matrix tonnages were estimated based on a constant density of 1.4 t/m^3 (dry basis) per the resource estimation methodology. The 3D block model was checked against both the 25 m by 25 m 2D Vulcan block model and Minescape reserves to confirm that data were honoured and that no volumes, tonnages, or assay data were altered. After review, the Minescape block model was compiled into a format that Vulcan software could read for optimization purposes.

15.3 Mineral Reserve Estimation Methodology

The assessment of surface mineable phosphate matrix reserves within the Project area was based on the 25-year mine plan and corresponding open pit design. The pit design was developed based on a pit optimization exercise that delineated the most economical 44 Mt of ROM material to feed a 25 year plan at a rate of 1.75 Mtpa on a dry basis. The development of the 25-year mine plan pit is covered more closely in Section 16.6.

A series of nested LG pits were developed over a range of commodity prices with the goal of targeting the most cost effective 44 Mt of plant feed. At the time the optimizations were performed, the expected average mass yield of the ROM matrix was 70%. Consequently the LG optimizations were developed using this recovery. Mining unit costs used in the optimizations were based on Golder's experience with similar projects and were adjusted for project specific diesel prices and labour costs. Beneficiation costs, port land costs, and ship loading costs were provided by Lycopodium Minerals Canada, Ltd. (Lycopodium). Table 15-1 summarizes the unit costs used in the pit optimization analysis.

Table 15-1 Summary of the Pro Forma Unit Costs used in the Pit Optimization Analysis

Description	Value (US$ / Unit)	
Total Overburden Stripping Cost[1]	$1.56	/ bcm
Total Matrix Mining Cost[2]	$4.01	/ ROM tonne
Beneficiation[3]	$7.64	/ ROM tonne
Port Land Costs[3]	$3.98	/prod. tonne
Shiploading[3]	$2.69	/ prod. tonne

Notes:
[1] Cost includes overburden stripping and haulage, operations support, and mine maintenance. Cost assumes a diesel price of $0.80/liter.
[2] Cost for the site includes matrix mining and haulage, stockpiling, pit dewatering, reclamation, and mine supervision and administration. Cost assumes a diesel price of $0.80/liter.
[3] Cost provided by Lycopodium Minerals Canada, Ltd.

The 3D block model loaded into Vulcan and adjusted with dilution and mining losses was used along with the unit costs to calculate total costs associated with each block. Revenue was calculated based on the calculated rock product at varying commodity prices.

As per the Mineral Resource Estimation methodology, a true phosphate cut-off grade was not applied to the Mineral Reserve Estimate. However, Golder applied a penalty to blocks with ROM grade values lower than 29% P_2O_5 and rewarded blocks with a ROM grade value greater than 29% P_2O_5 in the optimizations. Because the effects of beneficiation on phosphate rock P_2O_5 grade at Farim were not well defined at the time the optimization exercise was performed, this proration better ensures that minimum specifications for phosphate rock P_2O_5 grade can be achieved as P_2O_5 recovery generally increases with higher ROM (plant feed) P_2O_5 grade.

Based on the needs of the 25 year, 1.75 million tonnes per annum (Mtpa) mine plan, the final pit configuration was a slight modification of the USD $52/t of phosphate rock pit optimizations. The USD $52/t price pit resulted in two distinct pits. The resulting pit shell limits for these incremental pits were exported from Vulcan and imported into Mincom as the basis for pit designs, mine planning and reserve estimation.

The design criteria for the final pit configuration are shown in Table 15-2.

Table 15-2 Summary Table of Mine Design Parameters

Description	Value
Permanent wall angle	20°
Permanent wall operational FOS	>1.3
Bench Height	10 m
Short-Term Bench Face (Batter) Angle	65°
Short-Term Berm Width	14.9 m
Long-Term Bench Face (Batter) Angle (After Sloughing)	25°
Long-Term Berm Width (After Sloughing)	6.5 m
Overburden angle of repose OSF/IOB/SOS	1V:4H / 1V:6H / 1V:6H
Overburden spoil swell factor	27%
Total Moisture (As-Received Basis), Overburden	20%
Overburden Density (As-Received Basis)	2.10 t/m^3
Overburden Density (Dry Basis)	1.68 t/m^3
Total Moisture (As-Received Basis), Matrix	20%
Matrix Density (As-Received Basis)	1.75 t/m^3
Matrix Density (Dry Basis)	1.40 t/m^3
Minimum mineable matrix thickness	1 m
Mining roof loss	100 mm
Mining floor dilution	75 mm
Geology and mining recovery factor	95%
Buffer between pit and river	100 m
Full production mining months per year	9 months
Reduced production mining months per year	3 months
Mine dewatering possible	Yes
Material to support truck traffic	Yes
Spoil Stackability	Yes

15.4 Mineral Reserve Estimation Statement

Estimated ROM phosphate matrix reserves and phosphate rock reserves for the proposed 25 year, 1.75 Mtpa pit are listed in Table 15-3 below. Golder considers the criteria used to define the 25 year mineral inventory to be reasonable for public reporting. However, adequate financing and permitting will be required prior to the commencement of the project.

Table 15-3 Proven and Probable Reserves

Category	Units	Phosphate Matrix Reserves		
		Proven	Probable	Total/Average
ROM FPA Tonnes (Dry Basis)	Mt	44.0	-	44.0
ROM %P_2O_5 (Dry Basis)	%	30.0	-	30.0
ROM %Al_2O_3 (Dry Basis)	%	2.6	-	2.6
ROM %CaO (Dry Basis)	%	41.0	-	41.0
ROM %Fe_2O_3 (Dry Basis)	%	4.7	-	4.7
ROM %SiO_2 (Dry Basis)	%	10.6	-	10.6

The Measured and Indicated Resource estimates as stated in Section 14 are inclusive of the resources comprising the Proven and Probable Reserve estimates described in Table 15-3.

For the Farim Phosphate Deposit Beneficiation Option the total estimated Proven and Probable Reserves are 44.0 Mt (dry basis) with an average ROM P_2O_5 grade (dry basis) of 30.0%. The overall ROM strip ratio is estimated to be 10.26 bank cubic meters (bcm) per tonne of ROM phosphate matrix, requiring the removal of approximately 451.7 million bcm of overburden over the life of the mine.

A drawing showing the breakdown of Proven and Probable reserves within the 25-year mine plan pit is provided as Figure 15-1.

Golder subsequently used the 25-year mine plan pit extents as the basis for the preparation of a mine scheduling database. This involved estimates of phosphate matrix and overburden volumes and tonnages on detailed bench and block splits to allow subsequent simulation of mine development by excavator and truck methods.

15.5 Discussion of Potential Impacts of Relevant Factors on Mineral Reserve Estimate

As stated in Section 14, Golder used a dry density of 1.4 t/m^3 for the resource and reserve estimates. However, a previous resource estimate by others used a value of 1.43 t/m^3 and 1.50 t/m^3 for the FPA and FPB mineralized units, respectively. In future resource and reserve estimations, further density measurements should be taken.

A basic assumption of this Report is that the estimated phosphate matrix resources and reserves at the Project have a reasonable prospect for development under the existing circumstances and assuming a reasonable outlook for all issues that may materially affect the mineral resource estimates.

Failure to achieve reasonable outcomes in the following areas could result in significant changes to the resources and reserve estimates presented in this Report.

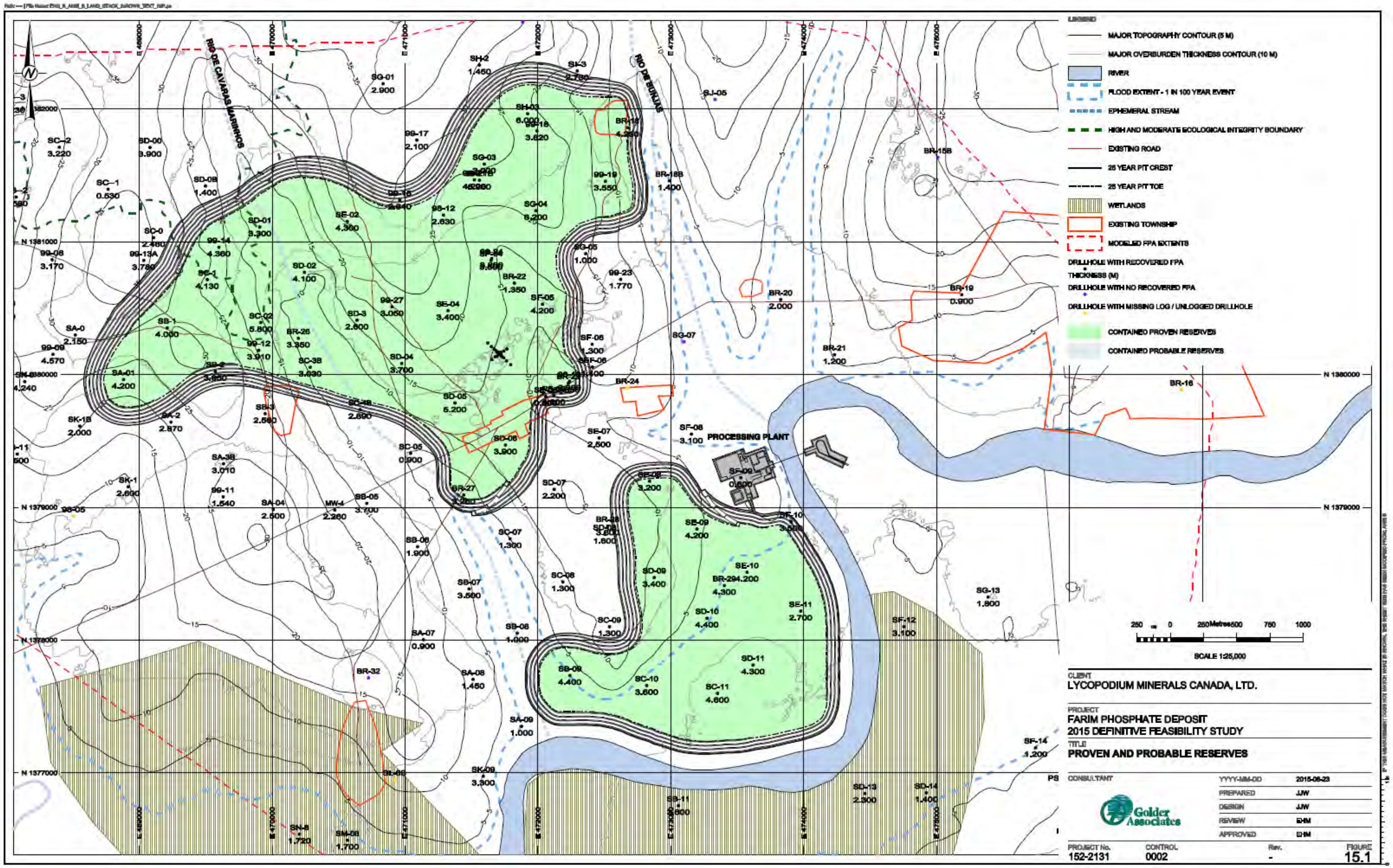
The Mineral Reserve Estimate anticipated a roof mining loss of 100 mm, a floor dilution gain of 75 mm, and a geology and recovery factor of 95%. These anticipated dilution and mining loss factors are based on extracting the matrix with small excavators. Additionally, due to the lack of sampling of dilution material was assumed to have 0% P_2O_5 concentration and identical contaminant concentrations as the FPA matrix directly above it. Should any one of these dilution or mining factors materially change, a new Mineral Reserve Estimation must be performed to account for its effects on tonnages and/or qualities.

A market for the product at current and forecast prices for the product phosphate rock at the Farim Phosphate Project is required to begin mining. The necessary mining licenses are in place, and contingent upon approval of this Technical report and the ESIA.

15.6 Potential for Future Reserve Expansion

As stated in Section 15.3, the Mineral Reserves Estimation is based solely on the 25-year mine plan open-pit design with highwall laybacks and a production rate of 1.75 Mtpa (dry basis). Resources outside of the 25 year pit extents were not considered in the Mineral Reserve Estimation dated 24 June 2015. There is strong indication of future reserve expansion through further economic evaluation. Future studies should investigate expanding reserves to include current resources outside of the 25 year pit.

Figure 15-1 Proven and Probable Reserves



16.0 MINING METHODS

16.1 Mining Method Options

The Project site is contained within a low lying, generally flat area. The surface is open, semi-arid savannah woodland with active subsistence agriculture throughout the Project area. The site contains a high-grade sedimentary, flat-lying phosphate deposit located within a single phosphate matrix bed known as the FPA matrix zone. Due to the geological and topographic characteristics of the deposit, three conventional surface mining methods were analyzed:

- Dredging overburden and matrix;
- Dragline with matrix slurry transport;
- Excavator/truck with matrix truck transport; and

A brief description of each method analysis follows.

16.1.1 Dredging Mining Method

Golder reviewed previous project studies which considered multiple level dredging operations to remove the overburden and matrix in a linear sequence. This method uses cutter-suction dredges, which float above the digging operation within contained water impoundments. Phosphate matrix is pumped to large storage tanks at the process plant.

Golder rejected this possible mining method for the following reasons:

1) Poor mining control of the matrix layer, resulting in lower grade of product;
2) Anticipated low angle of repose of dredge spoils;
3) Anticipated high overburden swell factors due to dredging soils with high clay contents;
4) Amount of storage area required to contain dredged spoils due to Items 1 and 2; and
5) Amount of embankment construction required to contain dredged spoils due to Items 1 and 2.

Golder rejected dredging in favor of a dry mining method, which will provide better spoil containment control, mining grade control, and product quality.

16.1.2 Dragline Mining Method

This method is commonly and successfully employed in United States east coast phosphate mines.

Large electric walking draglines remove the overburden directly above the phosphate. The overburden material is cast into a previous mined-out location within the operating radius of the dragline. The dragline then carefully extracts the phosphate matrix and places the matrix onto the operating bench in large piles.

The matrix is typically placed as far from the operating face as possible, given the operational reach of the machine. A sectional view of a typical dragline operation is shown in Figure 16-1 on the coming pages.

The matrix piles are then slurried using a slurrifier, or pit car, using multiple high pressure water jets. The matrix slurry flows into a prepared matrix well adjacent to the matrix pile and pit car. From the matrix well, the matrix slurry is pumped to large storage tanks at the process plant. A plan view of this configuration is provided in Figure 16-2.

The pit car relocates with the dragline as the operation moves down the face over the width of the mine. Once the pit reaches the end of the pit limit, the entire operation is relocated toward the matrix and water return pipelines for a new 40-metre (m) pit. Pipelines are typically located approximately 200 m to 250 m from the start of an operation, so multiple dragline/slurry setups can be achieved without relocation of the main pipelines.

The pre-strip operation (shovel/truck stripping above the working bench) typically advances 500 m or more ahead of the dragline mining face. Pre-strip overburden (i.e., overburden above the dragline working bench) is handled by shovels/excavators and haul trucks. Overburden is initially hauled from the pit to an external waste dump (WD) until a large enough mined-out area can be established. Once sufficient mined-out area is developed, pre-strip overburden is backhauled into the mined-out area for placement and final storage.

The advantages of the dragline mining method include the following:

- Ability to allocate some of the overburden to typically lower cost dragline operations;
- No trafficability or operating/travel surface requirements for equipment below the dragline working bench;
- Minimal support equipment (additional bench equipment) required; and
- Lower number of required trucks, haul road maintenance, etc.

Disadvantages of this mining method include the following:

- Higher up-front capital investment for draglines;
- Greater investment risk;
- Limited grade control and blending capability of the matrix due to the fixed linear progression of the pit;
- Reliance on stackability of spoil material in mined-out;
- Higher required standards for bench preparation (levelling, dewatering, etc.); and
- Power requirements to run draglines, pit cars, pumps and boosters.

Golder rejected the dragline mining method due to excessive up-front required capital investment, power requirements for the mining operation, and limited grade control or blending capability of the matrix for consistent product requirements. Golder rejected the dragline mining method in favor of an open pit, excavator/truck mining method which requires less capital investment, reduces investment risk, and can use diesel mining equipment.

Figure 16-1 Typical Dragline Range Diagram

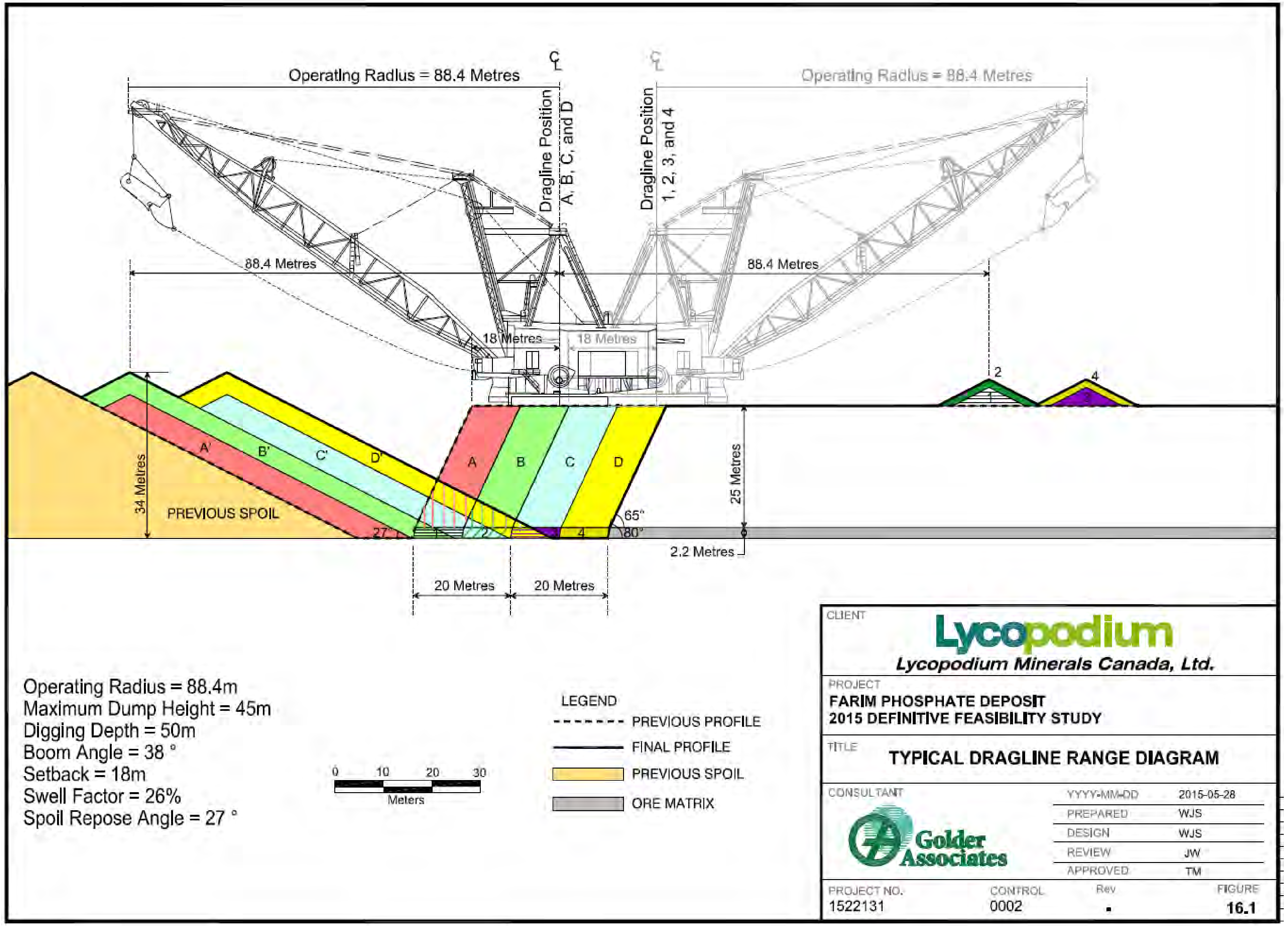
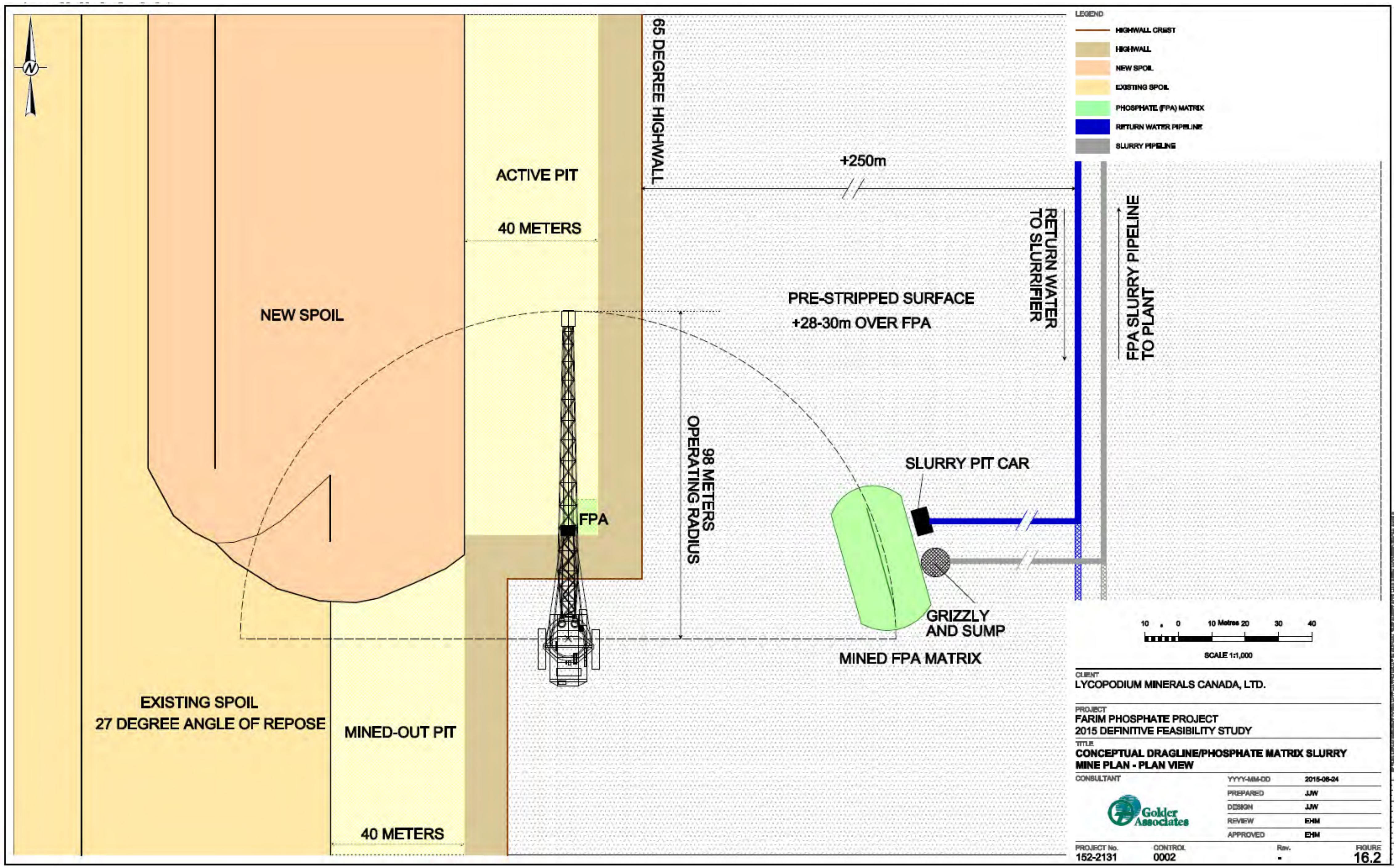


Figure 16-2 Conceptual Dragline/Phosphate Matrix Slurry Mine Plan – Plan View



16.1.3 Excavator/Truck Mining Method

This method uses excavators and trucks to handle 100 percent of the overburden and matrix.

The FPA matrix is mined by a multiple bench open pit haul back mine using excavators and trucks. This method uses a boxcut that requires storage of overburden outside the pit while the initial pit is developed. Once a sufficient volume has been excavated, the overburden is back-hauled into the mined-out area. Based on in-pit overburden backfill (IOB) design slopes and required mined-out area necessary to allow overburden be backfilled within the pit, it is estimated that some in-pit backfilling will become feasible in the first year of matrix production. Overburden not stored in-pit will either be sent to an ex-pit WD or to surcharge overburden storage (SOS) located above the existing IOB. The benching and excavation depths will depend on the actual overburden depth and will be altered to accommodate thicker overburden.

For the 1.75 million tonnes per annum (Mtpa) (dry basis) open pit, it is planned that overburden will be stripped and removed with 12 cubic metre (m^3) front end loaders (FEL) or other similar excavator matched with 97 tonne (t) capacity haul trucks. The matrix will be mined with 5 m^3 bucket class backhoes matched with 36 t capacity trucks to minimize mining dilution and maximize matrix recovery. The matrix will be hauled to a 175,000 t (dry basis) ROM stockpile adjacent to the plant, and segregated by quality. The matrix will be reclaimed and carefully blended into a plant feed hopper by front-end wheel loaders with 12 m^3 buckets to achieve the desired product P_2O_5 grade. The plant feed hopper will be installed so that matrix haul trucks can directly feed matrix to the plant if possible.

Overburden excavation will advance ahead of the matrix extraction in maximum 10 m height production benches. Because the overburden thickness is greater than 30 m within the 25 year pit, multiple overburden stripping benches will be developed and maintained in advance of the matrix extraction.

Figure 16-3 shows a typical pit configuration for this method of mining.

The advantages of the excavator/truck mining method include:

- Lower up-front capital investment;
- Better control of stackability of spoil material in the mined-out;
- Can run on diesel power;
- More operational flexibility on mining layouts, if necessary, for consistent grade control; and
- Lower investment risk.

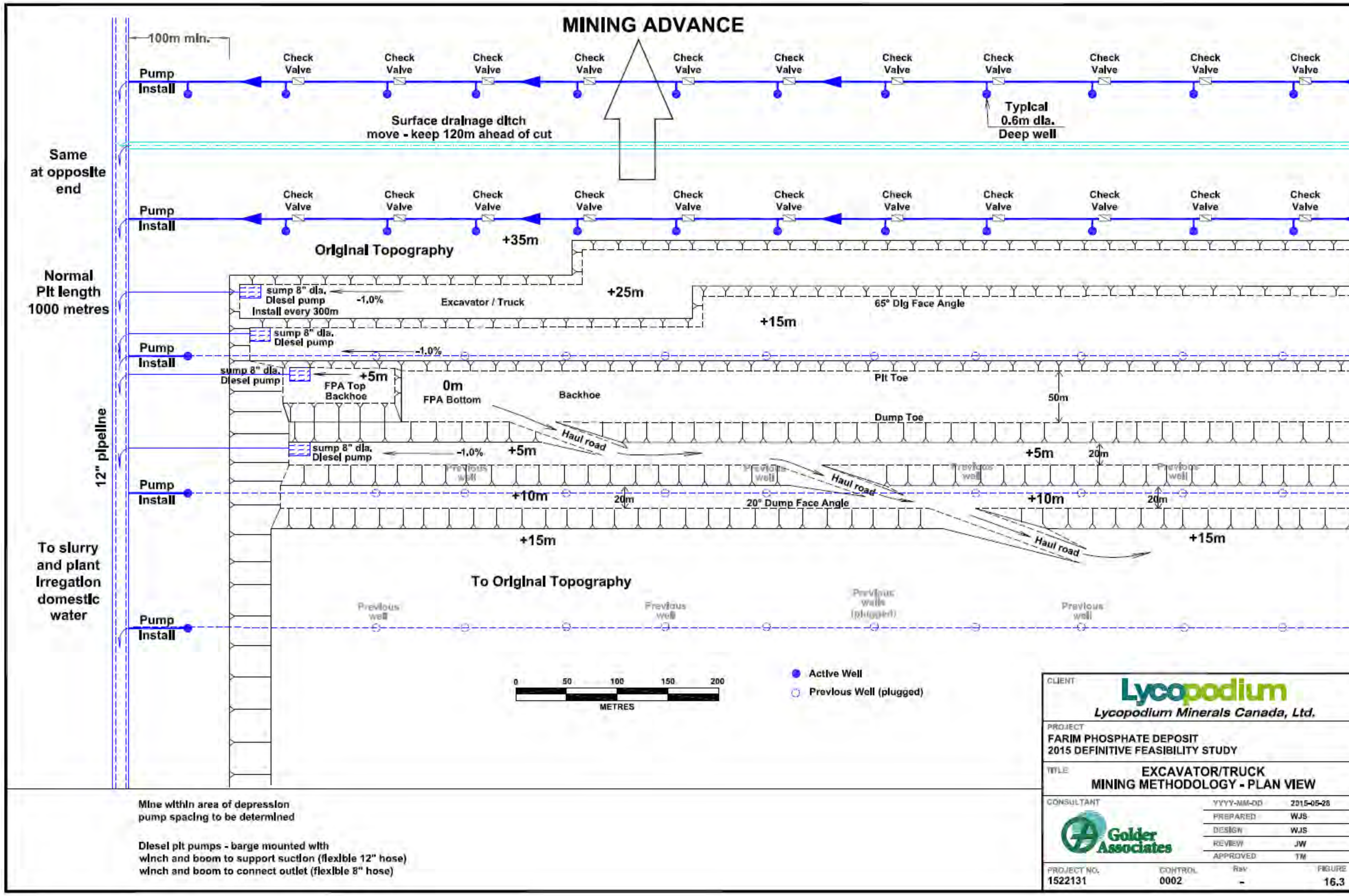
Disadvantages of this mining method include:

- Typically higher operating costs;
- Higher equipment trafficability requirements on operating/travel surfaces due to reliance on truck haulage, including access to the bottom (FPA matrix zone) of the pit;

- More support equipment required; and
- More required trucks, additional haul road maintenance, etc.

Golder selected the excavator/truck mining method for the 1.75 Mtpa Option based on lower initial capital, lower investment risk, increased grade control, limited power supply, and flexibility to adapt to a smaller scale Direct Shipping Option (DSO) operation if needed.

Figure 16-3 Base Case Excavator/Truck Mining Methodology – Plan View



16.2 Surface and Groundwater Constraints

The Project area experiences a five month rainy season, occurring from June to October, but is concentrated in mid-July to mid-September. From December through April, the country experiences drought with no significant rainfall. The 2006 rainfall data provided by the Bissau metallurgical office indicates that Guinea-Bissau experienced around 1,400 millimeters (mm) of rainfall for the year, with 88 percent of the annual rainfall occurring from July to September. The heaviest rain occurs in August, which experiences approximately 36 percent of the total annual rainfall. See the total monthly and daily average rainfall charts from 2006 in Figure 16-4 and Figure 16-5.

Figure 16-4 Bissau Total monthly Rainfall in 2006

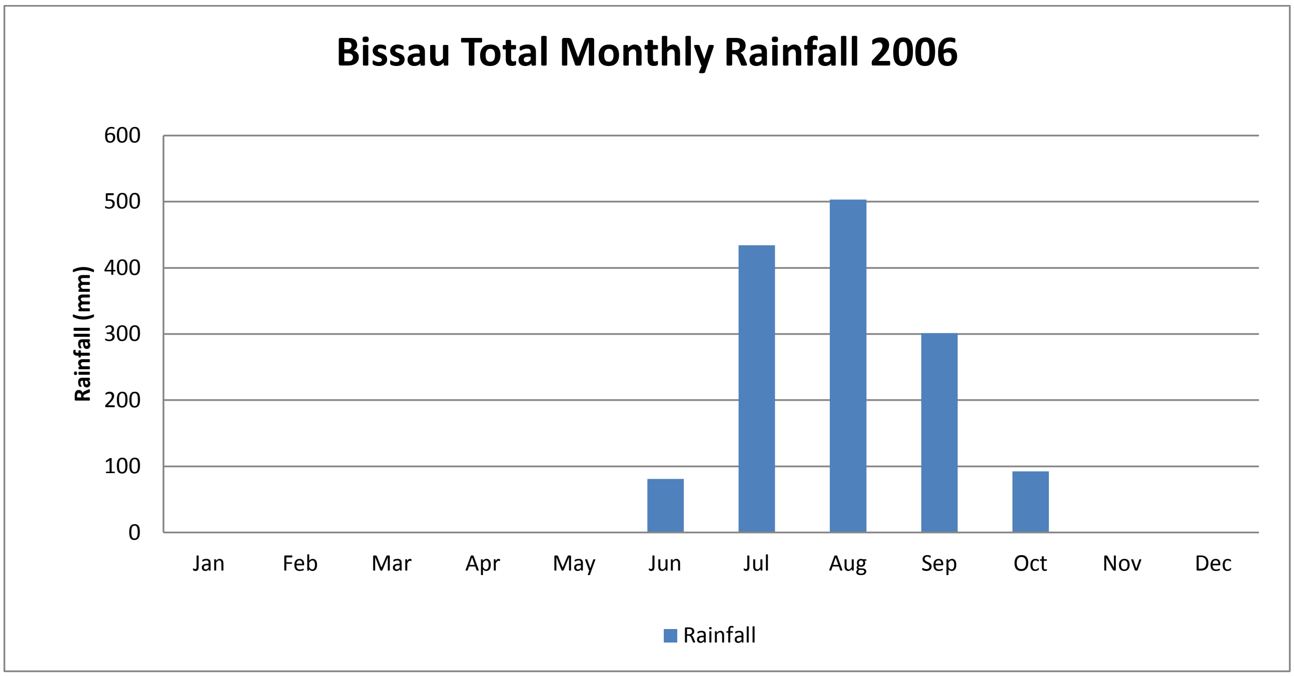
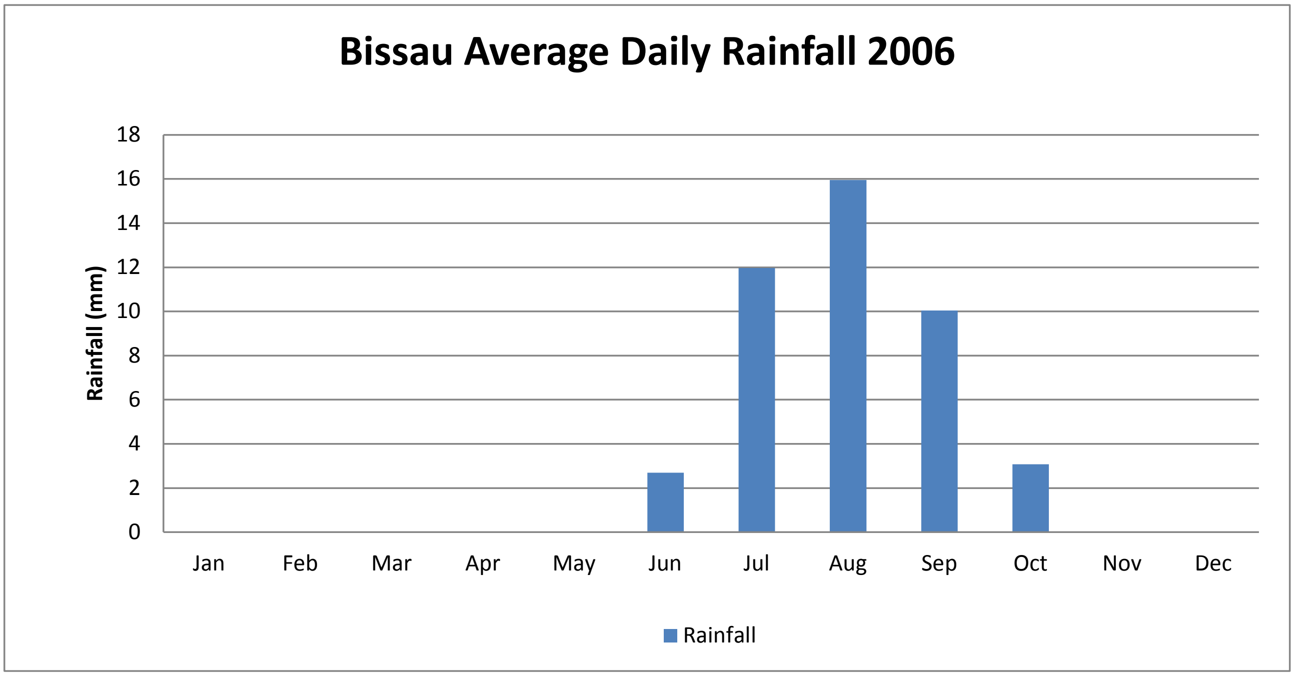


Figure 16-5 Bissau Average Daily Rainfall in 2006



The River Cacheu is the major water feature in the area, along with several tributaries. The river is broad, measuring approximately 500 m wide at the Project area, with typical water elevation around 5 mamsl. The river and its tributaries are tidal, with an approximate 2 m depth range during the tidal cycle.

The most critical design element of the proposed mining plan is water management. All mining areas must be fully dewatered in advance of mining activities. Dewatering of the overburden and phosphate matrix zone must be done approximately six months prior to mining activities to accommodate dry mining of the deposit. Section 18.9.6 details the dewatering assessment.

The River Cacheu must be considered in the surface water management design. The river rise must be controlled to avoid flooding the pits. A flood protection bund must be constructed (in stages) along the south border of the pit and the northern bank of the river. The initial boxcut could serve as a source for the material used in the construction of this bund.

Two ephemeral streams run approximately north-south through the mining areas. The eastern ephemeral stream (Rio de Bunjas) should be avoided if possible to minimize anticipated costs and environmental impact of diversion of this stream. An optimized pit design has been developed to avoid this stream. The western ephemeral stream (Rio de Cavaras Marinhos) will require a diversion plan design later in the mine life.

In addition to advance dewatering, in-pit water management is critical. Mine perimeter ditches and protection bunds with water storage ponds and pumps must be established and rigorously maintained to keep surface water from entering the mining areas. Roads must be well-graded and crowned with a thick layer of pervious crushed rock. In-pit roads and pit floors should be designed to drain to pit sumps located at 300 m intervals; sumps should be equipped with large, well-maintained pumps and float-level controls to operate when needed. Mining will continue at decreased productivities during the wettest two months of the heavy rainy season, and it will remain critical to maintain strict pumping and drainage plans to drain pits and roads as rapidly as possible to maintain equipment trafficability and access to the production faces. Failure to do so will result in operational inefficiencies and delays.

Mobile equipment (excavators, trucks, and auxiliary mobile equipment) will require trafficability at all times throughout the active mine to maintain productive operation of the equipment. Operating benches and running surfaces will be required to withstand the bearing pressure of the equipment. Due to the heavy rainfalls from mid-July to mid-September, Golder has applied de-rating factors to the mining equipment to account for standing down equipment after rain events and lower productivity in the rainy season.

16.3 Mine Design Criteria

Golder has performed an update to the pit slope design geotechnical study completed by Golder Associates UK (GAUK) as detailed in Section 16.5. In the Study, Golder recommends a 20° overall permanent wall angle at an operational factor of safety (FOS) of >1.3. This wall design will be temporarily dug with a 65° bench face (batter) angle, 14.9 m wide berms, and 10 m high benches. The overall permanent wall angle recommendation is based on maintaining a FOS of 1.3 for the overall slope. The bench design allows the bench face to ravel to angles as flat as 25° while maintaining a 6.5 m wide safety bench. Additional information about the geotechnical findings is included in Section 16.5. Golder based this recommendation on a geotechnical analysis of the four main soil units above the FPA matrix zone.

Dewatering pump test data indicates that dry open-pit mining will be feasible. Dry mining the deposit will allow 65° temporary dig face angles. Based on material density and moisture content lab results for the clay and sand horizons, Golder recommends an average overburden swell factor of 27 percent. This swell factor is applied to the ex-pit WD, IOB facilities, and SOS facilities. SOS facilities are areas of overburden storage within the pit footprint but overfilled a maximum of 25 m above original topography. Overburden will be stacked in external WD early in the mine life and backfilled into the mined-out pit when pit advance provides sufficient room for backfilling. External WDs are designed to an overall slope of 1V:4H, and SOS and IOB are designed to an overall slope of 1V:6H. WD will be built in lifts and compacted with a dozer and compactor. Section 16.7.2 details the overburden storage design criteria.

Mining recovery of the phosphate matrix was estimated based on an anticipated 100 mm of mining roof loss and 75 mm of floor dilution gain. An additional geology and mining recovery factor of 95 percent was applied to estimate the tonnage of ROM matrix recovered. The overall mining recovery is dependent upon the matrix thickness. The mining recovery factors reflect the scale of the operation and equipment used to mine the matrix.

The mining method for the Farim Phosphate Deposit will require mine haulage trucks. Excavator/truck mining will require stable haul roads and mine working surfaces for all pit levels and for all material, including the extraction of the FPA matrix. Furthermore, the excavator/truck method will require the construction and maintenance of permanent rock haul roads to the ex-pit WDs, maintenance facility, and ROM stockpile storage area adjacent to the processing plant. The design of these haul roads are covered in Section 16.5.6.

The proximity of the mine site to the Cacheu River will require the construction of a protection bund to prevent in-pit flooding. Sufficient overburden material from pre-stripping operations (Year 0) will be diverted to construct a bund between the mine site and the tidal extents of the river. This bund will be constructed for flood control and will serve as the primary barrier between the river and mining areas. The tidal nature of the river will require the construction of a bund to an elevation of 4 mamsl. The total buffer between the river and the open pit will be 100 m in width to allow construction of the bund to an elevation of 4 mamsl with 1V:3.5H slopes, a crest width of 20 m, a river buffer of 20 m ± 5 m, and a pit side buffer of 50 m to allow for a 33 m wide haul road and a 12 m offset from pit crest to road. There is sufficient buffer on the open pit side of the bund to allow pit haulage access as needed.

Because of the concentrated annual rainfall from July through September, the mine plan limits mining activities at full production to nine months out of the year; the other three months will be mined at reduced productivity. Operations must be vigilant with in-pit dewatering to prevent pit flooding and maintain pit stability.

The remote nature of the Farim operation, with limited power supply, precludes the use of electric mining equipment. All mining equipment selected for the plan is diesel mobile equipment.

Table 16-1 summarizes mine plan parameters and factors.

Table 16-1 Summary Table of Mine Plan parameters

Description	Value
Permanent wall angle	20°
Permanent wall operational FOS	>1.3
Bench Height	10 m
Short-Term Bench Face (Batter) Angle	65°
Short-Term Berm Width	14.9 m
Long-Term Bench Face (Batter) Angle (After Sloughing)	25°
Long-Term Berm Width (After Sloughing)	6.5 m
Overburden angle of repose WD/IOB/SOS	1V:4H/1V:6H/ 1V:6H
Overburden spoil swell factor	27%
Total Moisture (As-Received Basis), Overburden	20%
Overburden Density (As-Received Basis)	2.10 t/m^3
Overburden Density (Dry Basis)	1.68 t/m^3
Total Moisture (As-Received Basis), Matrix	20%
Matrix Density (As-Received Basis)	1.75 t/m^3
Matrix Density (Dry Basis)	1.40 t/m^3
Minimum mineable matrix thickness	1 m
Mining roof loss	100 mm
Mining floor dilution	75 mm
Geology and mining recovery factor	95%
Buffer between pit and river	100 m
Full production mining months per year	9 months
Reduced production mining months per year	3 months
Mine dewatering possible	Yes
Material to support truck traffic	Yes
Spoil Stackability	Yes

16.4 Geological Block Model

The two-dimensional (2D) geological model created in Maptek®'s Vulcan™ software, as detailed in Section 14, was used for the mine design and LOM production plan. Data extracted from the 2D geological block model included the Project area topographic surface from LiDAR survey data, block centroid easting and northing coordinates, overburden thickness, matrix thickness and assayed quality data. Assayed qualities for the matrix include P_2O_5 grade and the contaminants Al_2O_3, CaO, Fe_2O_3 and SiO_2. The geological model data were constructed on a 25 m by 25 m grid. Triangulation surfaces for the FPA roof and floor were also provided. The 2D geological model data was imported into Ventyx®'s Minescape™ software (version 5.8) to construct a three-dimensional (3D) block model for optimization purposes and to develop geological surfaces of overburden and matrix to aid in mine planning work. All geological model data imported into Minescape were checked to ensure original data were honoured and that the conversion of the 2D block model to a 3D model was successful.

After reviewing the model import, ROM mining surfaces were created to account for an anticipated 100 mm roof mining loss and 75 mm floor dilution gain where the FPA seam was greater than the minimum mineable thickness of 1 m. These anticipated dilution and mining loss factors are based on extracting the matrix with small backhoes. Additionally, ROM quality surfaces were developed to account for the mining losses and dilution gains. Given the lack of dilution sampling, dilution material was assumed to have 0% P_2O_5 concentration and identical contaminant concentrations as the FPA matrix directly above it. Like the FPA, dilution was also assumed to have a density of 1.4 t/m^3 (dry basis). An example of the effects of mining losses and dilution gains on ROM (recovered) P_2O_5 grades on matrix intervals of various thicknesses is provided in Table 16-2.

Table 16-2 ROM Recovery Factors at Various Matrix Thicknesses

FPA Thickness (m)	Roof Loss (m)	Floor Dilution (m)	Mining Loss	Recovered Thickness (m)	Recovery	In Situ %P_2O_5 (Dry Basis)	ROM %P_2O_5 (Dry Basis)	P_2O_5 Recovery
0.500	0.100	0.075	5%	0.475	90%	30	25.3	76%
1.000	0.100	0.075	5%	0.975	93%	30	27.7	86%
1.500	0.100	0.075	5%	1.475	93%	30	28.5	89%
2.000	0.100	0.075	5%	1.975	94%	30	28.9	90%
2.500	0.100	0.075	5%	2.475	94%	30	29.1	91%
3.000	0.100	0.075	5%	2.975	94%	30	29.2	92%
3.500	0.100	0.075	5%	3.475	94%	30	29.4	92%
4.000	0.100	0.075	5%	3.975	94%	30	29.4	93%
4.500	0.100	0.075	5%	4.475	94%	30	29.5	93%
5.000	0.100	0.075	5%	4.975	95%	30	29.5	93%
5.500	0.100	0.075	5%	5.475	95%	30	29.6	93%
6.000	0.100	0.075	5%	5.975	95%	30	29.6	93%

Notes:
[1] In situ P_2O_5 grade for demonstration purposes only

As demonstrated in Table 16-2 above, mining losses and dilution gains result in a loss of P_2O_5 quality from in situ to ROM; given a constant roof loss and dilution gain, the overall loss of P_2O_5 is dependent on seam thickness.

An example of the overall effects of mining losses, dilution gains and beneficiation on an interval of matrix is shown in Table 16-3.

Table 16-3 Effects of Mining Methodology and Beneficiation on FPA Matrix Recoveries and Grades

Parameter	Value
FPA Thickness (m)	3.500
Roof Loss (mm)	100
Floor Dilution (mm)	75
In Situ %P_2O_5 (Dry Basis)	30.0
In Situ %Fe_2O_3 (Dry Basis)	5.4
In Situ %Al_2O_3 (Dry Basis)	2.5
In Situ %SiO_2 (Dry Basis)	10.9
In Situ %CaO (Dry Basis)	40.3
Mining Loss	5%
ROM (Recovered) Thickness (m)	3.301
ROM %P_2O_5 (Dry Basis)	29.4
ROM %Fe_2O_3 (Dry Basis)	5.4
ROM %Al_2O_3 (Dry Basis)	2.5
ROM %SiO_2 (Dry Basis)	10.9
ROM %CaO (Dry Basis)	40.3
Processing Plant Mass Recovery (%)	75.5%
Product %P_2O_5 (Dry Basis) [1]	34.0%

The expected mass recovery of 75.5% and product grade of 34% P_2O_5 have been confirmed in pilot plant test work. Further details on metallurgical test work are provided in Chapter 13.

After developing the ROM surfaces, a 3D block model with blocks measuring 25 m by 25 m by 1 m in the X, Y, and Z, respectively, was created from the grid-based Minescape model. Using the same limits as the original 2D Vulcan block model, approximately 4.6 million blocks were created. The relevant geological and quality assay data for each block were populated using Minescape's resource estimation functions; matrix tonnages were calculated based on a constant density of 1.4 t/m^3 (dry basis) per the resource estimation methodology. The 3D block model was thoroughly checked against both the original 2D block model and Minescape reserves to ensure that original data were honoured and that no volumes, tonnages, or assay data had changed. After review, the Minescape block model was compiled into a format that Vulcan software could read for optimization purposes.

16.5 Geotechnical Parameters

Golder completed a review of the ground characterization data and slope design recommendations that were developed by Golder Associates Ltd. in 2012. The purpose of the review was to provide an update of the geotechnical recommendations based on any new information or revisions and re-evaluate the available information to assess if there are opportunities to further optimize the open pit slope designs.

Golder's review for this Feasibility Study is provided in the Farim Phosphate Geotechnical Assessment Technical Memorandum (Golder, 2015). The final pit limits accommodate a 1.75 Mtpa mine plan with a 25 year mine life. There is no new geotechnical data or information that would affect the open pit design. Therefore, this latest update is based solely on a review of the data contained in the 2012 Ground Characterization – Factual Report (Golder Associates UK, 2012).

16.5.1 Open Pit Ground Investigation

The geotechnical investigation of the Open Pit Area (OPA) was completed between October 17, 2011, and March 1, 2012. The previous geotechnical characterization activities completed in the OPA that are relevant to the open pit design include 22 geotechnical rotary core holes logs with standard penetration tests (SPT), typically at 3 m intervals.

Additional activities included a geophysical program including:

- Seismic Refraction Profiling;
- Multichannel Analysis of Surface Waves (MASW);
- Electrical Resistivity Tomography; and
- Electrical conductivity.

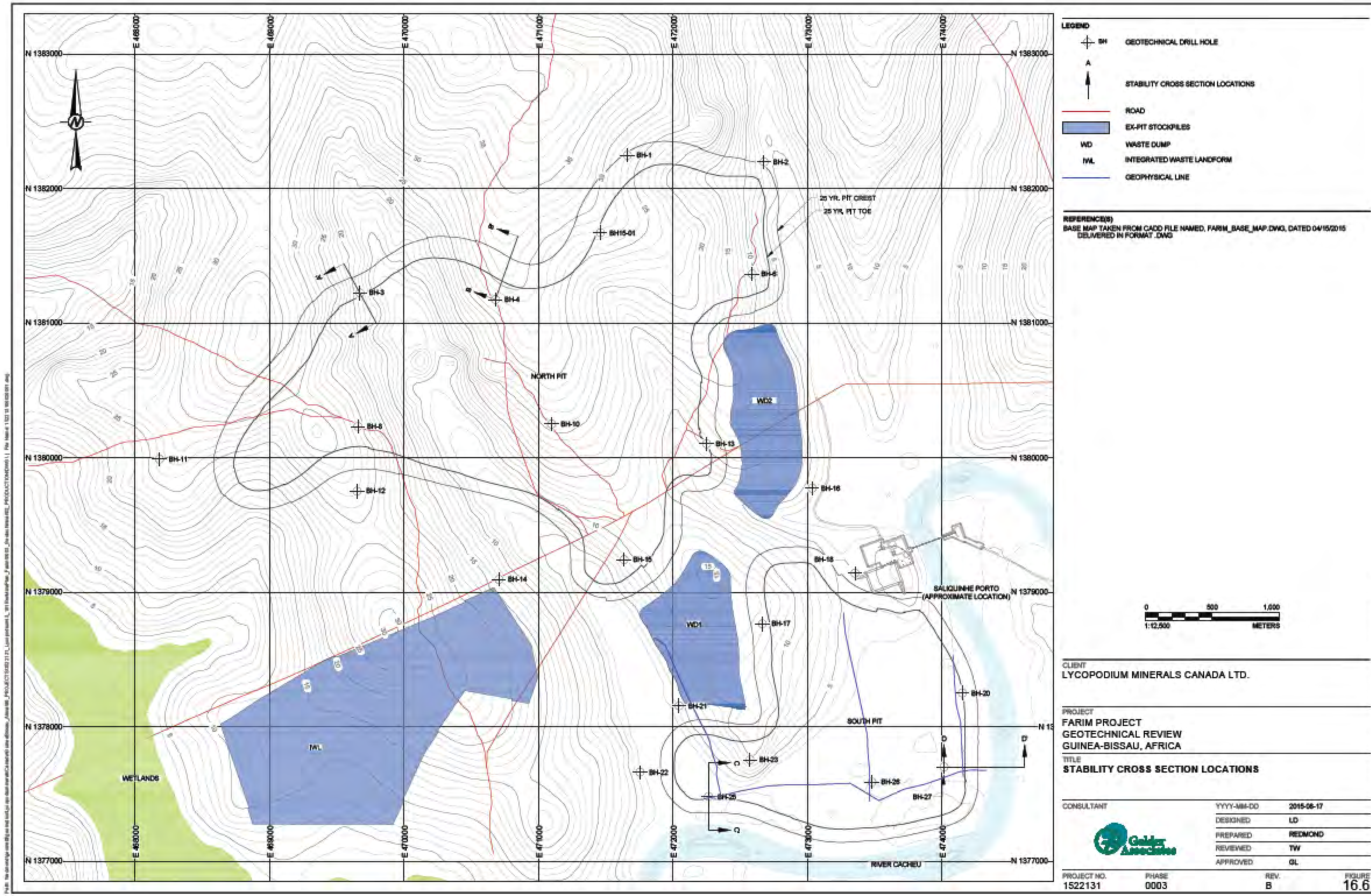
Laboratory testing included:

- Sieve analyses;
- Atterberg limits tests;
- Moisture content and density tests;
- Oedometer (Consolidation) tests;
- Undrained, unconsolidated (UU) triaxial tests;
- Consolidated undrained (CU) triaxial tests with pore pressure measurements;
- Direct shear tests;
- Moisture-Density Relation Tests; and

- California Bearing Ratio (CBR) tests.

The geotechnical drillholes, shown on Figure 16-6, were generally sited near the final 1.75 Mtpa open pit limits. The spacing between holes was 500 m to 1500 m. The borehole and the geophysical survey lines (light blue) are shown on Figure 16-6 in relation to the planned open pit.

Figure 16-6 OPA Boreholes, Geophysical Lines and Stability Cross Section Locations



16.5.2 Open Pit Ground Summary

Golder (2012) defined four principal stratigraphic units in the overburden on the basis of the field logs, field tests, and laboratory testing. The units vary in thickness, are not laterally extensive, and may occur in repeated sequences. The overburden is interpreted to have been deposited in a deltaic environment. The geotechnical units defined in the 2012 report are the following:

- OPA-1 – Firm Clay Unit. Red brown, orange, mottled CLAY. Some laminations of fine sand layers were observed. Occasional small content of gravel. Maximum encountered thickness was 23 m.

- OPA-2 – Clayey Sand Unit. Yellow brown, orange mottled fine to medium clayey SAND. Some laminations were observed with red and dark sand. Occasional small content of fine gravel. Lenses of pale brown clay are present in this unit. Maximum encountered thickness was 24 m.

- OPA-3 – Hard Clay Unit. Light gray very stiff to hard CLAY. Laminations with occasional gray sand. Medium gravel content, fine to medium in size. Occasional cobbles. Plasticity medium to high. Maximum encountered thickness was 25 m.

- OPA-4 – Fine Sand Unit. Light gray, mottled with light brown, very dense fine SAND. Laminations of black clay are encountered in this unit. Some of the clay presents forms lenses. Typically present above the Phosphate matrix bed. Maximum encountered thickness was 12 m.

- OPA-5 – Bedrock. Very weak to medium strong fractured white, light gray or pale brown dolomite or limestone. The material appears to have been weathered to a sandy or clayey matrix containing gravel or cobbles or intact rock.

These main geologic units are further grouped into predominantly sand units (OPA-2 and OPA-4) and predominantly clay units (OPA-1 and OPA-3). OPA-4 is distinguished from OPA-2 by higher density and typical occurrence deeper in the drillhole. Clay unit OPA-3 is distinguished from OPA-1 by higher stiffness.

16.5.3 Strength Parameters

Shear strength parameters were developed for the stratigraphic units described above based on in-situ standard penetration testing (SPT), and laboratory testing data from the 2012 Ground Characterization-Factual Report (Golder, 2012). The details of the selection of the strength parameters are provided in Golder, 2015). Table 16-4 summarizes the strength parameters applied to the stratigraphic units.

Table 16-4 Summary of Recommended Strength Values

Unit	Unit Weight (kN/m^3)	Total Strength Parameters		Effective Strength Parameters	
		Cohesion, c (kPa)	Phi (Degrees)	Cohesion, c' (kPa)	Phi (Degrees)
OPA-1	19	80	0	10	20
OPA-2	18	10	30	10	30
OPA-3	18	80	0	40	20
OPA-4	18	5	35	5	35
Bedrock	18	70	25	70	25

16.5.3.1 Open Pit Area Geophysics

Apex Geosciences performed a geophysical investigation at the Farim project including seismic refraction, MASW, and electrical resistivity. The results of this program were included in a report dated April 17, 2012, Appendix D, of the 2012 Ground Characterization Report (Golder, 2012). Three north-south and one east-west geophysical survey lines were completed in the south pit area totaling approximately 5,500 m. The locations of the survey lines are shown on Figure 16-6.

The seismic velocity information derived from the seismic refraction surveys can be used to obtain approximate engineering characteristics of soil units including soil stiffness. It can also be used to estimate engineering characteristics of the rock below the overburden in the OPA. The geophysical interpretation by Apex outlined three overburden units in the south pit area based on seismic velocities summarized below in Table 16-5.

Table 16-5 Seismic Refraction Interpretation by Apex

Layer	Seismic Velocity (m/s)	Average Seismic Velocity (m/s)	Thickness (m)	Interpretation	Stiffness/Rock Quality
1	421 - 1,119	776	0.8 - 6.0	Overburden	Soft-Firm / Loose-Medium Dense
2	1,176 - 1,674	1,453	0.1 - 7.1	Overburden	Stiff / Dense
3	1,444 - 1,995	819	23.6 - 4.7	Overburden	Stiff-Very Stiff / Dense-Very Dense
4	2,011 - 2,420	2,163	-	Slightly Weathered-Fresh Bedrock	Good

Golder used the seismic refraction information to evaluate whether the overburden units are laterally continuous. Golder did not identify continuous zones with similar seismic velocities that clearly correspond to the overburden units encountered in drillholes that could be traced over significant distances.

Inspection of the seismic profiles show indications of lower velocity soils in close proximity to the River Cacheu. Inspections of the geotechnical logs for drillholes BH-20 and BH-27, which are sited at the southeast crest of the south pit approximately 200 m to 350 m from the river in the plane of Profile R6, indicate the presence of soft clay in the upper 7 m of both of these holes. Drillhole BH-20 has 8 m of loose sand below the soft clay layer then the overburden becomes a very stiff clay below 15 m. Drillhole BH-27 has medium stiff clay below the soft clay, becoming stiff to very stiff below 18 m. Drillhole BH-25 is also located close to the planned southwest pit crest approximately 200 m from the river but does not have the soft clay present at the near surface; the overburden consists of stiff clay to a depth of 11 m becoming a compact sand to the base of the overburden.

The seismic refraction data and comparison of that data to drill log information suggests the possible presence of a soft clay and loose sand in the upper 15 m to 18 m within several hundred metres of the river. While it is typical that the upper several metres of the drillholes have loose sands or soft clays, drillholes BH-20 and BH-27 have unusually thick zones of soils with low SPT blow counts. Stability analyses have been carried out to evaluate the possible effect of this weak layer on the stability of the south wall of the South Pit; this analysis is discussed in Section 16.5.4.2. However, if the character of the soils encountered in the upper 15 m to 18 m in drillholes BH-20 and BH-27 is laterally continuous, these soils may represent a separate soil unit with strength parameters distinct from the soil units that have been defined to date. The presence of weak clay soils in the zone between the river and the open pit will also be a consideration on the stability of the retaining berm and the rate that the berm should be constructed.

16.5.4 Stability Analyses

Two-dimensional, limit equilibrium slope stability models were developed using the software program SLIDE™. The bench designs use 10-m high benches with design bench face angles of 25 degrees and 6.5-m wide benches resulting in overall slope angles of 20 degrees between haul ramps. Stability models were set up to evaluate bench scale and overall slope stability to verify the factor of safety remain above the design criteria value of 1.3. The seismicity of the region is low so Golder has not completed seismic analyses of slope stability.

Undrained shear strength parameters have been applied to evaluate the stability of temporary bench faces and the impact of equipment loading on the bench stability. Effective stress strength parameters are applied to evaluate the stability for long term conditions such as at the phased and final pit walls assuming the pit slopes are effectively depressurized in advance of mining and pore pressures in the clay units will come to equilibrium.

16.5.5 Bench Scale

A bench scale model was prepared to evaluate the stability of a temporary dig face angle for the design strength parameters developed for each of the overburden units (see Figure 16-7 on the following page). The actual performance of the dig face will be strongly influenced by the local conditions and degree of depressurization and time required for excess pore pressures to dissipate in the clays. Hydraulically isolated sand lenses are anticipated to retain water pressures. Dewatering of these localized zones through horizontal drive point drains or other means may be necessary to safely mine the bench face. A maximum dig face angle of 65° is assumed based on the planned use of large excavators or loaders to mine the benches. Benches 10 m high in the clay soil units are predicted to remain stable when excavated at 65° face angles for short term periods (Figure 16-7). Based on the design parameters for sand units OPA-2 and OPA-4 and assuming depressurized conditions behind the slope face, the excavated dig face will be below a FOS of 1.0 (0.7 and 0.8 for OPA2 and OPA4, respectively) during short term conditions and sloughing of the excavation slope face soon after it is excavated should be anticipated. If isolated areas with pore pressures are encountered they will need to be allowed to drain or measures taken to depressurize the slope.

The slope stability analyses for bench scale were also used to evaluate the impact of equipment loading on the excavated bench face and to define a minimum setback from the bench crest to the haul roads. Figure 16-8 in the coming pages shows the components of the bench design at the phase final wall. As a bench is being actively mined, the assumed 65° dig face will be excavated and a set back from crest of the dig face will be maintained so that equipment loads do not impact stability of the dig face. We have evaluated the equipment loading for a Loaded CAT 777 haul truck set back a distance of 12.0 m from the cut bench crest (based on 8.4 m to the design bench crest plus an additional 3.6 m to accommodate a 1.3 m high safety berm). Equipment loading this distance behind the dig face crest had no effect on the FOS in the sand units, and FOS in the clay units remained above 1.6. Haul roads should not be aligned closer than 12 m from a 10 m high cut bench crest.

Figure 16-7 Bench Scale Stability

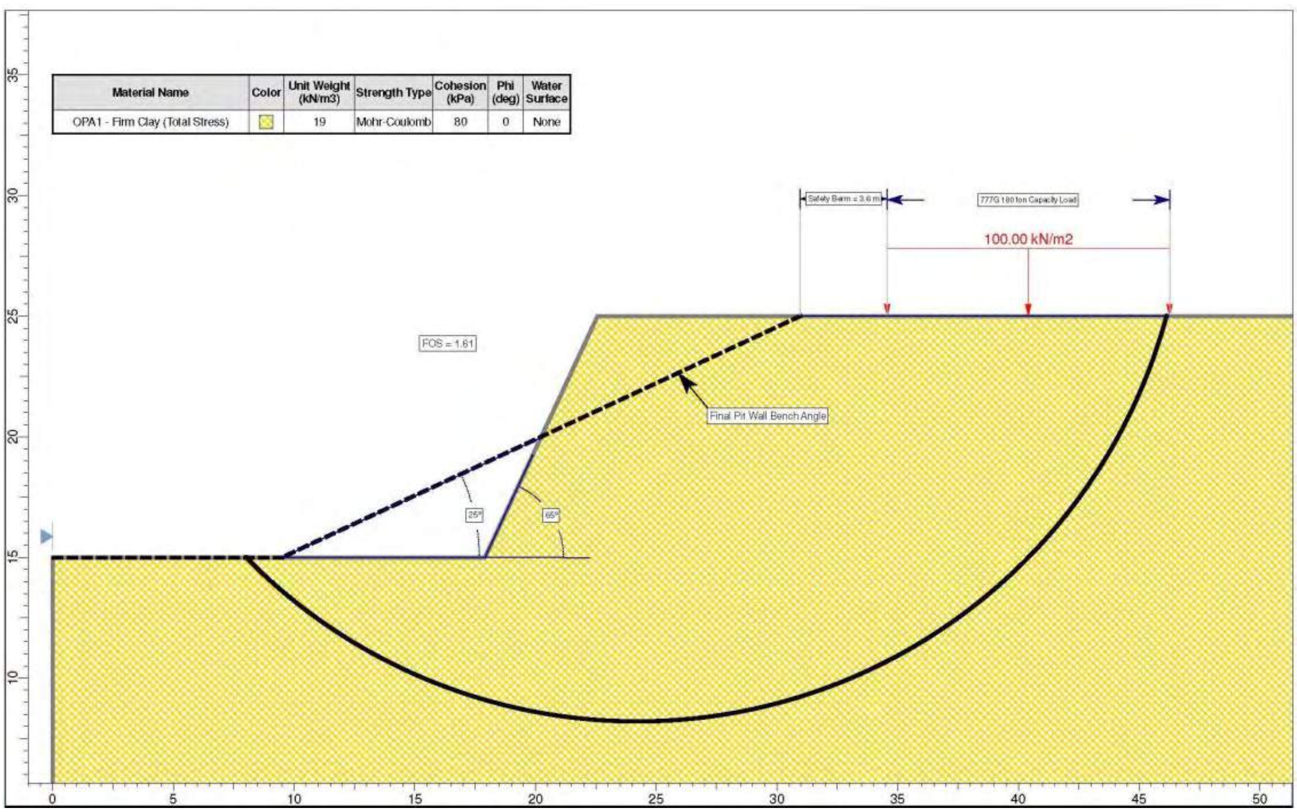
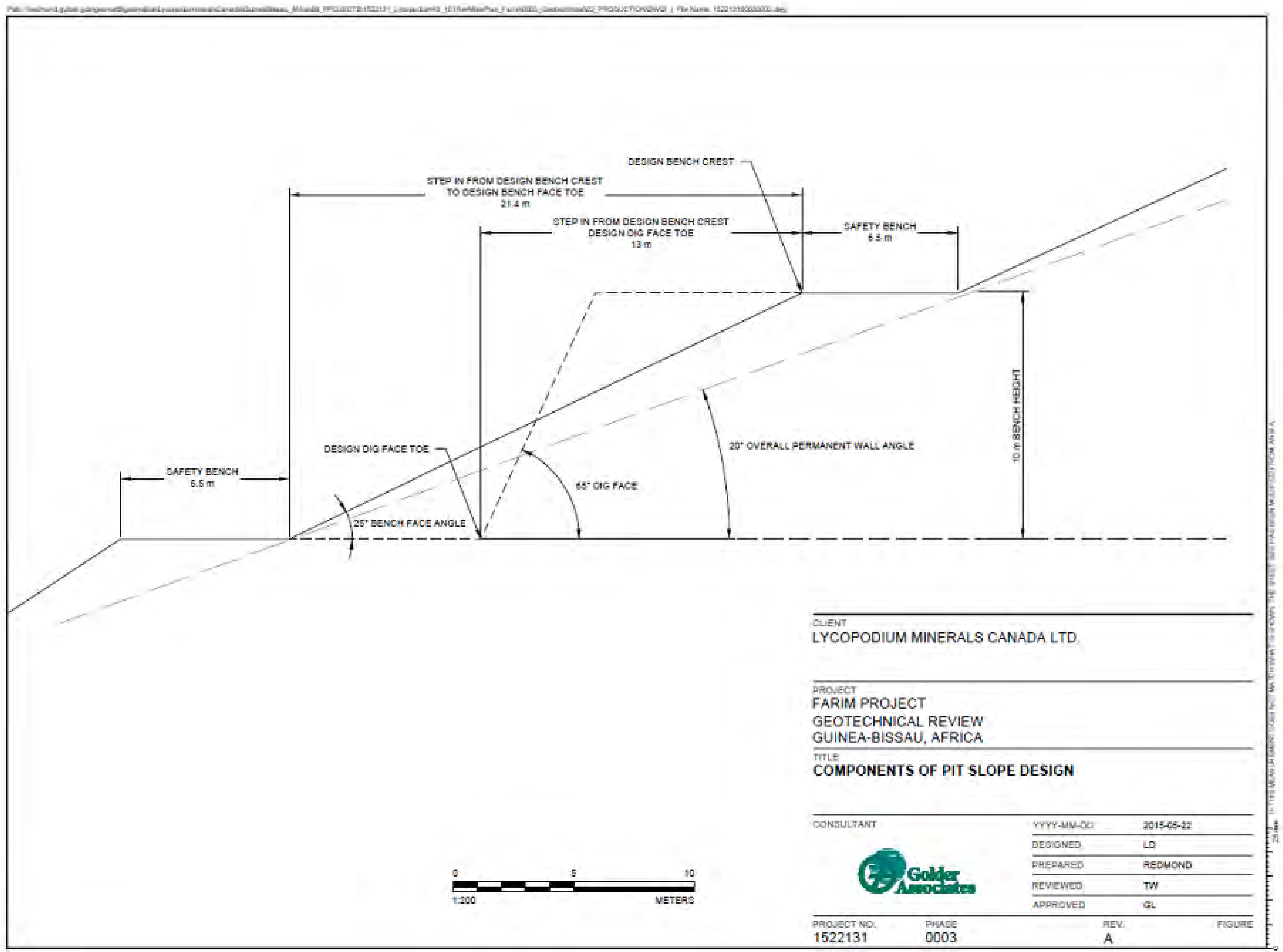


Figure 16-8 Components of Pit Slope Design



16.5.6 Overall Slope

Four overall slope stability cross-section models were set up through the planned final pit walls with the overburden units defined based on drillhole log information in the plane of the cross-sections. Each cross section model is based on information from one borehole, so Golder has assumed that the units are horizontal and laterally continuous in the preparation of these models. The stability models were run using a water table that is drawn down to the final pit floor below the crest of the pit sloping up to the regional water table at an assumed slope. This dewatered condition is an assumed condition and is not based on any hydrogeologic analyses or model. The locations of the overall slope stability cross section models are shown in Figure 16-6.

Stability analyses evaluated the recommended a slope design consisting of 10 m high benches with 25 degree bench face angles and 6.5 m safety benches to result in an overall 20° overall slope. Location of haul ramps will vary over time and the typical slope profile will not have a haul ramp. Therefore, Golder has not included haul ramps in the cross section models. Inclusion of haul roads would improve overall slope stability if they are placed such as to make the overall slope flatter. The stability cross-sections and minimum computed factors of safety for the critical shear surface (shear surface with the lowest FOS based on a search over the full height of the slope) are shown in Figure 16-9 through Figure 16-12. The stability cross-section locations were selected to evaluate maximum slope heights and maximum encountered thicknesses of soil units in the bore holes. Table 16-6 summarizes the results of the stability analyses.

Table 16-6 Critical Runs for Overall Slope Stability

Section	Associated Drill Hole	Minimum Factor of Safety	Reason for Selection as Critical Section
A	BH-3	1.39	High Slope height. Maximum thickness of OPA-1
B	BH-4	1.59	Maximum Slope height. To compare to stability section completed in 2012 study
C	BH-25	1.45	Maximum thickness of OPA-2
D	BH-27	1.39	Maximum thickness of OPA-4. Located in South Wall of South Pit. Soft clay present in the upper 7m.

The results of the stability analyses for overall slope stability for long term conditions indicate that appropriate factors of safety (FOS > 1.3) are maintained using the slope design geometry described above, applying the material properties developed for the defined stratigraphic units.

Based on the information currently available and the interbedded and laterally discontinuous soil conditions, it is not possible to predict the overburden units that will form the pit walls in different areas of the pit. The pit slope design recommendations apply to the most critical anticipated slopes. Overburden unit OPA-1 is the weakest unit and high slopes composed of OPA-1 are the controlling factor on the overall slope height. Section A (see Figure 16-9 on the following page) has a 23 m thickness of unit OPA-1 and is the thickest interval of OPA-1 encountered in the geotechnical bore holes that have been completed. Slopes higher than 35 m potentially composed entirely of unit OPA-1 would have a FOS of slightly below 1.3 but remain greater than 1.2 for slopes of 60 m, the maximum overall slope height. Based on the relatively conservative strength estimates applied and the expectation that slopes greater than 35 m high would be very rare, Golder recommends the 20° overall slope angle.

A multi bench total stress analysis using short term strengths for the clayey soils based on the slope configuration in Figure 16-13 in the coming pages was completed to evaluate the stability of the final pit wall for the condition when the clayey soils have not had sufficient time to become sufficiently drained and undrained conditions exist. Results indicate that final pit walls composed entirely of clay units OPA-1 or OPA-2 (applying total stress strength parameters of 80 kPa) that are greater than 30 m high will not meet the FOS criteria of 1.30. At final slope heights of approximately 40 m, it is expected that the FOS will be less than 1.0. Therefore, an important component of the pit slope development will be the verification that sufficient drainage has occurred through installation of piezometers particularly where clay units comprise a continuous slope greater than three bench heights. If the pore pressures in these clay slopes do not have sufficient time to come to equilibrium or do not decrease due to the effects of unloading, then it would be necessary to construct flatter slopes in some areas.

Figure 16-9 Stability Section A based on Borehole HC-3 Long Term Stability Analysis

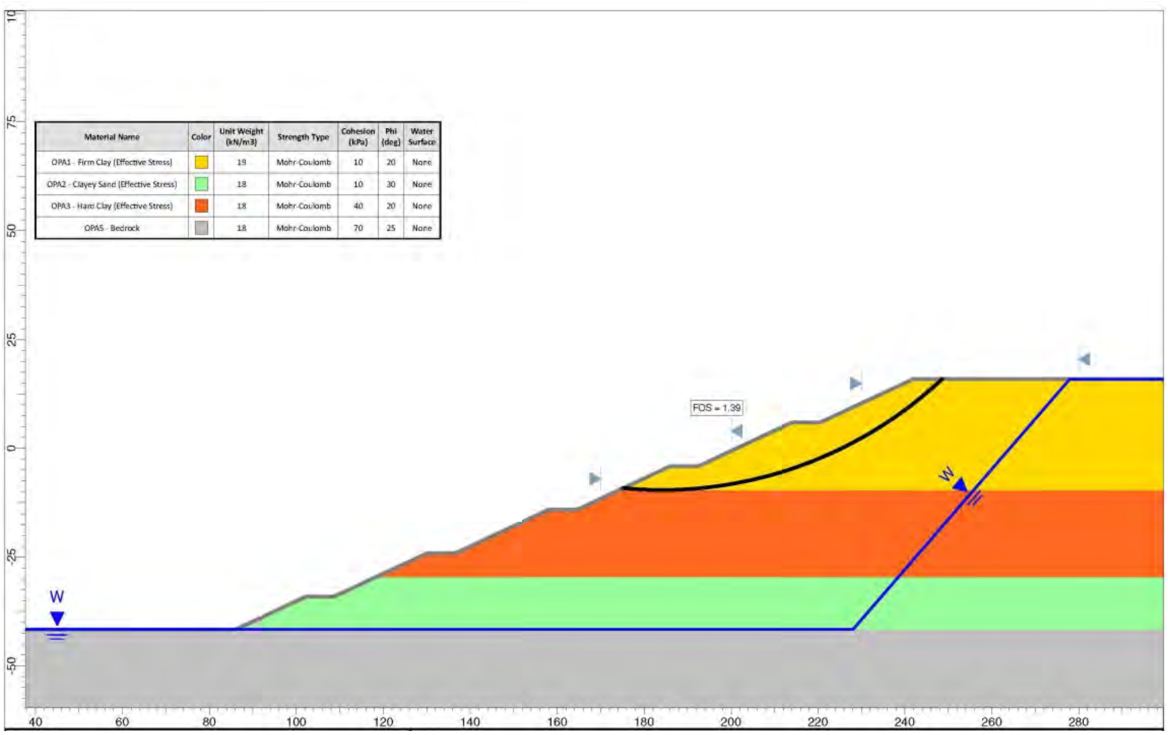


Figure 16-10 Stability Section B based on Borehold BH-4 Long Term Stability Analysis

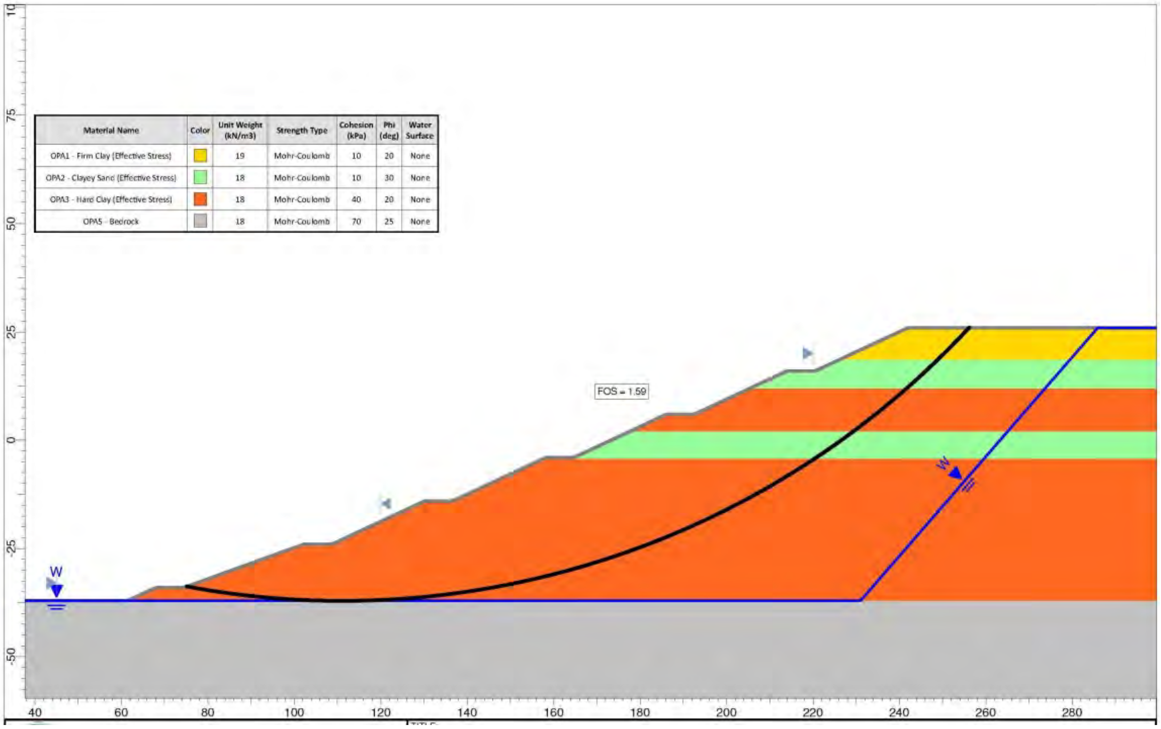


Figure 16-11 Stability Section C based on Borehole BH-25. Long Term Stability Analysis

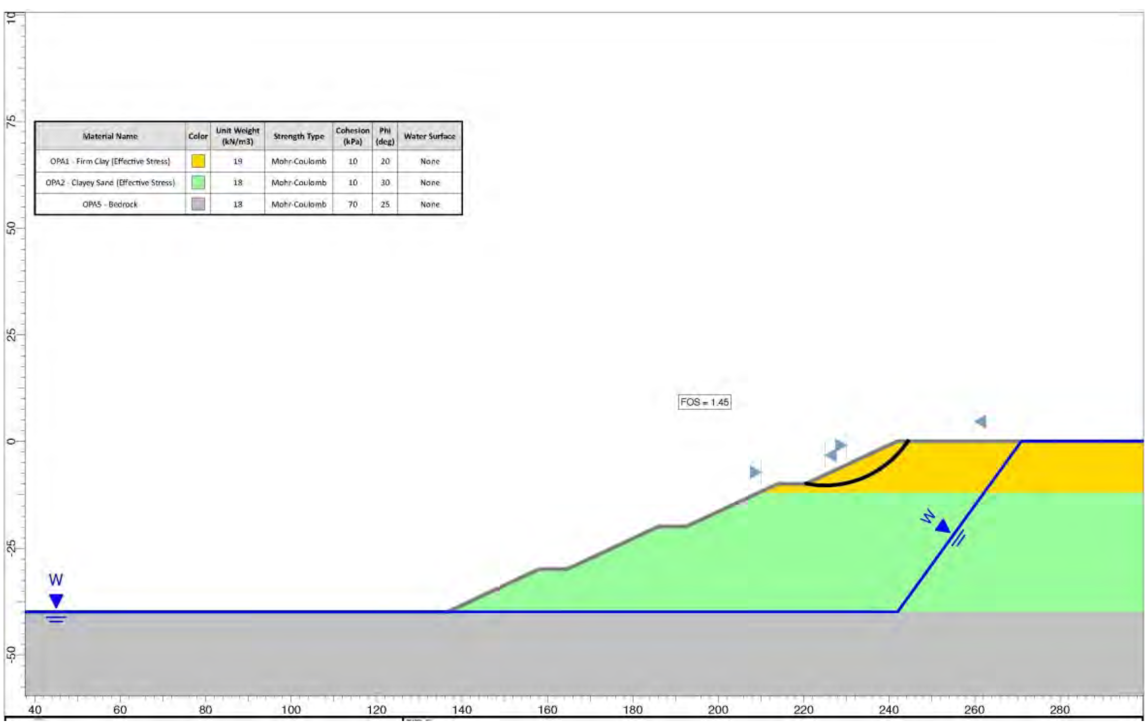


Figure 16-12 Stability Section D based on Borehole BH-27. Long Term Stability Analysis

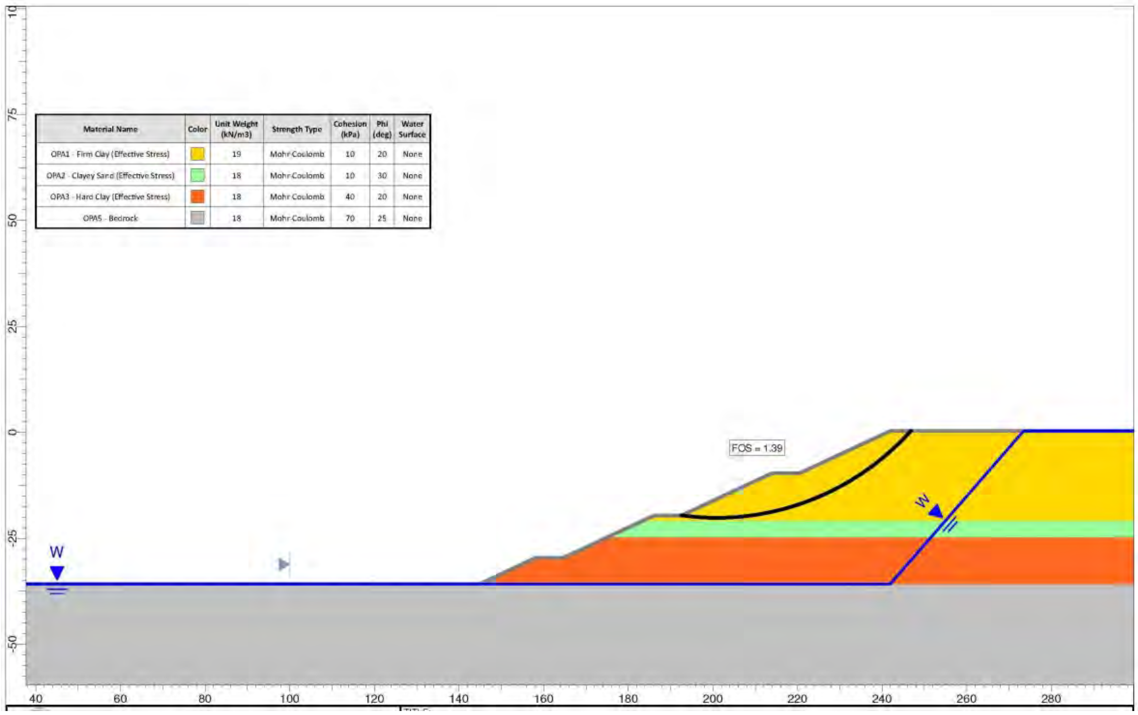
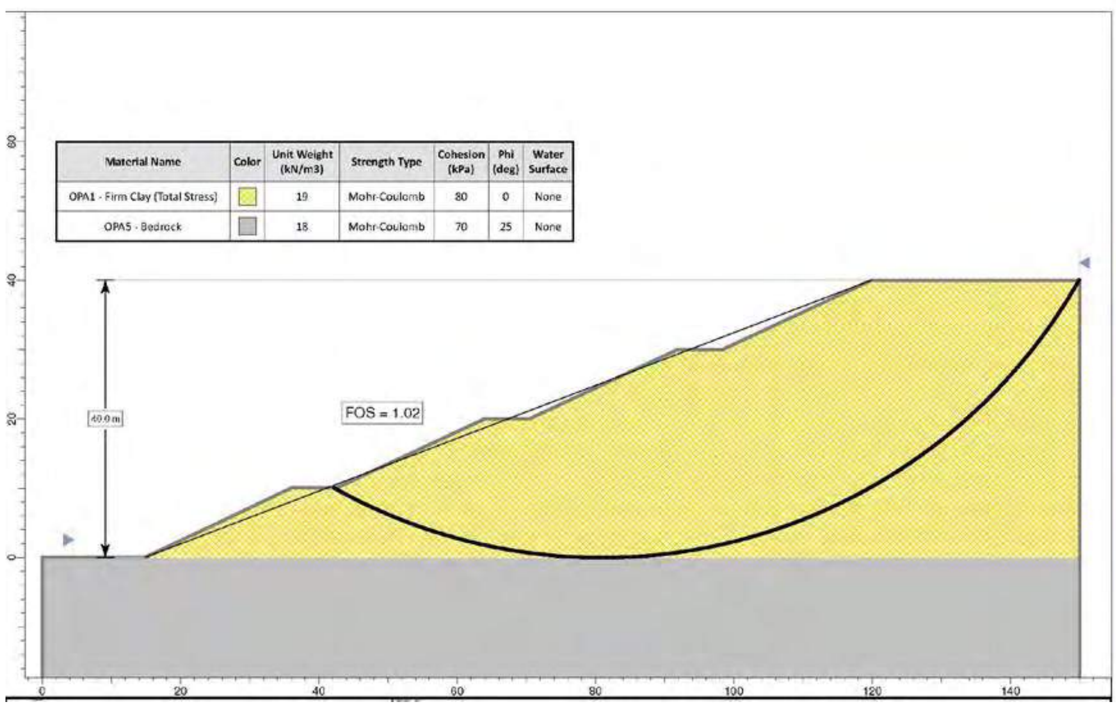


Figure 16-13 Multi-bench Analysis for Undrained Conditions, 40 m Slope Height in Clay Soils



16.5.7 Slope Design Recommendations

Based on the review of the previous geotechnical characterization data and the results of slope stability modeling, Golder recommends applying the slope design illustrated on Figure 16-8 and summarized in Table 16-7. The overall 20° permanent slope angle is the controlling factor for the slope recommendations. The temporary dig face angle of 65° is an assumed typical temporary slope angle cut by an excavator or loader that, over time, will slough and erode to a flatter slope angle. The benches in the higher cohesion clay soils will maintain steeper bench faces over the lifetime of the pit wall. Near surface soils may be expected to have additional cohesion from laterite formation and cementation by iron oxides. Cohesionless sand will reach flatter bench face angles over time. The intent of the slope design is to maintain an effective safety bench through the duration of the phased final pit walls. The 25° permanent bench face angle represents the minimum expected long term bench face angle and provides a 6.5 m wide safety bench.

Table 16-7 Open Pit Slope Design Parameters

Description	Value
Bench Height	10 m
Temporary Dig Face Angle	65°
Permanent Bench Face (Batter) Angle	25°
Permanent Inter-Ramp Wall Angle	20°
Step-in From Design Bench Crest to Design Bench Face Toe	21.4 m
Step-in From Design Bench Crest to Temporary Design Dig Face Toe	13 m
Safety Bench Width	6.5 m

16.5.8 Ground Water Management

The open pit design is based on the implementation of groundwater and surface water control measures detailed in Sections 18.19.5, 18.19.7 and 18.19.8.

16.5.9 Dewatering Induced Settlement

Dewatering of the OPA will lead to a lowering of groundwater within the overburden soils. This will lead to an increase in effective stress conditions within the dewatered soils and will induce settlement that will lower ground surface levels. The lowered ground levels could affect surface water management and the design levels of flood protection or earthwork containment structures.

Two conditions were previously assessed relating to dewatering on the southern and northern perimeters. The soil profile used for the slope stability analyses was used together with appropriate geotechnical parameters based on SPT values for granular soils and oedometer results for cohesive soils.

It was estimated that settlement up to 500 mm could occur due to the dewatering of the mine site. Sand units are likely to settle quickly where they are not confined by clay layers. The majority of the settlement will be due to the longer term consolidation of the clay layers within the ground profile and the total settlement that will be realized will be controlled by the duration of dewatering.

16.5.10 Pore Water Pressures

A critical assumption in the stability analyses is that the slopes are depressurized in advance of mining. Section 18.19.5 describes the planned dewatering which will include the following:

- A system of pumped wells around the perimeter of the OPAs;
- A grid of “sacrificial” internal pumped wells to form dewatered “cells” approximately aligned with the edges of the proposed mining cells, to dewater ahead of the working strip;
- Additional shallow perimeter wells in the overburden in areas where significant sand layers are identified and where pumping from the deep aquifer does not provide sufficient drawdown in the overburden;
- Drainage ditches or trenches in pit slopes to collect residual seepage water where it emerges from isolated sand zones exposed within the slopes;
- An in-pit pumping system to manage residual seepage water and surface water from precipitation; and
- Monitoring boreholes to allow external groundwater levels to be observed.

The pore pressure conditions within a bench planned for mining should be investigated through installation of piezometers or push-in probes before mining into a new area. The water table condition at the periphery of the pit limits should be monitored to verify the dewatering expectations are being met. It is assumed that dewatering will take place ahead of the mine excavations to allow sufficient depressurization and dissipation of pore water pressures within the clay layers. Particular attention is required in the locations of proposed haul roads where cyclical loading are anticipated and where confined sand lenses that may retain groundwater are exposed.

16.5.11 Open Pit Trafficability

Haul roads will be needed to transport overburden from the open pit face to the in-pit overburden backfill, or ex-pit stockpiles as well as for transporting phosphate to the beneficiating plant. A stable road base will be important for safety and truck efficiencies. Procedures that may be needed to provide a stable road base such as compaction and import of base course and surface course materials are important cost factors.

In addition to road base design based on the California Bearing Ratio (CBR) test data and rating curves, the current study has compiled and reviewed the laboratory CBR test data and reviewed the sub-base thickness recommendations. This review is summarized below.

Based on the moisture contents (average 25%) determined from undisturbed samples, the plastic indices and CBR testing undertaken on remolded samples (Golder, 2012), a CBR of 4 to 5 was adopted for clay sub-grade for pavement design of the haul roads. Where granular sub-grades are encountered the CBR value is expected to be higher with reduced moisture contents following dewatering and a CBR value of 10 to 15 was recommended.

Based on the 2015 review of the 2012 testing data, the use of CBR values of 5 for clay subgrade and 10 to 15 for sand are considered appropriate and consistent with our understanding of the soil conditions. The subgrade conditions in the open pit are expected to be highly variable with lenses of sand occurring within clay units and variably clayey sands in the sand units where the natural stratigraphy is intact. Varying the haul road subgrade design for the subgrade condition may not be feasible and a clay subgrade condition should be assumed for all haul roads in the open pit. The in-pit and ex-pit dumps can be expected to be composed of a mixture of the overburden soils and an intermediate CBR value of 10 is recommended assuming the mixing is sufficient to prevent extensive zones of just one type of soil in any particular location and the subgrade is compacted to 95% of the maximum standard Proctor density.

Drawings showing the typical design of overburden and matrix haul roads have been provided as Figure 16-14 and Figure 16-15, respectively.

Figure 16-14 Typical Overburden Haul Road Design

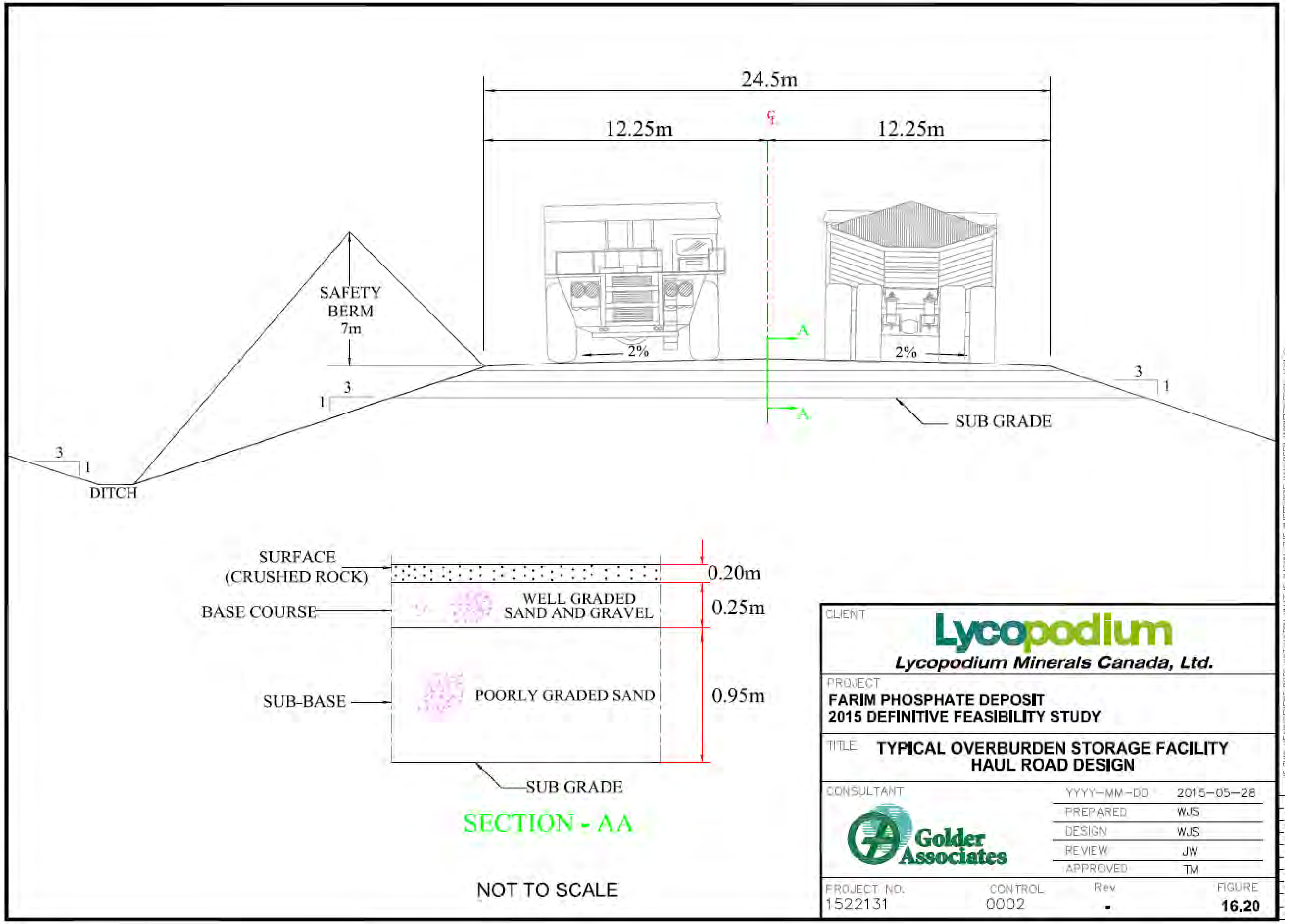
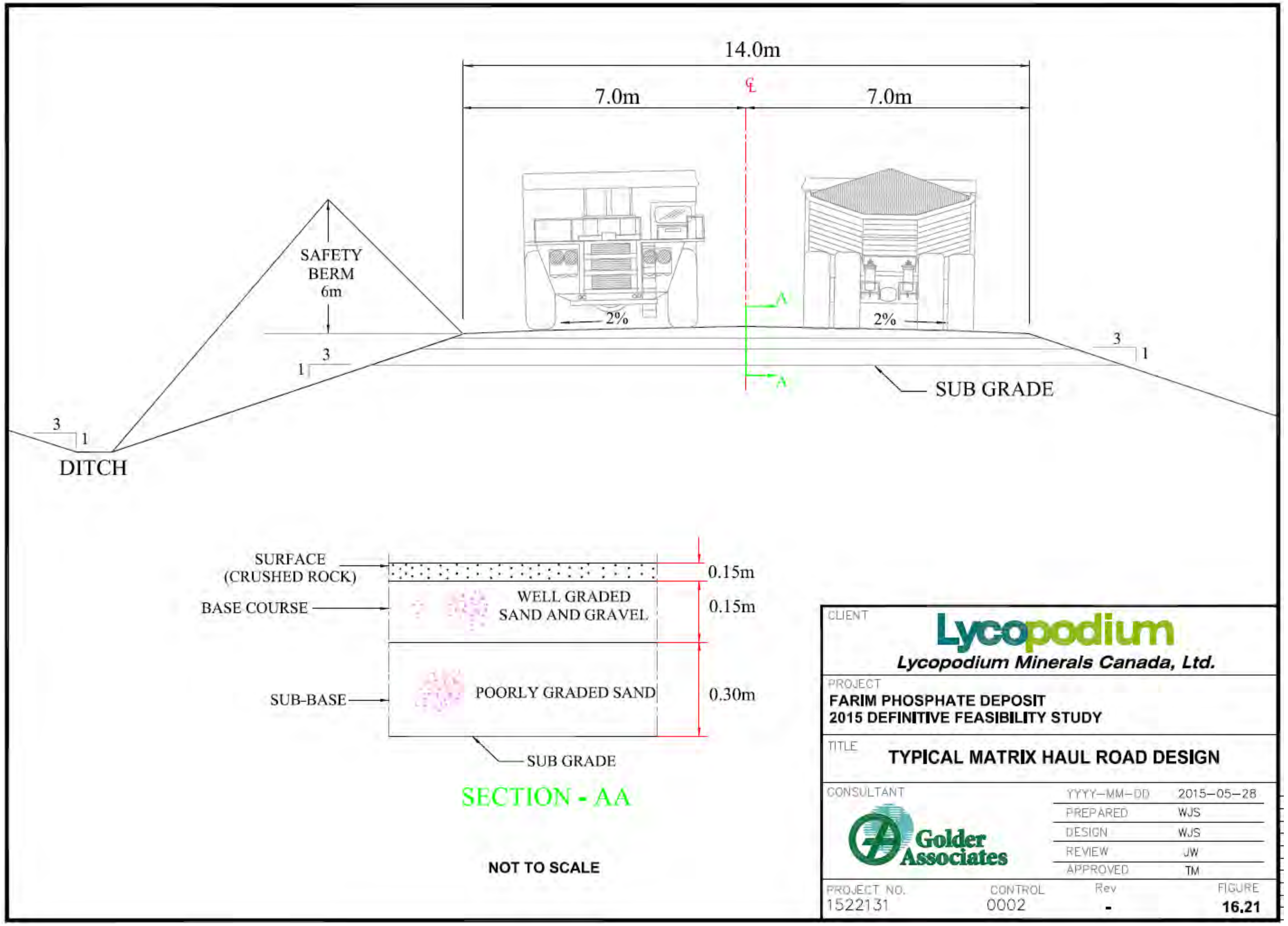


Figure 16-15 Typical Matrix Haul Road Design



16.5.12 Open Pit Monitoring

Due to the variability in the extent and types of soils, the excavations will require a high degree of visual inspection and monitoring of ground conditions. This is anticipated to include the following:

- Regular inspections of slope conditions;
- Installation and regular inspection of monitoring piezometers to confirm groundwater conditions;
- Regular surveying to monitor settlement;
- Installation and measurement of slip indicators (inclinometers) in areas of suspected slope instability and to verify design assumption, particularly in high slopes consisting primarily of clay units; and
- Undertaking and monitoring the timely remediation of identified unstable areas.

A monitoring and action plan should be developed before mining begins to define the monitoring goals, activities, and frequencies and actions to be taken at certain reading levels.

16.6 Optimized Pit Design

The 3D block model created in Minescape was imported into Vulcan for pit optimization and queried to determine the optimized resource for the mine plan. This optimized resource is located within a pit shell design, utilizing all the resource data contained within the 3D block model and the plan design parameters and factors summarized in the Table 16-1. Based on Golder's geotechnical review in Section 16.5, an overall highwall slope angle of 20° was applied to the optimizations.

The goal of the resource optimization analysis was to determine the optimized Resources that satisfy the mine production plan. The target annual production rate for the mine plan is 1.75 Mt (dry basis) of ROM matrix for 25 years, or a total of 43.75 Mt of ROM matrix for the planned mine life. The optimized resource was defined as the matrix with the best available P_2O_5 grade and lowest resultant strip ratio.

The 25 year mine plan pit design was established using the Lerchs-Grossman (LG) 3D algorithm function in Maptek's Vulcan mine planning software. Unit costs were compiled using up to date diesel and labour costs and applied to the waste volumes and ROM matrix tonnages in each block to calculate an overall mining cost. The table below summarizes the unit costs used in the pit optimization analysis.

Table 16-8 Summary of Unit Costs used in the Pit Optimization Analysis

Description	Value (USD / Unit)
Total Overburden Stripping Cost [1]	$1.56 / bcm
Total Matrix Mining Cost [2]	$4.01 / ROM tonne
Beneficiation [3]	$7.64 / ROM tonne
Port Land costs [3]	$3.98 / tonne rock
Shiploading [3]	$2.69 / tonne rock

Notes:
[1] Cost includes overburden stripping and haulage, operations support, and mine maintenance. Cost assumes a diesel price of USD $0.80/litre.
[2] Cost includes matrix mining and haulage, stockpiling, pit dewatering, reclamation, and mine supervision and administration. Cost assumes a diesel price of USD $0.80/litre.
[3] Cost provided by Lycopodium Minerals Canada, Ltd.

A simple script, or program, was written into the optimization analysis to calculate the mining costs associated with the matrix and overburden based upon the unit costs. For each block, a total cost of mining was calculated using the recovered waste volume, ROM tonnes, and expected rock (product) tonnes. If there was matrix within the block, revenue was assigned to it based on the estimated rock tonnes and ROM P_2O_5 grade.

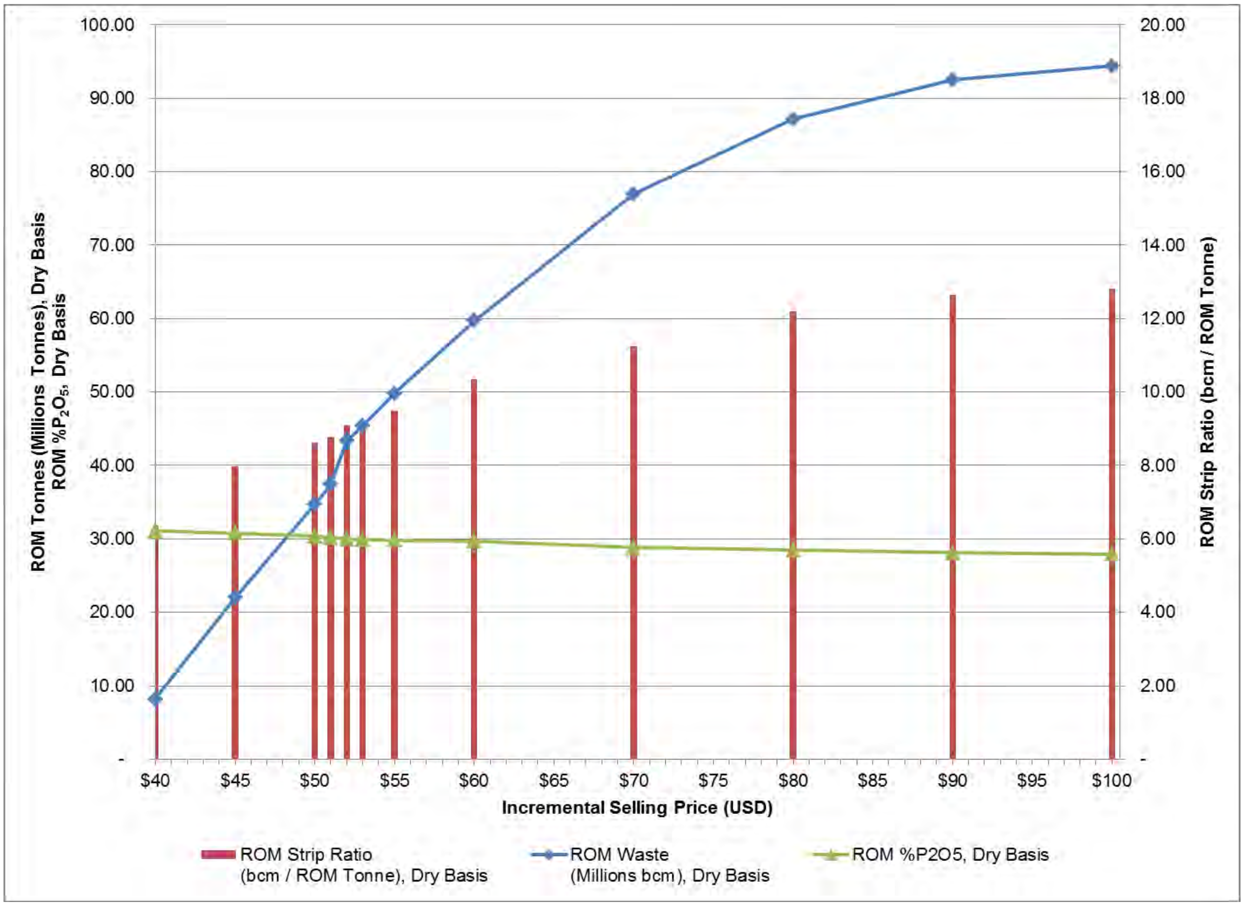
For each block, a total positive or negative value was calculated. Contained within the script was a feature to penalize ROM grade values that were lower than 29% P_2O_5 and reward blocks with a ROM grade value greater than 29% P_2O_5. Because the effects of beneficiation on phosphate rock P_2O_5 grade at Farim were not well defined at the time the optimization exercise was performed, this proration better ensures that minimum specifications for phosphate rock P_2O_5 grade can be achieved as P_2O_5 recovery generally increases with higher ROM (plant feed) P_2O_5 grade.

Optimization was conducted on Measured and Indicated Resources only; Inferred Resources were treated as waste. To prevent the optimized pits from taking blocks in other unwanted areas, the pit optimization analysis was limited to Measured and Indicated Resources north of the River Cacheu and excluded any Resource within the 100 m river control buffer that contains the flood protection bund. Additionally, a 300 m wide zone adjacent to the eastern ephemeral stream (Rio de Bunja) that flows near the current plant site was also flagged as overburden to better manage surface water issues that would occur while mining through from the ephemeral stream. Furthermore, the outermost blocks within the block model were flagged to confine the pit optimization within the block model.

For the LG optimization analysis, a phosphate rock selling price of USD $100/t phosphate rock at a grade of 29% ROM P_2O_5 by weight was used in the modelling process as an upper pit shell limit. Using this price as a baseline for the analysis resulted in matrix tonnages far in excess of the mine life resource target of 43.75 Mt (dry basis) of ROM matrix. Multiple iterations of the LG optimization were carried out using incrementally lower prices until a reasonable range was determined. The price range used for the LG optimization was from USD $40/t phosphate rock to USD $100/t phosphate rock in USD $10 increments, USD $40/t phosphate rock to USD $60/t phosphate rock in USD $5 increments, and from USD $50/t phosphate rock to USD $53/t phosphate rock in USD $1 increments.

At the time the optimizations were performed, the expected average mass yield of the ROM matrix was 70%. A graph comparing the incremental selling prices (at an assumed 70% mass yield of the ROM matrix) and resulting ROM tonnes and P_2O_5 grade contained within the incremental value pits is shown in Figure 16-16. Additionally, maps comparing the 25-year mine plan pit to the incremental value pit outlines have been provided to help better demonstrate the optimization process; these maps are provided as Figure 16-17 through to Figure 16-20.

Figure 16-16 Incremental Value Pit Reserves Comparison



The incremental LG pit optimizations from USD \$52 to USD \$53/t phosphate rock range produced pits containing phosphate rock tonnages in the 43.4 Mt to 45.4 Mt range. Both LG optimizations in this range resulted in two different pits, indicating that the optimized resource for the Project would require two distinct pits: a South Pit, and a North Pit.

Because the LG optimized pit resulting from a USD \$52/t rock selling price was within 0.5 Mt of the desired 25-year ROM tonnage, Golder made minor edits to the pit extents to create the detailed pit shell for the 25-year mine plan in Vulcan. The 25-year mine plan pit was designed using the criteria provided in Table 16-1, including a bench height of 10 m, a long-term berm width of 6.5 m, and a bench face (batter) angle of 25°. Additionally, a 24.5 m wide ramp was included in both the North and South Pits at the locations of the initial box cut for each pit to provide access to the mine floor. Each of these ramps will have a life span of one to two years until in-pit backfilling operations begin and the ramps progress with the face of the in-pit overburden backfill. The resulting pit design was exported from Vulcan and imported into Minescape to check ROM tonnage estimates and develop the mine plan and schedule.

Figure 16-17 LG Pit Optimizations – Comparison of 25-Year Mine Plan Pit & US$10 Incremental LG Pits ($40-$70)

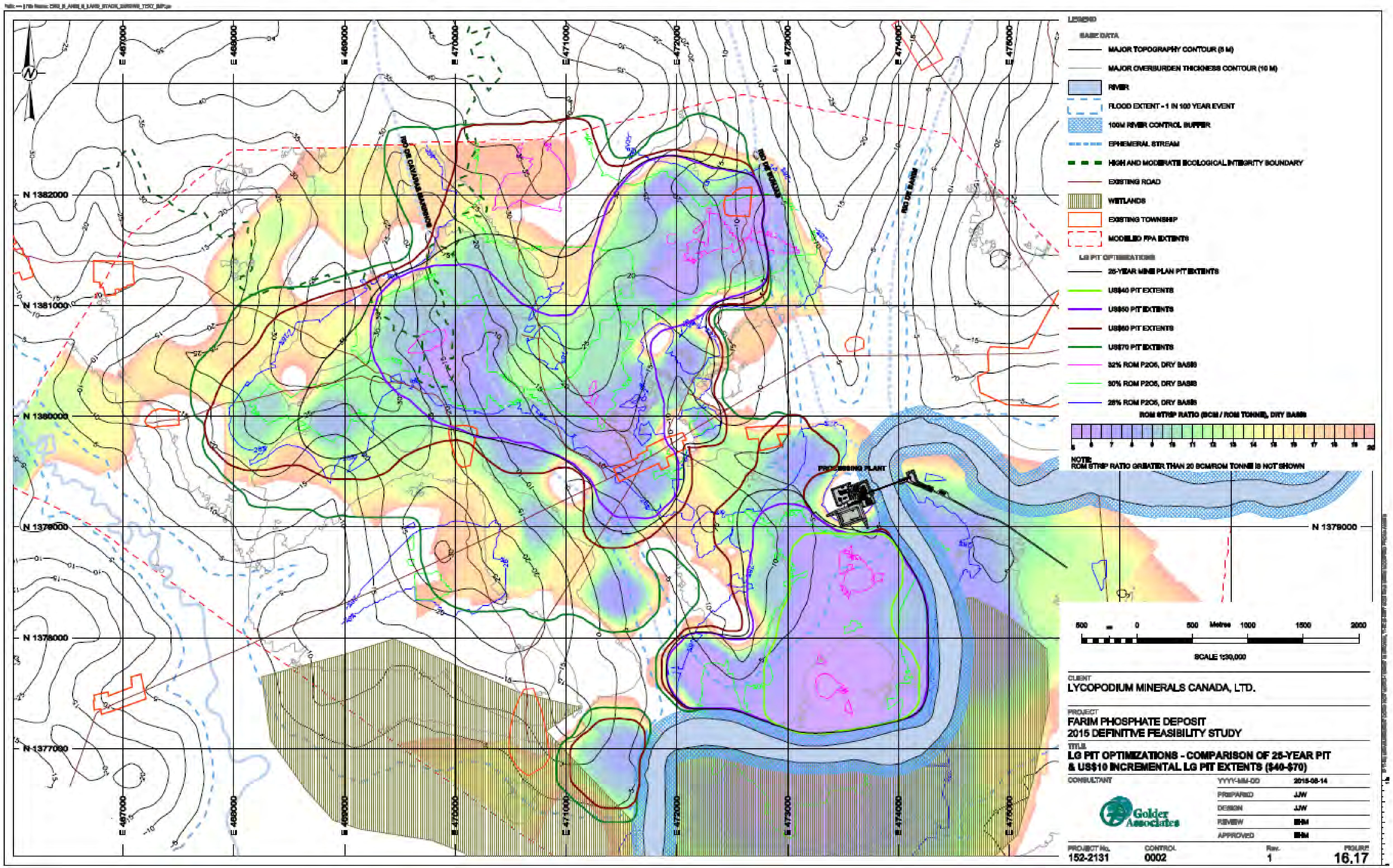


Figure 16-18 LG Pit Optimizations – Comparison of 25-Year Pit & US$10 Incremental LG Pits ($80-$100)

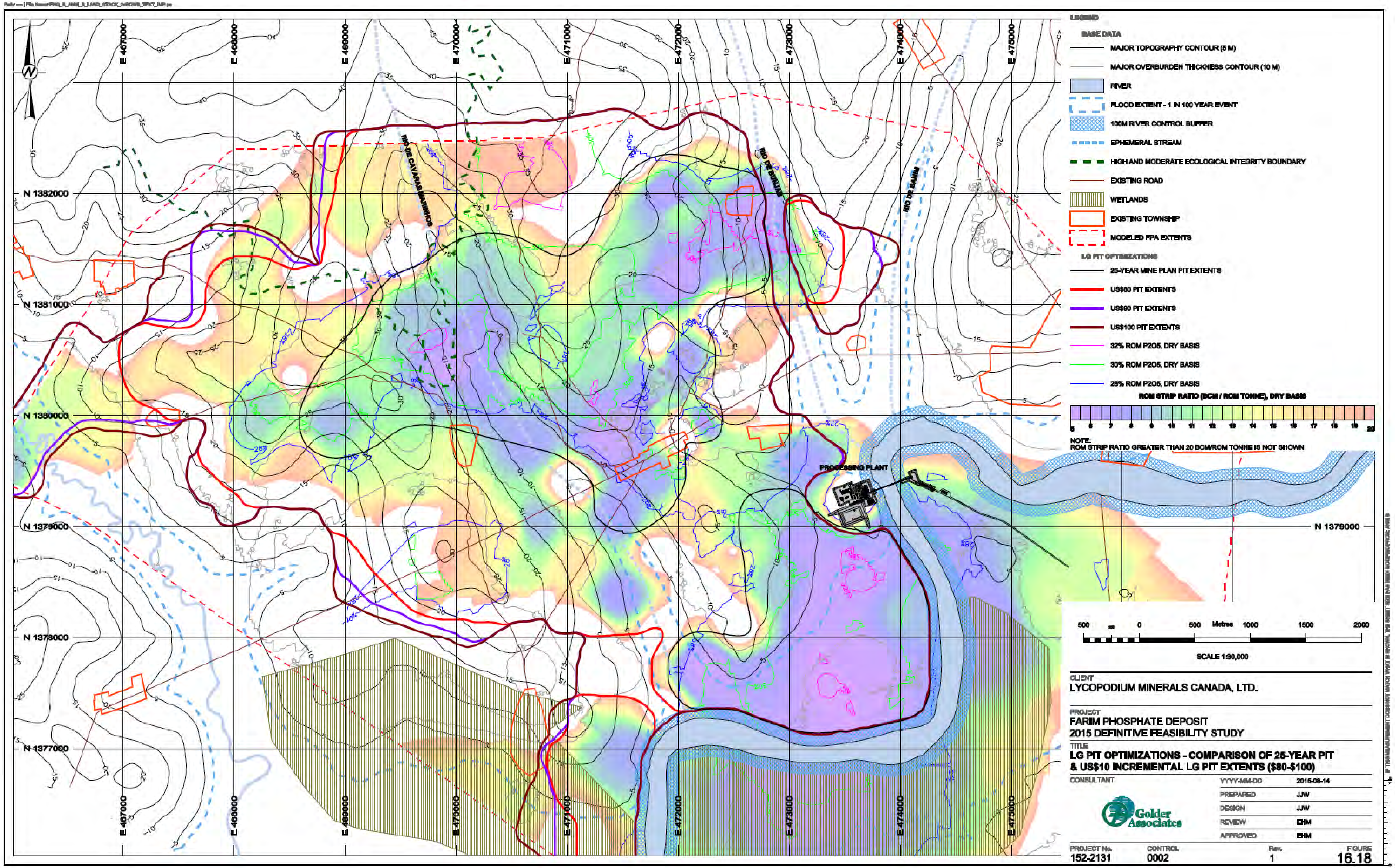


Figure 16-19 LG Pit Optimizations – Comparison of 25-Year Pit & US$5 Incremental LG Pits ($40-$60)

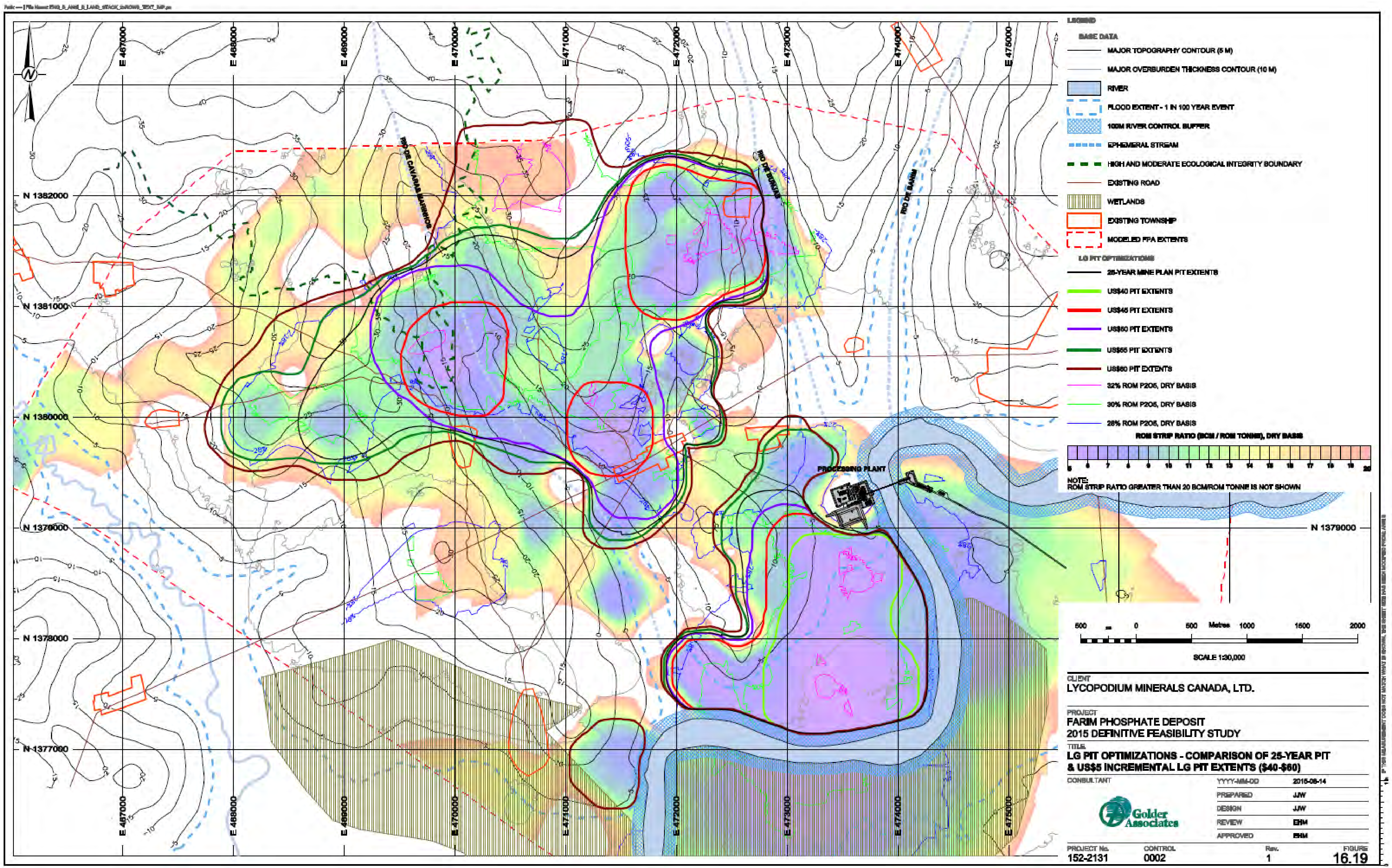
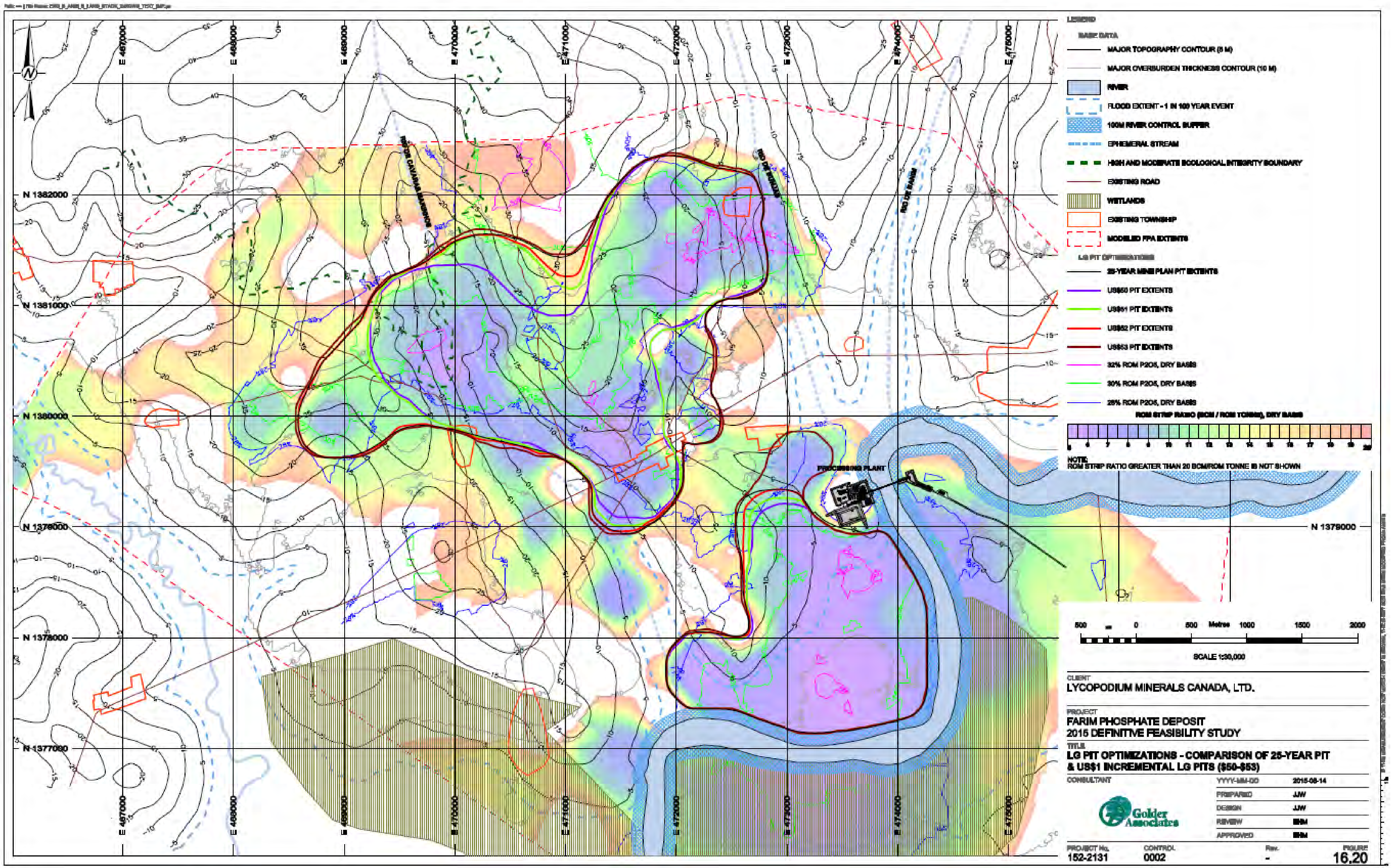


Figure 16-20 LG Pit Optimizations – Comparison of 25-Year Pit & US$1 Incremental LG Pits ($51-$53)



The resulting volumes, tonnages and grades contained within the 25-year mine plan pit design are shown in Table 16-9 below.

Table 16-9 25-Year Mine Plan Pit Resources

Category	Units	South Pit	North Pit	Total
In Situ Overburden Volume	000s BCM	102,866	319,082	421,948
In Situ FPA Tonnes (Dry Basis)	000s tonnes	14,442	32,247	46,689
Mean In Situ FPA Thickness	m	3.87	3.94	3.92
Mean In Situ %P_2O_5 (Dry Basis)	%	31.22	30.30	30.59
Mean In Situ %Al_2O_3 (Dry Basis)	%	2.34	2.65	2.55
Mean In Situ %CaO (Dry Basis)	%	40.51	41.16	40.96
Mean In Situ %Fe_2O_3 (Dry Basis)	%	3.77	5.14	4.72
Mean In Situ %SiO_2 (Dry Basis)	%	11.20	10.36	10.62
ROM Waste Volume (Dry Basis)	000s BCM	103,651	320,847	424,498
ROM (Plant Feed) FPA Tonnes (Dry Basis)	000s tonnes	13,611	30,396	44,007
ROM Strip Ratio (Dry Basis)	BCM / ROM Tonne	7.62	10.56	9.65
Mean ROM %P_2O_5 (Dry Basis)	%	30.61	29.71	29.99
Mean ROM %Al_2O_3 (Dry Basis)	%	2.34	2.65	2.55
Mean ROM %CaO (Dry Basis)	%	40.51	41.16	40.96
Mean ROM %Fe_2O_3 (Dry Basis)	%	3.77	5.14	4.72
Mean ROM %SiO_2 (Dry Basis)	%	11.20	10.36	10.62
Processing Plant Mass Yield	%	75.5	75.5	75.5
Rock (Product) Tonnes (Dry Basis)	000s tonnes	10,276	22,949	33,225
Mean Rock %P_2O_5 (Dry Basis)[1]	%	34.0	34.0	34.0
Tailings Tonnes (Dry Basis)[1]	000s tonnes	3,335	7,447	10,782

Notes:

[1] The expected product and tailings tonnes are based off an average plant mass yield of 75.5%.

[2] The 25-year pit outline is provided in Figure 16-21 through Figure 16-23.

Figure 16-21 General Arrangement of the 25-Year Mine Plan – ROM Strip Ratio (BCM / ROM Tonne), Dry Basis

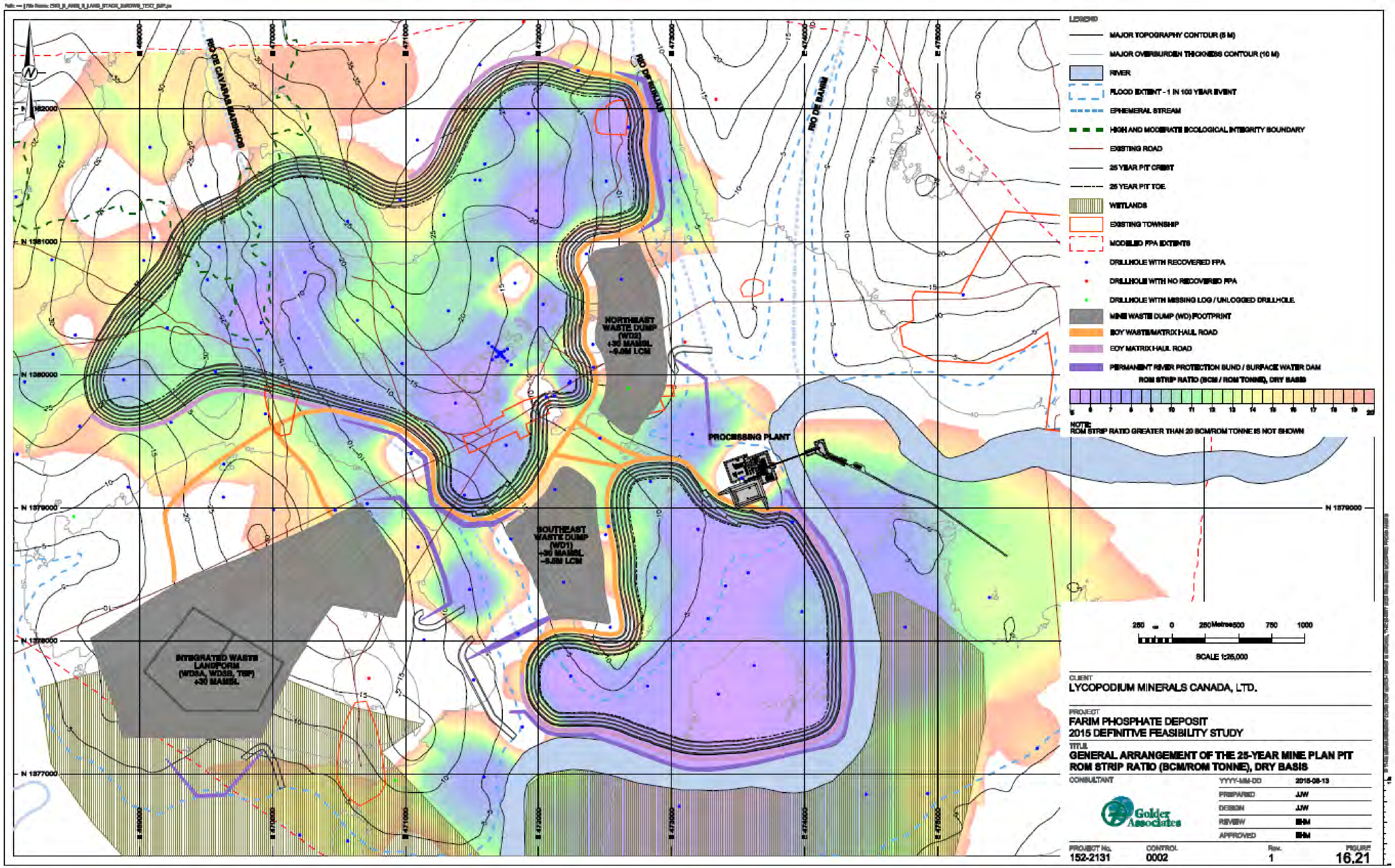


Figure 16-22 General Arrangement of the 25-Year Mine Plan – ROM %P_2O_5, Dry Basis

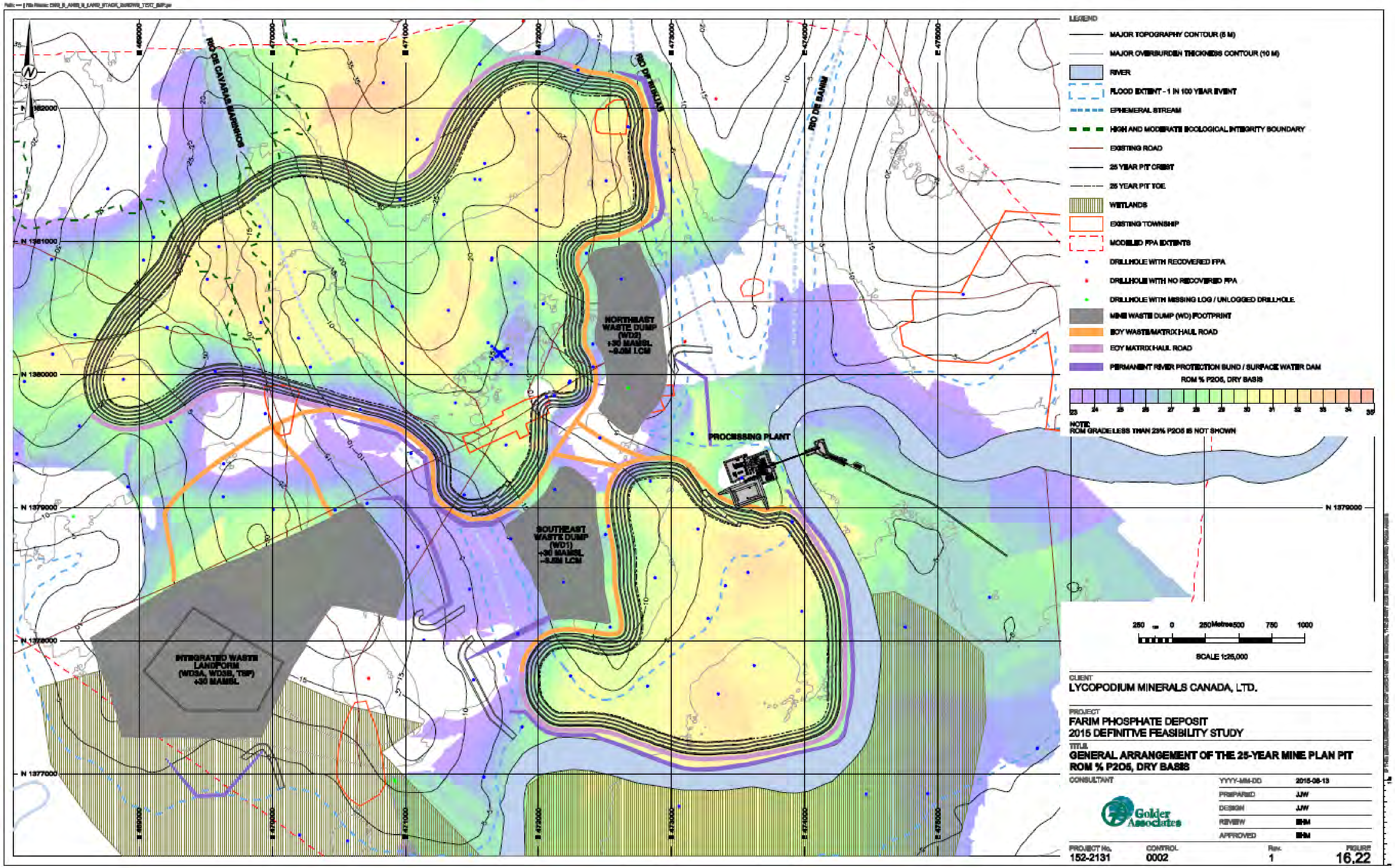
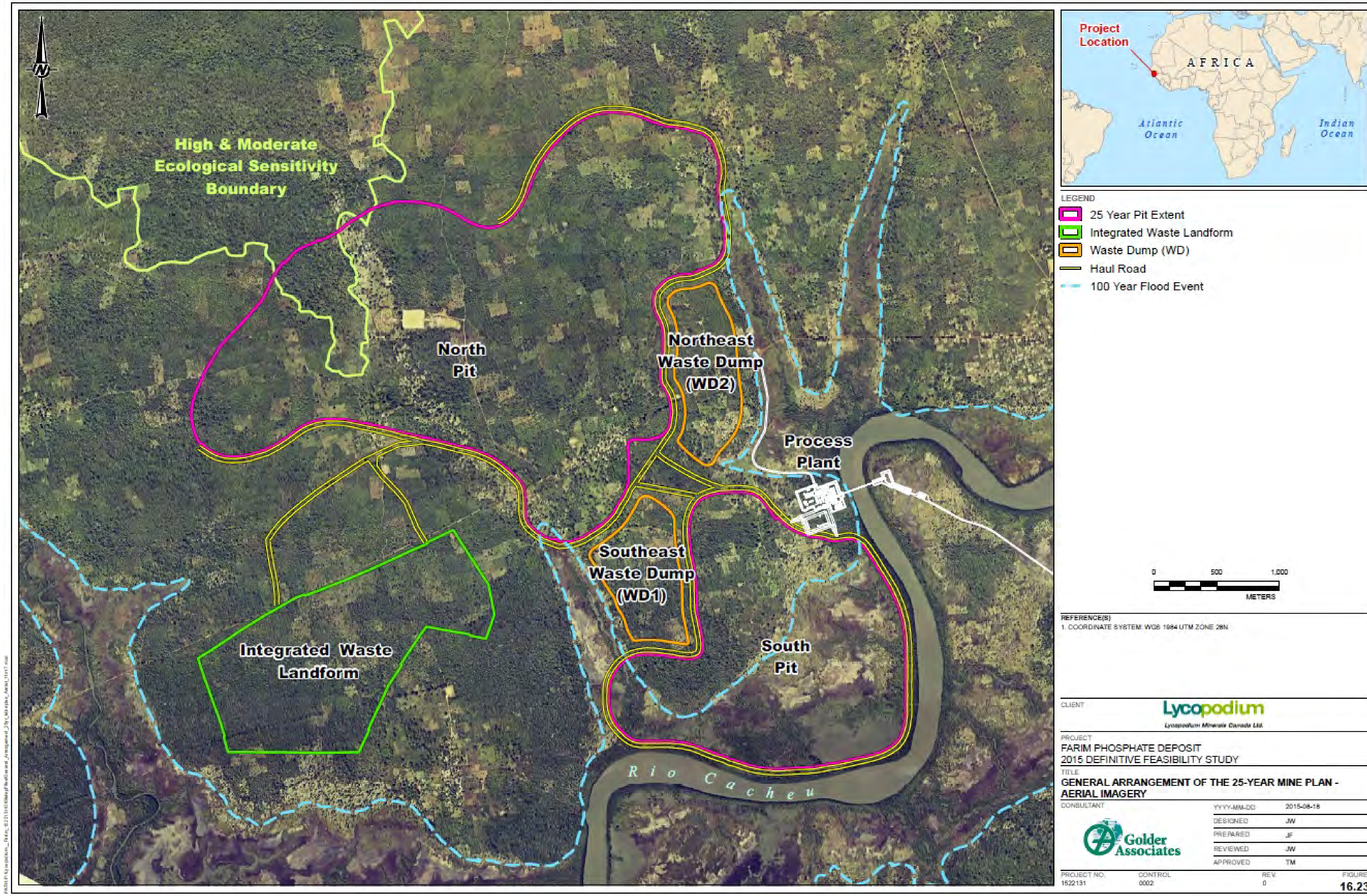


Figure 16-23 General Arrangement of the 25-Year Mine Plan – Aerial Imagery



16.7 Mining Plan Sequence

The mine plan production scenario was targeted to produce approximately 2.19 Mtpa of ROM phosphate matrix on an as-received basis (at approximately 20% moisture), or 1.75 Mtpa ROM phosphate matrix on a dry basis. The mine production schedule was developed to achieve these targets and to optimize the plan to defer costs and maximize net present value (NPV) while also providing a reasonable lead-in time for pit dewatering and surface water management activities.

A Minestar database was built to develop the LOM plan. Minestar is a customizable scheduling tool within the Minescape software that provides the user with mine production statistics and a graphical representation of the mining advance. This allows the user to interactively sequence the mining progression and to optimize the production schedule. Minestar allows the user to query the database to determine remaining volumes, tonnages, and grades to ensure that future production targets can be met. This dynamic feedback allows the user to fine-tune a mine plan to meet production needs, and to manage difficult mining areas due to grades or geology. The Minestar reporting functionality provides detailed production summaries and data for development of the cost models, and creates blocks color coded by scheduling period for the development of end-of-period maps.

Separate scheduling blocks 50 m by 50 m in size were developed for the FPA matrix and each 10 m overburden interval. This block size was chosen to provide a high degree of resolution while maintaining the ability to analyze an alternative scheduling option in a timely manner. The scheduling blocks were confined by the 25-year mine plan pit shell and topographic surfaces to exclude volumes or tonnages outside of the pit. All necessary volume, tonnage, and grade data were calculated by block, processed in a Microsoft Access database, and loaded into Minestar for scheduling. Approximately 26,300 total scheduling blocks containing overburden and matrix were created for the North and South pits.

The mine plan was developed to produce 1.75 Mtpa of ROM matrix for 25 years. The mine sequence includes six months of pre-stripping in “Year 0” to allow for immediate matrix production in Year 1. Approximately three months of matrix inventory was pre-stripped in Year 0. The complete yearly matrix advance sequence is shown in Figure 16-24. The yearly production statistics associated with the sequence are shown in Table 16-10. Note that a Year 26 was added to the production schedule to mine out the remaining 257,000 t of matrix in the designed pit shell.

Four key factors drove the progression of the sequence. In decreasing order of importance, these were: annual ROM production, stripping ratio, dewatering and surface water management, and backfill opportunities. The mine was sequenced with stripping ratio increasing from low-to-high to the extent possible to defer capital and operating costs and to minimize investment risk. The current plan was revised from the proposed mine plan of 2012 to mine the initial South Pit from north to south. This revision increases strip ratio in the initial years but allows mine operations to gain experience in groundwater and geotechnical conditions as mining approaches the River Cacheu. The revision also minimizes haul distances to the processing plant and results in a pit geometry that is more conducive to in-pit backfilling in the initial years, thus minimizing haul cycle times and haul truck fleet requirements to the extent practical. The sequence through the North Pit also accounted for dewatering demands associated with the western ephemeral stream (Rio de Bunja) to reduce stress on the open pit.

Table 16-10 Annual Mine Plan Production Statistics – Page 1

Category	Units	Production Year															
		0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
In Situ Overburden Volume	000s BCM	5,811	11,077	14,819	14,214	12,978	11,798	10,849	13,146	17,598	18,253	17,153	17,854	19,391	19,412	19,373	19,355
In Situ FPA Tonnes, Dry Basis	000s tonnes	-	1,855	1,857	1,858	1,856	1,856	1,856	1,857	1,860	1,875	1,855	1,857	1,856	1,854	1,854	1,855
In Situ FPA Thickness	m	-	4.04	3.75	3.54	3.88	3.95	4.15	3.80	3.65	3.69	4.27	3.62	3.69	4.23	4.24	3.89
In Situ P_2O_5, Dry Basis	%	-	31.56	31.85	30.94	30.87	31.10	31.78	31.23	29.55	29.27	28.85	28.71	30.96	30.90	31.05	31.93
In Situ Al_2O_3, Dry Basis	%	-	2.50	2.27	2.21	2.16	2.37	2.22	2.17	2.72	2.82	2.47	2.72	2.04	1.75	1.61	1.50
In Situ CaO, Dry Basis	%	-	40.67	41.37	41.60	40.87	39.90	39.44	39.89	38.97	39.41	40.47	39.85	42.83	43.49	43.33	42.22
In Situ Fe_2O_3, Dry Basis	%	-	3.96	3.67	3.70	4.03	4.32	3.68	3.28	5.36	7.31	5.53	4.81	3.80	3.63	3.98	5.61
In Situ SiO_2, Dry Basis	%	-	11.21	10.66	11.12	11.46	11.41	11.54	11.24	11.78	10.46	11.89	13.31	10.09	8.86	8.73	8.90
ROM Waste Volume	000s BCM	5,818	11,172	14,922	14,318	13,079	11,898	10,947	13,247	17,701	18,369	17,252	17,959	19,494	19,512	19,472	19,454
ROM (Plant Feed) Tonnes, Dry Basis	000s tonnes	-	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	-	6.38	8.53	8.18	7.47	6.80	6.26	7.57	10.11	10.50	9.86	10.26	11.14	11.15	11.13	11.12
Cumulative ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	-	9.71	9.12	8.81	8.47	8.14	7.82	7.79	8.08	8.35	8.50	8.66	8.87	9.04	9.19	9.32
ROM P_2O_5, Dry Basis	%	-	30.97	31.21	30.26	30.26	30.51	31.20	30.61	28.95	28.62	28.31	28.08	30.30	30.31	30.47	31.31
ROM Al_2O_3, Dry Basis	%	-	2.50	2.27	2.21	2.16	2.37	2.22	2.17	2.71	2.80	2.47	2.72	2.04	1.75	1.61	1.50
ROM CaO, Dry Basis	%	-	40.67	41.37	41.60	40.87	39.91	39.44	39.89	38.97	39.43	40.47	39.85	42.83	43.49	43.33	42.22
ROM Fe_2O_3, Dry Basis	%	-	3.96	3.67	3.70	4.03	4.32	3.68	3.28	5.35	7.33	5.53	4.81	3.80	3.63	3.98	5.61
ROM SiO_2, Dry Basis	%	-	11.21	10.66	11.13	11.46	11.41	11.54	11.24	11.78	10.45	11.88	13.31	10.09	8.86	8.72	8.90
Rock (Product) Tonnes, Dry Basis	000s tonnes	-	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321
Rock %P_2O_5 [1], Dry Basis	%	-	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00
Tailings Tonnes [1], Dry Basis	000s tonnes	-	429	429	429	429	429	429	429	429	429	429	429	429	429	429	429

Notes:
1 Expected product tonnages are based off of an average 75.5% plant mass yield.

Table 16-10 Annual Mine Plan Production Statistics - Page 2

Category	Units	Production Year											25 Year Total / Average	26 Year Total / Average
		16	17	18	19	20	21	22	23	24	25	26		
In Situ Overburden Volume	000s BCM	19,458	18,707	17,504	17,476	14,995	15,081	15,270	16,944	19,425	21,182	2,827	419,121	421,948
In Situ FPA Tonnes, Dry Basis	000s tonnes	1,855	1,854	1,856	1,856	1,857	1,857	1,855	1,854	1,855	1,856	273	46,417	46,689
In Situ FPA Thickness	m	4.31	4.55	3.56	3.66	3.42	3.45	3.95	4.53	4.19	3.91	3.80	3.92	3.92
In Situ P_2O_5, Dry Basis	%	31.12	31.34	29.96	29.39	29.28	30.01	31.27	31.68	30.17	29.89	30.83	30.59	30.59
In Situ Al_2O_3, Dry Basis	%	1.93	2.33	3.06	3.15	3.37	3.21	3.15	3.33	3.43	3.18	3.91	2.55	2.55
In Situ CaO, Dry Basis	%	41.30	42.31	41.68	40.80	39.27	39.90	41.32	41.72	40.48	40.83	41.26	40.96	40.96
In Situ Fe_2O_3, Dry Basis	%	6.14	4.62	4.67	5.26	5.54	5.66	5.08	4.39	4.66	5.25	4.87	4.72	4.72
In Situ SiO_2, Dry Basis	%	8.42	8.40	9.17	9.48	11.82	11.65	10.18	9.92	12.85	11.12	9.23	10.63	10.62
ROM Waste Volume	000s BCM	19,555	18,806	17,605	17,580	15,099	15,186	15,370	17,040	19,521	21,282	2,841	421,657	424,498
ROM (Plant Feed) Tonnes, Dry Basis	000s tonnes	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	257	43,750	44,007
ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	11.17	10.75	10.06	10.05	8.63	8.68	8.78	9.74	11.15	12.16	11.06	9.64	9.65
Cumulative ROM Strip Ratio, Dry Basis	BCM / ROM Tonne	9.43	9.51	9.54	9.57	9.52	9.48	9.45	9.46	9.53	9.64	9.65	9.64	9.65
ROM P_2O_5, Dry Basis	%	30.58	30.81	29.32	28.78	28.63	29.35	30.66	31.15	29.63	29.32	30.24	29.98	29.99
ROM Al_2O_3, Dry Basis	%	1.93	2.33	3.06	3.15	3.37	3.21	3.15	3.33	3.43	3.18	3.91	2.55	2.55
ROM CaO, Dry Basis	%	41.30	42.31	41.68	40.80	39.28	39.91	41.32	41.72	40.48	40.83	41.26	40.96	40.96
ROM Fe_2O_3, Dry Basis	%	6.14	4.62	4.67	5.26	5.54	5.66	5.08	4.38	4.66	5.25	4.87	4.72	4.72
ROM SiO_2, Dry Basis	%	8.42	8.40	9.17	9.48	11.82	11.65	10.17	9.92	12.85	11.12	9.23	10.63	10.62
Rock (Product) Tonnes, Dry Basis	000s tonnes	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	194	33,031	33,225
Rock %P_2O_5 [1], Dry Basis	%	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00	34.00
Tailings Tonnes [1], Dry Basis	000s tonnes	429	429	429	429	429	429	429	429	429	429	63	10,719	10,782

Notes:

1 Expected product tonnages are based off of an average 75.5% plant mass yield

16.7.1 Pit Progression

For the 2015 FS, the South Pit was opened up in a relatively high grade area adjacent to the plant that is outside the extents of the 1 in 100 year flood event. This approach minimizes risk by providing adequate time for the incremental construction of a bund around the perimeter of the River Cacheu to prevent pit flooding, and allows appropriate lead-in time for pit dewatering ahead of the mining advance. However, this approach also increases strip ratio early in the mine life as the lowest strip ratio material is within the extents of the 1 in 100 year flood even adjacent to River Cacheu. The effects of this approach on strip ratio early in the mine life are evident in Figure 16-25.

The overburden material pre-stripped in Year 0 will be used for construction of both the River Cacheu protection bund and the tailings embankment located west of the township of Canico.

After pre-stripping overburden in Year 0, the mine progresses to the southwest along the South Pit highwall towards the River Cacheu to chase the Resource with the lowest strip ratio in Years 1 and 2. The higher strip ratio Resource is incrementally mined in Years 2 and 3 to provide a pit geometry that maximizes backfill opportunities while minimizing strip ratio to the extent possible. Independent access is maintained to each mining bench throughout the mine life, allowing flexibility for mining advance and matrix extraction. After the mining advance expands across the full width of the 25-year South Pit extents by the end of Year 3, the mine face advances in a linear progression to the southwest until the South Pit is mined out in Year 8.

The North Pit is opened up in Year 8 as the tail end of the South Pit is mined out to allow for a complete transition of matrix production to the North Pit in Year 9. The mine face then progresses north-northeast from Years 9 through 14 to avoid disturbing the western ephemeral stream and allow time for a diversion ditch to be constructed through the IOB. High strip ratio Resource from the North Pit is incrementally mined with the lower strip ratio Resource in Years 12 through 17 to balance strip ratio and equipment requirements to the extent possible. In Year 18, the mining face shifts to the west across the full width of the 25-year North Pit extents and progresses linearly from east to the west through the remainder of the mine life.

Year 20 represents a critical juncture in the mine life as the overburden advance progresses through the western ephemeral stream (Rio de Cavaras Marinhos). At this time enough of the North Pit must be backfilled to reroute the ephemeral stream through the IOB using a diversion channel. Failure to reroute the western ephemeral stream through the IOB ahead of the mining advance will necessitate the use of different management methods to divert the large volumes of water from the heavy rainy season away from the pit. Previous studies have investigated the construction of a large impoundment structure with additional pumps and pipeline to manage surface water runoff from the ephemeral stream but have shown this to be a high-risk and high-cost approach (Surface Water Management 1.3 Mtpa Open Pit Feasibility Report, Golder 2012).

The mine plan met production and scheduling goals. At least 1.75 Mt of ROM matrix (dry basis) are delivered to the plant each year with a surplus of approximately 257,000 t over the LOM. As seen in Figure 16-25, the yearly strip ratio sees an initial peak in Year 2 and gradually decreases as the South Pit is developed and more of the low strip ratio Resource adjacent to River Cacheu is mined. The strip ratio then sees another large increase from Years 7 to 8 as production transitions from the South Pit to the higher strip ratio North Pit. A comparison of the yearly ROM (plant feed) grades is provided in Figure 16-26 in the coming pages.

End-of-period maps showing the mine progression, access, haul road progression, and facilities annually for Years 0 through 5, and Years 8, 10, 15, 20, and 26 have been provided as Figure 16-27 through Figure 16-37

An additional end-of-period map showing the status of the mine after the void in the North Pit at the end of the mine life has been backfilled is provided as Figure 16-38.

Figure 16-24 General Arrangement of the 25-Year Mine Plan – Yearly Matrix Extraction

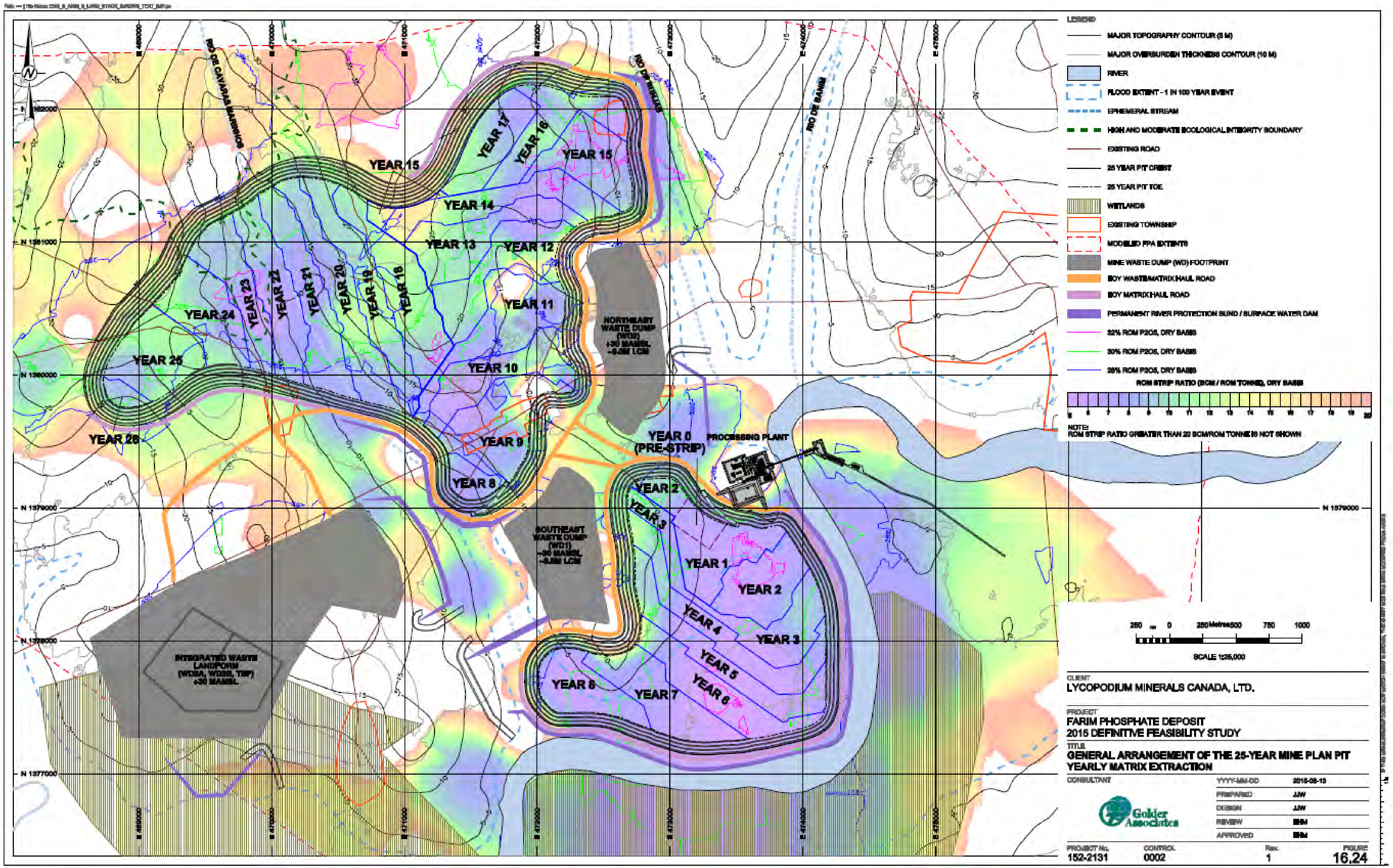


Figure 16-25 Annual Mine Production

Figure 16-26 Annual ROM Qualities

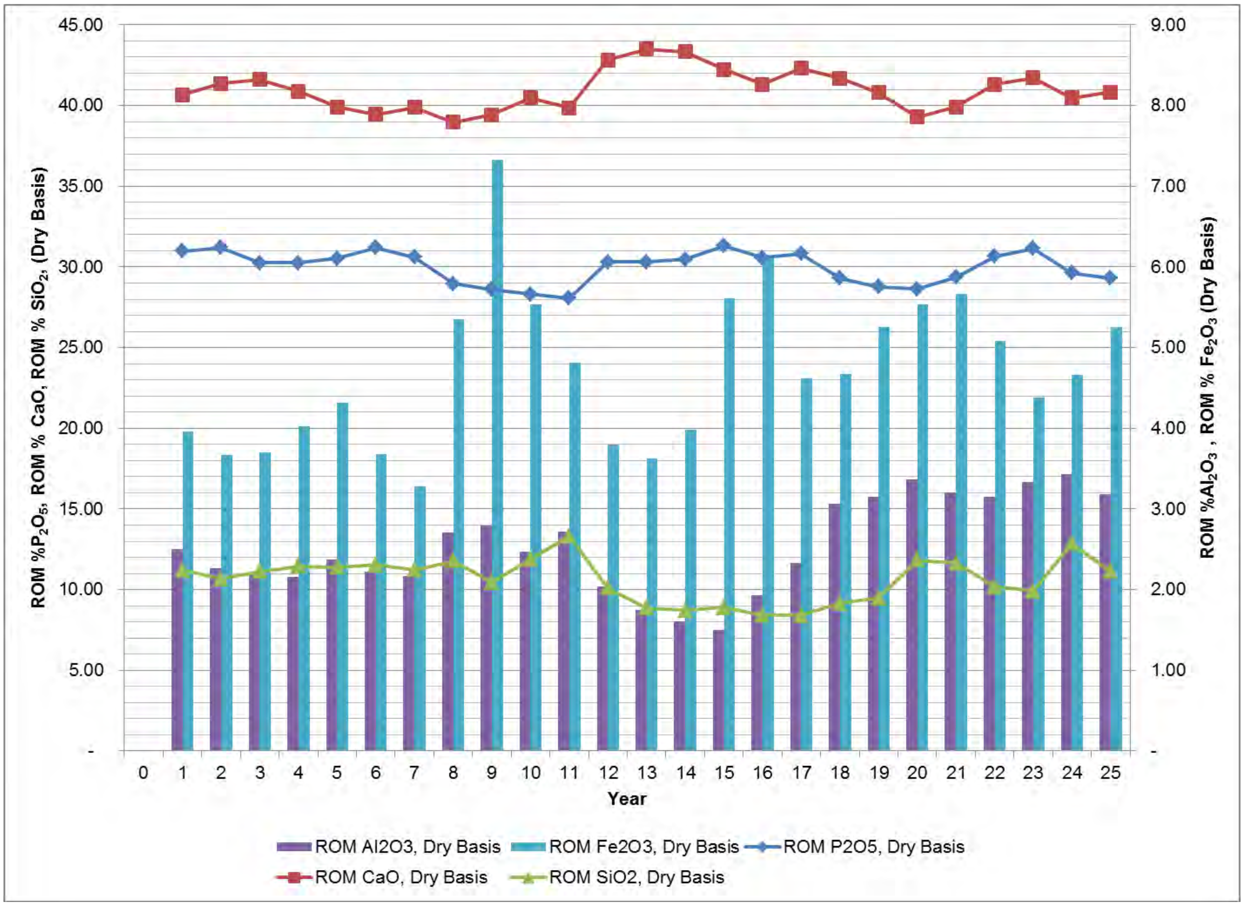


Figure 16-27 Mine Status Map — End-of-Year 0

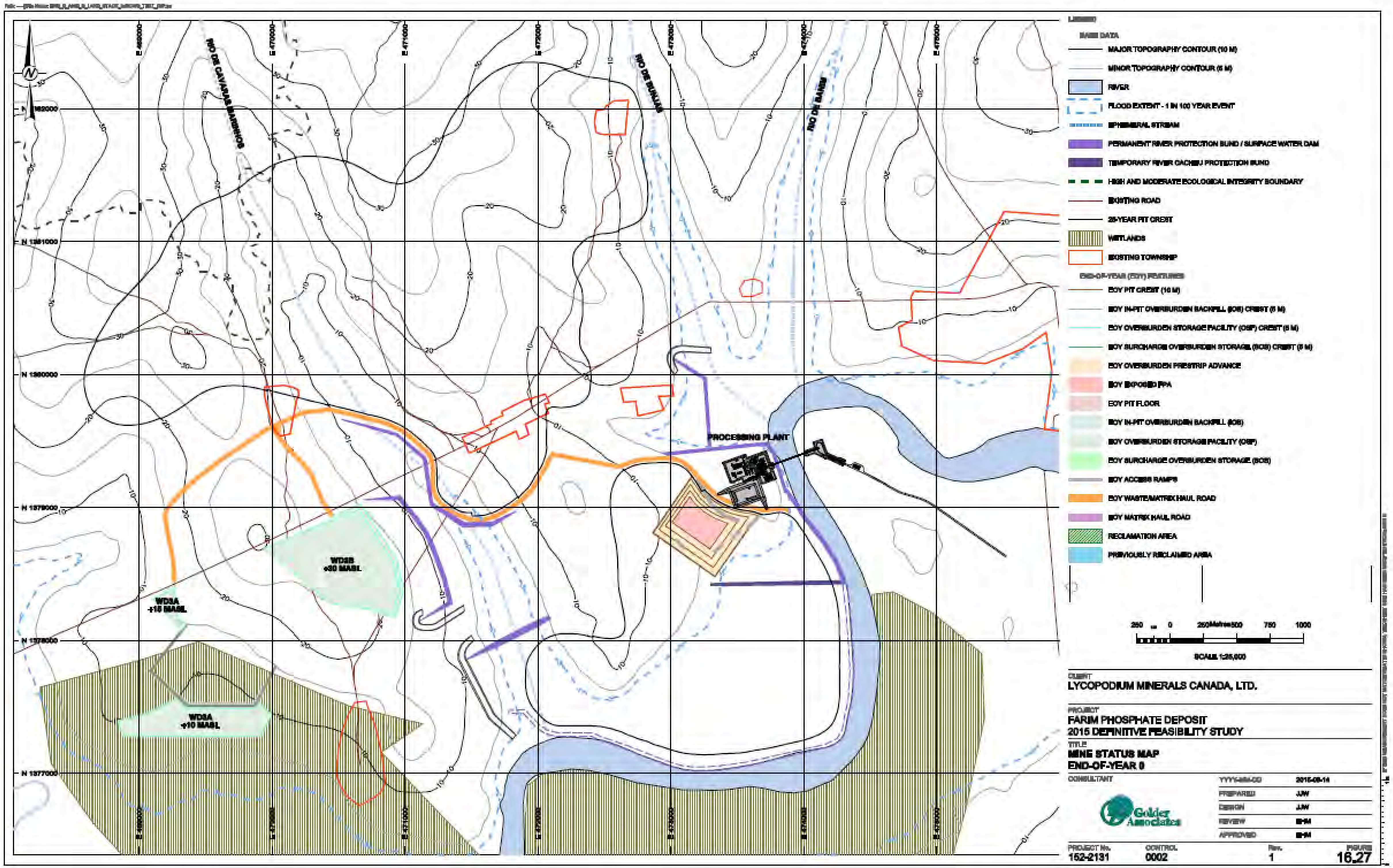


Figure 16-28 Mine Status Map — End-of-Year 1

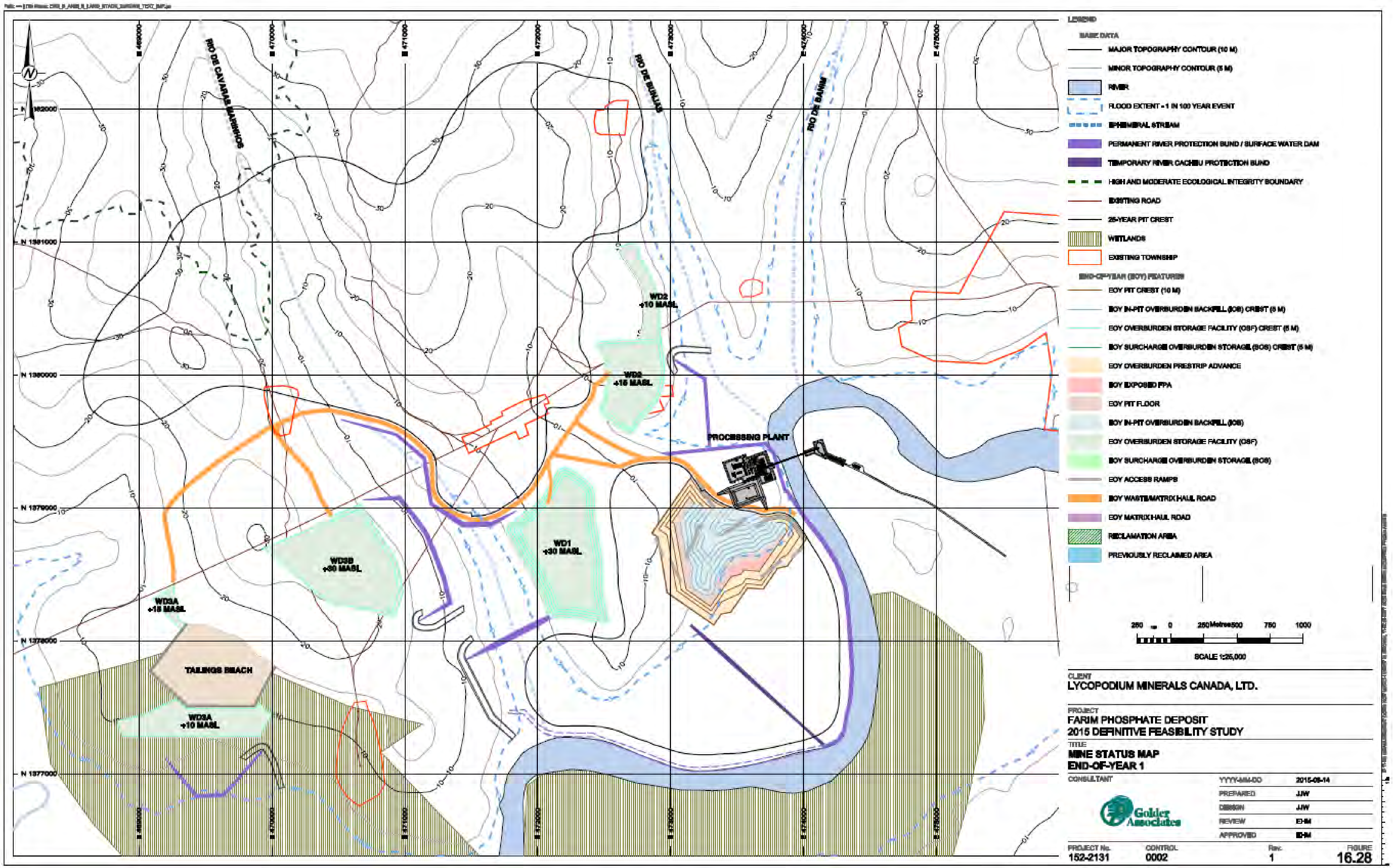


Figure 16-29 Mine Status Map — End-of-Year 2

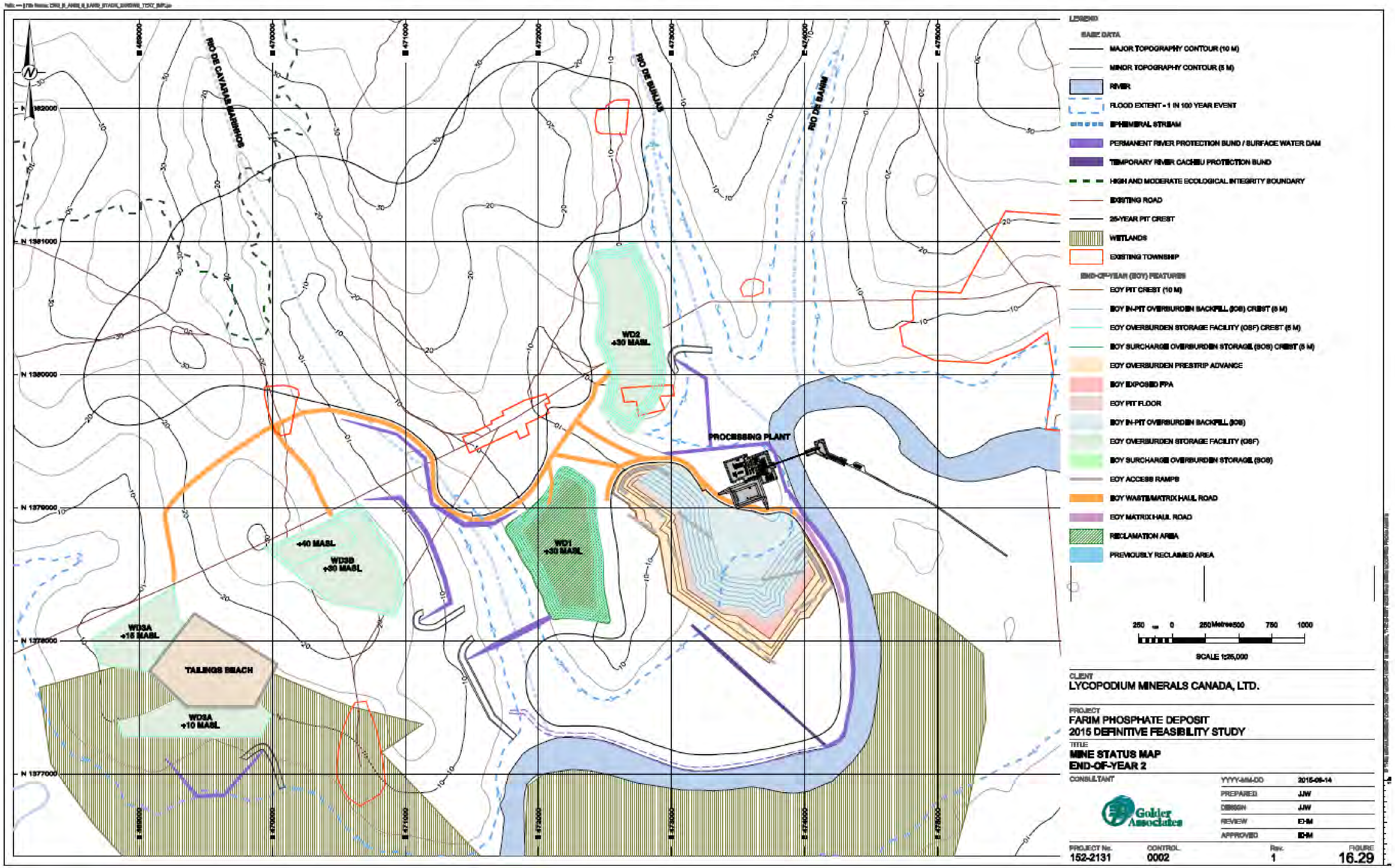


Figure 16-30 Mine Status Map — End-of-Year 3

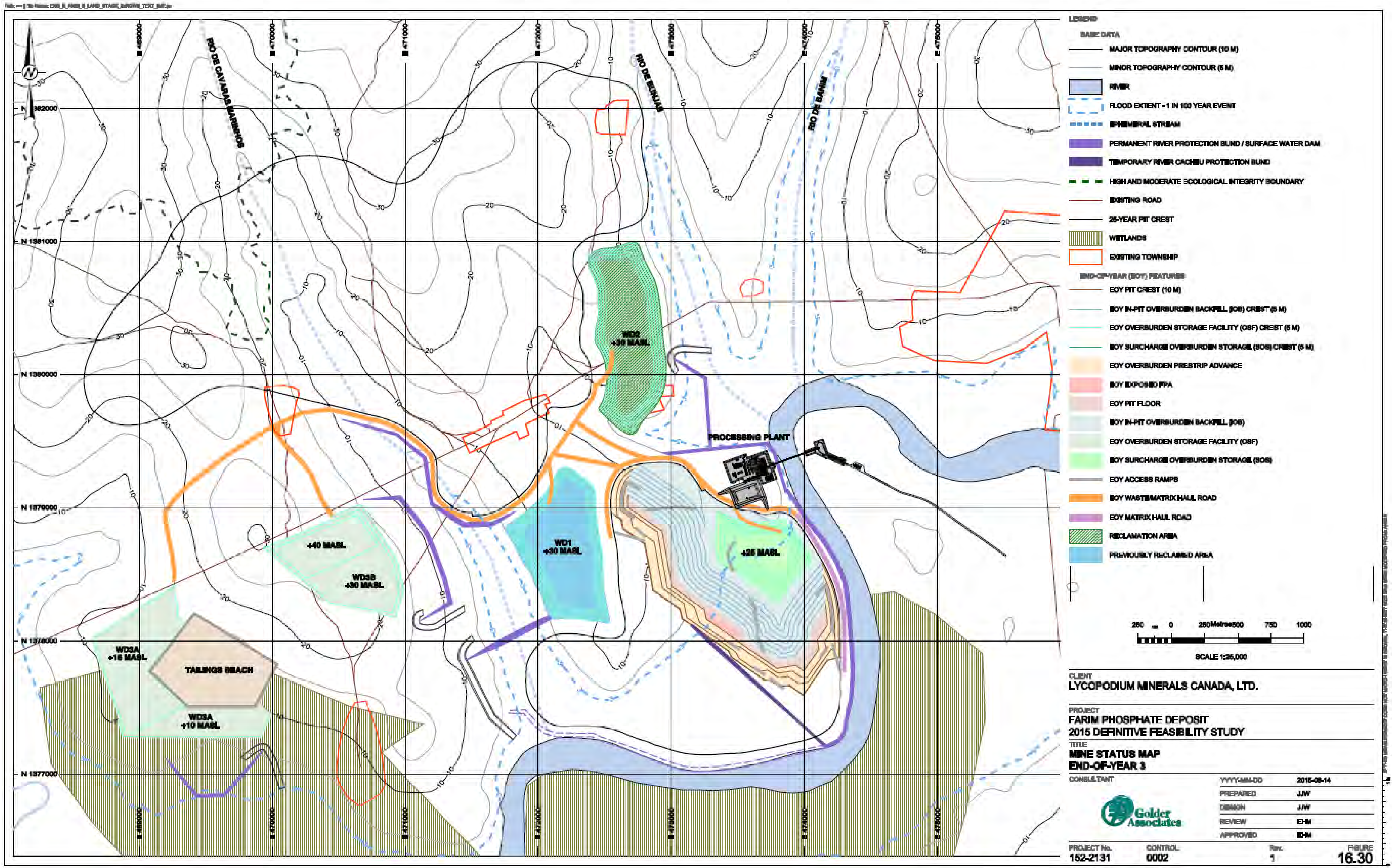


Figure 16-31 Mine Status Map — End-of-Year 4

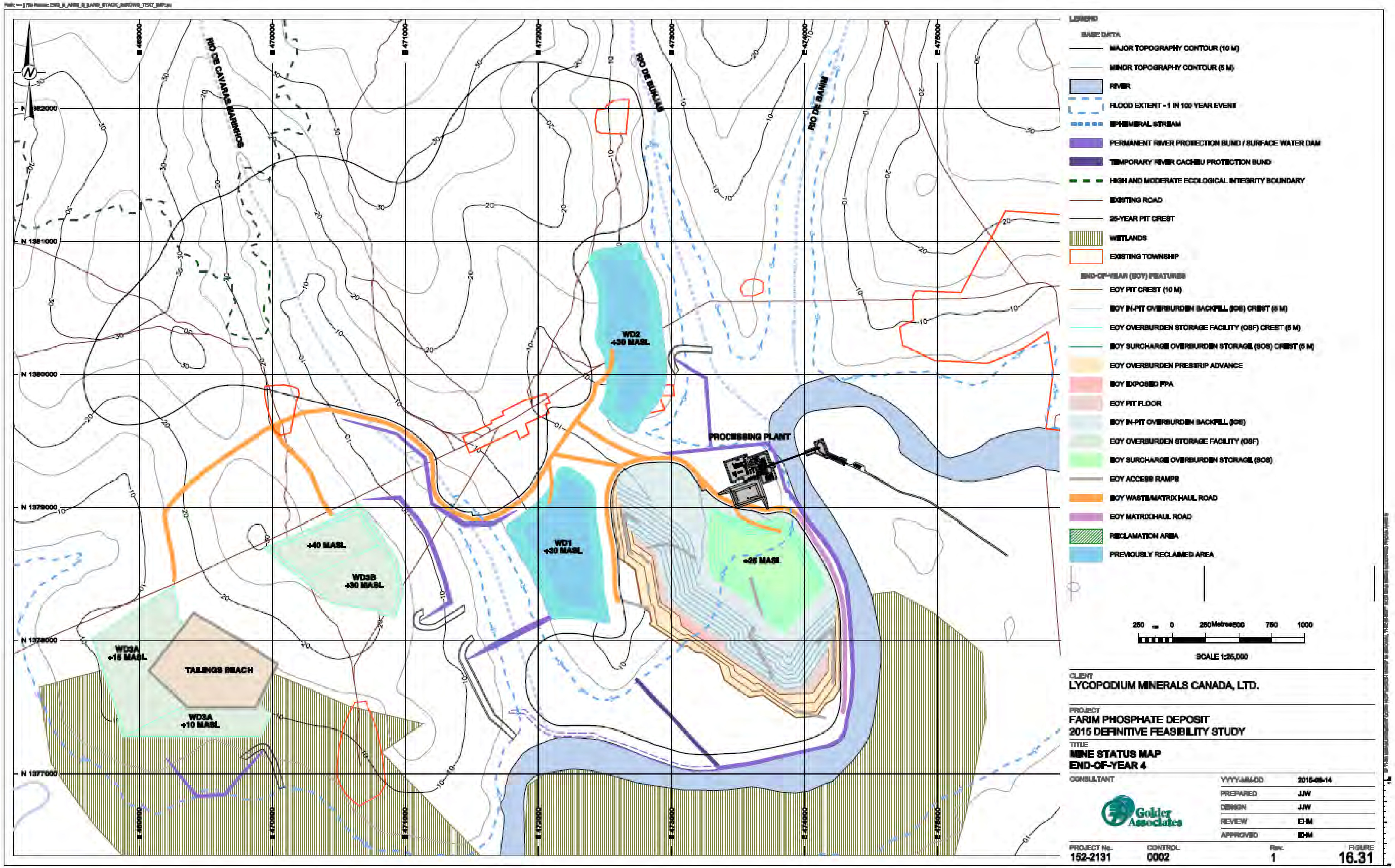


Figure 16-32 Mine Status Map — End-of-Year 5

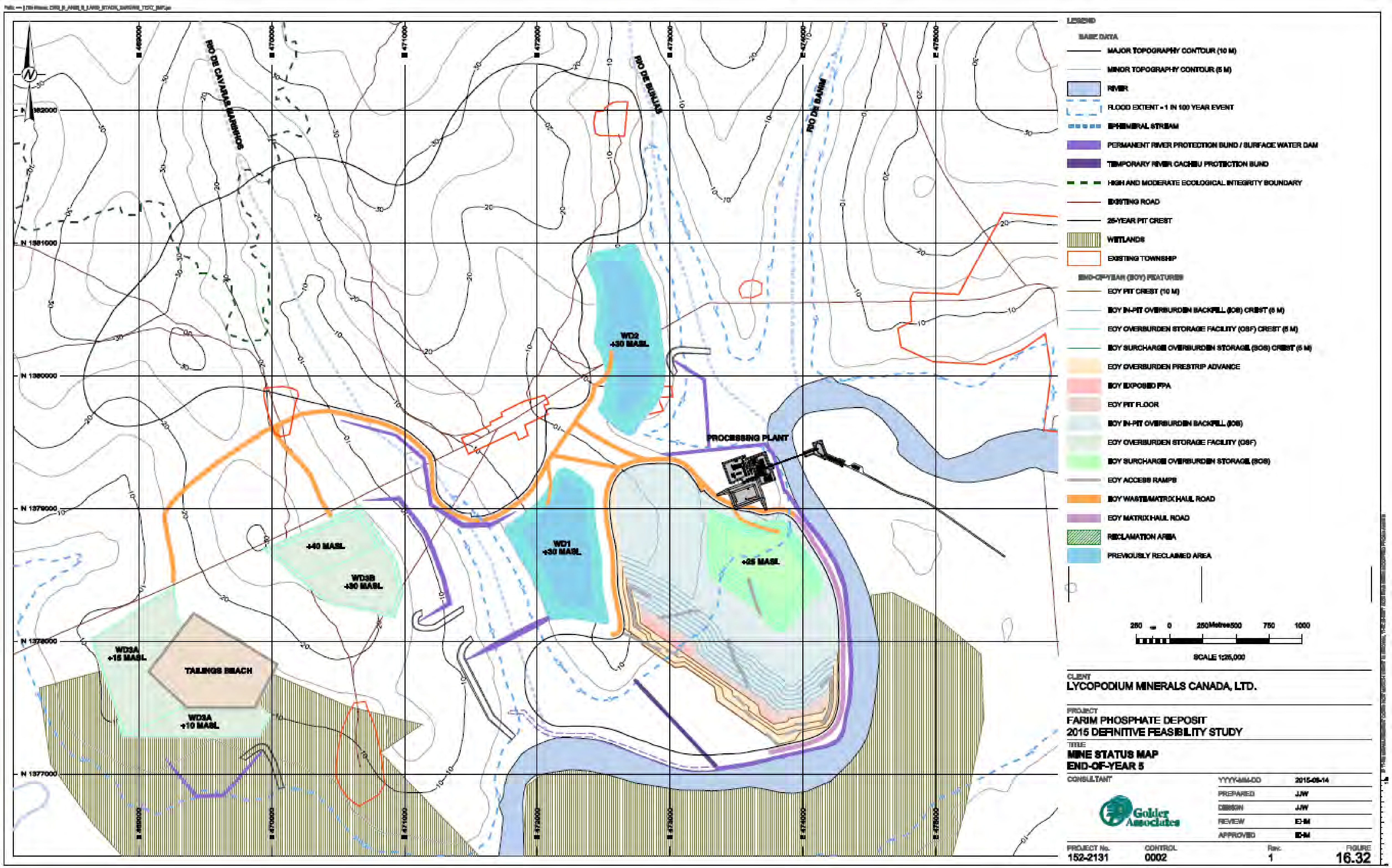


Figure 16-33 Mine Status Map — End-of-Year 8

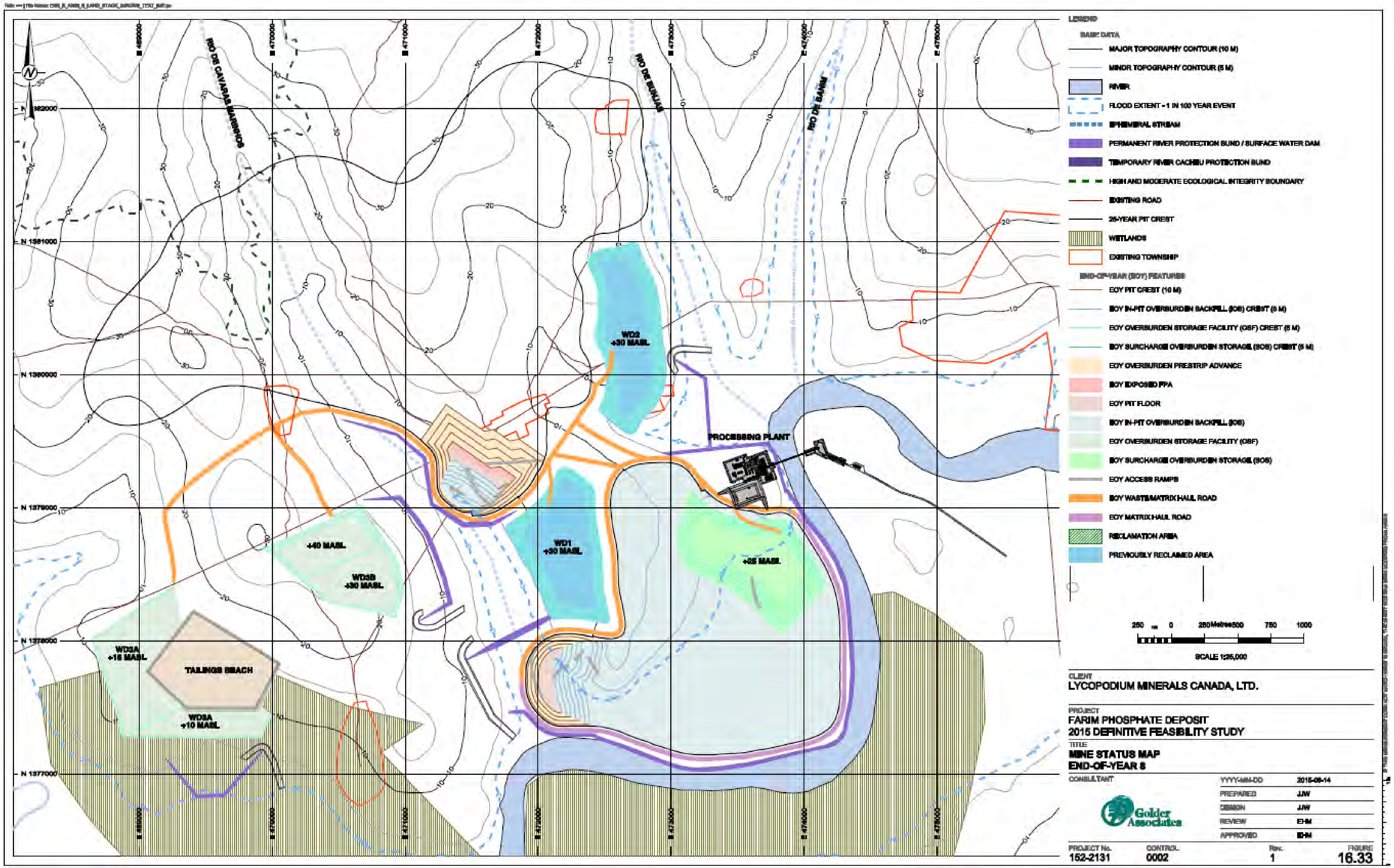


Figure 16-34 Mine Status Map — End-of-Year 10

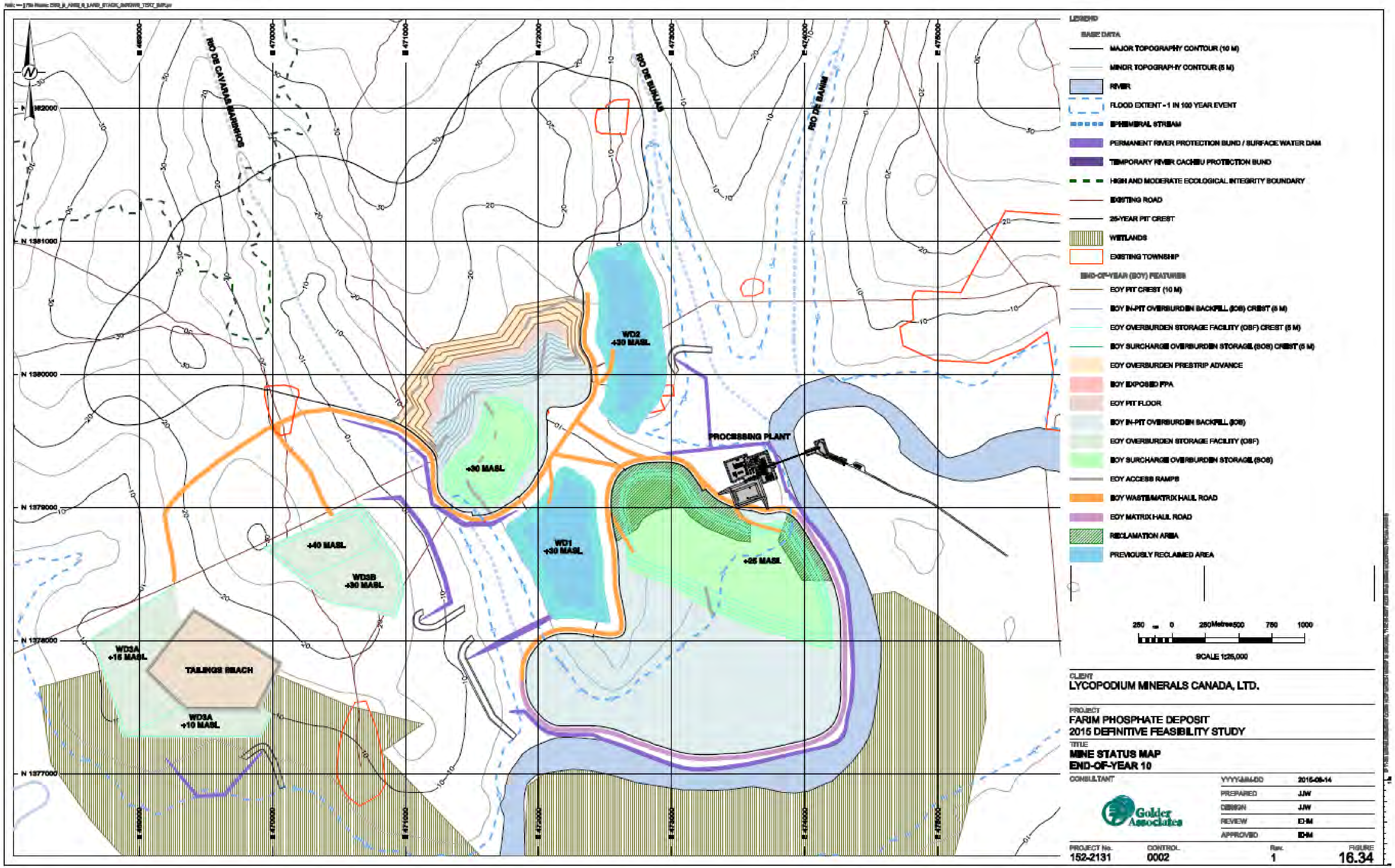


Figure 16-35 Mine Status Map — End-of-Year 15

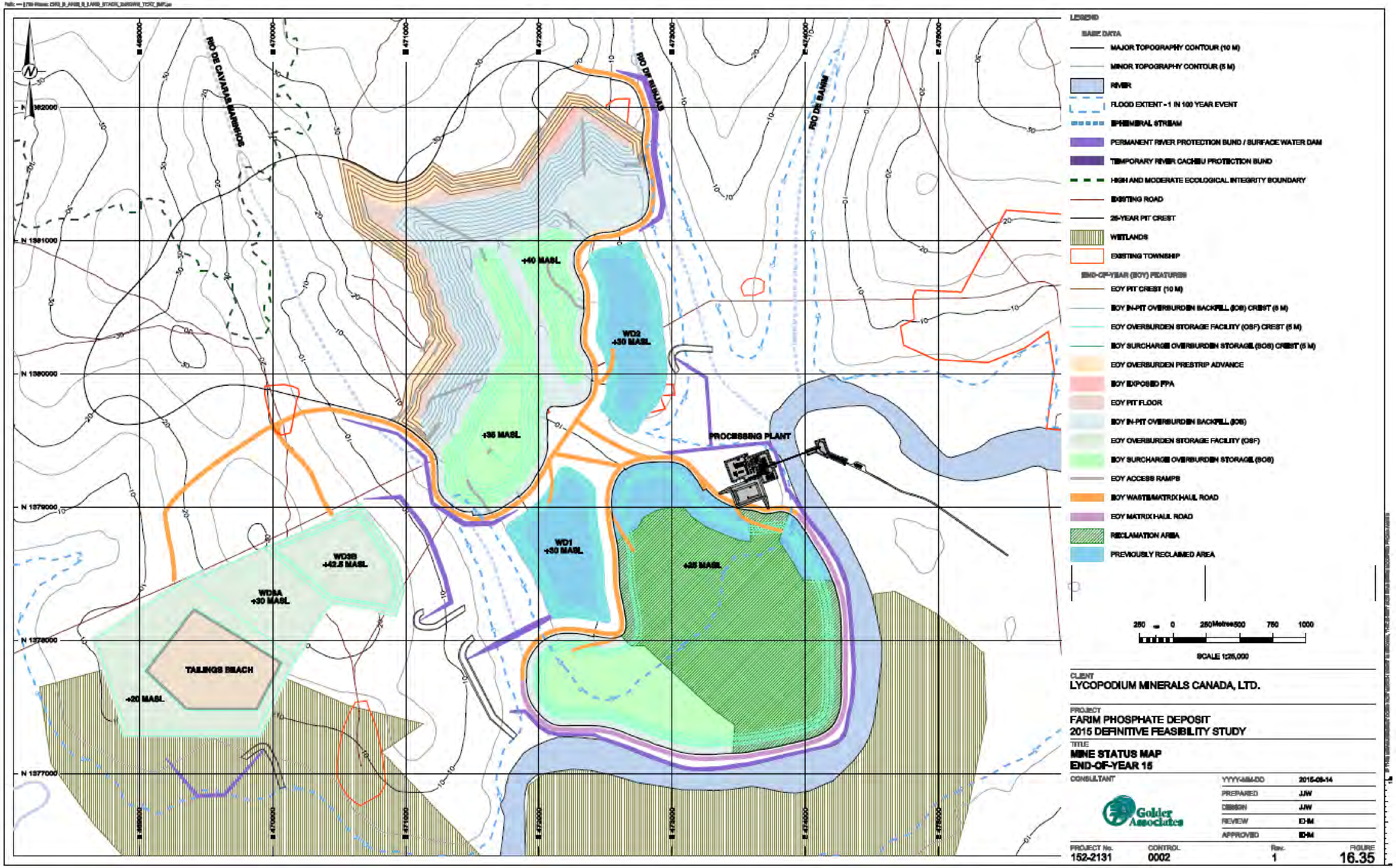


Figure 16-36 Mine Status Map — End-of-Year 20

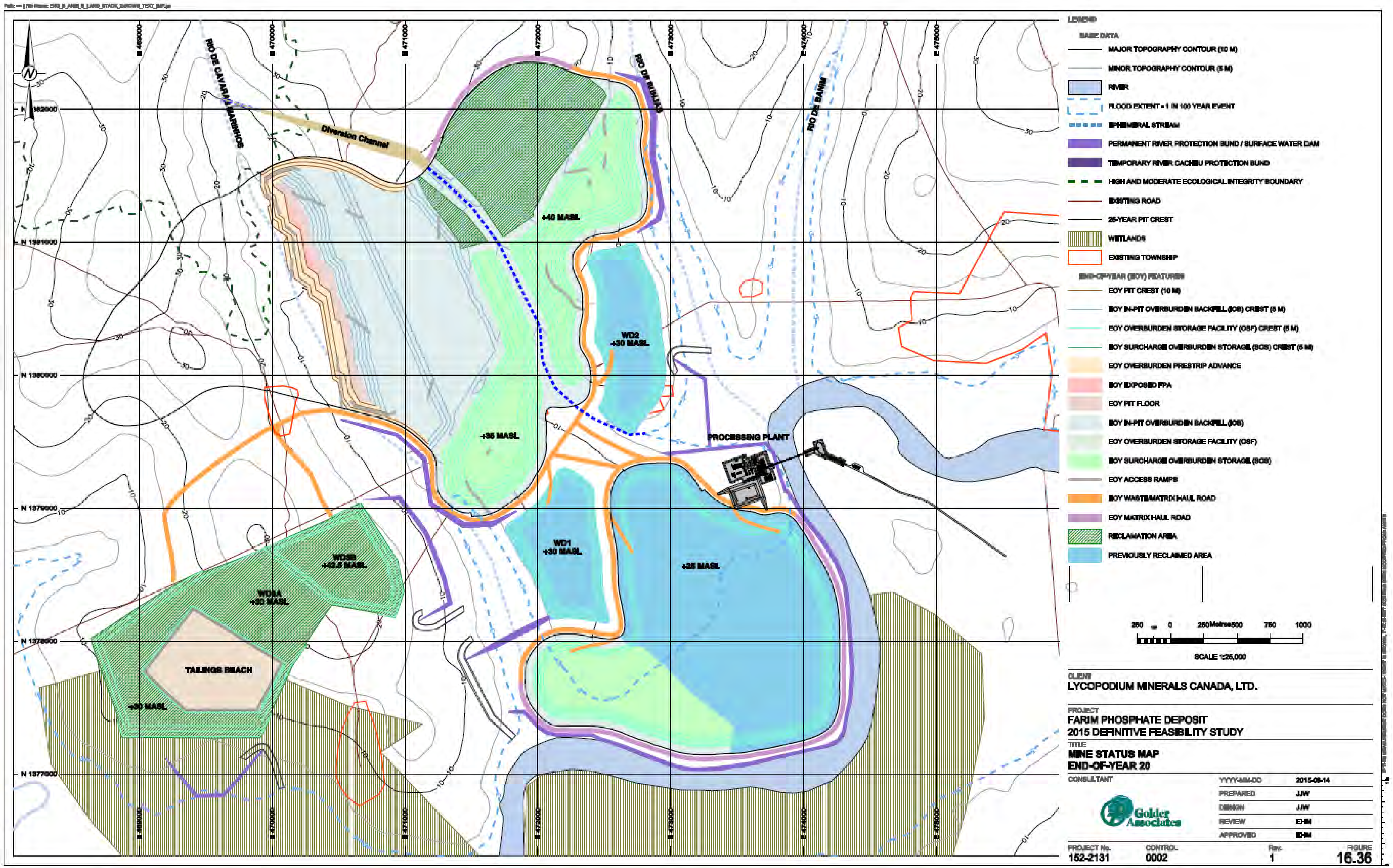


Figure 16-37 Mine Status Map — End-of-Year 26

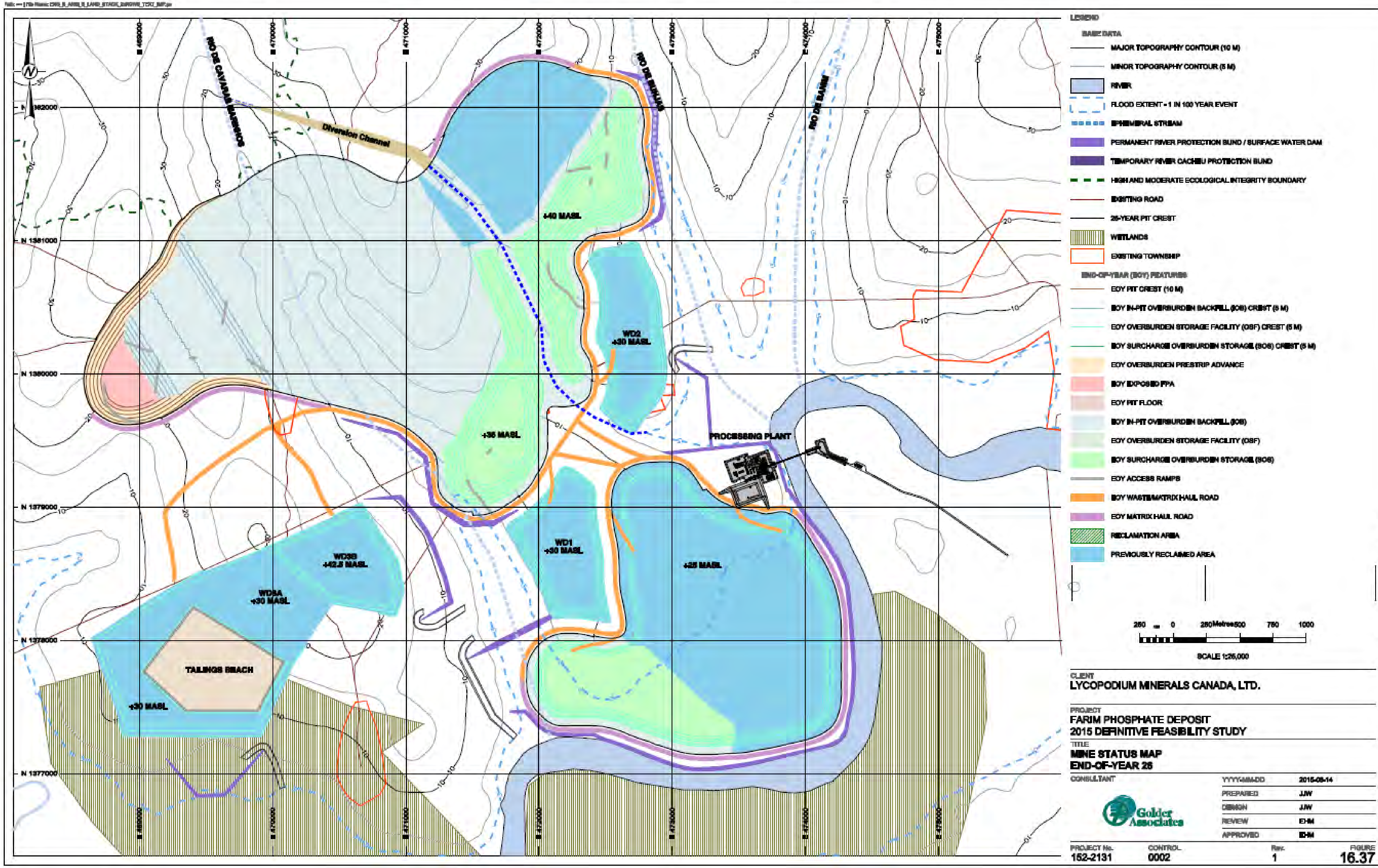
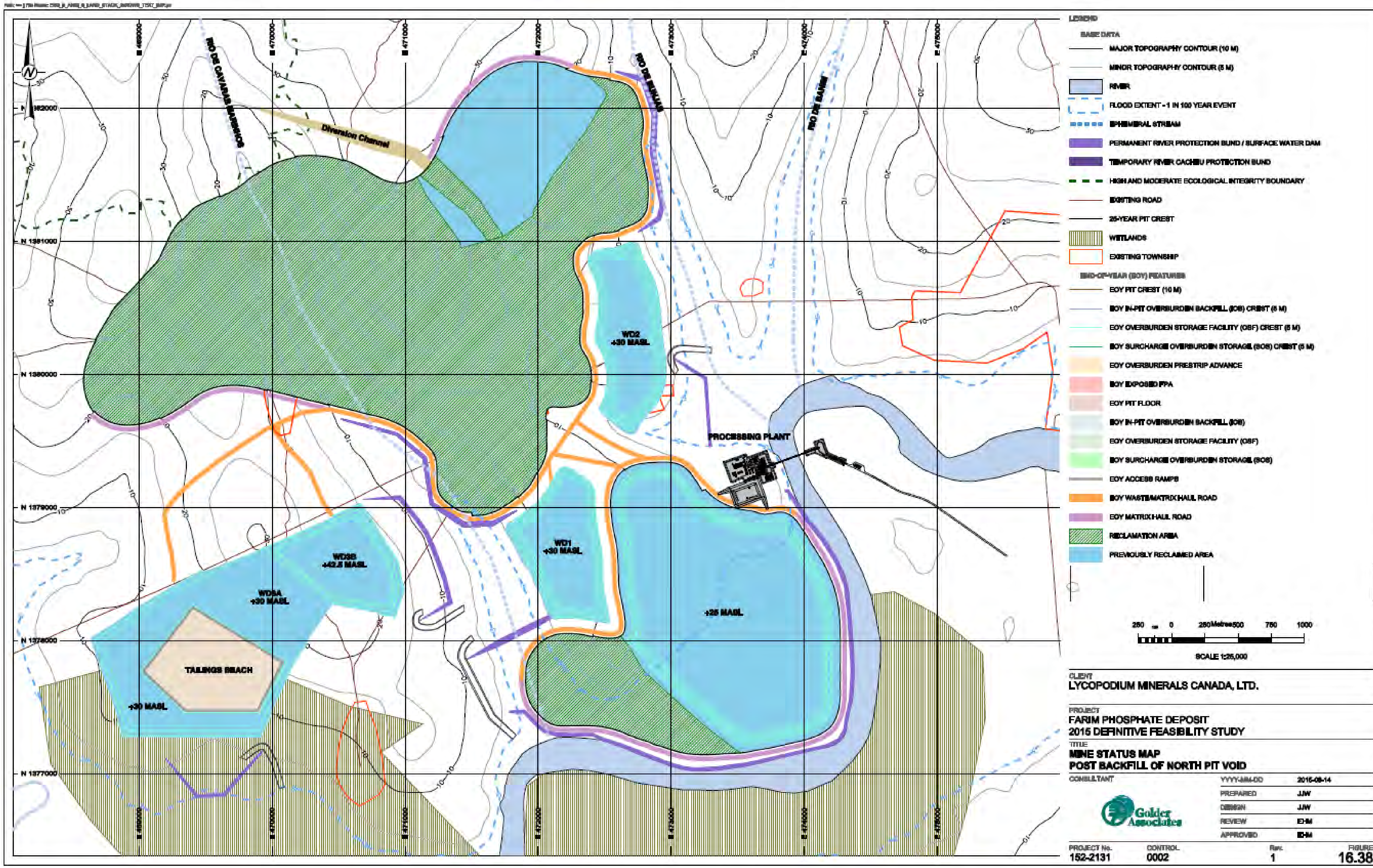


Figure 16-38 Mine Status Map — Post Backfill of North Pit Void



16.7.2 Overburden Storage Facilities

Four different types of overburden storage facilities are required to accommodate mine waste: in-pit overburden backfill (IOB), ex-pit waste dump (WD), ex-pit integrated waste landform (IWL), and surcharge overburden storage (SOS). IOB facilities are located within the open pit area (OPA) and are preferred as they help to minimize haul distances and reduce costs. IOB facilities are backfilled to original ground level and have been designed by Golder on an annual basis. SOS facilities are located above original ground level on top of IOB facilities and are the second best option to IOB facilities as they help to limit the area of disturbance outside of the pit. Ex-pit WDs are least desirable as they generally have longer haul distances, higher associated costs, and greater environmental and socio-economic impacts related to the increased area of disturbance.

The IWL, which combines the tailings storage facility (TSF) with a WD, has been designed by Knight-Piésold to accommodate tailings and overburden. The main WD at the IWL (WD3a) forms around the tailings embankment and has an estimated capacity of 23 million loose cubic metres (lcm) when dumped to an elevation of 42.5 mamsl. The northeastern extension of the WD at the IWL (WD3b) has an estimated capacity of 8.6 million lcm when dumped to an elevation of 40 mamsl. Additional information pertaining to the IWL is provided in Section 18.19.

The WDs were designed and volumetrically balanced in Vulcan and Minescape using the facility design criteria specified in Section 16.3. While the facilities will be compacted in lifts in the field, compaction was not accounted for in the overburden mass balances as its effects on overburden swell are not well constrained. Maximum IOB facility volumes were determined for each year by offsetting the pit toe 50 m and building lifts in 5 m increments until the facility crest intersected original topography or the total stack height of the IOB reached 40 m, whichever comes first. This maximum annual stack height of 40 m, which only plays a factor in IOB dump sequence in the North Pit, is based on Golder’s experience in similar projects, and Golder considers it to be a reasonable operational constraint for the Project. A sectional drawing demonstrating this concept has been provided in Figure 16-39. Available IOB facility volumes by year were calculated as the difference in the cumulative IOB volume for the previous year and the maximum IOB volume at year end. The ex-pit WD and SOS facilities were then designed to accommodate any remaining overburden that could not be fit in the IOB facilities. A generic section of an ex-pit WD design is shown in Figure 16-40.

Figure 16-39 Excavator/Truck Mining Methodology — Profile View

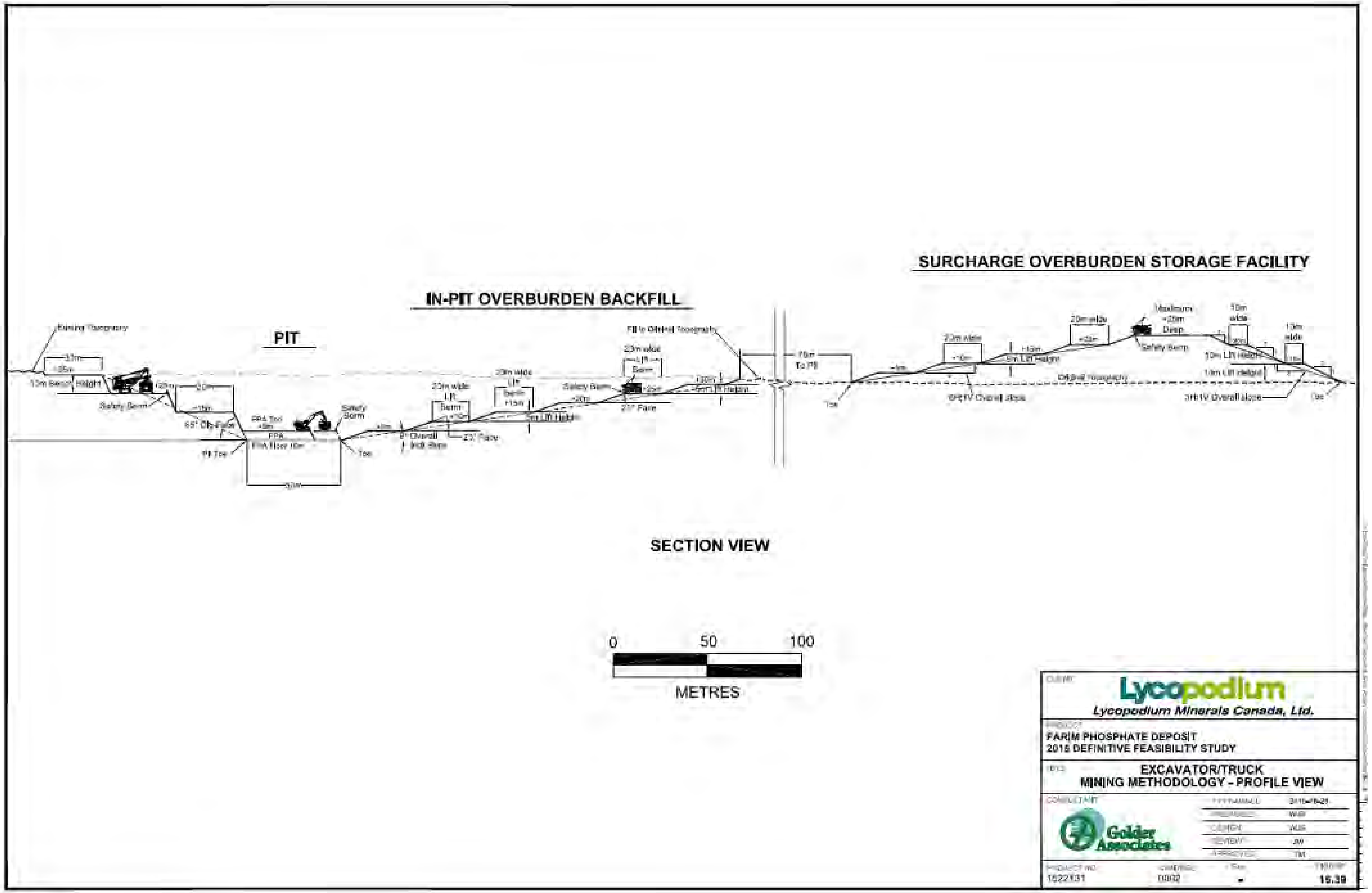
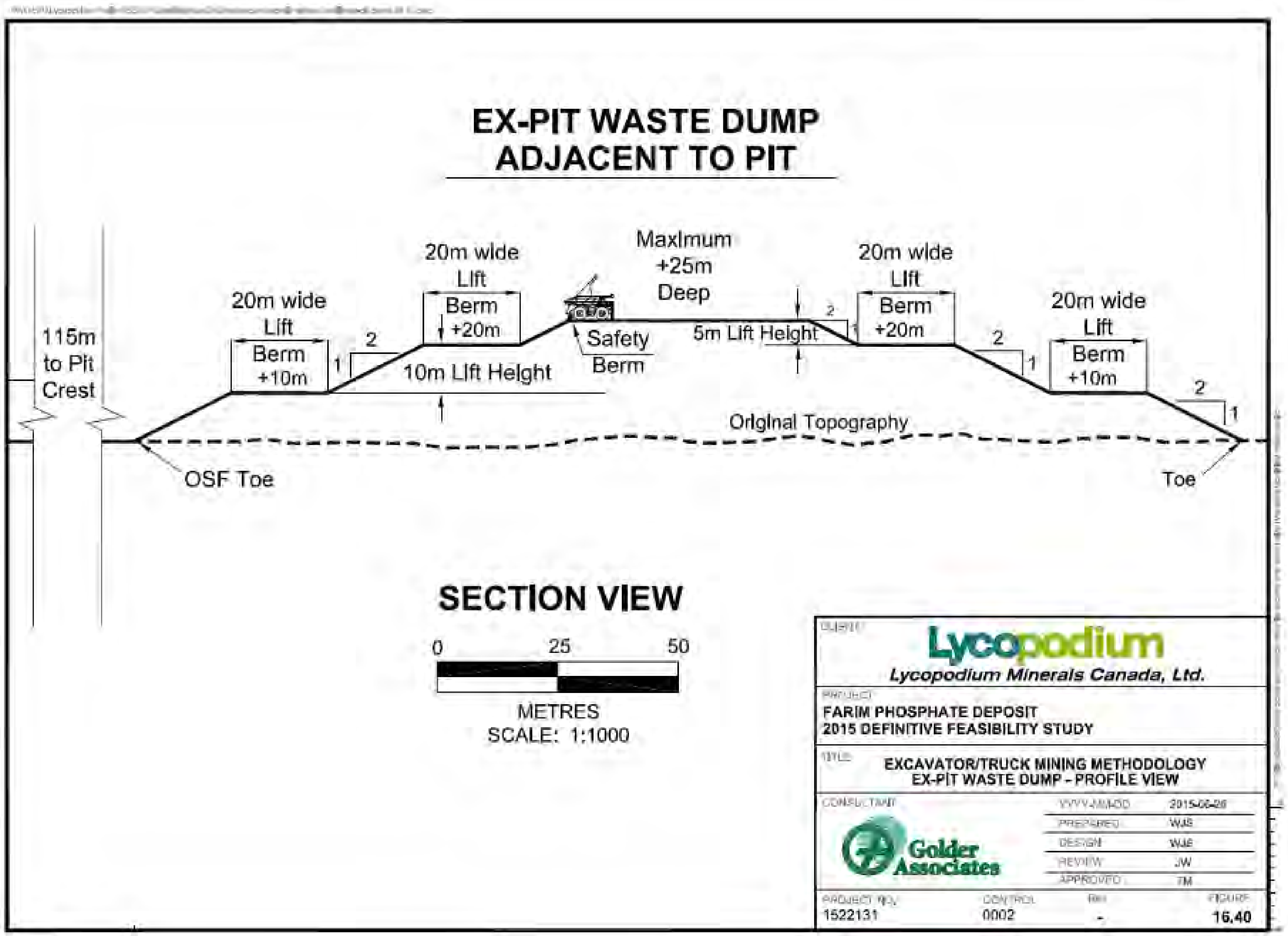


Figure 16-40 Excavator/Truck Mining Methodology Ex-Pit Waste Dump – Profile View



A zone of potentially leachable overburden exists 5 m to 10 m above the roof of the FPA seam. This overburden must be deposited to IOB and covered with inert overburden, or it must be deposited to WD3b at the IWL, where surface water runoff can be treated before release back into the environment. Only inert overburden will be placed in WD1, WD2, WD3a and SOS facilities to minimize the number of locations where water can become contaminated from exposure to potentially leachable overburden. For the purposes of this exercise, Golder assumed that all overburden 7.5 m above the FPA seam is potentially leachable and calculated the annual volumes recovered from the pit. The annual volumes of potentially leachable overburden are presented in Table 16-11.

The results of the volumetric balance provided in Table 16-11 and Figure 16-41 show that approximately 78 percent of all overburden stripped is deposited to IOB, 8 percent is deposited ex-pit, and 14 percent is deposited to an SOS facility. Overburden deposited ex-pit can refer to an ex-pit WD, the tailings embankment located at the IWL, or the Cacheu River protection bund along the perimeter of the South Pit. Given the relatively small volume any additional bunds may represent in comparison to IOB, SOS, and WD, Golder only accounted for the Cacheu River protection bund in the volumetric balance.

It should be noted that WD3a was designed by Knight-Piésold to have a maximum capacity of 22 million lcm of waste, but Golder only utilized 13.4 million lcm in an effort to minimize effective haul distances and cycle times. This extra capacity in WD3a provides flexibility to the mine plan should there be a need to alter the design of another WD, SOS, or IOB advance without altering the footprint of the IWL.

In Years 0 through 2 of the mine life approximately 26.3 million lcm of overburden must be deposited outside of the South Pit; this equates to 65 percent of all overburden stripped in those years. Some of this material will be required to build the first stage of the tailings embankment at the IWL and River Cacheu protection bund, but Golder estimates these facilities only require 2.9 million lcm for construction. Golder initially allocated all other ex-pit overburden to the WD located at the IWL, which has a one-way haul distance of approximately 7.6 km and haul cycle time of 26 minutes from the overburden stripping operations in Years 0 through 2. The combination of haul distance, haul cycle time, and volume of material required 23 CAT 777D haul trucks in Year 2. This number plunged to 15 haul trucks in Year 3 due to increased backfilling opportunities.

To alleviate haul cycle times and minimize haul truck requirements early in the mine life to the extent possible, Golder designed two WDs with a close proximity to the South Pit. The first WD, henceforth referred to as WD1, is located along the western extent of the South Pit between the pit and the western ephemeral stream (Rio de Cavaras Marinhos). The second WD is located north of the South Pit between the eastern boundary of the North Pit and eastern ephemeral stream (Rio de Bunjas); this WD is henceforth referred to as the WD2. The locations of these WDs are provided in Figure 16-43 in the coming pages. Both of these WDs were designed to a maximum 25 m above original ground level (aGL) at a 1V:4H overall slope. The overall slope for these two WDs as measured from the eventual pit toe to the crest of the WD is approximately 1V:5H. WD1 and WD2 have approximate capacities of 8.5 million lcm and 9.0 million lcm, respectively, for inert overburden.

The addition of WD1 and WD2 in Years 1 and 2 of production helped reduce the number of required haul trucks to 16, which equates to deferral in capital of approximately USD $8 million in haul trucks. Actual reduction in cash due to shorter hauls was roughly USD $2 million. Please note that these estimates of savings are exclusive of any surface water management or dewatering costs incurred by the addition of these WDs.

By Year 3 the pit geometry increases backfilling opportunities, and from Years 4 through 7 most overburden is deposited to IOB, though some deposition to SOS is required.

As mining transitions to the North Pit, increasing amounts of overburden are allocated to SOS from Years 8 through 14 as the geometry of the mining advance is not as conducive to backfilling. All SOS facilities have been designed at an overall slope of 1V:6H with a maximum height of 25 m aGL to mitigate geotechnical risk and reduce visual impact, and maintain a minimum 50 m offset from the crest of the IOB advance and the 25-year pit extents. Additionally, the SOS facilities within the North Pit have been offset from the limits of the diversion channel used to reroute the western ephemeral stream. While some of the North Pit overburden is deposited to the SOS within the North Pit, much of the North Pit overburden must be sent to the SOS in the South Pit due to these design constraints. A total of 62.5 million lcm of overburden are stored in SOS between Years 8 and 14, and 81 percent of this is allocated to the South Pit SOS.

In Years 14 through 17 some ex-pit storage of overburden is required as the designed IOB and SOS are at maximum capacity. This waste will be allocated to WD3a and WD3b located at the IWL. After the northeast section of the North Pit is mined out in Year 17, a large volume becomes available for in-pit backfilling, and all overburden stripped thereafter is allocated to IOB.

Year 20 represents a critical juncture in the mine plan as all overburden stripped must be used to construct an IOB to original ground level sufficient to accommodate the diversion channel required to reroute the western ephemeral stream. Failure to do so will necessitate the use of different management methods to divert the large volumes of water from the heavy rainy season away from the pit.

At the end of mining a large void remains in the North Pit. This void, which is approximately 35.4 million cubic metres (m^3) in size, must be backfilled with overburden material from SOS. This requires the complete re-handle of both SOS at the North Pit and the re-handle of approximately 15.5 million lcm of waste from the South Pit SOS. The final arrangement of the mine and all associated WD and SOS facilities after the North Pit has been completely backfilled is provided as Figure 16-38 in Section 16.7.1.

Table 16-11 Waste Dump Volumetric Balance

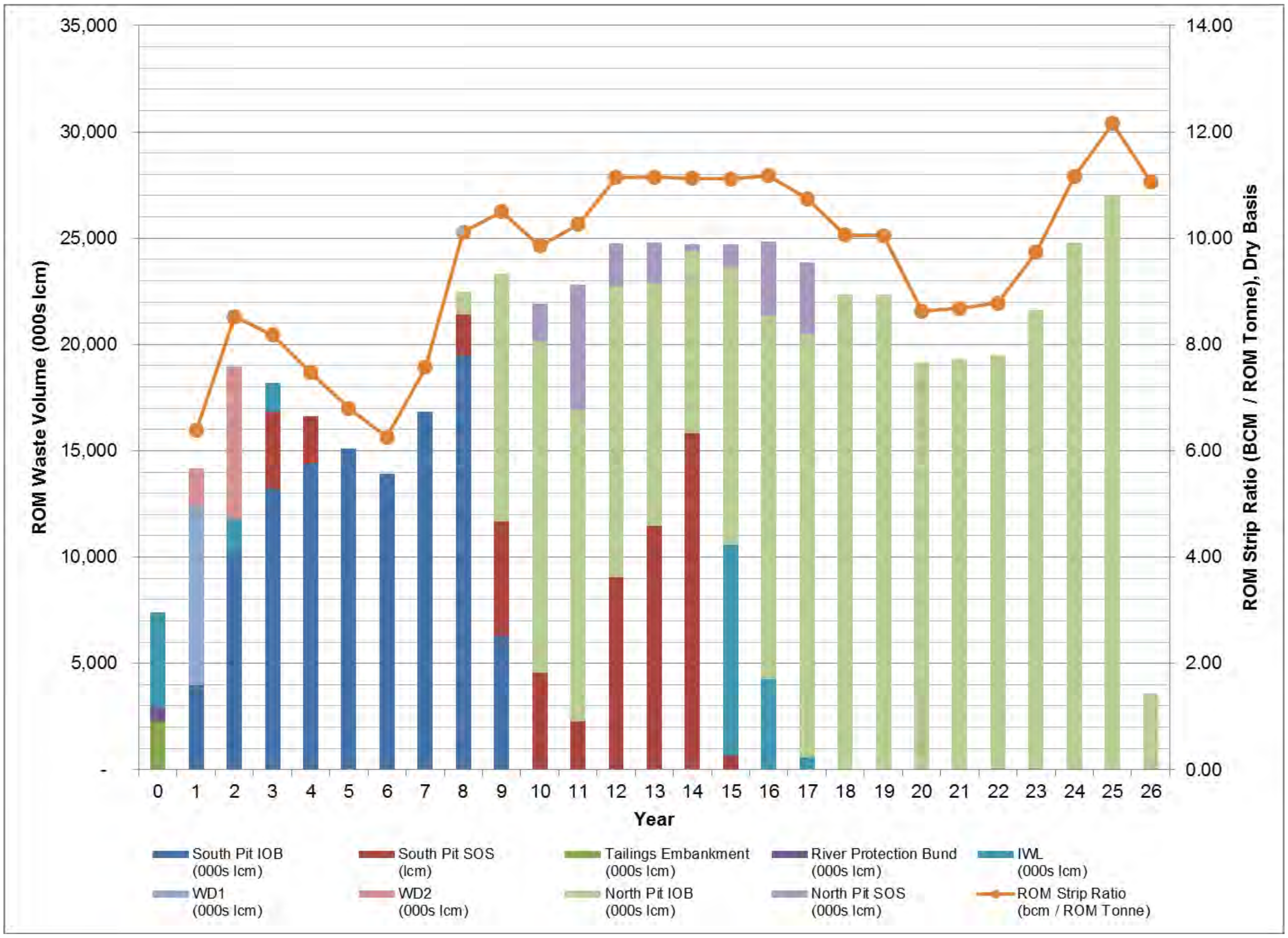
Production Year	Annual Overburden Stripping (000s bcm)	Annual Overburden Stripping (000s lcm)	Annual Potential Overburden Leach Volume [1] (000s bcm)	Annual Potential Overburden Leach Volume [1] (000s lcm)	Potentially Leachable Overburden Deposited to IOB (000s lcm)	Total Overburden Deposited to In-Pit Overburden Backfill [IOB] (000s lcm)	Overburden Deposited to Surcharge Overburden Stockpile [SOS] (000s lcm)	Potentially Leachable Overburden Deposited to IWL (000s lcm)	Total Overburden Deposited to Integrated Waste Landform [IWL] (000s lcm)	Total Overburden Deposited Ex-Pit [2] (000s lcm)	% Overburden Deposited to IOB	% Overburden Deposited to SOS	% Overburden Deposited Ex-Pit [1]
0	5,818	7,389	537	682	-	-	-	682	4,477	7,389	0%	0%	100%
1	11,172	14,188	2,079	2,640	2,640	3,956	-	-	-	10,232	28%	0%	72%
2	14,922	18,950	2,682	3,406	2,923	10,293	-	483	1,449	8,658	54%	0%	46%
3	14,318	18,184	2,834	3,599	3,144	13,199	3,620	455	1,365	1,365	73%	20%	8%
4	13,079	16,610	2,559	3,250	3,250	14,406	2,203	-	-	-	87%	13%	0%
5	11,898	15,110	2,515	3,194	3,194	15,110	-	-	-	-	100%	0%	0%
6	10,947	13,903	2,413	3,065	3,065	13,903	-	-	-	-	100%	0%	0%
7	13,247	16,824	2,591	3,290	3,290	16,824	-	-	-	-	100%	0%	0%
8	17,701	22,481	2,593	3,293	3,293	20,554	1,926	-	-	-	91%	9%	0%
9	18,369	23,328	2,947	3,743	3,743	17,898	5,430	-	-	-	77%	23%	0%
10	17,252	21,911	2,551	3,239	3,239	15,552	6,359	-	-	-	71%	29%	0%
11	17,959	22,807	2,859	3,632	3,632	14,666	8,141	-	-	-	64%	36%	0%
12	19,494	24,757	2,714	3,447	3,447	13,665	11,092	-	-	-	55%	45%	0%
13	19,512	24,781	2,523	3,204	3,204	11,431	13,349	-	-	-	46%	54%	0%
14	19,472	24,729	2,455	3,118	3,118	8,571	16,158	-	-	-	35%	65%	0%
15	19,454	24,707	2,493	3,166	-	13,033	1,782	3,166	9,892	9,892	53%	7%	40%
16	19,555	24,835	2,195	2,788	1,369	17,097	3,480	1,419	4,258	4,258	69%	14%	17%
17	18,806	23,884	1,836	2,331	2,137	19,883	3,417	195	584	584	83%	14%	2%
18	17,605	22,359	3,324	4,222	4,222	22,359	-	-	-	-	100%	0%	0%
19	17,580	22,326	2,814	3,573	3,573	22,326	-	-	-	-	100%	0%	0%
20	15,099	19,176	2,780	3,531	3,531	19,176	-	-	-	-	100%	0%	0%
21	15,186	19,286	2,989	3,797	3,797	19,286	-	-	-	-	100%	0%	0%
22	15,370	19,520	2,513	3,192	3,192	19,520	-	-	-	-	100%	0%	0%
23	17,040	21,640	2,084	2,647	2,647	21,640	-	-	-	-	100%	0%	0%
24	19,521	24,792	2,222	2,822	2,822	24,792	-	-	-	-	100%	0%	0%
25	21,282	27,028	2,447	3,108	3,108	27,028	-	-	-	-	100%	0%	0%
26	2,841	3,608	340	431	431	3,608	-	-	-	-	100%	0%	0%
Total	**424,498**	**539,112**	**64,890**	**82,411**	**76,011**	**419,776**	**76,958**	**6,400**	**22,025**	**42,378**	**78%**	**14%**	**8%**

Notes:

[1] Assumes all overburden 7.5 m above the roof of the FPA seam is potentially leachable.

[2] Overburden deposited ex-pit refers to the river protection bund, tailings embankment at the IWL, or to an ex-pit WD.

Figure 16-41 Annual Waste Dump Volume Allocations



16.7.3 Haul Road Requirements

Based on the haul road design criteria specified in Figure 16-14 and Figure 16-15, two different types of haul roads will be constructed for mining activities: overburden haul roads that support a 97 t capacity end-dump truck (180 t fully loaded), and matrix haul roads that support a 36 t capacity end-dump truck (approximately 72 t fully loaded).

Heavy traffic to the IWL in Year 0 will require the construction of a permanent overburden haul road approximately 6.6 km long leading from the pit to the IWL. This haul road will be regularly used from Years 0 through 3 and again in Years 15 through 17 to transport overburden from the pit to the ex-pit IWL or the West WD; it will also be used for reclamation purposes in following years. Additional overburden haul roads and ramps will be constructed in-pit in Year 0 for overburden truck access and will progress with the pit face for the LOM.

Matrix haul roads must be incrementally built along the entire southern perimeter of the South Pit adjacent to River Cacheu from Years 1 through 8 to allow for haulage of the matrix to the processing plant. This haul road will also be used for the incremental construction and maintenance of the River Cacheu protection bund. Haul roads will be incrementally built along the northern perimeter of the South Pit from Years 1 through 8 on an as-needed basis to allow for additional haulage of matrix to the plant. Because this haul road will also be used extensively from Years 8 through 15 to haul North Pit overburden to the SOS above the South Pit, Golder suggests that this haul road be built to overburden haul road specifications. This will also allow CAT 777 haul trucks convenient access to maintenance facilities at the processing plant.

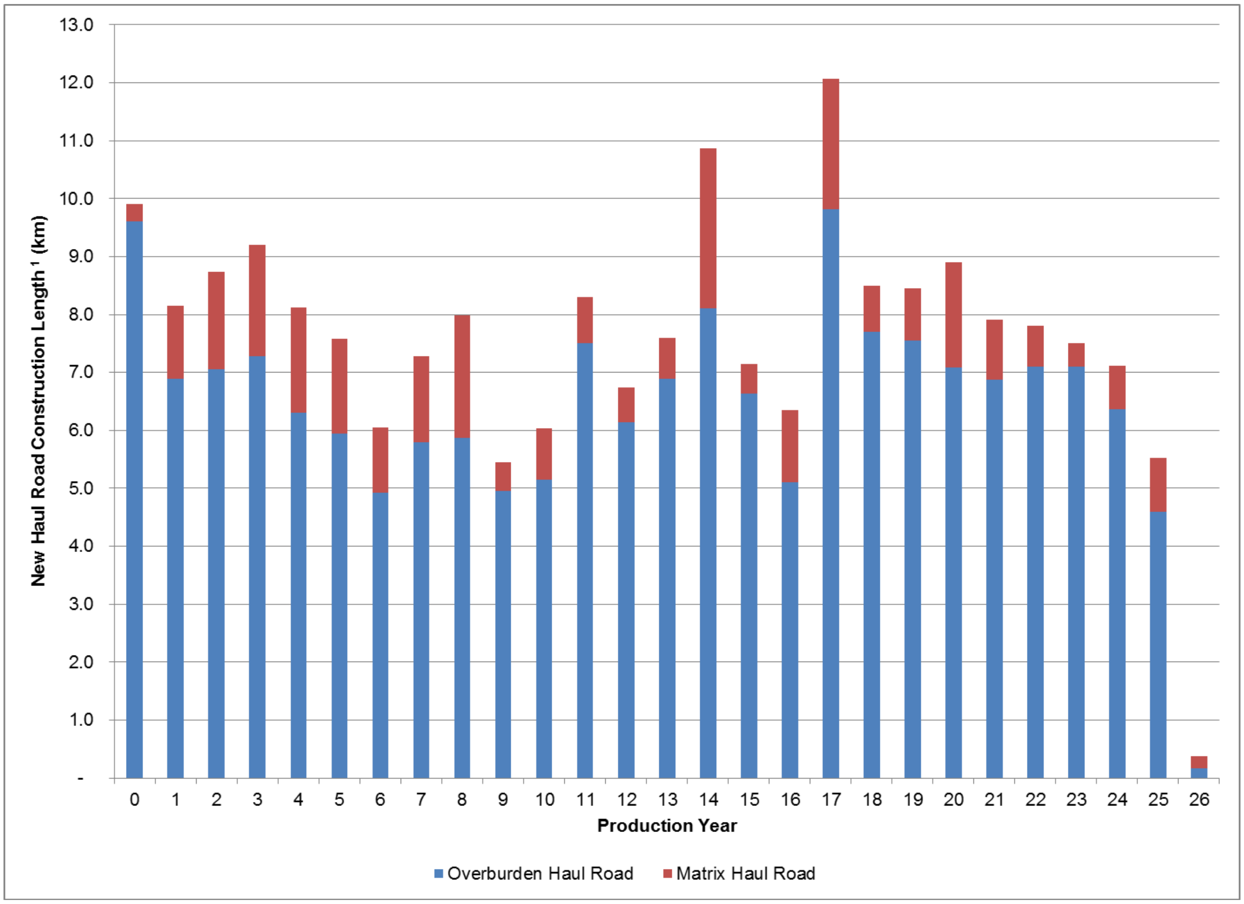
Haul roads providing access to the pit floor will be constructed to provide matrix mining equipment access to the mining face. In-pit backfill ramps/roads and haul roads will be built to overburden haul road specifications, and will progress with the pit face for the LOM. In-pit access ramps at a 10% grade will be built along the in-pit backfill face to provide access to the pit floor for matrix haul trucks. This access ramp will be built to the overburden haul road specifications provided in Figure 16-14 of Section 16.5.11, and will progress with the IOB facility face for the LOM.

Additional ex-pit haul roads will be built along the 25-year pit limit crest on an as-needed basis as the pit progresses. Roads servicing the North Pit will be tied in with existing haul roads wherever possible to minimize construction requirements. In instances where the haul road will be used extensively by both matrix and overburden haul trucks, the haul roads will be built to overburden haul road specifications. The progression of the mine will require additional construction of haul roads along the entire eastern and southern perimeters of the North Pit and a portion of the northern perimeter. The progression of these overburden and matrix haul roads is shown in the end-of-period maps provided.

Due to the heavy rains experienced in Guinea-Bissau during the rainy season, continuous and vigilant oversight of haulage roads will be required to ensure roads are well maintained. A small backhoe, truck, and mobile screen will provide maintenance of the various haul roads as needed. This equipment may also be used to reclaim road rock and potentially reduce the costs associated with rebuilding in-pit roads. However, Golder’s experience with similar phosphate mines with clayey soil conditions indicate that road rock recovery may be minimal and may not provide a meaningful cost savings. For this reason, Golder has assumed no recovery or reuse of road rock for the mining cost estimate.

Approximate yearly haul road construction lengths are shown in Figure 16-42.

Figure 16-42 Total Yearly Haul Road Construction Lengths



Notes:
[1] Total lengths include the in-pit and ex-pit lengths of both matrix and overburden haul roads. This figure is not intended to indicate the amount of new rock needed each year for haul road construction.

16.7.4 Haul Profile Simulations

Pit centroids, IOB centroids, ex-pit WD and IWL centroids, and SOS centroids were approximated for each year using the facility surfaces and representative end-of-period pit surfaces, when available. When end-of-period pit surfaces were not available to approximate centroids, centroids were developed from mining sequence using the weight-averaged centroids of the scheduling blocks.

Haul profile strings from the yearly pit centroids to the corresponding IOB, ex-pit WD and IWL, and SOS centroids were created to represent the haul route. A maximum grade of 10% was used based upon the truck specifications. IOB hauls were developed by drawing a line string along an excavation bench on the pit face, then back-hauled along the nearest in-pit overburden facility face to minimize elevation changes and to reduce costs.

SOS hauls were developed by taking the shortest path possible from the pit centroid to the nearest ex-pit overburden haul road using a network of ramps to the pit crest, then followed an ex-pit haul road to the appropriate SOS centroid. Ex-pit WD and IWL hauls were similarly developed using the same network of in-pit ramps from the pit centroid to the nearest overburden haul road to be taken to the appropriate ex-pit WD or IWL.

Matrix haul profiles were created using profile strings from the pit centroid to the crest of the in-pit facility by ramping up the in-pit overburden facility face at a maximum 10% grade. The haul profile string then followed the crest of the IOB to the nearest haul road leading to the plant. The in-pit overburden facility face ramp used to access the pit floor progressed along with the pit and backfill advances.

The haul profile strings were allocated into XYZ text files, processed in a Microsoft Access database to check for errors, and imported into Caterpillar's Fleet Production and Cost Analysis (FPC) software to estimate overburden and matrix haul times. Maximum grades of ±10% were assumed based on equipment specifications. The remaining assumptions used in FPC to develop haul times are listed in Table 16-12 below. The results of the FPC haulage simulations for matrix and overburden are provided in Table 16-13 on the following page, and their effects on haul truck fleet requirements are detailed in Section 16.9.

Table 16-12 FPC Haul Simulation Assumptions

Grade	Maximum Speed (kph)
10% to -5%	40
-5% to -10%	20
Sharp Turns	10
From - To	**Rolling Resistance**
0 to 25 m	6%
25 to 125 m	5%
125 to 225 m	4%
225 m to Last 75 m	3%
Last 75 m to Last 25 m	5%
Last 25 m	6%

Table 16-13 Results of the FPC Haulage Simulations

Production Year	Effective Matrix Haul Distance (km)	Matrix Haul Cycle Time (min)	Effective Overburden Haul Distance (km)	Overburden Haul Cycle Time[1] (min)
0	-	-	6.24	21.29
1	1.61	7.83	2.60	10.00
2	1.39	6.82	2.46	9.18
3	1.94	8.74	2.42	8.88
4	2.07	9.19	2.75	9.71
5	2.33	10.14	1.89	6.97
6	2.58	10.96	1.75	6.66
7	2.96	11.99	1.51	5.79
8	3.29	13.23	2.25	8.39
9	2.50	10.40	2.29	8.73
10	3.10	12.75	2.12	8.37
11	2.46	10.49	1.72	7.32
12	2.87	12.31	2.63	10.21
13	3.17	12.95	3.47	12.97
14	3.32	13.63	4.42	15.83
15	3.04	12.27	4.73	16.93
16	3.54	15.25	2.89	10.93
17	3.87	15.66	1.76	7.23
18	3.19	12.84	2.73	10.45
19	3.45	13.59	2.50	9.53
20	3.92	15.21	2.37	8.93
21	4.18	16.15	2.18	8.41
22	4.34	17.55	1.81	7.31
23	4.58	18.73	1.79	7.36
24	5.88	21.97	1.79	7.62
25	5.98	22.80	1.54	6.70
26 [2]	6.18	23.47	3.72	13.31
27 [2]	-	-	4.92	16.74

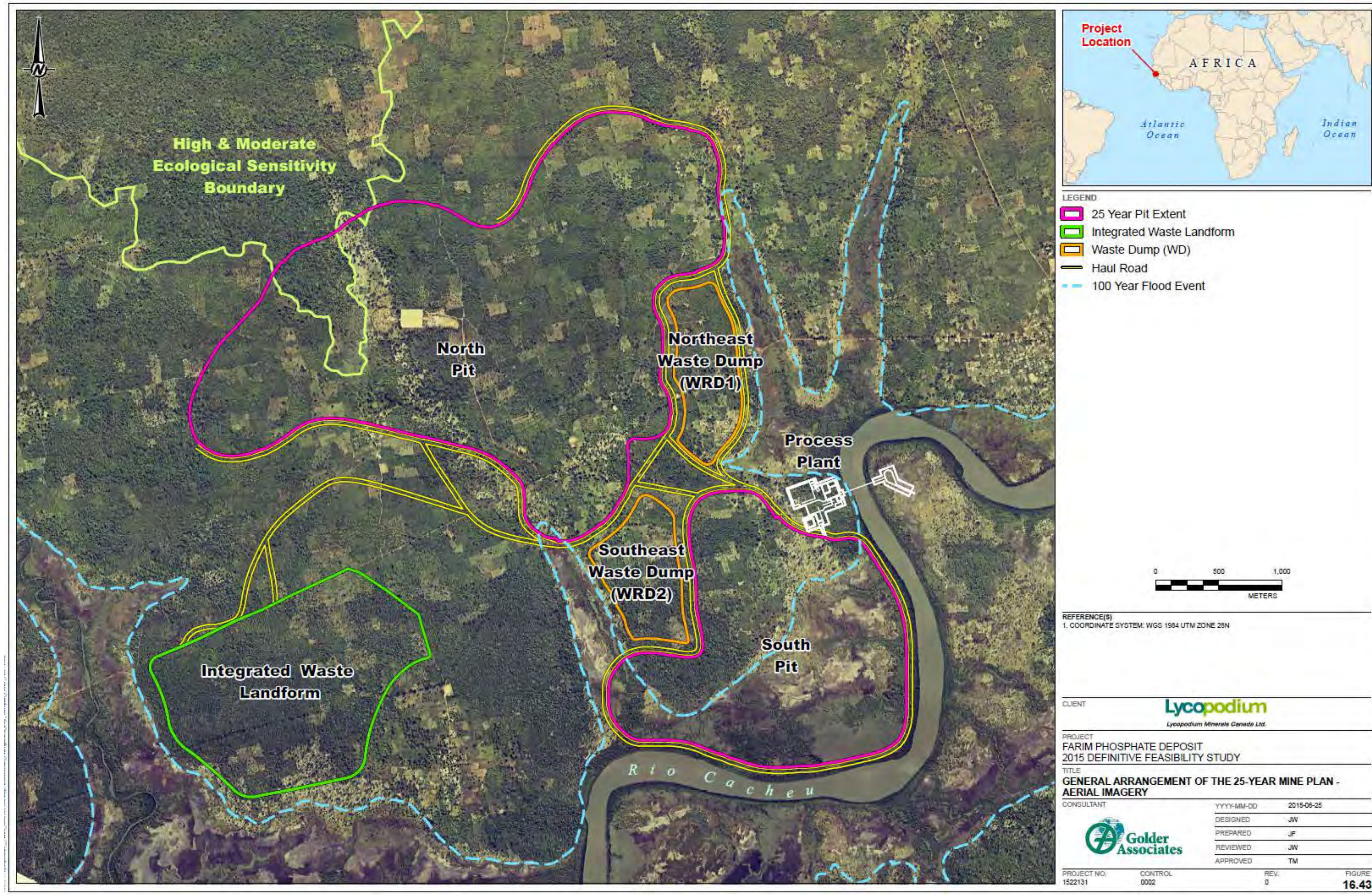
Notes:
[1] Effective Overburden Haul Distances and Haul Cycle Times have been weight-averaged to account for haulage to IOB, SOS, and ex-pit WD and IWL.
[2] Years 26 and 27 include effective haulage distances and cycle times to rehandle approximately 35.4M lcm of overburden from SOS to backfill the void left behind in the North Pit at the end of mining.

16.7.5 ROM Stockpile

A 175,000 t ROM (dry basis) stockpile area was designed to provide phosphate matrix storage near the plant ROM Bin. This stockpile capacity is necessary to ensure continuous plant feed operation during unscheduled downtime of mine production if weather, equipment availability, pit water issues, and other unforeseen conditions occur. The stockpile can also be used to blend ROM matrix as required to meet production quality specifications.

The ROM Bin is designed for direct plant feed for the mine haul trucks. Haul trucks directly feeding the plant can access the ROM Bin via a ramp from ground level to the top of the hopper. In the event trucks cannot directly feed the plant, the matrix will be sent to the stockpile. Mining costs include cost of reclaiming FPA from the ROM stockpile and loading into the plant feed hopper. Figure 16-43, shows an overall mine plan general arrangement of the project, with pits, overburden and storage facilities, ex-pit haul roads, and general facility locations underlain with an aerial photograph of the project area.

Figure 16-43 General Arrangement of the 25-Year Mine Plan Pit – Aerial Imagery



16.8 Major Equipment Requirements

The equipment selection for the Project was dependent on a variety of factors, including annual material movement requirements, bench height, pit configuration and number of mining faces, and the required selectivity of the mining equipment in overburden and matrix. Based on these conditions, 5 m^3 bucket-class hydraulic backhoes were selected as the primary loading fleet for matrix. These machines are large enough to produce the annual tonnages required and are able to efficiently load the 36 t class of trucks selected for the Project. A 12.2 m^3 bucket-class wheel loader was included to feed matrix into the ROM Bin at the stockpile and as an alternative matrix loading machine.

Primary overburden stripping will be performed with 12.2 m^3 bucket-class front end loaders (FELs) matched with 97 t haul trucks. These shovels were assigned to excavate full bench height stripping.

Table 16-14 below lists the equipment by class and models by manufacturer.

Table 16-14 Summary of Available Equipment Models

Equipment Type	Size Class	Applicable Models
Wheel loader	12.2-m^3 bucket	Caterpillar 992K, Komatsu WA900-3 & Liebherr L580
Backhoe	5.0-m^3 bucket	Caterpillar 374DL & Komatsu PC1250
Haul truck	97-tonne payload	Caterpillar 777G & Komatsu HD 785
	36-tonne payload	Caterpillar 770 & Komatsu HD 325
Water truck	34-liter tank capacity	Caterpillar 770 & Komatsu HD325
Bulldozer	405 hp	Caterpillar D9R & Komatsu D275AX
Grader	297 hp	Caterpillar 16M
Compactor	147 hp	Caterpillar CS-56

A typical excavator operating configuration for the Project is depicted in Section 16.1.3, Figure 16-3. The large (12.2 m^3) wheel loaders are used to efficiently expose matrix leaving a temporary face angle of approximately 65°. Dozers in the 405 HP (horsepower) class are used to prepare the working surface and to create access to the work area. They also provide support for the loader at mining faces. Overburden haulage is accomplished with a fleet of 97 t capacity end-dump trucks.

Equipment productivity calculations are based on mining conditions, equipment capacity, availability, and utilization with non-productive time being a key factor in equipment utilization.

The 12.2 m^3 wheel loader can load these trucks with overburden in five passes. Matrix is exposed and mined with the 5 m^3 backhoes, and the matrix is hauled to the ROM stockpile using 36 t capacity end-dump trucks; the backhoes can fill the 36 t trucks in six passes. Matrix was scheduled on a dry basis which is reflected in the five passes shown in Table 16-16; the sixth pass accounts for the estimated 20% moisture content in the ROM matrix. A relatively long 60 second load cycle time was used to ensure overall loading time would be accounted for.

Availability and utilization factors, as shown in Table 16-15 were applied to calculate scheduled hours, operating hours, and number of units required.

The rates outlined in Table 16-16 reflect effective productivities given estimated equipment parameters (e.g., material swell factors, material densities, bucket fill factors, cycle times, and mechanical availabilities), machine usage, truck saturation, and loading configurations. Truck saturation factors (i.e., the percentage of time that a truck is available for loading at the excavator or wheel loader) were estimated to be in the range of 87% to 97% for the various haulage applications. Mechanical availability is a measure of time that a piece of equipment is physically (mechanically) capable of operating. Mechanical availability is a function of the intensity of equipment usage and machine application. Additional de-rating factors were applied to account for weather delays during the rainy season.

Equipment availabilities, as outlined in Table 16-15 and utilized in this Study, reflect Golder's experience, engineering estimates, and file data. Estimated availabilities are intended to reflect average levels of mechanical availability over the effective life of a particular piece of equipment for the level of utilization stipulated by the respective production scenarios.

As previously indicated, equipment productivities are affected by the mined material, operating conditions, and the mining application. Estimated excavator production rates for different mining applications are summarized in Table 16-16.

The bucket fill factors in Table 16-16 reflect the effectiveness of shovel and backhoe bucket filling. The fill factor is a function of the characteristics of the excavated material, machine application, and operator skill, and is expressed as a percentage of the rated (heaped) bucket capacity.

The swell factor is defined as the adjustment used to de-rate rated bucket capacity in loose cubic metres to an equivalent capacity in bank cubic metres for a given percent material swell. Based on available data, swell factors of 27% and 12% were assigned for overburden material and matrix, respectively.

Truck fleet sizes and other major equipment requirements for this Feasibility Study are summarized in Table 16-17.

Support equipment for the operations included 405 HP bulldozers assigned to the wheel loader to perform pit clean up, prepare benches for excavators, and other support activities at mining faces. It was also assigned for WD maintenance and final grading operations. A small backhoe (2.1-m^3 bucket) was assigned to load rock material for road construction from a mobile rock screen plant or on-site aggregate loading point into a fleet of 36 t payload end-dump trucks. Compactors and scrapers were used primarily for road construction and maintenance as well as WD maintenance.

Graders, water trucks, cranes, forklifts, backhoe loaders and other services vehicles were scheduled as required to support the mining operation.

Table 16-15 Summary of Equipment Delays and Performance Factors – Page 1

DELAY	DELAY & IDLE TIME (minutes) PER SHIFT														
	Shovel		Backhoe		Wheel Loader			Haul Truck		Water Truck	Scraper	Drill	Dozer		
	Overburden	Matrix	Overburden	Matrix	Overburden	Matrix	Stockpile	Overburden	Matrix				Overburden	Matrix	Support
OPERATING DELAYS ("D")	**115.0**	**125.0**	**110.0**	**120.0**	**65.0**	**85.0**	**55.0**	**60.0**	**70.0**	**42.5**	**43.5**	**35.0**	**35.0**	**35.0**	**35.0**
Fuel & Lube ("F&L")	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0
Walking / Moving	30.0	30.0	30.0	30.0	10.0	15.0	10.0	5.0	5.0	5.0	10.0	15.0	10.0	10.0	10.0
Waiting On Trucks ("WOT")	30.0	30.0	25.0	25.0	10.0	15.0	-	-	-	-	-	-	-	-	-
Waiting On Other Equipment	10.0	10.0	10.0	10.0	5.0	5.0	5.0	5.0	5.0	15.0	15.0	5.0	5.0	5.0	5.0
Queuing	-	-	-	-	-	-	-	10.0	10.0	-	-	-	-	-	-
Misc. / Other / De-rating Factor for Rainy Season	30.0	40.0	30.0	40.0	25.0	35.0	25.0	25.0	35.0	7.5	3.5	-	5.0	5.0	5.0
IDLE TIME ("I")	**75.0**	**90.0**	**75.0**	**90.0**	**75.0**	**90.0**	**75.0**	**75.0**	**90.0**	**75.0**	**75.0**	**80.0**	**75.0**	**90.0**	**75.0**
Weather ("WTH")	20.0	30.0	20.0	30.0	20.0	30.0	20.0	20.0	30.0	20.0	20.0	5.0	20.0	30.0	20.0
Meal / Break ("M/B")	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	50.0	30.0	30.0	30.0
Shift Change ("SC")	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0
Misc./Other	5.0	10.0	5.0	10.0	5.0	10.0	5.0	5.0	10.0	5.0	5.0	5.0	5.0	10.0	5.0
TOTAL DELAYS & IDLE TIME (minutes)	**190.0**	**215.0**	**185.0**	**210.0**	**140.0**	**175.0**	**130.0**	**135.0**	**160.0**	**117.5**	**118.5**	**115.0**	**110.0**	**125.0**	**110.0**
% Of An 8-Hour Shift for Waste/Support or a 12-Hour Shift for Matrix	39.6%	29.9%	38.5%	29.2%	29.2%	24.3%	27.1%	28.1%	22.2%	24.5%	24.7%	24.0%	22.9%	17.4%	22.9%
Mechanical Availability ("MA")	80.0%	80.0%	80.0%	80.0%	82.0%	82.0%	82.0%	90.0%	90.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Operational Usage ("OU")	50.5%	62.7%	51.8%	63.5%	64.4%	70.4%	67.0%	68.8%	75.3%	69.4%	69.1%	70.1%	71.4%	78.3%	71.4%
Effective Pit Utilization ("EPU")	**40.4%**	**50.1%**	**41.5%**	**50.8%**	**52.8%**	**57.7%**	**54.9%**	**61.9%**	**67.8%**	**55.5%**	**55.3%**	**56.0%**	**57.1%**	**62.6%**	**57.1%**
Working Hours Per 8-hr. Shift for Waste/Support or Per 12-hr. Shift for Matrix ("W")	3.2	6.0	3.3	6.1	4.2	6.9	4.4	5.0	8.1	4.4	4.4	4.5	4.6	7.5	4.6
Consuming Delays Per Shift ("CD")	1.7	1.8	1.6	1.8	0.8	1.2	0.7	0.8	0.9	0.5	0.5	0.3	0.3	0.3	0.3
Total Engine Hours Per Shift ("EH")	4.9	7.9	4.9	7.9	5.1	8.1	5.1	5.7	9.1	4.9	4.9	4.8	4.9	7.9	4.9
Engine Factor ("EF")	1.52	1.30	1.48	1.29	1.20	1.17	1.15	1.15	1.11	1.10	1.11	1.07	1.07	1.04	1.07
Consumption Factor ("CF")	**61.3%**	**65.4%**	**61.3%**	**65.4%**	**63.3%**	**67.4%**	**63.3%**	**71.3%**	**75.4%**	**61.3%**	**61.3%**	**60.2%**	**61.3%**	**65.4%**	**61.3%**
Truck Saturation ("TS")	86.6%	92.3%	88.8%	93.6%	96.2%	96.5%	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a

Notes:

MA = A / S
OU = W / A
EPU = MA x OU
W = S x EPU

EH = W + CD
EF = EH / W
CF = EPU x EF
TS = W / (W + (WOT/60))

A = Available Hours Per Shift = W + D + I
S = Scheduled Hours Per Shift
CD = (D - F&L) + WTH + SC + (0.75 x M/B)
TS = W / (W + (WOT/60))

EH = W + CD
EF = EH / W
CF = EPU x EF

Table 16-15 Summary of Equipment Delays and Performance Factors- Page 2

DELAY	DELAY & IDLE TIME (minutes) PER SHIFT					
	Compactor	**Support Backhoe**	**Motor Grader**	**Service Truck**	**Light Plant**	**Misc. Equipment**
OPERATING DELAYS ("D")	**50.0**	**38.5**	**39.5**	**21.5**	**9.5**	**9.5**
Fuel & Lube ("F&L")	15.0	9.0	15.0	4.5	4.5	4.5
Walking / Moving	15.0	10.0	5.0	5.0	-	-
Waiting On Trucks ("WOT")	-	-	-	-	-	-
Waiting On Other Equipment	10.0	15.0	15.0	10.0	-	-
Queuing	-	-	-	-	-	-
Misc. / Other / De-rating Factor for Rainy Season	10.0	4.5	4.5	2.0	5.0	5.0
IDLE TIME ("I")	**95.0**	**95.0**	**75.0**	**75.0**	**95.0**	**75.0**
Weather ("WTH")	20.0	20.0	20.0	20.0	20.0	20.0
Meal / Break ("M/B")	30.0	30.0	30.0	30.0	30.0	30.0
Shift Change ("SC")	20.0	20.0	20.0	20.0	20.0	20.0
Misc./Other	25.0	25.0	5.0	5.0	25.0	5.0
TOTAL DELAYS & IDLE TIME (minutes)	**145.0**	**133.5**	**114.5**	**96.5**	**104.5**	**84.5**
% Of An 8-Hour Shift for Waste/Support or a 12-Hour Shift for Matrix	30.2%	27.8%	23.9%	20.1%	21.8%	17.6%
Mechanical Availability ("MA")	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Operational Usage ("OU")	62.2%	65.2%	70.2%	74.9%	72.8%	78.0%
Effective Pit Utilization ("EPU")	**49.8%**	**52.2%**	**56.1%**	**59.9%**	**58.2%**	**62.4%**
Working Hours Per 8-hr. Shift for Waste/Support or Per 12-hr. Shift for Matrix ("W")	4.0	4.2	4.5	4.8	4.7	5.0
Consuming Delays Per Shift ("CD")	0.6	0.5	0.4	0.3	0.1	0.1
Total Engine Hours Per Shift ("EH")	4.6	4.7	4.9	5.1	4.7	5.1
Engine Factor ("EF")	1.15	1.12	1.09	1.06	1.02	1.02
Consumption Factor ("CF")	**57.1%**	**58.3%**	**61.3%**	**63.4%**	**59.3%**	**63.4%**
Truck Saturation ("TS")	100.0%	n/a	n/a	n/a	n/a	n/a

Notes:

MA = A / S	EH = W + CD	A = Available Hours Per Shift = W + D + I	EH = W + CD
OU = W / A	EF = EH / W	S = Scheduled Hours Per Shift	EF = EH / W
EPU = MA x OU	CF = EPU x EF	CD = (D - F&L) + WTH + SC + (0.75 x M/B)	CF = EPU x EF
W = S x EPU	TS = W / (W + (WOT/60))	TS = W / (W + (WOT/60))	

Table 16-16 Summary of Available Equipment Models

MACHINE	Truck Fleet Capacity (tonnes)	Eff. Truck Payload Capacity (tonnes)	Number of Passes	Rated Bucket Capacity (bcm) [B1]	Bucket Fill Factor [FF]	Swell Factor [S]	Effective Bucket Capacity (1) (bcm) [EB1]	Effective Bucket Capacity (1) (tonnes) [EB2]	Cycle Time (secs) [CT]	Nominal Shift Schedule	Mech. Avail. (2)	Oper. Usage (3)	Estimated Prod. Rate Per Shift (4)	Estimated Annual Production Capacity (5)
STRIPPING MACHINES														
Caterpillar 992K - Wheel Loader - Waste	90.5	90.8	5	12.2	0.900	0.787	8.6	18.2	48.0	3 x 8	82.0%	64.4%	2,740 bcm	3,000,500 bcm
MATRIX LOADING MACHINES														
Caterpillar 374DL - Backhoe - Matrix	36	29.7	5	5.0	0.950	0.893	4.2	5.9	60.0	1 x 12	80.0%	63.5%	2,175 tonnes	794,000 tonnes
Caterpillar 992K - Wheel Loader - Stockpile	n/a	n/a	n/a	12.2	0.900	0.893	9.8	13.7	120.0	3 x 8	82.0%	67.0%	1,810 tonnes	1,982,000 tonnes
ROCK LOADING MACHINES														
Caterpillar 336DL - Backhoe - Support	36	34.0	9	2.1	0.900	1.000	1.9	3.8	60.0	3 x 8	80.0%	65.2%	475 bcm	520,000 bcm

Notes:
(1) Effective Bucket Capacity In Bank Cubic Metre ("bcm") = EB1 = B x FF x S, Effective Bucket Capacity In Tonnes = EB1 x Material Weight
(2) Mechanical Availability = Avail. Hours / Sched. Hours
(3) Operational Usage = Working Hours / Avail. Hours
(4) Rate At Given Mech. Avail. and 90% to 95% Truck Saturation = EB1 or EB2 x (3600 / CT) x Hours Per Shift x Mech. Avail. x Oper. Utilization
(5) Based On 7 x 3 schedule with 8-hour shifts for Waste and 7 x 1 schedule with 12-hour shift for Matrix

Table 16-17 Summary of Primary Equipment Requirements – Page 1

Description	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
Caterpillar 374DL - Backhoe	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Caterpillar 336DL - Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Caterpillar 992K - Wheel Loader	3	5	6	6	6	5	5	6	7	8	7	7	8	8	8
Caterpillar D9R - Dozer	2	4	5	5	4	4	4	4	6	6	5	6	6	6	6
Caterpillar 777G - End Dump Truck	12	13	17	16	15	11	10	12	19	20	18	18	23	27	31
Caterpillar 770 - End Dump Truck	1	6	6	7	7	7	7	8	8	7	8	7	8	8	9
Caterpillar 16M - Motor Grader	2	2	3	3	3	3	2	3	3	3	3	3	3	3	3
Caterpillar CS-56 - Compactor	3	4	6	6	5	5	4	5	7	7	7	7	7	7	7
Caterpillar 428F - Backhoe Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Caterpillar 770 - Water Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel/Lube Truck	2	3	3	3	3	3	3	3	4	4	4	4	4	4	4
Mechanic's Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	3	3
Pickup Truck	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11
Liebherr LTM 1095 - Mobile Crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
10-tonne Forklift	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Welding Machine	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Light Plant	5	8	10	10	9	9	8	9	12	12	12	12	13	13	13
Screening Plant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Table 16-17 Summary of Primary Equipment Requirements – Page 2

Description	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28
Caterpillar 374DL - Backhoe	3	3	3	3	3	3	3	3	3	3	3	1	0	0
Caterpillar 336DL - Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Caterpillar 992K - Wheel Loader	8	8	8	7	7	6	6	7	7	8	8	8	3	0
Caterpillar D9R - Dozer	6	6	6	5	5	5	5	5	5	6	7	6	3	0
Caterpillar 777G - End Dump Truck	32	24	18	21	20	17	16	15	17	19	20	30	13	0
Caterpillar 770 - End Dump Truck	8	9	10	8	9	9	9	10	10	12	12	1	0	0
Caterpillar 16M - Motor Grader	3	3	3	3	3	3	3	3	3	3	3	3	2	1
Caterpillar CS-56 - Compactor	7	7	7	7	7	6	6	6	7	7	8	8	4	0
Caterpillar 428F - Backhoe Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Caterpillar 770 - Water Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel/Lube Truck	4	4	4	4	4	3	3	3	4	4	4	4	2	1
Mechanic's Truck	3	2	2	2	2	2	2	2	2	2	2	3	2	1
Pickup Truck	11	11	11	11	11	11	11	11	11	11	11	11	11	11
Liebherr LTM 1095 - Mobile Crane	1	1	1	1	1	1	1	1	1	1	1	1	1	0
10-tonne Forklift	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Welding Machine	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Light Plant	13	13	13	12	12	11	11	11	12	13	14	13	6	0
Screening Plant	1	1	1	1	1	1	1	1	1	1	1	1	0	0

16.9 Mine Cost Estimates

As part of the Study, Golder estimated the costs of FPA matrix production and capital requirements associated with the 1.32 Mtpa (product tonnage) Definitive Feasibility Study. Production costs and project capital estimates were developed on an annual basis to reflect mine plan production schedules.

The cost model assumed the pre-production at Year 0 for overburden stripping would be directly performed by the mining operator, GB Minerals using company-owned equipment and company employees, while both the onsite and offsite infrastructures were under construction. Golder estimated the 25-year mine plan for production of 1.75 Mtpa (dry basis) of feed for the process facility. The estimates encompassed all costs associated with all mining, matrix and overburden handling, ROM stockpile processing, and other mine support services. Capital expenditures include all mining equipment costs and haul road development but do not include infrastructure development. The estimate does not include product transport costs to the port. All cost estimates related to processing and other activities after the matrix is placed into the hopper were provided by other parties. In addition, overhead or indirect mine operating costs was not included in this cost model. Golder included ongoing reclamation costs during the mine life including dozer work for backfill pit re-grading and re-vegetation during mining and backfill of the final pit void. However, final mine closure and infrastructure demolition were not included in the mining cost model and were covered by others.

The cost model does not include capital expenditures for the stockpile base or any mine infrastructure, such as maintenance facilities, offices, wash house, worker camps, warehousing and storage, fuel storage and islands, etc.

The cost model reflects zero-based principles for each year of production. The annual mine operating costs are estimated by combining the annual production statistics from the mine plan with the estimated equipment productivities, utilizations, and mine operating schedules.

Two costing alternatives were prepared for consideration. A 100% Equity Case was developed assuming 100% equity ownership of all Project assets and that all equipment would be purchased on a new basis. A Lease Case was developed whereby major mining equipment was obtained through a five year lease during which time the equipment was paid off and ownership was transferred to the mine. For the purposes of this Study, Golder addressed and estimated the following cost components:

- Direct mine operating costs - labour, materials, and supply;
- Indirect mine operating costs – not capitalized;
- Capital expenditures – all capital costs required to purchase the equipment and infrastructure necessary to operate the mine; and,
- Non-cash costs – the plan depreciates capital expenditures on a straight-line basis.

While Golder has identified specific equipment suppliers for several equipment classes within the capital cost estimates, the use of these equipment suppliers in the model does not represent a recommendation from Golder that GB Minerals use these suppliers. Equipment suppliers' names represent only the equipment sizes and capacities typically used within the mining industry, simplifying cost estimation and documentation.

All costs and dollar amount referenced in this section are expressed in terms of Second Quarter 2015 US dollars (USD$).

16.9.1 Direct Mine Operating Costs

16.9.1.1 Direct Mine Operating Costs – Overview

As stated previously, direct mine operating costs include the required labour, supply, and materials costs based on the mine plan schedule. Labour costs include wages for hourly production, maintenance and support employees, and salaries for mine administration and supervisory staffs. Labour calculations also include payroll burdens (e.g., payroll taxes and fringe benefits). Supply and materials costs include expenditures necessary for operating equipment and infrastructure, including costs for consumables, tires, repair parts, and other miscellaneous operating supplies. A zero-based budgeting approach estimated labour and materials costs in developing the cost model. The estimates of the quantity of labour and materials necessary to fulfill the requirements of the mine plan became the basis of all cost estimates.

Operating costs for the preproduction Year 0 are shown as such for the purposes of showing all costs associated with mine operation. However, for this Study, operating costs associated with initial overburden pre-stripping, pit dewatering, and costs associated with material placement for site elevation in the vicinity of facilities and haul roads in advance of mine production during Year 0 are considered as capital costs.

Golder categorized direct operating activities in the following designated functions for reporting and cost analysis purposes:

- Overburden stripping and topsoil removal;
- FPA mining;
- Pit dewatering;
- Reclamation;
- Mine maintenance;
- Operations support;
- FPA stockpiling; and,
- Mine supervision and administration.

The overburden stripping cost centre represents overburden excavation and removal of overburden by the front-end loader fleets and truck haulage of excavated overburden material to designated WD areas.

FPA mining activities include the costs involved in mining the phosphate (matrix) in-pit by the hydraulic backhoe fleet, cleaning of loading faces by smaller dozers, and haulage of ROM matrix to the stockpile. FPA stockpiling encompasses the costs to handle material between the point haul trucks place the matrix at the ROM stockpile until the delivery of the ROM matrix into the plant feed hopper by wheel loader. The cost model does not include costs associated to all activities beyond the ROM Bin (i.e., transporting the phosphate rock (product) from the plant loadout to an off-site location).

For reporting purposes, pit dewatering was treated as single cost centre, separated from the operation support function. Pit dewatering functions include pump and well installations, operating power required, and in-pit pumping activities. The mine pit dewatering effort was based on rainfall within the active pit area measured annually from the crest of the advancing pit to the crest of the advancing in-pit backfill. Capital and operating costs associated with ground water and surface water were developed by others and not included in the mining cost estimate.

The operations support includes estimates for road grading, scraping, dust suppression, haul road maintenance, and other miscellaneous support activities. Reclamation includes estimates for hauling back overburden to the void left behind in the North Pit after mining, overburden stockpiles, in-pit backfill grading, and re-vegetation monitoring. Mine maintenance functions include in-pit equipment fuelling and lubrication, repairing equipment in the field, bulk fuel handling, servicing haul truck tires, light plant operation, and shop maintenance activities such as component replacements and routine equipment maintenance.

The supervision and administration function encompasses the cost of salaried supervisory and administrative personnel stationed at the mine, mine office operating supplies and pickup truck fleet operations and maintenance.

16.9.1.2 Direct Mine Operating Costs – Labour

Golder estimated operating labour requirements using a zero-based approach with annual staffing levels determined by the level of equipment or facility usage dictated by the mine plan. Golder allotted maintenance labour, support labour, mine administration, and supervisory staff to ensure adequate support for production activities and to facilitate effective mine operations. Manpower requirements necessary for the operation of primary production equipment (such as wheel loaders, hydraulic backhoes, overburden and matrix haul trucks, bulldozers, and graders) were based on the respective equipment operating shifts derived using established equipment scheduling parameters. Maintenance and support labour and mine supervisory and administrative personnel were assigned as deemed necessary to adequately support production.

For the Study, Golder assumed mining operations, other than mining and hauling FPA matrix, scheduled on seven days per week, three 8-hour shifts per day basis. Mining matrix is scheduled on the day shift only, one 12 hour shift per day, seven days per week basis due to higher mining dilution consideration during a night shift. Four rotating crews working 12 hour shifts would accomplish continuous coverage. The mine was assumed to operate 365 days per year with 10 holidays covered by overtime. Production during the two month rainy season was de-rated to account for delays and lower productivity from equipment.

Labour cost comprises wages for hourly employees, and salaries for supervisory and administrative personnel were provided by GB Minerals. The compensation rates were provided on an annual basis and included Base Rate, car loans (salaried only), housing, medical and dental, interest on loans, funeral assistance, social security, provident fund, death and disability, workman's compensation, bonuses, and overtime. With the exception of some selected expatriate positions, the compensation rates reflect local conditions. Consequently, the mine will need to develop comprehensive training programs to ensure the development of the workforce sourced locally.

Golder used the information to estimate operating labour costs for six pay-grade categories using hourly operating labour rates. Higher pay-grade categories were assigned to maintenance personnel and equipment operators having greater skill level or work responsibility. General labourers or lower responsibility personnel filled lower pay grade categories. Total hourly costs formed the yearly equivalent of the base rate charges.

Table 16-18 below shows the hourly wage rates for the six pay-grades and the respective job descriptions. Hourly rates are the base rates and annual rates include the additional burden items previously noted.

Table 16-18 Summary of Hourly Wage Rates

Pay Grade	Job Classification	Wage Rate
1	Spotter	$1.77 per hour $7,739 annually
	Pumper	
	Reclamation Laborer	
2	Lasser Man, Operations Trainee	$1.77 per hour $7,739 annually
	Operations Trainee	
3	Loader Operator	$1.60 per hour $7,059 annually
	Compactor Operator	
	Scraper Operator	
	Roller Operator	
	Water Bowser Operator/Water Truck Driver	
	Fuel Truck Driver	
	Tire Serviceman	
	Crane Operator	
	Forklift Operator	
	Maintenance Trainee	
4	Haul Truck Driver	$1.60 per hour $7,059 annually
	Dozer Operator	
	Grader Operator	
	Backhoe Operator	
5	Front Shovel/Excavator (> 5-cubic meter) Operator	$1.94 per hour $8,436 annually
	Wheel Loader (> 5-cubic meter) Operator	
6	Electrician	$1.94 per hour $8,436 annually
	Welder	
	Mechanic	

GB Minerals Ltd. supplied Golder with in-country labour rates based on salary surveys. Expatriate salaries were estimated using base salaries deemed competitive within the region. Table 16-19 on the following page lists a summary of base salaries for mine administration and supervisory staff.

Table 16-19 Summary of Salaried Labour Positions

Position	Base Salary	Other Benefits	Total Compensation
Mining Manager (Expat)	$202,940	$97,060	$300,000
Maintenance Manager (Expat)	$148,190	$76,810	$225,000
Mining Engineer/Geologist (Expat)	$148,190	$76,810	$225,000
Mining Engineer	$129,940	$70,060	$200,000
Mining Superintendent	$48,728	$26,273	$75,000
Geologist	$111,690	$63,310	$175,000
Mine Clerk	$48,728	$26,273	$75,000
Maintenance Superintendent	$6,200	$4,655	$10,855
Surveyor	$111,690	$63,310	$175,000
Foreman	$19,147	$10,853	$30,000
Senior Storekeeper	$11,400	$7,660	$19,060
Blending Control Technician	$8,400	$5,710	$14,110
Warehouse Supervisor	$8,400	$5,710	$14,110

16.9.1.3 Direct Mine Operating Costs – Material & Supply

The material and supply component of the direct mine operating cost represents the expenses incurred for equipment such as fuel, lubricants, rubber tires, and repair/replacement parts, and non-equipment operating supplies including maintenance supplies and other miscellaneous general mine items.

Annual equipment operating supply requirements were estimated on a cost per machine engine hour basis. Note that an engine hour is herein defined as a scheduled hour adjusted for non-consuming mechanical and operating delays to reflect the portion of total scheduled time that a piece of equipment is consuming operating supplies. Unit costs for diesel fuel ($/L) and lubricants ($/L or $/kg) were based on vendor budgetary pricing data. Table 16-20 lists the unit costs applied in the cost model for consumable items.

For non-equipment specific supply cost items (e.g., welding tools, testing equipment, and miscellaneous supplies), parameters other than machine engine hours were utilized in the estimation of annual material and supply expenditures. Unit costs for these items were based on vendor budgetary pricing, available file information, and engineering estimates.

Table 16-20 Summary of Unit Consumable Costs

Lubricant	Grade	Equipment Usage	Unit Cost
Engine Oil	SAE 5W-40	Excavators /Loaders	$4.90 / L
	SAE 10W-30	Mobile equipment	$4.90 / L
	SAE 15W-40	Haul trucks	$4.90 / L
Transmission Fluid	SAE 0W-30	Excavators /Loaders	$4.05 / L
	SAE 30	Mobile equipment	$4.05 / L
	SAE 5W-30	Haul trucks	$4.05 / L
Final Drive & Differential Fluids	SAE 50	Excavators	$4.20 / L
	SAE 60	Haul trucks	$4.20 / L
	SAE 75W-90	Small excavators	$4.35 / kg
	SAE 80W-90	Mobile equipment	$4.35 / kg
Hydraulic Fluid	SAE 0W-20	Excavators	$3.95 / L
	SAE 5W-30	Mobile equipment	$3.95 / L
	SAE 10W	Haul trucks	$3.95 / L
	32	Haul trucks	$3.95 / L
Grease	Multi-purpose	Mobile equipment	$6.15 / kg
	3% Moly	Hydraulic /Excavators	$9.10 / kg
	5% Moly	Shovels / Loaders	$9.10 / kg
	Synthetic	Excavators	$9.10 / kg
Fuel, Diesel	n/a	All Equipment	$0.60 / L Year 0
			$0.80 / L Years 1-25

16.9.1.4 Direct Mine Operating Costs – Equipment Hourly Rates

Equipment hourly operating costs are a function of the estimated hourly consumption or usage of fuel, lubricants, rubber tires, filters, and repair/replacement parts. Estimated consumption rates of fuel and lubricants for individual pieces of equipment were based on manufacturer/dealer specifications and guidelines, engineering estimates, and actual operating data on file at Golder. Where applicable, the total hourly cost of operating various types of equipment was determined by applying unit consumable costs to equipment usage estimates. Other elements included in determining the hourly operating cost estimate for each equipment type were hourly tire costs, undercarriage costs, and rebuild and replacement costs.

Hourly tire costs for rubber-tired equipment were developed using vendor budgetary tire price data and estimated tire lives. Equipment hourly repair/replacement and filter costs reflect manufacturer/dealer cost information and engineering estimates based on Golder's experience. Table 16-21 lists the unit costs applied in the cost model for consumable items.

Golder estimated annual operating costs for mining and support equipment by multiplying the operating hours derived for a particular piece of equipment in a given year by the respective machine hourly operating cost. Operating hours for major production equipment (e.g., hydraulic backhoes, wheel loaders, haul trucks, dozers, and graders) are a function of the scheduled material volumes or tonnages to be moved and estimated estimated equipment production rates. Support equipment was assigned as deemed necessary to facilitate an effective mining operation.

Table 16-21 also lists the base price of each equipment obtained from major equipment suppliers or manufacturers. Golder secured the quotes for support equipment, such as welding machine, light plant, or small forklift, from local manufacturers in the region to maintain more accurate prices. The base price included tire costs for trucks, wheel loaders, graders, and other wheeled machines, but excluded taxes, freight, comissions, and other applicable fees.

Table 16-21 Summary of Equipment Base Price and Hourly Operating Costs

EQUIPMENT DESCRIPTION					HOURLY COSTS						
Equipment Type	Manufactorer & Model	Size Class		Total Capital Cost	Fuel	Lube	Tire	Filter	U.C.	R&R	Total
				($)	($)	($)	($)	($)	($)	($)	($)
Backhoe	Caterpillar 374F - Backhoe	5.0	m^3 bucket	$860,700	$33.20	$4.56	n/a	$1.15	$13.76	$46.78	**$99.45**
Backhoe	Caterpillar 336D - Backhoe	2.1	m^3 bucket	$385,500	$31.20	$2.75	n/a	$1.10	$3.71	$14.99	**$53.75**
Wheel Loader	Caterpillar 992K - Wheel Loader	12.2	m^3 bucket	$2,165,400	$78.72	$8.40	$14.71	$1.80	n/a	$85.79	**$189.42**
Dozer	Caterpillar D9R - Dozer	405	hp	$785,600	$43.92	$2.91	n/a	$1.00	$15.71	$23.57	**$87.11**
End Dump Truck	Caterpillar 777D - End Dump Truck	97	tonnes	$1,296,250	$60.00	$6.92	$25.20	$0.90	n/a	$31.42	**$124.44**
End Dump Truck	Caterpillar 770 - End Dump Truck	36	tonnes	$631,650	$29.28	$6.92	$7.20	$0.45	n/a	$12.34	**$56.19**
Motor Grader	Caterpillar 16M - Motor Grader	297	hp	$802,700	$26.32	$2.00	$2.93	$0.50	n/a	$21.36	**$53.11**
Compactor	Caterpillar CS-56 - Compactor	147	hp	$164,000	$13.60	$3.91	$0.00	$0.60	n/a	$3.60	**$21.71**
Scraper	Caterpillar 637G - Scraper	26.0	m^3 bucket	$1,276,200	$75.36	$3.91	$0.00	$1.35	n/a	$28.21	**$108.84**
Water Truck	Caterpillar 770 - Water Truck	34,000	liter	$781,650	$32.64	$6.92	$7.20	$0.40	n/a	$18.74	**$65.90**
Fuel/Lube Truck	Fuel/Lube Truck	400	hp	$353,000	$21.52	$2.52	$0.00	$1.10	n/a	$26.48	**$51.61**
Mechanic's Truck	Mechanic's Truck	150	hp	$72,000	$7.28	$0.85	$0.00	$0.45	n/a	$3.12	**$11.70**
Pickup Truck	Pickup Truck	128	hp	$43,000	$4.56	$0.41	$0.00	$0.50	n/a	$0.68	**$6.15**
Crew Bus	Crew Bus	94	hp	$59,000	$4.56	$2.52	$0.00	$0.55	n/a	$0.78	**$8.41**
95-tonne Crane	Liebherr LTM 1095 - Mobile Crane	95	tonne	$1,155,000	$55.68	$4.19	$0.00	$1.15	n/a	$20.63	**$81.65**
10-tonne Forklift	10-tonne Forklift	10	tonne	$60,947	$15.76	$2.60	$0.00	$0.55	n/a	$1.09	**$20.00**
2 - 4 ton Forklift	2 - 4 ton Forklift										**$0.00**
Welding Machine	Welding Machine	24	hp	$9,493	$1.20	$0.07	n/a	n/a	n/a	$0.36	**$1.63**
Light Plant	Light Plant	2,300	m^2	$8,536	$2.16	$0.12	n/a	$0.10	n/a	$0.26	**$2.64**
Wash Plant	Beneficiation Wash Plant										**$0.00**
Screening Plant	Screening Plant	266	hp	$22,495	$20.92	$0.12	n/a	$0.00	n/a	$1.52	**$22.56**

16.9.1.5 Direct Mine Operating Costs – Base Summary

Direct operating expenses represent the single largest component of estimated total production costs, typically accounting for over 50% of the total costs of production. The direct operating estimates included direct operating pre-production activities associated with initial stripping in advance of first production at Year 0. The costs include the construction of mine road maintenance, ROM stockpile facilities, and the pre-stripping of low-ratio pits.

The annual direct mine operating costs for GB Minerals are listed in Table 16-22. As the mining consultant for the project, Golder calculated and reported all cash costs directly associated with mining, including preproduction Year 0 and backfill of the void in the North Pit after mining (“void backfill”). As part of the integration into the total project cost estimate, Lycopodium, as the lead consultant and under the direction of GB Minerals, elected to categorize preproduction and void backfill costs as capital expenses.

The operating costs comprise of material and supply estimates and labour cost. The total direct mine operating cost, including preproduction and void backfill costs, is $903.2 million, or $27.18/t product. Annual costs range from $15.1 million (M) in preproduction Year 0 to $46.1M in Year 15. When the estimated preproduction costs of $15.1 million and void backfill costs of $57.0 million are excluded from the direct operating costs and categorized as capital, the total becomes $831.0 million, or $25.01/t product.

The primary direct mine operating cost drivers in Table 16-22 are overburden stripping and FPA (matrix) mining. These costs account for 79% of the total average direct mine operating costs over the life of the mine. Fluctuations in annual direct operating costs are primarily attributed to changes in physical mining parameters such as stripping ratios, haulage distance, and lift. Overburden and matrix haulage costs exhibit the greatest variability.

Dewatering costs only include the cost to pump rainwater from the active pit, which was assumed to be the area at the end of each year between the crest of the mining pit(s) and the crest of the advancing in-pit backfill dump(s). Dewatering costs varied from $46,000 in Year 0 to $424,000 in Year 18.

Table 16-23 details annual labour requirements and indicates that productivity is approximately 3,752 product tonnes per employee over the life of mine.

As previously noted, expenses related to haul road construction were not included in direct operating expenses. These were categorized as capital cost and are explained later in Section 16.9.3. However, the costs for road maintenance were included in the operations support component and comprise costs to operate a grader, utility backhoe and water truck.

Table 16-22 Summary of Direct Mine Operating Costs – Page 1

DESCRIPTION	Year 1	Year 2	Year 3	Year 4	Year 5	Years 6 - 10	Years 11 - 15	Years 16 - 20	Years 21 - 26	TOTAL
PRODUCTION STATISTICS										
Total ROM Production (000s tonne - Dry Basis)	1,750	1,750	1,750	1,750	1,750	8,750	8,750	8,750	9,007	44,007
Total Product Tonnage (000s tonne - Dry Basis)	1,321	1,321	1,321	1,321	1,321	6,606	6,606	6,606	6,800	33,225
Total Stripping Volume (000s bcm)	11,172	14,922	14,318	13,079	11,898	77,517	95,891	88,645	91,240	418,680
Rehandle Volume (000s bcm)	-	-	-	-	-	-	-	-	-	-
Total Effective Stripping Volume (000s bcm)	11,172	14,922	14,318	13,079	11,898	77,517	95,891	88,645	91,240	418,680
Stripping Ratio (bcm/ROM Tonne)	6.38	8.53	8.18	7.47	6.80	8.86	10.96	10.13	10.13	9.51
Productivity (ROM Tonne/Total Employees)	5,105	4,280	4,410	4,641	5,063	4,291	3,241	3,629	3,668	3,905
DIRECT OPERATING COSTS										
Waste Stripping ($000s)	$16,351	$21,136	$20,065	$18,925	$15,355	$103,696	$155,745	$126,967	$121,143	$599,383
Cost Per Product tonne ($/Tonne)	*$12.38*	*$16.00*	*$15.19*	*$14.32*	*$11.62*	*$15.70*	*$23.58*	*$19.22*	*$17.81*	*$18.04*
FPA Mining ($000s)	$1,766	$1,694	$1,829	$1,864	$1,932	$10,269	$10,436	$11,212	$13,046	$54,048
Cost Per Product tonne ($/Tonne)	*$1.34*	*$1.28*	*$1.38*	*$1.41*	*$1.46*	*$1.55*	*$1.58*	*$1.70*	*$1.92*	*$1.63*
Pit Dewatering ($000s)	$110	$163	$181	$180	$233	$659	$1,290	$1,821	$1,925	$6,562
Cost Per Product tonne ($/Tonne)	*$0.08*	*$0.12*	*$0.14*	*$0.14*	*$0.18*	*$0.10*	*$0.20*	*$0.28*	*$0.28*	*$0.20*
Reclamation ($000s)	$469	$831	$815	$549	$499	$3,411	$4,791	$4,551	$4,871	$20,788
Cost Per Product tonne ($/Tonne)	*$0.35*	*$0.63*	*$0.62*	*$0.42*	*$0.38*	*$0.52*	*$0.73*	*$0.69*	*$0.72*	*$0.63*
Maintenance ($000s)	$1,534	$1,899	$1,827	$1,696	$1,578	$9,823	$12,258	$11,093	$11,482	$53,189
Cost Per Product tonne ($/Tonne)	*$1.16*	*$1.44*	*$1.38*	*$1.28*	*$1.19*	*$1.49*	*$1.86*	*$1.68*	*$1.69*	*$1.60*
Operations Support ($000s)	$1,125	$1,125	$1,125	$1,125	$1,125	$5,623	$5,626	$5,626	$6,747	$29,248
Cost Per Product tonne ($/Tonne)	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.99*	*$0.88*
FPA Processing ($000s)	$959	$959	$959	$959	$959	$4,794	$4,794	$4,794	$4,935	$24,111
Cost Per Product tonne ($/Tonne)	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*
Mine Supervision & Administration ($000s)	$1,679	$1,679	$1,679	$1,679	$1,679	$8,396	$8,396	$8,396	$10,075	$43,657
Cost Per Product tonne ($/Tonne)	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.48*	*$1.31*
DIRECT OPERATING COSTS ($000s)	**$23,993**	**$29,487**	**$28,481**	**$26,977**	**$23,360**	**$146,670**	**$203,335**	**$174,460**	**$174,224**	**$830,986**
Cost Per Product tonne ($/Tonne)	***$18.16***	***$22.32***	***$21.56***	***$20.42***	***$17.68***	***$22.20***	***$30.78***	***$26.41***	***$25.62***	***$25.01***

Table 16-23 Summary of Labour Requirement – Page 1

DESCRIPTION	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
PRODUCTION STATISTICS															
Total Product Tonnage (000s tonne - DB)	**-**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**
Total Prime Stripping Volume (000s bcm)	**5,818**	**11,172**	**14,922**	**14,318**	**13,079**	**11,898**	**10,947**	**13,247**	**17,701**	**18,369**	**17,252**	**17,959**	**19,494**	**19,512**	**19,472**
Rehandle Volume (000s bcm)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Effective Stripping Volume (000s bcm)	**5,818**	**11,172**	**14,922**	**14,318**	**13,079**	**11,898**	**10,947**	**13,247**	**17,701**	**18,369**	**17,252**	**17,959**	**19,494**	**19,512**	**19,472**
Stripping Ratio (bcm/ROM Tonne)	0.0	9.1	12.2	11.7	10.7	9.7	8.9	10.8	14.4	15.0	14.1	14.7	15.9	15.9	15.9
Productivity (Product Tonne/Total Employees)	**0**	**5,105**	**4,280**	**4,410**	**4,641**	**5,063**	**5,302**	**4,815**	**3,880**	**3,526**	**3,932**	**3,876**	**3,267**	**3,125**	**2,992**
OPERATIONS LABOR															
Shovel/Backhoe/Small Excavator Operators	4	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Loader Operators	10	20	25	24	23	21	20	23	29	30	29	30	32	32	32
Haul & Water Truck Operators	58	76	90	87	85	70	66	72	104	106	101	97	122	139	157
Compactor Operators	9	17	23	22	20	18	17	21	27	28	27	28	30	30	30
Dozer Operators	8	14	18	18	16	15	13	16	22	23	21	22	24	24	24
Grader, Scraper & Utility Equipment Operators	6	9	10	10	10	9	9	10	11	11	11	11	12	12	12
Pumper / Laborer	1	3	5	5	5	7	5	5	4	3	4	5	7	7	9
SUBTOTAL - OPERATIONS LABOR	***96***	***148***	***180***	***176***	***168***	***149***	***138***	***156***	***206***	***210***	***202***	***202***	***235***	***254***	***272***
MAINTENANCE LABOR															
Fuel Truck Driver / Serviceman	12	23	30	28	25	23	23	25	33	35	32	35	37	37	37
Electricians	3	3	3	3	3	3	3	3	3	6	3	3	6	6	6
Crane / Forklift Operators	4	9	12	11	10	10	9	11	13	13	13	13	14	14	14
Mechanics / Welders	10	10	10	10	10	10	10	10	10	20	10	10	20	20	20
Tire Servicemen & Maintenance Helpers / Trainees	24	36	44	42	38	36	36	40	46	60	46	48	62	62	62
SUBTOTAL - MAINTENANCE LABOR	***53***	***81***	***98***	***94***	***87***	***82***	***81***	***89***	***105***	***134***	***104***	***109***	***139***	***139***	***139***
TOTAL HOURLY LABOR	**150**	**229**	**279**	**270**	**255**	**231**	**219**	**244**	**311**	**345**	**306**	**311**	**374**	**393**	**412**
SUPERVISION & ADMINISTRATION															
Mine Supervision & Administration	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
TOTAL SUPERVISION & ADMINISTRATION	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**
TOTAL WORKFORCE	**180**	**259**	**309**	**300**	**285**	**261**	**249**	**274**	**341**	**375**	**336**	**341**	**404**	**423**	**442**

Table 16-23 Summary of Labour Requirement – Page 2

DESCRIPTION	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	TOTAL / AVG.
PRODUCTION STATISTICS															
Total Product Tonnage (000s tonne - DB)	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**1,321**	**180**	**-**	**-**	**30,805**
Total Prime Stripping Volume (000s bcm)	**19,454**	**19,555**	**18,806**	**17,605**	**17,580**	**15,099**	**15,186**	**15,370**	**17,040**	**19,521**	**21,282**	**21,141**	**8,900**	**-**	**451,698**
Rehandle Volume (000s bcm)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0
Total Effective Stripping Volume (000s bcm)	**19,454**	**19,555**	**18,806**	**17,605**	**17,580**	**15,099**	**15,186**	**15,370**	**17,040**	**19,521**	**21,282**	**21,141**	**8,900**	**0**	**451,698**
Stripping Ratio (bcm/ROM Tonne)	15.9	16.0	15.4	14.4	14.4	12.3	12.4	12.5	13.9	15.9	17.4	117.6	0.0	0.0	10.3
Productivity (Product Tonne/Total Employees)	**2,946**	**3,179**	**3,443**	**3,671**	**3,759**	**4,095**	**4,089**	**4,117**	**3,874**	**3,322**	**3,197**	**498**	**0**	**0**	**3,752**
OPERATIONS LABOR															
Shovel/Backhoe/Small Excavator Operators	9	9	9	9	9	9	9	9	9	9	9	4	4	4	9
Loader Operators	32	32	31	29	29	26	26	26	28	32	35	31	13	0	28
Haul & Water Truck Operators	162	129	104	114	110	96	95	92	100	114	116	134	62	9	104
Compactor Operators	30	30	29	27	27	23	24	24	26	30	33	33	14	0	26
Dozer Operators	24	24	23	22	22	19	19	19	21	24	26	26	11	0	20
Grader, Scraper & Utility Equipment Operators	12	12	12	11	11	10	10	10	11	12	13	11	7	4	11
Pumper / Laborer	11	11	11	13	9	11	12	12	11	7	7	7	7	0	8
SUBTOTAL - OPERATIONS LABOR	***279***	***246***	***219***	***225***	***217***	***194***	***195***	***192***	***207***	***228***	***239***	***247***	***117***	***16***	***206***
MAINTENANCE LABOR															
Fuel Truck Driver / Serviceman	37	37	35	33	33	30	30	30	32	37	40	36	17	1	32
Electricians	6	6	6	3	3	3	3	3	3	6	6	3	3	0	4
Crane / Forklift Operators	14	14	14	13	13	12	12	12	13	14	15	13	5	0	13
Mechanics / Welders	20	20	20	10	10	10	10	10	10	20	20	10	10	0	14
Tire Servicemen & Maintenance Helpers / Trainees	62	62	60	46	46	44	44	44	46	62	64	50	28	0	50
SUBTOTAL - MAINTENANCE LABOR	***139***	***139***	***135***	***105***	***105***	***98***	***99***	***99***	***104***	***139***	***145***	***112***	***63***	***1***	***111***
TOTAL HOURLY LABOR	**419**	**386**	**354**	**330**	**322**	**293**	**293**	**291**	**311**	**368**	**383**	**359**	**180**	**17**	**317**
SUPERVISION & ADMINISTRATION															
Mine Supervision & Administration	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
TOTAL SUPERVISION & ADMINISTRATION	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**
TOTAL WORKFORCE	**449**	**416**	**384**	**360**	**352**	**323**	**323**	**321**	**341**	**398**	**413**	**389**	**210**	**47**	**347**

16.9.2 Indirect Mining Costs

Indirect mine operating costs are those costs incurred by the mining operation and not directly attributable to the production of matrix. Indirect costs include: property and liability insurance; permitting fees; bonding; engineering consulting fees; exploration drilling; legal and auditing fees; freight and postage fees; communications fees; government and environmental relations fees; lab sampling and quality control; employee related training; industry dues; royalty costs; and other miscellaneous expenses.

Methods commonly used to estimate indirect operating costs include estimation as a constant yearly expense, estimation as a fraction of the capital asset net book value, expense per employee-year, and estimation as a unit rate per product tonne. However, Golder did not include overhead expenses in the cost model assuming these expenses would be provided by GB Minerals Ltd. or Lycopodium.

16.9.3 Capital Expenditures and Non-Cash Costs

16.9.3.1 Capital Expenditures

Capital costs represent the investment in physical assets required to facilitate matrix production, processing and delivery of the finished rock product to the port. Capital assets include mobile mining equipment, service and support equipment, material handling, processing, facilities, and infrastructure including those required to sustain the operations and those required for environmental protection. The capital expenditures developed for Item 16 refer to those items directly related to mining and include mining equipment, road development and water management within the developing pit.

Capital outlays to bring the property to full production are referred to as initial capital. Capital spending to periodically replace equipment as it becomes worn out and equipment that is required to meet increasing task and ongoing development of roads is referred to as sustaining capital. Capital costs were developed on an annual basis to meet the mining requirements.

Two capital cost estimates were developed. A 100% Equity Case was developed assuming 100% equity ownership of all Project assets and that all equipment would be purchased on a new basis. A Lease Case was also developed whereby major mining equipment was obtained through a five year lease during which time the equipment was paid off and ownership was transferred to the mine. The following considers the 100% Equity Case first with the Lease Option alternative following.

16.9.3.2 Capital Basis

Unit capital costs for primary production equipment were generally based on dealer/manufacturer budgetary price quotes and Golder file data. Golder obtained prices from dealers or suppliers during the second quarter of 2015. The quoted prices for major mining equipment such as front end loaders, hydraulic backhoes, haul trucks, dozers, and graders included costs of typical standard performance and safety options. Additionally, a spare parts allowance of 5% was assumed for major equipment purchases. Capital costs for support equipment, service vehicles, and ancillary mine support equipment such as light plants, a rock screening plant, and welding machines were also based on manufacturer quotes, primarily from local manufacturers in Africa.

Mine capital estimates assume that capitalized rebuilds will be employed to extend the effective service life of hydraulic backhoes, wheel loaders, haul trucks, water trucks, dozers and graders. Estimated rebuild parameters for these units are outlined later in this section as Table 16-27.

Due to the remoteness of the mine site, it is assumed that the Project will have to be self-supporting and provide the necessary facilities and services. Most infrastructure capital expenditures, such as the processing plant, offices, and maintenance facilities, were the responsibility of Lycopodium.

Golder estimated capital costs using two categories for analytical purposes: (1) initial capital expenditures, and (2) sustaining capital expenditures. Initial capital represents the estimate of capital required to progress the operation to a production stage including road construction, infrastructure, and accumulated miscellaneous expenses at Years 0 and 1. Sustaining capital represents the capital required over the remainder of the mine operation's life. Sustaining capital comprises equipment replacement and rebuilds, equipment capital additions, haul road development, and minor miscellaneous capital requirements.

Table 16-24 summarizes estimated initial and sustaining capital requirements for the Project mine plan. Initial estimated capital requirements total $50.0 M (exclusive of capital development stripping) with primary mining equipment accounting for 85% of the initial requirements. The Support Equipment & Spare Parts accounts for an additional 7% of initial capital requirements, making initial equipment capital 92% of the initial expenditure. Other major capital expenditures including early road construction and the internal mine dewatering system estimated at $4.0 M. Dewatering of groundwater in advance of mining and surface water management costs were covered by others.

Estimated sustaining capital totals $187.9 M, with nearly all expenditures associated with additional equipment purchases, equipment replacement, or major equipment rebuilds. Incremental haul road development takes up almost 16% of the total sustaining capital. Annual estimates of mine plan capital expenditure over the LOM are included in Table 16-25 and Figure 16-44.

Estimated capital did not take into account the costs related to maintenance facility construction and other support infrastructures, such as office buildings and fuel islands. These costs were accounted for by Lycopodium.

Table 16-24 Summary of Capital Expenditures—100% Equity Case

Description	Initial Capital	Sustaining Capital	Total
Primary Mining Equipment ($000's)			
Wheel Loaders	$10,827	$60,668	$71,495
Excavators/Backhoes	$2,968	$310	$3,277
Dozers	$3,142	$12,054	$15,197
Haul Trucks	$20,641	$60,108	$80,749
Motor Grader	$1,605	$6,124	$7,729
Water Trucks	$1,563	$788	$2,351
Compactors and Scrapers	$656	$3,240	$3,896
Support Equipment & Spare Parts ($000's)	$4,543	$11,693	$16,236
Processing, Infrastructure & Miscellaneous ($000's)			
Dewatering System	$1,000	$3,000	$4,000
Haul Road Construction	$3,030	$29,887	$32,916
TOTAL ESTIMATED CAPITAL EXPENDITURE ($000s)	$49,976	$187,872	$237,847

The costs associated with initial haul road construction and progression development throughout the operation represent a significant component of capital expenses. It accounts for 14% of the total estimated capital. Based on haul road design criteria as shown in Section 16.5.11, the roads were designed for use of 97 t overburden and 36 t matrix trucks. As seen in Table 16-26, Golder calculated the unit cost of road construction to be $178 and $57 per metre of road built for overburden and matrix trucks, respectively. These costs incurred for the aggregates supply of about $17.29/m^3, the use of 2.1 m^3 bucket backhoe and 36 t truck fleet, a mobile rock screening plant, grader and compactor. Labour cost to operate the required equipment was also included in the estimate.

Estimated capital did not take into account the costs related to maintenance facility construction and other support infrastructures, such as office buildings and fuel islands. These costs are covered by others.

Table 16-25 Summary of Yearly Estimated Capital Expenditures Page 1

Machine / Item	Capital New / Rebuild	Replace-ment Life (hours)	Unit Cost ($000s)	Deprec-iation Life (Years)	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
STRIPPING & LOADING MACHINES																			
Caterpillar 992K - Wheel Loader	New/Replace	36,000	$2,165	7	$6,496	$4,331	$2,165	-	-	-	-	$6,496	$6,496	$4,331	-	-	-	-	$6,496
Caterpillar 992K - Wheel Loader	Rebuild	18,000	$297	3	-	-	-	$891	$594	$297	-	-	-	-	-	$1,188	$891	$297	-
Caterpillar 374DL - Backhoe	New/Replace	60,000	$861	10	-	$2,582	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar 374DL - Backhoe	Rebuild	30,000	$103	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar 336DL - Backhoe	New/Replace	48,000	$386	7	$386	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar 336DL - Backhoe	Rebuild	24,000	$45	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar D9R - Dozer	New/Replace	48,000	$786	10	$1,571	$1,571	$786	-	-	-	-	-	$786	-	-	$1,571	$1,571	$786	-
Caterpillar D9R - Dozer	Rebuild	24,000	$75	5	-	-	-	-	-	$151	$151	-	$75	-	-	-	-	$75	-
Spare Parts Inventory (@ 5%)	Other	n/a	n/a	5	$423	-	$148	-	-	-	-	-	$364	$217	-	$79	$79	$39	$325
HAUL TRUCKS																			
Caterpillar 777G - End Dump Truck	New/Replace	60,000	$1,296	10	$15,555	$1,296	$5,185	-	-	-	-	-	$2,593	$1,296	-	$15,555	$5,185	$10,370	$5,185
Caterpillar 777G - End Dump Truck	Rebuild	30,000	$105	5	-	-	-	-	-	$1,257	$105	-	$419	-	-	-	-	$209	$105
Caterpillar 770 - End Dump Truck	New/Replace	60,000	$632	10	$632	$3,158	-	$632	-	-	-	$632	-	-	-	-	-	-	$632
Caterpillar 770 - End Dump Truck	Rebuild	30,000	$53	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Parts Inventory (@ 5%)	Other	n/a	n/a	5	$809	-	$259	$32	-	-	-	$32	$130	$65	-	$778	$259	$519	$291
MOBILE EQUIPMENT																			
Caterpillar 16M - Motor Grader	New/Replace	42,000	$803	7	$1,605	-	$803	-	-	-	-	-	-	-	$1,605	-	$803	-	-
Caterpillar 16M - Motor Grader	Rebuild	28,000	$56	3	-	-	-	-	-	-	-	$112	-	$56	-	-	-	-	-
Caterpillar 770 - Water Truck	New/Replace	LOM	$782	10	$1,563	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar 770 - Water Truck	Rebuild	24,000	$66	5	-	-	-	-	$131	-	-	-	$131	-	-	-	-	$131	-
Caterpillar CS-56 - Compactor	New/Replace	42,000	$164	10	$492	$164	$328	-	-	-	-	-	$164	-	$656	-	$328	-	-
Caterpillar CS-56 - Compactor	Rebuild	28,000	$14	5	-	-	-	-	-	-	-	$58	-	$29	-	-	-	$14	-
Caterpillar 637G - Scraper	New/Replace	42,000	$1,276	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar 637G - Scraper	Rebuild	28,000	$122	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Parts Inventory (@ 5%)	Other	n/a	n/a	5	$158	-	$40	-	-	-	-	-	-	-	$80	-	$40	-	-
SERVICE & SUPPORT EQUIPMENT																			
Caterpillar 428F - Backhoe Loader	New/Replace	60,000	$108	10	$108	-	-	-	-	-	-	-	-	-	-	-	$108	-	-
Caterpillar 428F - Backhoe Loader	Rebuild	30,000	$10	5	-	-	-	-	-	-	$10	-	-	-	-	-	-	-	-
Fuel/Lube Truck	New/Replace	40,000	$353	10	$706	$353	-	-	-	-	-	-	$353	$706	$353	-	-	-	-
Fuel/Lube Truck	Rebuild	20,000	$53	5	-	-	-	-	$106	$53	-	-	-	-	-	-	$53	$106	$53
Mechanic's Truck	New/Replace	30,000	$72	10	$72	$72	-	-	-	-	-	$72	$72	-	-	-	-	$72	$72
Mechanic's Truck	Rebuild	30,000	$11	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pickup Truck	New/Replace	30,000	$43	7	$516	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pickup Truck	Rebuild	30,000	$6	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crew Bus	New/Replace	LOM	$59	7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crew Bus	Rebuild	36,000	$9	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Liebherr LTM 1095 - Mobile Crane	New/Replace	LOM	$1,155	10	$1,155	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Liebherr LTM 1095 - Mobile Crane	Rebuild	35,000	$231	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10-tonne Forklift	New/Replace	LOM	$61	10	$61	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10-tonne Forklift	Rebuild	35,000	$11	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Welding Machine	New/Replace	20,000	$9	10	$19	-	-	-	-	-	-	-	-	-	-	$19	-	-	-
Welding Machine	Rebuild	20,000	$1	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Screening Plant	New/Replace	10,000	$22	20	$22	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Screening Plant	Rebuild	10,000	$3	n/a	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Light Plant	New/Replace	40,000	$9	7	$43	$26	$17	-	-	-	-	-	$60	$26	$17	-	$9	-	-
Light Plant	Rebuild	20,000	$1	5	-	-	-	-	$5	$3	$2	-	-	-	-	$2	$5	$3	$2
INFRASTRUCTURE & MISC.																			
Dewatering System	New/Replace	LOM	n/a	25	-	$1,000	$1,000	-	-	-	-	-	-	-	$2,000	-	-	-	-
Haul Road Construction	New/Replace	LOM	n/a		$1,730	$1,300	$1,382	$1,444	$1,289	$1,187	$967	$1,139	$993	$912	$968	$1,384	$1,130	$1,269	$1,601
TOTAL ESTIMATED CAPITAL EXPENDITURE ($000s)					**$34,123**	**$15,853**	**$12,113**	**$2,998**	**$2,125**	**$2,947**	**$1,234**	**$8,541**	**$12,636**	**$7,637**	**$5,680**	**$20,575**	**$10,460**	**$13,891**	**$14,762**
CUMULATIVE CAPITAL EXPENDITURE ($000s)					**$34,123**	**$49,976**	**$62,088**	**$65,086**	**$67,211**	**$70,158**	**$71,392**	**$79,933**	**$92,568**	**$100,205**	**$105,885**	**$126,460**	**$136,920**	**$150,811**	**$165,572**

Table 16-25 Summary of Yearly Estimated Capital Expenditures Page 2

Machine / Item	Capital New / Rebuild	Replace-ment Life (hours)	Unit Cost ($000s)	Deprec-iation Life (Years)	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Total
STRIPPING & LOADING MACHINES																			
Caterpillar 992K - Wheel Loader	New/Replace	36,000	$2,165	7	$6,496	$4,331	-	-	-	-	-	$6,496	$6,496	$4,331	-	-	-	-	$64,962
Caterpillar 992K - Wheel Loader	Rebuild	18,000	$297	3	-	-	-	$1,188	$891	$297	-	-	-	-	-		-	-	$6,533
Caterpillar 374DL - Backhoe	New/Replace	60,000	$861	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$2,582
Caterpillar 374DL - Backhoe	Rebuild	30,000	$103	5	-	-	-	-	-	-	-	-	$310	-	-	-	-	-	$310
Caterpillar 336DL - Backhoe	New/Replace	48,000	$386	7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$386
Caterpillar 336DL - Backhoe	Rebuild	24,000	$45	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar D9R - Dozer	New/Replace	48,000	$786	10	-	-	-	$786	-	-	$1,571	$1,571	$786	-	$786	-	-	-	$14,141
Caterpillar D9R - Dozer	Rebuild	24,000	$75	5	-	$151	$151	$75	-	-	-	-	$75	-	-	-	$151	-	$1,056
Spare Parts Inventory (@ 5%)	Other	n/a	n/a	5	$325	$217	-	$39	-	-	$79	$403	$364	$217	$39	-	-	-	$3,355
HAUL TRUCKS																			
Caterpillar 777G - End Dump Truck	New/Replace	60,000	$1,296	10	$1,296	-	-	-	$2,593	$1,296	-	-	-	-				-	$67,405
Caterpillar 777G - End Dump Truck	Rebuild	30,000	$105	5	-	$1,257	$314	$524	$419	$419	-	$105	-	-	-	-	$209	-	$5,341
Caterpillar 770 - End Dump Truck	New/Replace	60,000	$632	10	-	-	$632	-	-	-	-	-	$632	$632	-	-	-	-	$7,580
Caterpillar 770 - End Dump Truck	Rebuild	30,000	$53	5	$53	$264	-	$53	-	-	-	$53	-	-	-	-	-	-	$423
Spare Parts Inventory (@ 5%)	Other	n/a	n/a	5	$65	-	$32	-	$130	$65	-	-	$32	$32	-	-	-	-	$3,527
MOBILE EQUIPMENT																			
Caterpillar 16M - Motor Grader	New/Replace	42,000	$803	7	-	-	-	-	$1,605	-	$803	-	-	-	-	-	-	-	$7,224
Caterpillar 16M - Motor Grader	Rebuild	28,000	$56	3	-	$112	-	$56	-	-	-	-	-	-	$112	-	-	$56	$505
Caterpillar 770 - Water Truck	New/Replace	LOM	$782	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$1,563
Caterpillar 770 - Water Truck	Rebuild	24,000	$66	5	-	-	$131	-	-	-	-	$131	-	-	-	$131	-	-	$788
Caterpillar CS-56 - Compactor	New/Replace	42,000	$164	10	-	$164	-	$492	$164	$328	-	-	-	-	$164	$164	-	-	$3,608
Caterpillar CS-56 - Compactor	Rebuild	28,000	$14	5	$43	$14	$29	-	-	-	-	-	$14	-	$43	$14	$29	-	$288
Caterpillar 637G - Scraper	New/Replace	42,000	$1,276	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Caterpillar 637G - Scraper	Rebuild	28,000	$122	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Parts Inventory (@ 5%)	Other	n/a	n/a	5	-	-	-	-	$80	-	$40	-	-	-	-	-	-	-	$439
SERVICE & SUPPORT EQUIPMENT																			
Caterpillar 428F - Backhoe Loader	New/Replace	60,000	$108	10	-	-	-	-	-	-	-	-	-	-	$108	-	-	-	$323
Caterpillar 428F - Backhoe Loader	Rebuild	30,000	$10	5	-	-	-	-	$10	-	-	-	-	-	-	-	-	-	$20
Fuel/Lube Truck	New/Replace	40,000	$353	10	-	$353	-	$1,059	-	-	-	-	-	-	-	$353	-	-	$4,236
Fuel/Lube Truck	Rebuild	20,000	$53	5	-	-	-	-	-	-	$53	-	$106	$53	-	-	-	-	$582
Mechanic's Truck	New/Replace	30,000	$72	10	$72	-	-	-	-	-	$72	-	$72	$72	-	-	-	-	$720
Mechanic's Truck	Rebuild	30,000	$11	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pickup Truck	New/Replace	30,000	$43	7	-	-	-	-	-	-	-	-	$516	-	-	-	-	-	$1,032
Pickup Truck	Rebuild	30,000	$6	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crew Bus	New/Replace	LOM	$59	7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crew Bus	Rebuild	36,000	$9	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Liebherr LTM 1095 - Mobile Crane	New/Replace	LOM	$1,155	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$1,155
Liebherr LTM 1095 - Mobile Crane	Rebuild	35,000	$231	5	-	-	-	$231	-	-	-	-	-	-	-	-	-	-	$231
10-tonne Forklift	New/Replace	LOM	$61	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$61
10-tonne Forklift	Rebuild	35,000	$11	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Welding Machine	New/Replace	20,000	$9	10	-	-	-	-	-	$19	-	-	-	-	-	-	-	-	$57
Welding Machine	Rebuild	20,000	$1	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Screening Plant	New/Replace	10,000	$22	20	-	-	$22	-	-	-	-	-	-	-	-	-	-	-	$45
Screening Plant	Rebuild	10,000	$3	n/a	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Light Plant	New/Replace	40,000	$9	7	$17	$43	$26	$17	-	$9	-	-	-	$17	$51	$26	$17	-	$418
Light Plant	Rebuild	20,000	$1	5	$1	-	-	-	-	$7	$3	-	$2	$1	-	-	-	-	$35
INFRASTRUCTURE & MISC.																			
Dewatering System	New/Replace	LOM	n/a	25	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$4,000
Haul Road Construction	New/Replace	LOM	n/a		$1,213	$980	$1,878	$1,419	$1,397	$1,366	$1,285	$1,305	$1,289	$1,177	$872	$42	-	-	$32,916
TOTAL ESTIMATED CAPITAL EXPENDITURE ($000s)					**$9,581**	**$7,886**	**$3,215**	**$5,938**	**$7,289**	**$3,805**	**$3,905**	**$10,065**	**$10,693**	**$6,530**	**$2,175**	**$730**	**$406**	**$56**	**$237,847**
CUMULATIVE CAPITAL EXPENDITURE ($000s)					**$175,153**	**$183,039**	**$186,253**	**$192,192**	**$199,481**	**$203,286**	**$207,191**	**$217,256**	**$227,950**	**$234,480**	**$236,655**	**$237,385**	**$237,791**	**$237,847**	

Figure 16-44 Total Estimated Capital Expenditures

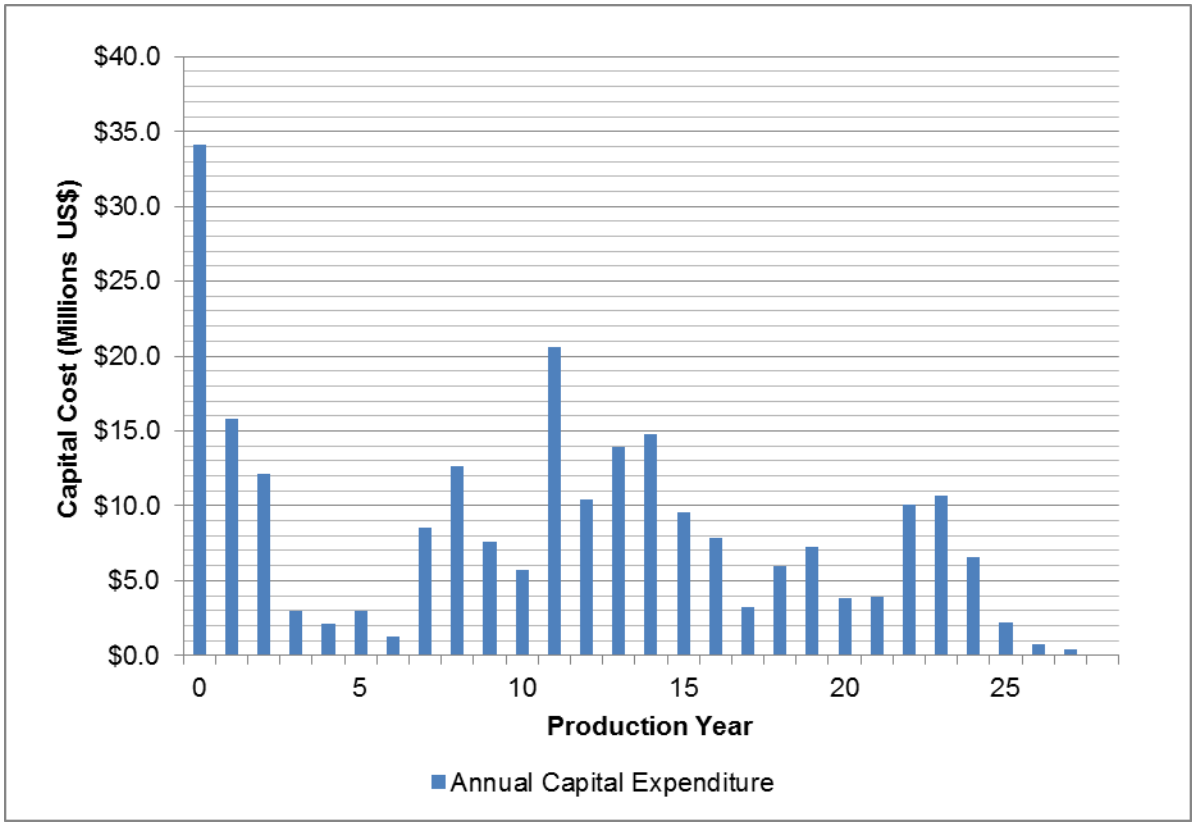


Table 16-26 Unit Cost of Road Construction

<table>
<tr><th colspan="2">Road Purpose</th><th>Overburden Haulage</th><th>Matrix Haulage</th></tr>
<tr><td colspan="2">Truck Payload (tonnes)</td><td>97</td><td>36</td></tr>
<tr><td colspan="2">Driving Surface (Road) Width (m)</td><td>24.5</td><td>14</td></tr>
<tr><td rowspan="2">Sub Grade / Sub Base / Base Course</td><td>Thickness (m)</td><td>1.20</td><td>0.45</td></tr>
<tr><td>Estimated Unit Cost per meter of road built</td><td>$93.64</td><td>$20.19</td></tr>
<tr><td rowspan="2">Surface Course</td><td>Thickness (m)</td><td>0.20</td><td>0.15</td></tr>
<tr><td>Estimated Unit Cost per meter of road built</td><td>$84.72</td><td>$36.31</td></tr>
<tr><td rowspan="2">Total</td><td>Thickness (m)</td><td>1.40</td><td>0.60</td></tr>
<tr><td>Estimated Unit Cost per meter of road built</td><td>$178.36</td><td>$56.50</td></tr>
</table>

Notes:
A 2.1 m^3 bucket backhoe, 36 tonne dump trucks, 45 t/h mobile screening plant, 297 hp grader and 147 hp compactor are assumed to be used for road construction

16.9.3.3 Sustaining Capital

As previously indicated, the sustaining capital includes equipment replacement and rebuild. Equipment replacement and rebuild represents a major component of sustaining capital expenditures. Equipment replacement/rebuild expenditures are necessary to ensure that equipment remains in proper working condition. Equipment was scheduled to be replaced or rebuilt when the estimated operating hours for that particular piece of equipment approached or exceeded the designated machine service life. It is necessary when equipment eventually becomes unserviceable and/or non-functional during the normal course of operations. Where possible, Golder used major equipment rebuilds to extend the effective lives of the excavators, wheel loaders, haul trucks, water trucks, graders, compactors and dozers.

Golder quantified equipment replacement lives in terms of cumulative machine operating hours. Actual operating hours are a function of operating conditions, intensity of equipment use, and basic machine design. Golder based replacement and rebuild intervals operations on experience and available manufacturer/dealer guidelines. Golder's equipment replacement occurred once the cumulative operating hours for an individual equipment unit surpassed or approached the estimated machine life. Estimated rebuild parameters for major equipment are outlined Table 16-27.

Primary equipment requirements for the 1.75 Mtpa Case for this study are listed in Table 16-28. The detailed equipment requirements were previously shown in Section 16.8. The table shows the initial units required for the pre-production period and first year of production, as well as the number of units required throughout the remainder of the mine plan. The additional required units include both additions to the existing fleet and replacement units over the life of mine.

Table 16-27 Summary of Equipment Replacement and Rebuild Parameters

Equipment Description		Service Life	
Equipment Type	**Size Class**	**Machine Replacement Life (Hours)**	**Machine Rebuild Life (Hours)**
Wheel Loader	12.2 m^3 bucket	36,000	18,000
Backhoe	5 m^3 bucket	60,000	30,000
Backhoe	2.1 m^3 bucket	48,000	24,000
Dozer	405 hp	48,000	24,000
End Dump Truck	97 t	60,000	30,000
End Dump Truck	36 t	60,000	30,000
Motor Grader	297 hp	42,000	28,000
Compactor	147 hp	42,000	28,000
Scraper	26 m^3 bed	42,000	28,000

Table 16-28 Summary of Equipment Requirements

Description	Initial Units Required	Additional/ Replacement Units	Total Units Purchased
Primary Mining Equipment			
Wheel Loader - 12.2 m^3 class	5	25	30
Backhoe - 5.0 m^3 class	3	0	3
End Dump Truck – 97 t capacity	13	39	52
End Dump Truck – 36 t capacity	6	6	12
Major Support Equipment			
Backhoe - 2.1 m^3 class	1	0	1
Dozer – 405 hp class	4	14	18
Compactor – 147 hp class	4	18	22
Motor Grader – 297 hp class	2	7	9
Scraper - 26.0 m^3 bed			
Water Truck - 34,000 L tank capacity	2	0	2

16.9.3.4 Non-Cash Costs – Depreciation and Final Reclamation

Non-cash costs include the depreciation charges expensed, in accordance with cost accounting practices, to compensate for the decline in value of capital items over time. A typical depreciation method would be using a straight-line basis over a one-year to 20-year depreciation life. Golder normally derives the duration of the depreciation life from the life span of the equipment and estimates of how the value of the asset would decline over time. Golder depreciated primary mining equipment over a period of 20 years or less as outlined in Table 16-29.

Table 16-29 Summary of Asset Depreciation Lives

Asset Depreciation Life (Years)	Assets Included in Depreciation Class
20	Preparation plant, facilities, and infrastructure
10	Excavators (> 5 m^3), end-dump trucks, water trucks, dozers, fuel and tire service trucks, cranes and forklifts, welding machines.
7	Excavators (< 5 m^3)' wheel loaders, graders, lube trucks, mechanic trucks, pickups, light plants and pumps
5	Shovel rebuilds, excavator rebuilds, end-dump truck rebuilds, water truck rebuilds, dozer rebuilds, supply trucks
3	Grader and wheel loader rebuilds

The final reclamation accrual usually includes costs for re-grading the areas affected by the mining operation to a stable configuration, replacing topsoil, if applicable, and re-vegetation. All facilities constructed to support the mining operation can be assumed raised, and the demolition material hauled offsite for disposal. In addition, some roads can be planned as remaining in place to provide all-weather access to the property or be regarded as the original condition. Costs for long-term monitoring and maintenance are usually included in accrual to provide for any post-closure environmental monitoring and reporting as well as site maintenance for few years after closure.

The accrual rate in most cases will be treated as a liability for the Project and added to the cost summary as a non-cash cost.

Golder included ongoing reclamation costs during the mine life including dozer work for backfill pit re-grading and re-vegetation during mining and backfill of the final pit void. However, final mine closure and infrastructure demolition were not included in the mining cost model and were covered by others.

16.9.3.5 Lease Option

Golder prepared an alternative cost estimate to lease major mining equipment as an opportunity to reduce the initial mining capital. The assumptions for leasing were based on information supplied by the local Caterpillar dealer and recommendation from a company specializing in providing insurance for political risk. The five year payment schedule for equipment leasing is as follows:

- Year 1 of Lease – 20% of equipment purchase price plus 2.5% Arrangement Fee.
- Year 2 of Lease – 20% of equipment purchase price plus 4.2% Interest on outstanding balance Year 1.
- Year 3 of Lease – 20% of equipment purchase price plus 4.2% Interest on outstanding balance Year 2.

- Year 4 of Lease – 20% of equipment purchase price plus 4.2% Interest on outstanding balance Year 3.

- Year 5 of Lease – 20% of equipment purchase price plus 4.2% Interest on outstanding balance Year 4.

Payment was assumed to occur the beginning of the year. The interest rate included 0.72% Libor (cost of borrowing from central bank) and 3.5% margin. The payments were simplified from what would likely be quarterly payments. The equipment was assumed to be paid off and became the property of GB Minerals in Year 5. The Lease Option resulted in a reduction of initial capital cost of about $26 M and an increase in overall cost of about $31 M over the project life to cover the cost of additional fees, interest, and insurance.

16.9.4 Contractor Rate Model

As part of the Study, Golder has requested and received a contractor quote to perform the mining at the site. The quote, provided by NRW, a civil and mining firm headquartered in Belmont, Western Australia, provided the quote based on using similar-sized mining equipment and truck shovel methods through Year 5 of the mining plan. The cost provided included a fixed cost of mobilization and set up of $2.52 M in Year 0 plus another $1.71 M in the first 5 years to ramp up to full capacity. Additionally, a capital cost of $4 M has been included to upgrade the 150 person man camp used for construction activities in preproduction years to a 300 person man camp.

A unit cost of $5.90 per bcm for Year 0 and $4.17 through Year 5 was provided by NRW for waste. NRW provided a cost of $3.01 per dry tonne for matrix mining. Total cost of delivered matrix for the contractor option averaged $39.01/ROM tonne (dry basis) for Years 0 through 5. A summary of the contractor costs is provided in Table 16-30 on the following page.

Table 16-30 Summary of Contractor Cost Model

DESCRIPTION		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
PRODUCTION STATISTICS	**Units**							
Total ROM Production (Dry Basis)	000s tonnes	**-**	**1,750**	**1,750**	**1,750**	**1,750**	**1,750**	**8,750**
Total Product Tonnage (Dry Basis)	000s tonnes	-	1,321	1,321	1,321	1,321	1,321	6,606
Total Stripping Volume	**000s bcm**	**5,818**	**11,172**	**14,922**	**14,318**	**13,079**	**11,898**	**71,206**
Rehandle Volume	000s bcm	-	-	-	-	-	-	-
Total Effective Stripping Volume	**000s bcm**	**5,818**	**11,172**	**14,922**	**14,318**	**13,079**	**11,898**	**71,206**
Waste Cost	$/bcm	$5.90	$4.17	$4.17	$4.17	$4.17	$4.17	$4.31
Matrix Cost	$/tonne	-	$3.01	$3.01	$3.01	$3.01	$3.01	$3.01
Total Waste Cost	$000s	$34,301	$46,553	$62,180	$59,664	$54,500	$49,579	$306,777
Total Matrix Cost	$000s	-	$5,274	$5,274	$5,274	$5,274	$5,274	$26,369
Fixed Costs	$000s	$2,518	$1,710					
Total Cost	**$000s**	**$40,819**	**$53,537**	**$67,454**	**$64,938**	**$59,774**	**$54,852**	**$341,375**
Matrix Cost	**$/ROM tonne**	**n/a**	**$30.59**	**$38.54**	**$37.11**	**$34.16**	**$31.34**	**$39.01**
	$/product tonne	***n/a***	***$40.52***	***$51.05***	***$49.15***	***$45.24***	***$41.52***	***$51.67***

17.0 RECOVERY METHODS

17.1 Process Flow Sheet Selection

The design of the processing facility for this feasibility phase of the Project is based on the metallurgical test work conducted to date combined with industry best practises.

The Farim Composite sample consisted of four subsamples, or drill holes, SB9, SC10, SC11, and SE10 with each subsample further subdivided into several cuts corresponding to sequential drilling depths. The subsample composition was based on the block model and assay model data of the deposit and it was considered representative of at least the first seven years of production of the deposit. After discussion and clarification on the handling and analyses of these subsamples, it was decided to select three cuts of each drill hole (top, middle, and bottom) to be sent for chemical analysis. The selected cuts were analyzed to confirm the block model assay data of the deposit and to determine the main contaminants in the ore for the first seven years of mining. The main impurities were determined to be acid insoluble, Fe_2O_3 (iron bearing minerals), and Al_2O_3 (clays and slimes).

The sample preparation procedure was designed to obtain blended composites of each drill hole: SB9, SC10, SC11, and SE10 proportional to the weight of each cut of the corresponding hole. Initially, each cut of subsample was blended and then split in half. One half of each blended subsample cut was then placed in a plastic bag, sealed and stored as a reserve sample. Approximately 50 kg of reserve samples was preserved, while the remaining half of each cut was used to prepare the composites.

In addition to the individual hole composites, a composite of all the subsamples was blended to represent the Farim Phosphate ore for the first seven years. Thus, five samples were obtained: SB9 Composite, SC10 Composite, SC11 Composite, SE10 Composite, and a general composite, called the Farim Composite. Care was taken during this process to maintain the moisture content of each cut by keeping it in sealed containers after blending and splitting. The prepared samples were also stored in sealed containers.

The flowsheet is developed based on the use of scrubbing and sizing technologies while avoiding the use of grinding and wet high intensity magnetic separation to reduce capital and operating costs for the project. The testwork results have successfully proven that the proposed flowsheet is able to achieve the required product specifications.

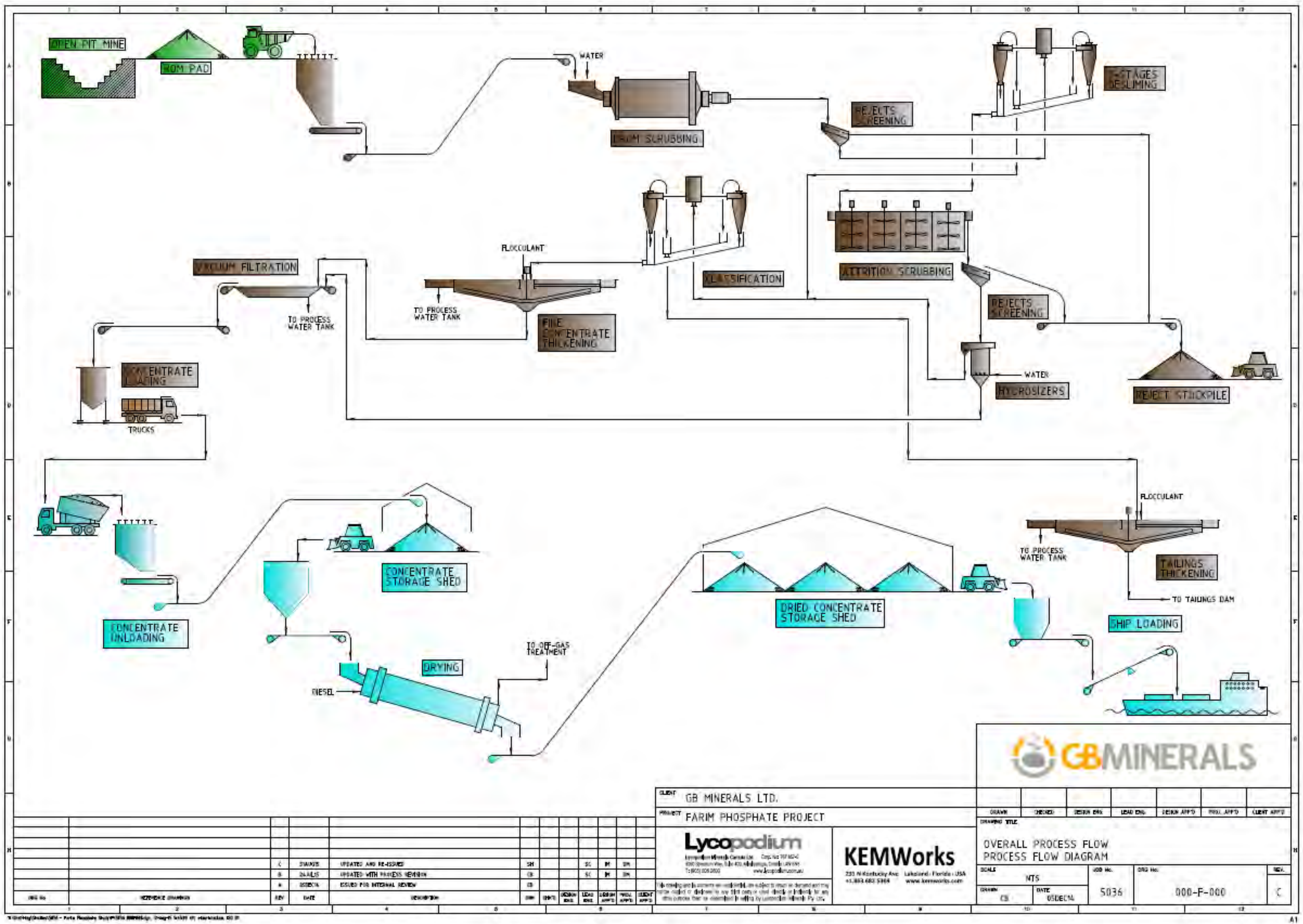
An overall schematic diagram of the process is shown in Figure 17-1 below.

The following steps are included in the selected flowsheet:

- Ore storage and reclaiming of Run-of-Mine (ROM) ore;
- Two stage scrubbing and screening to reject +1.18 mm material;
- Sizing with hydrosizers and cyclones to separate -1.18 x 0.106 mm material to a coarse concentrate, -0.106 x 0.020 mm material to a fine concentrate material, and to reject -20 µm material to tailings;

- Fine concentrate thickening;
- Concentrate filtration, storage and reclaim;
- Thickening and disposal of tailings (reject material) to the Integrated Waste Landform (IWL) and return of decant water to the beneficiation plant;
- Concentrate reclaim, drying and stockpile at Port site; and
- Dried concentrate shiploading at Port site.

Figure 17-1 Overall Process Flow Diagram



17.2 Process (Beneficiation) Plant Description

The process description details the 1.75 Mtpa beneficiation plant for the production of 1.32 Mtpa of phosphate concentrate. A complete set of process flow diagrams can be found in Appendix A.

17.2.1 Feed Preparation

ROM will be delivered by 36 tonne dump trucks from the open pit. ROM will either be dumped directly into the ROM Bin or dumped onto the ROM stockpile. The ROM stockpile will have a five weeks storage capacity equivalent to 175,000 live tonnes.

ROM ore (P_{80} 25 mm) will be dumped by haul trucks or loaded by wheel loaders directly into the ROM Bin. The ROM Bin will be equipped with a static grizzly to prevent oversized rocks from entering the bin. A belt feeder will extract ROM rock from the bin to be conveyed to the horizontal scrubber.

For metallurgical accounting and plant control purposes, weightometers will be installed on the scrubber feed conveyor.

17.2.2 Scrubbing & Sizing

The scrubbing and sizing circuit will include a horizontal scrubber, an attrition scrubber, two reject screens, two-stage desliming cyclones, two hydrosizers, and a classification cyclone cluster and associated equipment.

Blended ore with a F_{80} of 25 mm will be fed into the horizontal drum scrubber feed chute at an instantaneous rate of 219 dry t/h. Process water will be added to maintain a scrubber discharge slurry density of approximately 35% solids. The horizontal drum scrubber will be bi-directional and will be driven by five 90 kW motors. The horizontal scrubber will be 3.6 m diameter by 10.0 m EGL (effective grinding length) and is sized based on 5 minutes of retention time.

The product from the horizontal scrubber will discharge onto a vibrating screen with 5 mm slotted openings to remove +5 mm material. The 1.8 x 3.6 m^2 screen will be fitted with water sprays to remove clay balls and fine slimes from the surface of oversize rocks. The oversize material from the vibrating screen is considered reject and will be conveyed to the reject bin, to be transported off-site. The screen undersize will be deslimed with a two-stage cyclone cluster circuit using a cut point at 75 µm. The overflows of the cyclone clusters will combine with the overflow of the hydrosizers (-106 µm material). The underflow of the secondary cyclone cluster will flow into the attrition scrubber. The attrition scrubber will have four compartments – each 3.8 m^3 in volume to give a total retention time of 5 minutes. The normal retention time required for majority of the ore types is 2.5 minutes for attrition scrubbing hence only two of the four cells will be in operation. In the event where some deposits may require more retention time, all four cells will be in operation. Process water will be added to the underflow of the desliming cyclones to maintain a slurry of approximately 55% solids in the attrition scrubber.

Attrition scrubber products will discharge onto a vibrating screen with 1.18 mm slotted openings to remove +1.18 mm material. The 1.8 x 3.6 m^2 screen will be fitted with water sprays to clean the surfaces of the material and release agglomerated clay, iron and phosphate particles and to produce a 40% solids density undersize. Oversize material from the vibrating screen is considered reject and will be combined with the +5 mm material on a rejects conveyor to be conveyed to the rejects bin. A weightometer will be installed on the rejects conveyor for accounting purposes. Vibrating screen undersize will be pumped to two hydrosizers for additional separation at 0.106 mm.

The hydrosizers will be 3.3 m wide x 3.3 m long x 6.7 m high in size. Hydrosizer underflow at 1.18 x 0.106 mm and 70% solids density will be diluted to 55% solids in an agitated tank prior to being pumped to the concentrate filter feed tank. Hydrosizer overflow at -0.106 mm will be sent to a pump feed tank to be combined with the desliming cyclones overflow from which the material will be pumped to a cyclone cluster for classification at 0.020 mm. The cyclone cluster will consist of ten canisters and each canister will consists of eight 100 mm cyclones. Under normal conditions, 8 canisters will be in operation and 2 canisters will be on stand-by. Classification cyclone underflow at 45% solids will become the 0.106 x 0.020 mm fine concentrate and reports the fine concentrate pump tank for transfer to the fine concentrate thickener. The -0.020 mm cyclone overflow will be rejected as fines and will be sent to the coarse tailings tank.

17.2.3 Fine Concentrate Thickening

The fine concentrate thickening area will include a pump tank, a deep cone thickener and other associated equipment.

Classification cyclones underflow will be collected in the fine concentrate pump tank prior to being pumped to the fine concentrate thickener. Filter cloth wash return water and filter sump pump discharge will be periodically pumped to the fine concentrate pump tank to be re-processed. Underflow will be thickened to 55% solids and then will be pumped to the vacuum filter feed tank. Thickener overflow will flow by gravity back to the process water tank.

17.2.4 Concentrate Filtration & Storage

The concentrate filtration and storage area will include a vacuum belt filter, a product transfer conveyor, a concentrate bin and other associated equipment.

Coarse concentrate and thickened fine concentrate from fine concentrate thickener will be combined in the 22 m^3 live concentrate filter feed tank from which it will be gravity-fed to the 1.6 m wide x 18m long concentrate belt filter. Phosphate concentrate will be filtered to achieve 8% moisture and the filter cake will discharge onto a concentrate filter discharge conveyor. Filtrate will be collected in a filtrate receiver and will be pumped back to the process water tank.

A cloth wash system for washing the vacuum filter belt is included for cleaning of the belt cloth. A sump pump is provided to pump any spillage back to the fine concentrate pump tank feeding the fine concentrate thickener.

The concentrate filter discharge conveyor will transport the filtercake into the concentrate transfer conveyor feed bin. A belt feeder under the feed bin will feed the concentrate filtercake onto the concentrate transfer conveyor, which is equipped with a weightometer for accounting purposes. The concentrate transfer conveyor crosses the Cacheu River and discharges into a 2,000 m^3 live concentrate bin. Concentrate dump trucks, with 31 tonne payload, will drive under the concentrate bin to be loaded for transport to the port facility.

A truck wash station is provided near the entrance of the concentrate storage area to wash the concentrate dump trucks before entering the concentrate storage area. The under body and wheel wash will have a drive-through concept where the trucks will drive through the station at 5 km/h with no stopping required. Trucks will be weighed before and after concentrate loading for accounting purposes.

17.2.5 Tailings Handling

The tailings handling area will include a high rate thickener, an Integrated Waste Landform (IWL) and other associated equipment.

Tailings from the beneficiation plant will be collected in the coarse tailings tank before it will be pumped to the tailings thickener. Tailings will be thickened to 15% solids content and will be pumped to the tailings dam for storage. Tailings thickener overflow will flow by gravity back to the process water tank.

This final tailings stream, at approximately 50% solids, is stored at the IWL. Water reclaimed from the IWL is returned to the process water tank.

17.2.6 Process Plant Sampling

Samplers will be located at different points throughout the plant to monitor process conditions and to perform metallurgical accounting.

A total of six samplers will be installed and they will be located at:

- Scrubber feed conveyor discharge;
- Fine Concentrate;
- Coarse Concentrate;
- Concentrate storage bin feed;
- Tailings storage dam feed; and
- Concentrate ship loadout.

17.2.7 Water Distribution

A total of 3.0 m^3/h of raw water is required as make-up to the beneficiation plant. Raw water makeup at 1.7 m^3/h will be treated in the water treatment plant before it is used as filtered/gland water. The potable water treatment plant will receive 1.3 m^3/h for the production of potable water.

Truck wash water and process water tank overflow, at an approximate combined flow rate of 31 m^3/h, will be sent to an event pond for sedimentation. Event pond return water will be pumped to a water treatment plant for treatment before it will be recycled back to the filtered water tank.

17.2.8 Potable and Gland Seal Water (GSW)

This system will consist of a water treatment plant that treats and filters process water and a potable water treatment plant that sterilize treated water for potable consumption.

Excess process water will be pumped to an 18 m^3 excess water tank. Truck wash water and process water tank overflow will be sent to an event pond for storage. Return water from the event pond will be pumped to the excess water tank. Water from the excess water tank is pumped to be treated in the water treatment plant for the purpose of recycling the water back into the beneficiation plant and eliminating water discharge into the environment. The filtered water storage tank will have 76 m^3 capacity. Two gland water pumps (one standby) will draw gland water from the filtered water tank and distribute it to users throughout the plant. Two filtered water pumps (one standby) will distribute water to the potable water treatment plant and reagent make-up users.

Filtered water is further treated in a potable water treatment plant to maintain the level in the potable water storage tank. The potable water storage tank will have 35 m^3 capacity. Two potable water pumps (one standby) will draw potable water from the potable water storage tank and distribute it to potable water users.

17.2.9 Raw/Fire Water

The raw water system will consist of a raw/fire water tank and two raw water pumps (one standby). Raw water will be primarily used as makeup to the process water tank during start-ups and as required during normal operation.

The fire water system will consist of a raw/fire water tank and a set of three fire water pumps. The raw/fire water tank has a live capacity of 317 m^3 which is equivalent to 4 hours supply of fire water. A fire water pump vendor package will supply fire water to the fire water uses. The vendor package will include an electrical, a jockey and a diesel pumps piped in parallel.

17.2.10 Process Water

The process water system will consist of a mostly closed circulating loop to minimize makeup water requirements. Process water will be used primarily in the scrubbing circuit as dilution water. Two centrifugal pumps (one standby) will deliver process water to users distributed throughout the plant. A process water tank (PWT) with 642 m^3 live capacity will provide 15 minutes of residence time within the process water system. This tank will be replenished by the thickener overflow, tailings dam reclaim and filter filtrate. Excess process water will be sent to the water treatment plant for treatment.

17.2.11 Air Distribution

Compressed air provided to the Beneficiation Plant will be divided into two systems:

Plant air will be supplied by two compressors (one standby) rated at 500 m^3/h each. Compressed air is fed to a plant air receiver, from which plant air is distributed to users throughout the beneficiation plant

Instrument air will be supplied to the beneficiation plant by two compressors (one standby) and two air dryers rated at 500 m^3/h. Instrument air is fed to a plant air receiver, from which instrument air is distributed to users throughout the beneficiation plant.

17.2.12 Reagents

Room is provided in the layout for addition of future reagents, should reverse flotation be added to the process in later years to increase product grades.

17.2.13 Flocculant

There will be two flocculant preparation systems in the beneficiation plant. The two systems are identical in size and operation. They will prepare and store flocculant for addition to the fine concentrate thickener and tailings thickener separately.

Dry flocculant will be delivered in 25 kg bags. The bags will be added to the feed hopper. Flocculant will be pulled from this hopper by an eductor (jet wet mixing system) using filtered water. The initial flocculant mix strength will be 0.5% w/w. Mixed flocculant will be aged in the flocculant mixing tank while being stirred at a low intensity. Once ready, the batch will be transferred to the flocculant storage tank. Parallel standby and duty metering pumps will dose flocculant to the inline mixer where the flocculant will be diluted to 0.05% prior to entering the thickener. The flocculant addition rate will be pre-determined by the measured bed depth level and/or the clarity of the thickener overflow.

A sump pump will be provided in the flocculant preparation and mixing area.

17.2.14 Diesel

Diesel from trucks will be unloaded by a fuel transfer pump into a diesel storage tank with 1,000 m^3 capacity. Diesel will be transferred by a transfer pump from the diesel storage tank to two storage tanks. One of the tanks will be equipped with two fuelling stations to fuel light vehicles and has a capacity of 10 m^3. The other tank will be equipped with two fuelling pumps (one standby) for refuelling of heavy vehicles and has a capacity of 50 m^3.

Any diesel spillages around the diesel area will be collected and transported to an oil/water separator. Separated oil will be disposed and separated water will be sent to site drainage.

17.2.15 Effluents

An event pond will capture all untreated process water and slurry spillage. This spillage will be returned to the plant for treatment as required.

17.3 Port Site Process Description

Filtered concentrate from the beneficiation process plant will be trucked to the Port site to undergo drying before it is loaded onto ships to be transported to market.

17.3.1 Port Concentrate Unloading, Drying & Storage

Filtered concentrate will be delivered by 31 tonne dump trucks from the beneficiation process plant. The filtered concentrate will be dumped onto the concentrate unloading bin. A belt feeder will extract material from the concentrate unloading bin onto a concentrate stockpile conveyor. The concentrate stockpile conveyor will transfer the material to the concentrate stockpile inside a covered storage shed. The covered concentrate stockpile will have a 16 truck loads capacity equivalent to 500 live tonnes.

Dust collectors will be installed at all material handling transfer points to prevent fine concentrate dust from entering the working environment and to minimize product loss. A rotary valve will discharge the collected dust back onto the closest concentrate belt conveyor. Cleaned air will be discharged to the atmosphere.

Filtered concentrate at 8% moisture will be fed into the concentrate dryer feed hopper from which a belt feeder draws the concentrate out from the hopper onto a conveyor equipped with a belt weightometer. The conveyor discharges the material into a feed screw conveyor from which feeds the concentrate rotary dryer. The rotary dryer will be 2.7 m in diameter and 18.3 m in length. Hot air enters the dryer at 600ºC and exits the dryer at 105ºC. Hot air is produced by a burner through the combustion of diesel and air. Dried concentrate will exit the dryer at approximate 105ºC with target moisture of 3%. Dried concentrate from the rotary dryer will discharge onto a sacrificial conveyor and then onto a dried concentrate travelling conveyor. The travelling conveyor will transport the dried concentrate into a storage shed in which the material will be stockpiled until it is time for shiploading. The dried concentrate stockpile will have a live capacity of 60,000 t.

Hot rotary off-gas will be first treated in a dust collector to remove fine entrained concentrate. Cleaned hot gas from the dust collector is fed into a scrubber where raw water is used to reduce off-gas temperature for condensing the moisture in the off-gas. Condensate combined with the scrubber water will be collected in a scrubber seal tank and pumped to the port storm water settlement pond. Cooled scrubber off-gas will be discharged to the atmosphere. Collected fines in the dust collector will be pneumatically conveyed to a ribbon blender for treatment as discussed below.

A truck wash station is provided at the port near the entrance of the concentrate storage area to wash the concentrate dump trucks before entering the concentrate storage area. The under body and wheel wash will have a drive-through concept where the trucks will drive through the station at 5 km/h with no stopping required. Trucks will be weighed before and after concentrate unloading for accounting purposes.

17.3.2 Port Concentrate Loadout

When a ship is berthed, front-end loaders will transfer dried concentrate from the storage shed into three concentrate hoppers. Each concentrate hopper will be equipped with its own belt feeder for a regulated delivery of concentrate onto the port concentrate loadout conveyor. A belt weightometer will be installed on the loadout conveyor to accurately measure the tonnage of concentrate being loaded onto the ships. Reclaimed concentrate from the loadout conveyor will discharge onto the port concentrate shiploader. The port concentrate shiploader will be a traversing shiploader with luffing and shuttling boom.

A centralized dust collector will be installed at all material handling transfer points to prevent fine concentrate dust from entering the working environment and to minimize product loss. A pneumatic conveying system will transfer the collected fines to a ribbon blender where the dust will be mixed with a wetting agent to increase the bulk density of the fine particles and hence preventing the fines from becoming airborne. Treated fines will be collected in a hopper and a belt feeder under the hopper will transport the material to the dried concentrate conveyor. Cleaned air from the dust collector will be discharged to the atmosphere.

The shiploading system will have a nominal capacity of 630 t/h and maximum capacity of 1,200 t/h. Each ship will have a 35,000 DWT (dead weight tonne) of concentrate capacity. To transport an annual concentrate tonnage of 1.32 Mt, a total of 38 shipments will be required each year.

17.3.3 Water Distribution

The raw water requirement in the port area will be approximately 19 m^3/h.

Treated water to discharge will be approximately 24 m^3/h. The destination of the treated water will be determined during the next phase of the Project.

17.3.4 Potable Water

This system will consist of a water treatment plant that filters raw water and a potable water treatment plant that filters and sterilize raw waters for potable consumption.

Raw water is treated in the water treatment plant to maintain the level in the port potable water tank. The port potable water tank will have 17 m^3 capacity. Two potable water pumps (one standby) will distribute water to the potable water users.

17.3.5 Raw Water

The raw water system will consist of a raw water tank and two raw water pumps (one standby). Raw water will be primarily used as truck wash water and feed to the water treatment plant. The raw water tank with 247 m^3 live capacity will provide 24 hours of residence time within the raw water system.

17.3.6 Fire Water

The fire water system will consist of a fire water tank and a set of three fire water pumps. The fire water tank has a live capacity of 317 m^3 which is equivalent to 4 hours supply of fire water. A fire water pump vendor package will supply fire water to the fire water uses. The vendor package will include an electrical, a jockey and a diesel pumps piped in parallel.

17.3.7 Effluent Treatment

The effluent treatment system will consist of a storm water settlement pond, a storm water storage pond, an effluent treatment plant and associated pumps. Effluent from the port area such as site drainage, dirty truck wash water, dryer scrubber water and water treatment plant effluent will be sent to the settlement pond. Overflow from the settlement pond will be sent to the storage pond. Clean water from the oil/water separator will also report to the storage pond. Water from the storage pond will be pumped to the effluent treatment plant. Sludge produced by the effluent treatment plant will be disposed. Treated effluent from the effluent treatment plant will be discharged to the environment.

17.3.8 Compressed Air

Compressed air will be supplied to the port site by one compressor and one air dryer rated at 500 m^3/h. An air receiver will be utilized for compressed air distribution within the port areas.

17.3.9 Diesel

Diesel from trucks will be unloaded by a fuel transfer pump into one of two port diesel storage tank with 600 m^3 capacity each. Diesel will be transferred by a transfer pump from the diesel storage tanks to the light vehicle diesel storage tank, rotary dryer diesel day tank and power plant day tank.

The light vehicle storage tank will have 20 m^3 capacity and will be equipped with two fuelling pumps (one standby) for refuelling of vehicles. The rotary dryer diesel day tank will have a capacity of 36 m^3.

Any diesel spillages around the diesel area will be collected in a sump pump and pumped to an oil/water separator. Separated oil will be disposed and separated water will be sent to the storm water storage pond.

17.4 Process Control Philosophy

17.4.1 General Common Controls

The individual plant areas are described in the following sections. A number of elements in the beneficiation plant will share common controls. Some of these common controls are described below and the rest will be mentioned in their individual sections.

17.4.2 Agitators

Agitators in tanks will be controlled by amperage and/or torque to a manual RPM set point.

17.4.3 Bins

Bins will be equipped with level detectors. These will be either microwave or proximity type depending on the contents and dimensions of the bin.

17.4.4 Chutes

Chutes will be equipped with level detectors to indicate blockage, with high level switches, where appropriate. These level detectors will generally be interlocked with the upstream equipment feeding into the chute.

17.4.5 Screens

Spray water addition will be controlled manually. Operator will be able to control screen vibration amplitude, frequency and amperage manually.

17.4.6 Sump Pumps

Level instrumentation and controllers will activate the sump pumps under automatic control on high-level and deactivate on low-level detection. An alarm will be displayed on ultimate high level (High-High).

17.4.7 Slurry Pumps

Slurry pumps drawing from pump boxes will be equipped with manual on/off valves on pump suction and discharge lines. Manual water purge valves will be included on all critical slurry lines for line flushing on shut-down.

17.4.8 Pump Boxes

Level indication and controllers will control level by either adjusting the pump motor speed or by controlling the rate of water addition to the pump box.

17.4.9 Pump Gland Water

Slurry pumps will be fitted with stuffing boxes that require gland water. Controls for pump gland water will include manually adjusted flow regulators. There will be no interlock between seal water and pump motor, therefore under low flow conditions, the operator will manually adjust the flow valve.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

The project consists of an open pit mine, drum scrubber, attrition scrubber, classification cyclones, hydrosizer, concentrate thickener and filter, tailings thickener, transfer conveyer to transport concentrate across the Cacheu River, and truck loadout. The product is then trucked 75 km to the port of Ponta Chugue, where it is unloaded, conveyed through a rotary dryer, stockpiled, and conveyed via shiploader to direct load 35,000 DWT ships.

Both facilities in Farim and Ponta Chugue will produce their own power via diesel generating sets.

18.2 Site Plan

The mine site is located approximately 5 km west of the town of Farim. The mine site is bound by the Cacheu River to the east and south of the open pit. The beneficiation plant has been located between the southern and northern open pits, adjacent to the Cacheu River. The plant area, including site buildings, is approximately 200 m x 200 m. The beneficiation plant is located at the narrowest point of the Cacheu River, where it is approximately 150 m wide, to minimize the cost of the conveyor crossing. A conveyor is utilized to transfer dewatered phosphate rock into a storage bin on the east side of the river, where trucks are loaded. A 2 km gravel road will be constructed to connect the truck loading facility to the new highway to the east.

The Integrated Waste Landform (IWL), a tailings co-disposal facility, is located approximately 5 km west of the process plant. For overburden storage, two waste dumps (WD’s) are located west of the process plant and between the southern and northern open pits. The WD locations have been chosen to minimize haul distances, thus reducing capital and operating costs.

Refer to Figure 18-1 for the Farim Site Plan, and Figure 18-2 for a 3D view of the process plant.

Figure 18-1 Farim Site Plan

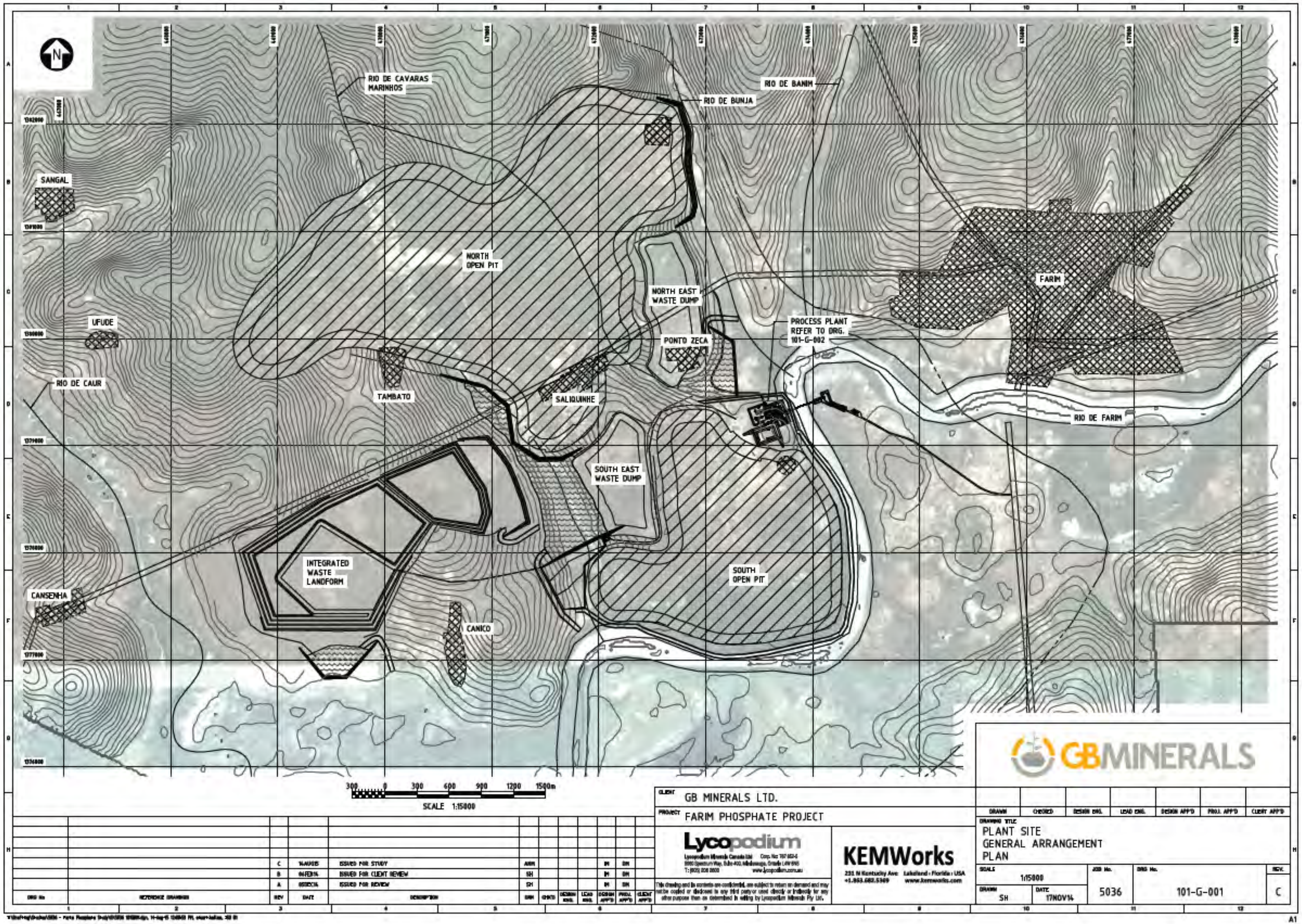


Figure 18-2 Farim Process Plant Pictorial View

Refer to Figure 18-3 for the Ponta Chugue Site Plan, and Figure 18-4 for a 3D view of the drying and storage facilities.

Figure 18-3 Ponta Chugue Site Plan

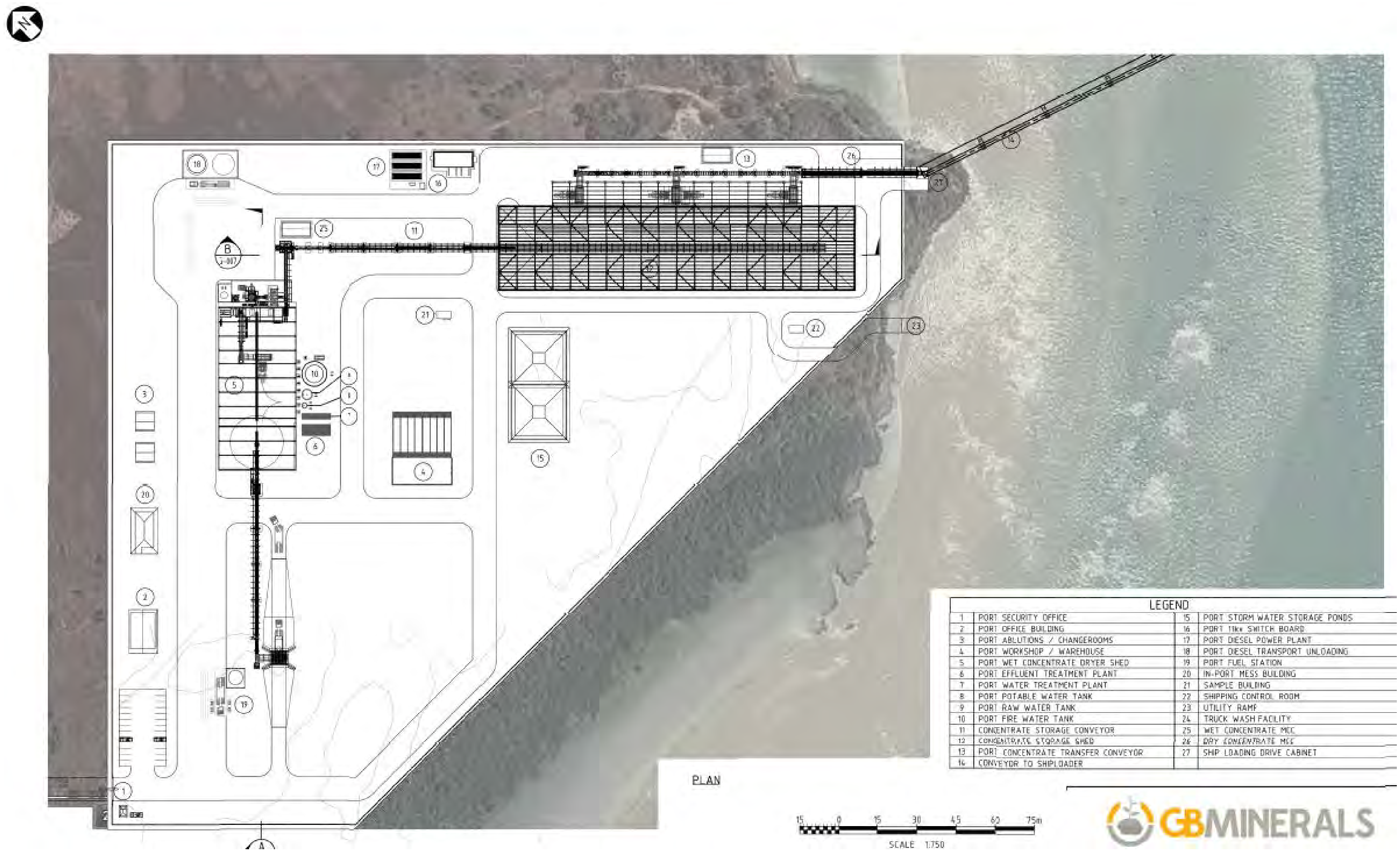


Figure 18-4 Ponta Chugue Pictorial View

18.3 Site Roads

The Farim and Ponta Chugue onsite and offsite roads will be constructed of crushed waste rock from existing quarries in Guinea-Bissau and from any naturally available materials. The onsite roads have been designed to connect the various process plant and port facility areas for operation and maintenance. At Farim, the offsite gravel road is approximately 2 km in length and connects the truck loadout facility to the new paved highway leading to Ponta Chugue. At the port in Ponta Chugue, the offsite gravel road is approximately 4 km in length and connects the port facilities to the paved highway.

See Figure 18-5, showing the new paved highway from Farim to Mansoa, in excellent condition.

Figure 18-5 New Highway from Farim to Mansoa

18.4 Site Power Supply

18.4.1 Power Supply

Power supply to the Farim plant site and the Ponta Chugue Port Facilites will be from a Diesel Onsite Power Plant (OPP). At Farim, the power plant will supply a Main HV switchroom inside the processing plant from which power will be distributed at 11 kV.

The configuration of the Farim Plant OPP is:

- 4 x 1.2 MW prime rated 11 kV generators (3 duty, 1 standby).
- 11 kV switchroom.

The configuration for the Port OPP is:

- 3 x 0.5 MW prime rated 0.4 kV generators (2 duty, 1 standby).
- Direct feed to the LV switchroom.

18.4.2 Electrical Distribution

The electrical system for the Project is based on 11 kV distribution and 400 V working voltage.

The 11 kV supply will be stepped down from 11 kV to 400 V at Motor Control Centres (MCC) via four separate 11 kV / 433 V distribution transformers. The LV switchrooms in the process plant area will house one LV MCC each, with the Plant Services & Reagents Switchroom holding two. These will be fed from an 11 kV feeder via an underground power cable. Power supply to the MSA will be fed from an 11 kV feeder via 11 kV underground power. Outdoor control panels and distribution boards have been allowed for plant lighting and small power distribution and UPS power distribution.

18.4.3 Installed Load and Maximum Demand

The installed load and maximum demand is shown in Table 18-1 for the Farim plant and Table 18-2 for the Ponta Chugue port facilities.

Table 18-1 Plant Power Demand

Plant Installed Load	Plant Maximum Demand	Plant Average Continuous Load
4800 kW	2982 kW	2502 kW

Table 18-2 Port Power Demand

Port Installed Load	Port Maximum Demand	Port Average Continuous Load
1500 kW	961 kW	769 kW

18.4.4 Electrical Buildings

The Main HV switchboard and all other plant electrical buildings will be prefabricated switchroom buildings:

- Three LV Switchrooms for the beneficiation plant;
- One LV Switchroom at the port;

- Plant and Port Control Rooms.

These electrical buildings will be installed with air-conditioners and sealed to prevent ingress of dust.

18.4.5 Transformers and Compounds

All the 11 kV / 433 V distribution transformers (1.6 MVA, 0.75 MVA, 0.5 MVA, 0.05 MVA) will be of ONAN cooling configuration and vector group Dyn11.

Fire rated concrete walls will be constructed around the pad mounted transformers.

An outdoor rated 11 kV / 433 V outdoor kiosk substation will be used to provide power to the mine services area.

18.4.6 11 kV Switchboards

The 11 kV switchboards will be fully withdraw-able design complete with protection, metering and earthing facilities.

The design fault level and circuit breaker ratings adopted are:

- 11 kV switchboard busbar 1,250 A, 40 kA at 3 sec.
- 11 kV circuit breakers 630 A.

Protection will be provided by microprocessor based protection relays.

18.4.7 Electronic Variable Speed Drives and Soft Starters

LV variable speed drive (VSD) units and soft starter (SS) ratings range from 2.2 kW up to 315 kW. These are floor or wall mounted (dependent upon size) along the internal wall of the LV substation.

18.4.8 400 V Motor Control Centre

The LV MCC’s will be double-sided (back to back) and housed in the LV switchroom. Construction of all MCC’s will have Form 4 segregation, Type 2 coordination. Starters in MCC’s will be of demountable design and main incoming circuit breakers will be of withdraw-able design complete with protection. All motor starters will be equipped with smart overload relays as per Lycopodium engineering standard. The LV MCC’s will supply power to the low voltage motors, low voltage variable speed drives and low voltage distribution boards.

18.4.9 Earth Fault Protection

Earth leakage protection will be applied to circuits with General Purpose Outlets (GPO’s, i.e. power points) and for lighting circuits.

18.4.10 Fire Protection

The HV switchroom, LV switchroom and the plant and port control rooms will be provided with fire detection systems. Signals from the fire detection system will be wired to the respective fire indication panel (FIP) in the switchrooms and all signals will be monitored by a master fire detection panel (MFIP) in the security / emergency services control room in the corresponding Administration Buildings. Each FIP will also be wired to a local siren with beacon to warn staff of the fire detection. The same fire and smoke activation alarm signals detected by the fire detection system will also be monitored in the plant and port control rooms.

18.4.11 Cable Ladders

Cable ladders will generally be laid horizontally, with vertical ladders used in areas where spillage may occur.

Cables of different voltage groups will be installed on separate ladders. If they need to be installed on the same ladder, then complete segregation of the ladders will be provided. Ladder routes will follow the mechanical pipe racks.

18.4.12 Cables

Direct buried cables will be provided with armouring.

Cables up to 16 mm^2 will be PVC insulated and bigger cables will be XLPE insulated.

VSD cables will be multiple core 3 x phase and 3 x earth cables symmetrically laid out within an overall shielded cable.

Cables within the plant and port areas will be installed above ground, on cable ladders and follow the mechanical pipe racks wherever possible.

18.4.13 Lighting

All lighting around the beneficiation plant and port is designed in a fit for purpose manner to suit the operational requirements for each area.

18.4.14 Earthing System and Lighting Protection

The following method of system earthing will be implemented at various voltage levels:

11 kV	Earthed via earthing transformers
415 V	Solidly earthed system / Multiple Earthed Neutral (MEN) / T-N-C-S
Note:	T – Terre (French for earth) N – Neutral C – Combined S - Separate

Lightning protection will be provided for all plant and port building structures. Plant and port substations/switchrooms and structural high points will be fitted with lightning masts of sufficient height and quantity to ensure that all exposed points will be covered. Lightning protection systems will have their own independent earthing electrodes and will be interconnected with the power earthing system.

18.5 Process Water

The process water system will consist of a mostly closed circulating loop to minimize makeup water requirements. Process water will be used primarily in the scrubbing circuit as dilution water. Two centrifugal pumps (one standby) will deliver process water to users distributed throughout the plant. A process water tank (PWT) with 642 m^3 live capacity will provide 15 minutes of residence time within the process water system. This tank will be replenished by the thickener overflow, tailings dam reclaim and filter filtrate. Excess process water will be sent to the water treatment plant for treatment.

18.6 Raw/Fire Water

The raw water system will consist of a raw/fire water tank and two raw water pumps (one standby). Raw water will be primarily used as makeup to the process water tank during start-ups and as required during normal operation. Raw water is pumped to various users throughout the beneficiation plant, including the reagent area and all pump gland seals.

The fire water system will consist of a raw/fire water tank and a set of three fire water pumps. The raw/fire water tank has a live capacity of 317 m^3 which is equivalent to 4 hours supply of fire water. A fire water pump vendor package will supply fire water to the fire water uses. The vendor package will include an electrical, a jockey and a diesel pumps piped in parallel. Level controls will assure that the level of the tank does not fall below the fire water minimum.

18.7 Potable Water

At both Farim and Ponta Chugue, fresh water will be supplied by local wells and pumped from the raw/fire water tank through a reverse osmosis unit to produce potable water for drinking. The potable water storage tank in Farim will have a 35 m^3 capacity, and the one in Ponta Chugue will have a 17 m^3 capacity. Two potable water pumps (one standby) will draw potable water from the potable water storage tanks and distribute it to potable water users for drinking, cooking, showers, and emergency eyewash stations throughout the corresponding facilities at Farim and Ponta Chugue. The reverse osmosis concentrate is pumped to a local area sump and periodically pumped back into the process circuit.

18.8 Tailings and Return Water System

The tailings delivery system will transport slurried tailings from the beneficiation plant to the Tailings Storage Facility (TSF) which is inside the Integrated Waste Landform (IWL), see Figure 18-1. The tailings delivery system will consist of tailings thickener underflow pumps, and a 100 mm diameter HDPE (high density polyethylene) pipeline, approximately 5 km long. The return water delivery system will pump recycle water from the TSF to the process water tank. The system will consist of barge pumps and a 100 mm diameter HDPE pipeline, 5 km long, which runs adjacent to the tailings pipeline.

18.9 Farim Plant Site and Administration Buildings

A single-storey administration building, 23 m x 12 m, will be located near the main site entrance gate. The building will have a reception area, offices, meeting rooms, a main conference room, medical clinic, kitchenette and washrooms. The offices are for managers, engineers, geologists, and clerks. A parking lot and transport and pick-up area is located adjacent to the administration building.

A laboratory, 12 m x 5 m, will be used to test metallurgical accounting samples from the beneficiation plant, mining and exploration operations.

A plant kitchen and dining hall, 18 m x 8 m, will include a seating area for up to 80 people with overhead fans, kitchen, and food storage.

The two plant change house and ablutions buildings, male and female, will be 8 m x 7 m. They include separate male and female showers, bathrooms, and change room with lockers.

The combined plant workshop/warehouse, used to store and maintain equipment and parts, will be 38 m x 24 m. The workshop/warehouse will house mechanical, electrical, instrumentation and general items. Internal offices will be supplied adjacent to the warehouse for warehouse and maintenance staff.

A main security gatehouse as well as a separate beneficiation plant security gatehouse will be included.

18.10 Port Site and Administration Buildings

A single-storey administration building, 15 m x 10 m, will be located near the port site entrance gate. The building will have a reception area, offices, meeting rooms, a main conference room, medical clinic, kitchenette and washrooms. The offices are for managers, engineers, and wharf personnel. A parking lot and transport and pick-up area is located adjacent to the administration building.

A wet concentrate shed, 109 m x 21 m, where the product will be unloaded and dried to 3% moisture.

A dry concentrate shed, 150 m x 36 m, where final product will be stored for shiploading. On ship arrival, the product will be unloaded via front end loaders onto a conveyor feeding the shiploader.

A combined shipping control room and sample building, 6 m x 3 m, will be used to check product moisture levels and store shipping records.

A port kitchen and dining hall, 8 m x 6 m, will include a seating area for up to 20 people with overhead fans, kitchen, and food storage.

18.11 Mine Truck Shop

The truck shop will service the mining fleet and include the necessary maintenance service bays as well as an office for supervision and planning. Initially, 4 truck bays will be required, but will be progressively expanded to 6 bays as the fleet requirement increases. The building will be a steel structure with metal cladding and a concrete slab on grade. A tire yard will be located adjacent to the truck shop.

18.12 Communication Systems

An integrated voice and data network infrastructure will be provided in the beneficiation plant and port. Telephone and voice mail system will provide voice functionality via this network. This system will be linked to the main telephone switchboard for connection to outside lines. Radio sets will be provided for operations personnel.

18.13 Ventilating and Air Conditioning (HVAC) Systems

The ancillary buildings will require varying degrees of air conditioning and ventilation. The beneficiation plant facility will be entirely outdoors, and only the main control room and electrical switch rooms will be air conditioned. The drying and storage buildings at the port will be ventilated only. The administration building, laboratory building, meals area, change house and gatehouses will be air conditioned. Ancillary buildings will require varying degrees of ventilation and air conditioning. Exhaust fans will be used to provide ventilation of the washroom areas.

18.14 Building Fire Protection Systems

Systems to be provided for personnel and property protection include: smoke/heat detectors and manual pull stations, fire extinguishers, fire hydrant coverage of all beneficiation plant area buildings, and internal fire hose coverage for all enclosed building areas.

Fire hose cabinets and external fire hydrants will be located so that all interior areas of the buildings are within reach of a fire hose stream.

A firewater header system will be provided at the Farim site and will cover the administration building, beneficiation plant and ancillary buildings, along with fire hose coverage throughout the facility, supplemented by hand held fire extinguishers. A separate stand pipe system will be installed to provide fire hose coverage throughout the reagent area, with hand held fire extinguishers. A firewater header system will also be provided at Ponta Chugue, and will cover the drying building, product storage buildings, shiploading, and wharf areas, supplemented by hand held fire extinguishers. For electrical rooms ionization type smoke detectors will be provided, with hand held fire extinguishers.

Hand held fire extinguishers will be provided for the control rooms.

18.15 Waste Disposal

Solid wastes will be disposed of in a manner complying with local regulations. Allowable products will be disposed of in a solid-waste landfill constructed on site. Products not allowed to be disposed of in the landfill will be transported to appropriate facilities off site.

A septic system will be utilized for sewage disposal at both facilities in Farim and Ponta Chugue. Septic tanks will be located at the Farim beneficiation plant, and near the product storage facilities in Ponta Chugue. The septic tank sludge will be removed by vacuum truck at regular intervals.

18.16 Fuel Supply

Diesel fuel will be shipped to the Port of Bissau which is approximately 18 km west of Ponta Chugue. From the Port of Bissau, diesel fuel will be transported via trucks to diesel fuel storage tanks at both Farim and Ponta Chugue. The estimated diesel fuel usage is approximately 3 Million litres/month. The diesel fuel is required primarily for the mining fleet, rotary dryer, and the power generation plants.

The diesel storage tanks will be above ground, designed per API 650 (American Petroleum Institute) standard, and inside a secondary containment berm. Both facilities will be equipped with fuel dispensing systems located on site.

18.17 Site Security

All entrants to both the plant and port sites will need to pass through the security guardhouse located at the front gates. The entrances to both sites, as well as the Farim plant and Ponta Chugue port facilities, will be fenced with chain-link security fencing.

18.18 Marine Design at Ponta Chugue

Two marine export scenarios were investigated at Ponta Chugue: direct loading onto bulk carriers and transhipment from barges to bulk carriers. Both options were developed with the intent of minimizing CAPEX. As a result, additional operational, safety, and maintenance measures were implemented when compared to a traditional wharf system. A high level description of each scenario is provided below.

18.18.1 Direct Load Scenario

Under the direct load scenario, bulk carriers navigate the River Geba 60 nautical miles to the project site at Ponta Chugue for loading at a fixed wharf. The loading wharf consists of a trestle and infrastructure to support one radial telescoping shiploader. An example of a similar direct loading facility (sans mooring buoys) is shown in Figure 18-6 below.

It is noted that the proposed minimum CAPEX configuration has drawbacks with respect to operational efficiency, safety (requires warping/shifting vessel), and maintainability.

Figure 18-6 Example of Direct Loading Facility

18.18.2 Barge Transshipping

Under the transshipping scenario, barges are loaded at Ponta Chugue via a nearshore loading wharf. Laden barges are then stored at a barge marshalling area prior to being transported by tugs to a deep water transshipping location in the River Geba.

At the transshipping location, bulk carriers arriving from the ocean are moored at a dedicated single point mooring (SPM), which allows them to swing freely with the prevailing current and wind conditions. After bulk carrier mooring, barges are berthed alongside and unloaded with the ship's gear (cranes aboard the ship). An example of the intended transshipping operation is shown in Figure 18-7.

Figure 18-7 Example of Transhipping Operation

18.18.3 Conceptual Design of Port

Detailed conceptual designs and corresponding CAPEX and OPEX estimates were created for the marine export scenario described above. Comparison of the estimates revealed that the transshipping scenario was considerably more expensive. As a result, the direct load scenario was selected by GB Minerals. The remainder of this section describes the major components of the direct load scenario.

18.18.4 Possible Navigation Route

Direct loading requires navigation of the River Geba 60 nautical miles to Ponta Chugue by bulk carrier. Establishing that a safe navigation route exists to access Ponta Chugue and the proposed transhipment locations is of critical importance for the feasibility of the export project. Previous study of the navigation route by Baird in 2012 (Baird, 2012a) included:

1. A literature search pertaining to the existing navigation of the Geba estuary (Geba Channel).

2. Interviews with local administrators and pilots from the Administration of Ports of Guinea-Bissau (APGB) and Portline Transportes Marítimos Internacionais, S.A. (Portline) based in Lisbon, Portugal, currently sailing to Bissau.

3. A hydrographic survey by Coastal Consulting & Exploration (CCE), covering the project site at Ponta Chugue and a length of the River Geba from Ponta Chugue to Banco do Alenquer.

A summary of the information obtained in 2012 related to possible navigation routes follows below.

Literature Search

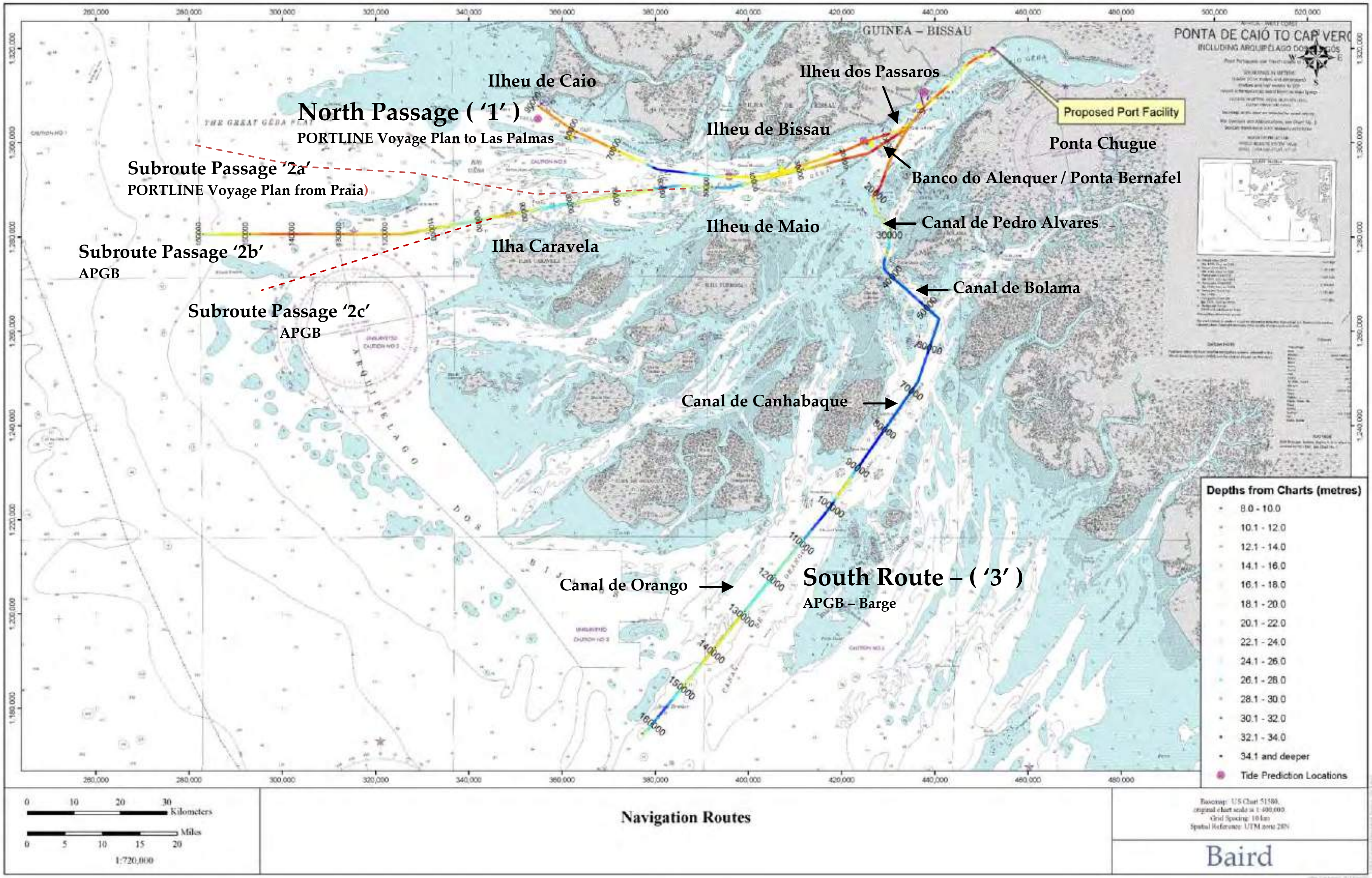
Gathered documents describing the existing conditions in the Geba estuary include UKHO Admiralty Chart No. 1724 &1726, NIMA Chart No. 51580, NIMA sailing directions, Instituto Hidrografico Portugal (IHP) charts, and shipping voyage plans.

The NIMA sailing directions indicate that the usual channels of approach to the River Geba are the Canal de Caio on the North side of the estuary and the Canal de Orango on the South side (Route 1 and Route 3 respectively in Figure 18-8 below). However, the Canal de Caio is the only passage recommended for navigating deep-draft vessels to the Port of Bissau.

In addition to Routes 1 and 3, various sub-passages between shoals following natural channels to the inner reach of the estuary are also indicated on the charts and NIMA sailing directions (Subroute Passages No. ‘2a’, ‘2b’, ‘2c’ in Figure 18-8 below).

After review of the information gathered in 2012, navigation sub-routes 2a, 2b, 2c and 3, shown in Figure 18-8, were discounted from further consideration due their shallow charted depths and general lack of information. Hence, the north route (Route 1 in Figure 18-8) to Bissau via the Canal de Caio was assumed as the main route for the Direct Load scenario.

Figure 18-8 Possible Transit Routes (Chart Depths Based on 1949-1969 Survey)



Administration of Ports of Guinea-Bissau (APGB) and Portline, SA

In addition to the documentation that was gathered regarding navigation, several discussions and meetings were held with the Bissau Port authorities during a site visit in February 2012. Salient points from those meetings include:

- APGB personnel identified similar sailing routes as those indicated in the NIMA sailing directions.
- According to the office of Marine Services, navigation through the River Geba Channel requires tidal assistance for ships that have drafts larger than 8m and are subjected to strong current. Squalls might affect navigation. Visibility is generally good, although low visibility events have been reported.
- The most recent dredging in the area was done in 1974.
- Official bathymetric information in the country is old and unreliable. Entering Bissau Port requires a pilot who uses his own directions, although it is common among captains to avoid the pilot directions since they are not reliable. There is no radio communication with pilots (request for pilot has to be done by the local shipping agent); vessel lifeboats have to be used to ship pilots in and out. Pilots do not speak English.
- There are few reliable navigational aids. There is only one buoy (ISO 4s) in position 11 48.9N 01622.4w, very close to the Ponta de Caio light house. Lighthouses in Ponta de Caio and Ponta Arlete are working with a range of about 6.5 km in normal visibility conditions.
- Night berthing at Bissau Port is not recommended, and tugs are not available.

An additional source for shipping information was the Portuguese company Portline, who operates a regular service to Bissau every two weeks. Salient points from conversations with Portline representatives include:

- 8.5 m maximum vessel draft is safely practicable to approach, transit, and pilot after mid-flood tide or before mid-ebb tide.
- Pilots confirmed two of the four passages indicated in the NIMA Sailing Directions; Route 1 and 2a.

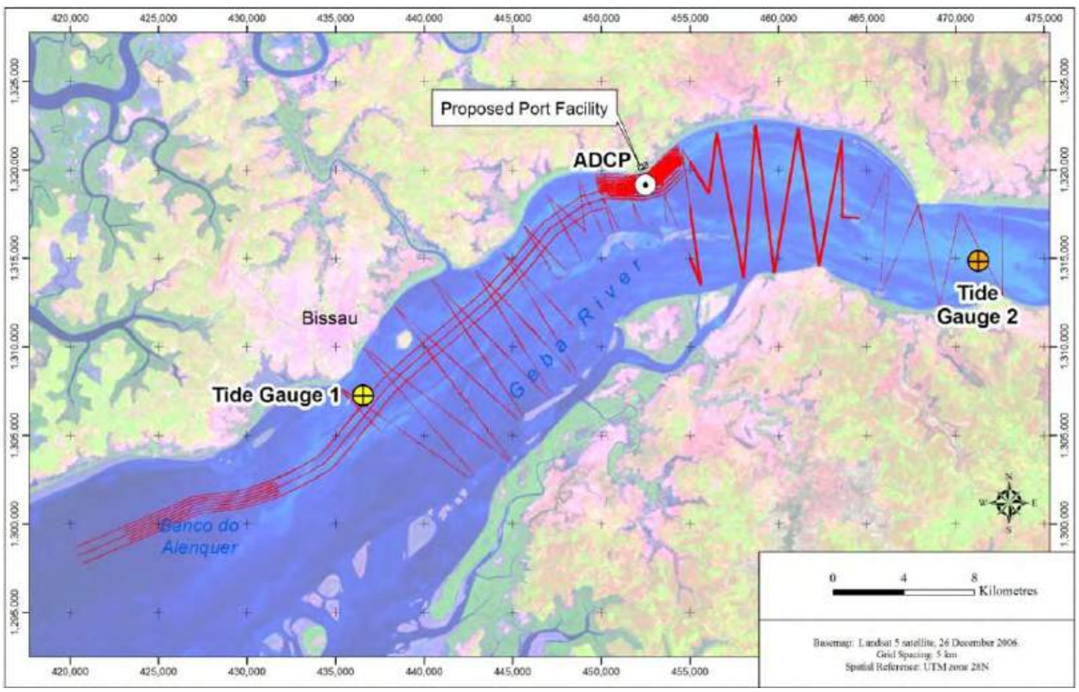
Hydrographic Data

New hydrographic data were not collected for this study. Instead, hydrographic data from the charts listed above, supplemented by a small set of historical hydrographic survey data, were utilized to understand the depth of the river.

The published navigation charts are based upon surveys completed between 1949 and 1971. From the available literature (IHO 2002, 2004 & 2010), it is understood that updated chart and maritime safety information was regularly sent to the Instituto Hidrografico of Portugal (IHP) until 1988. Since then, no maritime safety information was provided to IHP by Guinea-Bissau.

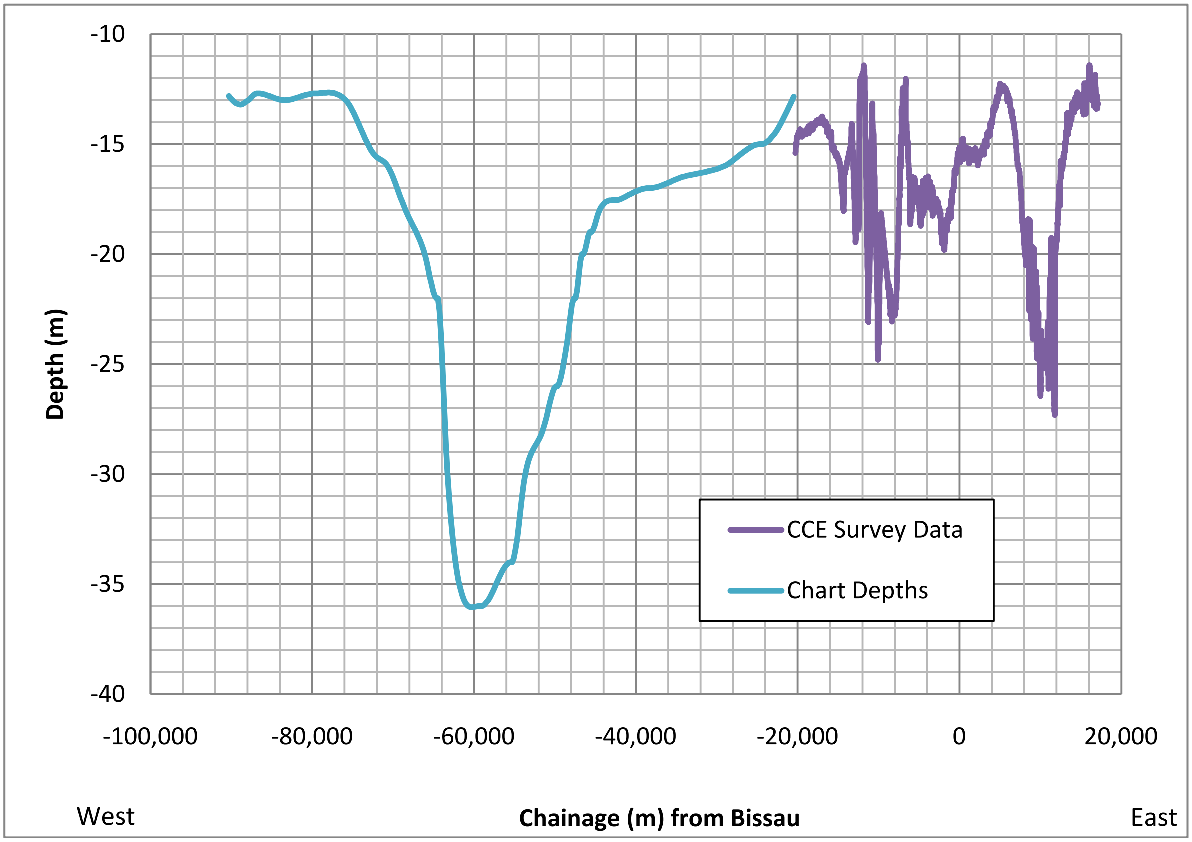
The hydrographic survey taken by CCE in 2012 consists of detailed coverage at Ponta Chugue and several lines from Ponta Chugue to Banco do Alenquer (Figure 18-9). Survey data were not gathered west of the approach to Bissau, hence a significant length of the navigation route does not have recent survey coverage (a project risk).

Figure 18-9 Hydrographic Survey and instrument locations by CCE during March-April 2012



The available depth along the north navigation route from the ocean to Ponta Chugue varies considerably (see Figure 18-10), as does the tide, which ranges between 3 m at the most eastern end of the Canal de Caio and 6 m near Ponta Chugue. Due to the limited water depth at constriction points, most laden bulk carriers in the world fleet would need to take advantage of the tide to navigate out to sea via the North Passage.

Figure 18-10 Hydrographic Data

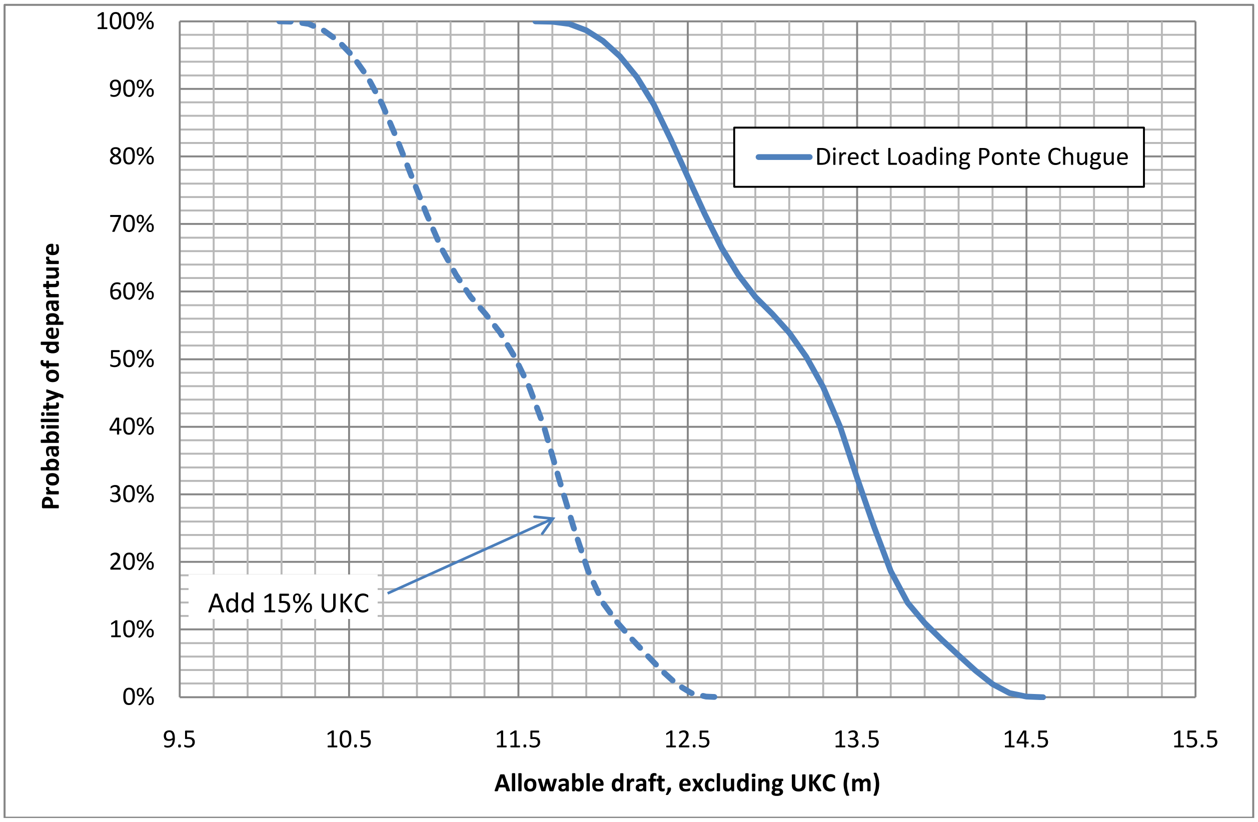


18.18.5 Tidally Assisted Transits

Baird's tidal assisted vessel departure model (TWCALC) was used to assess allowable vessel departure draft. The model predicts the time at which a vessel will pass particular points on its outbound transit route, given a specific departure window, and checks the water level at that point to verify that sufficient depth exists. This process is completed for an entire year, assuming departures at 10 minute intervals.

Output from the analysis, depicted in Figure 18-11, shows the probability that a vessel having a particular draft with an under keel clearance allowance of 15% will be able to depart given the existing hydrographic information. Note that vessels having laden drafts less than or equal to 10.1 m (generally vessels <= 35k DWT) can depart from Ponta Chugue 100% of the time, meaning that they are not tidally constrained. On the other hand, vessels having drafts greater than approximately 12.5 m can never leave Ponta Chugue fully loaded without grounding. Vessels having drafts between 10.1 m and 12.5 m will have to await an appropriate tidal window for departure.

Figure 18-11 Tidal window analysis results – Allowable draft for departures



18.18.6 Bulk Carriers

It is recommended that the direct load option utilize 30-35k dwt vessels, which generally should not require tidal assistance based on the available hydrographic information.

While larger vessels could access Ponta Chugue utilizing the tide, we recommend a conservative approach herein, utilizing vessels that are not tidally restricted, in light of:

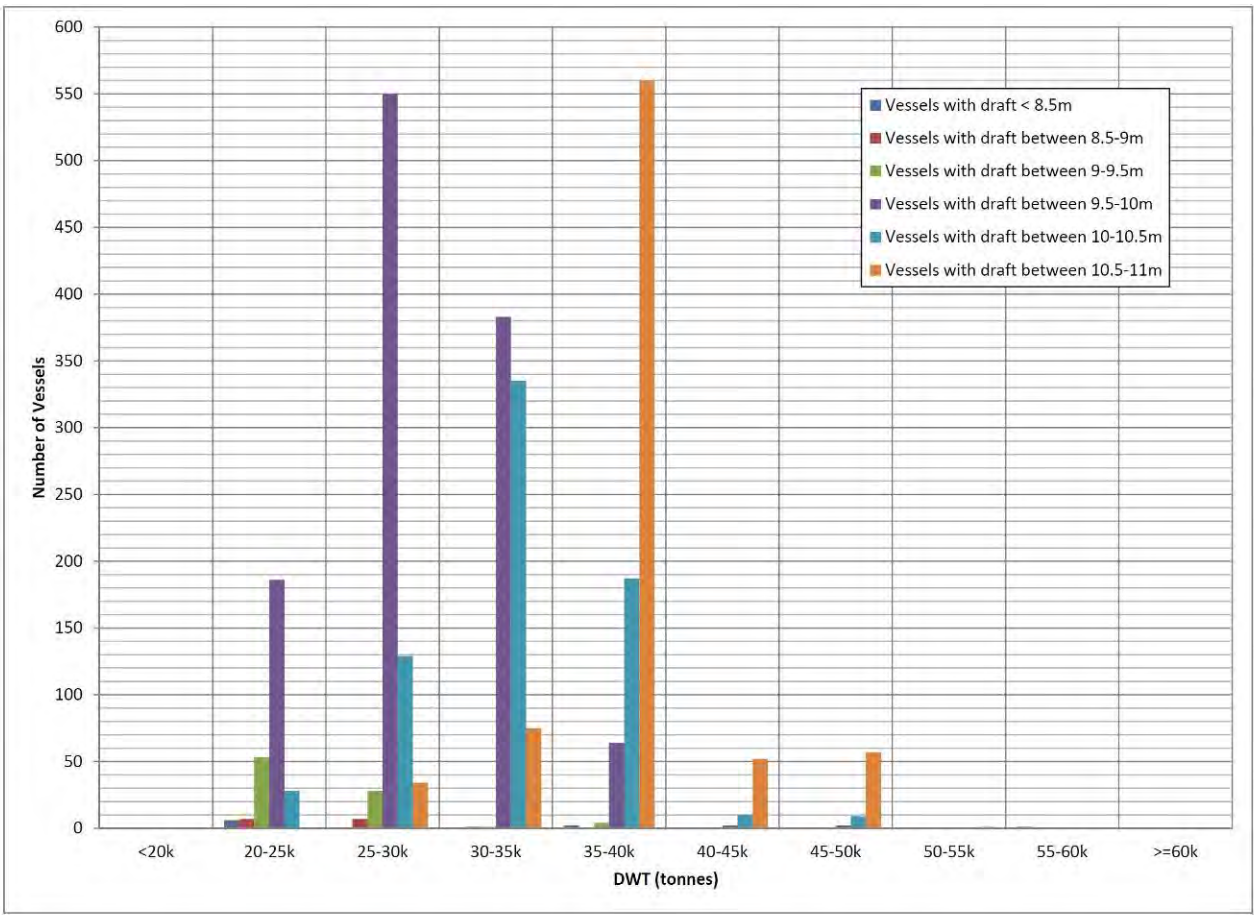
- The lack of survey information from Bissau to open ocean.
- Information gathered from APGB and Portline, which indicates vessels having drafts between 8 m and 8.5 m currently require tidal assistance to access the Port of Bissau.
- The remoteness and limited nature of infrastructure in Bissau.

Figure 18-12 shows that there are approximately 380, 30-35k DWT bulk carriers in the world fleet having drafts less than 10 m (not tidally restricted). Another 330 vessels exist having drafts between 10 and 10.5 m. Figure 18-11 above indicates that vessels having drafts less than or equal to 10.5 m will encounter a tidal condition that allows for departure 94% of the time. As such, it is acceptable to consider these vessels part of the available fleet for the purposes of this study.

In addition, a significant number of the 30-35k DWT vessels were built between 2010 and 2014, and a significant number are on order through 2016.

In summary, it appears that there are currently 710 acceptable 30-35k DWT vessels in the existing fleet with a substantial number of new builds. As such, a sufficient supply of vessels should be available for the project, pending market conditions.

Figure 18-12 Bulk Carrier Population



18.18.7 Wharf Layout and Infrastructure

This section describes the layout and conceptual design of the direct loading wharf at Ponta Chugue.

Geotechnical Assumptions

Available geotechnical data at Ponta Chugue are presented in Golder (2012), and a summary of subsurface conditions is provided in Baird (2012). Available data are limited to onshore investigations, which include seven boreholes, eight test pits, and several geophysical profiles. No offshore subsurface investigations were performed.

In general, soil profiles on land consisted of stiff to hard sandy and gravely clays with sands becoming more prominent below 10 m depth. Although bedrock was not encountered in the borings, geophysical logs indicate bedrock at depths of 30 to 40 m below grade.

For the purposes of preliminary pile foundation design (at all locations considered), design assumptions on subsurface conditions were consistent with Golder (2012), dense sand (N=40) below grade and a factor of safety of 2.5 on calculated pile capacities. Baird assumed bedrock at El. -30 m CD, which is generally consistent with the landside geophysical profiles.

It is noted that a rocky promontory along the shoreline at Ponta Chugue provides evidence of possible subsurface rock. In addition, the steep angles of repose of the existing riverbed just offshore of Ponta Chugue indicate rock.

The general lack of offshore geotechnical data potential presence of bedrock within relatively shallow depths presents significant uncertainty in the design process.

18.18.8 Facility Description

Facility Infrastructure

The primary elements of infrastructure making up the direct load wharf follow. For additional detail see Figure 18-13 and Figure 18-14.

- Steel pile bents to support the conveyor and truss system delivering phosphate to the wharf. Bents are constructed with two 600 x 16 mm pipe piles with opposing 1H:6V batters. Three HP360 x 152 beams are welded together to form the bent cap on which the conveyor and truss system is directly attached.

- Two steel pile supported platforms are used to support a single telescoping radial shiploader. The curved shiploader track platform is 4.0 m wide and approximately 47 m long on the outside arc. The platform is supported by nine bents of 600 x 16 mm pipe piles with opposing 1H:6V batters. Three HP360 x 152 beams are welded together to form the bent caps on which the superstructure is directly attached. The platform deck consists of five W360 x 122 steel beams evenly spaced and laterally restrained with heavy-duty steel grating.

- Four steel pile supported mooring dolphins. Each dolphin incorporates four 1,000 x 25 mm pile piles with 1H:6V vertical batters. The steel pile cap doubles as a driving template and is fabricated with 1,100 x 25 mm pile sleeves and 800 x 19 mm cross bracing. The central bollard mount consists of a 1,200 x 25 mm pipe fully grouted and sealed with 25 mm top and bottom plates. Steel grating is used as a permeable walking surface.

- Four steel pile berthing dolphins with steel decks. The berthing dolphins are identical to the mooring dolphins except for the addition of a braced 1,000 x 25 mm vertical pile. A parallel motion fender with two 800 mm Grade 3 rubber cones and a 2.5 x 5 m fender panel is affixed to each vertical pile through field-welded brackets.

- Steel gangways providing access to the mooring and berthing dolphins for stevedores.

- A floating wharf to moor tugs and the pilot boat. It is anticipated that all service vessels will be fuelled from the floating wharf.

- Guide piles and a graded access ramp for loading and unloading of a maintenance barge. Note this facility is recommended as there is no vehicular or heavy equipment access out to the wharf. As such, floating plant will need to be utilized in the event of breakdown. The maintenance barge will also be critical for maintain aids to navigation.

- Required aids to navigation at Ponta Chugue and along the anticipated navigation route.

Note the infrastructure does not include items such as the mechanical, electrical, utility, fuelling, and lighting requirements for the main wharf or floating tug wharf. It also does not include the intended shiploader or conveyor.

Figure 18-13 Wharf Plan View

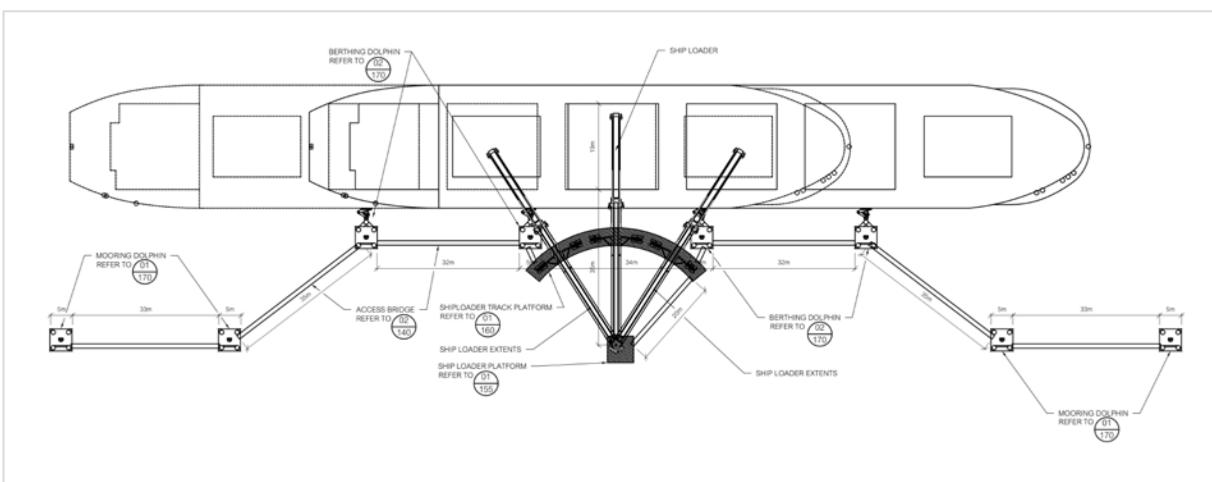
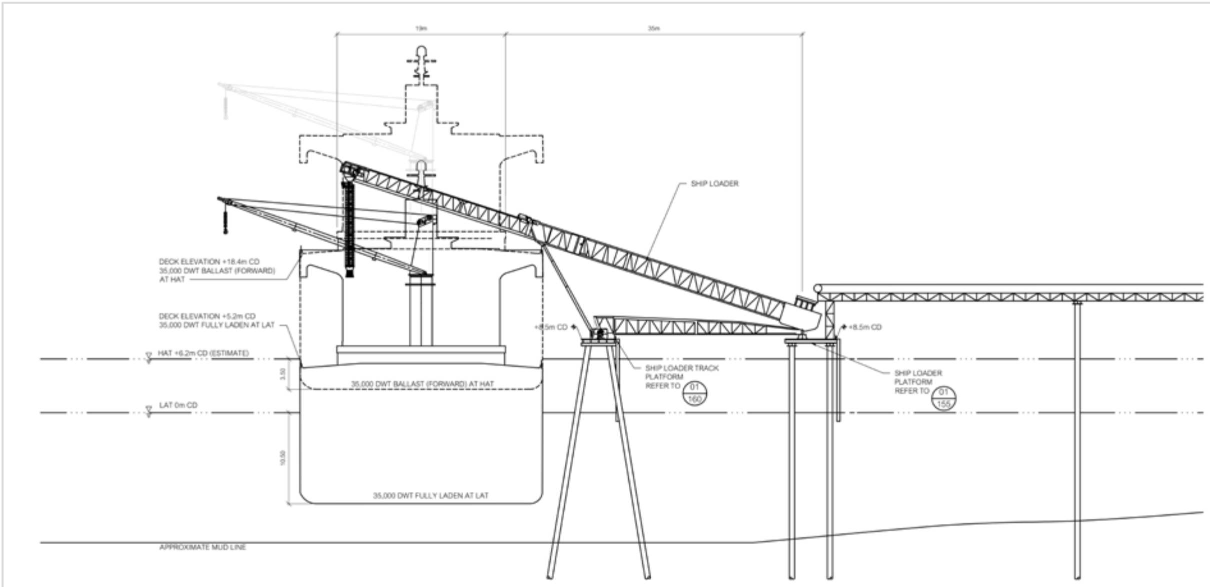


Figure 18-14 Wharf Cross-Section



Facility Alignment and Layout

The wharf was located east of the point at Ponta Chugue so that the facility could be both close to shore and vessels could still avoid the shoal shown in Figure 18-15 below. The facility is generally aligned with the -15 m contour. It is noted that a detailed geomorphology and sediment transport study was not completed as part of the feasibility study. As such, the location of the wharf may be subject to sedimentation.

The facility, and thus the vessel at berth, is generally aligned with the tidal currents (Figure 18-16). This alignment reduces the tendency of the current to draw the vessel away from or onto the berth. In addition, it should minimize the amount of vessel motion during loading making for a safer warping operation.

Figure 18-15 Direct Load Facility Aligned with Current

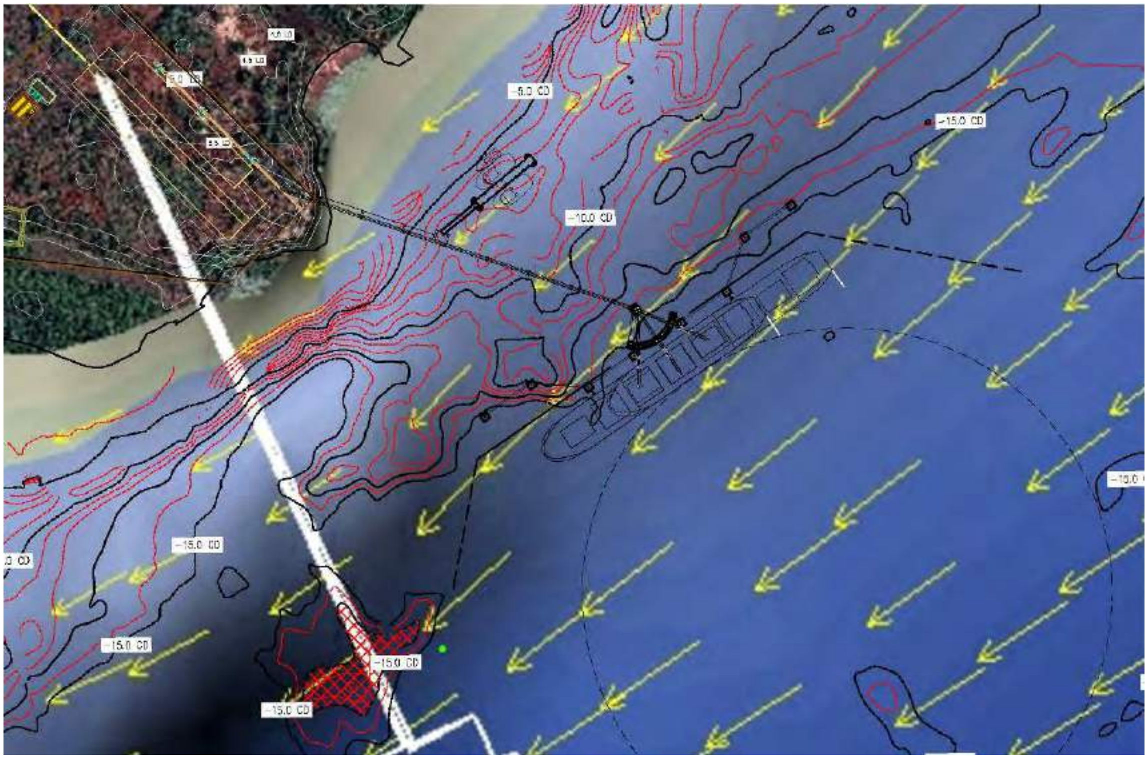
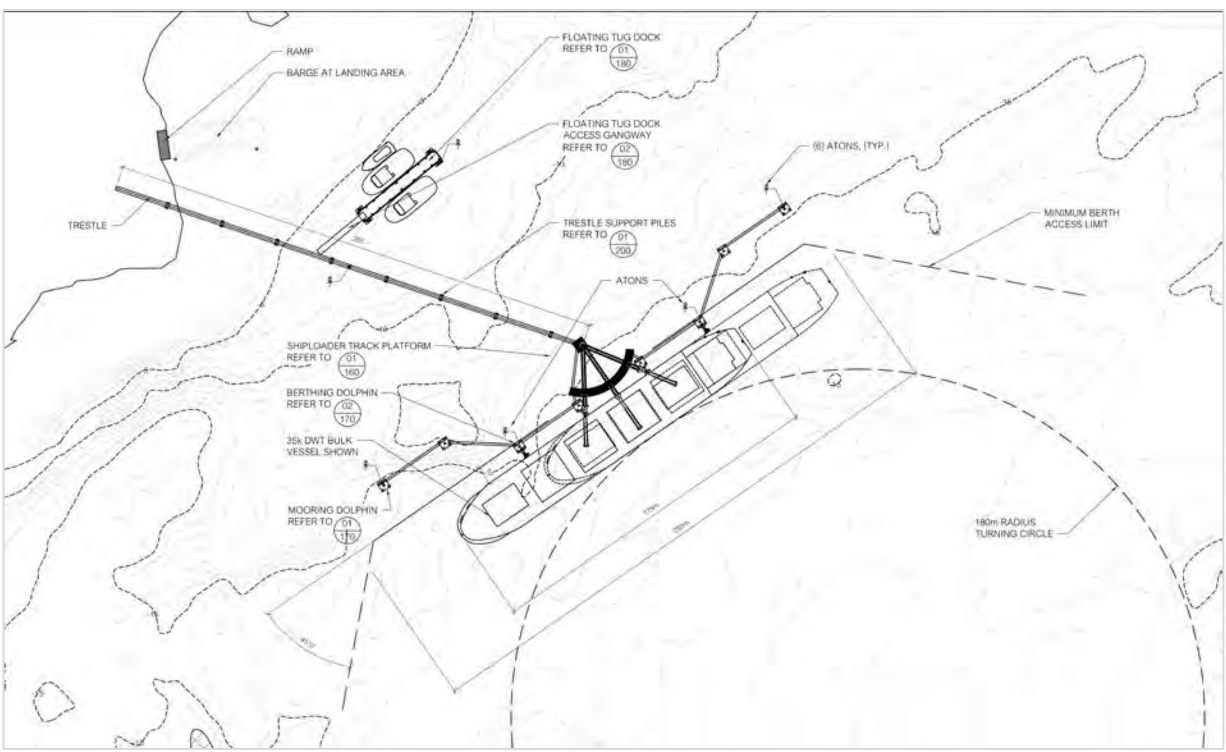


Figure 18-16 Direct Load Facility General Layout



18.18.9 Support Vessels

A fleet of support vessels will be required at Ponta Chugue to transport the pilot to and from bulk carrier, safely berth and deberth the vessel at the wharf, and to warp the vessel while at berth. Further description is provided below.

Tug Boats

Tugs will be required at Ponta Chugue to assist the bulk carrier into the berth and to warp the bulk carrier along the berth during loading. The tugs must be sized to handle the wind and potentially large current speeds at Ponta Chugue. It is recommended that two tugs with bollard pull between 30 and 40 tonnes be utilized for these operations. The specifications are based on Baird's past experience with similar projects and the Sea-web database. Additional work will be required during final design to recommend final specifications and tug propulsion type. For the purposes of this study, twin screw or Azimuth Stern Drive (ASD) propulsion units are generally thought to be sufficient.

The Sea-web database was queried to investigate tugs meeting the above requirements. Statistics for vessels with a bollard pull between 30 and 40 tonnes were compiled for engine power as well as tug length and draft. Tugs for the direct shipping option will have the approximate characteristics listed in Table 18-3.

Table 18-3 Approximate Tug Characteristics for the Direct Shipping Option

Number of tugs	**2**
Bollard pull	30-40 tonnes
Engine power	1500-2500 kW
Length	20-35 m
Draft	2.5-4 m

Pilot Boat

A Pilot boat will be required to transport the pilot from Ponta Chugue to the pilot pick up point, where the pilot will board the bulk carrier for inbound vessel transit and return also to him to Ponta Chugue following outbound vessel transit. Pilotage is standard practice at ports and river passages throughout the world. The use of a pilot significantly reduces the risk of vessel accident, as the pilot will gain considerable knowledge concerning the local environmental conditions of the river (waves, currents, winds, hydrographic) through repeated vessel navigation.

Local pilots for the Port of Bissau currently board vessels at Ponta de Caio (approximately 60 nm from Ponta Chugue). It has been assumed that the dedicated Farim pilot will board at a similar location.

Maintenance Barge

A Maintenance Barge (crane barge) is needed to maintain the navigation aids and provide floating access to the direct load infrastructure in the event of damage or breakdown. A covered area and crew house is typically included on such a crane barge and is recommended. Many crane barges are also fitted with mooring / anchoring systems and / or spudwells and spuds to hold them in position while on the project; these items are also recommended. A 50 m length x 15 m breadth, (approximate depth 3.1, draft 1.8 m) maintenance barge has been included in the CAPEX estimate.

18.18.10 Operations

A general description of the anticipated operation of the direct load terminal follows:

- Pilots will be board the pilot boat at Ponta Chugue and sail to the pilot boarding location near Ciao. The ocean going vessel will then be piloted 60 nm to Ponta Chugue;
- Vessels ranging from 30-35k DWT will be piloted up the river, arrive at the facility, be berthed bow west, with tug assistance, and will undergo draft survey;
- Mooring lines will require nearly constant tending during loading to accommodate the changing tide;
- As the wharf utilizes one radial telescoping shiploader, the bulk carrier must be warped or shifted along the wharf throughout the loading process to access all the bulk carrier's holds. Warping requires careful preparation and a full mooring party available at the wharf and crew aboard the ship. If the ship moves off the wharf during warping, the bow or stern can swing in to the quay resulting in damage to the bulb, rudder, propeller, or to the wharf;
- Warping will be achieved with tug assistance;
- Warping will not be permitted after nightfall. In general, the vessel will be warped at dusk and can continue loading into the night until another warping movement is required;
- Vessel loading will be suspended during the rain to preserve the moisture content of the material. It is noted that this restriction should be further evaluated during final design;
- Following loading, the vessel will undergo another draft survey before deberthing under tug assist; and
- Pilots will navigate the laden vessel 60 nm from Ponta Chugue to the pilot boarding location near Ciao. The pilot will then disembark and return to Ponta Chugue via the pilot boat.

18.18.11 Discrete Event Simulation

A discrete event simulation (DES) of the direct load scenario was undertaken using Arena ® Simulation Software, developed by Rockwell Automation. The simulation included the processes of vessel arrival, vessel loading, and vessel departure.

A number of the inputs used in the model are shown in Table 18-4. It is noted that the vessel loading rate at Ponta Chugue assumes a net loading rate of 60% of the peak capacity of the material handling system (1,200 tph).

Table 18-4 Direct Load Model Inputs

Input	Value Option A
Vessel size	32.500 DWT
Laden draft + 15% UKC	12.075 m
Barge Loading rate	750 ton/hr
Drying plant feeding availability	0.91
Current limit for bulk carrier and barge berthing	3.5 kn
Loading suspended during rain events	Yes
Operations limited to daylight hours	Warping

Turnaround-Time

Average bulk carrier turnaround-time (time the vessel is Ponta Chugue) was estimated for use in approximate vessel freight rates. The major components comprising vessel turnaround-time are shown in Table 18-5 below.

It is noted that demurrage and despatch were not included in OPEX estimates as vessel chartering arrangements were uncertain at the time of the feasibility study.

Table 18-5 Average Turnaround-Time

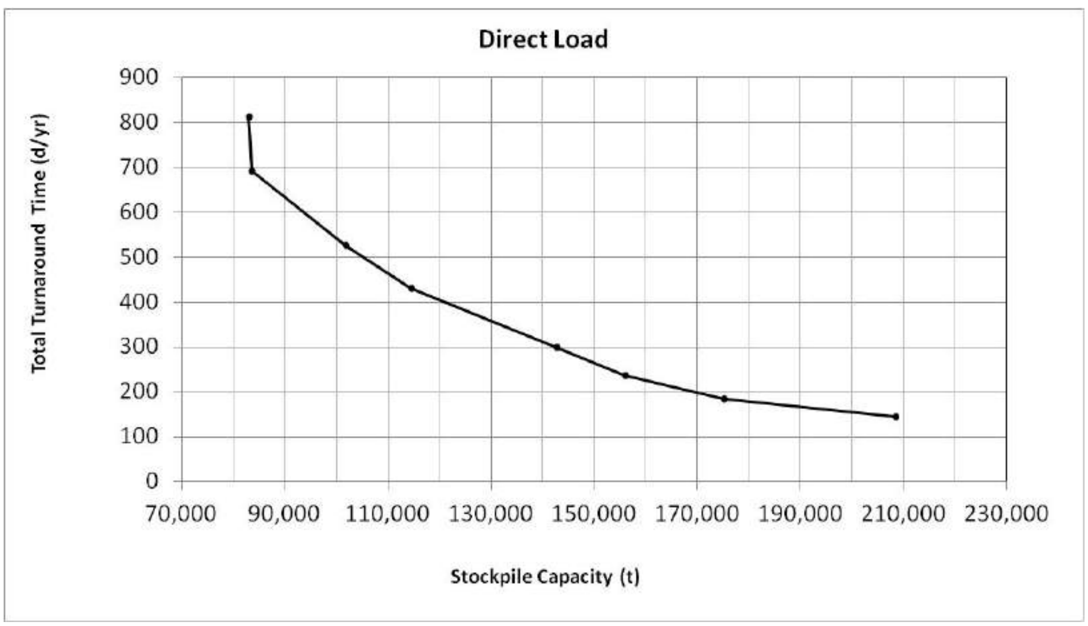
Process	12 months	
	Time (day/year)	Percentage
Waiting berth	16.5	11.3%
Rain downtime	29.5	20.3%
Holidays	2.5	1.7%
Berthing	1.7	1.2%
Loading	82.7	56.9%
Draft survey and deberthing	5.0	3.5%
Waiting daylight	7.5	5.2%
Waiting allowable draft	0.1	0.0%
Total TAT	145.4	100.0%
TAT (d/vessel)	3.5	

Minimum Land-Based Stockpile Size

The land-based stockpile level was monitored during simulations. A sensitivity analysis was conducted to investigate the impact stockpile size has on bulk carrier turnaround-time. The results of the analysis, shown in Table 18-6 below, indicate that substantial increases in turnaround time occur when the stockpile size is reduced to less than170,000 t. Increased turnaround time results in higher bulk carrier freight rates due to the additional time in port.

It is noted that the size of the stockpile in the model is highly influenced by the assumption that vessels will discontinue loading during all rain events and also by the irregular vessel arrival schedule that was assumed. Reductions in stockpile size can be achieved during final design in the event that vessels are allowed to load during rain events or if it is determined vessels will arrive at Ponta Chugue in a highly scheduled manner.

Figure 18-17 Sensitivity Analysis Turnaround Time vs. Stockpile Size



18.19 Tailings Storage Facility and Infrastructure

18.19.1 Site Characteristics

The Project comprises a high grade phosphate deposit which occurs within the Middle Eocene Lutetian Formation in a Cenozoic sedimentary basin that extends from Morocco in the north through Mauritania, Senegal, Guinea-Bissau and into Guinea to the south. The project will comprise an open pit mining operation with two individual pits, a tailings storage facility, a process (beneficiation) plant and other minor infrastructure. The process plant is designed to treat ore at a rate of 1.75 Mtpa over a mine life of 25 years and will generate 0. 256Mtpa of tailings.

The project site is located within and adjacent to the flood plain of the River Cacheu. Though the site is approximately 100 km from the coast, the river is tidal and has a tidal range of approximately 1.5 m at the site. There are almost continuous mangroves along the banks of the river and its tributaries. A grassland salt plain is commonly present beyond the mangrove which seasonally floods. The topography of the site area has very little relief and the highest point within the entire site area is approximately 40 mamsl.

The climate has distinct wet and dry seasons with almost all of the yearly rainfall occurring between May and October. Average annual precipitation and evaporation are approximately 1150 mm and 1600 mm respectively.

The seismicity of Guinea Bissau is typical of an intra-plate region, which is characterised by low levels of seismic activity and earthquakes that are randomly distributed in location and time. For design purposes Operational Basis Earthquake (OBE) and Safety Evaluation Earthquake (SEE) magnitudes of 0.04g and 0.12g have been adopted.

18.19.2 Geochemistry of Waste Rock

Three phosphate-bearing horizons (referred to as the FPO, FPA and FPB) have been identified at the Farim phosphate deposit. The FPA unit is the identified potentially economic phosphate bed. The FPB underlies the FPA and is of less economic interest due to the lower phosphate and high limestone content. The FPO is a clayey dolomitic limestone that is weakly phosphatic with limited economic potential. The phosphate deposit is underlain by a soft, white and porous limestone unit. The phosphate bearing strata are unconformably overlain by a sandy-argillaceous sequence comprising soft alternating sandy, clayey and sandy-clay layers with a blue/green soft clay or black lignitic clay at the base.

The site stratigraphy is summarized in Table 18-6.

Table 18-6 Indicative Site Stratigraphy

Age	Unit	Description	Thickness (m)
Post Eocene	Sandy/Argillaceous Overburden	Alternating sandy, clayey and sandy clayey layers	27 to 58
Eocene	Basal Clay Overburden	Blue/green soft clay and black lignitic clay (anoxic depositional environment)	
	Sand including FPO (phosphatic interval)	Grey/white fine grained sand including phosphate bearing clayey dolomitic limestone (FPO)	7 (single intercept)
	Upper Dolomitic Limestone	Clayey limestone	>2[1] (single intercept)

	Decarbonised Phosphate Unit (FPA)	Ore zone comprising beige to brown, poorly cemented very fine grained phosphatic sand.	1 to 11
	Calcareous Phosphate Unit (FPB)	Cemented phosphatic limestone	2 to 8
	Limestone	Soft, white and porous limestone	>6 to 17[1]

1 Base of unit not encountered

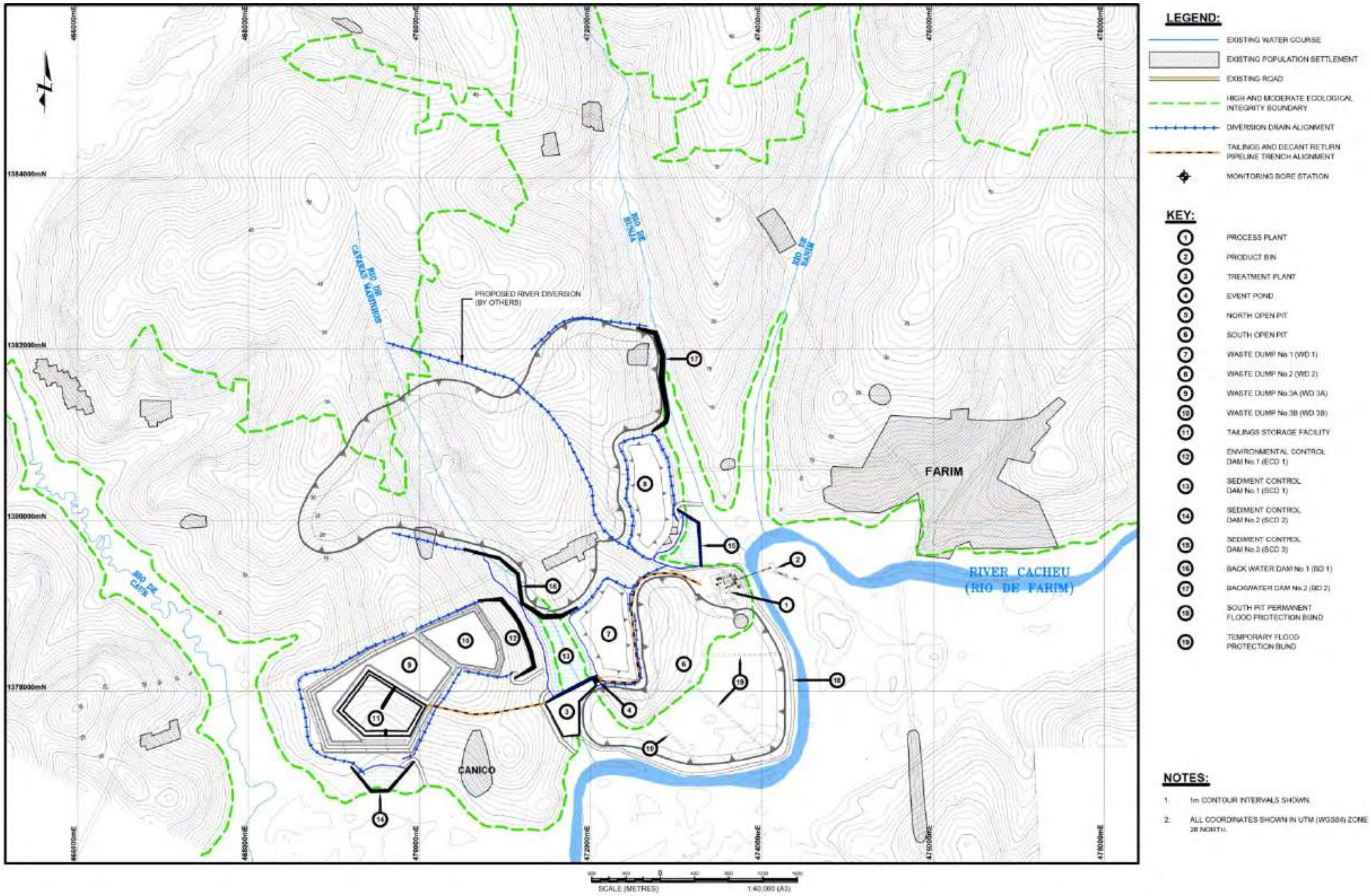
A geochemical assessment comprising testing of twenty composite samples of overburden was undertaken in 2012 as part of a previous phase of the project. It was concluded that the potential for acid generation through the oxidation of waste rock was expected to be low. In addition, it was reported that trace metal concentrations within the overburden waste are typically at or below crustal abundances. However, silver, arsenic, molybdenum, selenium and uranium were reported to exceed the crustal abundance by a factor of six in one or more samples. Distilled water extract testing results indicated that phosphate, ammonia, ammonium, sulphate, arsenic, cadmium, chromium, iron, manganese, nickel, lead and zinc exceed the reference surface water quality guidelines, with cadmium and nickel exceeding the World Health Organisation drinking water guidelines.

Most of the stripped overburden is envisaged to be benign and will be used to construct the Integrated Waste Landform (IWL), stored at surface or used to backfill mined out areas of the open pits. The overburden that may leach metals will be encapsulated within a separate waste dump as construction proceeds, or placed back in the mined out pit and encapsulated. Additional sampling and testing is currently in progress with the purpose of quantifying each of the waste lithologies present within the pit shells in more detail, so that the waste management plan can be developed further.

18.19.3 Waste Dump Design

The project mining plan indicates that of the 539 Mm^3 (loose cubic metres) of waste “rock” produced during the mining operation, 78% (420 Mm^3) will be placed as in-pit backfill, 13% (71 Mm^3) placed as surcharge overburden stockpile, and 9% (48 Mm^3) deposited in ex-pit waste dumps. Four ex-pit waste dumps are proposed, three of which (WD1, WD2, WD3a) are proposed to stockpile inert waste only, Two dumps will be located between the North and South pits (WD1 and WD2). A third (WD3a) will be formed around the perimeter of the proposed tailings storage facility, to form an integrated waste landform (Figure 18-18). A fourth waste dump, WD3b, is located directly to the northeast of WD3a but will contain potentially leachable waste in specially designed encapsulation cells contained within inert waste.

Figure 18-18 Tailings and Waste Rock Facilities



Based on the testing to date there is potential for elevated concentrations of environmentally significant metals and compounds in a portion of the waste rock. The relative quantities of each lithology and their geochemical characteristics are being determined in more detail at present. In the meantime, the following design philosophy has been adopted:

- Four ex-pit waste dumps are to be operated, WD1, 2, 3a and 3b. WD's 1 and 2 are sized to store 8.5 Mm^3 and 9.0 Mm^3 respectively. WD3a will store approximately 23 Mm^3 and WRD3b approximately 8.5 Mm^3 (nominally 18% of the total ex-pit waste).

- WD's 1, 2 and 3a will stockpile inert waste only. WD3b will contain inert waste plus potentially leachable waste in specially designed encapsulation cells contained within the inert waste.

- WD3b will be designed to reduce seepage of contact water from entering the surface water and groundwater environments. A basal low permeability soil liner will be formed beneath the footprint of WD3b and seepage/run-off flows at the base of WD3b will report to an environmental control dam (ECD) by means of re-shaping the ground surface to promote drainage to the pond, or provision of a network of above-ground sand/gravel drains. Any contaminated water collected in the ECD will be returned to the process plant for re-use. Uncontaminated water will be returned to the plant for re-use, if required, or discharged to the environment.

- Water collected from the WD1, WD2 and WD3a dumps will report to separate sediment control dams (SCD's) and will be discharged to the environment after reducing the sediment loading to an appropriate level.

The proposed geometry of the ex-pit waste dumps is as follows:

- Bench height – 10 m.

- Inter-bench slope - 1V:2H.

- Inter-bench berm width – 10 m.

18.19.4 Tailings Storage

Tailings Testing

A limited scope tailings testing programme was carried out on a small tailings sample slurried to 48% solids. The main findings of the tailings testing are as follows:

- The tailings will initially settle very slowly and only release a small quantity of water, based on the tested percent solids. Initial densities will therefore be low;

- The tailings will be slow to air dry and will need a large beach area and long exposure time, but benefits greatly with an increased dry density;

- Based on the grading size, it is predicted that the tailings will be of low permeability and high compressibility.

- The tailings were found to be Non-Acid Forming. As such, there is no perceived risk from acidification of the tailings and there are no specific controls required. However, the high sulphide content of the tailings has the potential to lead to high sulphate and saline drainage. Long-term leach tests will be required to assess this in more detail.

- The tailings solids contain elements with high levels of element enrichment, with bismuth, cadmium, phosphorous and selenium found to be highly enriched, silver, arsenic, fluoride, sulphur, antimony and uranium found to be significantly enriched, and calcium and chromium slightly enriched. As such the tailings storage facility must be designed and operated to fully contain the tailings solids and reduce dusting. As there are a number of environmentally significant elements which appear at elevated levels, in particular Uranium, a formal engineered containment and capping system may be required.

- The supernatant was found to exceed baseline surface water and groundwater concentrations for several parameters. As such, the TSF will require controls to limit seepage to surface water and groundwater, notably an engineered low permeability liner across the base and sides of the facility, and storage capacity sufficient to contain all stormwater run-off within the adopted design standard.

A larger representative sample will need to be tested at the revised design percent solids to determine the full range of permeability and consolidation parameters, and to confirm the geochemistry of the solids and supernatant liquor.

Tailings Design Parameters

The process design parameters for design of the tailings storage are summarized in Table 18-7.

Table 18-7 TSF design parameters

Design tonnage	**6.4 Mt**
Life of mine	25 years
Tailings output	256,000 tonnes per annum
Tailings beach slope	1V:80H

Tailings Storage Design

The tailings storage will consist of a single-cell paddock storage built as part of an integrated waste landform (IWL), located approximately 500 m west of Canico township (Figure 18-1). The IWL is designed to fully contain the design tonnage as well as rainfall run-off arising from storm events up to and including a 1 in 100 year average return interval (ARI) storm event or wet sequence.

The design incorporates a multi-zoned embankment, a compacted in situ soil-lined basin area, an underdrainage system covering the approximate extent of the supernatant pond, and an upstream toe drain. Due to the potential elevated levels of environmentally significant elements, a provisional allowance has been included to line the facility with an HDPE geomembrane, subject to further geochemical assessment of the tailings solids and supernatant liquor.

Tailings will be discharged into the facility by sub-aerial deposition methods, via a number of single point discharge points located along the embankment crest. The active tailings beach will be rotated around the facility so as to maximise tailings density and maintain the decant pond around the central decant. The upstream toe drains and underdrainage system will drain by gravity to a collection sump located at the toe of the southern embankment. Supernatant water will be decanted from the facility via a decant tower located at the centre of the facility. Solution recovered from the underdrainage and decant systems will be pumped back to the plant for re-use in the process circuit.

The embankments will be constructed in stages over the life of the facility to suit the storage capacity requirement. The embankment crest levels and design storage capacity at each stage are summarized in Table 18-8.

Table 18-8 Proposed TSF Embankment Staging

Construction Year	Stage	Crest Elevation (mASL)	Raise (m)	Cumulative storage (Mt)
Yr 0	1	15.0	8.5	0.3
Yr 2	2	16.2	1.2	0.7
Yr 4	3	17.4	1.2	1.2
Yr 6	4	18.6	1.2	1.7
Yr 8	5	19.6	1.0	2.2
Yr 10	6	20.7	1.1	2.8
Yr 12	7	21.7	1.0	3.3
Yr 14	8	22.6	0.9	3.8
Yr 16	9	23.5	0.9	4.3
Yr 18	10	24.4	0.9	4.8
Yr 20	11	25.3	0.9	5.3
Yr 22	12	26.2	0.9	5.8
Yr 24	13	27.1	0.9	6.4

The initial stage (Stage 1) will be constructed using selected mine waste and/or local borrow to provide 14 months of storage capacity. Thereafter the embankments will be raised on a bi-annual basis. The embankment will be built as a zoned earthfill embankment utilising material excavated as overburden from the open pit excavations and consisting of an upstream low permeability zone (Zone A), a transition fill zone (Zone B) and a downstream structural fill zone (Zone C). It will have upstream and downstream slopes of 3H:1V and a crest width of 10 m. The initial embankment (Stage 1) will be approximately 9 m high and the final embankment (Stage 13) will have a maximum height of approximately 25 m. The final length of the perimeter embankment will be approximately 2.8 km.

Mine waste from the open pit excavations will be placed into WD3a, outside the perimeter of the TSF, to form an integrated waste landform for efficient reclamation and rehabilitation. Capping of the TSF will be undertaken on de-commissioning to control dusting and water ingress.

18.19.5 Open Pit De-Watering

Groundwater Modelling

A groundwater flow model was developed for the open pits based on the proposed mining plan. The model was used to predict groundwater inflows during the various stages of development of the open pits and the associated drawdown extents. An improved de-watering infrastructure configuration was developed based upon the model findings.

Modelled pit inflows throughout the whole simulation period are shown graphically in Figure 18-19 and are summarised as follows:

- South Pit - average daily pit inflow is approximately 13,000 m^3/day (150 l/s), ranging between 9,800 m^3/day (113 l/s) and 16,700 m^3/day (193 l/s).

- North Pit - average calculated inflow is 6,500 m^3/day (75 l/s) ranging between a peak of 8,900 m3/day (103 l/s) and 5,100 m^3/day (59 l/s) at the end of mining (Year 26).

The drawdown resulting from pit de-watering has the potential to impact a significant number of nearby water users.

Assuming a yield per bore of 10.5 m^3/hr (3 l/s) the initial number of bores required is 40. As the pit is developed in phases, some bores will be de-commissioned and new bores will be constructed as new areas of the pit are developed. Over the operating life of the mine approximately 15 to 90 bores will operate at a time. A total of 540 bore locations will need to be developed and de-commissioned. The de-watering plan is shown in Figure 18-20. The bores would be drilled to the -70 mamsl level (about 70 m deep). In addition, in-pit sumps would also be required.

Figure 18-19 Dewatering Modelled Pit Inflow Rates

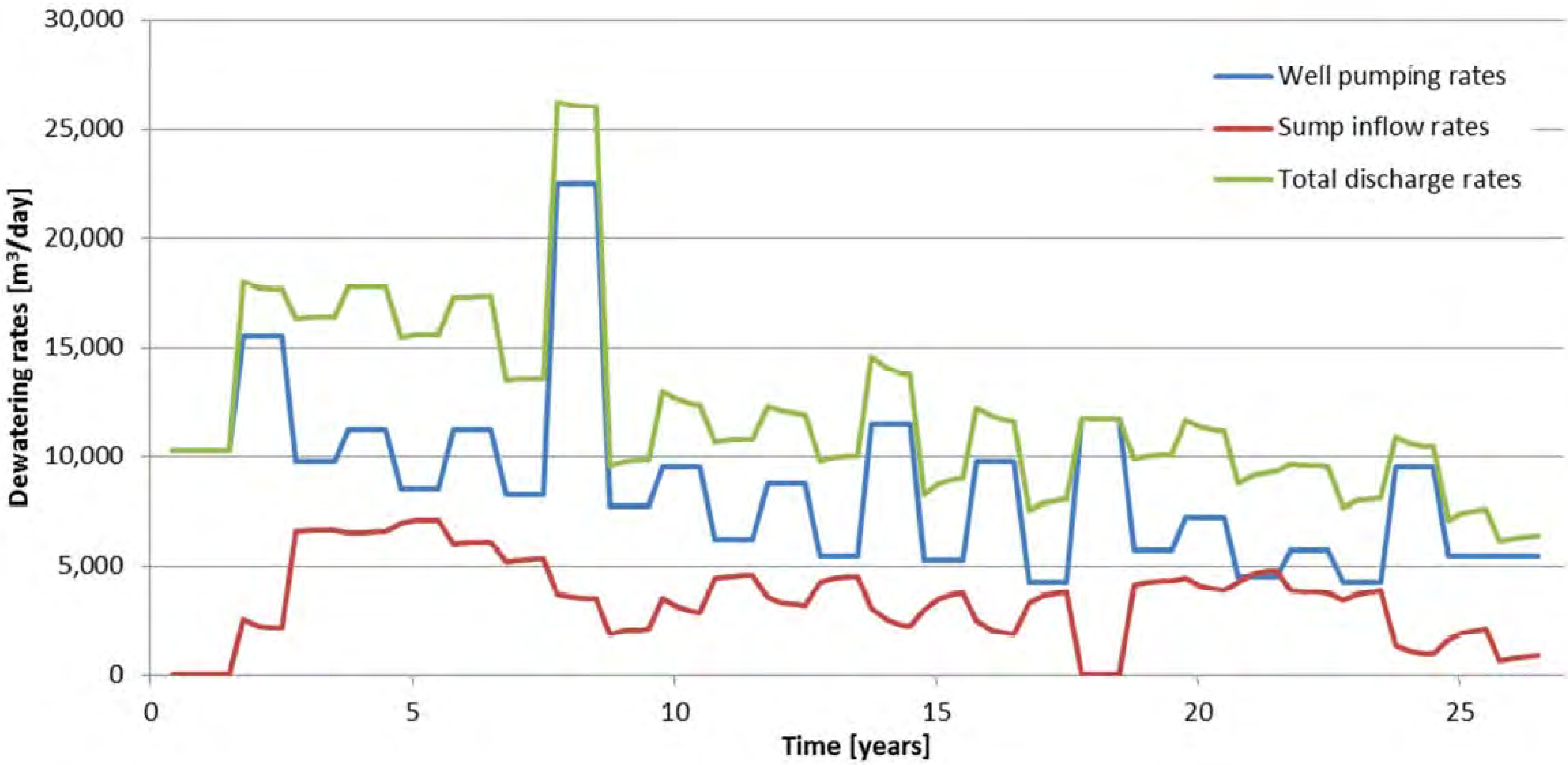
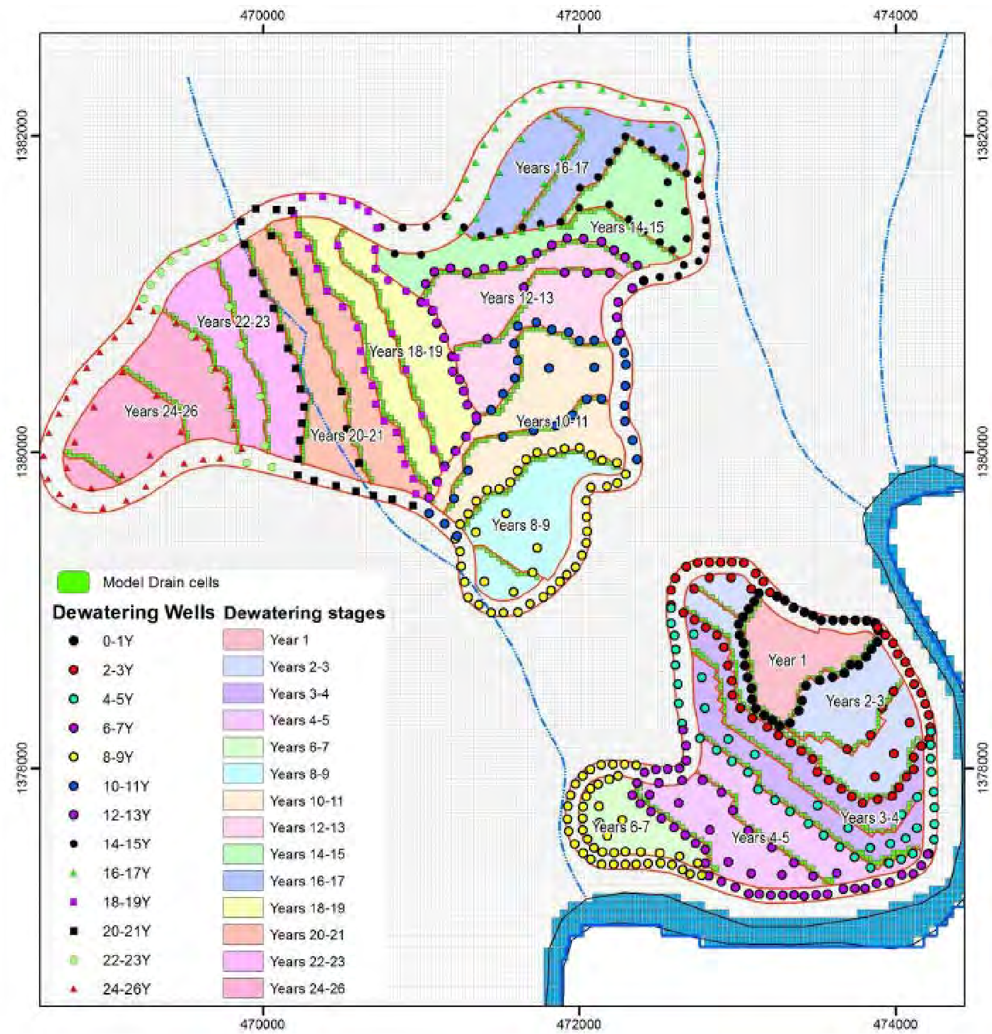


Figure 18-20 Dewatering Plan

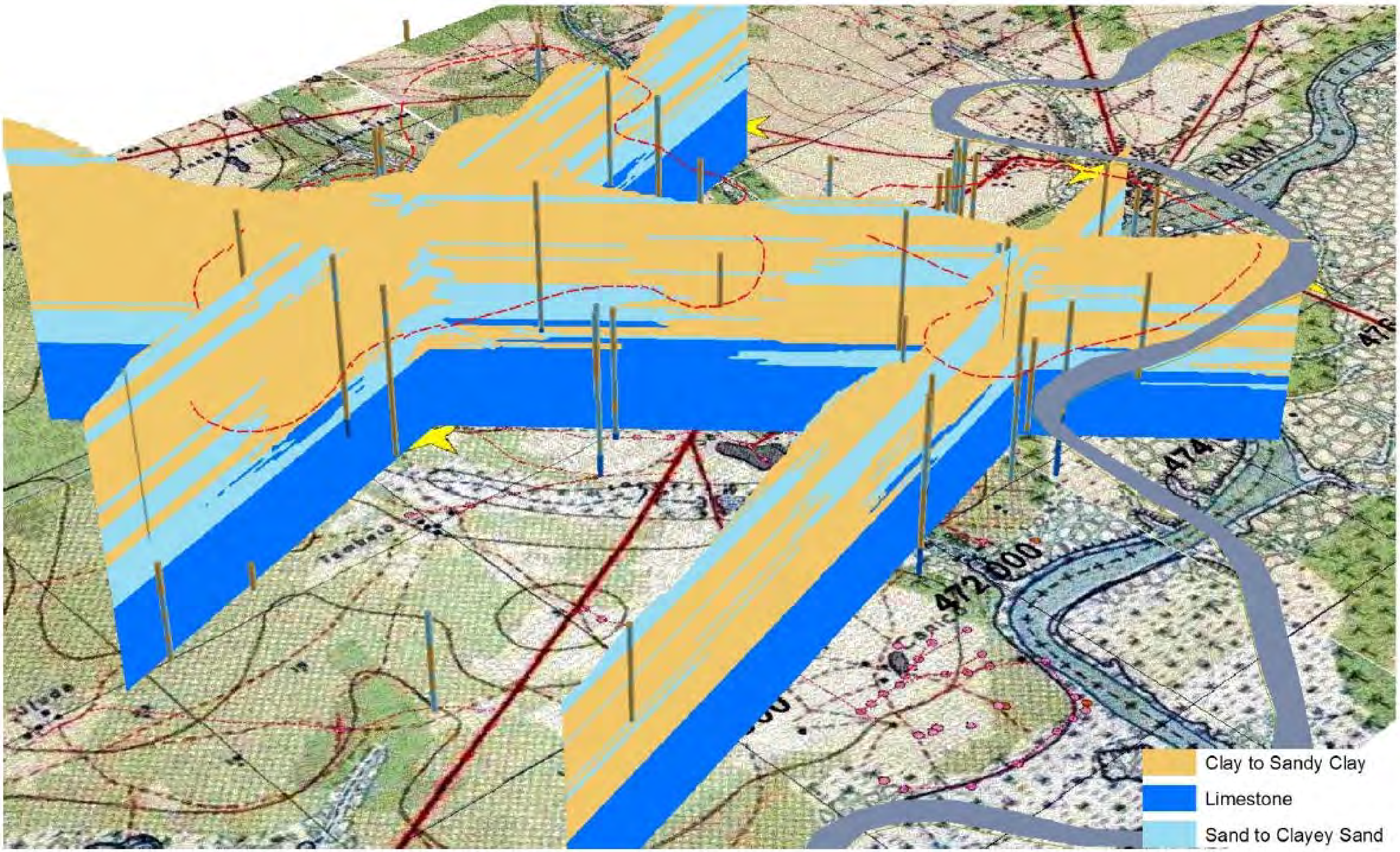


Conceptual Hydrogeology

Since the initial construction of the hydrogeological model the mine plan has changed and the understanding of the underlying geology has also improved significantly.

The river flood plain which occupies the southern part of the South Pit footprint is underlain by about 15 m to 20 m of clay (shown in light brown in Figure 18-21). This constrains the degree of hydraulic connection with the River Cacheu. In addition the deposit overburden comprises significant clay and sand lenses and it is now thought that these lenses are not as uniformly layered as was previously conceptualized. Leakage from the north to south flowing tributaries of the River Cacheu (Rio de Caur and Rio de Cavaras Marinhos to the west, and Rio de Bunja and Rio de Banin to the east) will constrain the extent of drawdown in an east-west direction, although the tributaries may contribute to pit inflows.

Figure 18-21 Dewatering Geological Fence Diagram



Conclusions

To further improve the understanding of the de-watering needs and the potential associated hydrogeological impacts, and following on from the recent field investigation, the groundwater model should be updated for the next phase of development, as follows:

- Provide an updated orebody geological model to allow a refinement of the hydrogeological model domain. In this regard, the development of a block geological model, to be used as a basis for the hydrogeological and other mining design and development purposes is essential.

- Completion of additional pumping tests in the southern pit to improve understanding of the groundwater flow regime, the hydraulic connection with nearby creeks and surface water bodies, water impacts associated with mining the pit and the variation of aquifer hydraulic properties over the area of interest.

- A Groundwater Management Plan is required to address the drawdown impacts on local water supplies. This would typically include:

  - The establishment of a groundwater monitoring network and a mitigation plan to ensure that water availability is maintained.

  - This would also include updating the hydrocensus and surveying nearby bores to determine use, depth, water level elevation.

  - The occurrence of at least two different water types, i.e. fresh groundwater and surface brackish river and creek water, support the need for a hydrogeochemical survey to understand baseline groundwater quality, the role of hydrogeochemical processes in the system and the degree of interaction between the brackish surface water bodies and groundwater both at current conditions and during dewatering.

- This will provide additional information for further model calibration and greater certainty around severity of likely dewatering impacts both with respect to groundwater levels and quality.

- Incorporate new pumping tests and hydrocensus data into the above model to provide additional calibration and refinement.

- The long term response of the aquifer to pumping, especially at the southern pit, should be tested by additional pumping tests.

- Investigate alteration of the mine plan to maximize dewatering efficiencies.

- Develop a borefield operating strategy to determine likely monitoring, equipment recycling and maintenance requirements.

- A surface water management plan is recommended to understand and mitigate the risk of surface water flooding of the southern pit and marshland inflows over the mangrove marshy area.

18.19.6 Geotechnical Assessment

The mine and port sites lie within an extensive sedimentary basin, on low lying ground and close to tidal rivers bordered with mangroves, mudflats and salt flood plains. The ground conditions encountered reflect the geological and topographical setting with normally to lightly consolidated and primarily cohesive alluvial deposits often encountered close to the rivers and more overconsolidated deposits at greater distance from the rivers.

The ground conditions along the southern perimeter of the Southern Pit and the product stockpile area of the plant site (east side of river) comprise very soft alluvial clays to depth. The ground conditions at the sites of the proposed tailings storage facility, processing plant (west side of river) and port site comprise more overconsolidated deposits. Near surface soils are predominantly cohesive and the safe bearing pressures afforded by these soils will be relatively low, particularly for small structures.

The subgrade at the proposed process plant site (west) is considered to offer reasonable stiffness but high settlements can be expected, a function of the magnitude of applied loading. The estimated settlements for structures within the processing plant are higher than the stated allowable settlements. Due to the high groundwater and saturated and low permeability soils, settlement is expected to take many years to complete. The impact this may have on long term maintenance will need to be considered. The ground conditions underlying the product stockpile (plant site east) are poor and similar to those identified along the southern perimeter of the South Pit. The subgrade is not considered suitable to support significant structures on spread footings, and piling or subgrade remediation measures will be required.

Selected sedimentary soils will be suitable for low permeability and general fill, and selected laterite sources may be suitable for structural fill and road sub-base, subject to testing. Outcrops of strongly indurated laterite suitable for erosion protection are present but potentially too small and few to provide adequate quantities for construction. Local sources of drainage sand/gravel, road base and concrete aggregate were not identified. The nearest known hard rock quarry is approximately 150 km from the site.

18.19.7 Surface Water Management

Sediment Control

Sediment control will be carried out using two primary methods comprising source control (i.e. reducing the generation of sediment) and the removal of sediment from run-off prior to discharge by means of sediment control dams (SCD’s). In addition to these controls disturbed areas will need to be limited as much as practicable, particularly during the wet season, and a continuous rehabilitation programme should be implemented to reduce the sediment load in run-off further.

Open Pits

The footprint of the South Pit lies partially within the tidal flood plain of the Cacheu River, with a tidal range at the site of approximately 1.5 m. It is proposed to construct a flood protection bund along the perimeter of the South Pit where it borders the Cacheu River. This will be constructed in stages with temporary bunds radiating out towards the river to de-lineate the proposed pit staging (and hence to defer construction costs over the life of mine), and to provide for construction access. The flood protection bund will be constructed to a crest elevation of 4 mamsl and to a width of 20 m using pre-strip mine waste placed directly by the mining fleet. An erosion protection layer will be placed on the river side of the bund.

The North Pit will require the construction of flood protection bunds during the initial stages as well as the construction of a and diversions to divert run-off from the upstream catchment past the active mining area in the later stages.

Process Plant

The flood protection bund will extend along the eastern and northern boundaries of the process plant site (west).

Waste Dumps

Sediment control dams will be located downstream of each of the four proposed inert waste dumps. Water collected in the WD1, WD2, and WD3a SCD's will be discharged to the environment after reducing the sediment loading to an appropriate level.

Waste dump WD3b is sized to contain 18% of the total ex-pit waste volume. Run-off from WD3b will be collected in ECD1 and then pumped to the process plant, if required, or to the treatment plant. Run-off from inactive or rehabilitated areas will be diverted around the sediment and environmental control dams to maximise the efficiency of those structures.

18.19.8 Site Water Management Model

Model Description

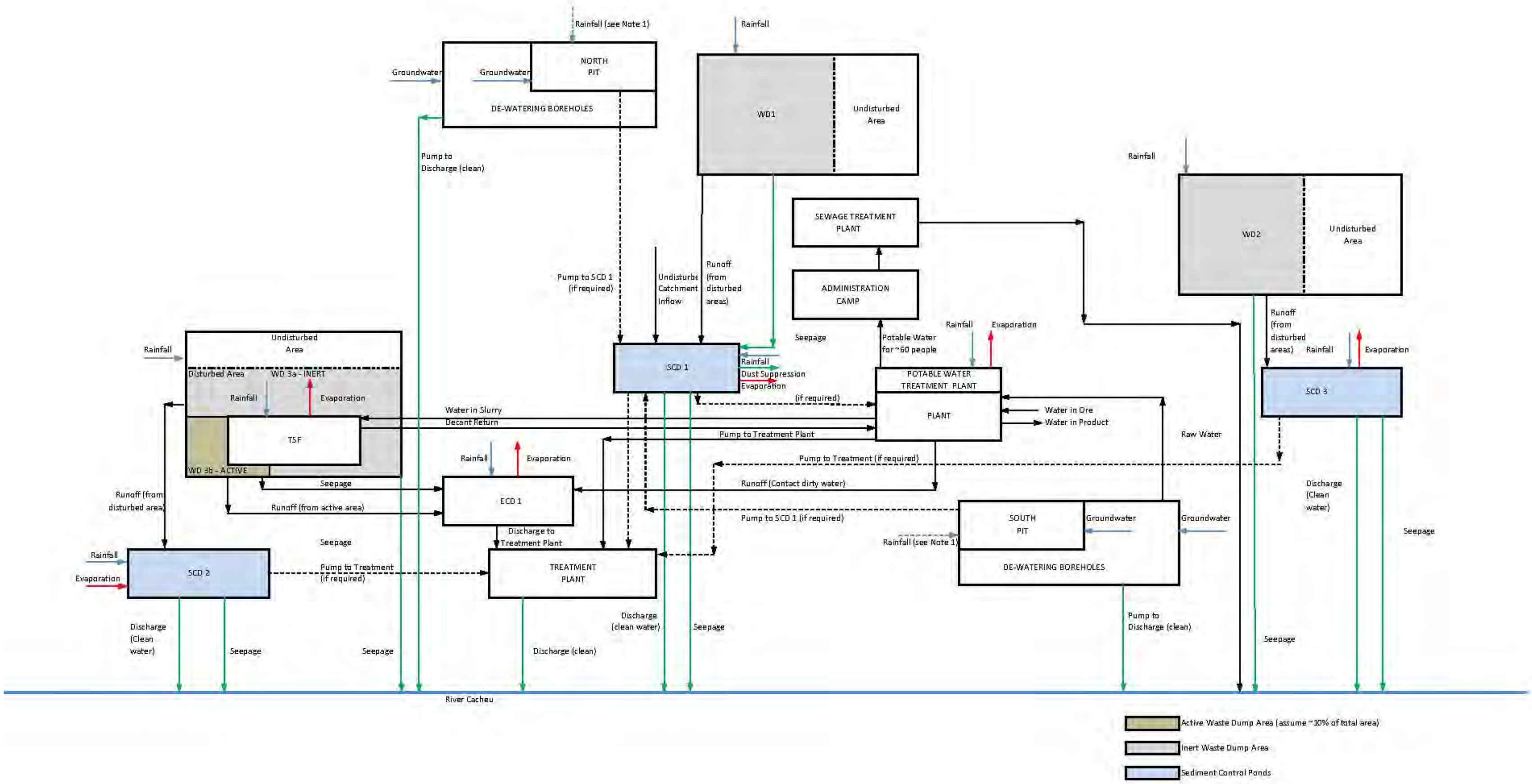
Management of water for the project site is critical in terms of the pumping, treatment and discharge requirements. A site water management model was developed in order to understand the various flows of water to and from the following locations:

- Open Pit De-Watering;
- Process Plant;
- Tailings Storage;
- Waste Dumps;

- Sediment Control Dams; and
- Environmental Control Dam

Figure 18-22 presents a conceptual block model of the site water management system. The model was used to simulate the expected water flows under average climatic conditions throughout the life of the project, and the impact of extreme rainfall events at critical times during the operation.

Figure 18-22 Farim Water Balance Block Model



The implications for the individual structures are discussed in the following sections.

Open Pit De-Watering

Pit inflows during the mining operation are expected to be of the following order of magnitude:

- South Pit - average daily pit inflow of approximately 13,000 m^3/day (150 l/s), ranging between 9,800 m^3/day (113 l/s) and 16,700 m^3/day (193 l/s).

- North Pit - average calculated inflow of 6,500 m3/day (75 L/s) ranging between a peak of 8,900 m^3/day (103 l/s) and 5,100 m^3/day (59 l/s) at the end of mining (Year 26).

Water generated from the pit de-watering operation is considered to be “clean” (as it is better quality than the river water) and will be pumped initially to a sediment control pond providing for settling of suspended sediment and water quality monitoring prior to release into the River Cacheu.

Waste Dumps, Sediment Control Dams and Environmental Control Dam

Of the four proposed waste dumps, three (WD1, WD2, and WD3a) will be used to store inert waste only. Water collected in the sediment control dams downstream of these dumps will provide for settling of suspended sediment prior to release of the water to the environment. Under average conditions the sediment control dams will be expected to generate spillway discharges during the wet season only.

The potentially leachable “active” mine waste will be placed into waste dump WD3b. Run-off from this dump will be contained in an environmental control dam (ECD1) and re-used in the process plant, if required, or treated and released. ECD1 is designed to contain run-off arising from storms up to 1:20 AEP 72 hour event. For storms in excess of this return period, ECD1 will discharge along an engineered spillway into sediment control dam no. 1 (SCD1).

Tailings Storage

The tailings storage is expected to be slightly water positive over the life of the facility. Approximately 2,100 m^3/day of water will need to be removed from the facility to maintain a balanced/slightly negative TSF water balance under average conditions. Supernatant water in the TSF will be impacted water and will therefore be returned initially to the process plant for use in the processing circuit. It is recommended that a decant return rate of 4,000 m^3/day is allowed for pumping to the process plant.

Process (Beneficiation) Plant

The process plant has a design throughput of 1.75 Mt/year and will produce 256,000 t/year of tailings and 172,750 t/y of oversized rejects. Ore will be fed into the process at a moisture content of 18% to 27% and the concentrate is to be shipped at 8% moisture content. Consequently, a surplus of water is expected from the ore processing. The process design criteria indicate that of the 2,232 m^3/hr of process water demand, 2,116 m^3/hr (95%) will be sourced from tailings and concentrate thickening, and filtrate processes. The balance of the process water demand (147 m^3/hr) will be sourced (in order of priority) from TSF decant return, the environmental dam (ECD1) and sediment control dam no. 1

(SCD1). Decant returns from the TSF under average climatic conditions are expected to vary between 30 m^3/hr in the dry season and 165 m^3/hr in the wet season, with an average over the life of the facility of approximately 85 m^3/hr. During periods of water deficit, make-up water will be sourced from ECD1 and SCD1, as required. Pumping capacities of 15 l/s and 30 l/s are recommended for ECD1 and SCD1.

During the wet season the site water balance is slightly water positive and discharge from the plant will be required. As this water is likely to be impacted, treatment may be required prior to discharge to the environment. A treatment rate of approximately 80 l/sec is recommended.

Conclusions

The site will generate continuous discharges of water throughout the operation. The quantities and quality of the water and the handling of the discharges vary according to the source. Recommended minimum pumping capacities between specific sources and destinations are summarized in Table 18-9.

Table 18-9 Recommended minimum pumping capacities

Source	Destination	Water classification	Purpose	Minimum pumping capacity (l/s)
Open pits	Environment	Clean	Discharge	3*
TSF decant	Process water tank	Process	Make-up	50
ECD1	Process water tank	Contact dirty	Make-up	15
ECD1	Treatment plant	Contact dirty	Treatment	80
SCD1	Process water tank	Contact clean	Make-up	25
Process plant	Treatment plant	Process	Treatment	15
Process plant	ECD1	Contact dirty	Storage^	20
Treatment plant	River Cacheu	Clean	Discharge	80

*Per de-watering bore

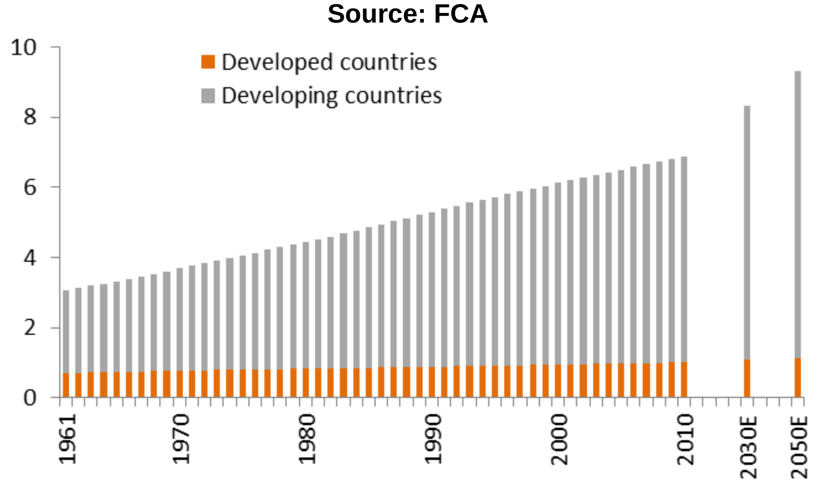
^ storage prior to treatment

19.0 MARKET STUDIES AND CONTRACTS

19.1 Global drivers of fertilizer demand

Phosphate is mainly used in the production of fertilizers (85% of world phosphate utilization). Animal feeds, human food complements (8%), industrial uses (5%) and specialty chemicals (2%) account for the remainder. Global fertilizer demand for phosphate is therefore the main driver for the growth of phosphate rock production as there are no known significant substitutes or alternatives to the application of phosphates in that field.

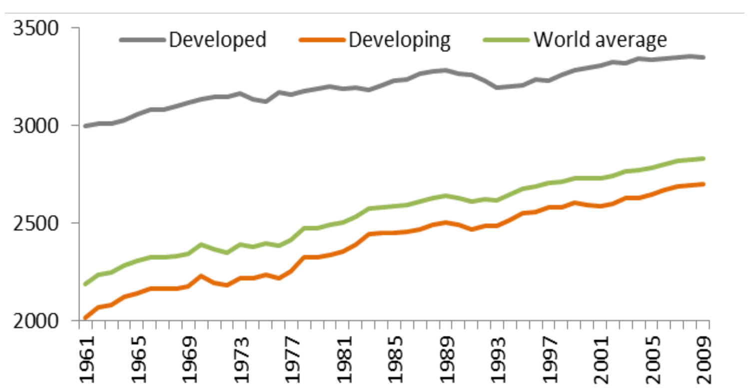
Figure 19-1 World population, bn people
Source: FCA



Fertilizer consumption is closely correlated to world population growth. World population has steadily increased in the past 60 years and is expected to reach 9.2 bn in 2050 (source: United Nations) up from 7.2bn currently (Figure 19-1).

Figure 19-2 Average diet, kcal/capita/day

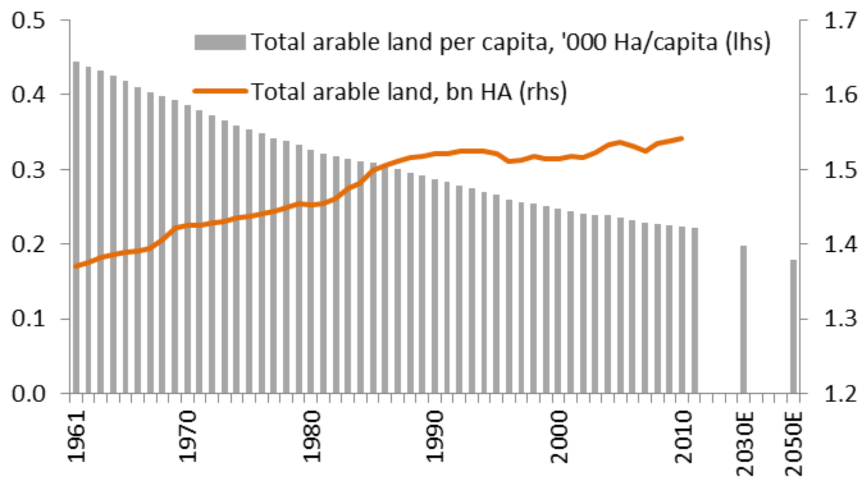
Source: FCA



An additional factor to fertilizer consumption growth is the worldwide increasing calorie intake per capita (Figure 19-2). This phenomenon is particularly strong in the developing world with China and India leading the change. High calorie intake diets are becoming widespread with increasing prevalence of meat, dairy, oilseeds which are all increasing the demand for grain, stockfeeds and agricultural production in general.

Figure 19-3 Total arable land and total arable land per capita

Source: FCA



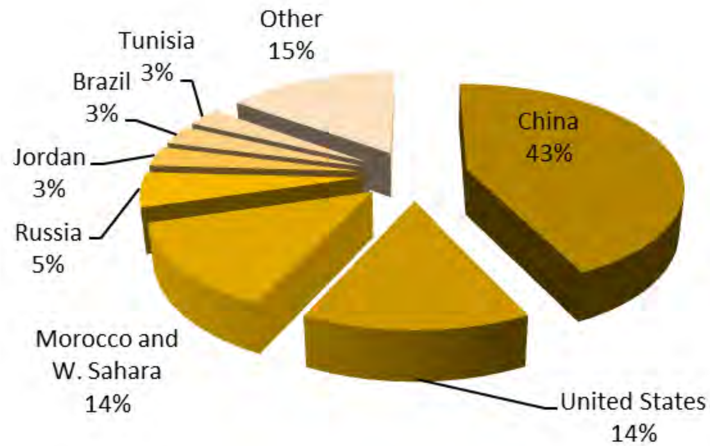
As arable land expansion is being superseded by population growth (see Figure 19-3), the only possibility left is an increase in agricultural production yields. This in turn is the main driver for the increase in fertilizer usage worldwide.

19.2 Global phosphate rock production and reserves

The major phosphate producing countries are China, US and Morocco as shown by Figure 19-4 below. China's production is mainly dedicated to the domestic market.

Figure 19-4 2012 Global phosphate rock production

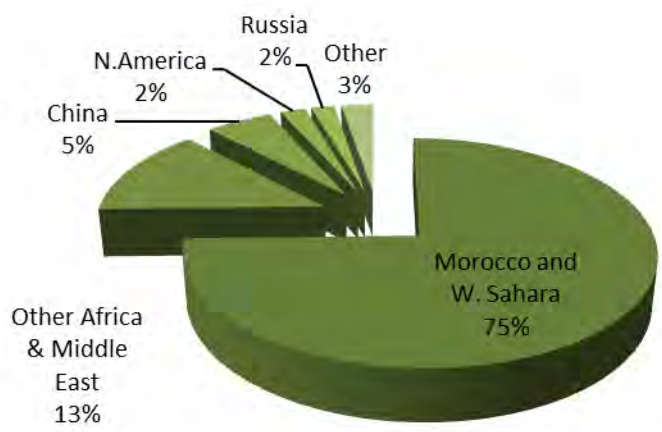
Source: Mosaic, CRU



More than 88% of global phosphate reserves are concentrated in North Africa and the Middle East with a single player, OCP from Morocco holding 75% of world reserves (Figure 19-5).

Figure 19-5 2012 Global phosphate rock reserves

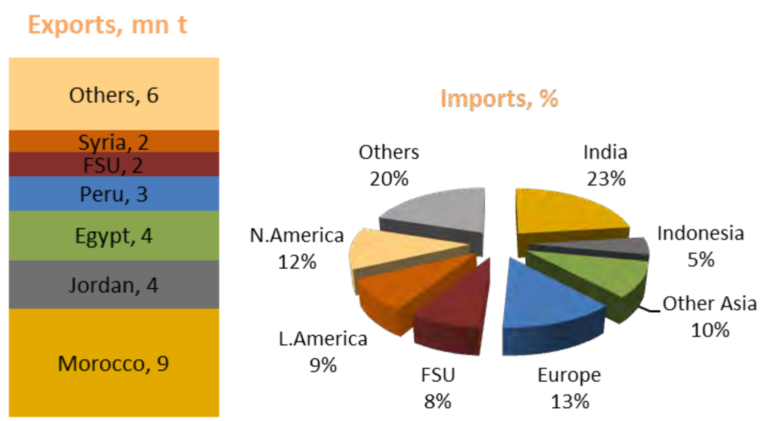
Source: Mosaic, CRU



The dominant players in North Africa and the Middle East also dominate the seaborne phosphate rock merchant market and represent 77% of exports and hence exert a strong control over merchant rock supply. (Figure 19-6).

Figure 19-6 2012 Phosphate rock exports and imports

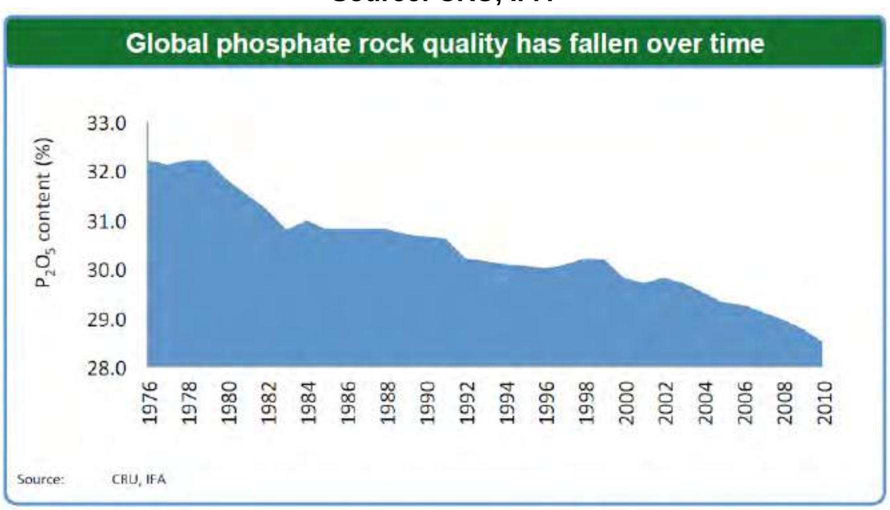
Source: IFA, Fertecon and CRU data, cited by PotashCorp web site



India is the major importer of phosphate rock and phosphoric acid. Morocco, the world's largest phosphate exporter aims to increase its downstream phosphate fertilizer production, reducing phosphate rock shipments to the merchant market. All in all, tight phosphate rock supply is supportive of phosphate rock prices.

Figure 19-7 Global phosphate rock P_2O_5 content

Source: CRU, IFA



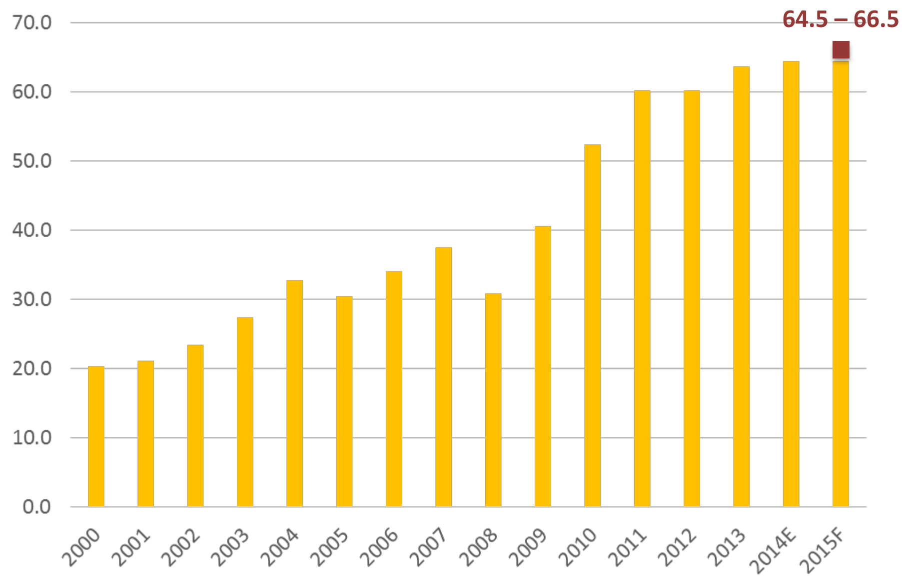
Finally, grades of existing phosphate mines have been steadily decreasing leading to a scarcity of high grade (above 32% P_2O_5 content) high quality merchant phosphate rock as shown by the graph on Figure 19-7.

19.3 Global phosphate rock markets

Phosphate rock consumption grew around 2% per annum in the last ten years. Consumption growth is driven mostly by Asian countries. As a result, global phosphate rock shipments have been growing steadily (Figure 19-8).

Figure 19-8 Million tonnes DAP/MAP/MES/TSP

Source: Mosaic, CRU



Phosphate rock consumption growth at 140 million tonnes in 2012 and a growth 2% per annum makes the addition of Farim's production of 1.32 Mtpa easier to absorb.

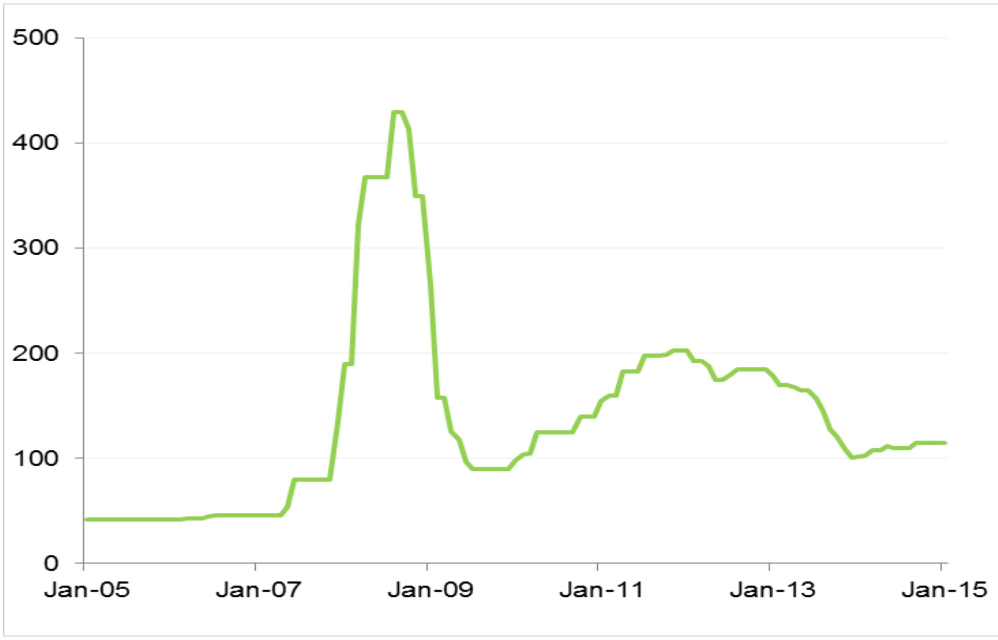
If the data is available, we should project to 2020 as it represents 3 yrs of production.

19.4 Historical prices

The main phosphate rock price reference is the Moroccan K10 FOB price as OCP is by far the largest exporter of phosphate rock.

Figure 19-9 Phosphate rock price, $/tonne

Source: Indexmundi



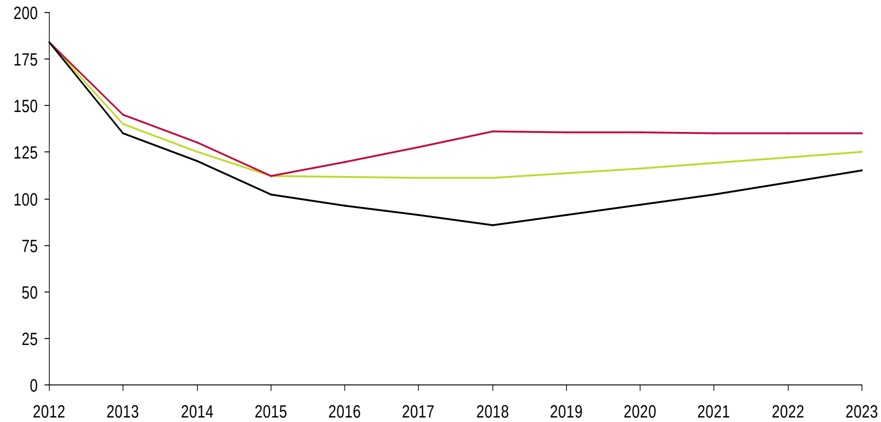
In line with other commodity prices, there has been a considerable movement in the reported phosphate rock price in the last ten years as evidenced by the graph in Figure 19-9. The average price increased from USD $30/t in 2005 to USD $430/t in 2008. Price changes of this magnitude are not the result of a series of supply and demand changes. The particular price shift upwards in 2008 was due to an overall increase in the cost of production of phosphate rock driven heavily by the rising cost of energy, tighter supply of phosphate rock, and a weaker dollar. At the same time the demand for fertiliser products increased strongly. After 2010, prices stabilised between USD $100 and $200/t. The average price for the 2013-2014 timeframe was USD $154/t.

19.5 Price projections

The following chart shows an independent price projection estimated by Integer Research until 2023 (Figure 19-10).

Figure 19-10 Phosphate rock price, $/tonne

Source: Integer



It is expected that the price of phosphate rock will not fall below USD $100/tonne which is assumed to be the floor price for the industry in real terms (downside scenario, in black). In the base case scenario (in green) the price is expected to remain around USD $125/tonne until 2023 due to planned capacity additions absorbed by moderate demand growth. In the upside scenario (in red) strong demand could bring back phosphate rock prices to USD $150/tonne.

19.6 Potential Expected Upside for Farim Rock

Farim's rock target specification is that of a high grade, low deleterious elements premium product. As such, and in line with market practice, a premium to the benchmark Moroccan K10 rock price has been estimated in Table 19-1. In total and according to internal estimates the expected premium of Farim's rock price over the benchmark K10 rock is 9.7%.

Table 19-1 Farim phosphate rock premium to K10 benchmark

COMPONENT	FARIM TARGET SPECIFICATION	K10 BENCHMARK SPECIFICATION	DIFFERENTIAL	ESTIMATED PREMIUM DISCOUNT IN %
P_2O_5 %	34.1%	32%	2.1%	+6.5%
CaO/P_2O_5	1.52	1.60	-0.08	+1.5%
HUMIDITY %	3%	1-3%	0%	0%
MER	0.04	0.03	0.01	-0.2%
C AND ORGANICS %	0.30%	0.17%	0.13%	-0.7%
F - REACTIVE SIO_2 %	0%	2.55%	-2.55%	+2.6%
TOTAL PROJECTED FARIM PREMIUM TO K10 BENCHMARK				+9.7%

The main contributor to Farim's rock premium is the P_2O_5 content that is 2% superior than the benchmark grade. This contributes to a 6.5% price premium over the K10 rock price. The second contributor is the low CaO/ P_2O_5 ratio of Farim's rock relative to its benchmark, adding a 1.5% price premium. The third aspect is the low fluorine level as well as the presence of reactive silica in Farim's rock which possibly adds a 2.6% premium. Higher MER ratios and slightly higher presence of organics reduce the premium by -0.9%.

Notes: Premium/discount formulas used:

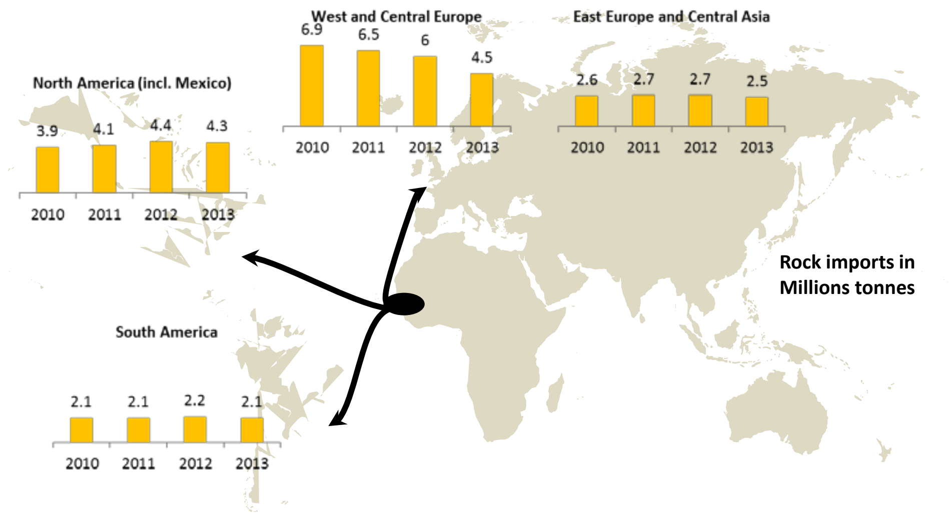
P_2O_5 content premium/discount = [P_2O_5 (Farim) - P_2O_5 (K10)] / P_2O_5 (K10) x Price K10

- CaO/P_2O_5 ratio premium/discount = [CaO/P_2O_5 - CaO/P_2O_5 (K10)] / CaO/P_2O_5 (K10) x Price of sulphur /3 xSulfuric acid requirement per tonne K10.
- MER premium/discount = [MER - MER (K10)] / MER (K10) x Additional ammoniation costs.
- C & Organics premium/discount = (% C- % C K10)x Discount.
- SiO_2 & F premium/discount = [(SiO_2 – F) – (SiO_2 (K10)- F (K10)] x Additional reactive sílica cost.
- Other elements have not been included in the premium calculations as they are expected to be relatively close to the K10 benchmark rock specification.

19.7 Attractive logistical position in the Atlantic basin

Farim's position in the middle Atlantic enables to serve a good variety of markets, see Figure 19-11.

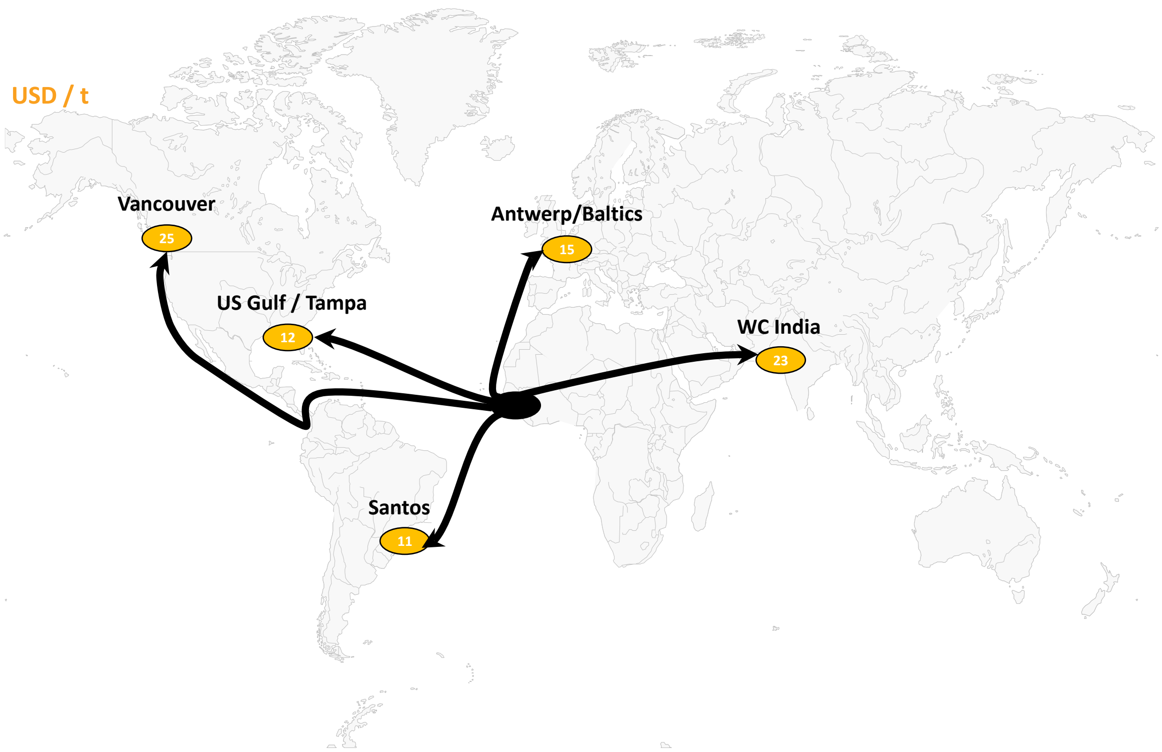
Figure 19-11 Rock imports in million tonnes
Source: IFA



The Atlantic basin market represents a total of 13.5 million tonnes of imports in 2013. While phosphate rock exports for Western Europe have been declining in 2013 due to economic slowdown, other areas such as North America, Eastern Europe and South America have evidenced steady demand or even increasing their rock imports marginally.

Figure 19-12 Dry bulk freight rates from Bissau in USD/t

Source: WF Baird and Associates Coastal Engineers ltd.



Farim's logistical position, see Figure 19-12, allows GB Minerals to bring product to coastal customer facilities in the Atlantic at competitive rates that are very similar to that of the dominant rock exporter in Morocco. As a result, Farim's final landed price which is ultimately what matters to clients should be competitive in the Atlantic basin.

The Qualified Person (QP) has reviewed the marketing studies and analyses, and the results support the assumptions in this Technical Report.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Below is a summary of the environmental and social considerations relevant to the Project, including:

- Environmental, cultural and socio-economic setting and studies conducted;
- Regulatory context; and
- Known environmental issues that could materially impact the issuer's ability to proceed with the planned/proposed mining development.

20.1 Environmental Studies and Assessments Completed to Date

Environmental studies were initiated by GB Minerals in 2011. Socio-economic and cultural heritage studies were undertaken from 2011 through 2012 by Tropica Environmental Consultants Ltd. of Senegal. Physical and biological baseline studies were undertaken by Golder Associates (UK) Ltd. and summarized in an environmental baseline study report (Golder 2014a). Previous environmental baseline studies undertaken between 2011 and 2013 focused mainly on the mine site, and included the Cacheu River (downstream of the mine site) as part of an option that was under consideration in 2012 to barge ore or concentrate down the Cacheu River to a coastal trans-shipment location. Limited baseline studies were also conducted along the highway between Farim and Bissau as part of a slurry pipeline option of moving ore or concentrate to a port.

A draft ESIA was prepared by Golder (2014b) in March 2014 for the mine site component of the Project that did not reflect all the aspects of the current design.

In October 2014, Lycopodium contracted Knight Piésold to complete a gap analysis of the environmental baseline data and the previous ESIA as part of the feasibility study team (Knight Piésold, 2014). This gap analysis formed the basis of supplemental baseline studies undertaken by Knight Piésold in April and May 2015. A summary of previous (2011 to 2013) and 2015 supplemental baseline studies is presented in Table 20-1.

Table 20-1 Summary of Environmental, Socio-economic and Cultural Heritage Baseline Studies

Discipline	Previous Baseline Studies (2011-2013)	Supplemental Baseline Studies (2015)
Meteorology	A meteorology station has operated nearly continuously at Farim since 2011.	Updated analysis of additional meteorological data completed; port site meteorology was described from Bissau climate records.
Air quality	Baseline measurements of particulate	No supplemental data collection deemed

Discipline	Previous Baseline Studies (2011-2013)	Supplemental Baseline Studies (2015)
	matter (PM), sulphur dioxide (SO_2), nitrogen oxides (NO_x) and dustfall collected in 2012 at representative locations at the mine site, along the transportation route and near the port site.	required.
Noise and vibration	Noise measurements collected at receptor locations near the site and along the transport route.	Noise measurements collected at the port site.
Geochemistry	50 overburden samples were collected from 2 boreholes in each of the north and south pit. Samples were composited and analyzed for acid rock drainage / metal leaching (ARD/ML) potential. Chemical analysis for metals completed on 50 ore samples.	The collection of additional overburden samples of overburden is currently underway for acid rock drainage/metal leaching (ARD/ML) potential. Tailings samples from bench scale testing (1 sample) completed and additional testing of tailings from a pilot plant testing underway. Ore and phosphate product undergoing testing including chromium (VI).
Soils	Comprehensive soil sampling program and land capability assessment within the mine site area.	Supplemental soil sampling program conducted at the mine site (metals only), and the port site (metals and soil fertility parameters).
Surface water	Surface water sampling conducted over multiple wet and dry seasons at the mine site.	Surface water sampling was conducted in the vicinity of the port site.
Groundwater	Comprehensive groundwater investigations completed, and one dry season and wet season sampling campaign completed.	Additional wells installed at the mine site and pump tests conducted. Revised groundwater model prepared. Supplemental groundwater quality sampling (dry season) conducted at select wells in the mine and port areas.
Aquatic ecology	Aquatic studies conducted in the River Cacheu and tributaries near the mine site.	Aquatic studies conducted in the River Geba at the port site, and supplemental aquatic studies at the mine site.
Terrestrial ecology	Terrestrial ecology studies conducted in the mine site area.	Terrestrial studies conducted at the port site, with supplemental terrestrial ecology studies at the mine site focusing on biodiversity.
Socio-economics	Preliminary socio-economic surveys and data collection.	Household surveys at the mine and port. Detailed land use mapping at the mine and port site areas.

Discipline	Previous Baseline Studies (2011-2013)	Supplemental Baseline Studies (2015)
Cultural heritage	The mine site area surveyed by a qualified international archaeologist.	The port site area surveyed by a qualified international archaeologist, and a follow-up survey was completed at the mine site.

A round of public meetings were held in May and June 2015 to present the Project plans and to solicit feedback from the Guinea-Bissau Government, local communities and other interested stakeholders. These meetings are in accordance with the company`s Stakeholder Engagement Plan (Knight Piésold, 2015a). The feedback from these engagement sessions will be incorporated into the ESIA that is currently under preparation.

The ESIA will be provided to Equator Principle Financial Institutions (EPFIs) potentially interested in financing part or the entire Project. The ESIA will be translated into Portuguese for submission to the Guinea-Bissau Government, as well as other stakeholders. Summary ESIA information will be prepared and presented to local stakeholders in Portuguese, or presented orally in the local languages of Creole and Mandinga.

20.2 Physical and Biological Setting

The following description of the project setting is heavily drawn from previous work completed by Golder (2014a, b).

20.2.1 Physical and Biological Setting

Geographic Location

Guinea-Bissau is located at approximately 12º Latitude and 15º Longitude. Much of the country is close to sea level, and the Farim mine site area is similarly flat with an elevation change of approximately 20 m over a distance of 4 km between the Cacheu River and the north western edge of the mine site area. The elevation of the wider project study area (mine to port) varies between 5 m and 50 m.

Natural resources found in Guinea-Bissau include: fish, timber, phosphates, bauxite, clay, granite, limestone and unexploited deposits of petroleum. Approximately 10.67% of the land is arable and 235.6 square kilometres is irrigated (Central Intelligence Agency, 2015).

Natural hazards include a hot, dry, dusty harmattan haze that may reduce visibility during the dry season and brush fires. Environmental issues include deforestation; soil erosion; overgrazing and overfishing.

Meteorological and Atmospheric Conditions

The land area of Guinea Bissau is mostly savannah with low coastal plains either colonised by freshwater wetlands (most converted to rice paddies), saltmarshes or fringing mangroves that line the river banks. The climatic and seasonal variations are very distinct in Guinea-Bissau and follow the general West African climate conditions. It is hot and humid all year round with little fluctuation in average temperature. Data collected at the Bissau station indicates that temperatures range from 16.6°C to 38.6°C, with the minimum temperatures occurring in January and the maximum temperatures occurring in April (Golder, 2014a).

There are two distinct seasons in Guinea-Bissau, the wet season and the dry season. During the wet season (June to October), most of the average rainfall is accounted for and the winds are predominately southwesterly. The dry season (November to May) accounts for very little rainfall and the winds are predominately northeasterly. The annual total rainfall at the Farim meteorology station in 2012 was 1,594 mm, which is representative of long-term annual precipitation values reported for Bissau. The majority of the rainfall events are short in duration and have a high intensity. Wind speeds are generally light all year round and are typically less than 5 m/s (18 km/h) (Golder, 2014a).

Air quality data collected at the mine site indicates that the air quality is representative of a natural environment with low concentrations of anthropogenic gases. Particulate matter is elevated at the Project sites (mine and port) due to naturally high concentrations of dust (Golder, 2014a). This is further elevated during Late November to Middle March when the Harmattan winds blow dust from the Sahara in the direction of the study region.

The daytime and night time noise levels in the vicinity of the Project sites regularly exceed the noise limits identified in the IFC`s noise guideline values of 55 and 45 $L_{Aeq\ 1\ hour}$, respectively (Golder, 2014a and Knight Piésold, 2015c). Baseline noise surveys indicate that measured daytime noise levels are typically higher than the lowest measured night-time noise levels. Daytime noise levels are most influenced by human activities. Noise levels increase around dusk due to the calling of crickets and toads, which steadily reduce as the night passes.

Tailings and Waste Overburden Geochemistry

A description of the geochemical properties of waste overburden and tailings is provided in Sections 18.19.2 and 18.19.4, respectively.

Mine Site Hydrogeological Conditions

Knight Piésold (2015b) has conceptualized the hydrostratigraphy of the Mine Site as follows:

- An overburden layer comprising sands, clays and gravels, extends from the land surface to the absolute elevation of -30 m RL. This unit can be considered an unconfined aquifer and is shown to be in limited hydraulic connection with the River (Rio) Cacheu due to the presence of extensive superficial clay in the lowland plain.

- A blue clay horizon is not continuous, occurring in localized areas only and ranging in thickness.

- A calcareous layer (limestone) lies beneath the orebody. Water levels in this unit sit at a higher elevation than those of the overburden suggesting that groundwater in this unit is under pressure with a vertically upwards hydraulic gradient. Field observations do not support this being a dolomitic limestone and instead indicate that the unit be better characterized as a calcareous clayey friable sandstone, justifying the low hydraulic conductivities of this layer.

Water levels respond to seasonal rainfall and decrease during the dry season and rise during the wet season. Groundwater provides baseflow to surface water bodies, including the River Cacheu and its tributaries. Groundwater elevations indicative of upward flow and groundwater discharge have been observed in the south area of the mine site. As a result, surface water bodies are potentially sensitive to losses in baseflow due to reductions in groundwater levels. These reductions could lead to a shorter duration for ephemeral stream flows (Golder, 2014c).

The quality of groundwater collected by Golder (2014a) in the mine site area is reflective of the undeveloped environment. Most of the samples collected met the World Health Organization drinking water guidelines (WHO, 2011). The salinity of the water measured as electrical conductivity is between 23.7 and 922 µS/cm. The chloride and sodium concentrations for all hydrogeological units are generally low, indicating rainwater recharge rather than a tidal influence from the River Cacheu. Groundwater recharge and quality immediately adjacent to the River Cacheu and Rio de Bunja (a tributary in the mine site area) are influenced by the tidal river during the wet season. Of the trace metal elements tested in groundwater, only the iron and manganese content were identified at concentrations above the aesthetic objectives for drinking water. The pH was also found to be outside the aesthetic objectives range in several of the samples.

Aquatic Environment in the Mine Site Area

The mine site is located adjacent to the Cacheu River, a major river that meanders through the study area in a southwesterly direction. There are also four streams within the mine site area that report to the Cacheu River: Rio de Banim, Rio de Bunja, Rio de Cavaras Marinhos, and Rio de Caur.

Adjacent to the mine site, the channel width of the Cacheu River is approximately 150 to 200 m and the maximum flow velocities in the river range between 1.1 to 1.5 m/s during the dry and wet seasons, respectively. The River Cacheu experiences a semi-diurnal tidal influence (two high tides and two low tides each day), thus forming part of the estuarine environment. By strict definition, any part of a river that is tidal is considered estuarine. This is further supported by the development of the mangroves and salt marsh areas, and the presence of marine/estuarine species of fish captured within the River Cacheu in the vicinity of the mine site (Scherman, Colloty & Associates (SC&A), 2015). The maximum tidal variation at Cacheu near the mouth of the estuary is about 3 m, and the tidal variation at a survey location downstream of Farim at Binta was measured at about 2 m (SC&A, 2015). Each of the streams transecting the mine site is also tide influenced.

The River Cacheu banks to the north and south are relatively flat and are susceptible to inundation during periods of high rainfall and/or high tides. The floodplain extends 1,500 m to the north and south. The river banks are composed of fine-grained sediments and are well vegetated with mangroves and other vegetation. Most of the river is in a relatively pristine state with the majority of the mangrove and beach shorelines being undisturbed. The fine-grained sediments are exposed at low tides and are subject to erosion by wave action and currents. The river is dominated by ongoing transport of fine-grained sediments; river bed sediments range from silt to clay at Farim, to sand sized particles near the estuary mouth. The monitored section of the river exhibits a quick response to rainfall events given sufficient antecedent rainfall. During the recent surveys in 2015, grab samples indicted areas of excessive scour within the larger river bends, which yield mostly weathered ferricrete in the samples (Golder, 2014a).

The aquatic ecosystem is driven primarily by the natural wet and dry season fluctuations resulting in nutrient and sediment transport to the coastal areas, coupled to the daily tidal regime. The highest diversity of fish and invertebrates (in faunal and water column) was observed in the more saline sections of the river. Seagrasses were not observed within the length of the river studied; their absence likely owing to the naturally high turbidity and lack of suitable intertidal habitat (sandy areas). Shellfish and other invertebrates reproduce throughout the length of the river and estuary. The mangrove ecosystems along the tributaries serve as nursery areas for fish. A study of fish tissue analysis shows that the system is not contaminated from anthropogenic sources/activities (Golder, 2014a).

Aquatic Environment at the Port Site

Aquatic ecology studies were undertaken within the River Geba at the Port Site area in May. Like the River Cacheu, the River Geba is estuarine and heavily influenced by ocean tides. At the port site location, the river is almost 7 km across, with depths measured during the spring high tide ranging from 3 m to 28 m (SC&A, 2015). The tide within the River Geba ranges from 3 m at the most eastern end of the Canal de Caio and 6 m near Ponta Chugue (Baird, 2015). The measurements were taken during spring tides and influenced by strong westerly winds that occurred during the sampling period. These winds combined with the large volume of water that moves during the tidal cycle accounted for strong currents (7 to 8 m/s) and local occurrences of large standing waves (0.6 to 1.2 m) during the sampling period.

The substrate in the vicinity of the port site consisted of fine mud with a depth of 5 to 8 m. The dominant fish species in the area captured during biological surveys was: Royal Threadfin, Sea catfish, Grunter, juvenile Sole and Kob (SC&A, 2015). All of these species are well adapted to turbid, high energy environments. Local fishermen confirmed that these fish are the dominant species and that they form an important part of their fisheries resource.

The wetlands areas are located along the floodplains, most of which have been converted to rice paddies, with only a band of mangrove (*Rhizophora* & *Avicennia*) remaining along the shorelines.

Terrestrial Ecology

Terrestrial ecology in the mine site area is influenced by the Cacheu River and human use of the area proximate to the river. Wetlands and mangrove communities line the river, and natural forest border the settled areas. Within the settled areas, secondary forest communities, rice paddies and upland agricultural fields dominate the landscape.

Mine Site

Baseline studies at the Mine Site focused on determining the diversity of mammals, birds, reptile, amphibians and arthropods within the study area. Overall, 259 species were recorded to be present in the study area distributed amongst the groups as follows (from Golder, 2014a):

- Mammals (15 species) - One species (Red Colobus) is listed as endangered on the IUCN Red List. This number of species represents a low diversity of mammal species in comparison to Guinea Bissau.

- Birds (75 species) - One species (Hooded Vulture) is listed as endangered on the IUCN Red List). This number of species represents a high diversity of bird species in comparison to Guinea Bissau.

- Arthropods (124 species) - No IUCN Red List species were identified.

- Reptiles (11 species) - No IUCN species were identified. Low numbers were observed of each species.

- Amphibians (five species) - No IUCN species were identified. This number of species represents a low diversity of amphibians in comparison to Guinea-Bissau.

It was evident during May 2015 terrestrial surveys that large areas (present estimates are between 600 to 800 ha) of natural forest have been converted into Cashew production areas since 2012 (SC&A, 2015). Large areas in the Northern Pit had been slashed and burned in order to clear these areas for crop production. The maps are currently being updated based on the most recent aerial images and ground-truthing.

Port Site

Terrestrial ecology studies were also undertaken SC&A at the Port Site area in May 2015 and results are pending. Land use is currently dominated by cashew trees, rice paddies and secondary thicket grassland areas. The grassland is covered mostly by a single species of thatching grass, which seems to be managed or promoted. The grass is then harvested in the dry season, bundled, and sold as roof thatching. One additional plant species was observed in the 2015 surveys compared to previous Golder surveys (Golder, 2014a), but none of these species appear to be of conservation concern.

20.2.2 Socio-Economic and Cultural Setting

National Socio-Economic Setting of Guinea-Bissau

The national socio-economic environment of Guinea-Bissau has been influenced by a history of political instability since the country gained its independence from Portugal in 1973. In 2012, the national population of the country was 1.7 million. Only 14% of the population speak the official language (Portuguese). Guinea-Bissau is ranked 177 out of 187 countries according to the 2014 UNDP Human Development Index and has one of the lowest per capita gross domestic products in the world

(United Nations Development Programme, 2015). Most of the population (44%) speaks Crioulo, a Portuguese-based creole language. There are many ethnic groups, with 7% of the population classified as an indigenous ethnic group (Papels). Golder (2014b) did not identify the presence of any indigenous ethnic group such as the Papels. This will be validated through the supplemental socio-economic studies currently underway by Knight Piésold.

Guinea-Bissau is divided into eight administrative regions in addition to the autonomous district of Bissau. The regions are subdivided into districts that are administered by District Administrators. In total, there are 37 districts. The region of Oio, where the project is located, is in the northern part of the country and consists of five districts: Bissora, Farim, Mansaba, Mansoa and Nhacra. The Oio region is predominantly rural, with a population estimated at approximately 215,000 inhabitants (15% of national population), and characterised by a diverse range of ethnic groups. The total population in the three districts (Farim, Mansoa and Mansaba) is estimated to comprise 64% of the population of the Oio Region. The populations of these districts live in rural villages, with only one or two towns in each district. Farim is the second most populous district in the region, with approximately 8,681 inhabitants. Outside of Farim, the population in the villages rarely exceeds 500 inhabitants.

The local social environment can be described as rural villages that are largely dependent on small-scale agriculture for both household subsistence and income generation, and larger peri urban settlements where there is more social infrastructure such as schools and religious establishments. In general, the whole project area lacks adequate social infrastructure such as health care facilities, schools, sanitation, water systems, and waste management. Many households reside in compounds and land ownership is followed through the integration of traditional law such as customary land management practices and legal forms of ownership. Decision-making is primarily through consensus facilitated by the village leaders or committees.

The larger villages have trade businesses and a more cash-based local economy. The smaller communities in the project area and along the transport route engage predominantly in subsistence agriculture, with the trade of any agricultural surplus for cash income. Natural resource-based livelihoods are also predominant. Livelihood activities entail cultivation of cashew, maize, millet, sorghum, rice and fonio, which are commonly grown in the area for consumption or sale; the production of natural resources use as home building materials and medicinal products; fishing, especially in villages along the River Cacheu and near the Port Site on River Geba; livestock rearing; and the production of salt, which is undertaken predominantly by women.

Socio-economic Conditions in the Local Study Area

Socio-economic studies were undertaken within the mine site and transport route study areas in 2011 and 2012 (Tropica, 2011 and 2012). Supplemental socio-economic studies are currently underway by Knight Piésold at the mine and port sites. Therefore, the discussion of socio-economic conditions in the local study area (LSA) focuses primarily on the communities close to the mine site. The conditions at the port site do not vary substantially from that at the mine, except that fishing activities are focused on the River Geba. The Cacheu River is considered to be a more important fisheries resource than the Geba River based on discussions with the local fishermen and the Department of Fisheries. Together both river systems only contribute a small fraction of the volumes yielded from offshore fishing activities. The following summarizes the social conditions within the mine site LSA:

- **Ethnicity:** The mine site area includes eight ethnic groups: the Mandingas (66% of the population), the Mansonkas (17%), Fula (7.6%), and Balantes (6%). Minority groups include the Manjak, Pepel and Mancagnes. Households in the Farim area are predominantly inhabited by Mandingo (40%), followed by the Fulani (27.6%) and Balante (21.5%).

- **Housing:** Households are located in clusters as rural villages rather than widely distributed. Households may comprise of a single family home with a single residential structure or a compound comprised of multiple buildings that support multi-generational family members. Household sizes vary between four members to over 25 members, with an average household size consisting of 10 members. Houses are predominately made of clay, corrugated iron roofing and have between four and seven rooms. With regard to ownership, 25% of households have title to the land, 11% have an occupancy permit and more than half (55%) have traditionally determined residential authorization.

- **Mobility:** Considerable mobility is experienced in Farim and its surrounding villages, especially among the young adult population. Mobility is often driven by a search for employment in Bissau, neighbouring countries (e.g., to Senegal, Gambia, and Cape Verde), and Europe (Portugal and France). The villages of Tambato, Canico, Tumana, Salikénié and Farim town are mostly affected by migration.

- **Religion:** Islam is the predominant religion (71% of the population) in the area and is practiced by the Mandingo and Fulani. Christians represent 25% of the population, while paganism is practiced by 4% of the population. These latter religions are mainly practiced by the Balante.

- **Social Organization:** Compounds or homesteads are often shared by more than one related family headed by a ‘chief’ who is the father or the grandfather. Families also share the agricultural land. Monogamy is more common (51.8% of respondents) than polygamy. In general, women and youth have the responsibility for most domestic tasks.

- **Decision-Making:** Decision-making is primarily through consensus facilitated by the village leaders or committees. The village chief (or committee) invites the heads of families and youth representatives and, in some cases, women's representatives when matters need to be discussed and decided upon. Decisions are made only after sufficient discussion and when each had the opportunity to express their opinion. Heads of villages are under the authority of the administrator of the district to which they report. The status of village head is usually held by the founding family of the village and is transferred within the family over generations.

- **Social Infrastructure/Amenities:** There is a basic hospital in Farim that has been supported by the Project to improve ward facilities. There is also a Christian church and mosque in Farim. There is a shortage of schools in the study area. Where schools are present, they are mostly temporary shelters.

- **Water Supply:** Villages and Farim town use traditional wells and hand pump-operated boreholes for domestic water. There is no reticulated sewerage system in the area and domestic (solid) waste is dumped in uncontrolled spaces.

- **Roads:** Roads within the LSA are generally unpaved dirt roads. Farim attracts daily visitors from surrounding villages to access services (mosques, churches, health care, education, and recreation) and commerce such as buying and selling at the market. Most travel is by foot or bicycle with motorcycles being the most frequently used form or motorized travel.

Economic Activities in the Mine Site Area

The following summarizes the economic conditions within the mine site LSA:

- **Access to Land:** Land is administered following traditional law by customary authorities. Thus, the law has changed the basis of ownership through the integration of customary land management practices with legal forms of ownership. Most households (93% of households surveyed in 2012) are actively cultivating land. Of this, only 13% reported holding title to the land they cultivate, while 55% were granted access to land through traditional administrative means, and 3% cultivate fields without any formal approval.

- **Subsistence Agriculture:** Maize, millet, sorghum, rice and fonio are commonly grown in the area for consumption or sale. Maize is the most important crop, being cultivated by more than 51% of the households. However, the cultivation of cashew plantations is critical to generating cash income. The strong market links in the region support significant local investment in cashew tree planting and processing of cashew nuts. In terms of land-take, Cashew trees are the dominant form of local land-use. The proportion of households involved in other crops (e.g., millet, beans.) is between 3% and 15%. Rice, although a staple food, is cultivated by only 12% of households. There are food gardens in several villages, managed mostly by women who have their gardens either around water sources (ponds, wells or boreholes) or in their own compound. Vegetables such as okra and tomatoes are intercropped with the main field crops.

- **Food Security and Income Generation:** Food deficit was widely reported by households despite the availability of farmland. Food shortages are caused by limited access to agricultural equipment and fertilizers, poor soil quality and impacts on productivity by local salt water intrusion from the River Cachue. Some of the produce that is cultivated in home gardens and fields is consumed by the growers and the remainder sold. Peanuts, cashews, cassava and beans are particularly important cash crops. The Project area is one of the most important regions in the country for producing peanuts, which are primarily sold in Senegal through a complex network of traders.

- **Salt Production:** Almost all women in the mining area are engaged in salt harvesting during the dry season. Using rudimentary equipment, the salt is mined from sand taken from rice fields that became salt-affected (tann) as a result of saltwater flooding the plains.

- **Livestock:** Almost 93% of surveyed households had livestock (cattle, sheep, and goats). Pig farming is generally practiced by the Balante and Manjak women, with an average of ten animals per household. Family ceremonies create the main opportunity for the sale of livestock.

- **Fisheries:** Fishing in the Farim area is practiced by 31% of households. Daily catches vary between 10 kg and 15 kg per individual and between 400 kg and 450 kg for group expeditions. There are roughly 43 fishermen grouped in an association in Farim, using a fleet consisting of 15 canoes. Within the River Geba and in the vicinity of the port site, preliminary results indicate that the local fishing groups are divided in fishing areas based on the location of their village and closest landing site (Porto). The proposed port is located within the Chugue community's fishing area, which is fished mainly between August and April, using 100 to 200 m long lines baited with small fish bought elsewhere. Due to the rocky nature of the river bed directly adjacent to the Chugue shoreline, the fisherman prefer to set their lines on the opposite bank near Jabada, which is 10 to 11 km from the proposed port site. The remaining months (May to July), all fishing activities are halted due to the strong currents and the presence of large numbers of shark that damage their long lines. These communities then revert to Cashew production/harvesting. The Chugue community also produces rice. Small nets are utilized when the paddies are flooded to catch the small fish trapped in the adjacent wetland/paddy areas.

- **Natural Resource Harvesting:** Forest products are used as food products, for home building material, and for medicinal products. Edible fruit (baobab fruit, palm fruit) is harvested in season, as are fibres, leaves (baobab leaf), sap extracts (palm wine), wood (90% of domestic energy), honey, and several medicinal plants. Products that are used and marketed include charcoal, baobab fruit, palm wine and palm fruit. Houses are built using material directly harvested from the natural surroundings (e.g., thatch, palm leafs, and wooden poles).

- **Landscape:** Four main landscape types were identified in the mine site area during the baseline assessment: river corridor, cultivated river valley, undulating farmland and woodland, and dense forest. None of these landscapes were determined to be particularly rare. Apart from the River Cacheu, there are no nationally or internationally recognized geographical features or landmarks in the mine site study area. There are many very old trees, including giant Baobab trees within the LSA, which have become the focus of the villages and the surrounding area. Some villages such as Tabandinto have been named after local tree species. Some of these mature specimens have spiritual and/or cultural significance.

Cultural Heritage

A number of cultural heritage features were identified within the mine site area, including archaeological sites, cemeteries and spiritual sites (living/intangible cultural heritage). Cultural heritage features were identified in the vicinity of the port site, but beyond the proposed footprint. No evidence of critical cultural heritage (as defined by IFC Performance Standard 8; IFC, 2012) was identified at either of the project development areas (ERM, 2015). Archaeological remains were predominantly pottery shards and other fragmentary remains of low to moderate cultural heritage significance. Field surveys confirmed the presence of three Muslim cemeteries within the mine study area (Golder, 2014a; ERM, 2015). One of these cemeteries is of high sensitivity and located near the village of Saliquenhe Ba. It contains the grave of a well-known imam who lived over 100 years ago and is sometimes visited by people from within and outside the region during an annual festival. In addition, a sacred grove (or holy forest) is located south of Saliquenhe Ba and is of local to regional importance (ERM, 2015).

20.3 Regulatory Context

20.3.1 Current Regulatory Status

A Mining Agreement was negotiated and signed between the Ministry of Energy and Natural Resources and GB Minerals AG on May 1, 2009. GB Minerals AG is a Switzerland-based entity 100% owned by GB Minerals Ltd. The Mining Agreement allowed for the subsequent issuance of the following mining leases to GB Minerals AG on May 28, 2009:

- Mining Lease No. 001/2009, issued on May 28, 2009, grants the company a Mining Production Licence;
- Mining Lease No. 004/2009, issued on May 28, 2009, provides GB Minerals AG with a Mining Licence.

GB Minerals is in good standing on both mining leases.

The Mining Agreement is considered the global agreement aggregating and coordinating the above licences and any other agreements or conditions relative to the Project. The Mining Agreement in its entirety includes:

- **Operating**, **Environmental** and **Social** plans which were submitted to the Government of Guinea-Bissau on July 1st, 2015 (approval pending – awaiting comments);
- Submission and approval of an **Economic** and **Technical** Feasibility Study as well as an **Environmental** and **Social Impact Assessment** (pending);
- **Mining Lease** (granted);
- **Mining License** (granted); and
- An **Incentive Annex** (pending).

The Mining Agreement provides GB Minerals with the right to construct and develop a mine to exploit the Farim phosphate deposit, and to construct and operate a port facility and any bridges, roads, transportation pipeline infrastructure required to connect the mine to the port site. The Government commits within the agreement to make immediately available the lands required for port infrastructure at the Ponta Chugue area.

In turn, the Mining Agreement requires GB Minerals to:

- Exploit the resource as per good international industry practices and in accordance with a Mining Operation Programme;

- Comply with environmental protection rules outlined in an Environmental Plan and the legislation and regulations applicable in Guinea-Bissau at the time of signing the Mining Agreement;

- Comply with the Social Program concerning employees who are national citizens and to train and to grant medical assistance to any person or employee used or working on the Project.

The key environmental provisions of the Mining Agreement are as follows:

- The licensee will take appropriate reasonable measures to ensure that its operations will not lead to any unnecessary adverse impacts to the environment, as per an approved Environmental Plan and any amendments;

- The licensee will compensate for damages caused by mining by rebuilding partially affected physical locations, where and when appropriate;

- The licensee shall have no responsibility for any environmental damage, except where gross negligence or wilful intent is demonstrated;

- The licensee shall not be held liable for environmental damages that may result from port infrastructure and roadways the licensee has undertaken to build as compensation for the Mining Rights granted under the Mining Agreement, except for in the instance of gross negligence or fault behaviour;

- Provisions regarding the timely issuance of permits/approvals to allow the mining project to proceed;

- Mining is to be undertaken according to a Mining Operations Plan.

An Environmental and Social Impact Assessment (ESIA) is currently being prepared by Knight Piésold. The ESIA will meet National (Guinea-Bissau) and international best practices, specifically the Equator Principles III (World Bank Group, 2013) and the International Finance Corporation (IFC) Performance Standards for Environmental and Social Sustainability (IFC, 2012).

There are no known national requirements to post performance or reclamation bonds. However, compliance with the IFC Performance Standards and World Bank Equator Principles requires that a Mine Reclamation and Closure Plan (MRCP) be prepared and that funding for closure be by either a cash accrual system or a financial guarantee by a reputable financial institution (IFC, 2007). Further discussions on mining closure planning and the posting of financial assurance are included in Section 20.6.

20.3.2 Applicable National Legislation and Regulatory Processes

The Constitution of Guinea-Bissau establishes sovereign rights for the Republic of Guinea-Bissau for the preservation or exploitation of living and non-living natural resources. Further to the constitution, a number of laws related to environmental protection and management have been passed. The legislation most relevant to the Project is summarized below.

Mining and Minerals Law

Law 1/2000 (the Mining and Minerals Law) regulates all issues related to the exploration and commercial production of mining substances that exist in the soil or subsoil and in the territorial waters, with the exception of oil. All mining resources in Guinea-Bissau belong to the State and property rights and the issuing of licenses/permits is the sole responsibility of the government. The Mining and Minerals Law sets out the procedures which enable individuals and entities (national or foreign) to be issued with mining leases, licenses, and rights.

Basic Law on the Environment

Guinea-Bissau has developed a framework law on the environment that lays the foundation for environmental policy and environmental assessments. Law No 1/2011 of 2 March 2011 approves the Basic Legislation on the Environment. This law defines the basic concepts, norms, and principles related to the protection, preservation and conservation of the environment. It aims to improve quality of life through the management and rational use of natural resources, to achieve the sustainable use of such resources.

Environmental Assessment Law

Law 10/2010 (the Environmental Assessment Law) regulates environmental and social impact assessment in Guinea-Bissau. The Environmental Assessment Law sets out the types of projects for which an ESIA is required. The project categories are consistent with the World Bank Group`s Equator Principle 1 and the IFC's practices. The Farim Phosphate project is classified as a Category A project due to the potential for negative impacts. As such, a full ESIA needs to be completed for this project.

Law 10/2010 details the ESIA processes to be followed, requirements for public consultation and disclosure, the components of the studies to be undertaken and resulting reports, and the government agencies that will be involved in the assessment process. Requirements are set for environmental and social management plans, which must present recommended mitigations, monitoring, capacity building, and a schedule and cost estimate to implement the mitigation measures.

National Environmental Assessment Process

The key national regulatory authorities involved in permitting and environmental management of extractive industries are as follows:

- **Ministry of Energy and Natural Resources** - Regulates the mineral industry in Guinea-Bissau, implements its mining policy and regulations, issues mining leases, and develops geological studies and maps.

- **Secretary of State of Environment and Tourism** - Responsible for implementing Guinea-Bissau's environmental policy.

- **Célula de Avalição de Impacte Ambiental (CAIA) -** The lead authority responsible for coordinating review of the Project's ESIA. This department is responsible for ensuring, through collaboration with other relevant government departments, that all development projects are analysed for their potential impacts. It is also responsible to ensure that follow-up monitoring is completed and that projects are compliant with the environmental assessment process during operations.

The Secretary of State for the Environment will make a recommendation to the Ministry of Natural Resources and Energy regarding the implementation of the Project based on CAIA's review of the ESIA. CAIA will then issue an environmental licence that is either a compliance declaration that gives the project proponent one year to implement initial management measures or a compliance certificate that gives the proponent a licence to operate for one to five years. The law further establishes the government's authority to conduct environmental audits (at the expense of the proponent) to check compliance with the conditions of the environmental licence.

20.3.3 International Standards and Guidelines

GB Minerals has elected to seek financing from Equator Principle Financial Institutions (EPFIs) and to complete an ESIA that meets the following standards:

- World Bank Group's (2013) Equator Principles III.

- IFC (2012) Performance Standards for Environmental and Social Sustainability.

In addition, the Project and ESIA will apply the IFC's Environmental, Health and Safety (EHS) Guidelines, and incorporate best practice guidance provided by the International Council of Mining and Metals (ICMM).

20.4 Waste and Management During Operations

Mine waste and water management planning is described in Section 18.19. The mine waste and water management plans are based on the following:

- Diversion of non-contact water away from mining areas and facilities;

- Backfilling of the open pits with waste overburden to the extent practical under the mine plan;

- Separate surface disposal of waste overburden based on the potential risk of the material to leach metals;

- Collection of runoff from clean overburden in surface waste dumps (surcharge waste dumps located over the backfilled open pit, and external to pit waste dumps) using sediment control dams (SCDs), for discharge to the local environment;

- Placement of tailings in a lined containment facility that will be capped at closure;

- Placement of potentially leachable waste overburden within a dedicated overburden stockpile associated with the TSF, forming an integrated waste landform;

- Consolidation of mine effluents with metals concentrations potentially above selected discharge limits (IWL runoff, process plant water, groundwater from dewatering wells, groundwater from in-pit sumps) into an environmental control dam (ECD) for monitoring prior to discharge; and

- Provision of treatment of water from the ECD prior to discharge, if required.

Mine effluent discharge guidelines will be adopted from the IFC`s Environmental, Health and Safety (EHS) Guidelines, or other site-specific water quality objectives developed in consideration of receiving water use and assimilative capacity, as permitted under the EHS Guidelines.

As described in Section 18.19, further geochemical evaluation is ongoing and will inform further design of mine waste management facilities.

20.5 Mine Closure

The National requirement to develop a Mine Reclamation and Closure Plan (MRCP) is embedded in Law 1/2000; the Mining and Minerals Law (see Section 20.3.2). The law also establishes, among other things: the conditions to be met for the issue of or an extension to mining leases or mining rights, requirements to assess any environmental impacts, and the requirement to develop an Environmental Plan to rehabilitate and compensate for environmental and social effects arising from mining activities. In addition, the environmental management plan should comply with all specification and practices established by international standards and regulations.

The IFC (2007) Environmental, Health and Safety (EHS) Guidelines for Mining requires a MRCP to be prepared in draft form prior to production that clearly identifies allocated funding to implement the plan. The costs associated with mine closure and post-closure activities, including post-closure care, should be included in business feasibility analyses during the planning and design stages. Funding to cover the cost of closure at any stage in the mine life, including provision for early, or temporary closure, should be by either a cash accrual system or a financial guarantee provided by a reputable financial institution. The two acceptable cash accrual systems are fully funded escrow accounts (including government managed arrangements) or sinking funds. Mine closure requirements should be reviewed on an annual basis and the closure funding arrangements adjusted to reflect any changes.

A preliminary MRCP and closure cost estimate has been prepared as part of the feasibility study. The MRCP adopts the IFC closure objectives, as follows:

- Future public health and safety are not compromised;

- The after-use of the site is beneficial and sustainable to the affected communities in the long term;

- Adverse socio-economic impacts are minimized and socioeconomic benefits are maximized.

The MRCP contemplates the progressive rehabilitation of a number of facilities at the mine site including the overburden waste dumps and the north and south open pits. The south pit and the majority of the north pit will be backfilled with waste overburden. A portion of the north pit will not be backfilled; the void in the north pit will be allowed to flood to form a small pit lake at closure. The IWL and onsite landfill will be capped with a suitable cover to prevent water ingress. Buildings, machinery and equipment will be decommissioned and removed from site for salvage or resale. Disturbed areas will be covered with stockpiled topsoil and revegetated. As much as practically possible, the land will be restored to provide stable landforms suitable for the agreed-upon future beneficial land uses.

At the port site, buildings, machinery and equipment will be decommissioned and removed from the site. Remediation will be undertaken, as required, so that the port site is compatible with future commercial or industrial land uses. The wharf structure will not be decommissioned, under the assumption that the Government or other private interests will wish to assume control of the site for future beneficial use.

Post closure monitoring and maintenance will take place for a period of at least five year to verify that the site has been returned to a physically and chemically stable state that is compatible with and capable of sustaining the agreed upon final land uses. Furthermore, the MRCP commits to developing post-closure social management plans to address potential adverse socio-economic impacts of closure as part of the company's Community Development Plan.

Closure costs were estimated at USD $5,550,243.

20.6 Water Management Post-Closure

Post-closure water management at the mine site will consist of the following:

- Seepage water from the IWL will continue to be collected and pumped back to the water treatment facility until the facility has been capped (to reduce infiltration) and seepage water quality meets discharge parameters.

- Surface and storm water will continue to be diverted away from the mining wastes and managed through the use of a constructed surface water management system.

- Dewatering boreholes will be decommissioned and groundwater levels are expected to recover quickly within the mining and nearby areas.

- The south pit will have been backfilled, covered with an external surcharge waste dump (WD-4), revegetated, and equipped with a surface water management system. The portion of the south pit surcharge waste dump located within the floodplain of the River Cacheu will integrate the existing flood protection berm to ensure stability.

- The majority of the north pit will be backfilled and two surcharge waste dumps (WD-5 and WD-6) will be constructed on top of the eastern portion of the pit. The backfilled portions and waste dumps will be revegetated and equipped with a surface water management system. The current study plans for the complete backfilling of the north pit at closure, using stockpiled overburden from WD-5 and WD-6. The option of not backfilling and allowing a small pit lake or pond to form at the most westerly portion of the north pit is being evaluated in the ESIA, and may present an opportunity to reduce closure costs.

- Waste dumps WD-1 and WD-2 will have been progressively rehabilitated within the first few years of mining and surface water management systems will have been constructed to manage storm water run-off. The Environmental Control Dams will have been decommissioned and the local drainage patterns (Rio de Caur, Rio de Cavars Marinhos and Rio de Bunja) will be re-established.

- The south and north pit diversion channels will be enhanced to ensure long-term physical stability.

- Surface and groundwater quality monitoring will be conducted until the site has been proven to be chemically stable.

20.7 Expected Material Environmental and Social Impacts

As noted above, an ESIA for the mine site component of the Project was completed in 2014 based on a previous mine design. An ESIA that includes all aspects of the Project proposed in this feasibility study is under preparation. A preliminary assessment of the key social and environmental risks and impacts of the Project is presented below and is based on the available baseline information, the project plans described in this feasibility study, and the work completed on the ESIA to date. Proposed mitigation measures to address the identified risks and impacts are described at a high level.

Based on the work to date, there have been no environmental issues that are expected to prevent GB Minerals from developing the Project. The Project is expected to result in adverse environmental, socio-economic and cultural heritage impacts that can be reduced to acceptable levels, through the implementation of mitigation measures. The most significant effects identified to date include:

- **Air Quality and Noise Impacts –** Previous air quality and noise modelling completed for the mine site area suggested that exceedances of WHO ambient air quality standards (referenced in the IFC General EHS Guideline) would occur off-site (Golder, 2014b). Revised air quality and noise modelling completed recently by Knight Piésold based on the revised project design found that while some exceedances of applicable thresholds will occur off-site, the magnitude and extent of these impacts is reduced compared with previous modelling results (Knight Piésold, 2015b). It is expected that there will be opportunity to apply further mitigation measures to reduce concentrations of air quality contaminants and noise levels to acceptable levels.

- **Management of Waste Overburden and Tailings and Potential Effects to Groundwater and Surface Water –** Geochemical evaluations suggest that both the overburden and tailings contain elevated concentrations of metals. Additionally, a preliminary radiological assessment indicates that the phosphate ore and tailings contain some measure of radioactivity that will require management (Northern Environmental Consulting and Analysis Inc., 2015). Containment of these materials will be necessary to prevent seepage of adverse quality effluent to groundwater and to prevent discharge to surface waters. The tailings storage facility will require sufficient cover at closure to shield any radioactivity.

- **Pit Dewatering Affecting Community Wells –** Dewatering of the open pits will create a drawdown cone that will likely affect existing community wells. The company is establishing plans to provide water of at least equivalent quantity and quality to affected groundwater users in the area. A hydrocensus monitoring program will be undertaken to monitor and verify the effects of dewatering on surrounding wells.

- **Ecological Impacts** – The project will result in the loss of mangroves, salt marsh and freshwater areas, as well as secondary forest. Small areas of indigenous forest will also be lost. Lost mangrove habitat may in turn contribute to riverbank instability and erosion coupled with a loss of crocodile habitat. A decrease in forest habitats represents a loss of habitat for primates. The potential ecological impacts of the dewatering process and any effluent discharge or spills will also be evaluated. It is expected that suitable engineering design, repositioning of infrastructure and habitat offsets can adequately mitigate the various ecological effects of the Project.

- **Involuntary Resettlement** – The Project will require the acquisition of approximately 3000ha of land resulting in the physical and/or economic displacement of an estimated 175 households in villages in the mine site area. A resettlement policy framework (RPF) has been developed within the ESIA, and a resettlement action plan (RAP) will be developed and executed in the future, in consultation with local and national authorities.

- **Community Health, Safety and Security** – The Project will interrupt the current flow of mostly pedestrian and bicycle traffic between the regional service centre of Farim to villages to the west and north of the mine site. In addition, the presence of the mine site and Project traffic to and from the mine will present safety hazards. Traffic safety and other community health and safety risks will extend along the transport route to the port site. There is also the possibility that the presence of the mine will result in an influx of people into the region, which will require management in conjunction with the regional and national governments. A Community Health, Safety and Security Management Plan is under preparation as part of the ESIA that will identify these issues and propose preliminary mitigation measures that can be discussed with the appropriate authorities.

- **Radiological Exposure to Workers** – The presence of uranium in the phosphate ore can be a human health and environmental concern due to the potential for exposure to elevated radiation. Preliminary estimates for the ore and tailings suggest that external doses to workers will be considerably less than the Health Canada (2011) dose constraint for occupationally-exposed workers (Northern Environmental Consulting and Analysis Inc., 2015). Further analyses are underway to improve the certainty of the estimates. Nonetheless, a monitoring program is recommended to verify that external doses to workers are within acceptable limits. A preliminary monitoring program will be described in an Occupational Health and Safety Plan.

- **Cultural Heritage Impacts** – Most cultural heritage features identified within the Project areas are judged to be of low sensitivity. There appears to be no “critical cultural heritage” under IFC Performance Standard 8 (IFC, 2012). There is a potentially sensitive cemetery within the proposed mine site footprint that contains the grave of a well-known imam who lived over 100 years ago and is sometimes visited by people from within and outside the region during an annual festival. In addition, a sacred grove (or holy forest) is also located within the mine footprint that is of local to regional importance. It will be necessary to develop mitigation plans for these features in consultation with the adjacent communities and other stakeholders.

- **Employment and Training Opportunities to Guinea-Bissau Nationals** – The education levels of the residents near to the Project sites is low, and the country does not have any prior experience with mining. Therefore, it will be necessary to develop a Human Resource Development Plan that establishes appropriate recruitment, training and employment policies and procedures to develop a predominantly local workforce over time.

The Project is expected to deliver positive socio-economic benefits to the Farim area and Guinea-Bissau broadly, including:

- Capacity building and institutional strengthening opportunities;

- Significant investment in Guinea-Bissau and the Farim region, with the potential for reinvestment into physical infrastructure and social services;

- Increased scientific knowledge and data for the project area;

- Provision of goods and services to the Project that will generate new jobs and economic growth in support of industries and spin-off businesses;

- Revenue to national government from royalty and income taxes;

- Guinea-Bissau will be internationally recognized as having a world class phosphate mine and being a supporter of international investment in mining.

The communities in the areas of all of the project components are likely to have elevated expectations about the benefits they may receive from this development. It is important that these expectations are managed carefully, requiring a comprehensive stakeholder engagement process.

20.8 Social and Community Related Requirements and Plans

An Environmental and Social Management Plan (ESMP) is being prepared as part of the ESIA. The ESMP will summarize the Company`s commitments to address and mitigate risks and impacts identified as part of the ESIA. Mitigation strategies will include avoidance, minimisation, and compensation/offset. The ESMP will consist of the following:

- **Level 1 - Management System** – This will include the environmental and social management system (ESMS) and the overarching environmental, social, health and safety management system that is applicable at the Project level. The management system will be designed to identify, assess, and manage on an on-going basis risks and impacts associated with the Project. The system consists of policies, management programs and plans, procedures, requirements, performance indicators, responsibilities, training and periodic audits, and inspections with respect to environmental or social issues.

- **Level 2 - Discipline-Specific Management Plans** – Conceptual level management plans identifying potential impacts, mitigation measures and monitoring programs by discipline.

- **Level 3 - Standard Operating Procedures (SOPs)** - Detailed instructions or operational standards for executing the discipline-specific management plans that will be developed as the Project moves into the detailed engineering design and construction phases.

The ESMP will be an integral part of the ESIA, but will act as a stand-alone document that will be updated as needed throughout the Project life to accommodate changes in Project circumstances, legislation and guidance, unforeseen events, and the results of monitoring.

The following Level 2 discipline-specific management plans will accompany the ESMP presented in the ESIA:

- Air Quality Management Plan
- Noise Management Plan
- Erosion Control and Sedimentation Plan
- Water Management Plan
- Biodiversity Management Plan
- Waste Management Plan
- Occupational Health and Safety Plan
- Emergency Preparedness and Response Plan
- Mine Reclamation and Closure Plan
- Stakeholder Engagement Plan
- Community Health, Safety and Security Management Plan
- Community Development Plan

- Cultural Heritage Management Plan

- Resettlement Policy Framework.

Select **Candidate Level 3 SOPs** are either currently under development or will be identified in the ESMP for future development. Consistent with IFC requirements, the Chance Finds Procedures (for cultural heritage site) and Grievance Mechanism are two Level 3 SOPs that will be developed as part of the ESIA and ESMP.

There have been no negotiations or agreements discussed with local communities or authorities to date, beyond the Mining Agreement signed in 2009 with the Ministry of Energy and Natural Resources. It is expected that the ESIA will provide an opportunity for GB Minerals to discuss socio-economic and community aspects of the Project with the Guinea-Bissau Government. It is expected that input will be received on the work needed to advance community-focused management plans (including the Stakeholder Engagement Plan and Grievance Mechanism); the Community Health, Safety and Security Management Plan; and the Community Development Plan.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE INPUT

The capital cost for mining, process (beneficiation) plant facilities, port facilities, marine services, tailings waste management facilities and infrastructure required to treat the throughput capacity of 1.75 Mtpa, for “Farim Phosphate Project”, is USD $193.8 million excluding Owner’s cost in third quarter 2015 US dollars, and is subject to the assumptions and exclusions listed below in section 21.2. The Owner’s cost is estimated at a cost of US$11.9 million and includes items such as Owner’s construction team cost, US$4.0 million Resettlement allowance, $2.0 million for insurance, etc. The parties below in Table 21-1 have contributed to the preparation of the capital cost estimate in specific areas as listed:

Table 21-1 Consultants and Specialities

Scope of Work	Consultant
Mining	Golder Associates
Tailings & Waste Management Facilities (TWF)	Knight-Piésold (Perth, Australia)
Process and Metallurgical Testwork	KEMWorks
Marine	Baird & Associates
Process Plant & Port Facilities	Lycopodium Minerals Canada

21.2 Capital Cost Summary

The accuracy of the CAPEX for the Farim Project, with consideration of the current state of the engineering design, procurement and other related tasks, is deemed sufficient to support a CAPEX with a target accuracy range from a high of +15% to a low of -15% of final Project costs at the summary level and is expressed in third quarter 2015 base currency of US dollars.

The capital cost is summarized in Table 21-2 and is inclusive of the costs to design, procure, construct up to and including plant commissioning and start up; Sunk cost, sustaining capital cost, interest during construction, deferred capital costs, escalation and foreign exchange fluctuations and owner’s costs are excluded from these estimates.

Sustaining capital has been computed for use in the financial model and is included in Section 22 of this report.

Table 21-2 Capital Cost Summary

WBS Code	Area Description	Total
	Contractor's P&G (Preliminary & General) Costs Including Mob & Demob Costs	
	Process (Beneficiation) Plant	3,159,000
	Port Facilities	931,000
	Marine Services	4,573,603
	Mining	
	Tailings Dam	2,191,642
Subtotal Contractor's P & G Costs Including Mob & Demob Costs		**10,855,245**
	Mining	
	Mining Equipment	50,113,000
	Mine Operations (Pre-strip)	15,096,000
400	Mining General	92,116
470	Mine Fuel Services	796,563
Subtotal Mining		**66,097,679**
	Process (Beneficiation) Plant	
100	Treatment Plant General	2,316,923
20	Feed Preparation	706,682
130	Reclaim	1,036,745
140	Scrubbing/Screening/Tailings	6,237,273
150	Fine Concentrate Thickening	1,010,146
190	Concentrate Filtration and Storage	3,389,425
210	Reagents	366,406
230	Water Services	2,210,614
240	Plant Services	4,155,957
250	Air Services	496,044
260	Plant Fuel Services	161,345
300	Plant Infrastructure	6,845,571
340	Tailings Line	825,928
	Plant Mobile Equipment	8,506,702
Subtotal Process Plant		**38,265,761**
	Tailings and Water Management Facilities	
	Tailings Storage Facilities & Associated Works	6,330,591
	Comminution Piping	815,280
	Hydrology	893,980
Subtotal Tailings and Water Management Facilities		**8,039,851**
	Port Facilities	
700	Port General	3,501,653
720	Port Product Loadout	3,373,735

WBS Code	Area Description		Total
740	Port Concentrate/Drying/Storage		5,530,335
752	Port Fuel Services		903,460
753	Port Air Services		167,982
770	Port Services		1,169,636
	Plant Mobile Equipment		1,000,000
Subtotal Port Facilities			**15,646,802**
	Marine Services		
	Marine Structures		11,690,442
	Aids to Navigation		2,732,961
	Vessel Allowances		7,850,000
	Special Studies / Field Investigations		1,563,462
Subtotal Marine Services			**23,836,865**
Total Direct Cost			**162,742,202**
	Indirect Costs		
	Construction Field Indirects		1,924,812
	Construction Support		1,283,208
	Construction Camp & Catering		1,117,430
	Construction Power Supply & Operations		641,604
	Spare Parts		1,267,572
	First Fills and Inventory		111,300
	Freight & Transport		1,262,823
	Vendor Supervision & Training		565,360
	Taxes, Duties and Permit	excluded	0
	Testwork	allowance	750,000
	EPCM Process Plant, Port Facilities, Tailings & Mining		4,757,869
	EPCM Marine Services		867,861
	Marine Facility Construction Observation		759,880
	Marine Vessel Delivery Fees		0
	2% Vessel Acquisition Fees		157,000
	Third Party Engineering & Inspection Services		1,248,250
	Pre-commissioning & Commissioning		729,537
Subtotal Indirect Cost			**17,444,509**
Subtotal for Contingency			**180,186,711**
	Contingency	7.6%	13,635,703
Subtotal Direct & Indirect Costs			**193,823,000**
	Owner's Cost		-
Project Total			**193,823,000**

The estimate is based on an EPCM execution approach outlined in section 21.4.2.4 and 21.4.2.5

The following qualifications and exclusions were made:

Assumptions

- All equipment and materials are new;
- The labour rate buildup was based on statutory law governing benefits to workers in effect at the time of the estimate as supplied by local Contractors;
- Buildup of craft labour benefits and burdens is based on current local labour law;
- Site construction contracts will be approached via a combination of lump-sum and unit price contracts;
- Kristal Font Inc. (KFI) assumes all information provided by GB Minerals and others for this FS compilation is accurately and comprehensively presented.

Exclusions

- Geology and resource estimation work (by others);
- Mining design and engineering work (by others);
- Mining capital cost estimate (by others);
- Topographic surveys for the plant site;
- Environment (by others), community relations, and heritage work;
- Metallurgical testwork and supervision at the laboratory (by others);
- Tailings disposal, hydrology, and site geotechnical work (by others);
- Marine design and cost estimates (by others);
- Government liaison, permits, licenses, approvals, including scheduling inputs.

Also excludes all Owner's costs such as:

- Pre-operations expense;
- Changes to design criteria;
- Hiring and relocation;
- Uplift cost;

- Legal;
- Public relations;
- Resettlement Costs;
- Geotechnical investigations;
- Sunk costs prior to and including this Study;
- Allowance for future expansion;
- Fees;
- Offsite facilities, except for power line;
- Allowance for risk;
- Cost of financing and interest during construction;
- Any provision for force major;
- Sustaining and operating costs;
- Working capital;
- Scope changes or accelerated schedule;
- Schedule Contingency;
- Escalation; and
- Foreign exchange fluctuation.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs) were identified and analyzed. To each of these categories, a percentage of contingency was allocated based on the accuracy of the data and an overall contingency amount derived for the process plant and the port facilities. Other consultants provided their own contingencies.

21.3 Capital Cost Estimate Scope

21.3.1 Mining

Golder estimated the costs of matrix production and capital requirements associated with producing FPA matrix from the two Farim mining pits. Production cost and project capital estimates were developed on an annual basis to reflect the yearly matrix release, waste removal (or “stripping”) requirements, and matrix/waste haulage parameters dictated by the respective mine plan.

The mining cost estimate assumes all mining functions are directly performed by GB Minerals using company-owned equipment and company employees. The mining cost includes only those costs directly related to mining and delivering matrix to the processing facility; all cost estimates related to processing and other activities after the matrix is placed into the hopper were assumed to be provided by other parties.

Golder included ongoing reclamation costs during the mine life including dozer work for backfill pit re-grading and re-vegetation during mining and backfill of the final pit void. However, final mine closure and infrastructure demolition were not included in the mining cost estimate as they were covered by others.

Additionally, Golder did not include overhead expenses or any other indirect mining costs (e.g., property and liability insurance, permitting fees, bonding, governments and environmental relations fees, royalties, and other miscellaneous expenses) in the mining cost estimate as these costs are covered by others. One hundred percent equity was assumed in the determination of Project capital requirements.

All mining costs and dollar amounts referenced in this section are exclusive of any taxes. The total mining cost has been included the total capital cost in Table 21-2. Details of the mining cost and associated costs are contained in section 16 of this report.

21.3.2 Process Plant & Port Facilities

The Farim beneficiation plant and associated facilities estimates have been prepared on a commodity basis (i.e. divided into earthworks, concrete, structural steel, architectural, etc.) and reported by area (i.e. Feed Preparation, Reclaim, Concentrate Stockpiling, etc.).The estimate is based on the purchase of new mechanical equipment and quantities have been assessed from first principles.

The estimate is based on the majority of the work being carried out under fixed price or unit price contracts under a normal development schedule. No allowance was included for contracts on a cost plus or fast-track accelerated schedule. The erection of tankage, structural steel, mechanical, piping, electrical, instrumentation and civil works will be performed by experienced contractors using local labour and/or third country nationals supervised by expatriates.

21.3.3 Tailings Storage Facility

Knight-Piésold established the scope and quantities for the Integrated Waste Landform (IWL), surface water management, and dewatering infrastructure. Knight-Piésold estimated the earthwork costs based on West African contractor rates. Kristal Font Inc. reviewed the estimate for completeness and comparison and align the unit cost with that used in this estimate.

21.3.4 Marine Services

Port marine costs were based on the scope of work established by Baird and Associates and the capital estimate has been prepared by same.

21.4 Basis of Estimate

The direct costs are all the costs associated with permanent facilities. This includes equipment and material costs, as well as construction and installation costs.

21.4.1 Direct Cost

Quantities

Engineering material take-offs (MTO's) were provided by Lycopodium based on "neat" quantities derived from Project drawings and sketches. The MTO quantities were based on the equipment lists, control system concepts, electrical load lists, sketches and other documents of record. Normal and accepted allowances were included in the estimate, as appropriate by the estimator. Conceptual quantities were prepared where drawing information was not available. Metric units are assumed throughout the estimate, with the exception of piping. Pipe sizing is described in inches of nominal diameter.

Unit Quantity Preparation

Quantity preparation (MTO's) were the responsibility of the engineering disciplines in Lycopodium for their scope of work. The MTOs, by discipline, were reviewed on an ongoing basis as the material quantities and takeoff assumptions were defined.

Takeoffs were produced by the following methodologies:

Autodesk Land Desktop Software

From documents, drawings and manual sketches.

Design Growth, Waste Factors and Material Take-off Allowance.

For the quantity based portions of the estimate, material prices or quantities were adjusted to cover normal construction waste (e.g., concrete over-pour, electrical raceway cut-offs), design growth and MTO allowance based on engineering and estimating experience, consistent with the development stage of the design and corresponding quantity definition on comparable projects.

These MTO allowances are typically included for waste, drop, breakage loss and the like which are not included in the engineering neat takeoffs. Steel connections, bracing and gusset plates are included in the steel allowance.

The following combined factors (as shown in Table 21-3) were used in the estimate.

Table 21-3 Design Growth, Waste Factor and MTO Allowance

Discipline	Factor	Remarks
Earthworks – Bulk Excavation	12%	To cover for comprehensive geotech, soil characterization, topography deficiencies
Earthworks – Bulk Backfill	10%	Swelling and compaction factors are included in the individual unit rates, by material type
Earthworks – Structural Excavation	10%	To cover for comprehensive geotech, soil characterization, topography deficiencies
Earthworks – Structural Backfill	10%	Swelling and compaction factors are included in the individual unit rates, by material type
Concrete	5%	Concrete supply costs includes for over-pour and wastage
Structural steel	10%	To cover connections, base plates, gussets and painting included in the material cost
Miscellaneous Steel	15%	To cover connections, cutoffs and wastage, bending and painting included in the material cost
Architectural	0%	Design build
Electrical Cabling	15%	Cable and wire overbuy, routing, sagging, loss of fittings – included in the material cost
Electrical Equipment	5%	Level of definition of load requirement
Instrumentation Cabling	15%	Cable and wire overbuy, routing, sagging, loss of fittings – included in the material cost
Instrumentation Equipment	5%	Level of P&ID definition
Piping	15%	Allowance for bends, cutoffs, etc., included in the material cost

The above wastage percentages were carried in the detailed estimate as a multiplier to the material cost at the work element level.

Design / Growth Allowance for Process Equipment

Some additional design development beyond the estimate basis documents is typically anticipated. Therefore, an equipment specification allowance (design development allowances) are included in the estimate for process equipment. Design development allowances are applied to the estimated cost for equipment.

This allowance has been included as a percentage on the value of the budgetary price for the above stated items. A 5% allowance was adopted for equipment with dollar value of less than USD $0.5 million, whilst 2% has been included for equipment with dollar value above USD $0.5 million.

Craft Wage Rates for Direct Installation

The construction craft wage rates were based on burdened hourly construction compensation rates that have been analyzed from rates recently provided (1Q2015) by at least three (3) local Contractors capable of executing the scope of work and engaged in similar or comparable work within the same locality or are very conversant with the area. The burdened labour rate by discipline is summarized in Table 21-4.

The local contractors confirmed that the following contactor's indirect costs were included in their all-in labour rate compensation:

- Accommodation and catering for construction personnel;
- Transportation for construction personnel;
- Overtime premiums;
- Vacation / leave allowance;
- Sick pay;
- Bonuses;
- Severance pay;
- Health insurance;
- Social security;
- Labour training;
- Small tools;
- Temporary facilities;
- Contractors site offices and associate running costs;
- Diesel generating plants;
- Contractor's overhead;
- Contractor's profit;
- All other local requirement.

Table 21-4 Unit Labour Rate by Discipline

Discipline	Average Unit Labour Rate by Discipline in USD/hr
Earthworks/Civil	10.00
Concrete	6.00
Structural Steel/Platework	17.00
Architectural	10.00
Mechanical Equipment Installation	17.00
Electrical	12.00
Instrumentation	12.00
Piping	17.00

The composite crew contractor labour rates were prepared based on typical Northern American crew configurations.

It has been assumed that the contractors will work 10-hour shifts for 6 days per week.

Labour Man-hour Units

The all-in contractor composite labour rate was applied against total man-hours per discipline item (labour productivity rate) to estimate the total labour cost.

Labour productivity rates based of North American work base hours for the mining industry, as compiled by various mining construction management companies on executed projects that has evolved to be the standard used in the mining industry was adopted as against the U.S. Gulf Coast productivity rate for mid-sized Greenfield projects which is more suited for the Oil and Gas industry.

These productivity units were then modified by multiplier (Labour Productivity Factors – PF) to adjust for local African Sub region with expatriate supervision for the Project. The Productivity Factors account for anticipated working conditions at the site, including but not limited to weather conditions, safety conditions, local labour skills and availability, work coordination, and local work practices.

The Labour Productivity Factors were computed using a proprietary empirical program, and then bench-marked against labour productivity factor produced by Richardson International Construction Factors Manual™ for the African sub region. Table 21-5 summarizes the Labour Productivity Factor by discipline.

Table 21-5 Craft Labour Productivity Calculation Summary

Discipline	Factor
Earthworks / Civil	1.48
Concrete	2.22
Structural Steel	2.18
Architectural	1.92
Mechanical	2.11
Piping	2.16
Electrical	2.13
Instrumentation	2.13

Construction Equipment Usage

The cost for construction equipment, estimated as dollars per direct work hour by prime account, provides for equipment ownership, depreciation, insurance, fuel oil, lubricants, maintenance, and service and repair.

Each prime account, except for D1 (earthworks), were priced per craft work hour. Equipment operator labour costs form a part of these hourly rates, as they are not included in composite crew labour mixes. Construction equipment costs for prime account D1 are calculated on dollars per type of work and unit, not on dollars per craft work hour.

The construction equipment usage unit rate is summarized in Table 21-6.

Table 21-6 Unit Construction Equipment Usage Rate Comparison By Discipline

Discipline	Unit Construction Equipment Usage Rate by Discipline in USD/mhr
Earthworks/Civil	Various
Concrete	3.00
Structural Steel	7.83
Architectural	5.00
Mechanical Equipment Installation	8.00
Electrical	8.00
Instrumentation	8.00
Piping	7.00

21.4.2 Indirect Cost

The indirect costs cover all the costs associated with temporary construction facilities and services, construction support, freight, Vendor representatives, spare parts, initial fills and inventory, Owner's costs, EPCM, commissioning and start-up assistance.

This account includes all temporary buildings and services required during construction and commissioning phases. These costs were estimated as per the construction execution plan and include the following:

- Offices;
- Temporary warehouse;
- Temporary construction services;
- Construction water supply;
- Sewage facilities;
- Construction communications;
- Lay-down areas;
- Roads and maintenance;
- Dust suppression; and
- Modular construction yard.

21.4.2.1 Temporary Construction Services

Temporary construction services include office janitorial and garbage services for the EPCM and Owner's Project teams; bottled water; QA surveying; site access control; material unloading; security services; personnel physicals, safety induction and badges, safety, first aid, medical supplies and services. The costs for these services are based on the EPCM and Owner's team organisation charts, Project construction schedule, and past experience with projects of similar size and duration.

An allowance was made for soils, concrete, and piping NDE (non-destructive examination) testing including HDPE NDE testing.

Freight and duty rates for construction management have been applied to the costs of the indirect materials.

21.4.2.2 Construction Equipment and Tools

Warehouse equipment and heavy-lift cranes have been estimated under temporary construction services.

Allowances for personnel protection equipment (PPE) for the EPCM and Owner's Project teams include hardhats, safety glasses and safety shoes.

21.4.2.3 Construction Field Office Expenses

Office supplies, consumables, reproduction printing, postage and courier service were estimated based on experience from other projects of this size and duration.

21.4.2.4 Engineering / Procurement

The engineering and procurement (EP) costs are based on the current execution plan. The discipline engineering costs are based on an estimate of man-hours required to complete an identified list of engineering deliverables. The engineering management, administration, Project services and procurement staff costs are based on staffing requirements and duration. The duration of each position was estimated and man-hours calculated accordingly. The engineering estimate is based on a 40-hour workweek. All-in charge-out labour rates were then applied to the estimated man-hours.

21.4.2.5 Construction Management

The construction management (CM) estimate is based on the Project execution plan. The construction execution basis of the CM estimate is that multiple contractors will contract the work on unit price, or lump-sum contracts.

The CM estimate covers the field- or site-based services required for constructing and commissioning the process facilities and associated infrastructure.

The CM estimate includes the following site-based services:

- Project management;
- Field engineering;
- Site document control;
- Construction management;
- Industrial relations;
- Construction supervision to general superintendent;
- Health, safety, environmental and community;

- Site administration;
- Field human resources;
- Site quality assurance and control;
- Site Project controls (cost control and schedule);
- Field accounting;
- Site computers and information technology services;
- Site procurement;
- Field receiving and warehousing; and
- Field contract administration.

The CM costs were calculated on a staffing requirement basis. The duration for each identified position was estimated and the hours calculated accordingly. The CM estimate is based on a 60 hour workweek.

Labour costs were applied to the hours estimated for each category. Average monthly assignment costs were calculated for the CM staff.

Support expenses for CM staff were included in the construction indirect field costs. These expenses include offices, vehicles, communications and transportation.

21.4.2.6 Commissioning

The commissioning account includes trade crews to support commissioning for a period of 6 months. The cost for commissioning assistance by the EPCM Contractor, based on providing seven technical staff, is included in the EPCM costs.

21.4.2.7 Freight / Duties

Freight costs information that was provided with the Vendor quotations have been included. The remaining freight were factored based on 2% Inland freight, 6% Ocean freight. An allowance was also made for air freight based on experience on other projects.

Bulk materials were deemed to be supplied FOB jobsite. Additionally, some equipment were also quoted FOB jobsite.

21.4.2.8 Vendor Representatives

Vendor representatives' costs were developed based on information as provided by the Vendors. Travel time of 1 day portal-to-portal was included with the Vendor time required onsite. Airfares, lodging, and other out-of-pocket expenses will be accounted for in the rate per round trip. All-in daily rates quoted by Vendors in their budgetary major equipment pricing were adjusted to reflect the planned 60-hour workweek.

Vendor representatives' costs were included where Vendors stated they required to be onsite for installation to maintain the equipment warranty. Vendor representative costs were also included for commissioning assistance.

21.4.2.9 Capital Spare Parts

Major spares were developed based on information as provided by the Vendors. These are critical spares to be maintained for the effective operation of the plant.

Where no information was provided by the Vendor, an allowance was included, based on experience for same equipment from previous project.

21.4.2.10 First Fills

The budgetary cost to supply plant first fills has been included and includes such items as lubricants, fuels, and flocculent. First fills does not include general warehouse inventory and staff.

21.4.2.11 Construction Fuel

Construction fuel will be purchased by the Owner and issued free of charge to the Contractors. This cost element will have been deducted from the Contractor's overhead.

Total fuel consumption for the power generating plants onsite and all Contractor's mobile equipment and machinery is computed and multiplied by the fuel price for the Project.

21.4.2.12 Owner's Cost

The following Owner's cost items although excluded from the initial capital cost were estimated and provided by the Owner and typical includes:

- Insurance.
- Owner's Site Staff Salaries.
- Permitting and Environmental Costs.
- Land acquisition.

- Marketing expenses.
- Consultants.
- Resettlement costs.

21.5 Operating Cost Estimate Introduction

The direct cash operating cost for the Farim Phosphate Project have been estimated under three functional headings: mining, process plant and general and administration (G&A). The operating costs have been estimated by the following parties:

- Mining – Golder and GB Minerals.
- Beneficiation Plant and Port Facilities – Lycopodium, Baird and GB Minerals.
- G&A – Lycopodium and GB Minerals.

The operating cost estimates are expressed in US dollars (USD) in first quarter 2015 terms and are expected to be accurate within ±15%.

A summary of the life-of-mine (LOM) operating costs are summarized in Table 21-7.

Table 21-7 Operating Cost Summary

COST CENTRE	Total Cost		
	USD/year	USD/t conc.	USD/t ore
Process & Admin. Labour	$ 6,626,034	$ 5.01	$ 3.78
Operating Consumables	$ 11,269,791	$ 8.53	$ 6.44
Power	$ 6,995,841	$ 5.30	$ 4.00
Maintenance	$ 1,360,007	$ 1.03	$ 0.78
Shiploading	$ 3,127,351	$ 2.37	$ 1.79
G&A Expenses	$ 3,535,000	$ 2.68	$ 2.02
Corporate Costs	$ 2,912,500	$ 2.20	$ 1.66
Mining Total	$ 33,044,463	$ 25.01	$ 18.88
TOTAL	**$ 68,870,097**	**$ 52.13**	**$ 39.35**

21.6 Mining Operating Costs

General

Golder estimated the costs of matrix production and capital requirements associated with producing FPA matrix from the two Farim mining pits. Production cost and project capital estimates were developed on an annual basis to reflect the yearly matrix release, waste removal (or “stripping”) requirements, and matrix/waste haulage parameters dictated by the respective mine plan.

The mining cost estimate assumes all mining functions are directly performed by GB Minerals using company-owned equipment and company employees. The mining cost includes only those costs directly related to mining and delivering matrix to the processing facility; all cost estimates related to processing and other activities after the matrix is placed into the hopper were assumed to be provided by other parties.

Golder included ongoing reclamation costs during the mine life including dozer work for backfill pit re-grading and re-vegetation during mining and backfill of the final pit void. However, final mine closure and infrastructure demolition were not included in the mining cost estimate as they were covered by others.

Additionally, Golder did not include overhead expenses or any other indirect mining costs (e.g., property and liability insurance, permitting fees, bonding, governments and environmental relations fees, royalties, and other miscellaneous expenses) in the mining cost estimate as these costs are covered by others. One hundred percent equity was assumed in the determination of Project capital requirements.

All mining costs and dollar amounts referenced in this section are exclusive of any taxes.

Direct Operating Costs

Direct operating costs encompass the labour and material and supply costs associated with matrix production and mine-to-process plant matrix trucking. Direct operating costs were estimated and reported by the following primary functional cost centers:

- Waste Stripping & Topsoil Removal;
- Matrix Loading & Haulage;
- FPA stockpiling;
- Mine Maintenance;
- Operations support;
- Pit dewatering;
- Interim Mine Reclamation; and
- Mine Supervision & Administration.

The waste stripping and topsoil removal cost center encompasses waste rock excavation by the loader fleet, truck haulage of excavated waste material to designated dump areas, removal, stockpiling, and direct replacement of topsoil.

Matrix loading and haulage (FPA Mining) activities include mining of the matrix by the hydraulic backhoe fleet and haulage of ROM matrix to the process plant or the ROM stockpile. FPA stockpiling includes the cost of re-handling matrix from the ROM stockpile.

Mine maintenance functions include in-pit equipment fueling and lubrication; repairing equipment in the field; servicing haul truck tires; and, shop maintenance activities including component replacements, major equipment rebuilds, and light vehicle maintenance.

The Operations Support function includes pit haul road maintenance, construction of in-pit ramps and bench access roads, and other general mine support activities. Pit dewatering, which includes pumping water from the pit, was included as a separate item. Dewatering/depressurization ahead of mining was considered separately with cost estimations prepared by others.

Interim mine reclamation involves the performance of various reclamation activities such as the rehandling of stockpiled topsoil and waste rock, waste dump grading, and revegetation while the mine is active.

The supervision and administration function encompasses the cost of salaried supervisory and administrative personnel stationed at the mine, mine office operating supplies, and pickup truck fleet operations and maintenance.

A summary of the direct mine operating costs is provided on the following page in Table 21-8. These costs are summed and presented on a gross basis as well as on a unit basis in $/product tonne. For a detailed breakdown of the primary functional cost centers that aggregate the direct operating costs please refer to Section 16.9.1.

Table 21-8 Summary of Direct Mine Operating Costs

DESCRIPTION	Year 1	Year 2	Year 3	Year 4	Year 5	Years 6 - 10	Years 11 - 15	Years 16 - 20	Years 21 - 26	TOTAL
PRODUCTION STATISTICS										
Total ROM Production (000s tonne - Dry Basis)	1,750	1,750	1,750	1,750	1,750	8,750	8,750	8,750	9,007	44,007
Total Product Tonnage (000s tonne - Dry Basis)	1,321	1,321	1,321	1,321	1,321	6,606	6,606	6,606	6,800	33,225
Total Stripping Volume (000s bcm)	11,172	14,922	14,318	13,079	11,898	77,517	95,891	88,645	91,240	418,680
Rehandle Volume (000s bcm)	-	-	-	-	-	-	-	-	-	-
Total Effective Stripping Volume (000s bcm)	11,172	14,922	14,318	13,079	11,898	77,517	95,891	88,645	91,240	418,680
Stripping Ratio (bcm/ROM Tonne)	6.38	8.53	8.18	7.47	6.80	8.86	10.96	10.13	10.13	9.51
Productivity (ROM Tonne/Total Employees)	5,105	4,280	4,410	4,641	5,063	4,291	3,241	3,629	3,668	3,905
DIRECT OPERATING COSTS										
Waste Stripping ($000s)	$16,351	$21,136	$20,065	$18,925	$15,355	$103,696	$155,745	$126,967	$121,143	$599,383
Cost Per Product tonne ($/Tonne)	*$12.38*	*$16.00*	*$15.19*	*$14.32*	*$11.62*	*$15.70*	*$23.58*	*$19.22*	*$17.81*	*$18.04*
FPA Mining ($000s)	$1,766	$1,694	$1,829	$1,864	$1,932	$10,269	$10,436	$11,212	$13,046	$54,048
Cost Per Product tonne ($/Tonne)	*$1.34*	*$1.28*	*$1.38*	*$1.41*	*$1.46*	*$1.55*	*$1.58*	*$1.70*	*$1.92*	*$1.63*
Pit Dewatering ($000s)	$110	$163	$181	$180	$233	$659	$1,290	$1,821	$1,925	$6,562
Cost Per Product tonne ($/Tonne)	*$0.08*	*$0.12*	*$0.14*	*$0.14*	*$0.18*	*$0.10*	*$0.20*	*$0.28*	*$0.28*	*$0.20*
Reclamation ($000s)	$469	$831	$815	$549	$499	$3,411	$4,791	$4,551	$4,871	$20,788
Cost Per Product tonne ($/Tonne)	*$0.35*	*$0.63*	*$0.62*	*$0.42*	*$0.38*	*$0.52*	*$0.73*	*$0.69*	*$0.72*	*$0.63*
Maintenance ($000s)	$1,534	$1,899	$1,827	$1,696	$1,578	$9,823	$12,258	$11,093	$11,482	$53,189
Cost Per Product tonne ($/Tonne)	*$1.16*	*$1.44*	*$1.38*	*$1.28*	*$1.19*	*$1.49*	*$1.86*	*$1.68*	*$1.69*	*$1.60*
Operations Support ($000s)	$1,125	$1,125	$1,125	$1,125	$1,125	$5,623	$5,626	$5,626	$6,747	$29,248
Cost Per Product tonne ($/Tonne)	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.85*	*$0.99*	*$0.88*
FPA Processing ($000s)	$959	$959	$959	$959	$959	$4,794	$4,794	$4,794	$4,935	$24,111
Cost Per Product tonne ($/Tonne)	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*	*$0.73*
Mine Supervision & Administration ($000s)	$1,679	$1,679	$1,679	$1,679	$1,679	$8,396	$8,396	$8,396	$10,075	$43,657
Cost Per Product tonne ($/Tonne)	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.27*	*$1.48*	*$1.31*
DIRECT OPERATING COSTS ($000s)	**$23,993**	**$29,487**	**$28,481**	**$26,977**	**$23,360**	**$146,670**	**$203,335**	**$174,460**	**$174,224**	**$830,986**
Cost Per Product tonne ($/Tonne)	***$18.16***	***$22.32***	***$21.56***	***$20.42***	***$17.68***	***$22.20***	***$30.78***	***$26.41***	***$25.62***	***$25.01***

Notes:

The reported product tonnages are based off an average plant mass yield of 75.5%

Labour Costs

Project labour requirements were estimated using a zero-based approach with annual staffing levels determined by the level of equipment or facility usage dictated by the mining plan. Manpower requirements necessary for the operation of primary production equipment (such as hydraulic excavators, wheel loaders, waste and matrix haul trucks, bulldozers, and graders) were based on the respective equipment operating shifts derived using established equipment-scheduling parameters. Maintenance and support labour and mine supervisory and administrative personnel were assigned as deemed necessary to adequately support production.

For the Study, waste mining was performed on a seven-days-per-week, three 8 hour shifts per day basis. Matrix mining was performed on a seven-days-per-week, one 12 hour shift per day basis. Continuous coverage for waste was scheduled to be accomplished with four rotating crews working 8-hour shifts. The mine was assumed to operate 355 days per year. However, the production was de-rated for the rainy season to account for weather related downtime and equipment delays.

Total labour costs were developed on an annual basis for both hourly and salaried personnel with 8 hour shift personnel providing 253 shifts of usable work annually and 12 hour shift workers providing 163 shifts of useable work annually. Details of the manpower levels and cost are available in Section 16.9.1.2.

Material and Supply Costs

Material and supply costs constitute expenditures for equipment operating supplies such as fuel, lubricants, rubber tires, filters, repair/replacement parts and other non-equipment specific items (e.g., road gravel, culverts, hardware, welding gases and rods, and small tools).

Annual equipment operating supply requirements were estimated on a cost per machine engine hour basis. Note that an engine hour is herein defined as a scheduled hour adjusted for non-consuming mechanical and operating delays to reflect the portion of total scheduled time that a piece of equipment is consuming operating supplies.

Equipment hourly operating costs are a function of the estimated hourly consumption or usage of fuel, lubricants, rubber tires, filters, and repair/replacement parts. Estimated consumption rates of fuel and lubricants for individual pieces of equipment were based on manufacturer/dealer specifications and guidelines; engineering estimates; and actual operating data on file at Golder. Unit costs for diesel fuel ($/litre) and lubricants ($/litre or $/kilogram) were based on vendor budgetary pricing data and information provided by GB Minerals. The major unit consumables costs assumed for material and supply cost estimates are summarized in Section 16.9.1.3.

Annual material and supply costs for mining, and support equipment were estimated by multiplying the operating hours derived for a particular piece of equipment in a given year by the respective machine hourly operating cost. Estimated equipment operating hours reflect the level of equipment usage dictated by the respective matrix production plan. Operating hours for major production equipment (e.g., hydraulic backhoes, wheel loaders, haul trucks, drills, dozers, and graders) are a function of the scheduled material volumes/tonnages to be moved and estimated equipment production rates. Support equipment was assigned as deemed necessary to facilitate an effective mining operation.

21.7 Process and G&A Operating Costs

21.7.1 Introduction

The process operating costs for the Farim Phosphate Project have been developed according to typical industry standards applicable to phosphate processing plants.

Quantities and cost data were compiled from a variety of sources including:

- Metallurgical test work;
- Supplier quotations;
- Advice from GB Minerals;
- Lycopodium data;
- KEMWorks data;
- Baird; and
- First principles.

The total process and G&A operating cost is USD $35.8 million per annum, USD $27.12/t concentrate or USD $20.47/t ore. A breakdown of the cost center is summarized in Table 21-9.

Table 21-9 Summary of Operating Costs

COST CENTRE	Total Cost		
	USD/year	USD/t conc.	USD/t ore
Total Labour	$ 6,626,034	$ 5.01	$ 3.78
Operating Consumables	$ 11,269,791	$ 8.53	$ 6.44
Power	$ 6,995,841	$ 5.30	$ 4.00
Maintenance	$ 1,360,007	$ 1.03	$ 0.78
Shiploading	$ 3,127,351	$ 2.37	$ 1.79
G&A Expenses	$ 3,535,000	$ 2.68	$ 2.02
Corporate Costs	$ 2,912,500	$ 2.20	$ 1.66
TOTAL	**$ 35,826,524**	**$ 27.12**	**$ 20.47**

21.7.2 Qualifications and Exclusions

The process operating cost estimate includes all direct costs associated with the Project to allow production of phosphate concentrate. Each cost estimate is presented with the following exclusions:

- Process operating costs battery limits are the ROM bin ahead of the scrubbing circuit to the tailings dam. All costs associated with areas beyond the battery limits of the study are excluded;
- All mining and exploration costs, except for laboratory assays;
- All taxes and import duties;
- Any impact of foreign exchange rate fluctuations;
- Any business interruption costs;
- Any escalation beyond the date of the estimate;
- Political risk insurance;
- First fill and opening stocks costs (included in the capital cost estimate);
- Tailings storage, rehabilitation or closure costs (included in sustaining capital);
- Product costs (transportation, refining, marketing, insurance);
- Licence fees or royalties (included in cash flow model);
- Environmental impact monitoring costs (environmental monitoring costs are included in financial model); and
- No contingency allowance.

21.7.3 Exchange Rates, Estimate Date and Escalation

Costs are presented in US dollars (USD) and are estimated on a pricing basis as of the first quarter of 2015. Unit rates for cost items that have been received from North American sources hence no conversion is required.

Escalation of operating costs from the time of the estimate is not considered for the Project.

21.7.4 Operating Cost Accuracy

The expected order of accuracy for the operating cost analysis is in the range of ±15%.

21.7.5 Plant Design Parameters

Operating costs have been developed according to the process design criteria. Table 21-10 summarizes the plant design criteria.

Table 21-10 Process Design Criteria

	Item	Unit	
Production	Total Annual ROM	t	1,750,000
	Total Annual Concentrate	t	1,321,250
Grade	ROM	$\%P_2O_5$	32.9
	Concentrate	$\%P_2O_5$	34.0
Recovery	Mass	%	75.5
	P_2O_5	%	78.5

21.7.6 Cost Categories

The operating cost estimate includes seven major categories as defined below:

1. Process Labour;
2. Consumables;
3. Power;
4. Maintenance;
5. Mobile Equipment;
6. Ship-loading; and
7. G&A.

A description of each cost category is provided in the following sections.

Process Plant Labour

The process labour is divided into the following areas: process department, process plant operations, port operations, metallurgy and maintenance. The process labour includes a combination of day and shift work. The estimated annual process plant labour cost is USD $4.18 million per annum.

Wages and Salaries

Labour cost for each position includes a salary and overhead costs. Overhead costs include provisions for health plan and medical examinations, life insurance, holidays, overtime, termination fees, etc. Wages and salaries have been provided by GB Minerals.

Consumables

The consumables category covers all wear parts and consumable materials in the process plant. Consumables include liners for equipment such as scrubbers, chemical reagents, as well as diesel fuel. The estimated annual consumables cost is USD $11.27 million per annum.

Consumption rates and pricing for consumables and reagents have been based on the following:

- Laboratory test work results are used, wherever possible to determine the reagent consumption rates. In the absence of test work data, reagent consumption rates are assumed based on first principle calculations, Lycopodium experience and generally accepted practice within the industry.

- Diesel fuel consumption rates for the mobile equipment, concentrate dryer, shiploading facility and power plants are based on first principles calculations and Lycopodium experience. A diesel price of USD $0.55/l was obtained from a vendor budgetary quotation in January 2015 however USD $0.80/l is used in the estimate as per GB Minerals' request.

- Consumables and reagents prices are obtained through supplier quotes.

- Antiscalant consumption rates and water treatment plant consumables are based on Lycopodium experience.

- Laboratory costs are allocated on a per sample basis. These costs (exclusive of labour costs) are included in the G&A cost category.

Power

The plant electricity consumption is determined based on the installed power, excluding standby equipment. The installed power for the processing plant at Farim and the port loadout facility are 4.8 MW and 1.5 MW respectively, which gives a total connected power of 6.3 MW. The estimated annual power cost is USD $7.00 million per annum.

Electrical load factors and utilization factors are applied to the installed power to arrive at the annual average power draw, which is then multiplied by total hours operated per annum and the electricity price to obtain the plant power cost.

Electricity is generated onsite by diesel power plants. A diesel price of USD $0.80/l is used in the calculations as per GB Minerals' request.

Maintenance

Maintenance material costs are estimated by applying factors to the ex-works mechanical equipment cost in each area of the plant. This is done to cover the cost of all maintenance materials and contract labour requirements. The factors applied are based on Lycopodium's database and experience, and are average costs over the life of the mine. As such, actual spares costs may be lower during the initial years but rise later. An overall factor of 4% is applied to the mechanical equipment supply cost ex-works. The estimated annual maintenance cost for process plant and mobile equipment is USD $1.36 million.

Mobile Equipment

The operating costs for mobile equipment are estimated and include diesel fuel, tires and maintenance parts. The fuel costs are included in the consumables cost centre while the other operating costs are included in the overall maintenance materials cost centre.

Ship-loading

The operating cost for the ship-loading facility is estimated by Baird and Associates. The cost includes diesel fuel usage, labour costs, equipment maintenance (pilot boat and tug boats) and contingency. The estimated annual ship-loading cost is USD $3.13 million.

G&A Costs

This category covers the G&A costs required for running the operation, which have been provided by GB Minerals.

The total estimated annual G&A cost is USD $8.9 million, $6.73/t concentrate or $5.08/t ore. Table 21-11 summarizes the three components of this cost category namely G&A expenses, G&A labour and corporate costs, which are based on the following:

- G&A expenses are provided by GB Minerals;
- Salaries and overheads are applied to the following administration areas: administration; security; as well as safety, health and environment. The G&A labour cost category includes mostly day work for the administration staff; with the exception of security staff whom perform shift work;

- Laboratory consumables are included in the G&A costs. Laboratory staffing is included in the process plant labour cost category;
- Corporate costs are provided by GB Minerals.

Table 21-11 Summary of G&A Costs

	G&A Operating Costs		
	USD/year	**USD/t conc.**	**USD/t ore**
G&A Expenses	$ 3,535,000	$ 2.675	$ 2.020
G&A Labour	$ 2,445,752	$ 1.851	$ 1.398
Corporate Costs	$ 2,912,500	$ 2.204	$ 1.664
Total G&A Cost	**$ 8,893,252**	**$ 6.730**	**$ 5.082**

22.0 ECONOMIC ANALYSIS

22.1 General Parameters

This financial model is prepared to reflect the revenue stream and corresponding operating cost for GB Mineral's Farim Phosphate green-field project which contains measured and indicated resources of 105.6 million tonnes at 28.4% P_2O_5, and additional inferred resources of 37.6 million tonnes at 27.7%% P_2O_5. The reserves are estimated at 44.0 million tonnes at 30.0% P_2O_5 based on a 25 year mining plan. 1.75 Mtpa of ore are mined with 1.32 Mtpa of beneficiated phosphate rock product produced. The final beneficiated phosphate rock concentrate will have a grade of 34%.

The financial analysis model covers the time span from years -3 through +27 with pre-production years of year -3, -2 and -1. Detailed engineering, construction and pre-stripping is assumed to occur during the pre-production period, it is envisaged that all the necessary permits to commence construction and execute this project will be in place at this time. Production years are from +1 to +25. Project closure is deemed to take place in years +26 and +27.

The capital cost for mining, process plant facilities, port facilities, marine services, tailings waste management facilities and infrastructure required to treat the throughput capacity of 1.75 Mtpa, for "Farim Phosphate Project", is USD 193.8million excluding Owner's cost in third quarter 2015 US dollars, and is subject to the assumptions and exclusions listed below in section 21.2. The Owner's cost is estimated at a cost of USD $11.9 million and includes items such as Owner's construction team cost, USD $4.0 million Resettlement allowance, $2.0 million for insurance, etc. Owner's costs have been included in the financial model.

Processing

The project consists of an open pit mine, drum scrubber, attrition scrubber, classification cyclones, hydrosizer, concentrate thickener and filter, tailings thickener, transfer conveyer to transport concentrate across the Cacheu River, and truck loadout. The product is then trucked 75 km to the port of Ponta Chugue, where it is unloaded, conveyed through a rotary dryer, stockpiled, and conveyed via a shiploader to direct load 35,000 DWT ships.

From the project execution plan and schedule it is assumed that mechanical completion and commissioning will be completed by the second quarter of year -1, which will be followed by a ramp-up period.

The mill ramp-up rate increases gradually over the next 6 months after commissioning in year -1 with full production assumed to be reached at the beginning of the last quarter of year -1. The total tonnage processed in Year -1 is 0.438 million tonnes compared with 1.75 million tonnes in full production years.

The operating cost in the model was calculated based on diesel price assumption of USD $0.80 per litre. Diesel accounts for 40% of the LOM operating cost. A sensitivity of the impact of diesel pricing was computed and is in Table 22-7 below.

Recoveries

The metallurgical program was conducted by KEMWorks Technology Inc. (KEMWorks), SGS Mineral Services (SGS) and ALS Metallurgy Kamloops (ALS).

The indicated P_2O_5 product grade was 34% with a mass recovery of 75.5%.

Production

It is assumed that the company will mine the south pit from the last quarter of -1 production year to production year +8 and then mine the north pit for the rest of the mine life.

The bench scale tests have been performed on samples from the South pit only. For the South pit, a 9.7% premium over the CRU Group's (CRU) estimate for Morocco K10 FOB price has been assumed.

Further bench scale tests on the North pit will be performed in the fourth quarter of 2015. Because of the modest differences in the ore in the South pit versus the North pit, a premium of 4.7% has been assumed for the North Pit until bench scale tests for the North pit can be completed.

Product Pricing

Product pricing was provided by CRU Group (CRU) in July 2015 for the period of 2015 to 2019, and include an average long term forecast of USD $123/tonne for the K10 Morocco P_2O_5 from 2019 onward. Added to this price are premium percentages for the higher grade of the Farim phosphate ore. From 2020 onward, the model pricing has been computed using the current K10 Morocco P_2O_5 CRU price from 2019 and then escalated on a yearly basis at a rate of 2% per annum. The product is priced on an FOB basis, it therefore includes all operating costs up to loading on the ocean vessels.

Assumptions

The financial model is based on the following assumptions:

- Start early works engineering by December 2015;
- Timely issuance of the required permits allowing exploitation of deposit and decision to commence construction by 1st quarter 2016;
- Escalation on product pricing and all operating cost including labour at 2% per annum;
- A 10 year income tax holiday will be granted from the commencement of production; and
- No allowance for foreign exchange fluctuation.

22.2 NPV, IRR and Payback Period

Table 22-1and Table 22-2 summarize the financial analysis modelled. NPV is calculated on an end basis.

Table 22-1 Financial Data

Revenue	USD $ X'000	5,476,899
Total Pre-Production Capital	USD $ X'000	205,279
Life of Mine Operating Cost	USD $ X'000	2,409,967
Total Sustaining Capital	USD $ X'000	366,597
Operating Margin Ratio (Op. Revenue / OpEx)		2.3
Royalties	USD $ X'000	109,538
Income Taxes	USD $ X'000	443,898
Pre-Tax Cumulative Cash flow	USD $ X'000	2,358,458
After-Tax Cumulative Cash flow	USD $ X'000	1,914,560

Table 22-2 Financial Statistics

		After Tax	Pre-tax
Cumulative net cash flow			
Undiscounted (BASE YEAR 2015)	USD $000	1,914,560	2,358,458
Net present value			
Discounted at 5%	USD $000	869,789	1,026,461
Discounted at 8%	USD $000	570,224	657,860
Discounted at 10%	USD $000	436,890	497,396
Discounted at 15%	USD $000	231,384	256,679
Internal rate of return	USD $000	34.5%	34.9%
Payback period	Years	4.3	4.3

Investment Incentives

GB Minerals is at an advanced stage of negotiating the unlisted investment incentives with the government of Guinea Bissau, namely amongst others:

- 10 year corporate income tax holiday (from commencement of production).
- Exemption from custom duties on imported equipment and machinery required for the operation and processing of the phosphate ore.

For the purpose of this report, we have only incorporated the corporate income tax holiday for the first 10 years from commencement of full production and assume zero duty on the imported equipment in the initial capital cost for the project. In the unlikely event that the corporate tax holiday is not approved Table 22-3 below shows the impact of such a denial. Should this happen, the project statistics are still robust and should proceed to the next stage.

Table 22-3 Income Tax Holiday Impact

ON AFTER TAX FINANCIAL STATISTICS		With 10 years Tax Holiday	Without Tax Holiday
Net present value			
Discounted at 10%	USD $000	436,890	336,626
Internal rate of return		34.5%	28.7%
Income Tax Payable	USD $000	443,898	671,281
Payback period	Years	4.3	4.8

Sustaining Capital

The project requires additional sustaining capital of USD $274.3 million in 2015 dollars, escalated at 2% per annum equals USD $366.6 million for LOM. These costs are largely to purchase additional mining equipment (Golder), for additional tailings facility storage and hydrology (Knight Piésold), ship loading tug boat, pilot boat, navigation aids and wharf maintenance (Baird) and plant and port mobile equipment.

Corporate Expense

As at the time of compiling this report, GB Minerals has only this Farim property, it has therefore been assumed that all corporate expenses of the head office will be borne by this project. The corporate expense has been included as part of the operating expense under “Other Costs”. The LOM corporate expense was calculated with 2% escalation at USD $115.7 million.

Closeout

The closeout cost estimated at USD $27.06 million LOM dollars. A salvage value expected to be realized from the sale of the equipment and structural steel materials of approximately USD $3.5 million has been credited into the closure cost. The closure is assumed to take place in Years +27 and +28. Under listed are the closure costs for the project calculated by various consultants:

Table 22-4 Close-out Cost

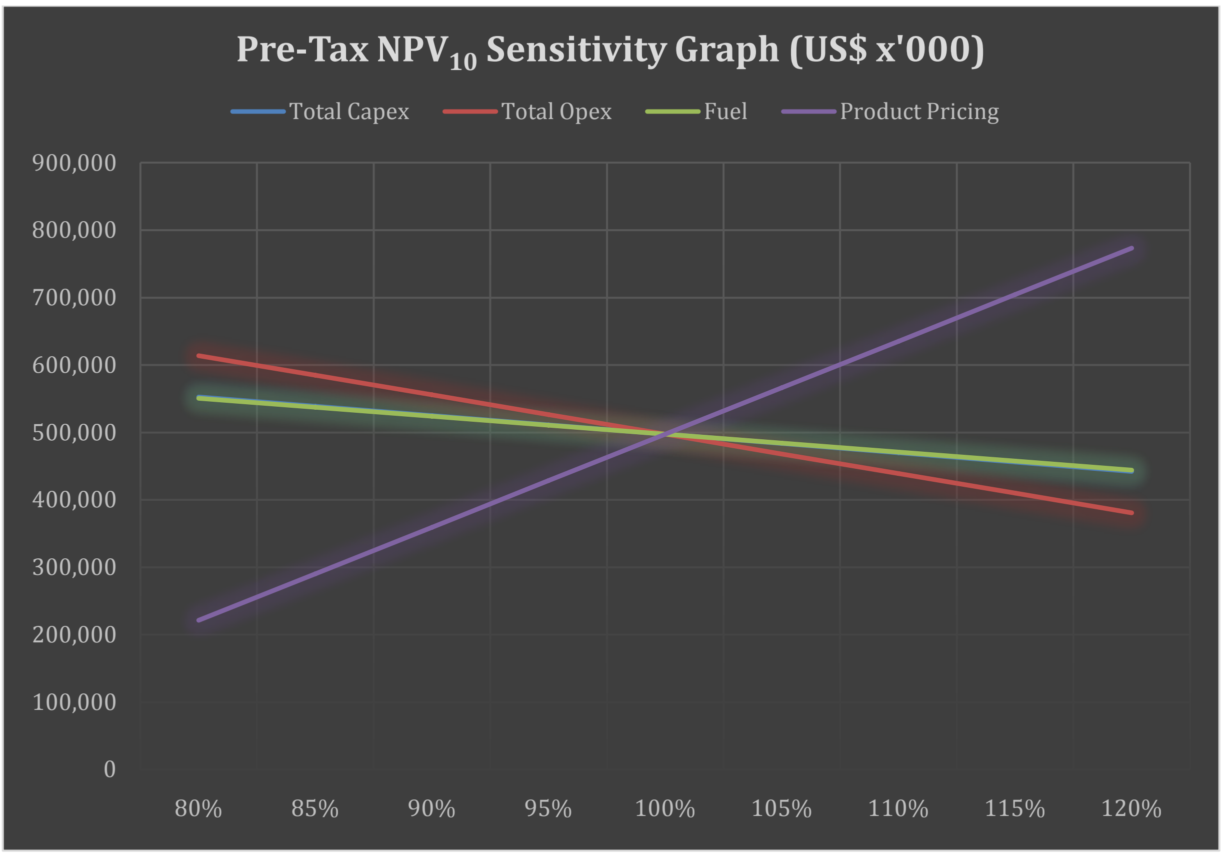
Closure Area	Amount Required (x ‘000)	Responsibility
Mine	USD $ 29,125	Golder
Tailings	USD $ 1,499	Knight Piésold
Salvage Value	USD $ 3,563	Kristal Font Inc.
Total	USD $ 27,061	

22.3 Sensitivity Analysis

The following Figure 22-1 Sensitivity Graph revolves around the pre-tax NPV@10% of USD $497,396,000.

The graph below shows the sensitivity of NPV_{10} (Pre-Tax) to capital costs, operating costs, fuel prices and revenue. The value of the project is more sensitive to revenue, operating costs, fuel prices and initial capital cost in that order respectively.

Figure 22-1 Sensitivity Graph



Additionally sensitivities are shown in Table 22-5 to Table 22-9 below and revolve around an after tax NPV_{10} of USD $436,890,000:

Table 22-5 Operating Cost vs revenue NPV Sensitivities

After Tax NPV @ 10% (USD x '000)	436,890			Opex-Mining, Non-mining, Ship-loading & Fuel (sensitivities shown in rows, top to bottom)						
		80%	85%	90%	95%	**100%**	105%	110%	115%	120%
Revenue, Product Pricing (sensitivities shown in columns, left to right)	120%	80,038	146,185	208,333	270,481	332,629	394,777	456,924	519,072	581,220
	115%	110,103	172,251	234,398	296,546	358,694	420,842	482,990	545,137	607,285
	110%	136,168	198,316	260,464	322,611	384,759	446,907	509,055	571,203	633,350
	105%	162,233	224,381	286,529	348,677	410,824	472,972	535,120	597,268	659,415
	100%	188,298	250,446	312,594	374,742	**436,890**	499,037	561,185	623,333	685,481
	95%	214,364	276,511	338,659	400,807	462,955	525,102	587,250	649,398	711,546
	90%	240,429	302,577	364,724	426,872	489,020	551,168	613,315	675,463	737,611
	85%	266,494	328,642	390,789	452,937	515,085	577,233	639,381	701,528	763,676
	80%	292,559	354,707	416,855	479,002	541,150	603,298	665,446	727,594	789,741
	75%	318,624	380,772	442,920	505,068	567,215	629,363	691,511	753,659	815,806

Table 22-6 Total Capital Cost vs Revenue NPV Sensitivities

After Tax NPV @ 10% (USD x '000)	436,890			Capex – incl. Owner's cost, Sustaining & Closure Costs (sensitivities shown in rows, top to bottom)						
		80%	85%	90%	95%	**100%**	105%	110%	115%	120%
Revenue, Product Pricing (sensitivities shown in columns, left to right)	120%	133,060	195,214	257,369	319,523	381,677	443,831	505,985	568,139	630,293
	115%	146,870	209,022	271,175	333,327	395,480	457,633	519,785	581,938	644,090
	110%	160,679	222,830	284,981	347,132	409,283	471,434	533,585	595,736	657,887
	105%	174,489	236,638	298,788	360,937	423,086	485,236	547,385	609,534	671,684
	100%	188,298	250,446	312,594	374,742	**436,890**	499,037	561,185	623,333	685,481
	95%	202,108	264,254	326,400	388,546	450,693	512,839	574,985	637,131	699,277
	90%	215,917	278,062	340,207	402,351	464,496	526,640	588,785	650,930	713,074
	85%	229,727	291,870	354,013	416,156	478,299	540,442	602,585	664,728	726,871
	80%	243,536	305,678	367,819	429,961	492,102	554,244	616,385	678,526	740,668
	75%	257,346	319,486	381,626	443,765	505,905	568,045	630,185	692,325	754,465

Table 22-7 Fuel Cost vs Revenue NPV Sensitivities

After Tax NPV @ 10% (USD x '000)	436,890			Fuel (sensitivities shown in rows, top to bottom)						
		80%	85%	90%	95%	**100%**	105%	110%	115%	120%
Revenue, Product Pricing (sensitivities shown in columns, left to right)	120%	140,606	202,754	264,902	327,049	389,197	451,345	513,493	575,640	637,788
	115%	152,529	214,677	276,825	338,972	401,120	463,268	525,416	587,564	649,711
	110%	164,452	226,600	288,748	350,896	413,043	475,191	537,339	599,487	661,634
	105%	176,375	238,523	300,671	362,819	424,966	487,114	549,262	611,410	673,558
	100%	188,298	250,446	312,594	374,742	**436,890**	499,037	561,185	623,333	685,481
	95%	200,221	262,369	324,517	386,665	448,813	510,960	573,108	635,256	697,404
	90%	212,145	274,292	336,440	398,588	460,736	522,883	585,031	647,179	709,327
	85%	224,068	286,215	348,363	410,511	472,659	534,807	596,954	659,102	721,250
	80%	235,991	298,139	360,286	422,434	484,582	546,730	608,877	671,025	733,173
	75%	247,914	310,062	372,209	434,357	496,505	558,653	620,801	682,948	745,096

Table 22-8 Operating Cost vs Total Capital Cost NPV Sensitivities

After Tax NPV @ 10% (USD x '000)	436,890			Opex - Mining, Non-mining, Ship-loading & Fuel (sensitivities shown in rows, top to bottom)						
		80%	85%	90%	95%	**100%**	105%	110%	115%	120%
Total Capital Cost (sensitivities shown in columns, left to right)	120%	388,355	374,423	360,492	346,560	332,629	318,697	304,766	290,834	276,903
	115%	414,292	400,392	386,493	372,593	358,694	344,795	330,895	316,996	303,096
	110%	440,229	426,361	412,494	398,627	384,759	370,892	357,024	343,157	329,290
	105%	466,165	452,330	438,495	424,660	410,824	396,989	383,154	369,319	355,483
	100%	492,102	478,299	464,496	450,693	**436,890**	423,086	409,283	395,480	381,677
	95%	518,039	504,268	490,497	476,726	462,955	449,184	435,413	421,642	407,870
	90%	543,976	530,237	516,498	502,759	489,020	475,281	461,542	447,803	434,064
	85%	569,912	556,206	542,499	528,792	515,085	501,378	487,671	473,964	460,258
	80%	595,849	582,174	568,500	554,825	541,150	527,475	513,801	500,126	486,451
	75%	621,786	608,143	594,501	580,858	567,215	553,573	539,930	526,287	512,645

Table 22-9 Operating Cost vs Revenue IRR Sensitivities

				Opex - Mining, Non-mining, Ship-loading & Fuel (sensitivities shown in rows, top to bottom)						
After Tax IRR	**34.5%**	80%	85%	90%	95%	**100%**	105%	110%	115%	120%
Revenue, Product Pricing (sensitivities shown in columns, left to right)	120%	14.8%	19.0%	22.6%	26.0%	29.2%	32.3%	35.3%	38.2%	41.0%
	115%	16.6%	20.6%	24.1%	27.4%	30.6%	33.6%	36.6%	39.4%	42.2%
	110%	18.3%	22.1%	25.5%	28.8%	31.9%	34.9%	37.8%	40.7%	43.4%
	105%	20.0%	23.6%	27.0%	30.2%	33.2%	36.2%	39.1%	41.9%	44.7%
	100%	21.5%	25.0%	28.3%	31.5%	**34.5%**	37.5%	40.3%	43.1%	45.9%
	95%	23.1%	26.5%	29.7%	32.8%	35.8%	38.7%	41.6%	44.4%	47.1%
	90%	24.5%	27.9%	31.1%	34.1%	37.1%	40.0%	42.9%	45.6%	48.3%
	85%	26.0%	29.3%	32.4%	35.5%	38.4%	41.3%	44.1%	46.9%	49.6%
	80%	27.4%	30.7%	33.8%	36.8%	39.7%	42.6%	45.4%	48.1%	50.8%
	75%	28.8%	32.0%	35.1%	38.1%	41.0%	43.8%	46.6%	49.3%	52.0%

Table 22-10 provides the life of project cash flow with time periods presented as years.

Table 22-10 Life of Project Cash Flow

Calendar year		2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Production	000 mt			330	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321
Revenue	**USD$000**			**46,025**	**184,100**	**178,301**	**181,867**	**185,505**	**189,215**	**192,999**	**196,859**	**200,796**	**204,812**	**199,386**	**203,374**	**207,442**	**211,591**
Total Royalty & Freight	**USD$000**			**(920)**	**(3,682)**	**(3,566)**	**(3,637)**	**(3,710)**	**(3,784)**	**(3,860)**	**(3,937)**	**(4,016)**	**(4,096)**	**(3,988)**	**(4,067)**	**(4,149)**	**(4,232)**
Net Revenue	**US$000**			**45,104**	**180,418**	**174,735**	**178,230**	**181,794**	**185,430**	**189,139**	**192,922**	**196,780**	**200,716**	**195,399**	**199,307**	**203,293**	**207,359**
Operating costs																	
Mining																	
Mining, Labour	USD$000			3,403	3,880	3,881	3,832	3,703	3,673	3,974	4,662	5,087	4,808	4,951	5,695	5,994	6,308
Mining, Other Cost	USD$000			711	13,785	14,394	13,876	12,398	11,496	12,640	17,153	19,373	19,203	19,123	22,072	25,040	28,075
Non-Mining	USD$000																
G&A	USD$000	1,048	1,966	3,678	3,751	3,826	3,903	3,981	4,061	4,142	4,225	4,309	4,395	4,483	4,573	4,664	4,758
Labour (exc Mine & Shiploading)	USD$000			5,469	7,032	7,172	7,316	7,462	7,611	7,763	7,919	8,077	8,239	8,403	8,571	8,743	8,918
Operating Consumables (ex Fuel)	USD$000			359	1,466	1,496	1,526	1,556	1,587	1,619	1,651	1,684	1,718	1,752	1,787	1,823	1,860
Power (excluding fuel)	USD$000			268	1,095	1,117	1,139	1,162	1,185	1,209	1,233	1,258	1,283	1,309	1,335	1,361	1,389
Maintenance Materials	USD$000			354	1,443	1,472	1,502	1,532	1,562	1,593	1,625	1,658	1,691	1,725	1,759	1,795	1,830
Shiploading Costs	USD$000																
Labour	USD$000			443	1,806	1,842	1,879	1,917	1,955	1,994	2,034	2,075	2,116	2,159	2,202	2,246	2,291
Maintenance, Consumables & Others	USD$000			319	1,356	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278
Total Fuel Cost	USD$000			19,155	27,986	30,328	30,422	30,103	29,098	29,380	32,175	36,523	37,471	37,442	38,955	42,762	45,675
Other Costs																	
Corporate Overhead				3,438	3,506	3,576	3,648	3,721	3,795	3,871	3,949	4,028	4,108	4,190	4,274	4,360	4,447
Pre-production Cost (ramp up-Salaries)				2,721													
Financing Costs		300	3,000														
EBITDA	**USD$000**	**(1,348)**	**(4,966)**	**4,786**	**113,312**	**104,353**	**107,910**	**112,983**	**118,130**	**119,676**	**115,018**	**111,432**	**114,407**	**108,583**	**106,805**	**103,226**	**100,531**
Total Depreciation				**15,998**	**35,154**	**40,134**	**36,629**	**31,600**	**26,865**	**24,242**	**20,979**	**21,195**	**19,293**	**20,119**	**18,054**	**20,537**	**21,722**
Income tax payable	**USD$000**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**(24,464)**	**(22,635)**	**(21,605)**
Net earnings	USD$000	(1,348)	(4,966)	4,786	113,312	104,353	107,910	112,983	118,130	119,676	115,018	111,432	114,407	108,583	82,341	80,591	78,926
Total Capital Cost	**USD$000**	**5,588**	**149,886**	**63,533**	**23,858**	**6,299**	**5,890**	**5,022**	**3,383**	**14,602**	**20,995**	**25,040**	**9,934**	**28,476**	**17,649**	**22,763**	**24,557**
Net project cash flow																	
Pre-tax	USD$000	(6,936)	(154,852)	(58,747)	89,454	98,054	102,020	107,961	114,747	105,074	94,023	86,392	104,472	80,107	89,156	80,463	75,974
After tax	USD$000	(6,936)	(154,852)	(58,747)	89,454	98,054	102,020	107,961	114,747	105,074	94,023	86,392	104,472	80,107	64,692	57,828	54,369

Calendar year		2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044
Production	000 mt	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321	1,321		
Total Revenue	**USD$000**	**215,822**	**220,139**	**224,542**	**229,032**	**233,613**	**238,285**	**243,051**	**247,912**	**252,870**	**257,928**	**263,086**	**268,348**		
Total Royalty & Freight	**USD$000**	**(4,316)**	**(4,403)**	**(4,491)**	**(4,581)**	**(4,672)**	**(4,766)**	**(4,861)**	**(4,958)**	**(5,057)**	**(5,159)**	**(5,262)**	**(5,367)**		
Net Revenue	**USD$000**	**211,506**	**215,736**	**220,051**	**224,452**	**228,941**	**233,520**	**238,190**	**242,954**	**247,813**	**252,769**	**257,824**	**262,981**		
Operating costs															
Mining															
Mining, Labour	USD$000	6,508	6,285	6,061	5,900	5,923	5,704	5,823	5,913	6,282	7,138	7,482	7,291		
Mining, Other Cost	USD$000	29,626	26,784	23,447	23,954	23,950	22,097	21,312	21,192	23,164	26,744	29,048	32,967		
Non-Mining															
G&A	USD$000	4,853	4,950	5,049	5,150	5,253	5,358	5,465	5,574	5,686	5,800	5,916	6,034		
Labour (ex Mine & Shiploading)	USD$000	9,096	9,278	9,464	9,653	9,846	10,043	10,244	10,449	10,658	10,871	11,088	11,310		
Operating Consumables (ex Fuel)	USD$000	1,897	1,935	1,973	2,013	2,053	2,094	2,136	2,179	2,222	2,267	2,312	2,358		
Power (ex fuel)	USD$000	1,416	1,445	1,474	1,503	1,533	1,564	1,595	1,627	1,660	1,693	1,727	1,761		
Maintenance Materials	USD$000	1,867	1,904	1,942	1,981	2,021	2,061	2,103	2,145	2,187	2,231	2,276	2,321		
Shiploading Costs	USD$000														
Labour	USD$000	2,336	2,383	2,431	2,479	2,529	2,580	2,631	2,684	2,738	2,792	2,848	2,905		
Maintenance, Consumables & Others	USD$000	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278		
Total Fuel Cost	USD$000	48,352	48,841	45,775	44,395	45,861	45,482	44,182	44,847	45,845	48,820	52,197	54,925		
Other Costs															
Corporate Overhead		4,536	4,626	4,719	4,813	4,910	5,008	5,108	5,210	5,314	5,421	5,529	5,640		
Pre-production Cost (ramp up-Salaries)															
Financing Costs															
EBITDA	**USD$000**	**99,741**	**106,027**	**116,438**	**121,332**	**123,785**	**130,252**	**136,313**	**139,857**	**140,780**	**137,716**	**136,125**	**134,190**		
Total Depreciation		**21,041**	**20,850**	**19,247**	**21,159**	**19,257**	**18,380**	**19,162**	**16,269**	**13,917**	**12,851**	**12,535**	**12,426**	**0**	**0**
Income tax payable	**USD$000**	**(21,702)**	**(24,006)**	**(27,653)**	**(28,096)**	**(29,139)**	**(31,470)**	**(32,913)**	**(34,533)**	**(35,260)**	**(34,415)**	**(34,155)**	**(41,858)**	**0**	**0**
Net earnings	USD$000	78,038	82,021	88,785	93,236	94,647	98,782	103,400	105,324	105,520	103,301	101,970	92,333		
Total Capital Cost	**USD$000**	**17,277**	**13,635**	**5,543**	**25,718**	**24,342**	**8,641**	**8,261**	**19,670**	**20,868**	**16,302**	**7,617**	**(23,472)**	**24,389**	**2,671**
Net project cash flow															
Pre-tax	USD$000	82,463	92,392	110,895	95,615	99,443	121,610	128,052	120,186	119,912	121,414	128,508	157,662	(24,389)	(2,671)
After tax	USD$000	60,761	68,386	83,242	67,519	70,305	90,141	95,139	85,653	84,652	86,999	94,353	115,805	(24,389)	(2,671)
Cumulative Cashflow															
Pre-tax	USD$000	1,089,828	1,182,220	1,293,115	1,388,729	1,488,173	1,609,783	1,737,835	1,858,022	1,977,934	2,099,348	2,227,856	2,385,518	2,361,129	2,358,458
After-tax	USD$000	999,421	1,067,808	1,151,050	1,218,568	1,288,873	1,379,014	1,474,153	1,559,806	1,644,458	1,731,457	1,825,810	1,941,614	1,917,225	1,914,554

23.0 ADJACENT PROPERTIES

There are no material mineral properties adjacent to the Project.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Strategy

This section describes the proposed organization and philosophy that is considered most appropriate for the effective design, engineering, construction and commissioning of the Farim Phosphate Project and associated infrastructure.

The objective is to provide the most economical approach for GB Minerals. A priority has also been to ensure the shortest possible construction period is achieved without risking the quality of work.

A goal for the execution phase of the project will be the attainment of the best safety record possible. To accomplish this, it is required that all contractors and involved personnel adhere to defined safety objectives and standards developed by the Engineer. These will include all appropriate safety requirements of GB Minerals and as specified by acts and regulations in Guinea Bissau.

The proposed execution strategy for the Farim Phosphate Project is based on an engineering, procurement and construction management (EPCM) implementation approach and horizontal discipline based contract packaging. An experienced engineering firm (the Engineer) will be engaged to provide EPCM services for the development of the process plant and the associated infrastructure. Specialist consultants will be contracted to address specific elements of the Project outside the core competency of the Engineer, including mining, geotechnical, environmental and the tailings management facility (TMF). An integrated project team approach will be adopted for the Project.

In general the key execution aspects which have the most significant impact on the capital costs are the contracting plan and the overall project schedule. They are of course interrelated, however the project schedule will significantly affect the duration based EPCM management costs including owner's costs, whilst the contracting plan significantly impacts the fabrication and installation rates and the contractor indirect costs.

Any change to the execution approach i.e. if an EPC approach was contemplated then this will impact the capital cost and the level of contingency applied to cover the EPC contractor's margin and risk profile.

Project Objectives

The key objectives during the execution phase of the Project are listed below:

- Health and Safety – Meet or exceed safety targets. Attain zero harm incidents during construction. Design for a safe operating environment;
- Environment – Be environmentally and socially responsible following the Equator Principles and International. Have no serious environmental incidents. Comply with all permit requirements;
- Community – Maintain good community relations. Maximize utilization of local available resources and involvement of the local community. Leave a positive legacy;

- Capital Cost – Target to achieve the lowest cost outcome without compromising quality and schedule. Complete the Project within the Project control budget;

- Quality – Design the Project to be fit for purpose, easy to maintain, operator friendly and safe; and

- Schedule – Schedule to meet start-up requirements. Attain mechanical completion within 16 months. Meet ramp-up targets.

Engineer's Responsibility

An Engineer will be appointed by the Owner and will execute the engineering and procurement services project from their home office and will maintain a site office at the Project sites.

The scope of services for the Engineer will be inclusive of the following:

- Process design including the process flow sheets, design criteria, final mass balance and water balance;

- Engineering design including the preparation of technical specifications, material and equipment data sheets, equipment lists, line lists, valve lists, cable schedules, electrical load lists and instrument lists;

- Design and detail engineering and drafting including P&ID's, layout drawings, general arrangement drawings and detail drawings for the civil, structural steel, platework, mechanical, piping and electrical disciplines;

- Procurement services including the tender, adjudication and recommendation for award of all purchase orders and contracts required for the expenditure of the capital works on the project;

- Preparation and issue of tender packages for steelwork and platework fabrication supply contracts;

- Preparation and issue of tender packages for site construction packages based on horizontal contract packages for: earthworks, building works, concrete works, field erected tankage, structural, mechanical and piping installation, electrical and instrumentation supply and installation;

- Project management services to co-ordinate and manage all aspects of the project and interface with the various vendors, suppliers and contractors involved in the project. This would include the preparation and maintenance of the project quality plan, the safety management plan, the project budget allocation and control, the project implementation schedule, contract preparation and control and project reporting;

- Project services including inspection and expediting, transport co-ordination and invoice approval and control;

- Construction management including site management, construction supervision, site safety, industrial relations and site interface. This will involve the co-ordination and management of all construction activities from site establishment to final completion including the completion of all punch listing activities and the rectification of minor defects and omissions;
- Commissioning the Project including all testing and pre-commissioning, dry commissioning and wet commissioning;
- Provision of a set of as-built drawings at the conclusion of construction;
- Provision of a complete set of maintenance and operator manuals based on vendor supplied manuals for mechanical and electrical equipment;
- Preparation and submission of a Health, Safety, Environmental and Community Relations (HSECR) Management Plan in accordance with any GB Minerals policies, procedures, rules and regulations;
- Provision of a Project Quality Plan based on ISO9000 and ensuring that the quality plan is effectively implemented during the project; and
- Provision of records of design audits and HAZOP studies.

GB Minerals' Responsibilities

The scope of services being managed by GB Minerals will be inclusive of the following:

- Geology and mining;
- Obtaining government approvals;
- Obtaining building permits;
- Obtaining duty exception;
- Approval of purchase orders and contracts prepared by the Engineer on behalf of GB Minerals;
- Payment of project direct and indirect costs;
- Management of the EPCM Engineer;
- Assist the Engineer with the planning and implementation of the project;
- Provision of first fill, opening stocks and consumables, spare parts, office equipment and mobile equipment; and

- Provision and training of the geology, mining, plant and port operations and maintenance workforce and the administration team to operate the plant and port facilities pre and post commissioning.

24.2 Project Implementation Plan

24.2.1 Project Management

Project Planning

The Engineer's project manager, in conjunction with the Owner's project manager, will be responsible for the preparation and implementation of the project management plan (PMP) to complement the project execution plan (PEP). The PMP will be further detailed and updated as required during detailed engineering and project execution phases of the Project.

Risk Management

The Engineer's project manager, in conjunction with the Owner's project manager, will be responsible for the preparation and implementation of the risk management plan for the project which shall address the identification, qualification, quantification, mitigation, and successful management of Project business risks, as well as opportunities.

Project Quality Management

The Project will implement a quality management program in accordance with owner's requirements.

24.2.2 Health, Safety, Security and Environment

Objectives and Approach

GB Minerals considers safety of utmost importance in the delivery of the Farim Phosphate Project and as such will commit to provide leadership to all stakeholders to obtain excellent safety results.

The health, safety, security and environment (HSSE) strategy for the Project will aim to:

- Integrate HSSE delivery into all project disciplines;
- Ensure alignment by all parties (Owner, Engineer, specialist consultants, service providers, construction contractors and vendors) to the common set of goals and objectives;
- Identify HSSE issues as early as reasonably practicable in Project development;
- Manage the identified HSSE risks by avoidance, prevention, control, and mitigation;
- Manage risks to personnel to a level "As low as reasonably practicable" (ALARP);
- Pursue zero damage to the environment and challenge any deviations; and

- Promote awareness and manage health issues.

Health, Safety, Security and Environment Execution Plan

The Engineer's health and safety manager together with GB Minerals' health, environmental and security manager(s) will be responsible for preparing, implementing, and updating the project specific HSSE plan. The HSSE plan and organizational structure will be approved by the Engineer's project and construction manager, and the Owner's project manager.

A detailed HSSE plan will be prepared prior to the start of construction activities in accordance with the framework and Owner requirements.

The HSSE plan is intended to provide a project specific work plan to organize, perform, and execute the HSSE responsibilities for the Project. It contains key group-wide expectations, responsibilities, processes, and procedures for ensuring that Project activities, including engineering, design, construction, management, operations and maintenance operations, are undertaken safely in a consistent manner and in line with contractual and regulatory requirements and policy commitment.

24.2.3 Project Controls

Project Controls Scope of Work

Project controls cover the following work:

- Planning and scheduling;
- Estimating and cost control;
- Change management;
- Project reporting (progress, cost, etc.);
- Associated project management systems and coding structures;
- Document and information management; and
- Project accounting.

Project Controls Plan

The Engineer's project controls manager will be responsible for preparing, implementing, and updating the project specific project controls plan (PCP) and organizational structure in coordination with, and approval of the Engineer's project manager.

The PCP will be prepared at the start of the basic and detailed engineering phases. The project controls group will organize, perform, and execute the project controls common to all phases of the Project and will be applicable to project management, engineering, procurement and contracts management, materials management, construction, commissioning and HSSE.

Work Breakdown Structure

The work breakdown structure (WBS) is used to control the execution of the Project and to provide project related information to various project stakeholders to suit their needs. The WBS is comprised of several coding structures, which are applied to the central project data such that the information can be sorted, managed and reported in various ways.

24.2.4 Detailed Engineering and Design

An engineering plan will be prepared by the Engineer's design manager and will define the principles and execution guidelines that will be adopted by the Engineer's team and any sub-consultants during the design phase of the Project. The engineering plan will identify the various engineering deliverables required at the tender, procurement, construction, commissioning, close-out, and handover stages.

At the beginning of the basic engineering phase of the Project a priority will be given to freezing the process design criteria, the process flow sheets and the process plant layout, and issuing the deliverables supporting the procurement of the long lead equipment – attrition scrubber, rotary dryer, ship loader, and diesel power plants.

Design reviews will be undertaken at predetermined stages of production of the technical documents and 3D models. The main objective of the design reviews is to verify that:

- Statutory requirements, codes, and standards are complied with;
- The technical documents meet the design requirements and are suitable for their intended purpose;
- Conflicts, unresolved issues, potential problems are identified, responsibility for resolution assigned and completion dates scheduled;
- The design is constructible; and
- HSSE requirements and lessons learned have been addressed.

Hazards and operability (HAZOP), constructability, and other reviews will be scheduled as appropriate. Model reviews will be conducted at 30%, 60%, and 90% of engineering development.

Technical peer reviews for the Project will be undertaken of the principal design documents and for any significant technical risk items identified in the risk register.

The engineering discipline leads will ensure all the external documents are reviewed by all disciplines in accordance with the engineering procedure as set in the engineering plan.

24.2.5 Procurement Management

The Procurement scope will cover all formal tendering of packages to achieve competitive pricing and an effective negotiating position to provide value for money to GB Minerals. Tendering will be based on a lump sum basis for equipment design and supply and on a “schedule of rates” basis for bulk materials supply, where applicable. Equipment suppliers will be selected on the basis of previous experience, ability to meet design requirements and the project schedule.

An equipment bidders list will be developed using global sourcing of prequalified vendors and GB Minerals’ preferred suppliers list. Following the compilation of technical requirements, tender packages will be prepared and issued to the approved bidders. Bidder’s questions will be addressed, clarification notices will be prepared and issued as necessary, and bids formally received.

Bids will be evaluated on the basis of obtaining the best value in terms of price, delivery, and equipment quality. Evaluation criteria will be developed for critical packages.

Technical evaluation will fundamentally be based on technical compliance, but will also consider supplier experience, ongoing operational and maintenance support and consumable spares strategy. Meetings, where required, will be arranged with the bidders to clarify and confirm technical and commercial matters and to finalize and obtain a satisfactory agreement for GB Minerals whose representative will be invited to participate in these meetings.

Participation by regional suppliers will be pursued to the maximum possible extent on the basis of quality, schedule, overall cost effectiveness, previous experience, and availability to perform the work. Direct negotiations with smaller local business groups on specific packages will be planned to encourage local sourcing of equipment and material.

Transport, logistics, customs clearance, and expediting services will be managed by the Project’s transport and logistics contractor. The majority of the purchase orders will be based on ex-works price basis.

Purchase orders for the supply of materials and equipment will specify packaging requirements to cater for sea freight and inland transport. The Engineer will supply the specification to which packaging of all materials and equipment must comply.

GB Minerals will be responsible for off-loading and storage of the bulk materials and equipment needed by the Project. The warehouse foreman and workers will be directly employed by the Owner. The Engineer will be responsible for the interface with the transport and logistics contractor.

Procurement of long lead equipment will be given the highest priority. The tender package preparation, manufacturing, and installation of these items will form the critical path for the Project schedule. All the engineering deliverables associated with these packages will be identified early in the basic engineering design phase and expedited to the maximum possible extent.

24.2.6 Contracting Plan

Lycopodium has chosen a balance between EPCM effort and contractor effort in the selection of the contracting plan which the estimate is based on. The interface areas will be managed by the appointed EPCM Engineer.

Horizontal packages have been selected and in some instances the supply is split from the installation to realise cost savings.

Table 24-1 below provides a preliminary key contracting plan for the purposes of providing an estimate basis.

Table 24-1 Contracting Plan

Contract Package	Description	Type
Off-site Fabrication Contract #1	Misc Platework (Bins, chutes etc) Pkg #1	Fixed priced schedule of rates
Off-site Fabrication Contract #2	Misc Platework (Bins, chutes etc) Pkg #2	Fixed priced schedule of rates
Off-site Fabrication Contract #3	Structural Steelwork Pkg #1 Plant Site	Fixed priced schedule of rates
Off-site Fabrication Contract #4	Structural Steelwork Pkg #2 Port Site	Fixed priced schedule of rates
Field Erected Tankage Contract	Supply, fabricate and erection	Lump Sum
Transport & Logistics Contract	Sea and road transport of goods to Site	Fixed Schedule of Rates
Earthworks Contract	Bulk Earthworks and detailed earthworks and drainage ,site roadwork's etc at plant and port site	Fixed Schedule of Rates
TMF (Tailings Management Facility) Contract	Earthworks and misc services for TMF	Fixed Schedule of Rates
Concrete Contract	Detailed earthworks and concrete supply and installation	Fixed Schedule of Rates
Structural, Mech. and Piping Contract	Structural/Mechanical/Piping – Supply of piping and Installation of SMP at plant and port site	Fixed Schedule of Rates
Electrical/Instrumentation Contract	Electrical/Instrumentation - Supply of bulk materials and Installation at plant and port site	Fixed Schedule of Rates
Process Controls Contract	Controls - Supply and Installation	Lump Sum and Day Rates
Site Buildings – Pre-Fabricated Type	Supply ex-works	Lump Sum
Site Buildings – Steel Framed , Sheeted Type	Supply ex-works	Lump Sum
Site Buildings – Concrete Block work Type	Supply and Install	Lump Sum
Fencing Contract	Supply and install	Lump Sum
Building Installation Contract	Install buildings, fit out and services	Fixed Schedule of Rates
Overland Piping Contract	Supply and Install	Lump Sum

24.2.7 Construction Management

The construction methodology proposed for the Project has the following aims:

- To provide a safe working environment;
- To achieve cost and schedule targets;
- To adopt a cost effective and fit for purpose construction methodology in contracting and site management based on tried and proven philosophies;
- To allow optimization in constructability;

- To provide a management plan that complies with the requirements of both GB Minerals and the Engineer's safety and environmental policies; and

- To achieve maximum possible utilization of local resources.

A construction management plan (CMP) will be prepared to provide a project specific statement and work plan on how the construction management team will organize, perform, and execute the construction management responsibilities for the Project. The CMP will define the interfaces with engineering, procurement, and commissioning, to ensure construction is executed in a timely, cost effective manner in accordance with all project objectives. The CMP will document the intended construction approach to the Project starting with early site capture through commissioning and start-up. The intent of this document is to clearly lay out the planned approach to construction for understanding by all Project stakeholders.

An integrated construction management team led by the construction manager will manage and coordinate all construction activities within the scope of the Project to ensure control over cost, schedule, and quality. This includes coordinating and managing the work interfaces between construction contractors on site and GB Minerals. This integrated team will include representatives from GB Minerals and the Engineer.

The Engineer's construction manager will be located in the engineering office during the first few months of the Project and then transferred to the site when construction commences. The construction manager will define the specific duties of key construction personnel to suit the construction requirements of the Project. He will be responsible for the overall construction planning, cost, and scheduling. The construction manager will ensure that all aspects of the work are properly set up with the necessary project controls for items such as: planning and scheduling; cost control, document control, accounting, project risk analysis, forecasting, trending, and change control.

The field engineering team will be responsible for providing engineering design support on any technical matters arising during construction.

24.2.8 Commissioning

The main objective of commissioning is to safely introduce production material to the process plant on the earliest possible date and turn over to GB Minerals an integrated plant capable of continuous and reliable performance.

A commissioning manager in conjunction with GB Minerals' process plant and port managers and his team will plan, coordinate, and execute all pre-commissioning and commissioning activities. The pre-commissioning, commissioning, turnover and acceptance methodology to be used for the facilities will be a systems based approach. The total scope of facilities for the Project will be divided into sub-systems based on operational function. A systems index will be developed and maintained as a key control document. The sub-systems will be grouped into operable systems that will be identified on the systems index and on the Project schedule. Turnover packages including pre-commissioning and commissioning documentation will be managed by operable systems with detailed tasks being performed at the sub-system level.

Upon completion of work (system completion) for the facilities, critical pre-commissioning and commissioning activities shall be performed in order to confirm all parts of the facilities are in good working order and meet the minimum acceptable performance requirements.

Pre-Commissioning and Testing

After verification that the process plant and port have been constructed in accordance with the design (conformance with P&ID's and drawings), construction and installation testing would typically include hydrostatic pressure tests, flushing of lines, alignment checks, electrical point to point checks, and component identification checks. Dry commissioning includes motor direction tests, all drives run, conveyors run and tracked, instruments checked, control system verified and facility sequence testing. By the conclusion of pre-commissioning all equipment and systems must be cleaned out.

The construction manager, via the construction supervisors, is responsible for managing all pre-commissioning activities, along with recording and approval of results. The testing will be conducted by the appropriate contractor. The commissioning manager will assist with coordinating the dry commissioning phase of pre-commissioning.

Mechanical Completion

Mechanical completion of a section of the plant is achieved when pre-commissioning is complete and it meets all requirements with respect to design, safety, physical operability and specifications and the relevant module is ready for extended operation and/or the introduction of ore/process fluids.

Wet Commissioning

Wet commissioning consists of successfully testing and operating the equipment grouped together into systems or modules, but without ore or reagents or other process material. At the successful conclusion of wet commissioning, ore/process fluids are introduced into the circuit and process commissioning commences.

Process Commissioning

Process commissioning follows the successful completion of wet commissioning. During this time, the initial introduction of ore and reagents to the process will occur. The circuit will be operated to achieve nominal throughput and metallurgical performance. Process commissioning will be managed by the commissioning manager using GB Minerals' operating personnel. The commissioning will be performed at a multi-system level, incorporating systems defined in the process functional specifications. The system start-up sequence will follow the order defined in the process flow sheets for the start-up of the process plant under normal operation. Completion of this phase is achieved once the milestone production rates for each system within the process plant have been achieved.

24.2.9 Project Close-out and Handover

Project close-out involves finalising all outstanding issues and work items when the work is complete and the Engineer's responsibilities end. At the completion of all construction and commissioning activities, the Engineer will provide the following close-out information to GB Minerals:

- As-built drawings;
- Piping and instrumentation diagrams (P&ID's);
- Electrical as-built drawings;
- Commissioning data and records;
- Quality records;
- Project close-out report; and
- Operating manuals and recommended spares lists.

The Engineer will create and issue for GB Minerals' sign off a handover certificate reflecting the fact that the plant is complete and operational, has been commissioned, that all performance warranties have been achieved and is fully functional.

24.2.10 Project Organizational Structure

24.2.11 Governance Structure

The Project will report and be accountable to GB Minerals senior management. Reporting will take place through regular weekly and monthly progress meetings and reports.

24.2.12 Allocation of Responsibility - Alignment

GB Minerals will manage the overall project and will be directly responsible for the following aspects of the scope:

- Finance, governmental approvals, environmental approvals and licenses;
- Land purchases, permits, taxes and duties unless contractually assigned to other parties;
- Mining planning and development;
- Environmental construction and operational requirements (monitoring, testing, reporting, etc.);
- On-site security and medical services;
- Community liaison and public relations;
- Operations preparedness;
- Off-site infrastructure including roads, housing and power;

- Engagement of specialist consultants and contractors for mining, waste disposal and other specialist scopes;
- GB Minerals will engage an experienced Engineer, acting as agent, to plan, engineer, construction manage and commission the Project.
- The Engineer shall be responsible for the following:
- Project management and reporting;
- Schedule development and progress;
- Cost control and cost reporting;
- Detailed engineering design, drawings and documentation;
- Field engineering, as built drawings and documentation;
- Coordination of sub-consultants as assigned by GB Minerals;
- Farim site facilities including process facilities, infrastructure, utilities, TSF, and ancillary structures;
- Port site facilities including loadout and drying facilities, marine structures, utilities, and ancillary buildings;
- Procurement of equipment and materials;
- Inspection and expediting of equipment and materials;
- Logistics including freight forwarding and transportation of equipment and materials to site;
- Development and administration of fabrication contracts;
- Development and administration of construction contracts;
- Construction management;
- Construction health and safety;
- Commissioning; and
- Close out of contracts, and purchase orders and handover of documents and data to GB Minerals.

Individual contractors will be responsible for the safe and successful execution of their work in accordance with their contracts.

A PEP will be developed early in the Project to ensure all project participants work with a unified set of tools, including procedures, forms, file numbering systems, information distribution matrices, and reporting structures.

The levels of authority for the project will be documented in the PEP.

24.2.13 Stakeholders Management

The Owner's team will be responsible to identify all stakeholders, document stakeholder requirements, and ensuring that stakeholder requirements are met.

24.2.14 Project Roles and Responsibilities

Responsibilities of the various project functions will be developed as part of the basic engineering phase of the Project. The following key roles will be clearly defined prior to the execution phase of the Project:

Table 24-2 Key Roles

GB Minerals (Owner)	Engineer
Project director	Project manager
Engineering manager	Engineering manager
(Oversight and third party engineering)	Procurement / contracts manager
Human resources manager	Project controls manager
Mine manager	Construction manager
Construction oversight	Construction superintendents
Security manager	Commissioning manager
Environmental manager	Logistics manager

24.2.15 Project Schedule

A Project execution schedule has been prepared as part of this Feasibility Study. The schedule is provided in Figure 24-1. The overall schedule duration from the start of detailed engineering to the end of commissioning is 19 months. The engineering activities will take approximately 10 months, the site construction activities will be completed in 12 months followed by commissioning. This schedule is based on Lycopodium's understanding of the project scope, current lead times for the delivery of critical equipment, and typical duration of engineering and site activities based on similar size projects executed by Lycopodium. The major project milestones are summarized in Table 24-3 below.

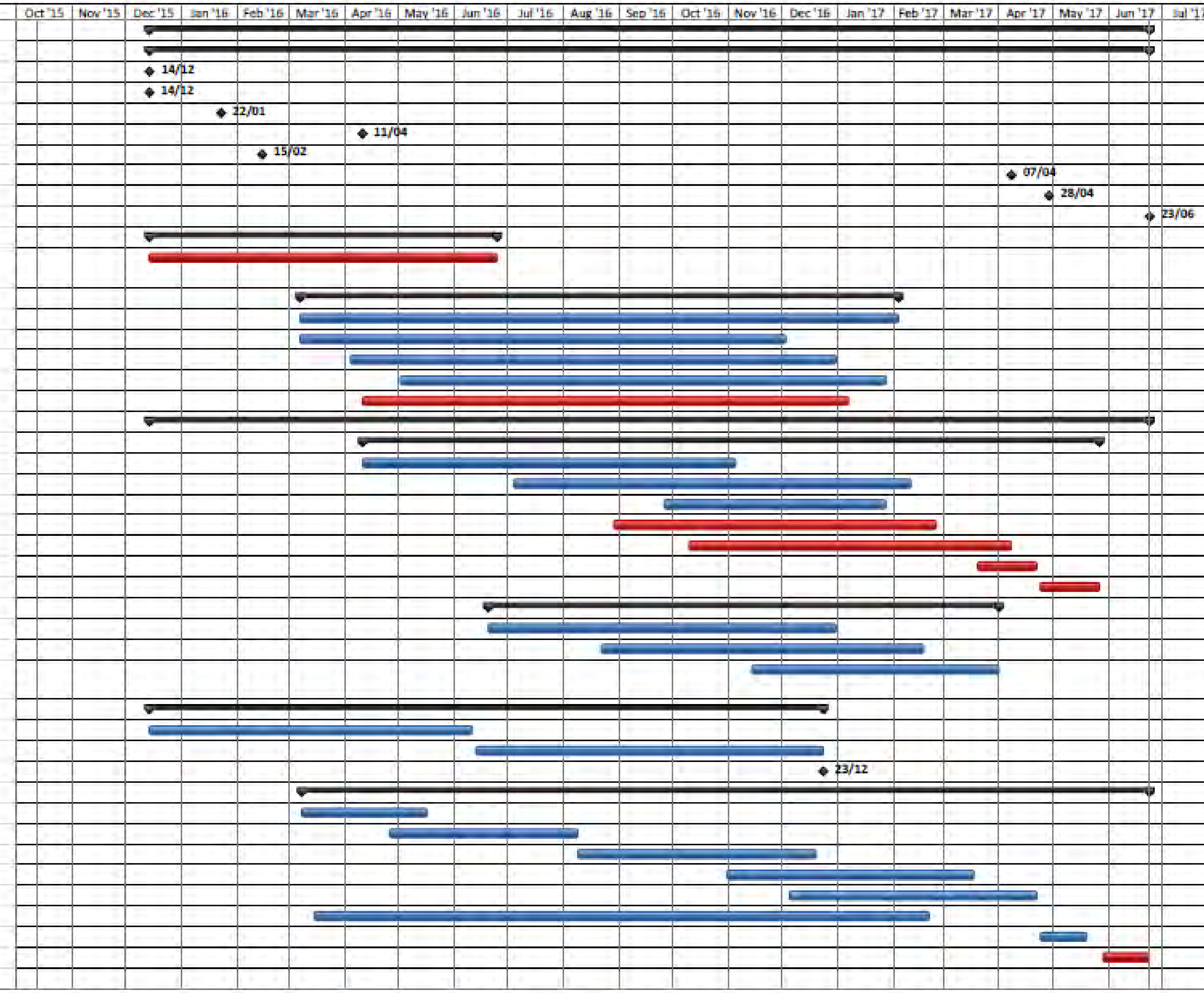
Table 24-3 Major Project Milestones

Major Milestone	Month
Start of Detailed Engineering	Month 1
Award Bulk Earthworks Contract	Month 4
Start Construction (Bulk Earthworks)	Month 5
Start Concrete Works	Month 8
Start SMP Installation	Month 9
Start Field Erected Tankage	Month 10
Detailed Engineering Complete	Month 11
Start E&I Installation	Month 11
Complete Bulk Earthworks (Farim site)	Month 12
Field Erected Tankage Complete (Farim site)	Month 14
Concrete Works Complete (Farim site)	Month 15
SMP Installation Complete (Farim and Port sites)	Month 16
E&I Installation Complete (Farim and Port sites)	Month 17
Start Commissioning	Month 17
Commissioning Complete	Month 19

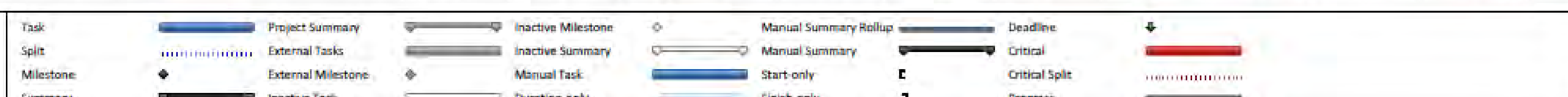
Figure 2 Implementation Schedule

FARIM PHOSPHATE PROJECT - IMPLEMENTATION SCHEDULE

ID	Task Name	Duration	Start	Finish
1	**FARIM PHOSPHATE PROJECT**	**400 days**	**Mon 14/12/15**	**Fri 23/06/17**
2	**KEY DATES**	**400 days**	**Mon 14/12/15**	**Fri 23/06/17**
3	Start Detailed Engineering	0 days	Mon 14/12/15	Mon 14/12/15
4	Environmental Permit Issued	0 days	Mon 14/12/15	Mon 14/12/15
5	Construction Permit Approval	0 days	Fri 22/01/16	Fri 22/01/16
6	Commence Bulk Earthworks	0 days	Mon 11/04/16	Mon 11/04/16
7	Commence Dewatering	0 days	Mon 15/02/16	Mon 15/02/16
8	Mechanical Completion Farim	0 days	Fri 07/04/17	Fri 07/04/17
9	Mechanical Completion Ponta Chugue	0 days	Fri 28/04/17	Fri 28/04/17
10	Process Plant and Port Operational	0 days	Fri 23/06/17	Fri 23/06/17
11	**Detailed Engineering**	**140 days**	**Mon 14/12/15**	**Fri 24/06/16**
12	Develop Engineering Specifications and Plant Drawings	140 days	Mon 14/12/15	Fri 24/06/16
13	**Procurement**	**240 days**	**Mon 07/03/16**	**Fri 03/02/17**
14	Ship Loader	240 days	Mon 07/03/16	Fri 03/02/17
15	Diesel Power Plant - Farim	195 days	Mon 07/03/16	Fri 02/12/16
16	Diesel Power Plant - Ponta Chugue	195 days	Mon 04/04/16	Fri 30/12/16
17	Rotary Dryer	195 days	Mon 02/05/16	Fri 27/01/17
18	Attriction Scrubber	195 days	Mon 11/04/16	Fri 06/01/17
19	**Construction**	**400 days**	**Mon 14/12/15**	**Fri 23/06/17**
20	**Beneficiation Plant**	**295 days**	**Mon 11/04/16**	**Fri 26/05/17**
21	Bulk Earthworks	150 days	Mon 11/04/16	Fri 04/11/16
22	Concrete	160 days	Mon 04/07/16	Fri 10/02/17
23	Field Erected Tanks	90 days	Mon 26/09/16	Fri 27/01/17
24	Structural, Mechanical, Piping	130 days	Mon 29/08/16	Fri 24/02/17
25	Electrical & Instrumentation	130 days	Mon 10/10/16	Fri 07/04/17
26	Pre-Commissioning	25 days	Mon 20/03/17	Fri 21/04/17
27	Commissioning	25 days	Mon 24/04/17	Fri 26/05/17
28	**Tailings Facility**	**205 days**	**Mon 20/06/16**	**Fri 31/03/17**
29	Bulk Earthworks	140 days	Mon 20/06/16	Fri 30/12/16
30	Starter Dam construction	130 days	Mon 22/08/16	Fri 17/02/17
31	Tailings and Decant pipelines (5 km long)	100 days	Mon 14/11/16	Fri 31/03/17
32	**Mining**	**270 days**	**Mon 14/12/15**	**Fri 23/12/16**
33	Dewatering	130 days	Mon 14/12/15	Fri 10/06/16
34	Prestrip	140 days	Mon 13/06/16	Fri 23/12/16
35	Commence FPA Matrix Mining	0 days	Fri 23/12/16	Fri 23/12/16
36	**Port Facilities**	**339 days**	**Tue 08/03/16**	**Fri 23/06/17**
37	Bulk Earthworks	50 days	Tue 08/03/16	Mon 16/05/16
38	Concrete	75 days	Tue 26/04/16	Mon 08/08/16
39	Buildings	95 days	Tue 09/08/16	Mon 19/12/16
40	Structural, Mechanical, Piping	100 days	Mon 31/10/16	Fri 17/03/17
41	Electrical & Instrumentation	100 days	Mon 05/12/16	Fri 21/04/17
42	Marine Structures	245 days	Tue 15/03/16	Mon 20/02/17
43	Pre-Commissioning	20 days	Mon 24/04/17	Fri 19/05/17
44	Commissioning	20 days	Mon 29/05/17	Fri 23/06/17
45				



Farim Phosphate Project



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The major long lead delivery items have been considered in the schedule, which are:

- Shiploader - 11 Months ARO (After Receipt of Order);
- Diesel Power Plants – 9 Months ARO;
- Rotary Dryer – 9 Months ARO;
- Attrition Scrubber – 9 Months ARO.

25.0 INTERPRETATION AND CONCLUSIONS

The following conclusions arise from the information provided in the previous sections:

25.1 Capital & Operating Costs, Economic Analysis

- The scope of design is estimated to require an initial capital investment of USD $193.8M, and sustaining capital of USD $366.6M.
- Life of mine operating costs for the project are estimated to be USD $52.13/t, which falls into the lowest quartile of the phosphate rock industry business cost curve (source: CRU Group)
- Based on a P_2O_5 price of USD $123/t plus a 9.7% premium over the CRU estimate for Morocco K10 FOB price, the after-tax NPV_{10} for the Project is USD $436.9M, while the after tax IRR is 34.5% and the payback period is 4.3 years. The economic analysis demonstrates robust economics and confirms the overall viability of the project. There is consequently justification for advancing to the next phase of detailed engineering.

25.2 Geology and Mining

- The data provided through various exploration and sampling programs combined with the detailed processing analysis, infrastructure and cost analysis is sufficient to support the feasibility level study and associated reserves.
- The reserves outlined in the study are based on a targeted mine life of 25 years. Additional Measured and Indicated resources have been delineated on the property, which have the potential to add substantial additional reserves.
- New drilling has been performed since the development of the current resource model. It is recommended that new drilling on the property be used to update the geologic resource model.

25.3 Metallurgical Testing and Recovery Methods

- The phosphate rock produced is a high grade, high quality, product that will attract a premium price.
- The samples used for this testwork were selected to represent the potential mining areas for the first seven years, ore grade, and mineralization types for the South Pit of the Farim deposit.
- The beneficiation plant is based on bench scale and pilot plant testwork designed for optimum recovery and minimum operating costs. The flowsheet is based upon unit operations that are proven in industry.
- The low CaO/P_2O_5 ratio means that sulphuric acid consumption in the phosphoric acid plant will be 15% lower than for benchmark Moroccan K10 phosphate.

- The low MER in the product means that the on-spec DAP fertilizer can be made with low ammonia consumption.

- The low levels of toxic heavy metals (including Cadmium, Lead, and Arsenic) in the phosphate product will give advantages in markets with strict fertilizer regulations.

- Moderate levels of carbonate and organics will make defoamer consumption low.

- Drill results indicate the presence of some DSO (direct shipping option) ore, which would only need to be dewatered and dried prior to shipping to market.

25.4 Marine

- Foundation design for the marine loading facility and navigation aids was based on geotechnical information from land-based boreholes. As such, the foundation design may require adjustment during final design to accommodate geotechnical conditions actually existing in the Geba River.

- While full bridge navigation simulations have not been conducted to verify the navigability of the Geba River to Ponta Chugue, some confidence in its navigability can be gained as deep draft vessels are currently visiting the Port of Bissau.

- Given the existing meteorological data and assumptions made with respect to bulk carrier arrivals the intended throughput at the vessel loading facility appears feasible.

- Given the existing hydrographic data available for the Geba River, it appears that bulk carriers of approximately 35,000 DWT will have acceptable keel clearance to navigate to Ponta Chugue without reliance on tide. As the existing hydrographic data over large stretches of the river is outdated, additional hydrographic data needs to be gathered during final design. As the riverbed is expected to be dynamic, hydrographic data must be gathered at regular intervals during operation.

- The design of the vessel loading facility and aids to navigation were developed with a goal of minimizing CAPEX, which has impacts on the required sustaining capital for the facility, and requires that stringent safety protocol be followed during vessel loading operations.

- While the marine loading facility was located in an area having suitable natural water depth for the design vessel, water depth may be reduced over time due to siltation.

- While the marine loading facility was oriented to minimize vessel response resulting from tidal currents during vessel shifting procedures, vessel shifts remain risky. Detailed planning by vessel experts and training for vessel operators is required to minimize risk.

25.5 Infrastructure

- The ground conditions at the TSF, processing plant (plant site west) and port site typically comprise overconsolidated clay interbedded with sand horizons and near surface laterite layers in places. These ground conditions are considered suitable for the proposed development.

- The ground conditions at the proposed product bin site (plant site east) are poor and similar to those identified along the southern side of the proposed South Pit. The ground is not considered suitable to support notable structures on spread footings, and therefore piling has been proposed and budgeted.

- Areas within the footprints of both the proposed South Pit and the product bin area appear to be subject to seasonal flooding and earthworks will become considerably more difficult outside of the dry season.

- Groundwater levels are typically close to ground surface and the site's materials are predominantly cohesive. The time taken for consolidation settlements to occur within the saturated soils is likely to be a number of years and extend well into the operational phase of the mining operation. Infrastructure designs should consider the potential impact of long-term settlement and the impact this may have on the amount and frequency of maintenance.

- Sources of construction materials, particularly drainage medium, concrete aggregate and select rock fill need to be confirmed.

25.6 Tailings, Waste Facilities, Hydrogeology, and Water Management

- A limited tailings testing program has been carried out to date. Consequently, the tailings physical behavior characteristics need further definition. This has implications for the TSF staging and water balance. Testing of a larger representative tailings sample at the nominated design tailings percent solids will be required to confirm the tailings properties for final design purposes. More definitive tailings testing should provide for optimization of the TSF staging and may provide embankment cost savings.

- It may be feasible to optimize the TSF embankment raise construction schedule by providing for increased inter-stage capacity, thereby reducing the total number of raises and hence the overall earthworks contractor mobilization costs.

- Waste and tailings volumes were significantly reduced in the later stages of the FS. Initial review indicates there is an opportunity to move the TSF and IWL to a more compact site, with the potential to simplify the surface water management aspects and realize significant capital and operating cost savings.

- The geochemistry assessment of the supernatant is based on preparation of tailings slurry from a dry sample using tap water. Consequently, the supernatant geochemistry results many not be accurate. The geochemistry of a representative sample of the tailings slurry requires further review, which may impact the design requirements for the TSF lining and capping system.

- The waste rock geochemical assessment to date is based on the testing of 20 composite samples. Additional geochemical assessments are in progress. Consequently, the relative quantities of mine waste by lithology and the geochemical risk associated with each lithology are not fully defined. Better definition of waste rock geochemistry will provide for optimization of the waste dump designs and of the impacted water management requirements.

- Whilst the understanding of the site hydrogeology has improved significantly, further work is required to provide additional data for modeling of mine inflows and drawdown impact predictions of the life of mine.

25.7 Environmental, Permitting, Social and Community Impact

- The political, location, environmental, social and permitting risks appear to be generally commensurate with other mining projects in West Africa. EIS work completed to date has not resulted in the identification of any fatal flaws or impacts that are expected to be of critical significance with mitigation measures applied.

- The Project is subject to a signed Mining Agreement, a mining lease (granted) and a mining licence (granted). The Project is also subject to an environmental review by the Government of Guinea-Bissau (GoGB). Successful completion of the Incentive Annex and the ESIA review both represent permitting risks that are judged to be low based on the priority the GoGB appears to place on seeing the Project be developed.

- An ESIA for the mine site area only was completed in December 2014, and a project-wide ESIA is near completion based on the project design presented in this Technical Report. The ESIA is being drafted to be compliant with the World Bank Equator Principles III (Equator Principles Association, 2013) and the IFC Performance Standards on Environmental and Social Sustainability (IFC, 2012).

- An Environmental and Social Management Plan (ESMP) is under development as part of the ESIA. A number of discipline-specific management plans have been developed to a conceptual level to address the impacts identified above as well as common environmental issues not listed above. Several additional plans have been identified but not developed.

- Key impacts of the Project include:

  - Air quality impacts – Modelling using worst-case assumptions suggests that adverse air quality will occur at several off-site receptors; additional mitigation measures will be required to moderate these impacts.

  - Impacts to Groundwater Levels Affecting Other Users - Mine dewatering is predicted to affect local village wells in the surrounding area. A monitoring program will be implemented and the project will be required to supply replacement water to affected users.

  - Water Quality Impacts - Limited geochemical evaluation of tailings and waste overburden suggests both materials are non-acid forming, though the tailings and a portion of the waste overburden have the potential to leach metals. Containment of the leachable materials will be necessary to prevent seepage of adverse quality effluent to groundwater and to prevent discharge to surface waters. Additional geochemical evaluation of both tailings and waste overburden will be completed shortly.

- Radiological Aspects - Uranium concentrations in the ore are elevated, as with many phosphate mines around the world. A preliminary radiological assessment indicates that the phosphate ore and tailings contain some measure of radioactivity that will require management. Preliminary estimates of exposure doses suggest that there is likely limited potential risk to workers and the public. Further radiological work is currently underway to further define this potential risk, and to identify the need for special occupational health and safety measures or design mitigation required (if any) to ensure that workers, the public and the environment are protected. The TSF closure cap has been designed to provide adequate shielding of low level radioactivity to ensure public safety post-closure.

- Loss of Critical Habitat – Development of the south pit will result in the loss of mangroves which provides important habitat for the Nile crocodile. A Biodiversity Management Plan developed as part of the ESIA has identified proposed biodiversity offsets to compensate for these losses.

- Physical and Economic Displacement Requiring Resettlement - Development of the Project will result in the physical displacement of an approximate 175 households in the Mine Site area. The host community for resettlement has not yet been identified. The successful development and implementation of a detailed Resettlement Action Plan (RAP) is required to address this key social risk of the Project. The time required to implement the RAP also has the potential to impact the development schedule.

- Impacts to Tangible and Intangible Cultural Heritage Features of High Significance – The Project will also displace two cemeteries, one of which is of high significance, a sacred forest, and a mosque. To mitigate the loss of cultural heritage, it will be necessary to develop specific mitigation plans compliant with IFC Performance Standard 8 (Cultural Heritage) in consultation with local communities and government, to relocate the cemeteries and mosque and compensate for the loss of the sacred forest.

- Traffic Safety – There will be potential risks to community safety resulting from project related truck traffic (shipping of product and the delivery of equipment and materials) given the amount of pedestrian road use and lack of familiarity with higher levels of traffic and heavy goods vehicles. Project traffic will need to be carefully managed.

- Uptake of Employment and Business Opportunities - The population of Guinea-Bissau has low levels of education and limited industrial experience with no previous mining experience. Therefore, focused effort on skills development and health and safety training will be necessary to establish the Project’s local workforce, and to keep the expatriate workforce to a minimum.

- Other Social Issues – Other common socioeconomic issues, such as influx in into the Farim area, will require management in consultation with local authorities.

- Significant Socioeconomic Benefit - The Project can be expected to provide significant benefits to the local area and Guinea-Bissau as a whole.

26.0 RECOMMENDATIONS

Recommendations for future work are as listed below.

26.1 Capital & Operating Costs, Economic Analysis

- The FS for the Farim Phosphate Project has been completed in sufficient detail to refine the economics to a +/-15% level of accuracy and outline the issues facing the project going forward. The project economics are sufficiently robust to warrant moving to the next phase of detailed engineering and construction.

26.2 Geology and Mining

- The results from the 10 recently completed beneficiation and metallurgical drill holes should be used to update the geologic resource model once the data and observations are available for these drill holes.

- Further investigation into the bearing capacity and wear characteristics of the material on site and proposed road construction methods to ensure adequate “trafficability” particularly in the rainy season.

- Consideration should be made to develop an onsite quarry to reduce the cost of road material.

- In the event that DSO ore may be attainable, consideration should be made for further infill drilling in the southern pit to better define the extent and quantity of DSO ore with resulting revised resource model and mine plan to determine quantities that may be shipped while maintaining acceptable grades for the plant feed.

26.3 Metallurgical Testing and Recovery Methods

- Determine the rheological characteristics of the products and tailings to determine the slurry and pumping characteristics, static and dynamic settling, and filtration characteristics.

- Evaluate the settling and filtration parameters in the presence of coagulants and/or flocculants for the design of the thickeners and filtration devices.

- Perform variability bench scale tests for different areas of the South Pit and of the North Pit of the deposit applying the beneficiation technology developed.

- Carry out extensive pilot plant tests for each the North and South Pit phosphate ore to obtain enough information on material balances, operating conditions, variability effects, products and their marketing, and to evaluate the use of column flotation cells for 0.106x0.020 mm size fraction when high iron bearing minerals are present.

- Implement a metallurgy testwork program to include:

  - Vacuum belt filter dewatering;

- Bulk material handling flowability tests for product bin design;
- Drying optimisation tests.

- Conduct continuous phosphoric acid plant tests to assess likely performance in an industrial plant. Conduct bench-scale phosphoric acid concentrations and clarification tests, and bench-scale fertilizer test work.

26.4 Marine

- Conduct initial geotechnical investigations for the bulk carrier loading facility at Ponta Chugue, aids to navigation in the Geba River, and the Cacheu River crossing structures at Farim
- Develop a marine operational readiness plan that details necessary training for vessel operators, logistics channels for sourcing spare parts, International Ship and Port Facility Security Code (ISPS) requirements, safety procedures, equipment and personnel required to maintain the marine facility.
- Conduct an analysis to estimate scour around the piles supporting the Cacheu River crossing structure.
- Conduct an investigation into the potential for sedimentation of the bulk carrier berth at Ponta Chugue.
- Conduct desktop and full bridge navigation simulations to better understand the navigability of the Geba River, the recommended vessel berthing procedures, and propulsion requirements of the assisting tugs.
- Gather additional hydrographic data between Banco do Alenquer and Ponta de Caio to validate the allowable vessel draft recommendation.
- Gather additional meteorological and oceanographic data to validate the hydrodynamic model (tides and currents) and better predict downtime associated with rainfall.
- Further investigate the likely variability associated with bulk carrier arrivals and the necessity to suspend bulk carrier loading during rain events, as these items have a substantial effect on the required stockpile size.

26.5 Infrastructure

- Conduct further geotechnical investigations for all surface infrastructure, including the beneficiation plant site, and Ponta Chugue port facilities.

26.6 Tailings, Waste Facilities, Hydrogeology, and Water Management

- Complete physical, geochemical and radiological testing programs on a representative sample of tailings in order to confirm the tailings characteristics for design.

- Complete a geochemical testing program on samples of specific geological lithologies in order to de-lineate and quantify mine waste in terms of material type and geochemical risk.

- Provide an updated orebody geological model to allow a refinement of the hydrogeological model domain. In this regard, the development of a block geological model, to be used as a basis for the hydrogeological and other mining design and development purposes is essential.

- Completion of additional pumping tests in the southern pit to improve understanding of the groundwater flow regime, the hydraulic connection with nearby creeks and surface water bodies, water impacts associated with mining the pit and the variation of aquifer hydraulic properties over the area of interest.

- A Groundwater Management Plan is required to address the drawdown impacts on local water supplies. This would typically include:

  – The establishment of a groundwater monitoring network and a mitigation plan to ensure that water availability is maintained.

  – Updating the hydrocensus and surveying nearby bores to determine use, depth, water level elevation.

  – The occurrence of at least two different water types, i.e. fresh groundwater and surface brackish river and creek water, support the need for a hydrogeochemical survey to understand baseline groundwater quality, the role of hydrogeochemical processes in the system and the degree of interaction between the brackish surface water bodies and groundwater both at current conditions and during dewatering.

- Conduct quarterly groundwater and surface water monitoring in the mine site study area over the next two years (twice during each of the dry and wet seasons), to establish a more robust baseline. Groundwater monitoring should include groundwater levels and quality, and should include sampling of representative village wells.

- Incorporate new pumping tests and hydrocensus data into the groundwater model to provide additional calibration and refinement.

- The long term response of the aquifer to pumping, especially at the southern pit, should be tested by additional pumping tests.

- Investigate alteration of the mine plan to maximise dewatering efficiencies.

26.7 Environmental, Permitting, Social and Community Impact

- Submit the ESIA (Environmental Plan under the Mining Agreement) and the Mining Operations Plan to the government of Guinea-Bissau (GoGB), and complete negotiation of the Incentive Annex with the GoGB, as soon as possible as intended.

- Initiate development of the Resettlement Action Plan and cultural heritage mitigation plans as soon as possible, to minimize the potential for this aspect to affect the development schedule.

- Complete the geochemical and radiological testing currently underway

- Complete the planned supplemental wet season biodiversity field program to identify any flowering plants or other species of conservation concern, and update the Biodiversity Management Plan accordingly.

- Stakeholder Engagement – The Project has conducted several rounds of public consultation meetings over the last several years, but intensification of this consultation in the near term will be necessary both before and after distribution of the ESIA. Key aspects of the Project requiring consultation to meet the requirements outlined in the IFC performance standards include: resettlement, cultural heritage, biodiversity management and offsets, radiological risk (or lack thereof), traffic safety and other project impacts.

- The ESMP including the discipline-specific management plans (some of which have not yet been developed) require further updating or preparation prior to construction, based on the outcome of further stakeholder engagement activities and detailed engineering design of the Project.

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APPENDIX A – PROCESS FLOW DIAGRAMS

CLIENT GB MINERALS LTD.

PROJECT FARIM PHOSPHATE PROJECT

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DRAWING TITLE
OVERALL PROCESS FLOW
PROCESS FLOW DIAGRAM

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NTS	5036	000-F-000	C

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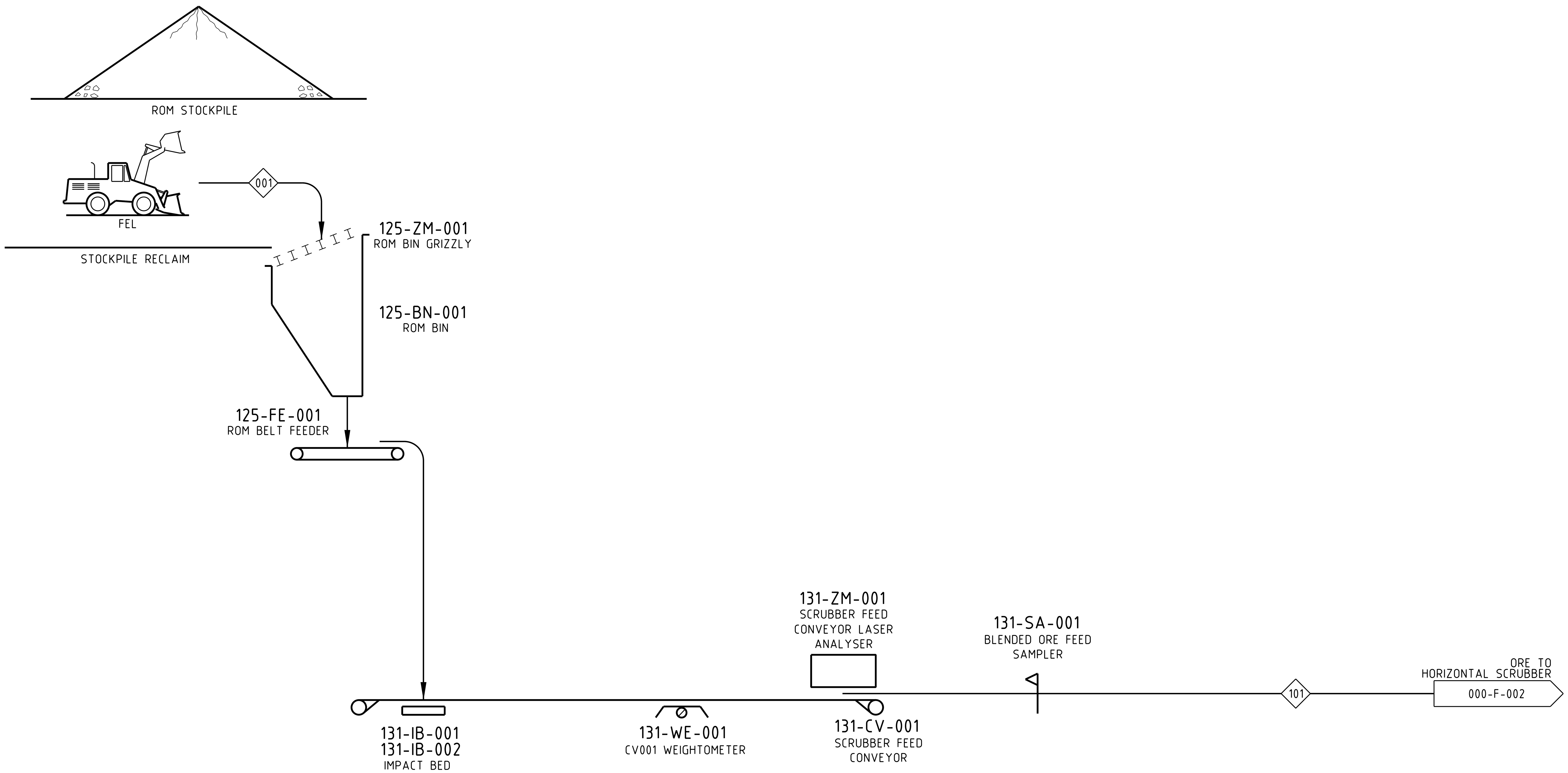

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		C	20JAN15	ISSUED FOR STUDY	RAM						
		B	20NOV14	ISSUED FOR CLIENT REVIEW-WITH PROCESS UPDATES	AG						
		A	22OCT14	ISSUED FOR CLIENT REVIEW	RAM						

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DRAWING TITLE
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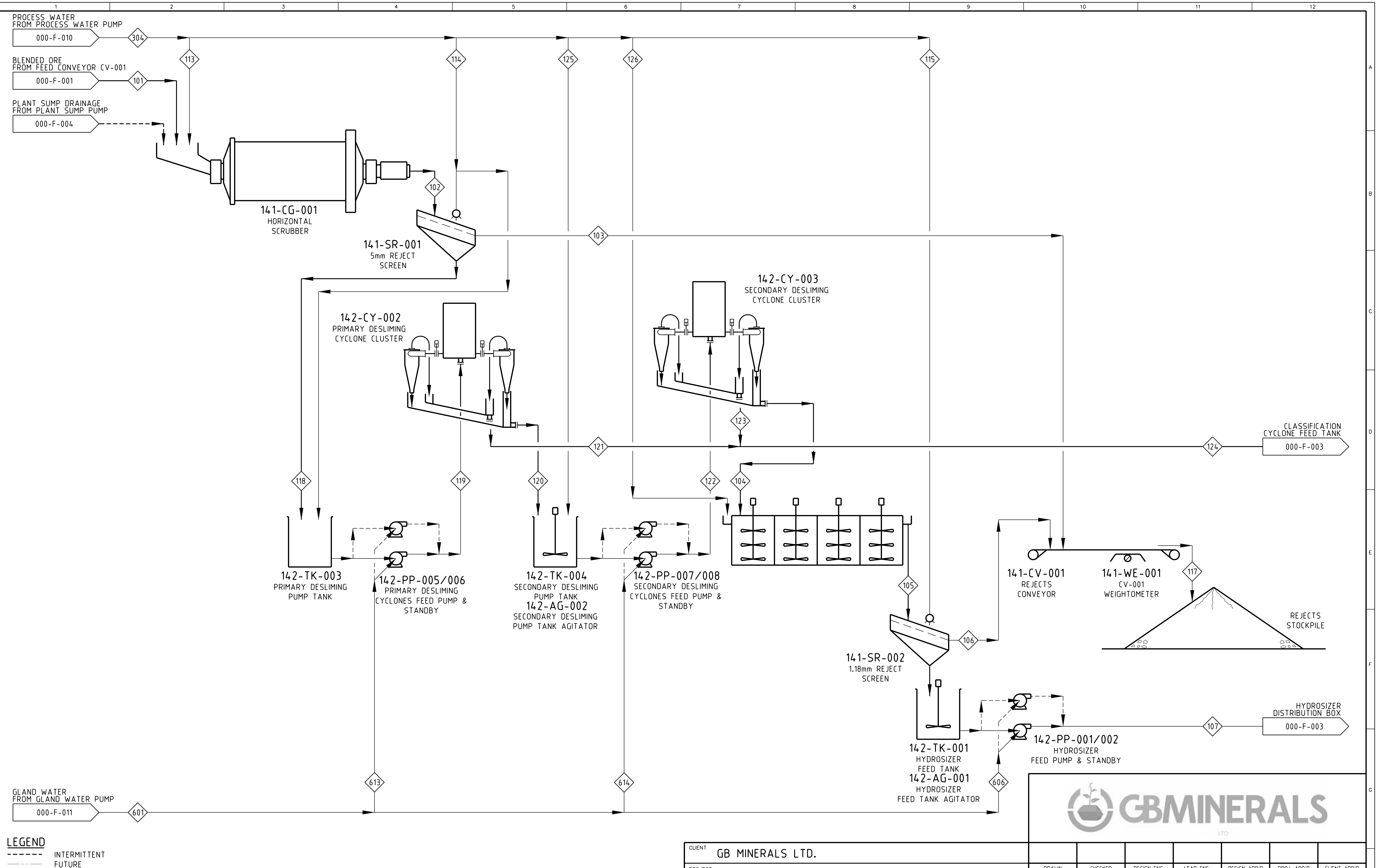

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SCRUBBING & CLASSIFICATION
(SHEET 1 OF 2)
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PROJECT: FARIM PHOSPHATE PROJECT

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DRG No	REFERENCE DRAWINGS	REV	DATE	DESCRIPTION	DRN	CHK'D	DESIGN ENG.	LEAD ENG.	DESIGN APP'D	PROJ. APP'D	CLIENT APP'D
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DRAWING TITLE
CLASSIFICATION
(SHEET 2 OF 2)
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
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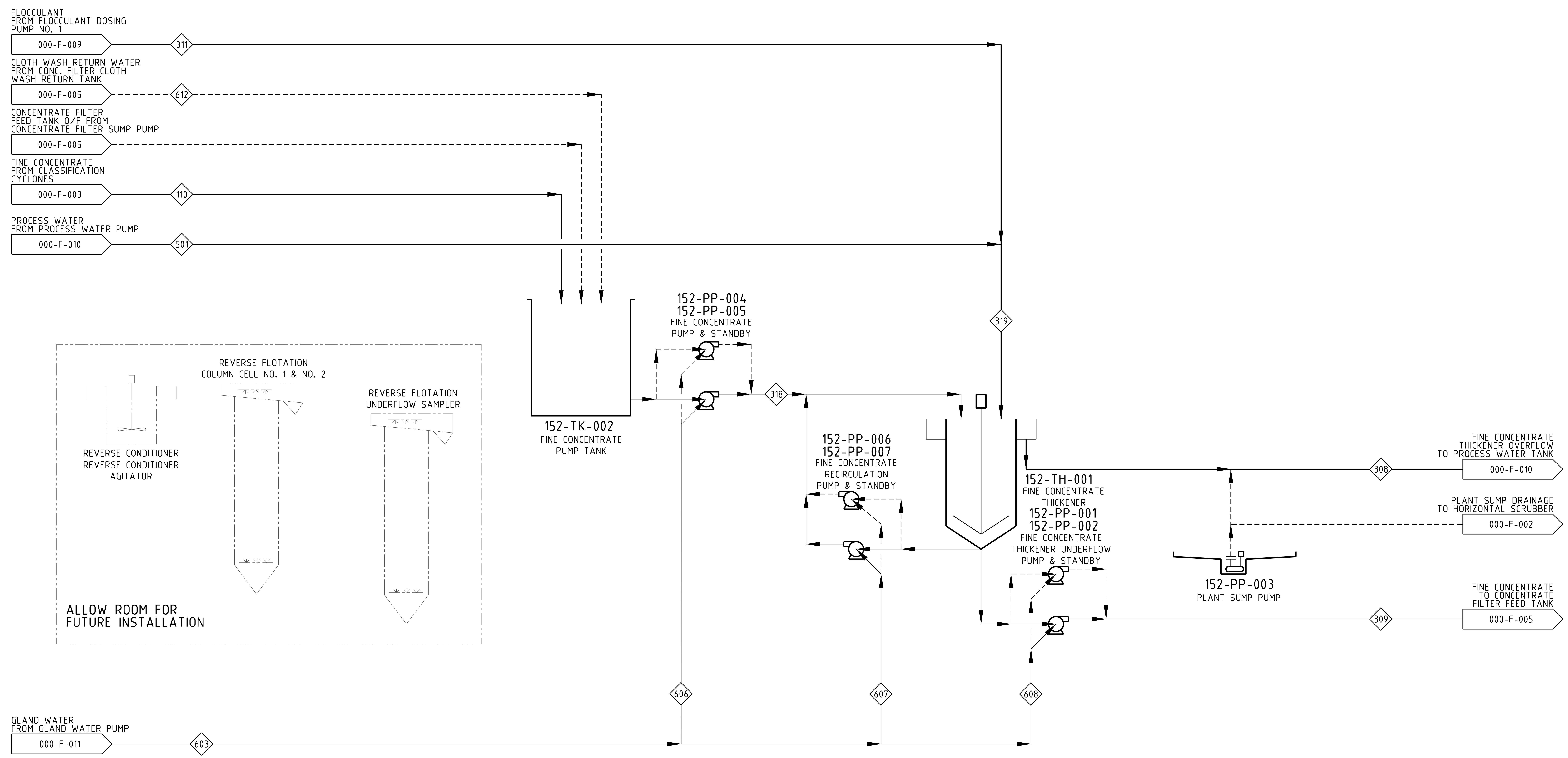
DRAWN: CB DATE: 08OCT14



LEGEND

INTERMITTENT
FUTURE

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		C	20JAN15	ISSUED FOR STUDY	RAM						
		B	20NOV14	ISSUED FOR CLIENT REVIEW-WITH PROCESS UPDATES	AG						
		A	22OCT14	ISSUED FOR CLIENT REVIEW	RAM						

CLIENT GB MINERALS LTD.

PROJECT FARIM PHOSPHATE PROJECT

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DRAWING TITLE
FINE CONCENTRATE THICKENING
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
NTS	5036	000-F-004	D
DRAWN CB / DATE 08OCT14			

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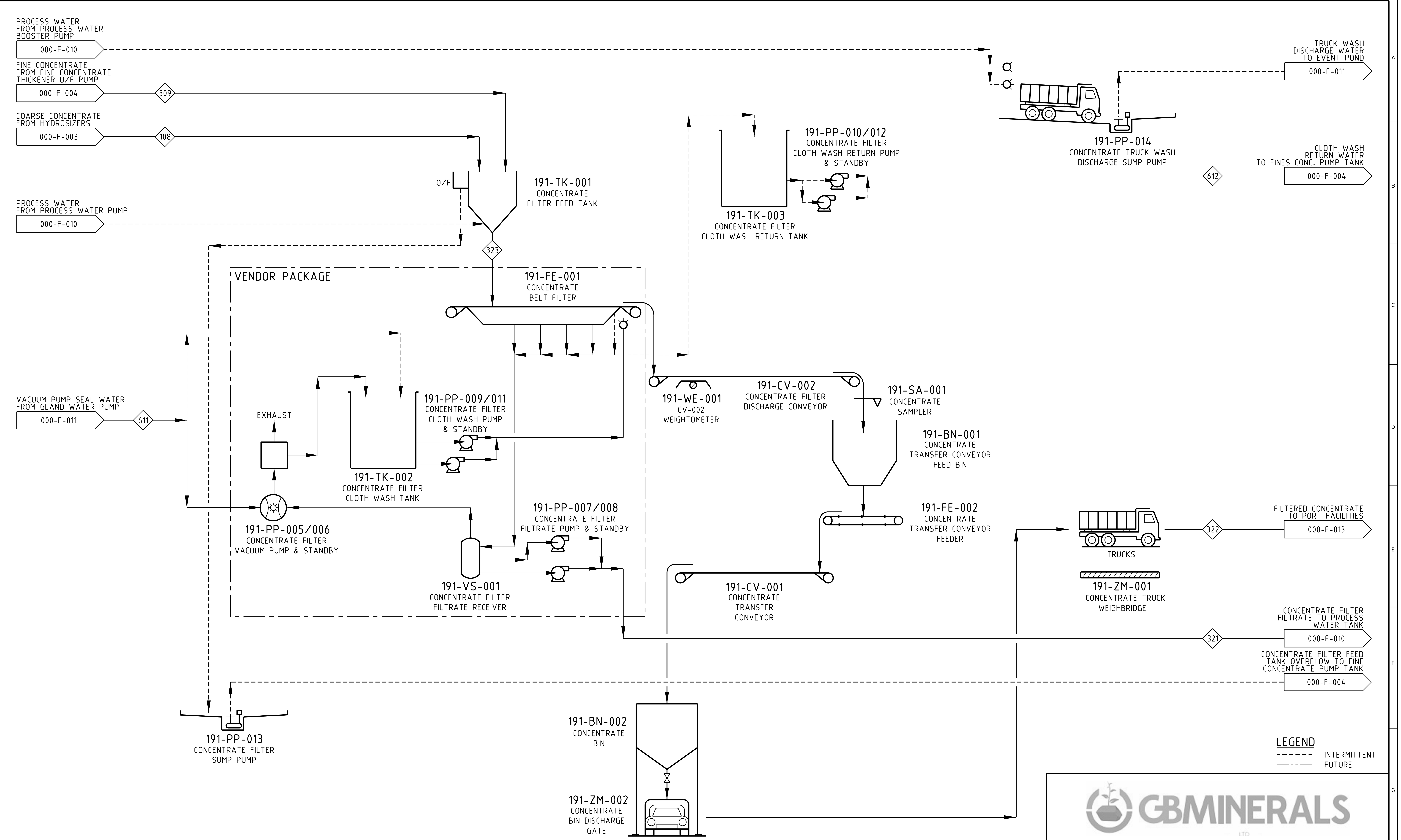

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D	24JUL15	UPDATED WITH PROCESS REVISION	CB			SC	IM	DM	
C	20JAN15	ISSUED FOR STUDY	RAM						
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DRG No	REFERENCE DRAWINGS

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DRAWING TITLE
CONCENTRATE FILTRATION & STORAGE
PROCESS FLOW DIAGRAM

SCALE NTS | JOB No. 5036 | DRG No. 000-F-005 | REV. D

DRAWN CB | DATE 08OCT14

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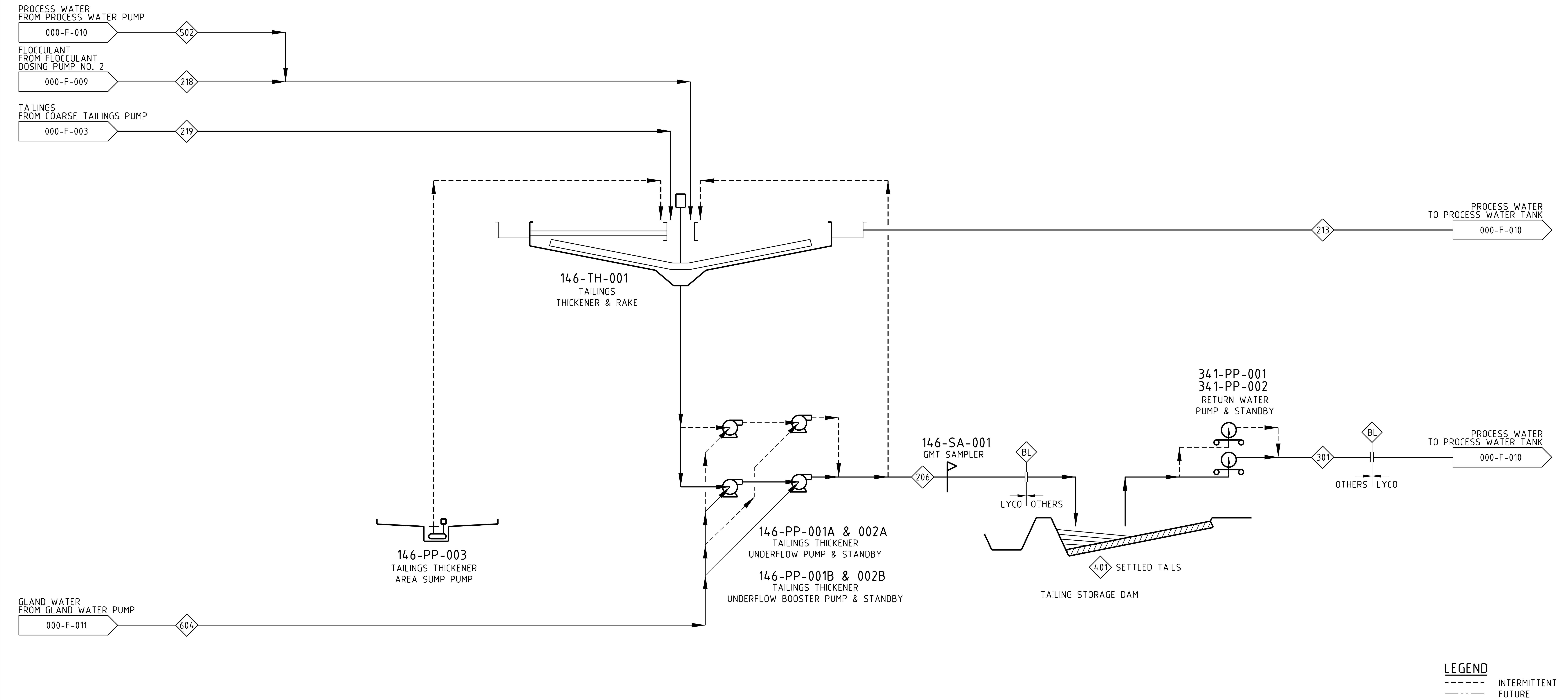

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DRAWING TITLE
TAILINGS HANDLING
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
NTS	5036	000-F-006	D

DRAWN CB | DATE 08OCT14

DRG No	REFERENCE DRAWINGS	REV	DATE	DESCRIPTION	DRN	CHK'D	DESIGN ENG.	LEAD ENG.	DESIGN APP'D	PROJ. APP'D	CLIENT APP'D
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		C	20JAN15	ISSUED FOR STUDY	RAM						
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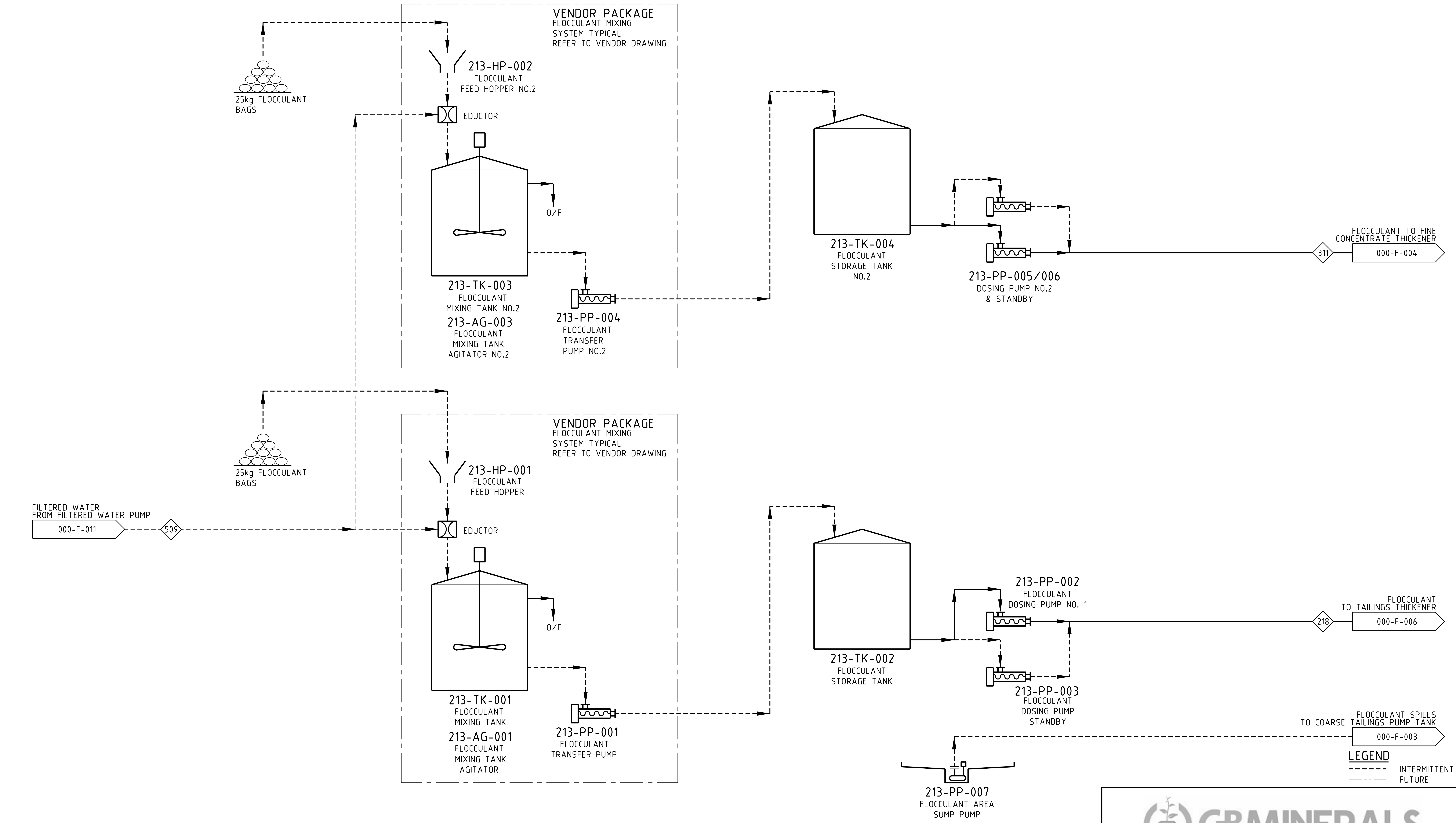

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		B	20NOV14	ISSUED FOR CLIENT REVIEW-WITH PROCESS UPDATES	AG						
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DRAWING TITLE
FLOCCULANT PREPARATION
PROCESS FLOW DIAGRAM

SCALE		JOB No.	DRG No.	REV.
NTS		5036	000-F-009	D
DRAWN: RRM	DATE: 08OCT14			

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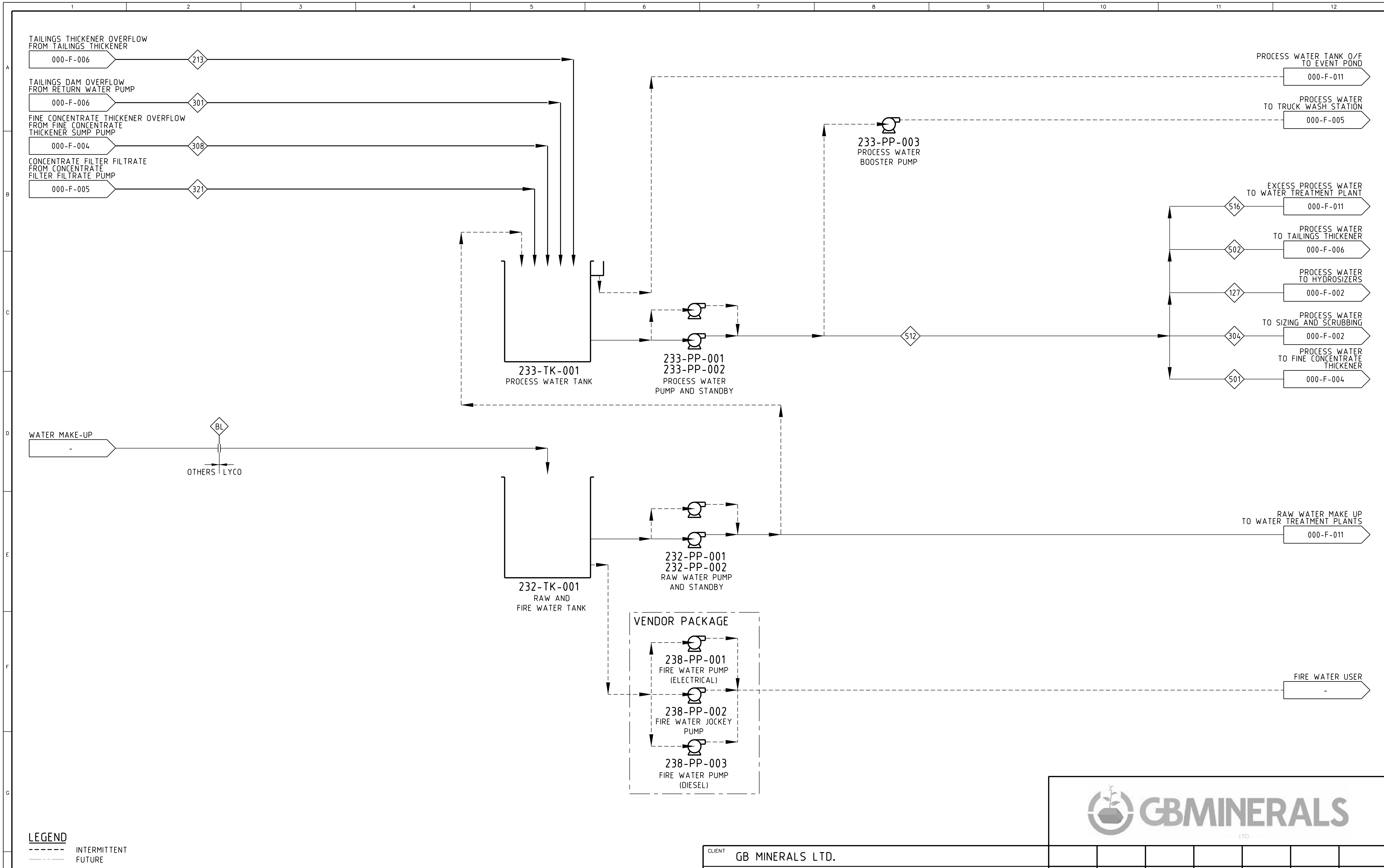

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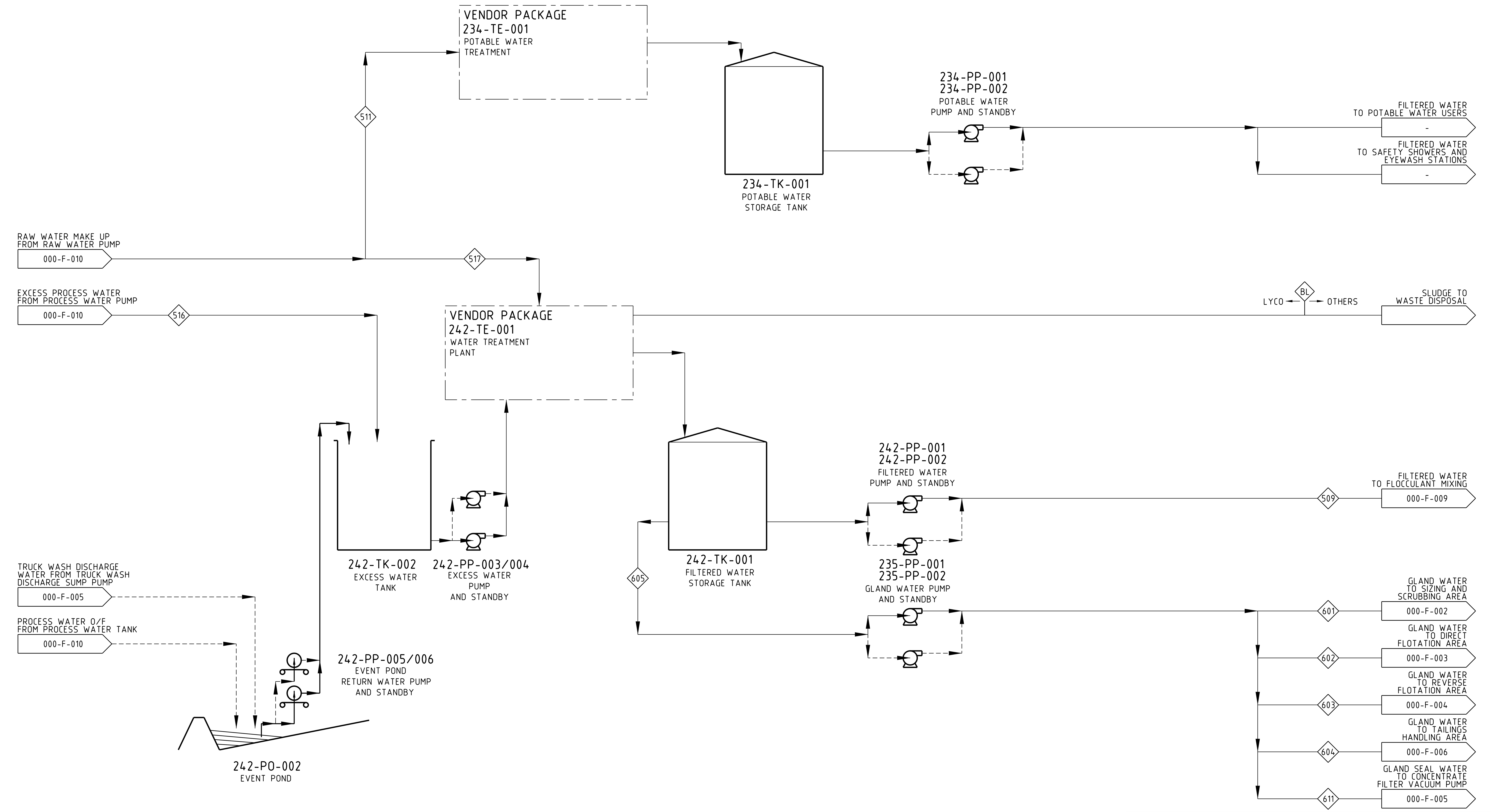
DRAWING TITLE
RAW, PROCESS AND FIRE WATER SERVICES
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
NTS	5036	000-F-010	D

DRAWN RRM DATE 08OCT14



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D	24JUL15	UPDATED WITH PROCESS REVISION	CB			SC	IM	DM	
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A	22OCT14	ISSUED FOR CLIENT REVIEW	RAM						

DRG No	REFERENCE DRAWINGS

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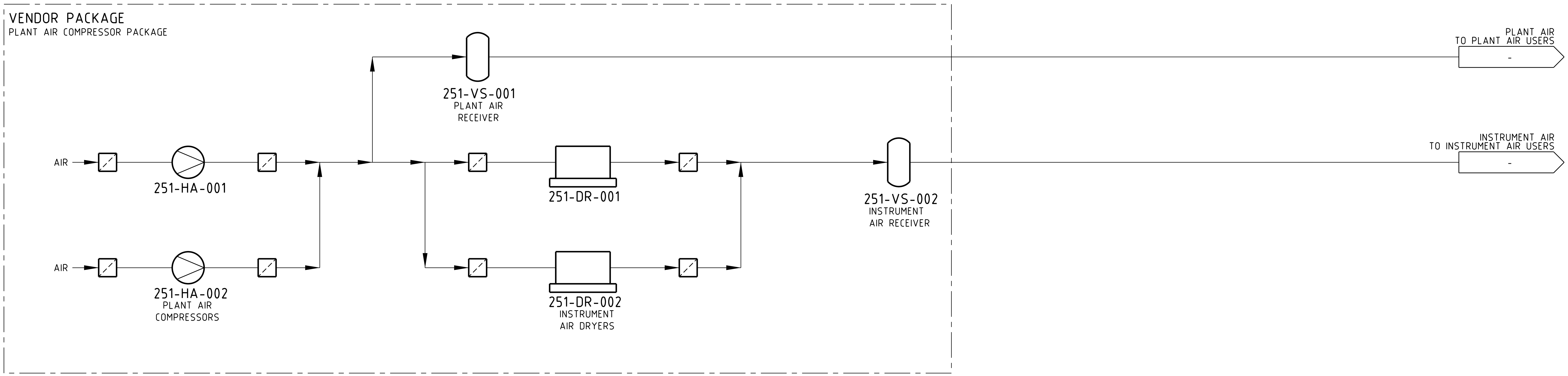
DRAWING TITLE
TREATED WATER SERVICES
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
NTS	5036	000-F-011	D

DRAWN	DATE
RRM	08OCT14



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REV	DATE	DESCRIPTION	DRN	CHK'D	DESIGN ENG.	LEAD ENG.	DESIGN APP'D	PROJ. APP'D	CLIENT APP'D
C	24JUL15	UPDATED WITH PROCESS REVISION	CB			SC	IM	DM	
B	20JAN15	ISSUED FOR STUDY	RAM						
A	20NOV14	ISSUED FOR CLIENT REVIEW	AG						

DRG No	REFERENCE DRAWINGS

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DRAWING TITLE

AIR SERVICES
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
NTS	5036	000-F-012	C

DRAWN	DATE
RRM	07OCT14

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DRAWING TITLE
PORT CONCENTRATE UNLOADING, DRYING & STORAGE
PROCESS FLOW DIAGRAM

SCALE	DRAWN	DATE	JOB No.	DRG No.	REV.
NTS	CB	27OCT14	5036	000-F-013	C

DRG No	REFERENCE DRAWINGS	REV	DATE	DESCRIPTION	DRN	CHK'D	DESIGN ENG.	LEAD ENG.	DESIGN APP'D	PROJ. APP'D	CLIENT APP'D
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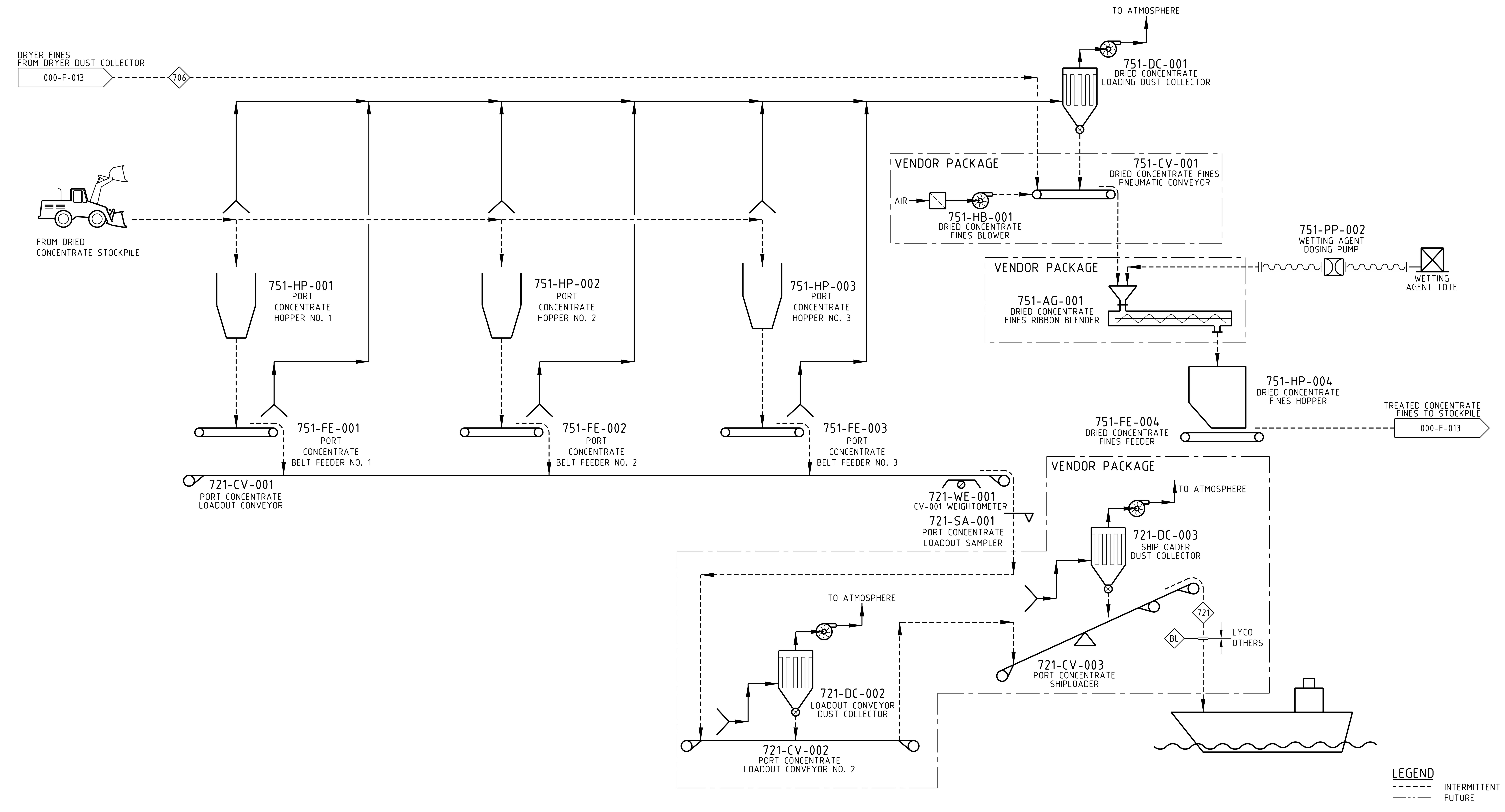

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DRG No	REFERENCE DRAWINGS

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B	20JAN15	ISSUED FOR STUDY	RAM						
A	20NOV14	ISSUED FOR CLIENT REVIEW	AG						

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DRAWN	CHECKED	DESIGN ENG.	LEAD ENG.	DESIGN APP'D	PROJ. APP'D	CLIENT APP'D

DRAWING TITLE
PORT CONCENTRATE LOADOUT
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
NTS	5036	000-F-014	C
DRAWN CB	DATE 27OCT14		

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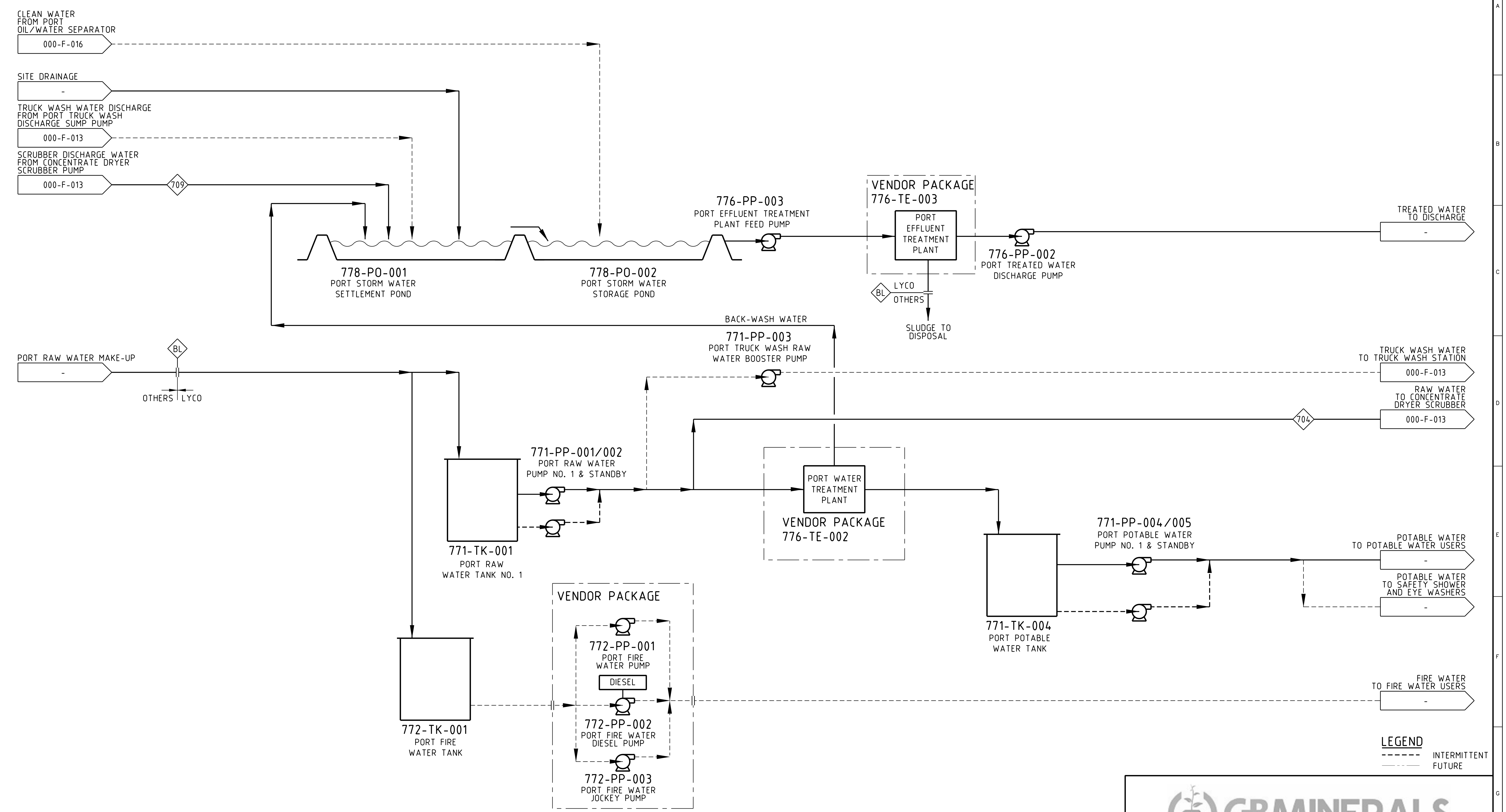

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		A	20NOV14	ISSUED FOR CLIENT REVIEW	AG						

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DRAWING TITLE
PORT SERVICES
PROCESS FLOW DIAGRAM

SCALE NTS | JOB No. 5036 | DRG No. 000-F-015 | REV. C
DRAWN CB | DATE 27OCT14

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		A	20NOV14	ISSUED FOR CLIENT REVIEW	MC						

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DRAWING TITLE
PORT SERVICES AND DIESEL
PROCESS FLOW DIAGRAM

SCALE	DRAWN	DATE	JOB No.	DRG No.	REV.
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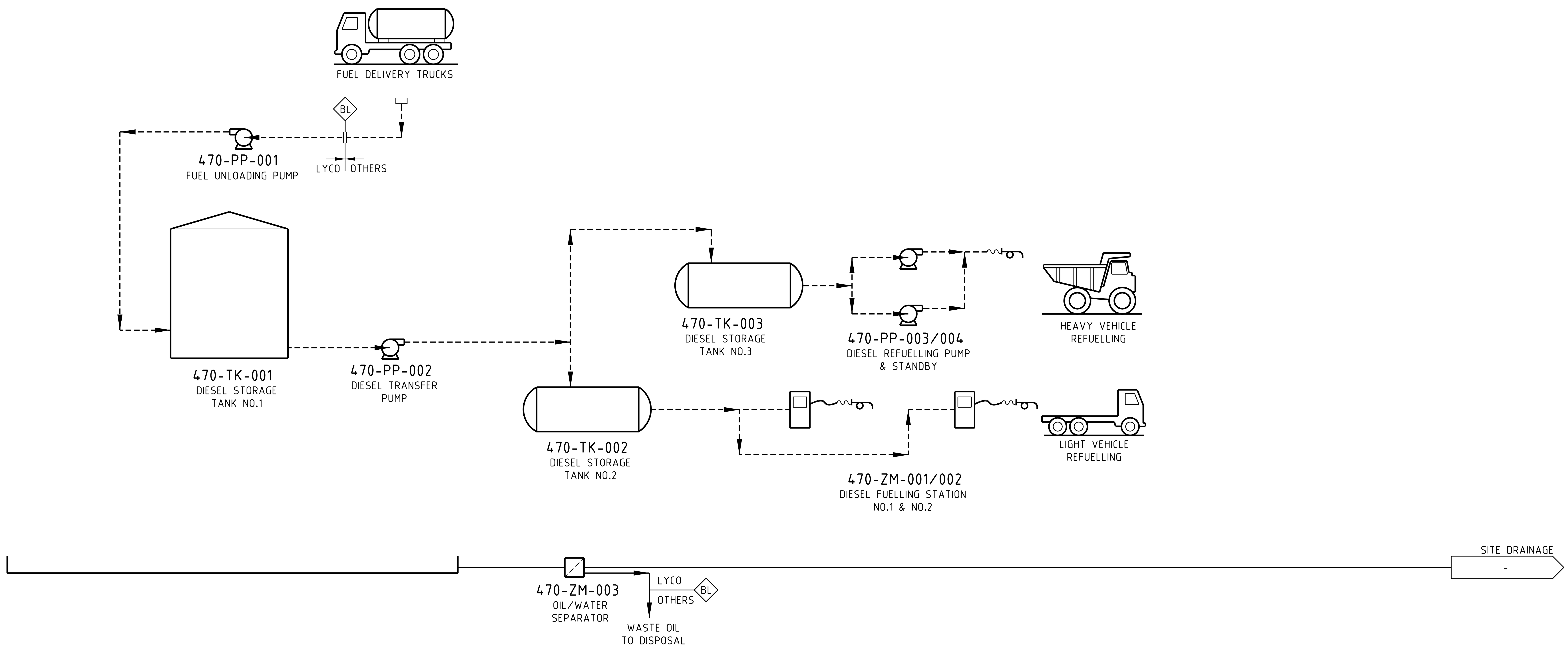

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DRAWING TITLE
MINE SERVICES
BULK FUEL STORAGE FACILITY
PROCESS FLOW DIAGRAM

SCALE	JOB No.	DRG No.	REV.
NTS	5036	000-F-017	D

DRAWN AG DATE 20NOV14

DRG No	REFERENCE DRAWINGS	REV	DATE	DESCRIPTION	DRN	CHK'D	DESIGN ENG.	LEAD ENG.	DESIGN APP'D	PROJ. APP'D	CLIENT APP'D
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		B	20JAN15	ISSUED FOR STUDY	RAM						
		A	20NOV14	ISSUED FOR CLIENT REVIEW	AG						

APPENDIX B – METALLURGICAL TESTWORK DATA

Appendix B – Metallurgical Test Work Data

Appendix A: Core Sample Reception, Preparation, and Characterization

ECHANTILLON COMPOSITE A PREPARER

ET ENVOYER 15/12/2014

SB 9					SC 10					SC 11					SE10				
Section	**kg**	**Percent of Hole**	**Hole Composite, g**	**Reserve, g**	**Section**	**kg**	**Percent of Hole Total**	**Hole Composite, g**	**Reserve, g**	**Section**	**kg**	**Percent of Hole Total**	**Hole Composite, g**	**Reserve, g**	**Section**	**kg**	**Percent of Hole Total**	**Hole Composite, g**	**Reserve, g**
32,15-32,35	3.0	8.33%	1500	1287	32,24-32,56	3.8	10.58%	1900	1809	30,47-30,82	0.42	9.38%	210	**202**	30,63-31,11	3.0	16.53%	1500	1360
32,35-32,65	4.2	11.67%	2100	2006	32,56-32,86	2.7	7.52%	1350	1406	30,82-31,17	0.47	10.49%	235	235	31,11-31,41	3.0	16.53%	1500	1535
32,86-33,08	3.3	9.17%	1650	1653	32,86-33,26	3.8	10.58%	1900	1936	31,17-31,52	0.47	10.49%	235	252	31,42-31,87	2.3	12.67%	1150	1180
33,08-33,46	5.4	15.00%	2700	2546	33,26-33,51	2.9	8.08%	1450	1436	31,52-31,64	0.16	3.57%	80	82	31,87-32,20	2.2	11.85%	1075	1106
33,46-33,79	4.8	13.33%	2400	2394	33,51-33,73	2.7	7.52%	1350	1280	31,93-32,28	0.39	8.71%	195	240	32,10-32,56	2.6	14.33%	1300	1244
33,79-34,09	4.2	11.67%	2100	1990	34,00-34,31	3.6	10.03%	1800	1741	32,28-32,58	0.37	8.26%	185	89	33,46-33,60	0.7	3.86%	350	376
34,09-34,27	3.0	8.33%	1500	1540	34,31-34,61	2.9	8.08%	1450	1472	32,58-32,93	0.39	8.71%	195	245	34,33-34,61	1.9	10.47%	950	986
34,50-34,82	4.2	11.67%	2100	2126	34,61-34,91	2.9	8.08%	1450	1511	32,93-33,20	0.34	7.59%	170	154	34,61-34,92	1.9	10.47%	950	949
34,82-35,12	3.9	10.83%	1950	1924	34,91-35,17	2.7	7.52%	1350	1396	33,20-33,60	0.47	10.49%	235	219	34,90-35,30	0.6	3.31%	300	243
					35,17-35,45	2.9	8.08%	1450	1528	33,60-34,00	0.53	11.83%	265	259					
					35,45-35,73	2.7	7.52%	1350	1366	34,00-34,55	0.47	10.49%	235	110					
					35,73-35,95	2.3	6.41%	1150	1111										
TOTAL	**36.0**		**18000**	**17466**	**TOTAL**	**35.9**		**17950**	**17992**	**TOTAL**	**4.48**		**2240**	**2085**	**TOTAL**	**18.2**		**9075**	**8980**

Samples as received, sorted by Hole Designation: SC-11, SB-9, SC10, SE-10

Moisture was preserved and the samples in good condition.

Photo of Screen Assays generated from the Farim Composite Sample

Individual Drill Hole Chemical Analyses

Project Name:	FARIM PHOSPHATE PROJECT					
PN:	2091-05				Date:	5/Mar/15
			SB9			
Sample Description	**Top**	**Middle**	**Bottom**	**Composite**	**Composite**	**"150g"**
Section No.	32,15-32,35	33,46-33,79	34,82-35,12	-	-	-
Thornton Lab Sample #	375950	375951	375952	376720	-	375359
		Analyzed			**Calculated**	**Analyzed**
Phosphorus - ICP - P2O5	35.28	30.83	26.82	30.99	30.63	29.58
Aluminum - Al2O3	0.78	1.46	0.59	0.87	1.00	1.88
Iron - Fe2O3	1.85	2.00	1.58	2.26	1.82	2.90
Sulfur (S), Total	0.95	1.12	0.99	1.32	1.03	1.67
Pyritic Sulfur (S)	0.73	0.92	0.55	0.95	0.75	1.02
$S_{pyritic}/S_{total}$ %	76.84	82.14	55.56	71.97	72.40	61.08
Pyritic Iron*	0.91	1.15	0.68	1.18	0.93	1.27
Calcium - CaO	49.57	43.86	46.74	46.13	46.28	39.08
Magnesium - MgO	0.02	0.32	3.70	0.85	1.37	0.42
Acid Insolubles	4.46	11.27	0.88	2.15	6.06	8.27
CaO/P_2O_5	1.405	1.423	1.743	1.489	1.511	1.321
MER	0.075	0.123	0.219	0.128	0.137	0.176
Adjusted MER *	0.049	0.085	0.193	0.090	0.106	0.133
Grade Potential, P_2O_5, %	38.36	36.64	27.95	33.18	34.00	34.69

Project Name:	FARIM PHOSPHATE PROJECT					
PN:	2091-05				Date:	5/Mar/15
			SC11			
Sample Description	**Top**	**Middle**	**Bottom**	**Composite**	**Composite**	**"150g"**
Section No.	30,47-30,82	32,28-32,58	34,00-34,55		-	-
Thornton Lab Sample #	375956	375957	375958	376722	-	375361
		Analyzed			**Calculated**	**Analyzed**
Phosphorus - ICP - P2O5	30.30	33.26	31.13	34.51	31.48	30.77
Aluminum - Al2O3	2.96	0.72	1.79	1.15	1.87	0.99
Iron - Fe2O3	2.21	1.16	2.58	1.95	2.04	1.31
Sulfur (S), Total	1.39	0.91	1.66	1.56	1.35	1.24
Pyritic Sulfur (S)	1.06	0.73	1.24	1.09	1.03	0.76
$S_{pyritic}/S_{total}$ %	76.26	80.22	74.70	69.87	76.33	61.29
Pyritic Iron*	1.32	0.91	1.54	1.36	1.28	0.95
Calcium - CaO	41.81	47.85	44.73	48.44	44.67	44.70
Magnesium - MgO	0.08	0.03	0.41	0.09	0.19	0.07
Acid Insolubles	10.99	9.86	5.72	3.88	8.69	10.80
CaO/P_2O_5	1.380	1.439	1.437	1.404	1.419	1.453
MER	0.173	0.057	0.154	0.092	0.130	0.077
Adjusted MER *	0.130	0.030	0.104	0.053	0.089	0.046
Grade Potential, P_2O_5, %	36.75	38.08	35.27	37.73	36.58	35.92

Project Name:	FARIM PHOSPHATE PROJECT					
PN:	2091-05				Date:	5/Mar/15
			SC10			
Sample Description	**Top**	**Middle**	**Bottom**	**Composite**	**Composite**	**"150g"**
Section No.	32,24-32,56	34,00-34,31	35,73-35,95	-	-	-
Thornton Lab Sample #	375362	375363	375364	376723	-	375360
		Analyzed			**Calculated**	**Analyzed**
Phosphorus - ICP - P2O5	31.26	33.35	31.11	35.03	32.00	35.42
Aluminum - Al2O3	2.26	1.21	0.68	0.92	1.50	0.67
Iron - Fe2O3	3.30	3.11	2.60	1.88	3.06	1.61
Sulfur (S), Total	2.11	1.67	1.78	1.43	1.87	1.52
Pyritic Sulfur (S)	1.63	1.41	1.28	1.03	1.47	0.94
$S_{pyritic}/S_{total}$ %	77.25	84.43	71.91	72.03	78.43	61.84
Pyritic Iron*	2.03	1.76	1.59	1.28	1.82	1.17
Calcium - CaO	43.75	47.21	46.00	49.52	45.57	49.06
Magnesium - MgO	0.02	0.19	0.26	0.13	0.14	0.04
Acid Insolubles	9.69	4.30	0.94	1.85	5.61	4.28
CaO/P_2O_5	1.400	1.416	1.479	1.414	1.424	1.385
MER	0.179	0.135	0.114	0.084	0.147	0.065
Adjusted MER *	0.114	0.083	0.063	0.047	0.090	0.032
Grade Potential, P_2O_5, %	37.83	37.18	33.10	37.30	36.38	38.53

			SE10			
Sample Description	**Top**	**Middle**	**Bottom**	**Composite**	**Composite**	**"150g"**
Section No.	30,63-31,11	32,10-32,56	34,90-35,30		-	-
Thornton Lab Sample #	375959	375960	375961	376721	-	375363
		Analyzed			**Calculated**	**Analyzed**
Phosphorus - ICP - P2O5	29.90	35.54	30.72	32.44	32.34	36.42
Aluminum - Al2O3	1.06	0.50	0.81	1.01	0.80	0.40
Iron - Fe2O3	3.57	1.17	4.45	3.44	2.65	1.15
Sulfur (S), Total	1.01	0.80	0.91	1.12	0.91	0.98
Pyritic Sulfur (S)	0.68	0.50	0.38	0.71	0.58	0.50
$S_{pyritic}/S_{total}$ %	67.33	62.50	41.76	63.39	63.08	51.02
Pyritic Iron*	0.85	0.62	0.47	0.88	0.72	0.62
Calcium - CaO	40.68	51.90	46.27	46.04	45.93	51.27
Magnesium - MgO	0.17	0.03	0.53	0.24	0.15	0.05
Acid Insolubles	11.59	3.92	2.31	4.22	7.48	4.17
CaO/P_2O_5	1.361	1.460	1.506	1.419	1.420	1.408
MER	0.161	0.048	0.188	0.145	0.111	0.044
Adjusted MER *	0.132	0.030	0.173	0.117	0.089	0.027
Grade Potential, P_2O_5, %	36.12	37.97	33.57	35.96	36.69	39.04

Appendix B: Horizontal Scrubbing

Test Summary

Test Number	% Solids	RPM	Retention Time, minutes
Baseline	50	36.8	5
HS #1	50	36.8	2.5
HS #2	50	36.8	5
HS #3	50	36.8	10
HS #4	35	36.8	2.5
HS #5	35	36.8	5
HS #6	35	36.8	10
HS #7/Confirmation	35	36.8	5

Normalized Horizontal Scrubbing Results for Tests at 50% Solids

Time, seconds	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Cum. Grades							
						P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
0.00	1180x75	475.4	100.00	100.00	0.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
150.00	1180x75	310.6	65.33	65.33	34.67	105.71	105.54	56.49	24.36	62.86	86.57	78.27	89.97
300.00	1180x75	332.2	70.41	70.41	29.59	105.84	105.45	45.67	26.78	69.93	77.43	66.49	85.66
600.00	1180x75	344.4	72.64	72.64	27.36	104.48	104.28	58.28	25.67	71.68	107.15	108.93	84.85

Time, seconds	Cum. Distribution								CaO/P_2O_5	MER	MER*	Grade Pot. P2O5, %
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %				
0.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.0	100.0	100.0	100.0
150.00	71.48	71.46	44.37	16.29	59.31	68.57	68.57	61.21	99.8	69.4	34.3	102.7
300.00	74.52	74.25	34.51	18.19	50.11	70.41	70.41	60.68	99.6	68.5	44.5	103.0
600.00	75.89	75.75	42.33	16.98	52.27	72.64	72.64	72.64	99.8	102.9	17.6	101.8

Normalized Horizontal Scrubbing Results for Tests at 35% Solids

Time, seconds	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Cum. Grades							
						P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
0.00	1180x75	475.4	100.00	100.00	0.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
150.00	1180x75	325.0	68.18	68.18	31.82	104.10	103.96	54.26	29.84	69.14	99.55	98.56	94.71
300.00	1180x75	349.2	73.66	73.66	26.34	104.89	104.87	59.90	26.22	55.08	87.80	73.05	88.31
600.00	1180x75	340.4	71.89	71.89	28.11	107.07	107.04	62.19	24.71	30.85	99.47	95.60	89.53

Time, seconds	Cum. Distribution								CaO/P_2O_5	MER	MER*	Grade Pot. P2O5, %
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %				
0.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.0	100.0	100.0	100.0
150.00	70.97	70.88	37.00	19.05	43.86	68.18	68.18	64.57	99.9	78.5	36.2	101.9
300.00	77.26	77.24	44.12	17.62	51.14	73.66	73.66	65.05	100.0	65.4	31.6	101.2
600.00	76.97	76.95	44.71	16.82	46.60	71.89	71.89	64.37	100.0	46.5	6.6	99.6

Highlighted Test reflects best results, conditions selected for further testing.

Baseline Test

Standard # 1 Test	
Baseline	**Observations:**
610 grams of wet ore + 390 mL H2O (50% solids) added to mill at 36.8 RPM for 5 minutes	lots of slimes released during attrition scrubbing became very thick and viscous
+6mm screened out, dried and weighed	
Remaining ore dewatered and attrition scrubbed at 41% solids at 560 RPM for 10 minutes	

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	35.60	7.56	7.56	92.44
3x16	1180	9.30	1.97	9.53	90.47
16x40	425	32.40	6.88	16.41	83.59
40x150	106	220.50	46.82	63.23	36.77
150x635	20	69.50	14.76	77.98	22.02
-635	6	103.70	22.02	100.00	0.00
Total		471.00	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	27.18	40.07	0.94	0.78	8.72	2.86	2.44	6.87	27.18	40.07	0.94	0.78	8.72	2.86	2.44	6.87
3x16	20.56	30.55	0.72	0.64	22.73	3.83	3.52	5.04	25.81	38.10	0.89	0.75	11.62	3.06	2.66	6.49
16x40	32.97	46.84	0.09	0.14	3.40	1.60	1.16	5.34	28.81	41.76	0.56	0.49	8.18	2.45	2.03	6.01
40x150	33.36	47.32	0.08	0.12	1.59	0.94	0.59	8.59	32.18	45.88	0.20	0.22	3.30	1.33	0.96	7.92
150x635	33.78	48.51	0.50	0.64	3.11	1.30	0.99	2.10	32.48	46.38	0.26	0.30	3.26	1.33	0.97	6.82
-635	29.11	41.38	0.60	3.65	2.50	0.93	0.56	9.47	31.74	45.28	0.33	1.04	3.10	1.24	0.88	7.40
Total	31.74	45.28	0.33	1.04	3.10	1.24	0.88	7.40								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	6.47	6.69	21.22	5.69	21.29	17.45	20.97	7.01	6.47	6.69	21.22	5.69	21.29	17.45	20.97	7.01
3x16	1.28	1.33	4.25	1.22	14.50	6.11	7.90	1.34	7.75	8.02	25.47	6.91	35.79	23.56	28.88	8.36
16x40	7.15	7.12	1.85	0.93	7.56	8.89	9.07	4.96	14.90	15.14	27.32	7.84	43.35	32.45	37.95	13.32
40x150	49.21	48.93	11.19	5.43	24.05	35.53	31.41	54.33	64.10	64.07	38.50	13.27	67.39	67.98	69.36	67.65
150x635	15.70	15.81	22.04	9.12	14.83	15.49	16.61	4.19	79.81	79.88	60.54	22.39	82.22	83.47	85.98	71.83
-635	20.19	20.12	39.46	77.61	17.78	16.53	14.02	28.17	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

HS #1

HS #1
Horizontal Scrubbing
610 grams of wet ore + 390 mL H2O (50% solids) added to mill at 36.8 RPM for 2.5 minutes
Then screened and sent for chemical analysis

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	61.90	13.02	13.02	86.98
3x16	1180	15.40	3.24	16.26	83.74
16x40	425	41.40	8.71	24.97	75.03
40x150	106	204.10	42.93	67.90	32.10
150x635	20	65.10	13.69	81.59	18.41
-635	6	87.50	18.41	100.00	0.00
Total		475.40	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	28.30	41.05	0.51	1.23	6.00	1.68	1.32	8.26	28.30	41.05	0.51	1.23	6.00	1.68	1.32	8.26
3x16	23.68	35.18	0.74	0.45	17.38	4.10	3.77	5.51	27.38	39.88	0.56	1.07	8.27	2.16	1.81	7.71
16x40	33.70	48.39	0.10	0.13	3.04	1.50	0.96	5.69	29.58	42.85	0.40	0.75	6.44	1.93	1.51	7.01
40x150	33.59	47.79	0.09	0.11	1.41	0.90	0.51	8.28	32.12	45.97	0.20	0.34	3.26	1.28	0.88	7.81
150x635	33.74	48.46	0.52	0.74	2.82	1.23	0.98	1.98	32.39	46.39	0.26	0.41	3.19	1.27	0.90	6.83
-635	29.29	41.49	0.61	3.64	2.49	0.95	0.67	9.65	31.82	45.49	0.32	1.00	3.06	1.21	0.85	7.35
Total	31.82	45.49	0.32	1.00	3.06	1.21	0.85	7.35								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	11.58	11.75	20.67	15.94	25.54	18.05	20.12	14.63	11.58	11.75	20.67	15.94	25.54	18.05	20.12	14.63
3x16	2.41	2.51	7.46	1.45	18.41	10.96	14.30	2.43	13.99	14.26	28.14	17.39	43.95	29.01	34.42	17.06
16x40	9.22	9.26	2.71	1.13	8.65	10.78	9.79	6.74	23.21	23.52	30.85	18.52	52.60	39.79	44.21	23.80
40x150	45.32	45.10	12.03	4.70	19.79	31.88	25.64	48.35	68.54	68.62	42.88	23.22	72.39	71.67	69.85	72.15
150x635	14.52	14.59	22.17	10.09	12.62	13.90	15.71	3.69	83.06	83.21	65.05	33.31	85.02	85.57	85.56	75.84
-635	16.94	16.79	34.95	66.69	14.98	14.43	14.44	24.16	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

HS #2

HS #2
Horizontal Scrubbing
610 grams of wet ore + 390 mL H2O (50% solids) added to mill at 36.8 RPM for 5 minutes
Then screened and sent for chemical analysis

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	26.60	5.64	5.64	94.36
3x16	1180	17.40	3.69	9.33	90.67
16x40	425	49.10	10.41	19.73	80.27
40x150	106	213.60	45.27	65.01	34.99
150x635	20	69.50	14.73	79.74	20.26
-635	6	95.60	20.26	100.00	0.00
Total		471.80	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	25.78	40.61	2.08	1.11	6.34	1.52	1.15	7.34	25.78	40.61	2.08	1.11	6.34	1.52	1.15	7.34
3x16	24.62	33.09	0.77	0.57	14.79	4.88	4.47	5.13	25.32	37.64	1.56	0.90	9.68	2.85	2.46	6.47
16x40	32.86	46.51	0.10	0.14	2.79	1.60	1.07	6.18	29.30	42.32	0.79	0.50	6.05	2.19	1.73	6.32
40x150	34.18	48.60	0.08	0.15	1.45	0.48	0.14	6.34	32.70	46.69	0.30	0.26	2.85	1.00	0.62	6.33
150x635	32.68	47.34	0.55	0.79	2.80	1.42	1.07	2.25	32.69	46.81	0.34	0.35	2.84	1.08	0.70	5.58
-635	28.35	40.56	0.61	3.81	2.46	1.13	0.73	9.51	31.81	45.55	0.40	1.05	2.76	1.09	0.71	6.37
Total	31.81	45.55	0.40	1.05	2.76	1.09	0.71	6.37								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	4.57	5.03	29.55	5.93	12.95	7.88	9.13	6.49	4.57	5.03	29.55	5.93	12.95	7.88	9.13	6.49
3x16	2.85	2.68	7.15	1.99	19.76	16.55	23.22	2.97	7.42	7.71	36.70	7.93	32.71	24.43	32.35	9.46
16x40	10.75	10.63	2.62	1.38	10.52	15.31	15.68	10.09	18.17	18.33	39.32	9.31	43.22	39.74	48.04	19.55
40x150	48.64	48.31	9.13	6.44	23.78	19.98	8.93	45.03	66.81	66.64	48.45	15.75	67.00	59.72	56.96	64.57
150x635	15.13	15.31	20.41	11.04	14.94	19.23	22.20	5.20	81.94	81.96	68.86	26.79	81.94	78.95	79.17	69.77
-635	18.06	18.04	31.14	73.21	18.06	21.05	20.83	30.23	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

HS #3

HS #3
Horizontal Scrubbing
610 grams of wet ore + 390 mL H2O (50% solids) added to mill at 36.8 RPM for 10 minutes
Then screened and sent for chemical analysis

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	13.50	2.85	2.85	97.15
3x16	1180	14.90	3.14	5.99	94.01
16x40	425	44.10	9.30	15.29	84.71
40x150	106	226.30	47.73	63.02	36.98
150x635	20	74.00	15.61	78.63	21.37
-635	6	101.30	21.37	100.00	0.00
Total		474.10	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	26.87	36.89	0.86	0.49	9.18	1.99	1.49	7.84	26.87	36.89	0.86	0.49	9.18	1.99	1.49	7.84
3x16	23.54	36.23	1.02	0.66	15.17	4.51	4.14	7.26	25.12	36.54	0.94	0.58	12.32	3.31	2.88	7.54
16x40	33.01	46.19	0.09	0.13	2.71	1.58	1.00	4.98	29.92	42.41	0.42	0.31	6.48	2.26	1.74	5.98
40x150	33.87	48.90	0.09	0.13	1.46	1.20	0.79	6.28	32.91	47.33	0.17	0.17	2.68	1.46	1.02	6.21
150x635	33.65	48.04	0.53	0.69	2.82	5.14	4.71	2.03	33.06	47.47	0.24	0.28	2.71	2.19	1.75	5.38
-635	29.36	41.89	0.59	3.55	2.53	1.10	0.68	8.89	32.27	46.28	0.32	0.98	2.67	1.96	1.52	6.13
Total	32.27	46.28	0.32	0.98	2.67	1.96	1.52	6.13								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	2.37	2.27	7.73	1.43	9.80	2.90	2.79	3.64	2.37	2.27	7.73	1.43	9.80	2.90	2.79	3.64
3x16	2.29	2.46	10.12	2.13	17.87	7.25	8.54	3.72	4.66	4.73	17.86	3.56	27.67	10.15	11.33	7.37
16x40	9.52	9.28	2.64	1.24	9.45	7.52	6.11	7.56	14.18	14.02	20.50	4.80	37.12	17.66	17.44	14.92
40x150	50.10	50.44	13.57	6.36	26.12	29.29	24.76	48.91	64.28	64.45	34.07	11.16	63.24	46.95	42.19	63.84
150x635	16.28	16.20	26.12	11.05	16.50	41.03	48.27	5.17	80.56	80.66	60.19	22.21	79.74	87.98	90.46	69.01
-635	19.44	19.34	39.81	77.79	20.26	12.02	9.54	30.99	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

HS #4

HS #4
Horizontal Scrubbing
610 grams of wet ore + 820 mL H2O (35% solids) added to mill at 36.8 RPM for 2.5 minutes
Then screened and sent for chemical analysis

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	45.40	9.52	9.52	90.48
3x16	1180	17.50	3.67	13.19	86.81
16x40	425	50.60	10.61	23.81	76.19
40x150	106	208.30	43.70	67.51	32.49
150x635	20	66.10	13.87	81.37	18.63
-635	6	88.80	18.63	100.00	0.00
Total		476.70	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	30.41	43.89	0.56	0.96	5.38	1.22	0.84	6.38	30.41	43.89	0.56	0.96	5.38	1.22	0.84	6.38
3x16	26.24	39.07	0.78	0.40	13.84	1.29	0.87	5.82	29.25	42.55	0.62	0.80	7.73	1.24	0.85	6.22
16x40	34.03	48.77	0.09	0.17	2.88	1.38	0.93	7.89	31.38	45.32	0.38	0.52	5.57	1.30	0.88	6.97
40x150	33.56	47.89	0.09	0.16	1.44	0.95	0.68	8.00	32.79	46.98	0.19	0.29	2.90	1.07	0.75	7.64
150x635	33.82	48.26	0.46	0.81	2.91	1.45	0.67	2.00	32.97	47.20	0.24	0.38	2.90	1.14	0.74	6.68
-635	29.70	42.20	0.59	3.64	2.58	1.06	0.68	9.17	32.36	46.27	0.30	0.98	2.84	1.12	0.73	7.14
Total	32.36	46.27	0.30	0.98	2.84	1.12	0.73	7.14								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	8.95	9.03	17.51	9.29	18.04	10.34	11.00	8.51	8.95	9.03	17.51	9.29	18.04	10.34	11.00	8.51
3x16	2.98	3.10	9.40	1.49	17.89	4.21	4.39	2.99	11.93	12.13	26.92	10.78	35.94	14.55	15.39	11.50
16x40	11.16	11.19	3.14	1.83	10.77	13.04	13.57	11.73	23.09	23.32	30.05	12.61	46.70	27.59	28.96	23.23
40x150	45.32	45.23	12.91	7.10	22.16	36.94	40.85	48.96	68.41	68.55	42.97	19.71	68.86	64.53	69.81	72.19
150x635	14.49	14.46	20.94	11.41	14.21	17.89	12.77	3.88	82.90	83.01	63.91	31.12	83.07	82.43	82.58	76.08
-635	17.10	16.99	36.09	68.88	16.93	17.57	17.42	23.92	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

HS #5

HS #5	
Horizontal Scrubbing	
610 grams of wet ore + 820 mL H2O (35% solids) added to mill at 36.8 RPM for 5 minutes	
Then screened and sent for chemical analysis	

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	10.70	2.26	2.26	97.74
3x16	1180	16.50	3.48	5.74	94.26
16x40	425	45.20	9.53	15.27	84.73
40x150	106	233.00	49.15	64.42	35.58
150x635	20	71.00	14.98	79.39	20.61
-635	6	97.70	20.61	100.00	0.00
Total		474.10	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	23.90	36.37	1.10	0.49	12.20	2.25	1.71	9.72	23.90	36.37	1.10	0.49	12.20	2.25	1.71	9.72
3x16	24.12	35.48	0.51	0.53	43.90	4.10	3.65	6.75	24.03	35.83	0.74	0.51	31.43	3.37	2.89	7.92
16x40	33.01	47.76	0.09	0.13	3.09	1.57	0.91	4.78	29.64	43.28	0.33	0.27	13.74	2.25	1.65	5.96
40x150	34.97	49.83	0.10	0.13	1.71	0.99	0.41	7.36	33.71	48.28	0.16	0.16	4.56	1.29	0.70	7.03
150x635	33.42	48.01	0.48	0.71	3.12	1.34	0.71	2.16	33.65	48.23	0.22	0.27	4.29	1.30	0.71	6.11
-635	29.50	41.83	0.59	3.56	2.64	1.28	0.84	9.26	32.80	46.91	0.29	0.95	3.95	1.29	0.73	6.76
Total	32.80	46.91	0.29	0.95	3.95	1.29	0.73	6.76								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	1.64	1.75	8.45	1.17	6.97	3.92	5.26	3.25	1.64	1.75	8.45	1.17	6.97	3.92	5.26	3.25
3x16	2.56	2.63	6.04	1.95	38.68	11.03	17.32	3.48	4.20	4.38	14.49	3.12	45.66	14.95	22.59	6.72
16x40	9.60	9.71	2.92	1.31	7.46	11.57	11.83	6.74	13.80	14.09	17.41	4.43	53.12	26.52	34.42	13.46
40x150	52.40	52.21	16.73	6.76	21.28	37.60	27.48	53.52	66.20	66.30	34.14	11.19	74.39	64.11	61.89	66.98
150x635	15.26	15.33	24.47	11.24	11.83	15.51	14.50	4.79	81.46	81.62	58.61	22.43	86.23	79.62	76.39	71.77
-635	18.54	18.38	41.39	77.57	13.77	20.38	23.61	28.23	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

HS #6

HS #6
Horizontal Scrubbing
610 grams of wet ore + 820 mL H2O (35% solids) added to mill at 36.8 RPM for 10 minutes
Then screened and sent for chemical analysis

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	20.30	4.29	4.29	95.71
3x16	1180	11.90	2.51	6.80	93.20
16x40	425	57.30	12.10	18.90	81.10
40x150	106	213.10	45.01	63.91	36.09
150x635	20	70.00	14.78	78.69	21.31
-635	6	100.90	21.31	100.00	0.00
Total		473.50	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	14.28	20.12	0.45	1.53	77.70	3.58	3.28	11.71	14.28	20.12	0.45	1.53	77.70	3.58	3.28	11.71
3x16	18.86	27.60	0.64	0.50	59.46	5.11	4.70	4.90	15.97	22.88	0.52	1.15	70.96	4.15	3.80	9.19
16x40	33.57	47.69	0.09	0.13	3.04	1.24	0.57	8.23	27.24	38.77	0.24	0.50	27.48	2.29	1.73	8.58
40x150	34.05	48.28	0.09	0.14	1.62	0.95	0.43	7.61	32.04	45.47	0.14	0.25	9.27	1.34	0.82	7.90
150x635	32.88	47.54	0.58	0.74	2.88	4.46	3.94	2.13	32.19	45.86	0.22	0.34	8.07	1.93	1.40	6.81
-635	28.95	41.23	0.63	3.72	2.43	0.99	0.59	9.37	31.50	44.87	0.31	1.06	6.87	1.73	1.23	7.36
Total	31.50	44.87	0.31	1.06	6.87	1.73	1.23	7.36								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	1.94	1.92	6.29	6.19	48.52	8.87	11.44	6.82	1.94	1.92	6.29	6.19	48.52	8.87	11.44	6.82
3x16	1.50	1.55	5.24	1.19	21.76	7.42	9.61	1.67	3.45	3.47	11.53	7.38	70.28	16.30	21.05	8.50
16x40	12.90	12.86	3.55	1.49	5.36	8.67	5.61	13.54	16.34	16.33	15.08	8.87	75.64	24.97	26.66	22.03
40x150	48.64	48.43	13.20	5.95	10.62	24.72	15.74	46.55	64.99	64.76	28.29	14.82	86.26	49.69	42.40	68.58
150x635	15.43	15.66	27.95	10.33	6.20	38.12	47.38	4.28	80.42	80.42	56.24	25.15	92.46	87.80	89.77	72.86
-635	19.58	19.58	43.76	74.85	7.54	12.20	10.23	27.14	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

HS #7 – Confirmation Test

HS #7 - Confirmation Test
Horizontal Scrubbing
610 grams of wet ore + 820 mL H_2O (35% solids) added to mill at 36.8 RPM for 5 minutes
Then screened and sent for chemical analysis

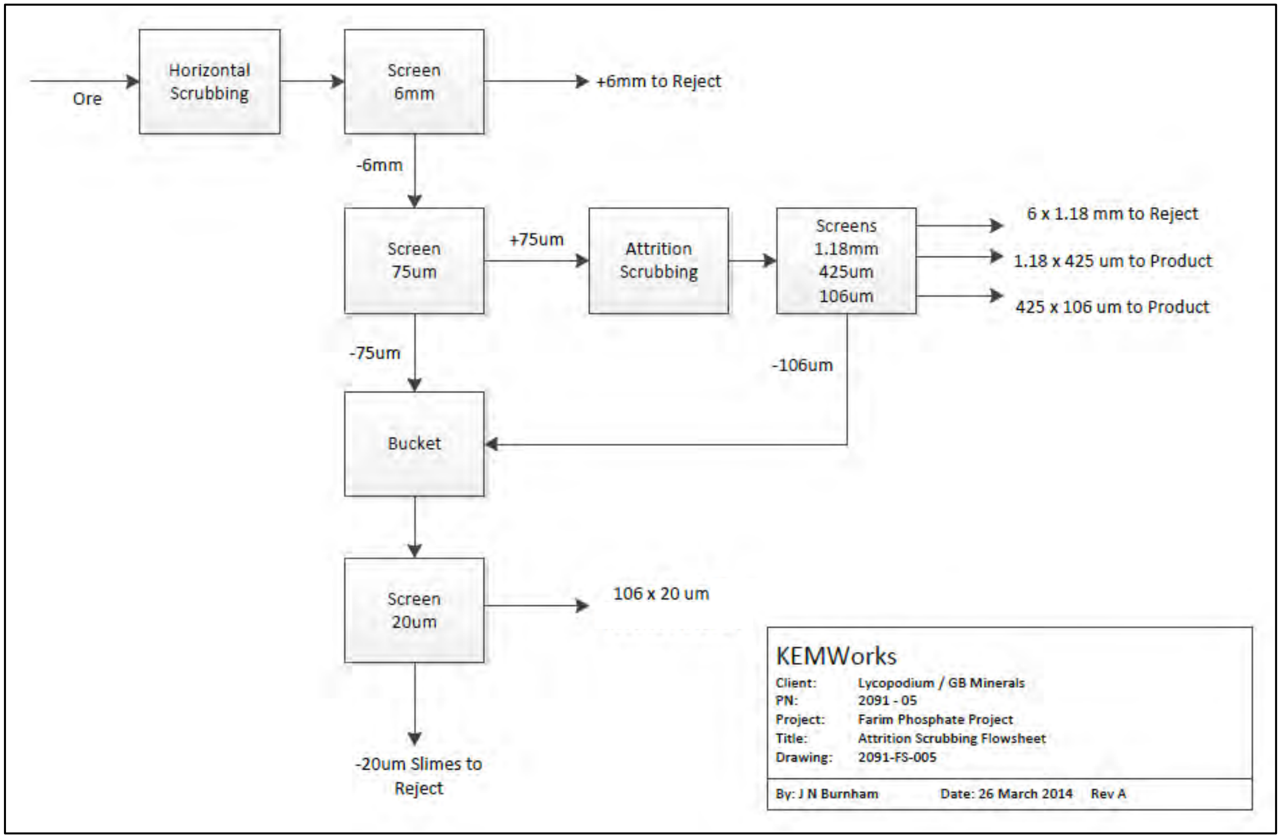
US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	13.00	2.74	2.74	97.26
3x16	1180	15.60	3.29	6.03	93.97
16x40	425	66.10	13.93	19.96	80.04
40x150	106	210.00	44.27	64.23	35.77
150x200	75	23.60	4.97	69.20	30.80
200x635	20	48.30	10.18	79.38	20.62
-635	6	97.80	20.62	100.00	0.00
Total		474.40	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	29.95	41.49	0.39	0.92	4.14	2.75	2.29	8.89	29.95	41.49	0.39	0.92	4.14	2.75	2.29	8.89
3x16	23.15	25.90	0.48	0.65	19.62	4.25	3.88	5.56	26.24	32.99	0.44	0.77	12.58	3.57	3.16	7.07
16x40	33.84	47.47	0.09	0.19	2.74	1.43	0.89	5.56	31.55	43.10	0.20	0.37	5.71	2.08	1.57	6.02
40x150	35.34	50.37	0.09	0.19	1.41	1.06	0.75	6.63	34.16	48.11	0.12	0.24	2.75	1.38	1.01	6.44
150x200	34.31	48.15	0.26	0.47	2.67	1.34	0.99	3.19	34.17	48.11	0.13	0.26	2.74	1.37	1.01	6.21
200x635	33.42	48.01	0.48	0.71	3.12	1.34	0.71	1.82	31.92	45.08	0.16	0.29	2.62	1.28	0.91	5.64
-635	29.50	41.83	0.59	3.56	2.64	1.28	0.84	9.09	31.42	44.41	0.25	0.96	2.63	1.28	0.89	6.35
Total	33.13	46.81	0.26	0.99	2.76	1.35	0.94	6.35								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	2.48	2.43	4.07	2.55	4.11	5.58	6.67	3.83	2.48	2.43	4.07	2.55	4.11	5.58	6.67	3.83
3x16	2.30	1.82	6.02	2.17	23.38	10.35	13.56	2.88	4.77	4.25	10.09	4.72	27.49	15.93	20.23	6.71
16x40	14.23	14.13	4.78	2.68	13.84	14.75	13.18	12.19	19.01	18.38	14.87	7.40	41.33	30.68	33.40	18.90
40x150	47.22	47.64	15.19	8.52	22.62	34.74	35.28	46.19	66.22	66.02	30.06	15.93	63.95	65.42	68.68	65.09
150x200	5.15	5.12	4.93	2.37	4.81	4.94	5.23	2.50	71.37	71.13	34.99	18.30	68.76	70.36	73.92	67.59
200x635	10.27	10.44	18.63	7.33	11.51	10.10	7.68	2.92	81.64	81.58	53.63	25.62	80.28	80.46	81.60	70.51
-635	18.36	18.42	46.37	74.38	19.72	19.54	18.40	29.49	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

Appendix C: Attrition Scrubbing

Attrition Scrubbing Flowsheet



Normalized Attrition Scrubbing Results for Tests at 45% Solids

Time, seconds	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Cum. Grades P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
Head	1180x75	475.40	100.00	100.00	0.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
150	1180x75	339.40	72.12	72.12	27.88	104.94	105.30	52.74	26.45	70.77	91.89	86.27	89.26
300	1180x75	341.10	72.19	72.19	27.81	104.38	105.34	56.92	23.57	66.10	79.99	72.66	85.44
600	1180x75	341.40	72.72	72.72	27.28	104.97	105.98	55.76	25.14	68.09	94.84	88.71	93.78

Time, seconds	Cum. Distribution P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	CaO/P_2O_5	MER	MER*	Grade Pot. P2O5, %
Head	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.0	100.0	100.0	100.0
150	75.68	75.94	38.04	19.08	51.04	66.27	62.22	64.38	100.3	73.9	38.0	102.3
300	75.35	76.04	41.09	17.01	47.71	57.74	52.46	61.68	100.9	72.2	41.2	100.8
600	76.33	77.06	40.55	18.28	49.52	68.96	64.51	68.19	101.0	74.6	35.1	102.3

Normalized Horizontal Scrubbing Results for Tests at 55% Solids

Time, seconds	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Cum. Grades P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
Head	1180x75	475.40	100.00	100.00	0.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
150	1180x75	346.90	73.90	73.90	26.10	104.43	104.95	61.05	26.63	76.26	95.54	92.50	84.24
300	1180x75	335.20	71.11	71.11	28.89	105.06	104.94	58.45	25.71	65.35	90.20	76.80	90.56
600	1180x75	338.50	71.55	71.55	28.45	104.83	104.79	61.14	23.69	60.94	96.10	95.14	100.59

Time, seconds	Cum. Distribution P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	CaO/P_2O_5	MER	MER*	Grade Pot. P2O5, %
Head	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.0	100.0	100.0	100.0
150	77.18	77.56	45.12	19.68	56.36	70.61	68.36	62.25	100.5	79.2	40.5	101.7
300	74.70	74.62	41.56	18.28	46.47	64.14	54.61	64.40	99.9	72.4	36.1	102.0
600	75.00	74.97	43.75	16.95	43.60	68.76	68.08	71.97	100.0	77.9	17.2	102.3

Normalized Horizontal Scrubbing Results for Tests at 60% Solids

Time, seconds	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Cum. Grades P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
Head	1180x75	475.40	100.00	100.00	0.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
150	1180x75	333.80	70.69	70.69	29.31	104.07	103.95	56.69	25.39	69.25	88.83	81.77	99.32
300	1180x75	340.70	71.79	71.79	28.21	105.27	105.26	57.08	28.69	61.60	67.02	55.06	88.81
600	1180x75	333.90	70.52	70.52	29.48	106.04	105.58	56.17	25.43	64.39	94.82	92.90	85.80

Time, seconds	Cum. Distribution P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	CaO/P_2O_5	MER	MER*	Grade Pot. P2O5, %
Head	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.0	100.0	100.0	100.0
150	73.56	73.48	40.07	17.95	48.95	62.79	57.80	70.21	99.9	75.3	39.4	101.9
300	75.57	75.56	40.98	20.60	44.22	48.11	39.53	63.75	100.0	70.2	40.9	102.0
600	74.78	74.45	39.61	17.93	45.41	66.86	65.51	60.50	99.6	60.9	43.0	102.7

AS #1

AS #1	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 35% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	

Test No.	AS #1
% Solids in Attrition Scrubber	45
Time (min.)	2.5
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	19.80	4.21	4.21	95.79
3x16	1180	9.20	1.95	6.16	93.84
16x40	425	39.80	8.46	14.62	85.38
40x150	106	227.80	48.41	63.03	36.97
150x635	20	71.80	15.26	78.28	21.72
-635	6	102.20	21.72	100.00	0.00
Total		470.60	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	25.98	36.33	1.69	1.02	8.26	1.20	0.78	5.90	25.98	36.33	1.69	1.02	8.26	1.20	0.78	5.90
3x16	22.09	23.12	0.66	0.49	22.79	4.87	4.24	4.11	24.75	32.14	1.36	0.85	12.87	2.36	1.88	5.33
16x40	34.46	48.19	0.10	0.20	3.16	1.35	0.75	5.54	30.37	41.42	0.63	0.47	7.25	1.78	1.23	5.45
40x150	34.44	48.85	0.09	0.17	1.37	0.89	0.46	6.91	33.49	47.13	0.22	0.24	2.73	1.10	0.64	6.57
150x635	33.85	48.65	0.53	0.79	3.09	1.38	0.78	2.15	33.56	47.42	0.28	0.35	2.80	1.15	0.67	5.71
-635	29.60	42.15	0.61	4.05	2.54	1.10	0.60	9.04	32.70	46.28	0.35	1.15	2.75	1.14	0.65	6.43
Total	32.70	46.28	0.35	1.15	2.75	1.14	0.65	6.43								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	3.34	3.30	20.35	3.73	12.65	4.43	5.04	3.86	3.34	3.30	20.35	3.73	12.65	4.43	5.04	3.86
3x16	1.32	0.98	3.69	0.83	16.22	8.35	12.73	1.25	4.66	4.28	24.05	4.56	28.88	12.78	17.77	5.11
16x40	8.91	8.81	2.42	1.47	9.73	10.01	9.74	7.28	13.57	13.09	26.47	6.03	38.61	22.79	27.51	12.39
40x150	50.98	51.10	12.47	7.14	24.15	37.79	34.20	51.99	64.55	64.18	38.94	13.17	62.75	60.58	61.71	64.38
150x635	15.79	16.04	23.15	10.46	17.17	18.47	18.28	5.10	80.34	80.22	62.08	23.64	79.92	79.05	79.99	69.48
-635	19.66	19.78	37.92	76.36	20.08	20.95	20.01	30.52	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

AS #2

AS #2	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	

Test No.	AS #2
% Solids in Attrition Scrubber	45
Time (min.)	5
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	17.50	3.70	3.70	96.30
3x16	1180	9.50	2.01	5.71	94.29
16x40	425	37.10	7.85	13.57	86.43
40x150	106	227.70	48.19	61.76	38.24
150x635	20	76.30	16.15	77.90	22.10
-635	6	104.40	22.10	100.00	0.00
Total		472.50	100.00		

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	2.47	1.98	9.85	1.17	21.05	17.47	23.33	6.72	2.47	1.98	9.85	1.17	21.05	17.47	23.33	6.72
3x16	1.25	0.91	4.22	0.97	15.88	10.12	13.16	1.56	3.72	2.89	14.07	2.14	36.93	27.59	36.49	8.27
16x40	8.50	8.43	2.75	1.17	7.92	9.38	8.71	5.69	12.22	11.32	16.82	3.31	44.84	36.97	45.19	13.97
40x150	49.53	50.06	13.49	5.50	24.63	31.32	25.53	51.19	61.76	61.38	30.30	8.81	69.47	68.29	70.72	65.15
150x635	17.31	17.56	24.86	10.34	15.17	17.04	18.22	4.81	79.07	78.93	55.16	19.15	84.64	85.33	88.94	69.96
-635	20.93	21.07	44.84	80.85	15.36	14.67	11.06	30.04	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	20.79	23.36	0.76	0.36	20.69	6.75	6.42	12.31	20.79	23.36	0.76	0.36	20.69	6.75	6.42	12.31
3x16	19.35	19.70	0.60	0.55	28.74	7.20	6.67	5.26	20.28	22.07	0.70	0.43	23.52	6.91	6.51	9.83
16x40	33.74	46.82	0.10	0.17	3.67	1.71	1.13	4.92	28.07	36.40	0.35	0.28	12.03	3.90	3.40	6.99
40x150	32.03	45.31	0.08	0.13	1.86	0.93	0.54	7.21	31.16	43.35	0.14	0.16	4.09	1.58	1.17	7.16
150x635	33.41	47.43	0.44	0.73	3.42	1.51	1.15	2.02	31.63	44.20	0.20	0.28	3.95	1.57	1.16	6.10
-635	29.52	41.59	0.58	4.17	2.53	0.95	0.51	9.23	31.16	43.62	0.29	1.14	3.64	1.43	1.02	6.79
Total	31.16	43.62	0.29	1.14	3.64	1.43	1.02	6.79								

AS # 3

AS #3	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	
Test No.	AS #3
% Solids in Attrition Scrubber	45
Time (min.)	10
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	14.60	3.11	3.11	96.89
3x16	1180	8.60	1.83	4.94	95.06
16x40	425	39.30	8.37	13.31	86.69
40x150	106	217.30	46.28	59.60	40.40
150x635	20	84.80	18.06	77.66	22.34
-635	6	104.90	22.34	100.00	0.00
Total		469.50	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	22.68	27.42	0.84	0.66	17.00	2.99	2.46	6.93	22.68	27.42	0.84	0.66	17.00	2.99	2.46	6.93
3x16	19.76	20.03	0.60	0.52	27.89	4.91	4.58	5.15	21.60	24.68	0.75	0.61	21.04	3.70	3.25	6.27
16x40	34.11	48.55	0.09	0.16	3.64	1.67	1.22	6.24	29.47	39.69	0.34	0.33	10.10	2.42	1.97	6.25
40x150	33.48	48.29	0.07	0.13	1.44	0.96	0.44	8.79	32.58	46.37	0.13	0.17	3.37	1.29	0.78	8.22
150x635	34.13	49.05	0.40	0.77	3.01	1.38	1.00	2.05	32.94	46.99	0.19	0.31	3.29	1.31	0.83	6.79
-635	29.25	41.53	0.57	4.12	2.26	0.86	0.48	8.97	32.12	45.77	0.28	1.16	3.06	1.21	0.75	7.27
Total	32.12	45.77	0.28	1.16	3.06	1.21	0.75	7.27								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	2.20	1.86	9.44	1.76	17.28	7.69	10.15	2.96	2.20	1.86	9.44	1.76	17.28	7.69	10.15	2.96
3x16	1.13	0.80	3.97	0.82	16.70	7.44	11.13	1.30	3.32	2.66	13.41	2.58	33.98	15.14	21.27	4.26
16x40	8.89	8.88	2.72	1.15	9.96	11.57	13.54	7.18	12.21	11.54	16.14	3.73	43.94	26.70	34.82	11.44
40x150	48.25	48.83	11.71	5.17	21.79	36.77	27.01	55.92	60.46	60.37	27.85	8.91	65.72	63.47	61.82	67.36
150x635	19.19	19.36	26.12	11.96	17.77	20.63	23.95	5.09	79.65	79.73	53.96	20.86	83.49	84.10	85.78	72.45
-635	20.35	20.27	46.04	79.14	16.51	15.90	14.22	27.55	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

AS #4

AS #4	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	
Test No.	AS #4
% Solids in Attrition Scrubber	55
Time (min.)	2.5
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	9.30	1.98	1.98	98.02
3x16	1180	10.50	2.24	4.22	95.78
16x40	425	37.00	7.88	12.10	87.90
40x150	106	226.60	48.27	60.37	39.63
150x635	20	83.30	17.75	78.12	21.88
-635	6	102.70	21.88	100.00	0.00
Total		469.40	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	27.47	41.36	1.83	0.94	4.93	1.40	0.86	6.53	27.47	41.36	1.83	0.94	4.93	1.40	0.86	6.53
3x16	20.95	23.22	0.70	0.42	22.14	4.27	3.64	6.13	24.01	31.74	1.23	0.66	14.06	2.92	2.33	6.32
16x40	33.28	47.40	0.10	0.17	3.88	1.61	0.93	5.29	30.05	41.94	0.49	0.34	7.43	2.07	1.42	5.65
40x150	34.40	50.33	0.09	0.15	1.42	0.91	0.27	6.13	33.53	48.65	0.17	0.19	2.62	1.14	0.50	6.03
150x635	32.57	47.00	0.60	0.80	2.66	1.41	1.09	2.18	33.31	48.27	0.27	0.33	2.63	1.20	0.63	5.16
-635	29.17	41.97	0.64	4.12	2.47	0.99	0.39	9.21	32.40	46.89	0.35	1.16	2.60	1.16	0.58	6.04
Total	32.40	46.89	0.35	1.16	2.60	1.16	0.58	6.04								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	1.68	1.75	10.37	1.61	3.76	2.40	2.93	2.14	1.68	1.75	10.37	1.61	3.76	2.40	2.93	2.14
3x16	1.45	1.11	4.48	0.81	19.07	8.26	14.02	2.27	3.13	2.86	14.84	2.42	22.83	10.66	16.95	4.41
16x40	8.10	7.97	2.25	1.16	11.78	10.98	12.62	6.90	11.22	10.82	17.10	3.58	34.61	21.63	29.57	11.31
40x150	51.25	51.81	12.42	6.26	26.40	37.99	22.44	48.96	62.47	62.63	29.52	9.84	61.01	59.63	52.01	60.26
150x635	17.84	17.79	30.44	12.27	18.18	21.64	33.30	6.40	80.30	80.42	59.96	22.10	79.19	81.27	85.31	66.66
-635	19.70	19.58	40.04	77.90	20.81	18.73	14.69	33.34	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

AS #5

AS #5	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	
Test No.	AS #5
% Solids in Attrition Scrubber	55
Time (min.)	5
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	25.20	5.35	5.35	94.65
3x16	1180	9.20	1.95	7.30	92.70
16x40	425	35.50	7.53	14.83	85.17
40x150	106	223.20	47.35	62.18	37.82
150x635	20	76.50	16.23	78.40	21.60
-635	6	101.80	21.60	100.00	0.00
Total		471.40	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	24.79	37.72	0.52	0.63	13.32	2.25	1.57	9.45	24.79	37.72	0.52	0.63	13.32	2.25	1.57	9.45
3x16	19.82	32.58	0.82	0.50	23.87	5.56	5.02	7.85	23.46	36.35	0.60	0.60	16.14	3.14	2.49	9.02
16x40	33.48	48.58	0.10	0.16	3.62	1.53	0.71	5.56	28.55	42.56	0.35	0.37	9.78	2.32	1.59	7.26
40x150	33.66	48.39	0.08	0.14	1.57	0.96	0.39	8.39	32.44	47.00	0.14	0.20	3.53	1.28	0.68	8.12
150x635	31.65	46.36	0.48	0.75	2.94	1.39	0.89	2.18	32.28	46.87	0.21	0.31	3.41	1.31	0.72	6.89
-635	29.07	41.42	0.60	3.94	2.51	1.00	0.63	9.10	31.58	45.69	0.30	1.09	3.21	1.24	0.70	7.37
Total	31.58	45.69	0.30	1.09	3.21	1.24	0.70	7.37								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	4.20	4.41	9.37	3.08	22.16	9.70	11.98	6.86	4.20	4.41	9.37	3.08	22.16	9.70	11.98	6.86
3x16	1.22	1.39	5.39	0.89	14.50	8.75	13.99	2.08	5.42	5.80	14.76	3.97	36.66	18.45	25.97	8.93
16x40	7.98	8.01	2.54	1.10	8.48	9.29	7.63	5.68	13.40	13.81	17.30	5.07	45.14	27.74	33.60	14.62
40x150	50.46	50.15	12.77	6.06	23.14	36.65	26.36	53.91	63.86	63.96	30.07	11.13	68.28	64.40	59.96	68.53
150x635	16.26	16.47	26.26	11.12	14.85	18.19	20.62	4.80	80.12	80.42	56.33	22.25	83.13	82.59	80.58	73.33
-635	19.88	19.58	43.67	77.75	16.87	17.41	19.42	26.67	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

AS #6

AS #6	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	
Test No.	AS #6
% Solids in Attrition Scrubber	55
Time (min.)	10
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	18.80	3.97	3.97	96.03
3x16	1180	9.00	1.90	5.88	94.12
16x40	425	36.10	7.63	13.51	86.49
40x150	106	222.40	47.01	60.52	39.48
150x635	20	80.00	16.91	77.43	22.57
-635	6	106.80	22.57	100.00	0.00
Total		473.10	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	18.65	30.94	0.92	0.60	20.96	5.64	5.36	9.08	18.65	30.94	0.92	0.60	20.96	5.64	5.36	9.08
3x16	18.47	30.89	0.89	1.04	26.20	5.78	5.62	4.38	18.59	30.92	0.91	0.74	22.66	5.69	5.44	7.56
16x40	33.09	47.98	0.13	0.13	3.48	1.79	1.36	4.89	26.78	40.56	0.47	0.40	11.82	3.48	3.14	6.05
40x150	32.30	46.84	0.09	0.13	1.43	1.69	1.36	11.56	31.07	45.44	0.17	0.19	3.75	2.09	1.76	10.33
150x635	32.95	48.54	0.56	0.72	3.07	1.37	1.12	2.09	31.48	46.12	0.26	0.31	3.60	1.93	1.62	8.53
-635	29.53	42.06	0.60	3.99	2.45	0.86	0.52	8.66	31.04	45.20	0.34	1.14	3.34	1.69	1.37	8.56
Total	31.04	45.20	0.34	1.14	3.34	1.69	1.37	8.56								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	2.39	2.72	10.89	2.10	24.93	13.25	15.55	4.22	2.39	2.72	10.89	2.10	24.93	13.25	15.55	4.22
3x16	1.13	1.30	5.04	1.74	14.92	6.50	7.81	0.97	3.52	4.02	15.93	3.84	39.84	19.76	23.35	5.19
16x40	8.13	8.10	2.95	0.87	7.95	8.08	7.58	4.36	11.65	12.12	18.88	4.71	47.79	27.83	30.93	9.55
40x150	48.92	48.71	12.60	5.37	20.12	46.98	46.67	63.49	60.57	60.83	31.48	10.08	67.91	74.82	77.60	73.03
150x635	17.95	18.16	28.19	10.71	15.54	13.70	13.83	4.13	78.52	78.99	59.67	20.79	83.45	88.52	91.43	77.16
-635	21.48	21.01	40.33	79.21	16.55	11.48	8.57	22.84	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

AS #7

AS #7	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	
Test No.	AS #7
% Solids in Attrition Scrubber	60
Time (min.)	2.5
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	26.00	5.51	5.51	94.49
3x16	1180	9.90	2.10	7.60	92.40
16x40	425	34.80	7.37	14.97	85.03
40x150	106	219.70	46.53	61.50	38.50
150x635	20	79.30	16.79	78.29	21.71
-635	6	102.50	21.71	100.00	0.00
Total		472.20	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	27.52	39.95	0.43	0.69	9.40	2.22	1.84	8.42	27.52	39.95	0.43	0.69	9.40	2.22	1.84	8.42
3x16	18.91	28.88	0.73	0.56	25.33	6.76	6.35	4.96	25.15	36.90	0.51	0.65	13.79	3.47	3.08	7.47
16x40	33.13	47.34	0.09	0.16	3.79	1.61	1.04	5.52	29.08	42.04	0.30	0.41	8.87	2.56	2.08	6.51
40x150	33.30	47.38	0.08	0.14	1.54	0.93	0.56	11.34	32.27	46.08	0.13	0.21	3.32	1.33	0.93	10.16
150x635	32.72	47.76	0.43	0.76	3.17	1.42	1.02	2.36	32.37	46.44	0.20	0.32	3.29	1.35	0.95	8.49
-635	29.98	42.86	0.62	4.08	2.51	0.94	0.63	9.27	31.85	45.66	0.29	1.14	3.12	1.26	0.88	8.66
Total	31.85	45.66	0.29	1.14	3.12	1.26	0.88	8.66								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	4.76	4.82	8.17	3.33	16.58	9.72	11.52	5.35	4.76	4.82	8.17	3.33	16.58	9.72	11.52	5.35
3x16	1.24	1.33	5.28	1.03	17.01	11.27	15.13	1.20	6.00	6.14	13.46	4.36	33.59	20.99	26.65	6.56
16x40	7.67	7.64	2.29	1.03	8.95	9.43	8.71	4.70	13.67	13.78	15.75	5.40	42.54	30.42	35.36	11.25
40x150	48.65	48.28	12.85	5.71	22.95	34.40	29.62	60.93	62.31	62.06	28.60	11.11	65.49	64.82	64.98	72.18
150x635	17.25	17.57	24.93	11.20	17.05	18.96	19.47	4.58	79.57	79.63	53.53	22.31	82.55	83.78	84.45	76.76
-635	20.43	20.37	46.47	77.69	17.45	16.22	15.55	23.24	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

AS #8

AS #8	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	
Test No.	AS #7
% Solids in Attrition Scrubber	60
Time (min.)	5
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	23.00	4.85	4.85	95.15
3x16	1180	11.10	2.34	7.18	92.82
16x40	425	39.60	8.34	15.53	84.47
40x150	106	225.50	47.51	63.04	36.96
150x635	20	75.60	15.93	78.97	21.03
-635	6	99.80	21.03	100.00	0.00
Total		474.60	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	23.19	36.13	1.31	0.49	13.92	1.39	1.07	6.94	23.19	36.13	1.31	0.49	13.92	1.39	1.07	6.94
3x16	19.25	29.17	0.72	0.63	30.51	6.69	6.32	5.54	21.91	33.86	1.12	0.54	19.32	3.12	2.78	6.48
16x40	34.18	49.00	0.10	0.15	3.72	1.47	1.02	5.16	28.50	42.00	0.57	0.33	10.94	2.23	1.83	5.77
40x150	34.53	49.82	0.09	0.18	1.52	0.89	0.47	7.14	33.05	47.89	0.21	0.22	3.84	1.22	0.81	6.80
150x635	32.99	47.98	0.57	0.81	2.98	1.39	0.97	1.97	33.03	47.91	0.28	0.34	3.67	1.25	0.84	5.83
-635	30.20	42.88	0.59	3.98	2.44	2.19	1.97	8.97	32.44	46.85	0.35	1.10	3.41	1.45	1.08	6.49
Total	32.44	46.85	0.35	1.10	3.41	1.45	1.08	6.49								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	3.46	3.74	18.33	2.15	19.79	4.64	4.82	5.18	3.46	3.74	18.33	2.15	19.79	4.64	4.82	5.18
3x16	1.39	1.46	4.86	1.34	20.93	10.78	13.73	2.00	4.85	5.19	23.20	3.49	40.73	15.42	18.54	7.18
16x40	8.79	8.73	2.41	1.14	9.11	8.45	7.90	6.64	13.64	13.92	25.60	4.63	49.83	23.87	26.44	13.81
40x150	50.58	50.52	12.35	7.76	21.19	29.14	20.74	52.28	64.22	64.44	37.95	12.38	71.02	53.01	47.18	66.10
150x635	16.20	16.31	26.22	11.70	13.93	15.26	14.35	4.84	80.42	80.75	64.17	24.09	84.95	68.27	61.53	70.93
-635	19.58	19.25	35.83	75.91	15.05	31.73	38.47	29.07	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

AS #9

AS #9	
Horizontal Scrubbing	
All samples were horizontally scrubbed at 36.8 RPM, 36% solids content for 5 minutes prior to the 6.3 x 0.075 mm size particles being subjected to attrition scrubbing.	

Test No.	AS #7
% Solids in Attrition Scrubber	60
Time (min.)	10
RPM Attri.	560

US Mesh	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %
3	6300	27.80	5.87	5.87	94.13
3x16	1180	8.90	1.88	7.75	92.25
16x40	425	34.60	7.31	15.06	84.94
40x150	106	224.80	47.48	62.53	37.47
150x635	20	74.50	15.73	78.27	21.73
-635	6	102.90	21.73	100.00	0.00
Total		473.50	100.00		

US Mesh	Grades								Cum. Grades							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	22.62	35.83	1.07	0.56	13.25	1.36	0.86	8.40	22.62	35.83	1.07	0.56	13.25	1.36	0.86	8.40
3x16	17.17	27.49	0.87	0.58	26.35	4.07	3.75	6.91	21.30	33.81	1.02	0.56	16.43	2.02	1.56	8.04
16x40	33.76	48.72	0.09	0.15	3.72	0.73	0.33	6.29	27.35	41.04	0.57	0.36	10.26	1.39	0.96	7.19
40x150	35.18	50.35	0.10	0.13	1.59	0.66	0.35	6.79	33.29	48.11	0.21	0.19	3.68	0.84	0.50	6.89
150x635	33.66	48.38	0.53	0.80	2.94	0.65	0.33	1.91	33.37	48.16	0.28	0.31	3.53	0.80	0.46	5.89
-635	30.38	43.33	0.60	3.98	2.38	0.35	0.03	9.10	32.72	47.11	0.35	1.11	3.28	0.70	0.37	6.58
Total	32.72	47.11	0.35	1.11	3.28	0.70	0.37	6.58								

US Mesh	Distribution								Cum. Distribution							
	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %	P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	S_{total}, %	$S_{pyritic}$, %	Insol, %
3	4.06	4.47	18.10	2.97	23.72	11.38	13.66	7.49	4.06	4.47	18.10	2.97	23.72	11.38	13.66	7.49
3x16	0.99	1.10	4.71	0.98	15.10	10.91	19.07	1.97	5.05	5.56	22.82	3.95	38.82	22.29	32.72	9.46
16x40	7.54	7.56	1.90	0.99	8.29	7.61	6.52	6.98	12.59	13.12	24.71	4.94	47.11	29.90	39.25	16.44
40x150	51.05	50.74	13.68	5.57	23.02	44.68	44.95	48.96	63.63	63.86	38.39	10.52	70.13	74.57	84.19	65.40
150x635	16.19	16.16	24.03	11.37	14.10	14.58	14.04	4.56	79.82	80.01	62.42	21.89	84.23	89.16	98.24	69.97
-635	20.18	19.99	37.58	78.11	15.77	10.84	1.76	30.03	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00								

Appendix D: Flotation

Summary of Test Conditions

Test	Reagent	Dose, ml	Dose, kg/ton	RPM	pH
FT #1*	CA-1208	2.4	0.470	1200	7.07
FT #2	CA-8032	1.2	0.227	1200	6.89
FT #3	CA-1260	1.2	0.228	1200	6.92
FT #4	CA-1208	2.4	0.465	1000	6.89
FT #5	CA-1208	4.8	0.976	1000	6.77
FT #6	CA-1208	3.6	0.725	1000	6.79
FT #7	CA-1208	6	1.168	1000	6.87

* Dose = two additions of 1.2 ml. Data for 1.2 ml of amine addition only.

Effect of Amine Type at Constant Dosage of 0.23 kg/ton

Amine Type	Weight grams	WT %	GRADES					RECOVERY	REJECTION			
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
CA-1208	249.40	97.65	34.41	7.13	0.18	1.60	0.09	97.71	0.73	11.69	2.44	4.83
CA-8032	262.10	99.36	34.02	7.44	0.21	1.52	0.08	99.40	0.43	4.65	1.31	2.53
CA-1260	262.90	99.73	33.97	6.91	0.20	1.64	0.10	99.75	0.15	2.11	0.30	0.84

FT #1

Product	Weight, g	Weight, %	GRADES					DISTRIBUTION				
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
1.18 x 106 Conc.	249.40	97.65	34.41	7.13	0.18	1.60	0.09	97.71	98.90	88.31	97.56	95.17
1.18 x 106 Tails	6.00	2.35	33.58	3.29	0.99	1.66	0.19	2.29	1.10	11.69	2.44	4.83
Calc. Head	255.40	100.00	34.39	7.04	0.20	1.60	0.09	100.00	100.00	100.00	100.00	100.00

FT #2

Product	Weight, g	Weight, %	GRADES					DISTRIBUTION				
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
1.18 x 106 Conc.	262.10	99.36	34.02	7.44	0.21	1.52	0.08	99.40	99.57	95.35	98.69	97.47
1.18 x 106 Tails	1.70	0.64	31.49	4.93	1.58	3.10	0.32	0.60	0.43	4.65	1.31	2.53
Calc. Head	263.80	100.00	34.00	7.42	0.22	1.53	0.08	100.00	100.00	100.00	100.00	100.00

FT #3

Product	Weight, g	Weight, %	GRADES					DISTRIBUTION				
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
1.18 x 106 Conc.	262.90	99.73	33.97	6.91	0.20	1.64	0.10	99.75	99.85	97.89	99.70	99.16
1.18 x 106 Tails	0.70	0.27	31.68	3.80	1.62	1.86	0.32	0.25	0.15	2.11	0.30	0.84
Calc. Head	263.60	100.00	33.96	6.90	0.20	1.64	0.10	100.00	100.00	100.00	100.00	100.00

FT #4

Product	Weight, g	Weight, %	GRADES					DISTRIBUTION				
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
1.18 x 106 Conc.	254.00	98.45	34.53	6.76	0.19	1.55	0.07	99.27	89.75	94.07	97.33	96.95
1.18 x 106 Tails	4.00	1.55	16.10	49.00	0.76	2.70	0.14	0.73	10.25	5.93	2.67	3.05
Calc. Head	258.00	100.00	34.24	7.41	0.20	1.57	0.07	100.00	100.00	100.00	100.00	100.00

FT #5

Product	Weight, g	Weight, %	GRADES					DISTRIBUTION				
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
1.18 x 106 Conc.	230.40	93.70	35.92	2.61	0.20	1.39	0.08	98.23	36.50	92.24	87.69	92.96
1.18 x 106 Tails	15.50	6.30	9.60	67.50	0.25	2.90	0.09	1.77	63.50	7.76	12.31	7.04
Calc. Head	245.90	100.00	34.26	6.70	0.20	1.49	0.08	100.00	100.00	100.00	100.00	100.00

FT #6

Product	Weight, g	Weight, %	GRADES					DISTRIBUTION				
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
1.18 x 106 Conc.	237.60	95.73	35.56	4.63	0.20	1.65	0.09	98.77	60.83	91.81	93.06	94.83
1.18 x 106 Tails	10.60	4.27	9.93	66.83	0.40	2.76	0.11	1.23	39.17	8.19	6.94	5.17
Calc. Head	248.20	100.00	34.47	7.29	0.21	1.70	0.09	100.00	100.00	100.00	100.00	100.00

FT #7

Product	Weight, g	Weight, %	GRADES					DISTRIBUTION				
			P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %	P_2O_5, %	Insol, %	Al_2O_3, %	Fe_2O_3, %	MgO, %
1.18 x 106 Conc.	234.30	91.24	36.70	2.20	0.16	1.48	0.08	97.30	26.64	89.52	82.98	89.08
1.18 x 106 Tails	22.50	8.76	10.60	63.10	0.20	3.16	0.10	2.70	73.36	10.48	17.02	10.92
Calc. Head	256.80	100.00	34.41	7.54	0.17	1.63	0.08	100.00	100.00	100.00	100.00	100.00

Appendix E: QEMSCAN Report

Photo of samples sent to SGS for QEMSCAN testing:

An Investigation into

MINERALOGICAL CHARACTERISTICS OF ONE PHOSPATE COMPOSITE SAMPLE

prepared for

LYCOPODIUM MINERALS CANADA LTD.

Project 13478-003 · Final Report
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Executive Summary

One feed composite sample labelled Farim Comp was submitted to the Mineral Services group within SGS for mineralogical characterization using QEMSCAN technology, chemical analysis, electron microprobe analysis (EMPA), and X-Ray Diffraction (XRD). This mineralogical characterization was originally requested by Marten Walters, from KEMWorks Technology, on behalf of Lycopodium Minerals Canada Ltd. The objective of this investigation was to determine the mineral assemblage of each sample, the liberation characteristics of the apatite, silicates, carbonates, oxides, and sulphides.

To aid with this objective, the deliverables from this size by size mineralogical study include:

- the mineral abundance of the sample (by size fraction),
- the liberation and association information of total apatite, silicates, oxides, sulphides, and carbonate minerals,
- determinative mineralogical parameters such as:
    - mineral release curves,
    - mineralogically limiting grade recovery curves, and
- grain size data.

The sample preparation and the details of the results are discussed in the main body of the report. Some points of interest are discussed in this summary.

- **Mass Distributions and Elemental Chemical Data**

The mass distributions and elemental chemical data by size fraction are summarized in Table 1. Note the higher abundance of aluminum and silicate in the -20 µm fraction and the much higher concentration of iron in the +1,180 µm fraction.

Table 1: Size Fractions for Analysis and Mass Distribution (%) of the Farim Comp

Fraction	Combined	+1180µm	-1180/+425µm	-425/+106µm	-106/+20µm	-20µm
Mass Size Distribution (%)	**100.0**	**15.5**	**19.3**	**26.0**	**15.4**	**23.8**
Mg (Chemical)	0.25	0.56	0.07	0.05	0.27	0.39
Al (Chemical)	0.70	0.39	0.13	0.12	0.40	2.20
Si (Chemical)	3.26	3.13	2.21	3.73	1.77	4.67
P (Chemical)	13.0	6.59	14.6	14.8	14.7	12.8
S (Chemical)	1.20	2.17	1.46	0.76	1.21	0.82
K (Chemical)	0.04	0.02	0.01	0.01	0.02	0.11
Ca (Chemical)	31.1	16.9	34.6	34.9	35.3	30.4
Fe (Chemical)	4.88	22.0	2.88	0.93	1.87	1.57

- **Mineral Abundances**

A summary of the mineral abundances is discussed below.

- ***Calculated Head***
  - The apatite content is 74.4%.
  - The "Apatite Impure" category accounts for 12.8% and predominately occurs in the -20 µm size fraction.
  - The gangue minerals are mainly:
    - *quartz (3.13 wt%)*
    - *Fe-oxides (5.58 wt%)*
    - *dolomite (0.50 wt%)*
    - *pyrite (2.83 wt%).*

- ***Size by Size Mineral Distributions***
  - Apatite abundance is highest in the +106 µm size fraction (91.2%) and the least in the -20 µm size fraction (48.3%).
  - The Fe-oxide content is much higher in the +1,180 µm fraction and accounts for ~28% by mass. This correlates well with the higher iron assay in this fraction.
  - Pyrite content is also highest in the +1,180 µm fraction and also correlated well with the sulphur assay.
  - The apatite impure phase is mainly composed of Ca-phosphate but it can have high levels of impurities. Aluminum and silica are the main ones bit it can also contain low levels of potassium & magnesium. This phase mainly occurs in the -20 µm fraction accounting for 48.9%.

- **EMPA**

The data from the electron microprobe analysis (EMPA) indicates that the average P_2O_5 content of the apatite is 37.21%. If a perfect concentrate of apatite was produced, this would be close to the maximum P_2O_5 grade that could be achieved. The EMPA also reveals that apatite contains significant SO_2 and fluorine at ~0.65% and 4.72%, respectively.

- **Liberation and Grain Size**

The liberation of the "Apatite Total" (which combines the apatite and apatite impure as one mineral group) is good, accounting for 96% (both "free" and "liberated" combined) of the calculated head. With the exception of the +1,180 µm size fraction, apatite liberation is very good in each of the other fractions. The non-liberated apatite particles are generally associated with the complex mineral class.

The calculated head for the carbonate liberation is poor, at 28%. The size by size liberation profiles of the carbonates shows poor liberation at the coarser sizes. Liberation generally increases with decreasing particle size. The non-liberated carbonate grains are commonly associated with the complex grains.

The liberation of the silicates for the comp is good, accounting for 77% (both “free” and “liberated” combined) of the calculated head. The liberation is poor in the +1,180 µm size fraction (13%) but is good in the remaining size fractions.

By mass, the oxide and sulphide are most abundant in the +1,180 µm size fraction and show poor liberation.

- **Grade-Recovery**

Grade-recoveries are calculated based on the liberation and chemistry (EMPA) of apatite. The mineralogical limiting grade recovery curves indicate that an 80% apatite recovery for a theoretical maximum P_2O_5 concentrate grade of 36%, respectively, would be possible at this grind target.

Introduction

This report describes a test program using a combination of mineralogical techniques including the QEMSCAN technology (Quantitative Evaluation of Materials by Scanning Electron Microscopy), X-Ray Diffraction (XRD), and electron microprobe analysis (EMPA). The QEMSCAN analysis was conducted to quantify the mineral abundance, determine the elemental contributors of phosphorous, silica, sulphur, and calcium, liberation characteristics of phosphates and gangue phases, and the theoretically achievable best concentrate grades.

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Testwork Summary

1. Sample Preparation and QEMSCAN Analysis Mode

The sample was labelled Farim Comp and received as five size fractions and the mass distributions were provided by KEMWorks. The size fractions and respective mass distributions are presented in Table 2. Sub-samples of each fraction were also submitted for chemical analyses for data validation purposes and program setup. They include WRA (whole rock analysis of major elements including silica, aluminum, iron, magnesium, calcium, potassium, titanium, manganese, chromium, and phosphorous) by X-Ray Fluorescence (XRF), and sulphur by Leco. The chemical results presented as oxides are summarized in Table 3, and the certificate of analysis is presented in Appendix C.

Additional sub-samples of each fraction were used to prepare graphite-impregnated polished epoxy grain mounts for mineralogical analyses with the QEMSCAN Particle Map Analysis (PMA) mode of measurement. The PMA is a particle mapping measurement which gives a complete analysis of the mineralogy of the sample. It allows for a robust determination of the bulk mineralogy, with mineral identities and proportions, along with average grain size measurements. The PMA mode also provides an analysis of the spatial characteristics of minerals, including liberation, association, and grain size distribution, and it allows for determinative mineralogical analyses such as mineral release and grade-recovery curves.

The QEMSCAN calculated assay and the direct chemical assays from the WRA were compared as a quality control measure for each of the products, as presented in Figure 1
and Appendix A. All of the data that was generated from the QEMSCAN analysis are also presented in Appendix A.

Table 2: Sample Inventory, Size Fractions for Analysis and Mass Distribution (%)

Size Fraction	Farim Comp - Mass Distribution (%)
+1180 µm	15.5
-1180/+425 µm	19.3
-425/+106 µm	26.0
-106/+20 µm	15.4
-20 µm	23.8
Total	100

Table 3: Summary of the WRA and Sulphur Analysis by Size Fraction

Farim Comp	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	S %
+1180 µm	6.7	0.74	31.5	0.93	23.6	0.1	0.03	0.05	15.1	0.78	0.04	0.04	2.17
-1180/+425 µm	4.72	0.25	4.12	0.12	48.4	0.18	< 0.01	0.01	33.4	0.1	0.02	0.02	1.46
-425/+106 µm	7.98	0.22	1.33	0.08	48.9	0.17	< 0.01	0.01	33.8	0.03	0.03	0.03	0.76
-106/+20 µm	3.79	0.76	2.67	0.44	49.4	0.19	0.03	0.05	33.7	0.04	0.05	0.03	1.21
-20 µm	9.98	4.16	2.24	0.64	42.6	0.17	0.13	0.17	29.4	0.02	0.14	0.07	0.82

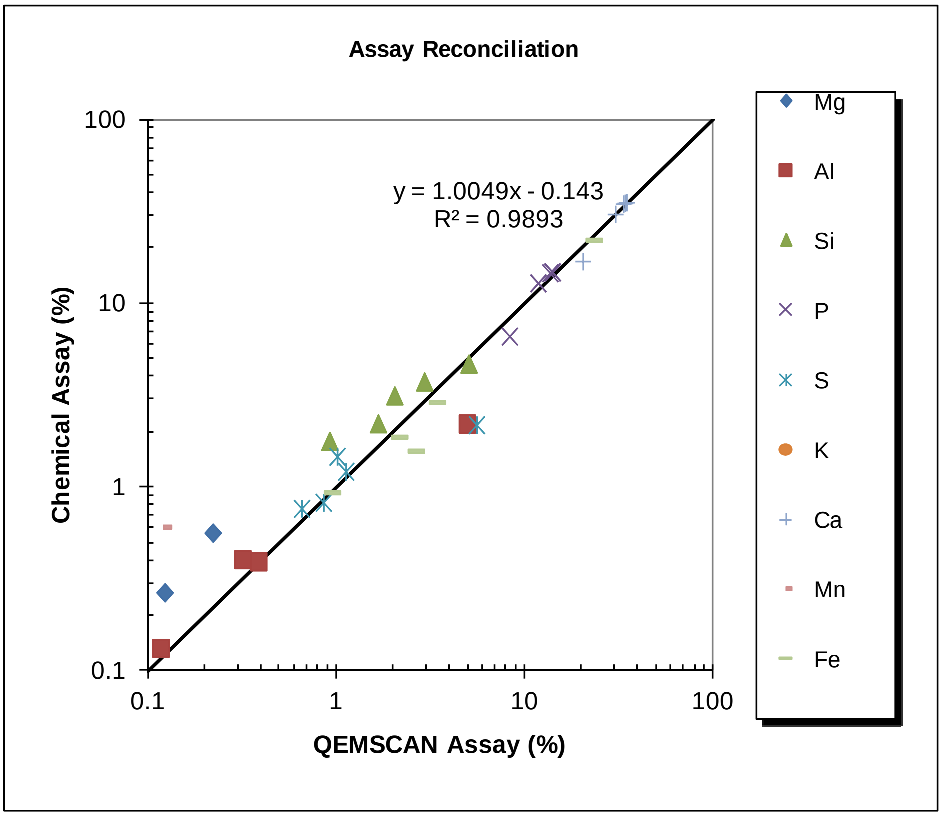


Figure 1: QEMSCAN Calculated vs. Chemical Assays for Each Size Fraction

2. X-Ray Diffraction (XRD) Analysis of the +425 µm and -20 µm Size Fractions

An additional aliquot of the +425 µm and -20 µm size fractions were also submitted for X-Ray Diffraction analysis to aid with gangue mineral speciation. Results are summarized in Table 4 and the full XRD report, including patterns and analytical conditions are presented in Appendix B. The results of each fraction are similar and include major amounts of apatite and minor quantities of quartz.

Table 4: Summary of the X-Ray Diffraction Analysis

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample ID	Major	Moderate	Minor	Trace
(13) Farim Comp -1180/+425un	apatite	-	quartz	-
(27) Farim Comp -20um	apatite	-	quartz	-

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

3. Discussion of Certain Mineral Phases and the Electron Microprobe Analysis (EMPA)

A mineral list generated with the iExplorer software, and the results of the electron microprobe analysis (EMPA) are discussed in this section: The deliverables include:

- the mineral abundances of each of the samples (by size fraction),
- the liberation and association of the total apatite, silicates, and carbonates,
- determinative mineralogical parameters,
- mineral release curves,
- mineralogically limiting grade recovery curves, and
- grain size data.

This data are discussed separately in Section 4 and all of the QEMSCNA Data is presented in Appendix A. A mineral list, developed with the QEMSCAN processing software iExplorer, and theoretical chemistries of each of the observed mineral phases are presented in Table 5.

Table 5: Mineral List Created with the iExplorer Software

Mineral	General Chemical Formula
Apatite	$Ca_5(PO_4)_3(F,Cl,OH)$,
Apatite Impure	$Ca_5(PO_4)_3(F,Cl,OH)$, with Mg and Si impurities
Quartz	SiO_2
Mica	$KAl_2(AlSi_3O_{10})(F,OH)_2$
Dolomite	$CaMg(CO_3)_2$
Calcite	$CaCO_3$
Fe-Oxides	*Magnetite* - Fe_3O_4, *Hematite* - Fe_2O_3 *Goethite* - $\alpha FeO{\cdot}OH$
Pyrite	FeS_2
Sphalerite	ZnS
Fe-Ca-Sulphate	(Fe,Ca)SO4*(H2O)
Gypsum	CaSO4*(H2O)

It should be noted that the energy dispersive X-ray characteristics for magnetite, hematite, and goethite are nearly identical. Therefore, these minerals cannot be distinguished reliably by QEMSCAN and only a total Fe-oxide abundance has been calculated.

The investigation revealed an impure phase, referred to as “Apatite Impure” (Table 5). This phase is mainly composed of calcium and phosphorous but it can have varied amounts of impurities, mainly silica and aluminum. However, by mass and abundance, this phase is most significant in the -20 µm size fraction.

Each of size fractions also subjected to an electron microprobe analysis (EMPA) to quantify the composition of the various mineral phases. Approximately 56 grains of “Apatite” and 5 grains of “Apatite Impure” were analyzed. A summary of the results is presented in Table 6 and Table 7, respectively. The average P_2O_5 content of the apatite is 15.86% (Or 37.21% P_2O_5). If a perfect concentrate of apatite were produced, then it would be close to the maximum P_2O_5 grade. Note that the SO_2 content averages ~0.65%, and fluorine 5.12%. These impurities will not be physically separated during processing of the ore and would be part of the concentrate.

The average values of phosphorous, silica, magnesium and calcium of the “Apatite Impure” are 11.25%, 4.2%, 0.03%, and 25.14%, respectively, but they range widely. Additional data from the EMPA is presented Appendix E.

Table 6: Summary of the Electron Microprobe Data for Apatite

	P	Si	S	Al	La	Ce	Mg	Ca	Mn	Fe	Na
Average	15.86	0.40	0.33	0.21	0.02	0.02	0.05	36.29	0.02	0.25	0.13
Max	17.27	5.40	0.71	3.22	0.07	0.08	0.46	40.57	0.17	1.12	0.35
Min	9.57	0.00	0.18	0.00	0.00	0.00	0.00	21.44	0.00	0.06	0.05
Std Dev	1.71	1.11	0.11	0.64	0.02	0.02	0.08	3.95	0.03	0.18	0.07

Table 7: Summary of the Electron Microprobe Data for Apatite (Impure)

	P	Si	S	Al	La	Ce	Mg	Ca	Mn	Fe	Na
Average	11.25	4.20	0.29	2.41	0.01	0.00	0.30	25.14	0.00	0.77	0.10
Max	14.69	5.40	0.34	3.22	0.01	0.02	0.46	32.27	0.00	1.12	0.10
Min	9.57	1.68	0.25	0.92	0.01	0.00	0.15	21.44	0.00	0.42	0.09
Std Dev	2.33	1.73	0.04	1.02	0.00	0.01	0.15	4.85	0.00	0.32	0.01

4. Mineralogical Details

4.1. Mineral Abundances

The size by size mineral distributions for Farim Comp and the calculated head sample are shown in Table 8 and graphically presented in Figure 2.

A summary of the mineral abundances is discussed below.

- ***Calculated Head***
  - The apatite content is 74.4%.
  - The "Apatite Impure" category accounts for 12.8% and predominately occurs in the -20 µm size fraction.
  - The gangue minerals are mainly:
    - *quartz (3.13 wt%)*
    - *Fe-Oxides (5.58 wt%)*
    - *dolomite (0.50 wt%)*
    - *pyrite (2.83 wt%).*
- ***Size by Size Mineral Distributions***
  - Apatite abundance is highest in the +106 µm size fraction (91.2%) and the least in the -20 µm size fraction (48.3%).
  - The Fe-oxide content is much higher in the +1,180 µm fraction and accounts for ~28%. This correlates well with the higher iron assay also in this fraction.
  - Pyrite content is also highest in this fraction and also correlated well with the sulphur assay.
  - The apatite impure phase mainly occurs in the -20 µm fraction at 48.9%.

Table 8: Mineral Distributions by Size Fraction

Survey		**13478-003 / MI5021-MAR14**										
Project		**Lycopodiunm**										
Sample		**Farim Comp**										
Fraction		**Combined**	**+1180um**		**-1180/+425um**		**-425/+106um**		**-106/+20um**		**-20um**	
Mass Size Distribution (%)			15.5		19.3		26.0		15.4		23.8	
Calculated ESD Particle Size		33	1241		405		126		33		10	
		Sample	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**	**Sample**	**Fraction**
Mineral Mass (%)	Apatite	74.4	8.17	52.6	17.1	88.6	23.7	91.2	14.0	90.8	11.5	48.3
	Apatite Impure	12.8	0.40	2.56	0.18	0.92	0.14	0.55	0.46	2.96	11.6	48.9
	Quartz	3.13	0.55	3.57	0.65	3.35	1.60	6.14	0.20	1.31	0.13	0.56
	Mica	0.04	0.03	0.17	0.00	0.02	0.00	0.01	0.00	0.01	0.00	0.01
	Other Silicates	0.12	0.07	0.48	0.01	0.07	0.01	0.05	0.01	0.07	0.01	0.04
	Dolomite	0.50	0.24	1.57	0.05	0.24	0.02	0.09	0.14	0.93	0.04	0.18
	Calcite	0.18	0.05	0.33	0.07	0.35	0.04	0.14	0.01	0.05	0.02	0.07
	Fe-Oxides	5.58	4.33	27.9	0.77	4.02	0.15	0.57	0.26	1.72	0.06	0.26
	Other Oxides	0.04	0.00	0.03	0.00	0.01	0.00	0.01	0.00	0.03	0.03	0.11
	Pyrite	2.83	1.57	10.1	0.30	1.58	0.31	1.19	0.32	2.06	0.33	1.37
	Sphalerite	0.03	0.01	0.09	0.00	0.01	0.00	0.00	0.00	0.01	0.01	0.03
	Fe-Ca-Sulphate	0.12	0.06	0.39	0.01	0.06	0.00	0.01	0.01	0.04	0.04	0.18
	Gypsum	0.17	0.02	0.15	0.13	0.66	0.01	0.03	0.00	0.00	0.00	0.02
	Other	0.04	0.01	0.04	0.02	0.09	0.00	0.01	0.00	0.01	0.01	0.03
	Total	100.0	15.5	100.0	19.3	100.0	26.0	100.0	15.4	100.0	23.8	100.0
Mean Grain Size by Frequency (µm)	Apatite	34	249		291		98		31		8	
	Apatite Impure	6	45		39		14		9		6	
	Quartz	78	141		241		193		51		6	
	Mica	18	32		31		13		7		4	
	Other Silicates	20	36		34		23		14		4	
	Dolomite	35	89		93		47		28		8	
	Calcite	19	31		34		17		7		8	
	Fe-Oxides	104	181		151		41		24		6	
	Other Oxides	5	35		31		12		9		4	
	Pyrite	36	141		97		60		25		7	
	Sphalerite	24	83		56		14		19		10	
	Fe-Ca-Sulphate	9	55		42		16		9		4	
	Gypsum	72	94		129		67		7		6	
	Other	10	30		34		12		6		4	

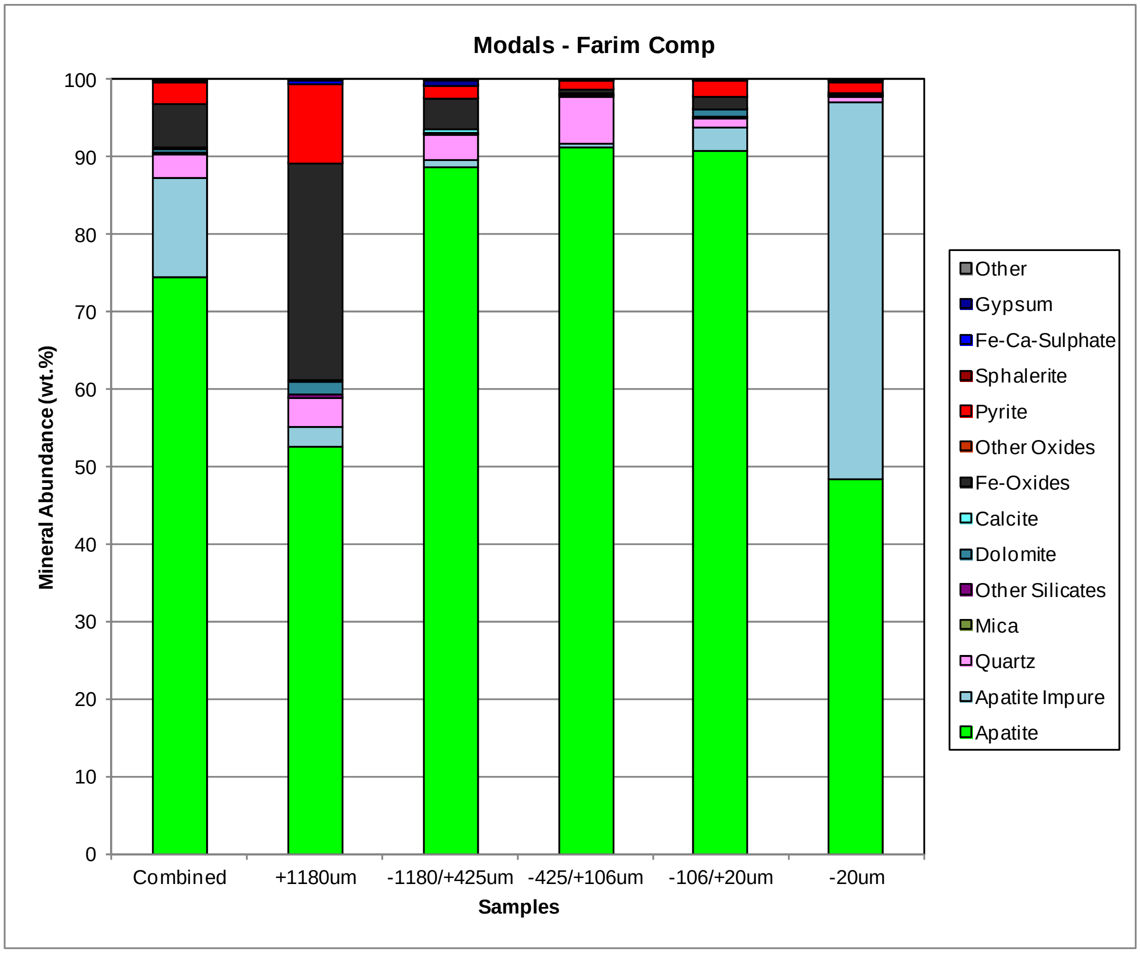
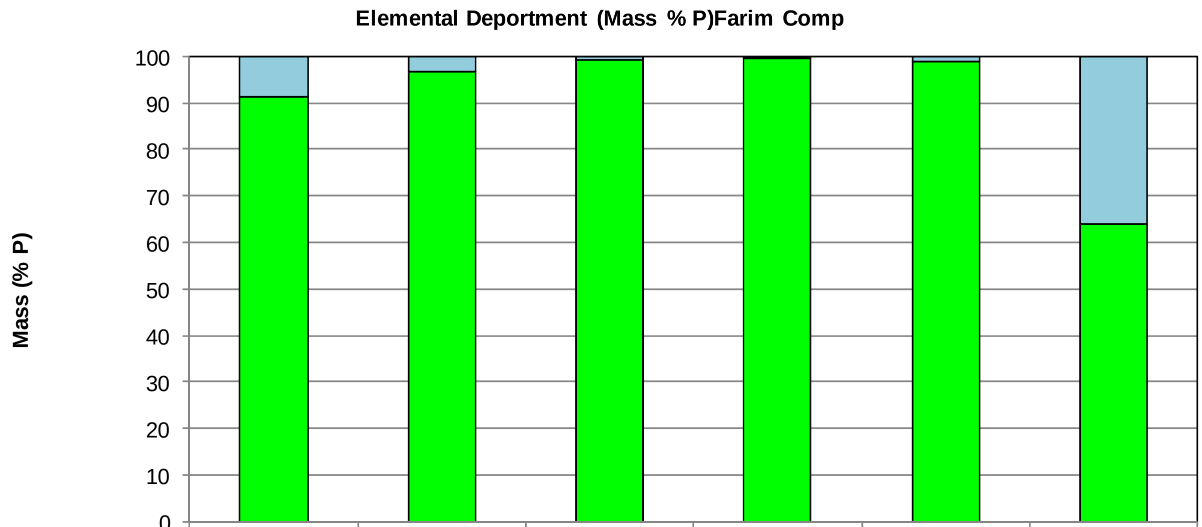


Figure 2: Graphical Display of the Normalized Mineral Distribution by Size Fraction

5. Elemental Deportment

Phosphorous elemental deportment for the calculated head and each size fractions is presented in Figure 3. Full results are presented in Appendix A.

In the calculated head, phosphorous is carried dominantly by apatite (91%), followed by apatite impure (8.75%). Throughout the size fractions, apatite is the prevailing phosphorous carrier but ranges 99% in the -1,180/+425 µm size fraction to 63% in the -20 µm size fraction.



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Other	0.00	0.00	0.00	0.00	0.00	0.01
Apatite Impure	8.75	3.28	0.70	0.34	1.27	36.1
Apatite	91.2	96.7	99.3	99.7	98.7	63.9

Figure 3: Phosphorous Elemental Deportment (Normalized Mass%)

6. Liberation and Association Data

For the purposes of this analysis, particle liberation is defined based on 2D particle area percent. In liberation analysis, particles are classified in the following groups (in descending order) based on mineral-of-interest area percent: free (>=95%), liberated (<95% and >=80%), middling (<80% and >=50%), sub-middling (<50% and >=20%) and locked (<20%). The mineral association data combines the middling and locked groups, sorting into binary association categories and complex groups. Binary association categories, for example Apatite:Silicates refer to particle area percent greater than or equal to 95% of the mineral groups. The complex group refers to particles with a combination of three or more minerals, including the mineral of interest. Definitions for the liberation and association are expressed in greater detail in Appendix D.

It should be noted, the “Apatite Total” class includes both the Apatite and “Apatite Impure” as one mineral entity.

Liberation and association graphs are also presented for:

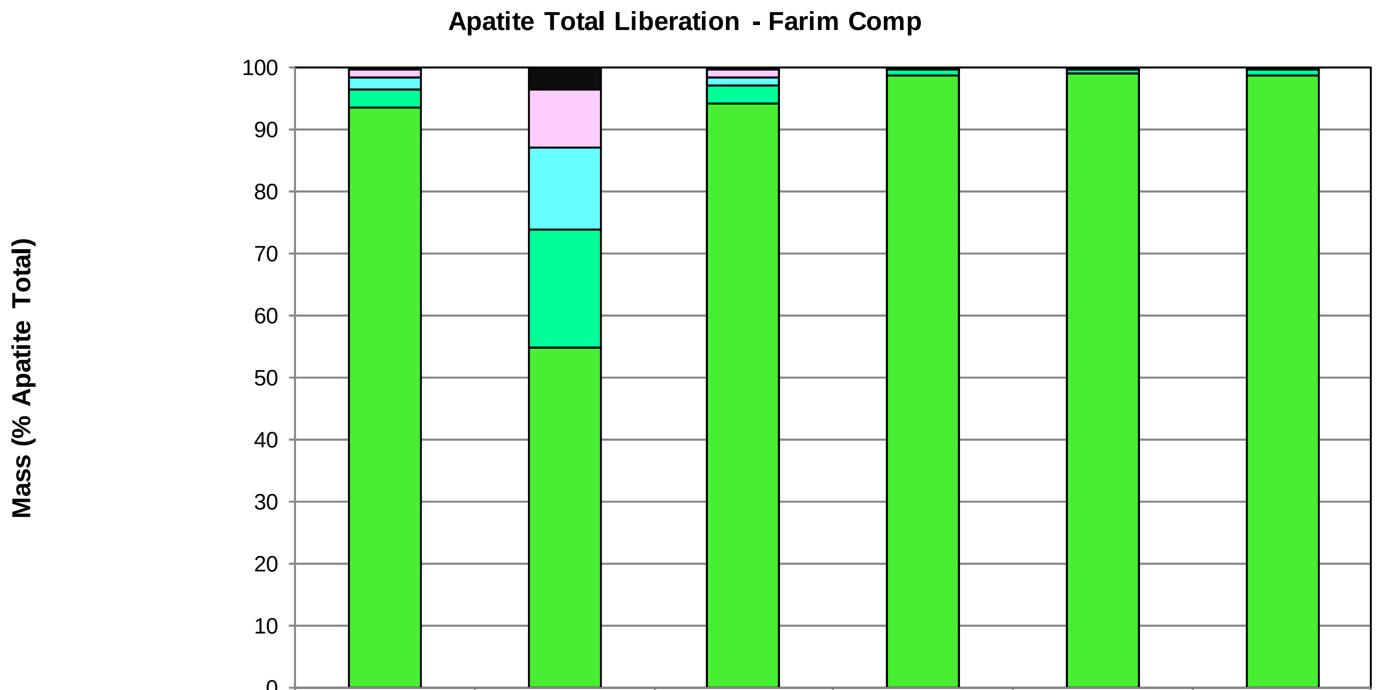
- Apatite Total - Figure 4 and Figure 5
- Carbonate - Figure 6 and Figure 7
- Silicates - Figure 8 and Figure 9
- Oxides - Figure 10 and Figure 11
- Sulphides - Figure 12 and Figure 13

In summary, the liberation of the “Apatite Total” for Farim Comp is good, accounting for 96% (both “free” and “liberated” combined) of the calculated head. The liberation is also good throughout each size fraction. The non-liberated apatite particles are generally associated with the complex mineral class.

In contrast, the calculated head for the carbonate liberation is poor, at 28%. The size by size liberation profiles of the carbonates shows poor liberation at coarse sizes. Liberation generally increases with decreasing particle size. The non-liberated carbonate grains are commonly associated with the complex grains.

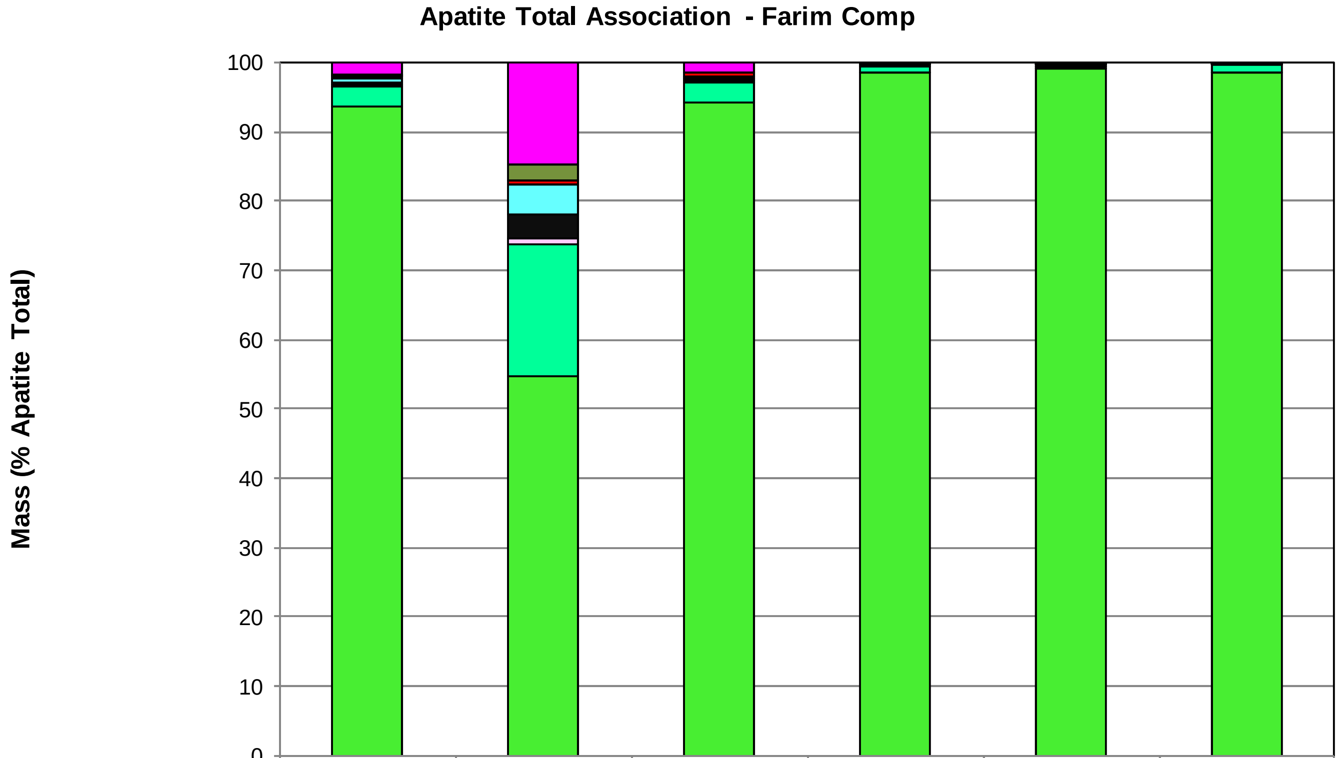
The liberation of the silicates for the comp is good, accounting for 77% (both “free” and “liberated” combined) of the calculated head. The liberation is poor in the +1,180 µm size fraction (13%) but is good in the remaining size fractions.

By mass, the oxide and sulphide are most abundant in the +1,180 µm size fraction and show poor overall liberation.



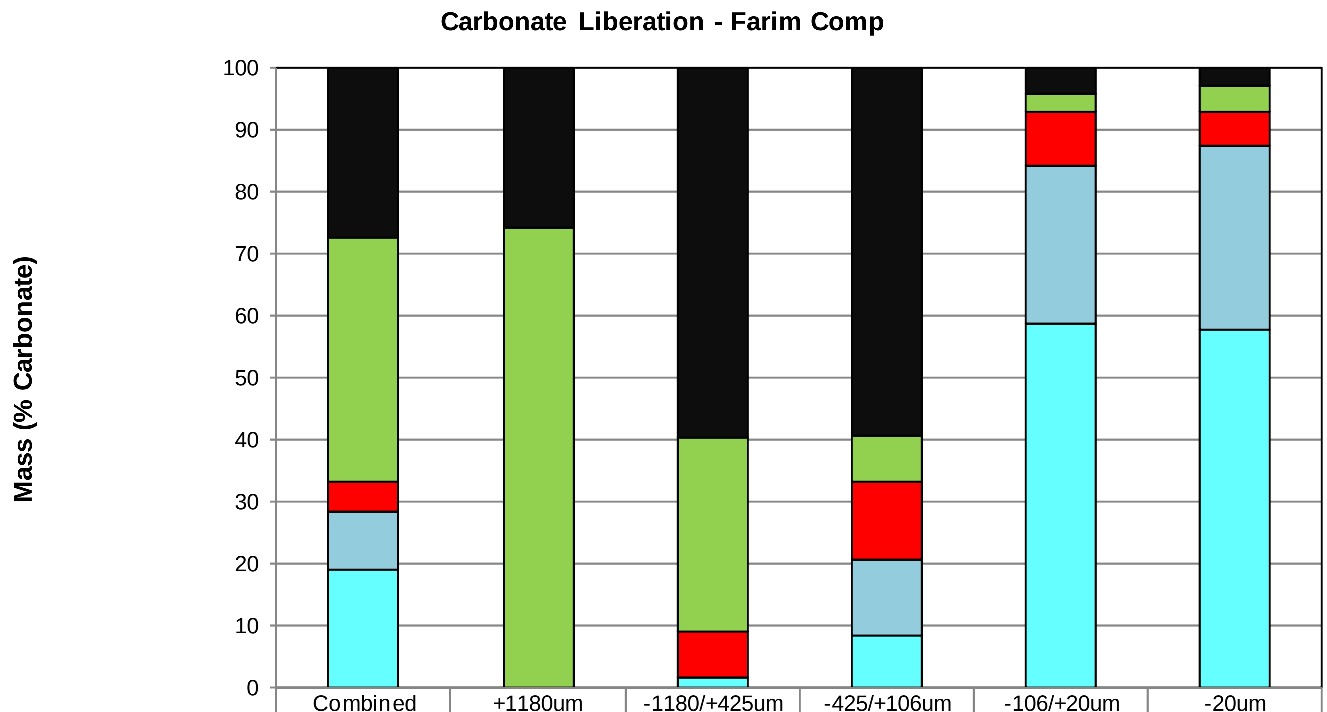
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Apatite Total	0.47	3.47	0.33	0.11	0.13	0.03
Sub-middling Apatite Total	1.26	9.44	1.23	0.14	0.17	0.08
Middling Apatite Total	1.76	13.4	1.36	0.27	0.20	0.25
Liberated Apatite Total	3.05	19.0	2.94	0.97	0.39	1.03
Free Apatite Total	93.5	54.7	94.1	98.5	99.1	98.6

Figure 4: Liberation Profile of the "Apatite Total" by Size Fraction (Normalized Distribution)



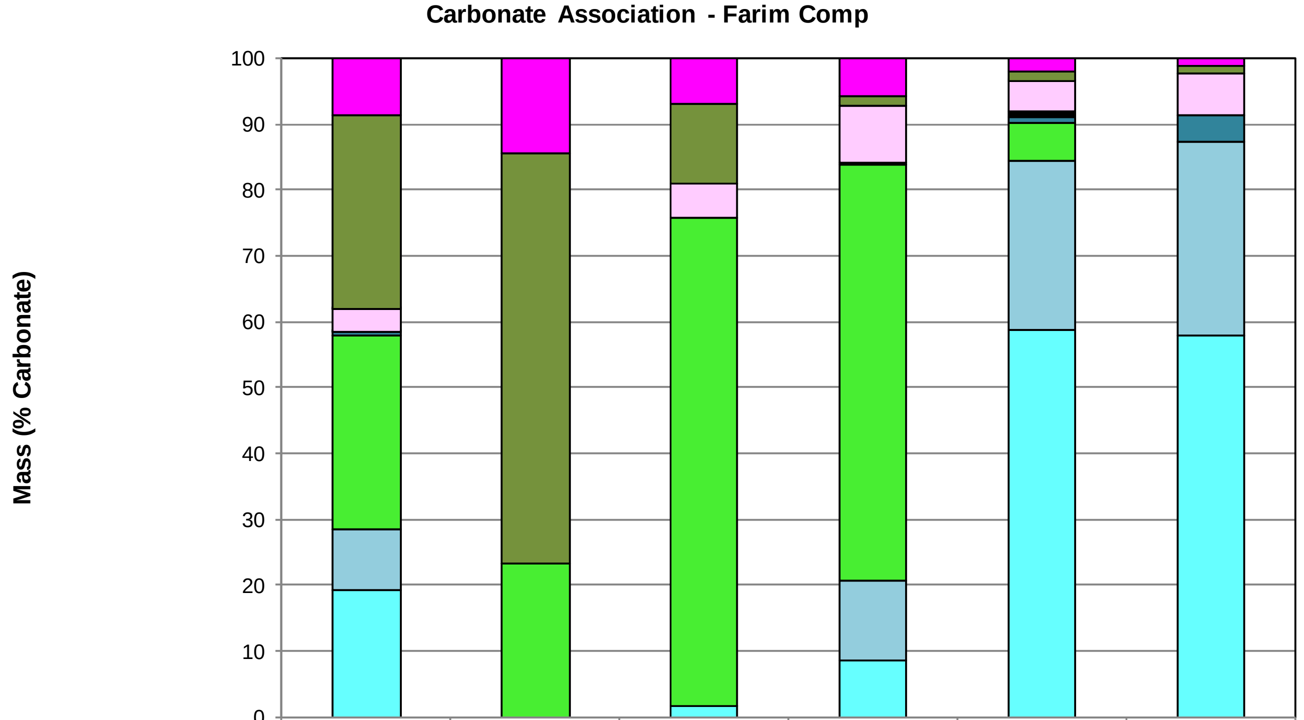
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	1.79	14.6	1.45	0.08	0.12	0.10
Apatite Total:Sil:Carb	0.24	2.44	0.00	0.00	0.00	0.00
Apatite Total:Other	0.00	0.00	0.00	0.00	0.00	0.00
Apatite Total:Sulphides	0.29	0.70	0.52	0.22	0.12	0.15
Apatite Total:Carbonates	0.52	4.19	0.34	0.06	0.13	0.03
Apatite Total:Oxides	0.45	3.46	0.43	0.02	0.06	0.05
Apatite Total:Silicates	0.18	0.91	0.17	0.14	0.08	0.03
Liberated Apatite Total	3.05	19.0	2.94	0.97	0.39	1.03
Free Apatite Total	93.5	54.7	94.1	98.5	99.1	98.6

Figure 5: Association Profile of the "Apatite Total" by Size Fraction (Normalized Distribution)



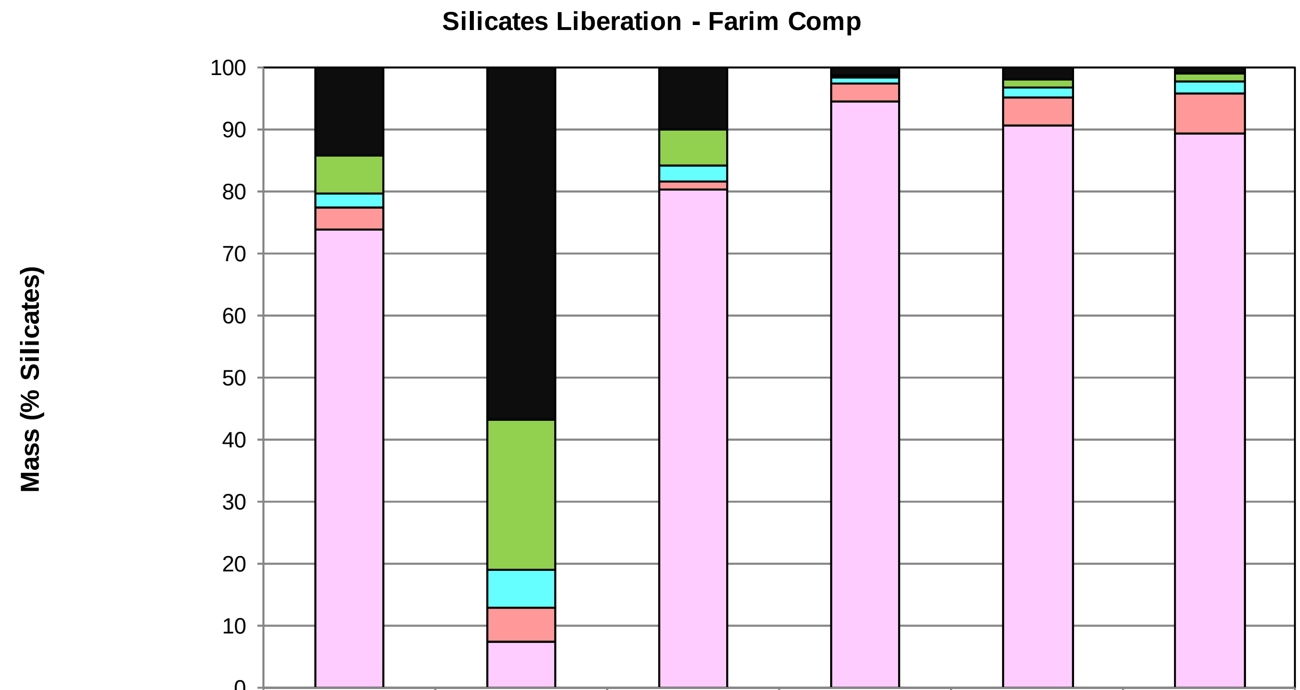
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Carbonate	27.6	25.7	59.7	59.4	4.35	2.85
Sub-middling Carbonate	39.2	74.3	31.2	7.45	2.82	4.21
Middling Carbonate	4.75	0.00	7.37	12.6	8.67	5.55
Liberated Carbonate	9.33	0.00	0.00	12.2	25.6	29.6
Free Carbonate	19.1	0.00	1.70	8.25	58.6	57.8

Figure 6: Liberation Profile of the Carbonates by Size Fraction (Normalized Distribution)



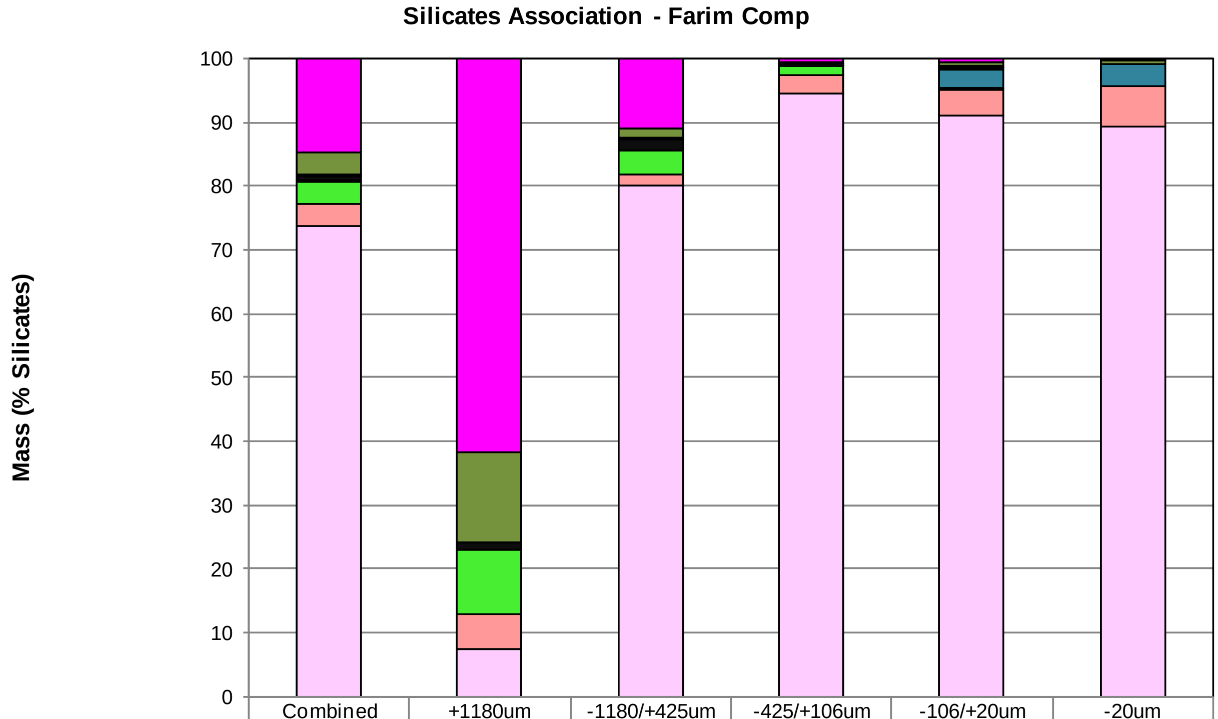
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	8.58	14.6	6.91	5.70	2.12	1.31
Carb: Sil: Ap	29.6	62.1	12.3	1.54	1.41	1.19
Carbonate: Other	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Sulphides	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Silicates	3.21	0.00	5.16	8.71	4.64	6.32
Carbonate: Oxides	0.20	0.00	0.00	0.29	0.78	0.00
Carbonate: Apatite Impure	0.55	0.00	0.00	0.00	1.00	3.79
Carbonate:Apatite	29.4	23.3	73.9	63.2	5.77	0.00
Liberated Carbonate	9.30	0.00	0.00	12.0	25.6	29.6
Free Carbonate	19.1	0.00	1.70	8.61	58.7	57.8

Figure 7: Association Profile of the Carbonates by Size Fraction (Normalized Distribution)



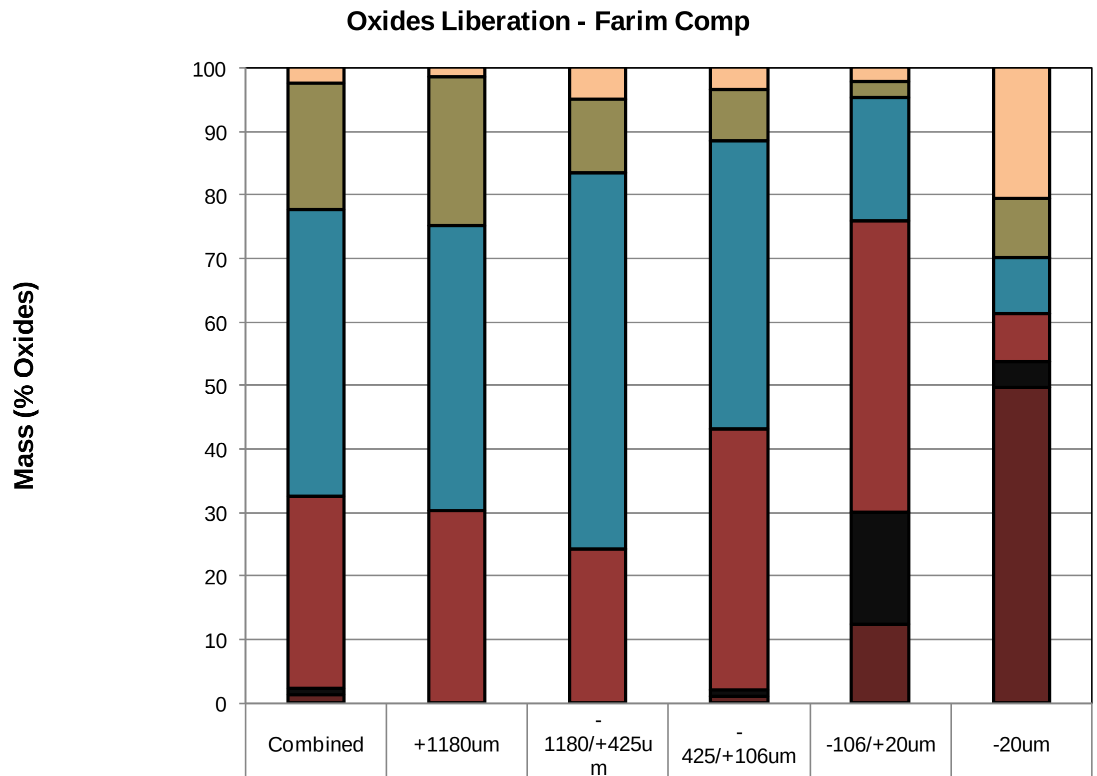
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Silicates	14.1	56.9	10.1	1.18	1.97	0.89
Sub-middling Silicates	6.38	24.2	5.61	0.58	1.39	1.37
Middling Silicates	2.26	5.90	2.59	0.74	1.58	2.10
Liberated Silicates	3.42	5.63	1.50	2.92	4.40	6.37
Free Silicates	73.8	7.36	80.2	94.6	90.7	89.3

Figure 8: Liberation Profile of the Silicates by Size Fraction (Normalized Distribution)



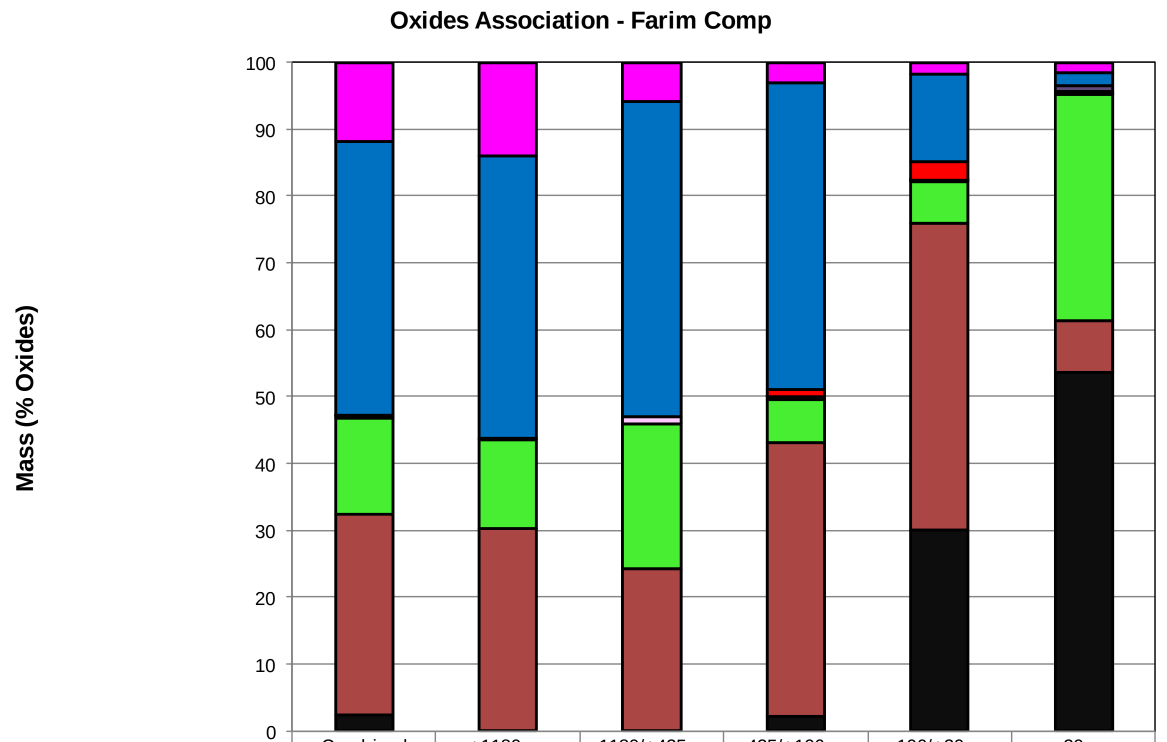
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	14.9	61.8	10.9	0.59	0.70	0.24
Sil:Carb:Ap	3.36	14.2	1.62	0.27	0.64	0.74
Silicates:Other	0.00	0.00	0.00	0.00	0.00	0.00
Silicates:Sulphides	0.18	0.00	0.32	0.20	0.17	0.00
Silicates:Carbonates	0.01	0.00	0.00	0.00	0.04	0.00
Silicates:Oxides	0.60	1.03	1.73	0.07	0.19	0.00
Silicates:Apatite Impure	0.37	0.00	0.00	0.07	2.96	3.37
Silicates:Apatite	3.40	10.0	3.76	1.29	0.25	0.00
Liberated Silicates	3.40	5.63	1.53	2.92	4.04	6.37
Free Silicates	73.8	7.36	80.2	94.6	91.0	89.3

Figure 9: Association Profile of the Silicates by Size Fraction (Normalized Distribution)



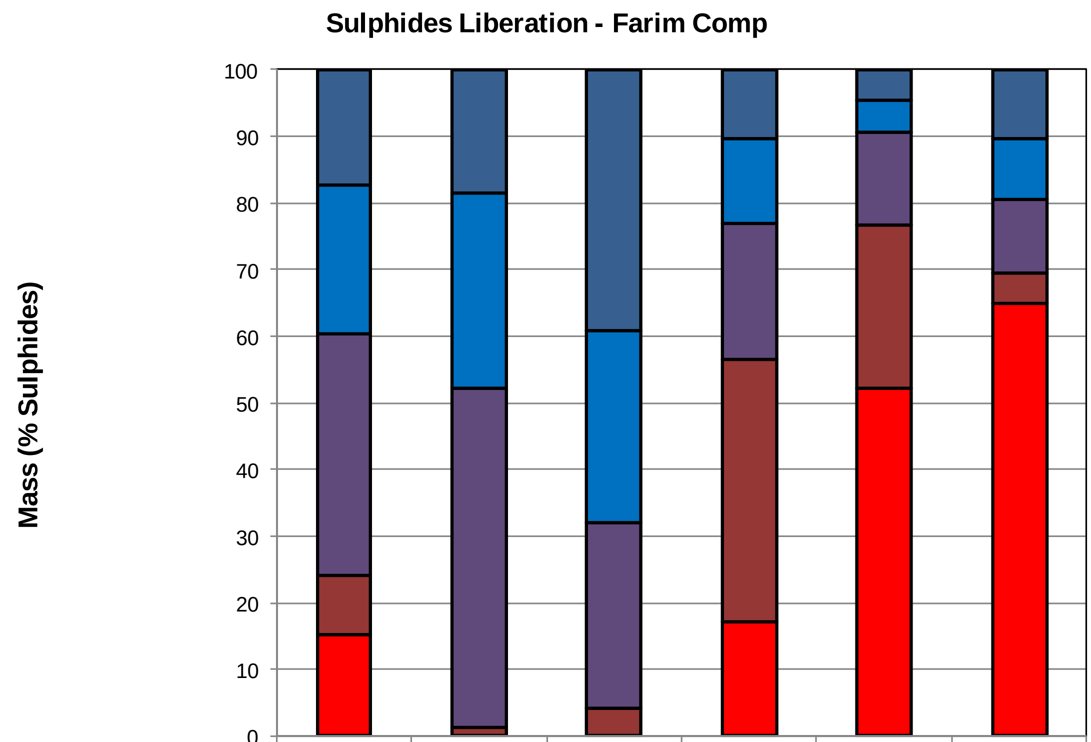
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Oxides	2.35	1.50	4.91	3.51	2.10	20.68
Sub-middling Oxides	20.04	23.26	11.68	7.91	2.62	9.13
Middling Oxides	45.16	44.96	59.25	45.46	19.41	8.85
Liberated Oxides	30.12	30.29	24.17	40.99	45.84	7.70
Free Oxides	0.94	0.00	0.00	1.24	17.68	3.87
Pure Oxide	1.39	0.00	0.00	0.89	12.35	49.77

Figure 10: Liberation Profile of the Oxides by Size Fraction (Normalized Distribution)



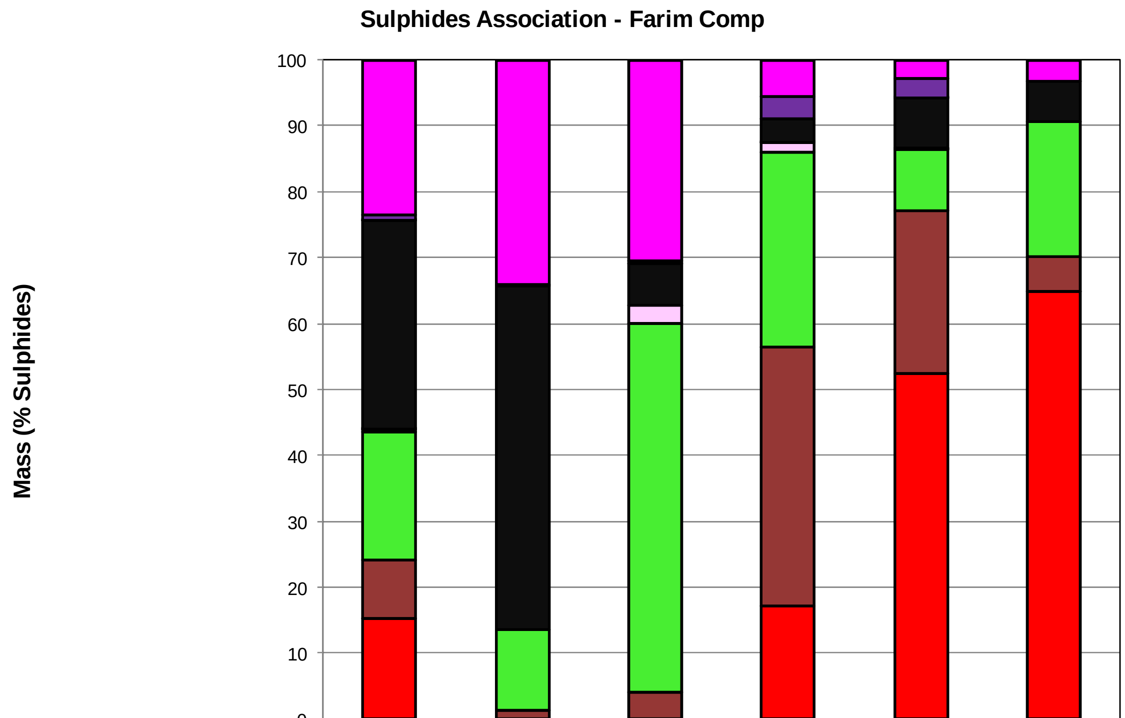
	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	11.81	14.03	5.79	3.00	1.72	1.58
Oxides:Sil:Carb:Apatite Total	41.00	42.24	47.17	45.95	13.20	2.00
Oxides:Other	0.01	0.00	0.00	0.00	0.00	0.70
Oxides:Sulphides	0.16	0.00	0.00	0.99	2.68	0.43
Oxides:Carbonates	0.00	0.00	0.00	0.01	0.01	0.00
Oxides:Silicates	0.31	0.17	1.12	0.45	0.28	0.09
Oxides:Apatite Total	14.25	13.27	21.75	6.49	6.25	33.87
Liberated Oxides	30.12	30.29	24.17	40.99	45.84	7.70
Free Oxides	2.33	0.00	0.00	2.13	30.03	53.64

Figure 11: Association Profile of the Oxides by Size Fraction (Normalized Distribution)



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Barren	0.00	0.00	0.00	0.00	0.00	0.00
Locked Sulphides	17.33	18.54	39.23	10.25	4.47	10.35
Sub-middling Sulphides	22.31	29.19	28.75	12.87	4.96	9.15
Middling Sulphides	36.36	51.02	27.86	20.38	13.76	10.93
Liberated Sulphides	8.77	1.24	4.16	39.33	24.72	4.70
Free Sulphides	15.24	0.00	0.00	17.17	52.09	64.86

Figure 12: Liberation Profile of the Sulphides by Size Fraction (Normalized Distribution)



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	23.53	34.12	30.53	5.62	2.85	3.26
Sulphides:Sil:Oxides:Apatite Total	0.82	0.16	0.49	3.28	2.85	0.00
Sulphides:Other	0.00	0.00	0.00	0.00	0.01	0.00
Sulphides:Oxides	31.60	52.17	6.25	3.70	7.67	6.04
Sulphides:Carbonates	0.03	0.00	0.00	0.01	0.26	0.00
Sulphides:Silicates	0.44	0.00	2.62	1.43	0.00	0.00
Sulphides:Apatite Total	19.47	12.31	55.95	29.45	9.30	20.48
Liberated Sulphides	8.82	1.24	4.16	39.33	24.54	5.37
Free Sulphides	15.28	0.00	0.00	17.17	52.51	64.86

Figure 13: Association Profile of the Sulphides by Size Fraction (Normalized Distribution)

6.1. Mineral Release Curves - Farim Comp

Mineral release curves are used to predict the amount of liberated mineral of interest at varying size distributions. This can be an indicator of optimum grind targets for metallurgical processes to achieve the most liberation for the least grind energy. The variation between value and gangue mineral release curves may sometimes be used to enhance separation.

Note: The size used for the mineral release is the mid-point screen size, which is calculated by the following: Midpoint = square root (top size) x square root (bottom size). For the top size, (e.g., +200 μm) the top size particle (e.g., 340 μm) is identified, then 340 μm will be the top size and 200 μm the bottom size. Thus, the point for the mineral release at this liberation would be calculated as: square root (340) x square root (200) = 18.4390 x 14.1421 = 260.76. For any mid-size, the size fraction μm is used for this calculation. However, for the bottom size, 3 μm is used because that is approximately the beam diameter limitation for the QEMSCAN. As per the liberation data discussed in previous sections, the mineral release curves have been expressed for "Apatite Total", carbonates, silicates, oxides and sulphides as in Figure 14.

Liberation of "Apatite Total" ranges from 73%, 97%, 99%, 99.5%, and 99% for grains sizes of 2,137 μm, 708 μm, 212 μm, 46 μm, and 9 μm, respectively.

Liberation of the "silicates" ranges from 13% to 81% to 97% to 95% and 95% for the same sizes, respectively.

Liberation of the "carbonates" ranges from 0% to 1.7% to 21% to 84% and 97% for the same sizes, respectively.

Liberation of the "oxides" ranges from Nil for the two top sizes to 2% to 30% and 53% for the same sizes, respectively.

Liberation of the "sulphides" ranges from 1% to 4% 56% to 77% and 70% for the same sizes, respectively.

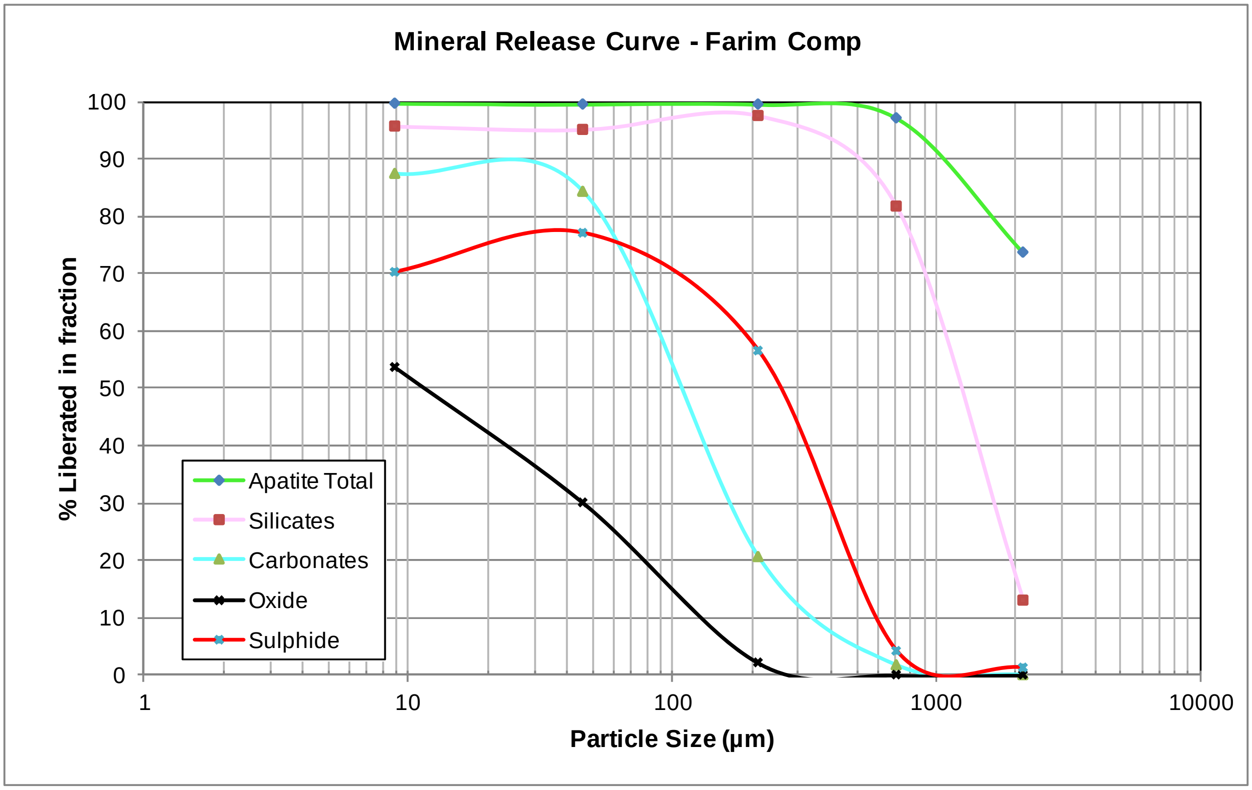


Figure 14: Mineral Release Curves for Farim Comp

6.2. Grade Recovery Curves

Another more functional method of presenting liberation is the mineralogically limiting grade-recovery curves, as shown below. They are based on the calculated mass of minerals and the total mass in each liberation category. Thus, the highest grade (>95% apatite) is contained in the >95% liberated apatite particles. Then the next category (60-80% liberation) is added and the combined grade is calculated. This is repeated until all apatite is accounted for. Mineralogically limited grade-recovery analyses provide an indication of the *theoretical maximum achievable* elemental or mineral grade by recovery, based on individual particle liberation and composition. These results, of course, do not reflect any other recovery factors that could occur in the actual metallurgical process. The graphs generated using QEMSCAN are merely a simple simulation of what is mineralogically possible. By sorting the minerals by their degree of liberation, the percentage of the mineral and the grade of that fraction can be calculated. First the fully liberated minerals are counted (>95% liberated), and that fraction will be the highest grade possible, but the recovery will be low. Then, incrementally, lesser liberated minerals are added to increase the recovery, but the grade will start to drop. This is done for each size fraction. It should be noted that all the calculations are done based on liberation which implies an ideal separation and most likely not attainable in a real plant operation. The plant grade-recovery curves will likely fall below the data presented in the graphs. However, the value of these plots is to demonstrate when liberation becomes a critical variable. The size by size grade recovery curves are presented in Figure 15 and illustrates that if an 80% apatite recovery is achieved, a maximum grade of 36% P_2O_5 is mineralogically possible at this grind target.

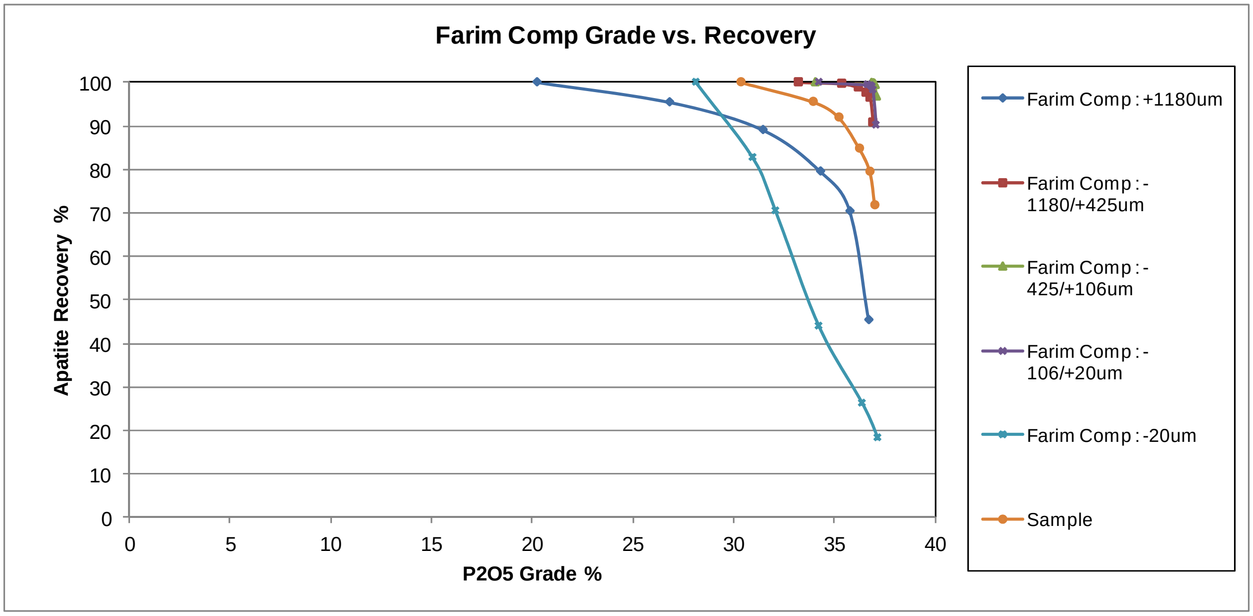


Figure 15: Grade Recovery of Apatite (P205) by Size Fraction

6.3. Cumulative Grain Size Distributions

Figure 16 illustrates the cumulative grain size distribution for the "Apatite Total", carbonates, silicates, particle, sulphides, and oxides. The curve referred to as "Particle" reflects all the measured minerals in the sample. A summary of the diameter at 50% passing (D_{50}) is presented in Table 9.

Table 9: D_{50} for Farim Comp

Mineral	D50 for Farim Comp
Apatite	109
Carbonates	59
Silicates	253
Particle	143
Sulphides	143
Oxides	505

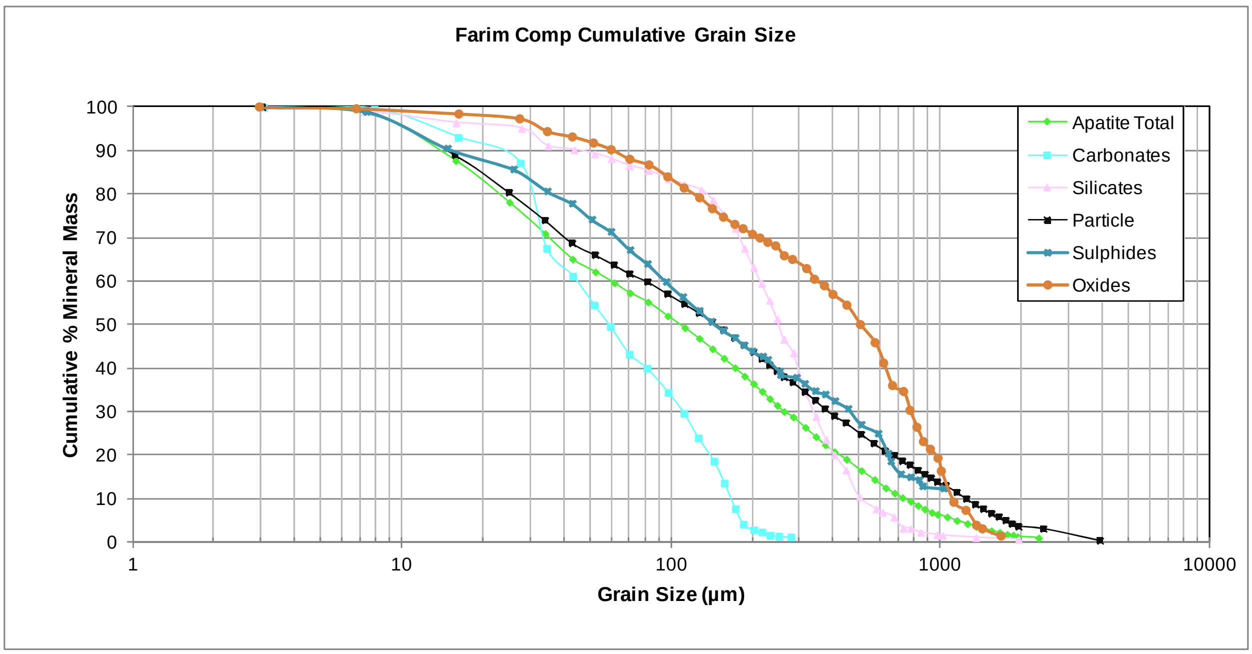


Figure 16: Cumulative Grain Size Distribution

Summary and Conclusions

The submitted sample has a calculated P_2O_5 head grade of ~14.6%.

The data from the electron microprobe analysis (EMPA) indicates that the average P_2O_5 content of the apatite is 38.33%. If a perfect concentrate of apatite was produced, this would be close to the maximum P_2O_5 grade that could be achieved. Apatite also contains SO_2, Na_2O and F contents at ~0.37%, 0.63%, and 10.91%, respectively.

QEMSCAN analysis yields apatite (45.9%), silicates (48.8% of which quartz accounts for 48.2%), carbonates (4.9% of which 4.8% is calcite), and trace amounts of Ti/Fe oxides, and sulphates.

The apatite content decreases from the +1,180 µm, +425 µm, and the +150 µm size fractions from ~72%,~ 71%, and ~17%, respectively, and increases again in the +75 µm and -75 µm size fractions from ~30% to ~60%. The -425/+150 µm fraction has the lowest apatite content and it consists of mainly quartz (81%).

The liberation of the apatite in the sample is good, at ~84%. The liberation remains above 80% in all size fractions. Liberation of the silicates is ~89% and that of carbonates is moderate, at ~67%.

The liberation data, in conjunction with the chemistry of the apatite, indicate that an 80% apatite recovery with a P_2O_5 grade of 33.7% is theoretically achievable.

Appendix A – QEMSCAN Data

QEMSCAN DATA

prepared for:

Lycopodiunm

Project 13478-003
MI5021-MAR14
April 27, 2015

Prepared by:

Katie Fairbairn/Kareen Fleury-Frenette
Mineralogist

High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy) (METH# 8.11.1) used by SGS Minerals Services

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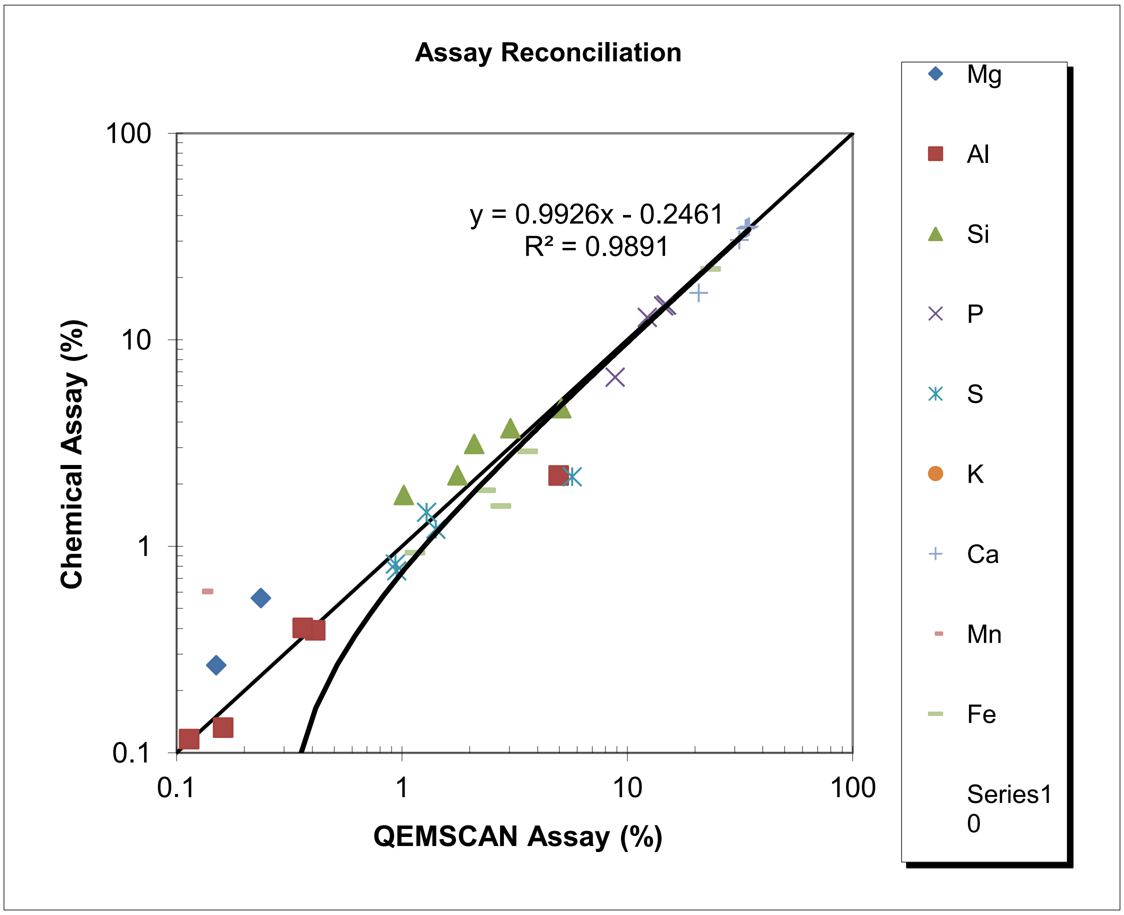
Member of the SGS Group (SGS SA)

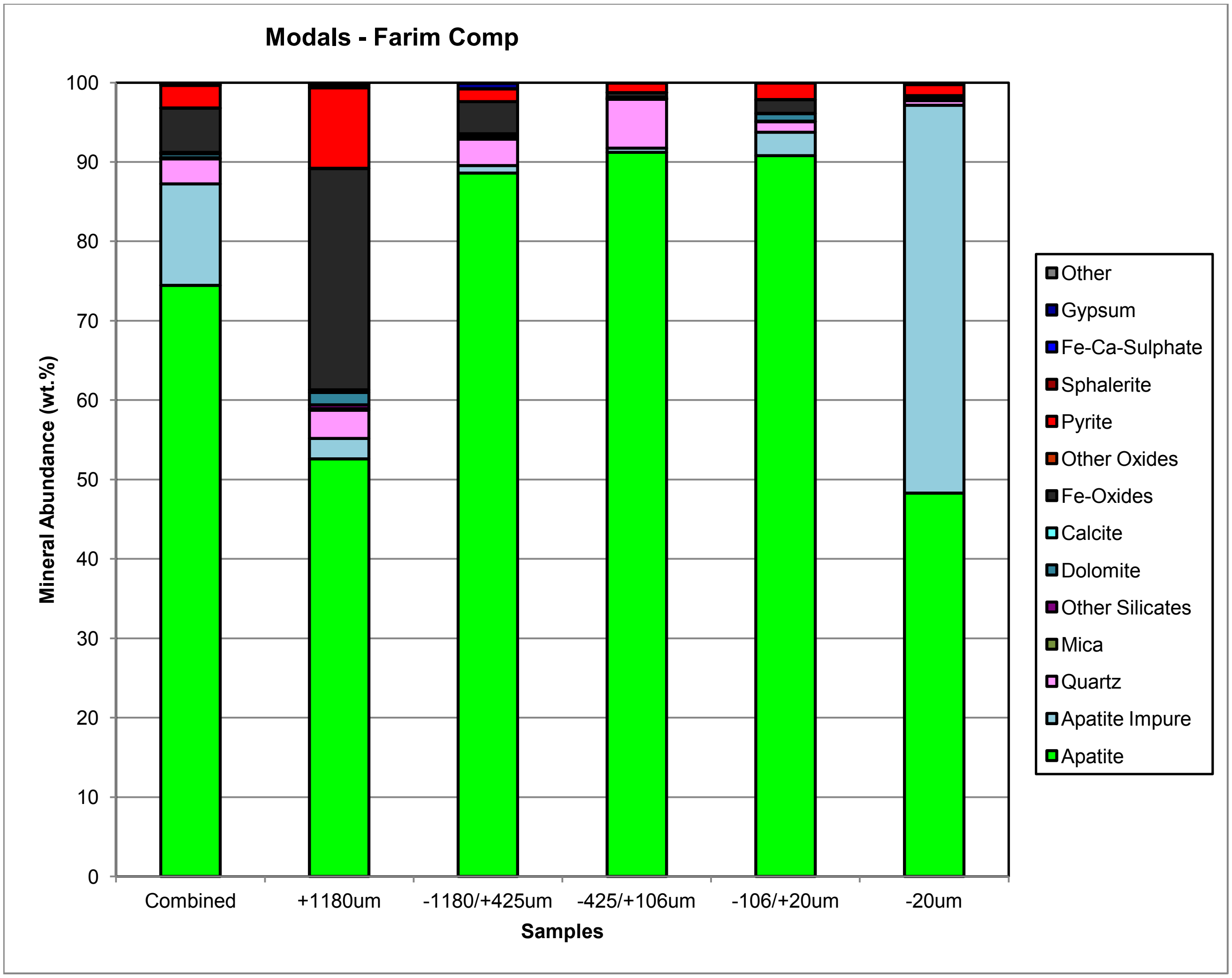
Lycopodiunm
13478-003
MI5021-MAR14

High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Assay Reconciliation



Sample	Farim Comp					
Element	**Combined**	**+1180um**	**1180/+425un**	**-425/+106um**	**-106/+20um**	**-20um**
Mg (QEMSCAN)	0.10	0.24	0.06	0.04	0.15	0.07
Mg (Chemical)	0.25	0.56	0.07	0.05	0.27	0.39
Al (QEMSCAN)	1.36	0.41	0.16	0.11	0.36	4.96
Al (Chemical)	0.70	0.39	0.13	0.12	0.40	2.20
Si (QEMSCAN)	2.82	2.09	1.76	3.03	1.02	5.09
Si (Chemical)	3.26	3.13	2.21	3.73	1.77	4.67
P (QEMSCAN)	13.25	8.83	14.49	14.86	14.93	12.27
P (Chemical)	12.99	6.59	14.58	14.75	14.71	12.83
S (QEMSCAN)	1.82	5.70	1.29	0.95	1.41	0.94
S (Chemical)	1.20	2.17	1.46	0.76	1.21	0.82
K (QEMSCAN)	0.01	0.04	0.01	0.00	0.00	0.02
K (Chemical)	0.04	0.02	0.01	0.01	0.02	0.11
Ca (QEMSCAN)	31.38	20.76	33.52	34.11	34.73	31.42
Ca (Chemical)	31.06	16.87	34.59	34.95	35.31	30.45
Mn (QEMSCAN)	0.53	0.13	0.06	0.05	0.19	1.93
Mn (Chemical)	0.12	0.60	0.08	0.02	0.03	0.02
Fe (QEMSCAN)	5.65	23.43	3.61	1.14	2.35	2.75
Fe (Chemical)	4.88	22.03	2.88	0.93	1.87	1.57


Modals - Farim Comp
Mineral Abundance (wt.%)
0
10
20
30
40
50
60
70
80
90
100
Combined
+1180um
-1180/+425um
-425/+106um
-106/+20um
-20um
Samples
Other
Gypsum
Fe-Ca-Sulphate
Sphalerite
Pyrite
Other Oxides
Fe-Oxides
Calcite
Dolomite
Other Silicates
Mica
Quartz
Apatite Impure
Apatite

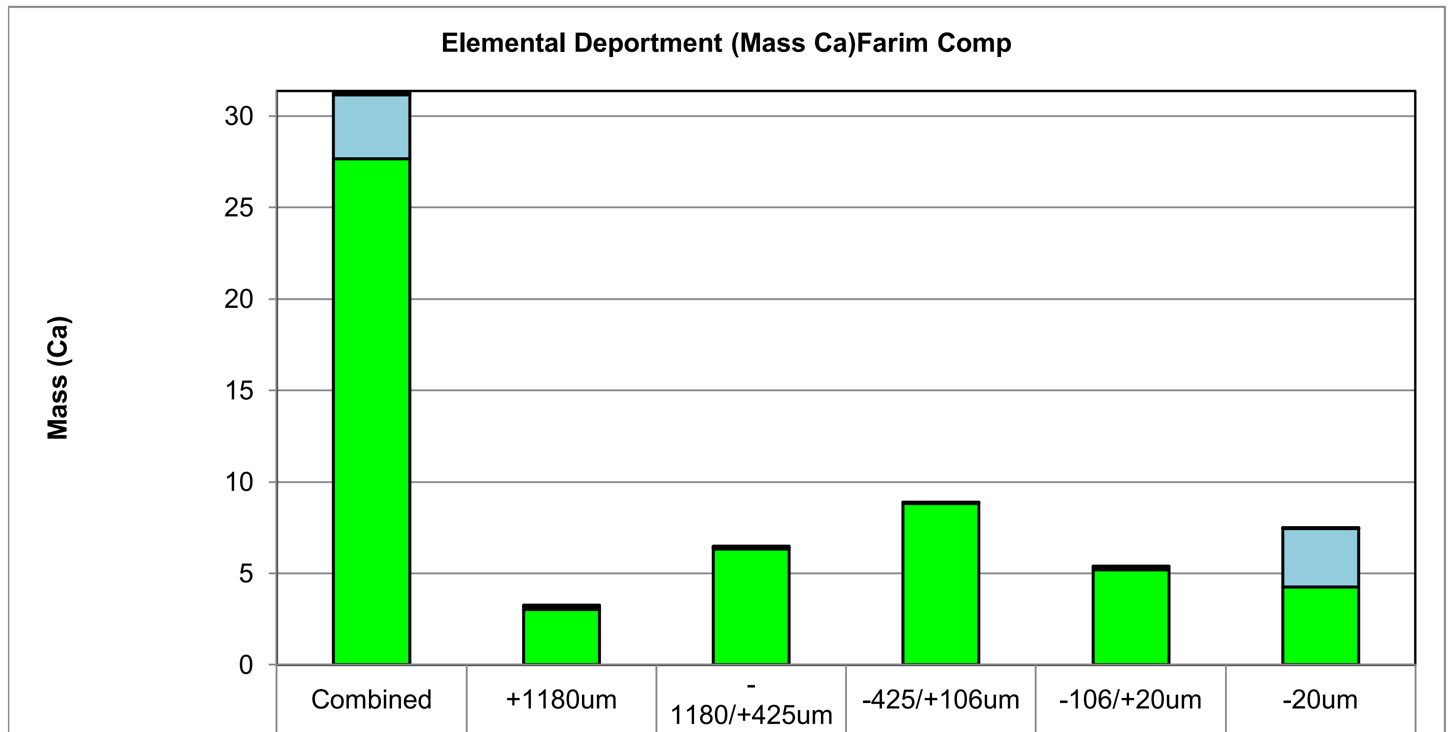
Lycopodiunm
13478-003
MI5021-MAR14

High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

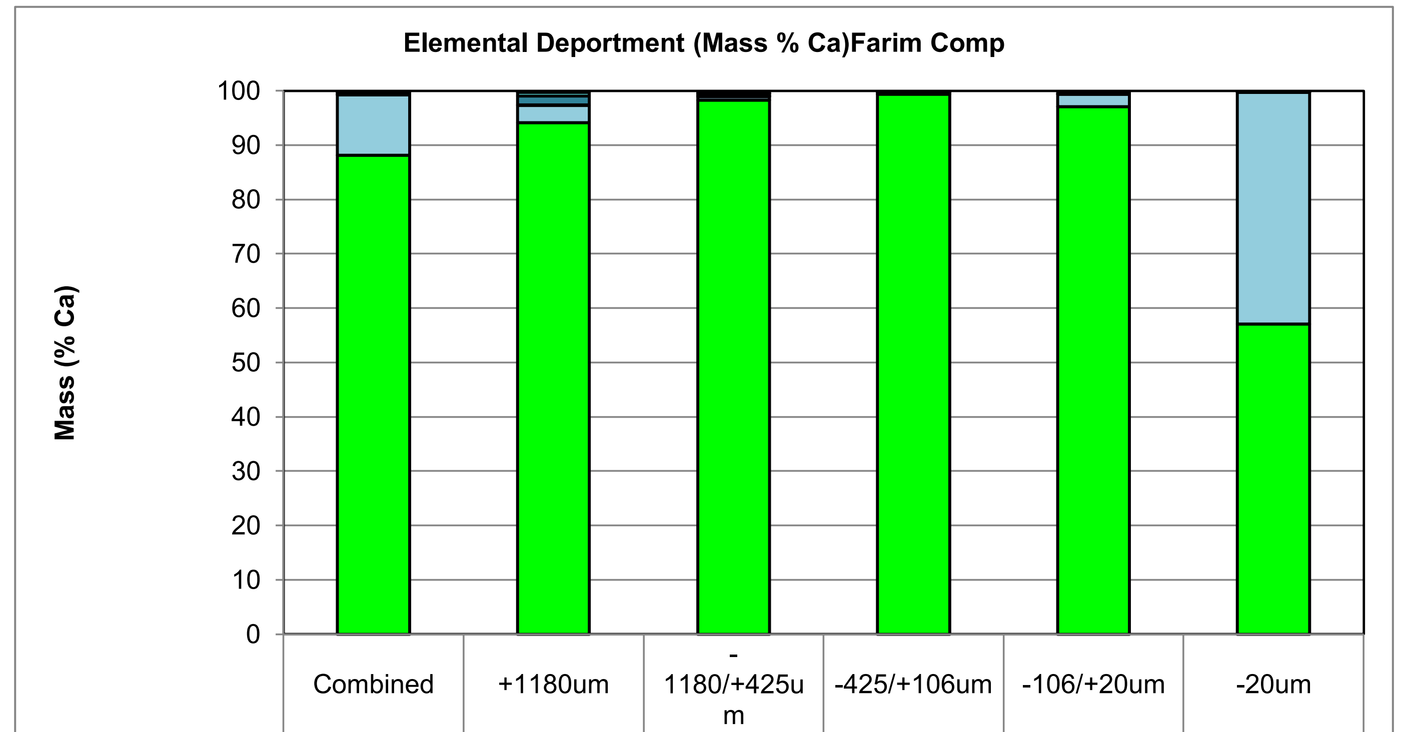
Modals

Survey		13478-003 / MI5021-MAR14										
Project		Lycopodiunm										
Sample		Farim Comp										
Fraction		Combined	+1180um		-1180/+425um		-425/+106um		-106/+20um		-20um	
Mass Size Distribution (%)			15.5		19.3		26.0		15.4		23.8	
Calculated ESD Particle Size		33	1241		405		126		33		10	
		Sample	Sample	Fraction	Sample	Fraction	Sample	Fraction	Sample	Fraction	Sample	Fraction
Mineral Mass (%)	Apatite	74.45	8.17	52.61	17.08	88.61	23.71	91.19	14.02	90.80	11.48	48.28
	Apatite Impure	12.79	0.40	2.56	0.18	0.92	0.14	0.55	0.46	2.96	11.62	48.87
	Quartz	3.13	0.55	3.57	0.65	3.35	1.60	6.14	0.20	1.31	0.13	0.56
	Mica	0.04	0.03	0.17	0.00	0.02	0.00	0.01	0.00	0.01	0.00	0.01
	Other Silicates	0.12	0.07	0.48	0.01	0.07	0.01	0.05	0.01	0.07	0.01	0.04
	Dolomite	0.50	0.24	1.57	0.05	0.24	0.02	0.09	0.14	0.93	0.04	0.18
	Calcite	0.18	0.05	0.33	0.07	0.35	0.04	0.14	0.01	0.05	0.02	0.07
	Fe-Oxides	5.58	4.33	27.88	0.77	4.02	0.15	0.57	0.26	1.72	0.06	0.26
	Other Oxides	0.04	0.00	0.03	0.00	0.01	0.00	0.01	0.00	0.03	0.03	0.11
	Pyrite	2.83	1.57	10.11	0.30	1.58	0.31	1.19	0.32	2.06	0.33	1.37
	Sphalerite	0.03	0.01	0.09	0.00	0.01	0.00	0.00	0.00	0.01	0.01	0.03
	Fe-Ca-Sulphate	0.12	0.06	0.39	0.01	0.06	0.00	0.01	0.01	0.04	0.04	0.18
	Gypsum	0.17	0.02	0.15	0.13	0.66	0.01	0.03	0.00	0.00	0.00	0.02
	Other	0.04	0.01	0.04	0.02	0.09	0.00	0.01	0.00	0.01	0.01	0.03
	Total	100.00	15.52	100.0	19.27	100.0	26.00	100.0	15.44	100.0	23.77	100.0
Mean Grain Size by Frequency (µm)	Apatite	34	249		291		98		31		8	
	Apatite Impure	6	45		39		14		9		6	
	Quartz	78	141		241		193		51		6	
	Mica	18	32		31		13		7		4	
	Other Silicates	20	36		34		23		14		4	
	Dolomite	35	89		93		47		28		8	
	Calcite	19	31		34		17		7		8	
	Fe-Oxides	104	181		151		41		24		6	
	Other Oxides	5	35		31		12		9		4	
	Pyrite	36	141		97		60		25		7	
	Sphalerite	24	83		56		14		19		10	
	Fe-Ca-Sulphate	9	55		42		16		9		4	
	Gypsum	72	94		129		67		7		6	
	Other	10	30		34		12		6		4	

Ca Deportment



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Other	0.00	0.00	0.00	0.00	0.00	0.00
Gypsum	0.04	0.01	0.03	0.00	0.00	0.00
Fe-Ca-Sulphate	0.01	0.01	0.00	0.00	0.00	0.00
Other Oxides	0.00	0.00	0.00	0.00	0.00	0.00
Calcite	0.07	0.02	0.03	0.01	0.00	0.01
Dolomite	0.10	0.05	0.01	0.00	0.03	0.01
Other Silicates	0.01	0.01	0.00	0.00	0.00	0.00
Apatite Impure	3.48	0.10	0.05	0.04	0.12	3.18
Apatite	27.66	3.03	6.35	8.81	5.21	4.26



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Other	0.00	0.00	0.00	0.00	0.00	0.00
Gypsum	0.12	0.17	0.46	0.02	0.00	0.02
Fe-Ca-Sulphate	0.03	0.16	0.01	0.00	0.01	0.05
Other Oxides	0.00	0.00	0.00	0.00	0.00	0.00
Calcite	0.23	0.64	0.42	0.16	0.06	0.09
Dolomite	0.33	1.57	0.15	0.05	0.56	0.11
Other Silicates	0.03	0.20	0.02	0.01	0.02	0.01
Apatite Impure	11.10	3.07	0.70	0.42	2.21	42.64
Apatite	88.15	94.19	98.23	99.33	97.14	57.10

Elemental Deportment (Mass Ca)Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Apatite	27.66	3.03	6.35	8.81	5.21	4.26
Apatite Impure	3.48	0.10	0.05	0.04	0.12	3.18
Other Silicates	0.01	0.01	0.00	0.00	0.00	0.00
Dolomite	0.10	0.05	0.01	0.00	0.03	0.01
Calcite	0.07	0.02	0.03	0.01	0.00	0.01
Other Oxides	0.00	0.00	0.00	0.00	0.00	0.00
Fe-Ca-Sulphate	0.01	0.01	0.00	0.00	0.00	0.00
Gypsum	0.04	0.01	0.03	0.00	0.00	0.00
Other	0.00	0.00	0.00	0.00	0.00	0.00
Total	31.38	3.22	6.46	8.87	5.36	7.47
Total (% in fraction)	100.00	10.27	20.58	28.26	17.09	23.80

Elemental Deportment (Mass % Ca)Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Apatite	88.15	94.19	98.23	99.33	97.14	57.10
Apatite Impure	11.10	3.07	0.70	0.42	2.21	42.64
Other Silicates	0.03	0.20	0.02	0.01	0.02	0.01
Dolomite	0.33	1.57	0.15	0.05	0.56	0.11
Calcite	0.23	0.64	0.42	0.16	0.06	0.09
Other Oxides	0.00	0.00	0.00	0.00	0.00	0.00
Fe-Ca-Sulphate	0.03	0.16	0.01	0.00	0.01	0.05
Gypsum	0.12	0.17	0.46	0.02	0.00	0.02
Other	0.00	0.00	0.00	0.00	0.00	0.00
Total	100.00	100.00	100.00	100.00	100.00	100.00

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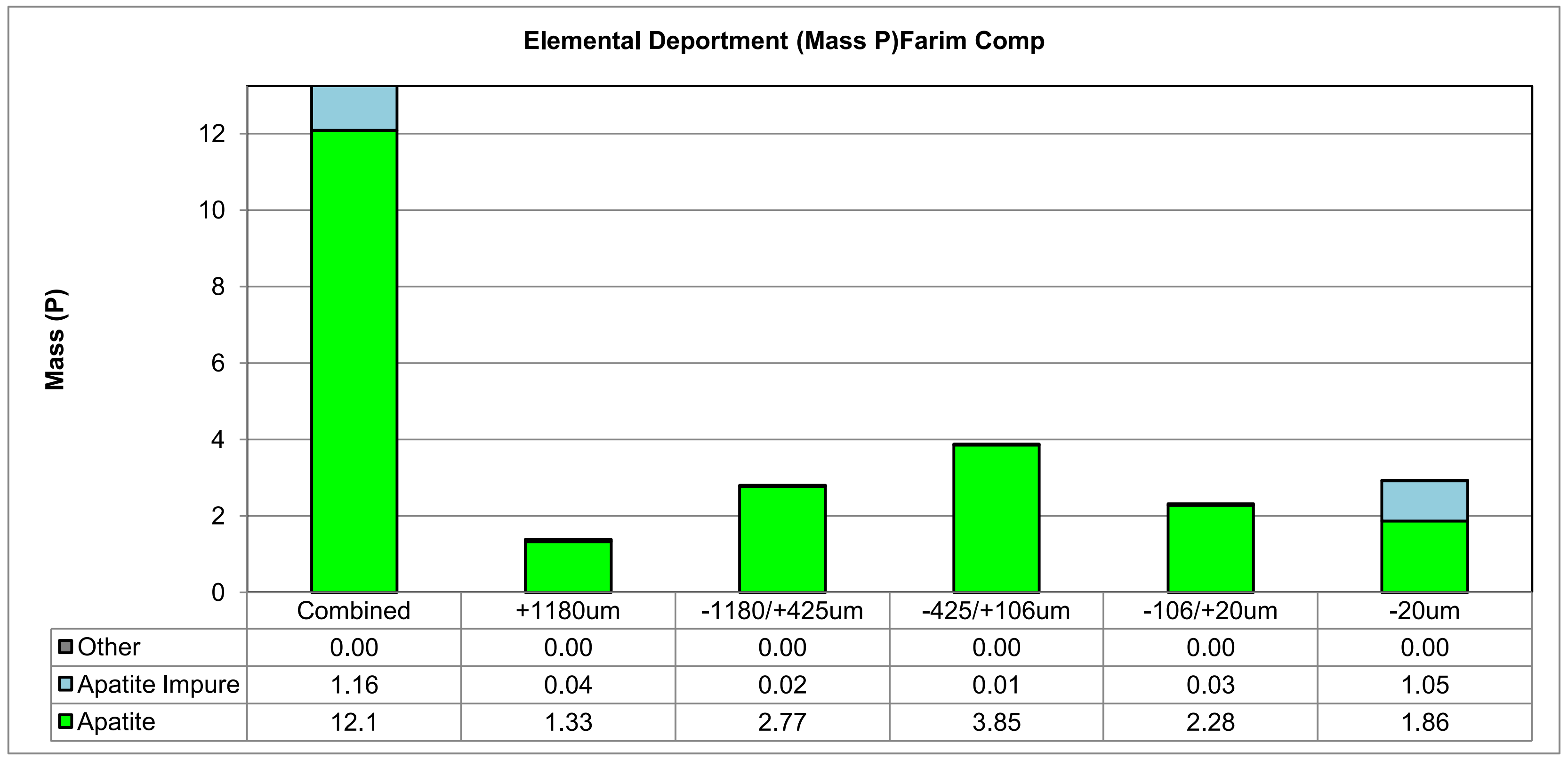
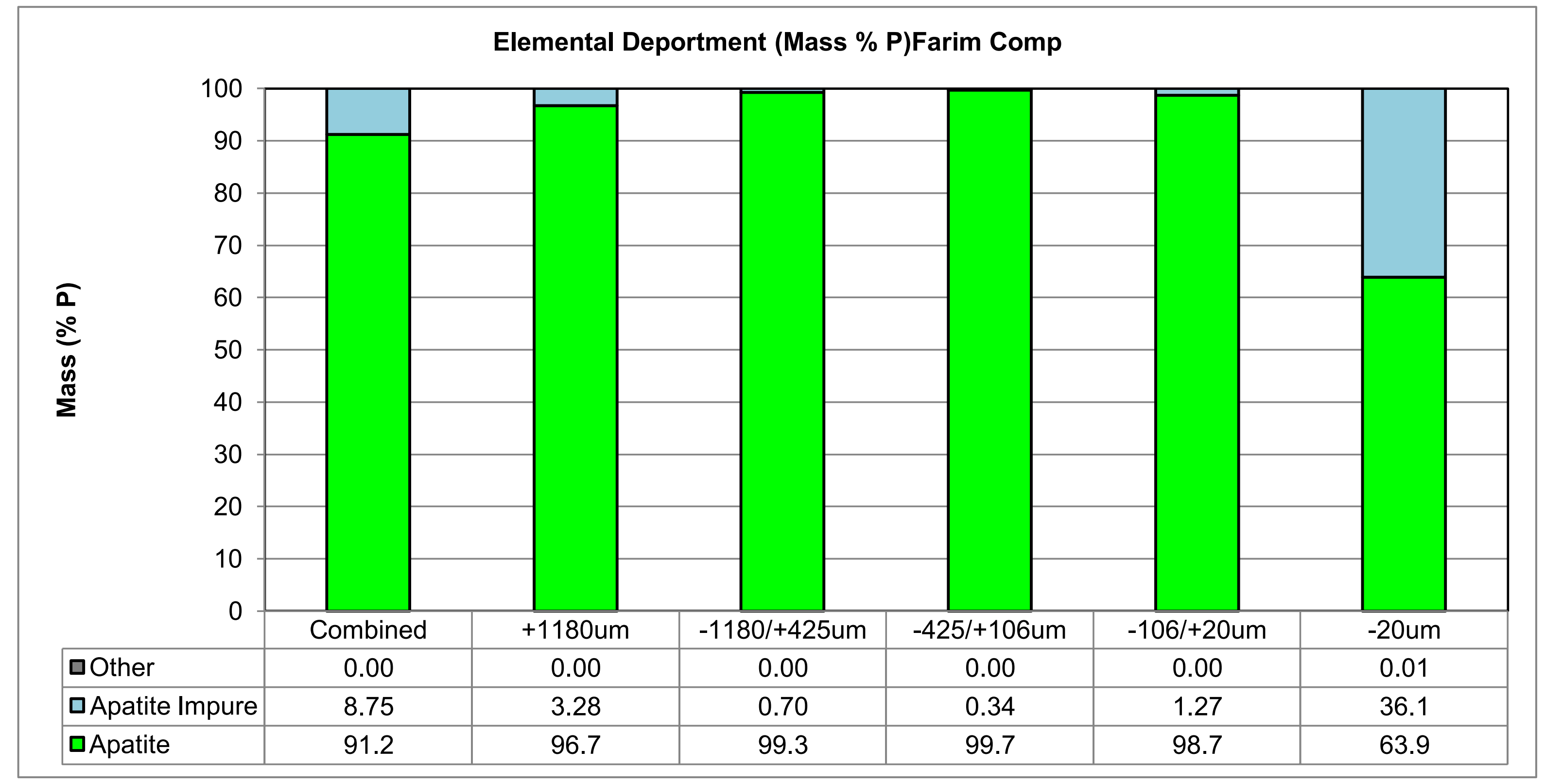
P Deportment





Elemental Deportment (Mass P)Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Apatite	12.1	1.33	2.77	3.85	2.28	1.86
Apatite Impure	1.16	0.04	0.02	0.01	0.03	1.05
Other	0.00	0.00	0.00	0.00	0.00	0.00
Total	13.3	1.37	2.79	3.86	2.31	2.92
Total (% in fraction)	100.0	10.3	21.1	29.2	17.4	22.0

Elemental Deportment (Mass % P)Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Apatite	91.2	96.7	99.3	99.7	98.7	63.9
Apatite Impure	8.75	3.28	0.70	0.34	1.27	36.1
Other	0.00	0.00	0.00	0.00	0.00	0.01
Total	100.0	100.0	100.0	100.0	100.0	100.0

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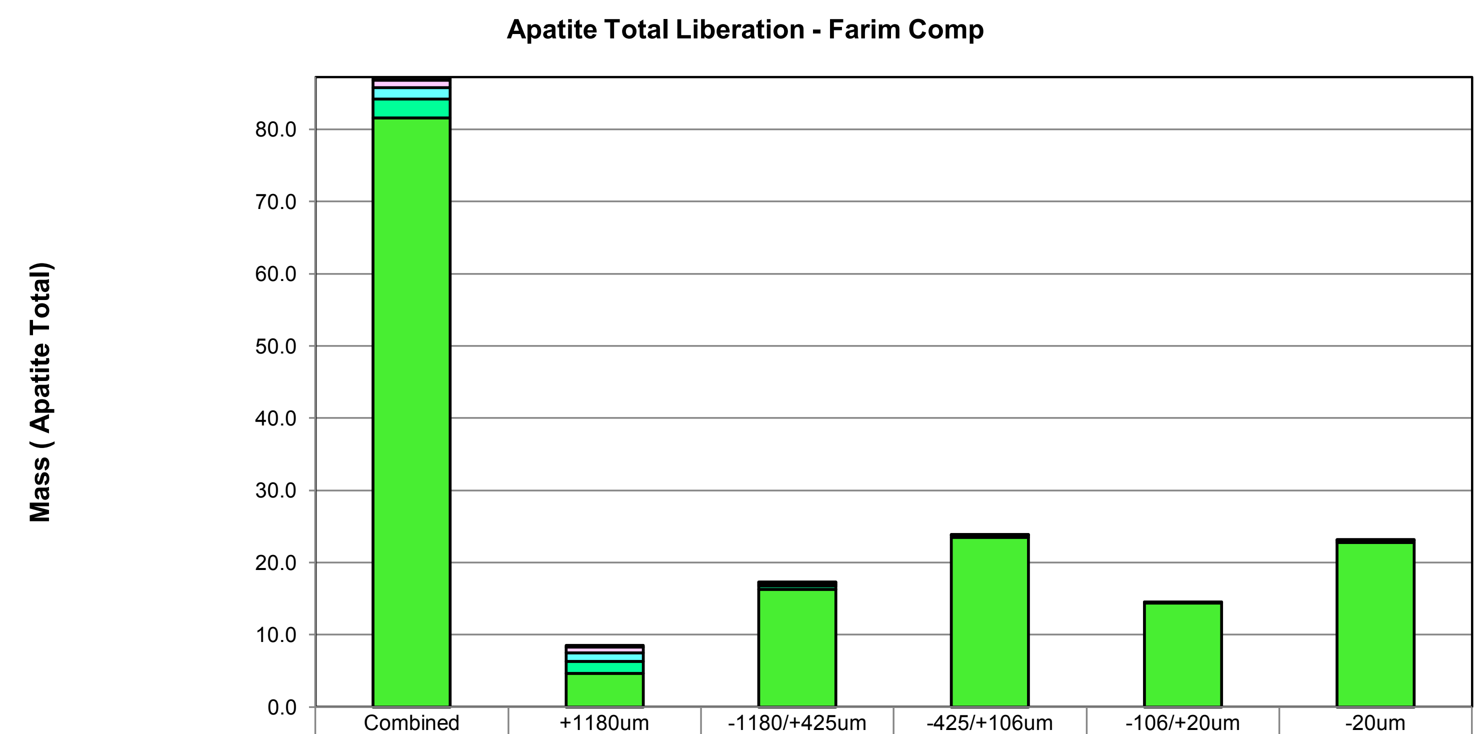
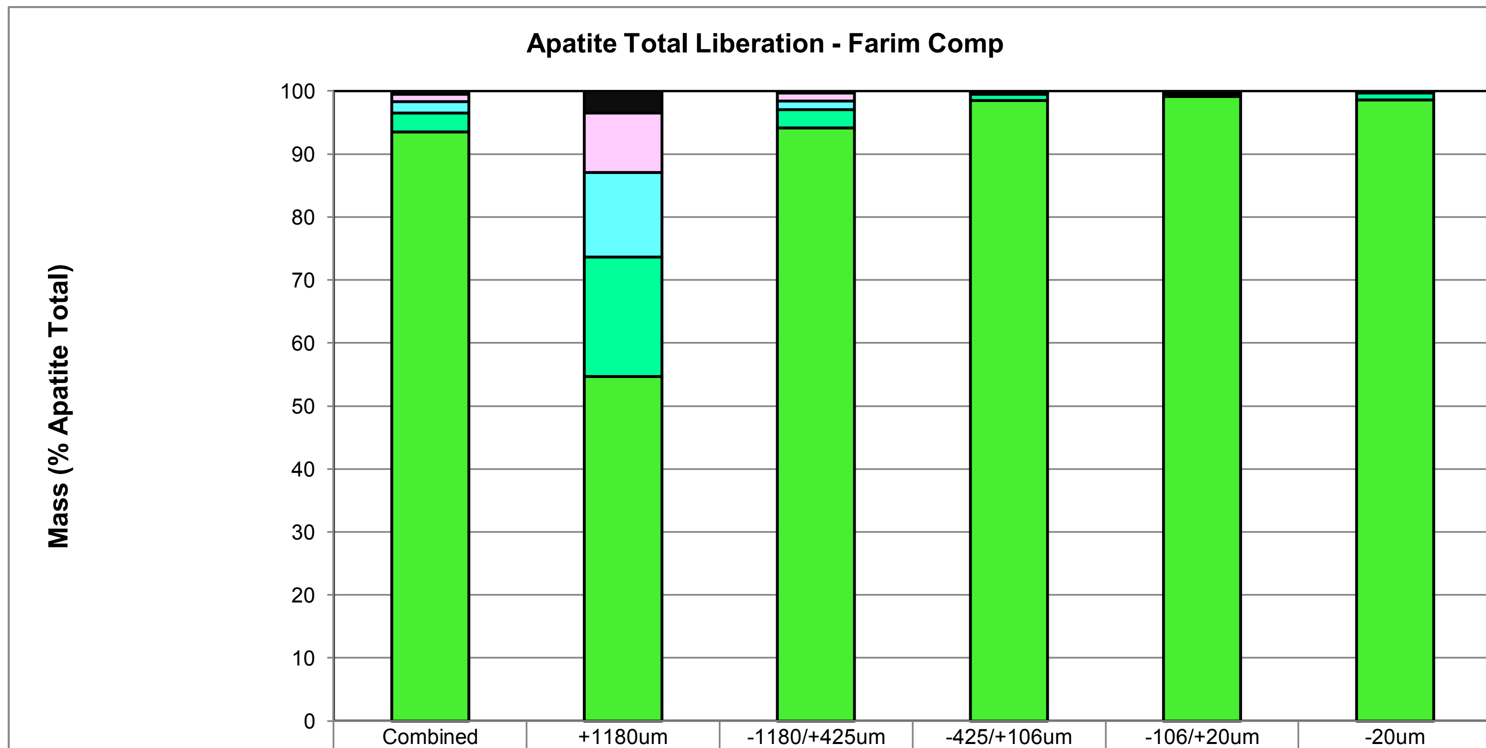
Apatite Total Liberation



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Apatite Total	0.41	0.30	0.06	0.03	0.02	0.01
Sub-middling Apatite Total	1.10	0.81	0.21	0.03	0.03	0.02
Middling Apatite Total	1.53	1.15	0.23	0.06	0.03	0.06
Liberated Apatite Total	2.66	1.63	0.51	0.23	0.06	0.24
Free Apatite Total	81.5	4.68	16.2	23.5	14.3	22.8



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Apatite Total	0.47	3.47	0.33	0.11	0.13	0.03
Sub-middling Apatite Total	1.26	9.44	1.23	0.14	0.17	0.08
Middling Apatite Total	1.76	13.4	1.36	0.27	0.20	0.25
Liberated Apatite Total	3.05	19.0	2.94	0.97	0.39	1.03
Free Apatite Total	93.5	54.7	94.1	98.5	99.1	98.6

Absolute Mass of Apatite Total Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Apatite Total	81.5	4.68	16.2	23.5	14.3	22.8
Liberated Apatite Total	2.66	1.63	0.51	0.23	0.06	0.24
Middling Apatite Total	1.53	1.15	0.23	0.06	0.03	0.06
Sub-middling Apatite Total	1.10	0.81	0.21	0.03	0.03	0.02
Locked Apatite Total	0.41	0.30	0.06	0.03	0.02	0.01
Total	87.2	8.56	17.3	23.8	14.5	23.1
Total (% in fraction)	100.0	9.82	19.8	27.3	16.6	26.5

Normalized Mass of Apatite Total Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Apatite Total	93.5	54.7	94.1	98.5	99.1	98.6
Liberated Apatite Total	3.05	19.0	2.94	0.97	0.39	1.03
Middling Apatite Total	1.76	13.4	1.36	0.27	0.20	0.25
Sub-middling Apatite Total	1.26	9.44	1.23	0.14	0.17	0.08
Locked Apatite Total	0.47	3.47	0.33	0.11	0.13	0.03
Total	100.0	100.0	100.0	100.0	100.0	100.0

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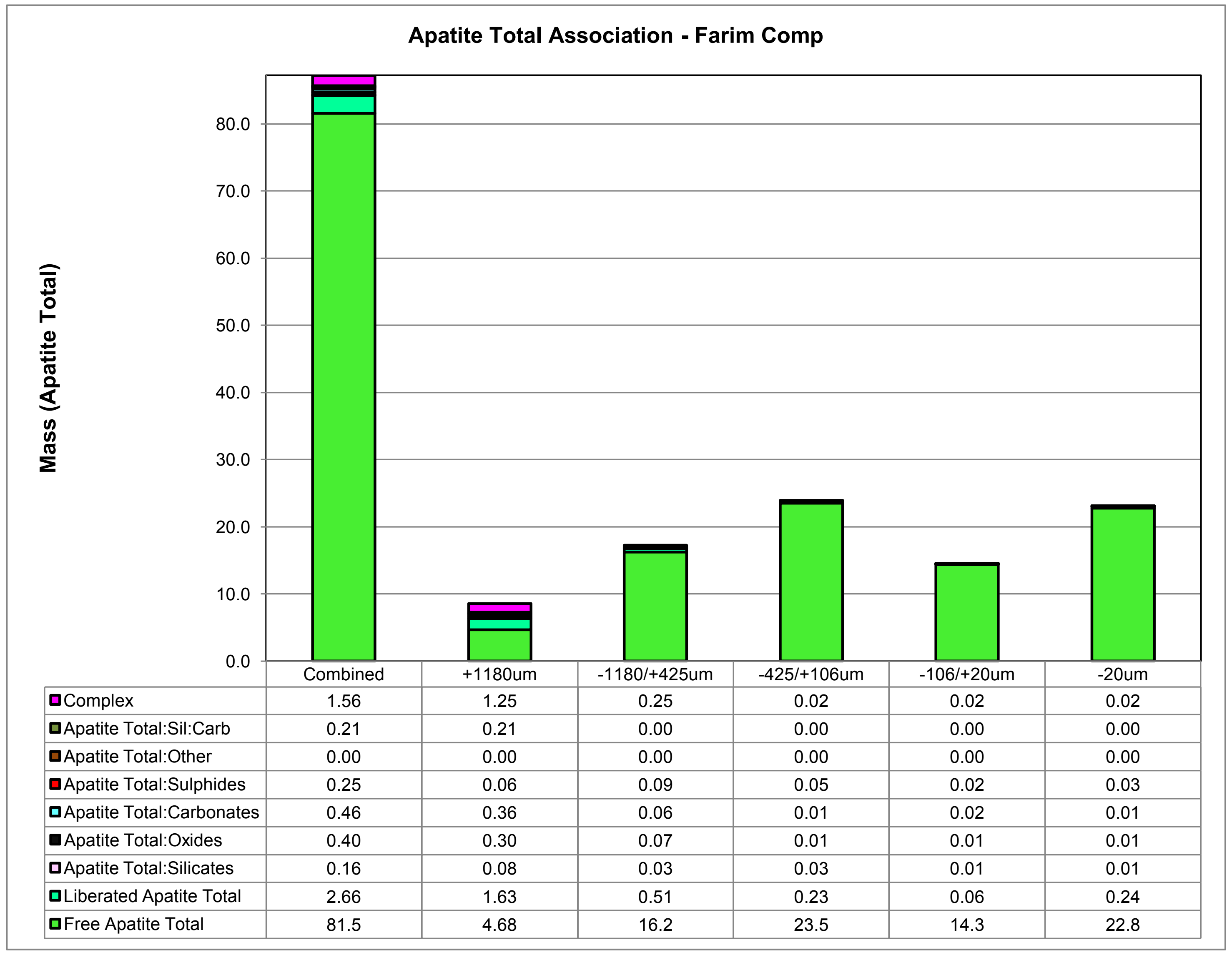
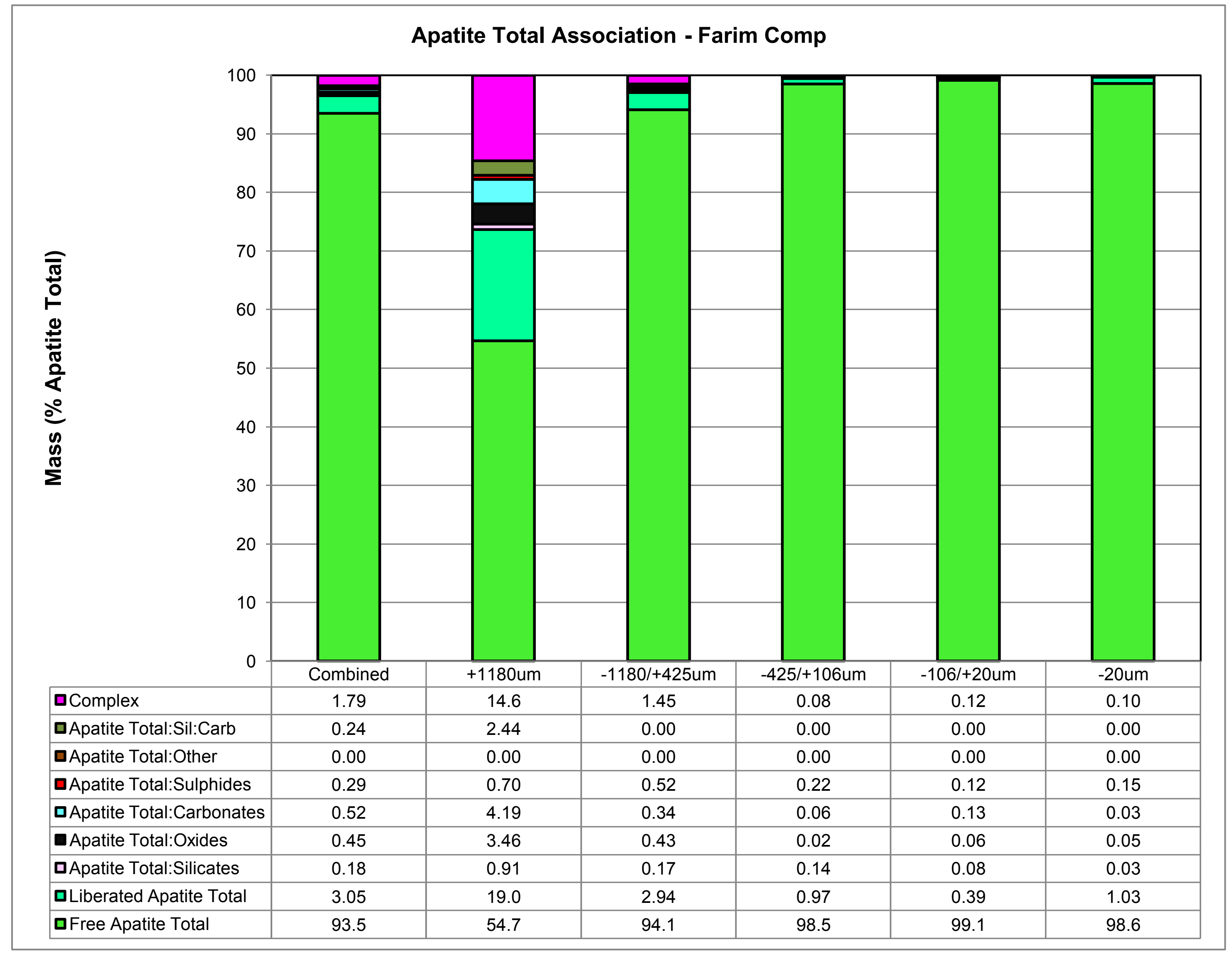
Apatite Total Association





Absolute Mass of Apatite Total Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Apatite Total	81.5	4.68	16.2	23.5	14.3	22.8
Liberated Apatite Total	2.66	1.63	0.51	0.23	0.06	0.24
Apatite Total:Silicates	0.16	0.08	0.03	0.03	0.01	0.01
Apatite Total:Oxides	0.40	0.30	0.07	0.01	0.01	0.01
Apatite Total:Carbonates	0.46	0.36	0.06	0.01	0.02	0.01
Apatite Total:Sulphides	0.25	0.06	0.09	0.05	0.02	0.03
Apatite Total:Other	0.00	0.00	0.00	0.00	0.00	0.00
Apatite Total:Sil:Carb	0.21	0.21	0.00	0.00	0.00	0.00
Complex	1.56	1.25	0.25	0.02	0.02	0.02
Total	87.2	8.56	17.3	23.8	14.5	23.1
Total (% in fraction)	100.0	9.82	19.8	27.3	16.6	26.5

Normalized Mass of Apatite Total Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Apatite Total	93.5	54.7	94.1	98.5	99.1	98.6
Liberated Apatite Total	3.05	19.0	2.94	0.97	0.39	1.03
Apatite Total:Silicates	0.18	0.91	0.17	0.14	0.08	0.03
Apatite Total:Oxides	0.45	3.46	0.43	0.02	0.06	0.05
Apatite Total:Carbonates	0.52	4.19	0.34	0.06	0.13	0.03
Apatite Total:Sulphides	0.29	0.70	0.52	0.22	0.12	0.15
Apatite Total:Other	0.00	0.00	0.00	0.00	0.00	0.00
Apatite Total:Sil:Carb	0.24	2.44	0.00	0.00	0.00	0.00
Complex	1.79	14.6	1.45	0.08	0.12	0.10
Total	100.0	100.0	100.0	100.0	100.0	100.0
Liberated	96.5	73.7	97.1	99.5	99.5	99.6

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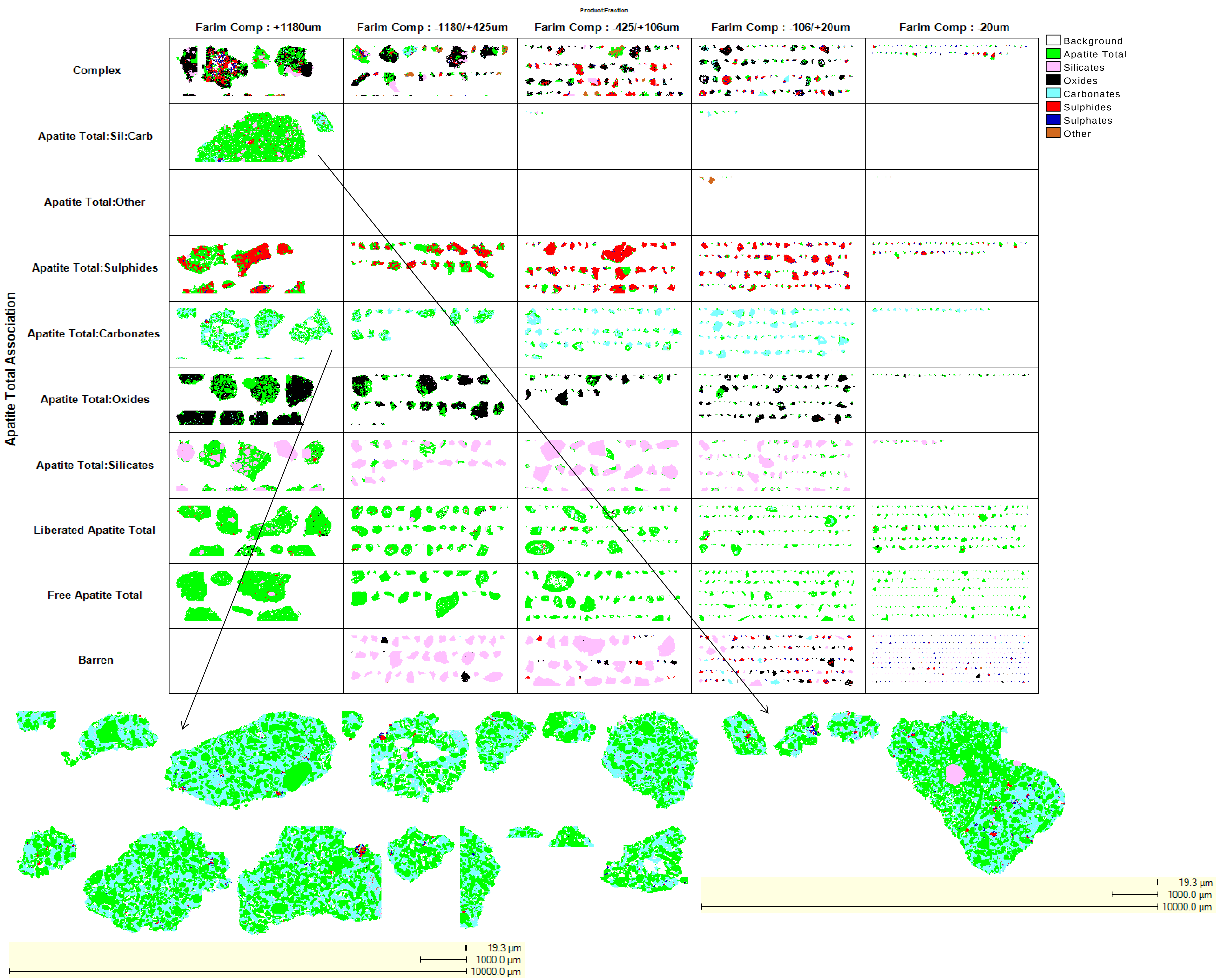
Apatite Total Association Image Grid



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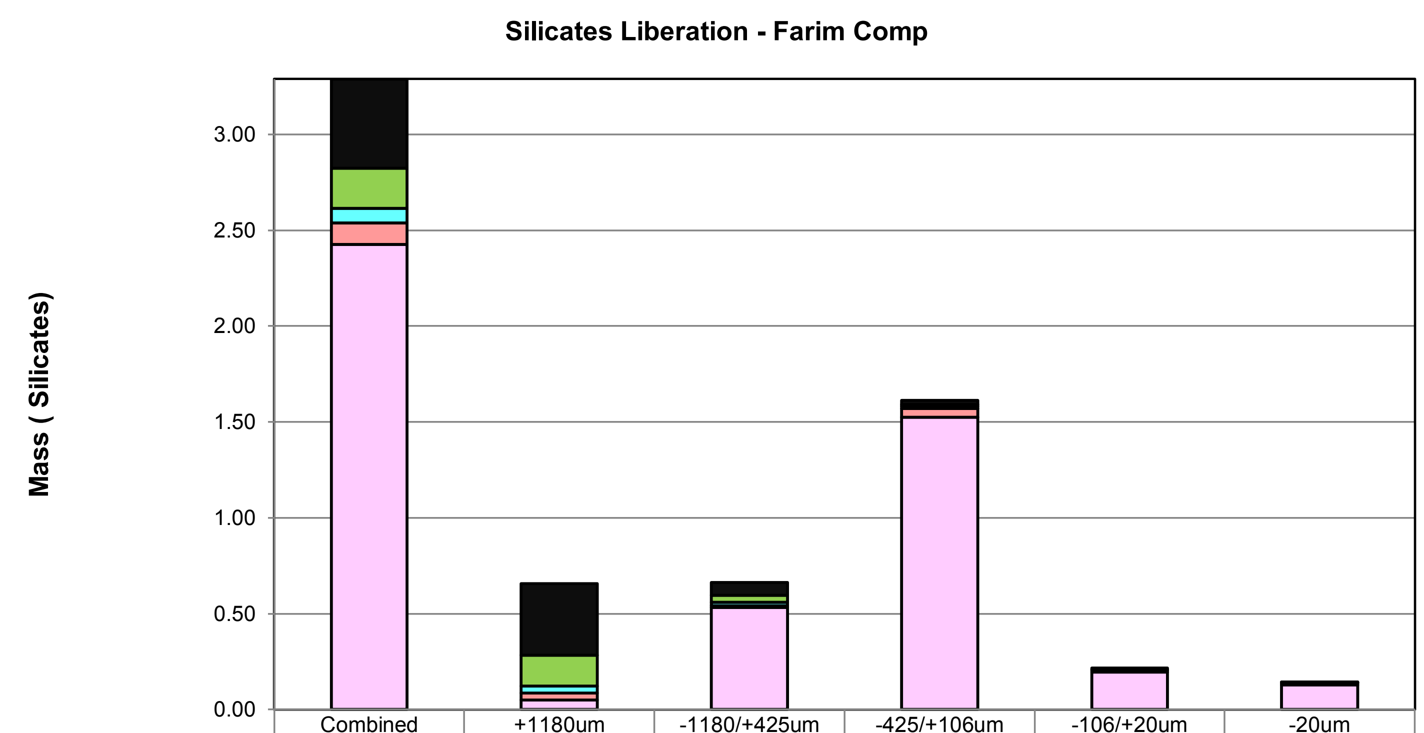
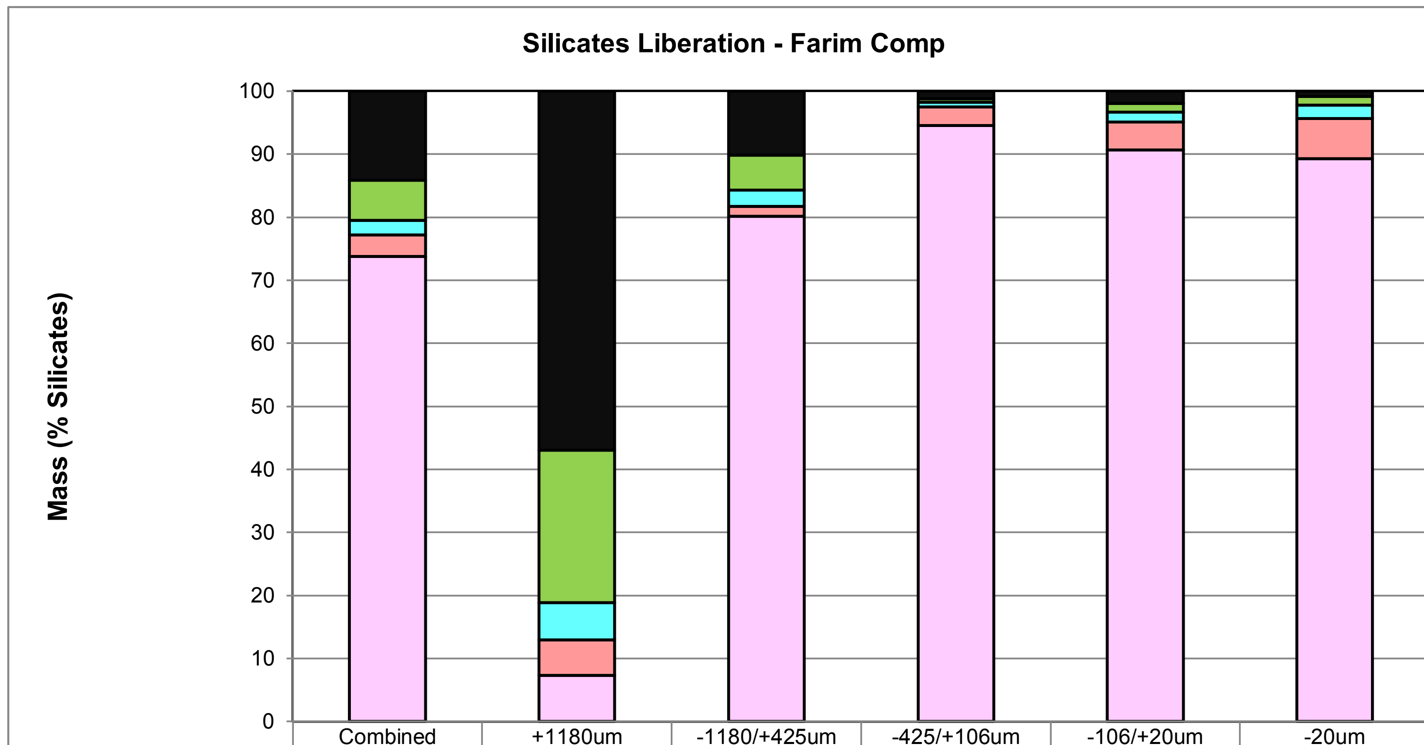
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Silicates Liberation



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Silicates	0.47	0.37	0.07	0.02	0.00	0.00
Sub-middling Silicates	0.21	0.16	0.04	0.01	0.00	0.00
Middling Silicates	0.07	0.04	0.02	0.01	0.00	0.00
Liberated Silicates	0.11	0.04	0.01	0.05	0.01	0.01
Free Silicates	2.43	0.05	0.53	1.52	0.20	0.13



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Locked Silicates	14.1	56.9	10.1	1.18	1.97	0.89
Sub-middling Silicates	6.38	24.2	5.61	0.58	1.39	1.37
Middling Silicates	2.26	5.90	2.59	0.74	1.58	2.10
Liberated Silicates	3.42	5.63	1.50	2.92	4.40	6.37
Free Silicates	73.8	7.36	80.2	94.6	90.7	89.3

Absolute Mass of Silicates Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Silicates	2.43	0.05	0.53	1.52	0.20	0.13
Liberated Silicates	0.11	0.04	0.01	0.05	0.01	0.01
Middling Silicates	0.07	0.04	0.02	0.01	0.00	0.00
Sub-middling Silicates	0.21	0.16	0.04	0.01	0.00	0.00
Locked Silicates	0.47	0.37	0.07	0.02	0.00	0.00
Total	3.29	0.66	0.66	1.61	0.22	0.14
Total (% in fraction)	100.0	19.9	20.2	49.0	6.56	4.34

Normalized Mass of Silicates Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Silicates	73.8	7.36	80.2	94.6	90.7	89.3
Liberated Silicates	3.42	5.63	1.50	2.92	4.40	6.37
Middling Silicates	2.26	5.90	2.59	0.74	1.58	2.10
Sub-middling Silicates	6.38	24.2	5.61	0.58	1.39	1.37
Locked Silicates	14.1	56.9	10.1	1.18	1.97	0.89
Total	100.0	100.0	100.0	100.0	100.0	100.0

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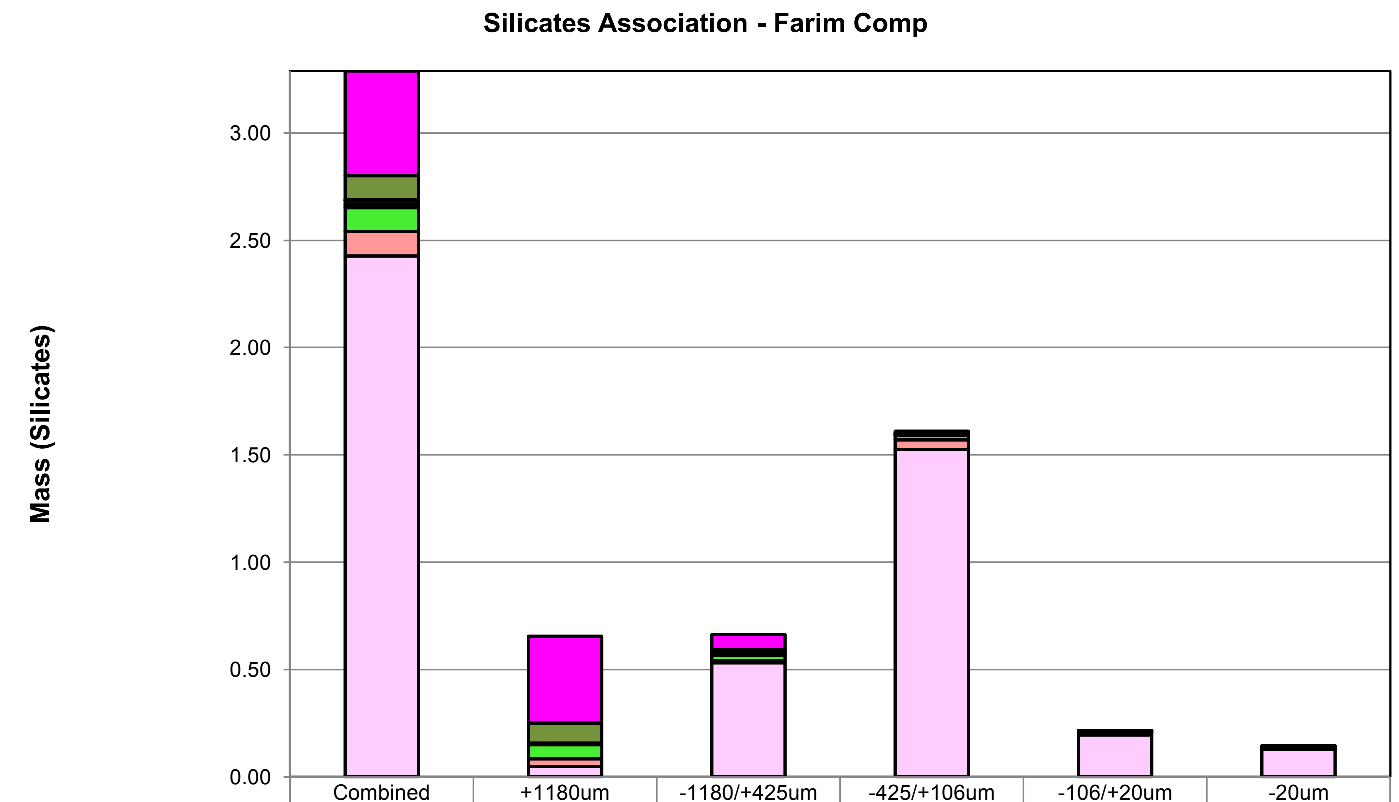
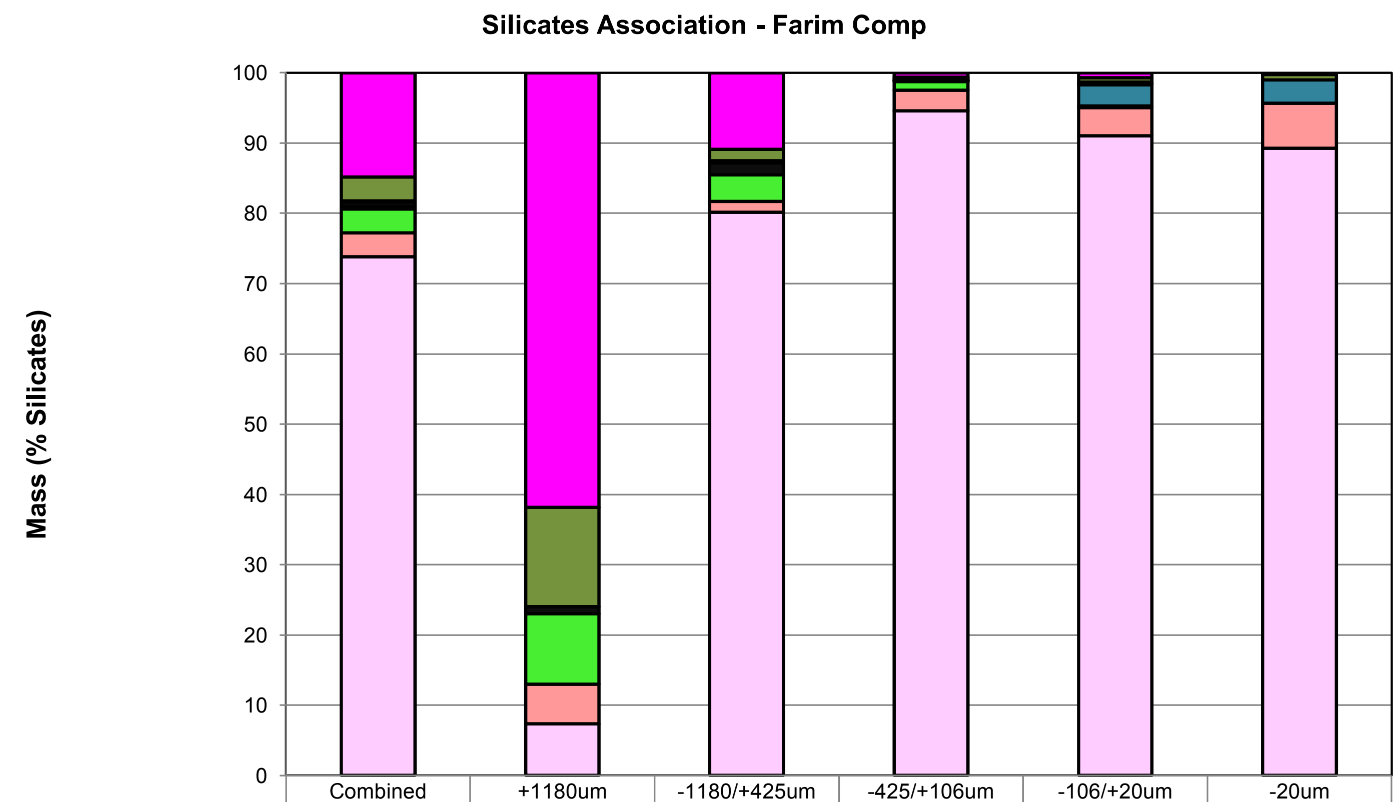
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Silicates Association



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	0.49	0.41	0.07	0.01	0.00	0.00
Sil:Carb:Ap	0.11	0.09	0.01	0.00	0.00	0.00
Silicates:Other	0.00	0.00	0.00	0.00	0.00	0.00
Silicates:Sulphides	0.01	0.00	0.00	0.00	0.00	0.00
Silicates:Carbonates	0.00	0.00	0.00	0.00	0.00	0.00
Silicates:Oxides	0.02	0.01	0.01	0.00	0.00	0.00
Silicates:Apatite Impure	0.01	0.00	0.00	0.00	0.01	0.00
Silicates:Apatite	0.11	0.07	0.02	0.02	0.00	0.00
Liberated Silicates	0.11	0.04	0.01	0.05	0.01	0.01
Free Silicates	2.43	0.05	0.53	1.52	0.20	0.13



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	14.9	61.8	10.9	0.59	0.70	0.24
Sil:Carb:Ap	3.36	14.2	1.62	0.27	0.64	0.74
Silicates:Other	0.00	0.00	0.00	0.00	0.00	0.00
Silicates:Sulphides	0.18	0.00	0.32	0.20	0.17	0.00
Silicates:Carbonates	0.01	0.00	0.00	0.00	0.04	0.00
Silicates:Oxides	0.60	1.03	1.73	0.07	0.19	0.00
Silicates:Apatite Impure	0.37	0.00	0.00	0.07	2.96	3.37
Silicates:Apatite	3.40	10.0	3.76	1.29	0.25	0.00
Liberated Silicates	3.40	5.63	1.53	2.92	4.04	6.37
Free Silicates	73.8	7.36	80.2	94.6	91.0	89.3

Absolute Mass of Silicates Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Silicates	2.43	0.05	0.53	1.52	0.20	0.13
Liberated Silicates	0.11	0.04	0.01	0.05	0.01	0.01
Silicates:Apatite	0.11	0.07	0.02	0.02	0.00	0.00
Silicates:Apatite Impure	0.01	0.00	0.00	0.00	0.01	0.00
Silicates:Oxides	0.02	0.01	0.01	0.00	0.00	0.00
Silicates:Carbonates	0.00	0.00	0.00	0.00	0.00	0.00
Silicates:Sulphides	0.01	0.00	0.00	0.00	0.00	0.00
Silicates:Other	0.00	0.00	0.00	0.00	0.00	0.00
Sil:Carb:Ap	0.11	0.09	0.01	0.00	0.00	0.00
Complex	0.49	0.41	0.07	0.01	0.00	0.00
Total	3.29	0.66	0.66	1.61	0.22	0.14
Total (% in fraction)	100.0	19.9	20.2	49.0	6.6	4.3

Normalized Mass of Silicates Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Silicates	73.8	7.36	80.2	94.6	91.0	89.3
Liberated Silicates	3.40	5.63	1.53	2.92	4.04	6.37
Silicates:Apatite	3.40	10.0	3.76	1.29	0.25	0.00
Silicates:Apatite Impure	0.37	0.00	0.00	0.07	2.96	3.37
Silicates:Oxides	0.60	1.03	1.73	0.07	0.19	0.00
Silicates:Carbonates	0.01	0.00	0.00	0.00	0.04	0.00
Silicates:Sulphides	0.18	0.00	0.32	0.20	0.17	0.00
Silicates:Other	0.00	0.00	0.00	0.00	0.00	0.00
Sil:Carb:Ap	3.36	14.2	1.62	0.27	0.64	0.74
Complex	14.9	61.8	10.9	0.59	0.70	0.24
Total	100.0	100.0	100.0	100.0	100.0	100.0
Liberated	77.2	13.0	81.7	97.5	95.1	95.6

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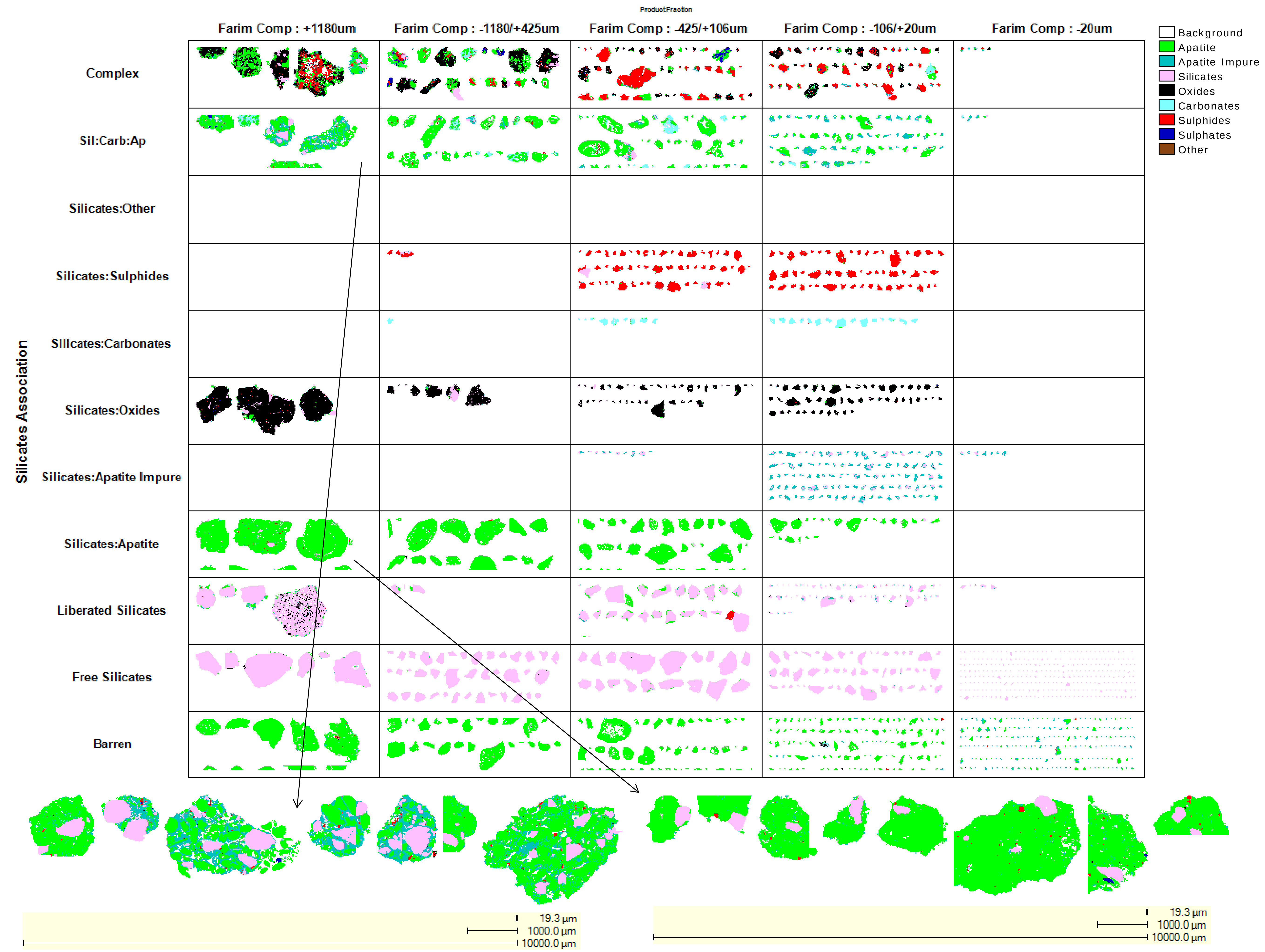
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Silicates Association Image Grid



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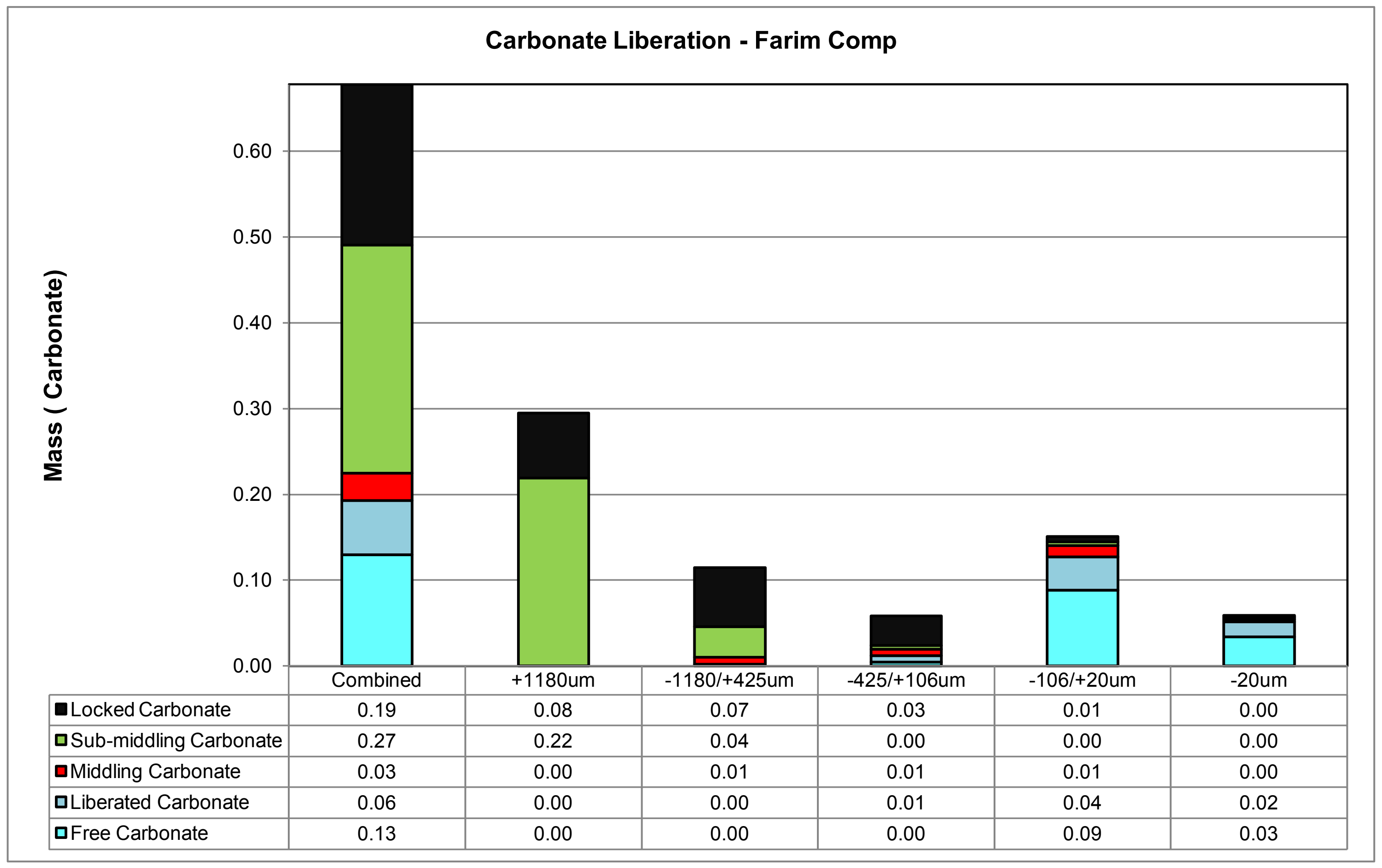
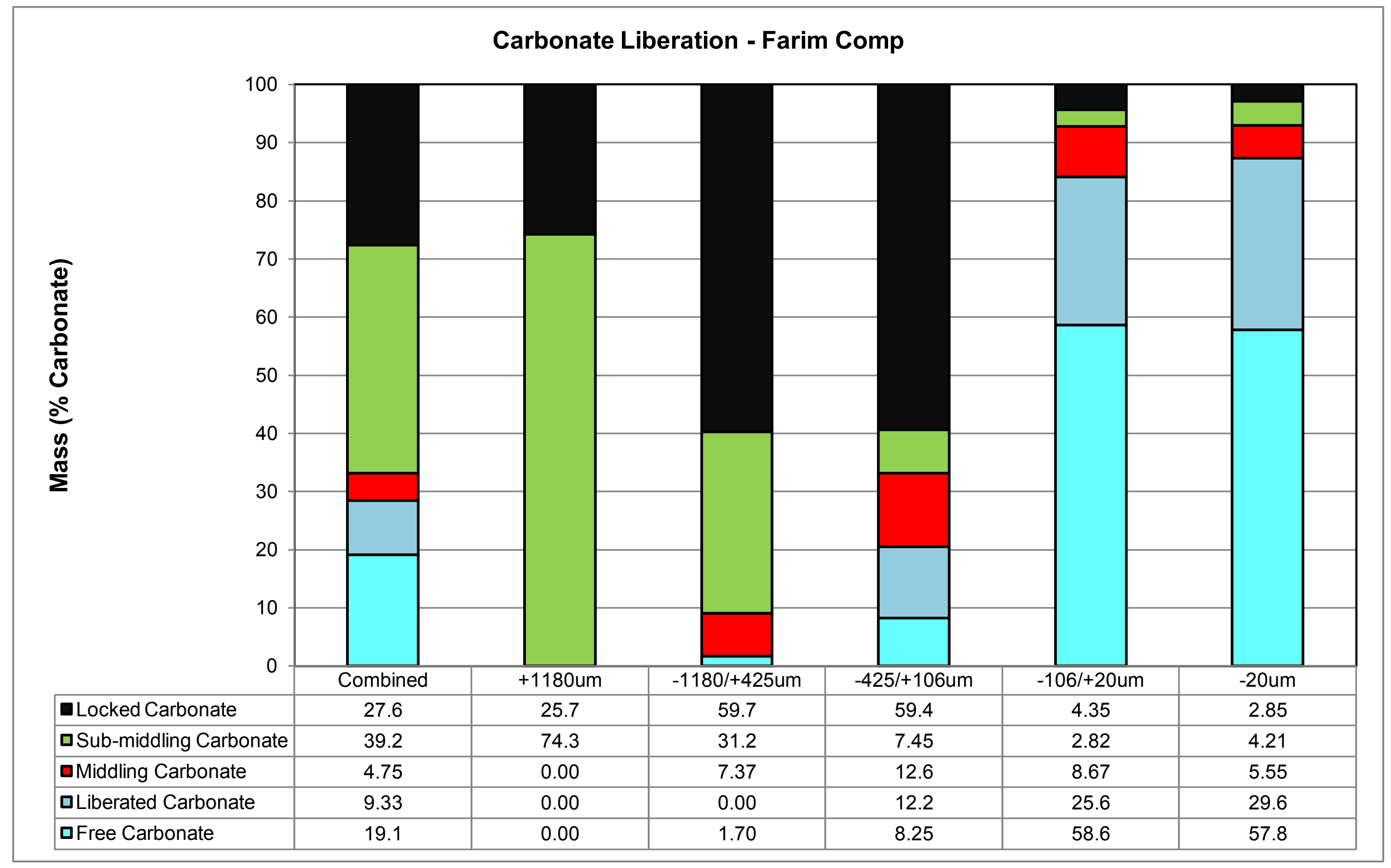
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Carbonate Liberation





Absolute Mass of Carbonate Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Carbonate	0.13	0.00	0.00	0.00	0.09	0.03
Liberated Carbonate	0.06	0.00	0.00	0.01	0.04	0.02
Middling Carbonate	0.03	0.00	0.01	0.01	0.01	0.00
Sub-middling Carbonate	0.27	0.22	0.04	0.00	0.00	0.00
Locked Carbonate	0.19	0.08	0.07	0.03	0.01	0.00
Total	0.68	0.29	0.11	0.06	0.15	0.06
Total (% in fraction)	100.0	43.5	16.9	8.62	22.3	8.68

Normalized Mass of Carbonate Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Carbonate	19.1	0.00	1.70	8.25	58.6	57.8
Liberated Carbonate	9.33	0.00	0.00	12.2	25.6	29.6
Middling Carbonate	4.75	0.00	7.37	12.6	8.67	5.55
Sub-middling Carbonate	39.2	74.3	31.2	7.45	2.82	4.21
Locked Carbonate	27.6	25.7	59.7	59.4	4.35	2.85
Total	100.0	100.0	100.0	100.0	100.0	100.0

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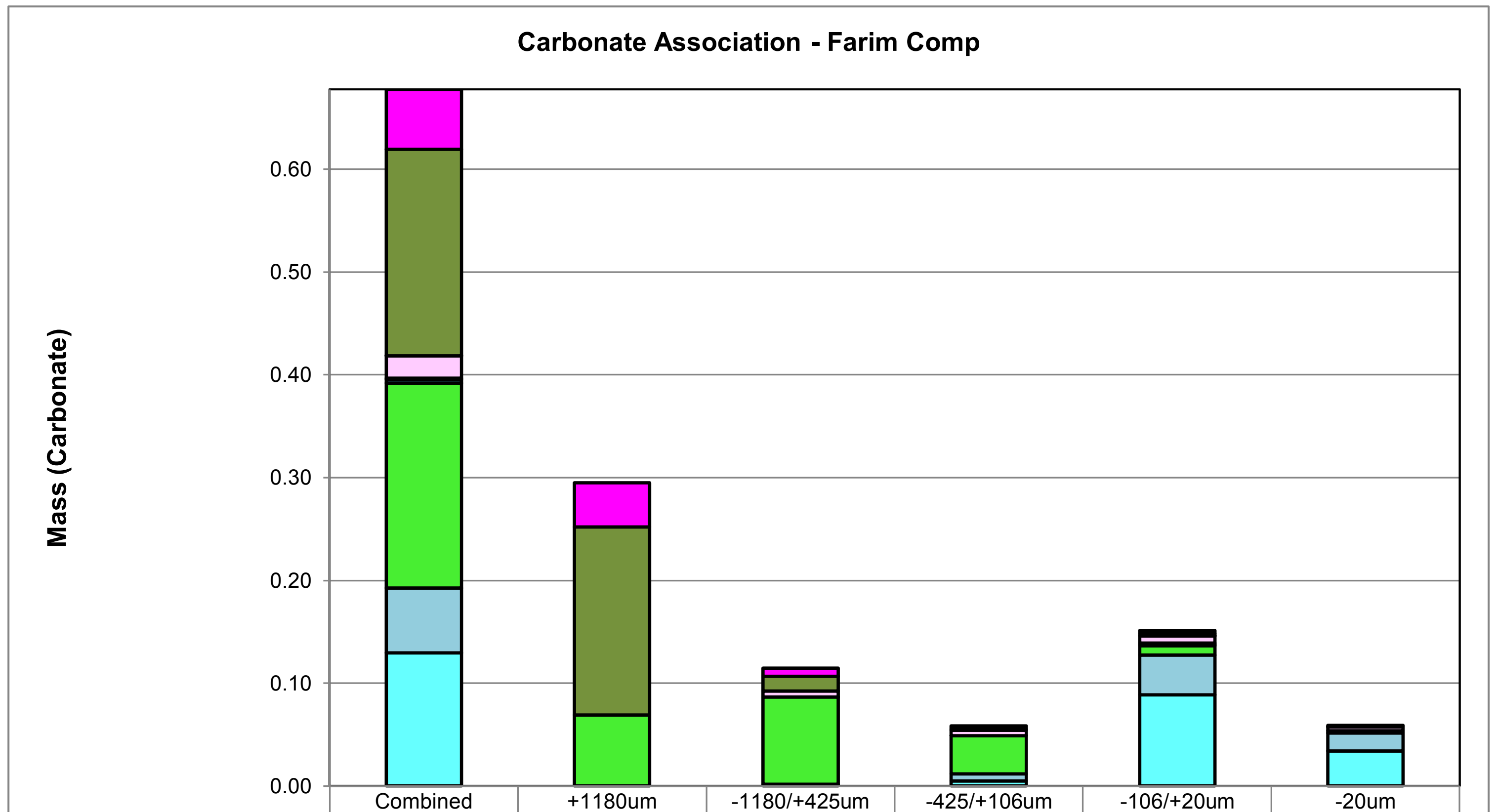
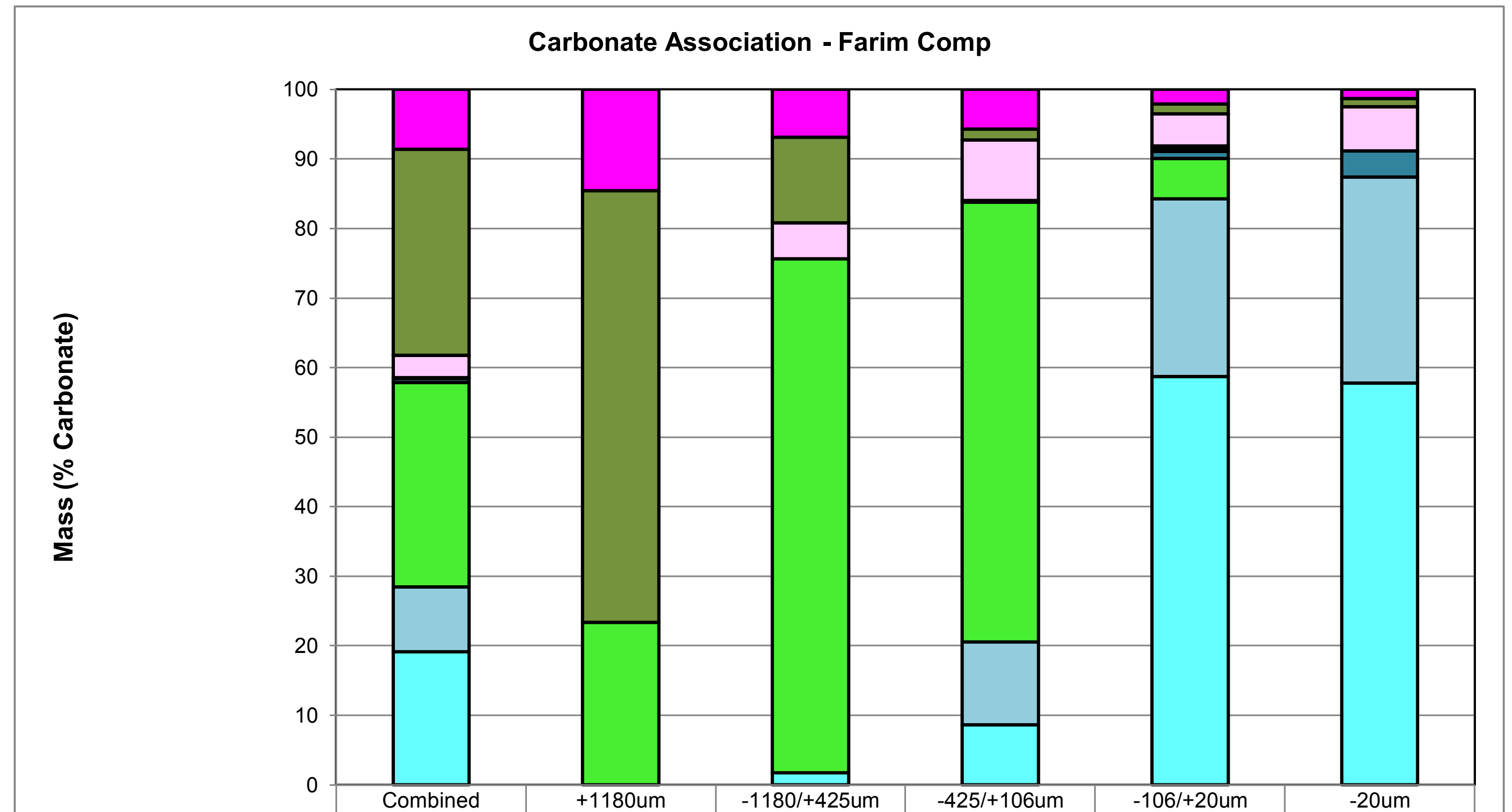
Carbonate Association



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	0.06	0.04	0.01	0.00	0.00	0.00
Carb: Sil: Ap	0.20	0.18	0.01	0.00	0.00	0.00
Carbonate: Other	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Sulphides	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Silicates	0.02	0.00	0.01	0.01	0.01	0.00
Carbonate: Oxides	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Apatite Impure	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate:Apatite	0.20	0.07	0.08	0.04	0.01	0.00
Liberated Carbonate	0.06	0.00	0.00	0.01	0.04	0.02
Free Carbonate	0.13	0.00	0.00	0.01	0.09	0.03



	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Complex	8.58	14.6	6.91	5.70	2.12	1.31
Carb: Sil: Ap	29.6	62.1	12.3	1.54	1.41	1.19
Carbonate: Other	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Sulphides	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Silicates	3.21	0.00	5.16	8.71	4.64	6.32
Carbonate: Oxides	0.20	0.00	0.00	0.29	0.78	0.00
Carbonate: Apatite Impure	0.55	0.00	0.00	0.00	1.00	3.79
Carbonate:Apatite	29.4	23.3	73.9	63.2	5.77	0.00
Liberated Carbonate	9.30	0.00	0.00	12.0	25.6	29.6
Free Carbonate	19.1	0.00	1.70	8.61	58.7	57.8

Absolute Mass of Carbonate Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Carbonate	0.13	0.00	0.00	0.01	0.09	0.03
Liberated Carbonate	0.06	0.00	0.00	0.01	0.04	0.02
Carbonate:Apatite	0.20	0.07	0.08	0.04	0.01	0.00
Carbonate: Apatite Impure	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Oxides	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Silicates	0.02	0.00	0.01	0.01	0.01	0.00
Carbonate: Sulphides	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Other	0.00	0.00	0.00	0.00	0.00	0.00
Carb: Sil: Ap	0.20	0.18	0.01	0.00	0.00	0.00
Complex	0.06	0.04	0.01	0.00	0.00	0.00
Total	0.68	0.29	0.11	0.06	0.15	0.06
Total (% in fraction)	100.0	43.5	16.9	8.6	22.3	8.7

Normalized Mass of Carbonate Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Carbonate	19.1	0.00	1.70	8.61	58.7	57.8
Liberated Carbonate	9.30	0.00	0.00	12.0	25.6	29.6
Carbonate:Apatite	29.4	23.3	73.9	63.2	5.77	0.00
Carbonate: Apatite Impure	0.55	0.00	0.00	0.00	1.00	3.79
Carbonate: Oxides	0.20	0.00	0.00	0.29	0.78	0.00
Carbonate: Silicates	3.21	0.00	5.16	8.71	4.64	6.32
Carbonate: Sulphides	0.00	0.00	0.00	0.00	0.00	0.00
Carbonate: Other	0.00	0.00	0.00	0.00	0.00	0.00
Carb: Sil: Ap	29.6	62.1	12.3	1.54	1.41	1.19
Complex	8.58	14.6	6.91	5.70	2.12	1.31
Total	100.0	100.0	100.0	100.0	100.0	100.0
Liberated	28.4	0.00	1.70	20.6	84.3	87.4

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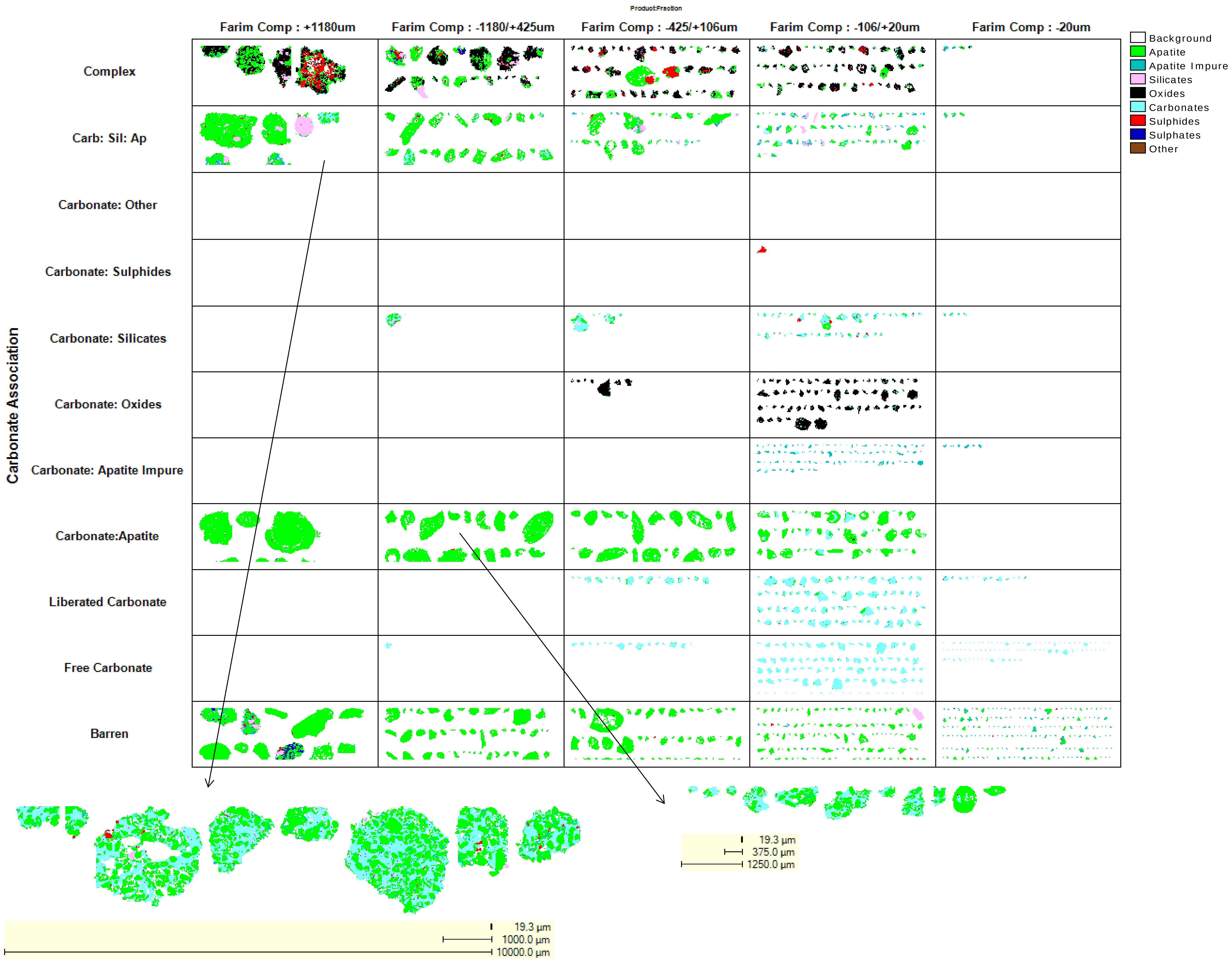
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Carbonate Association Image Grid



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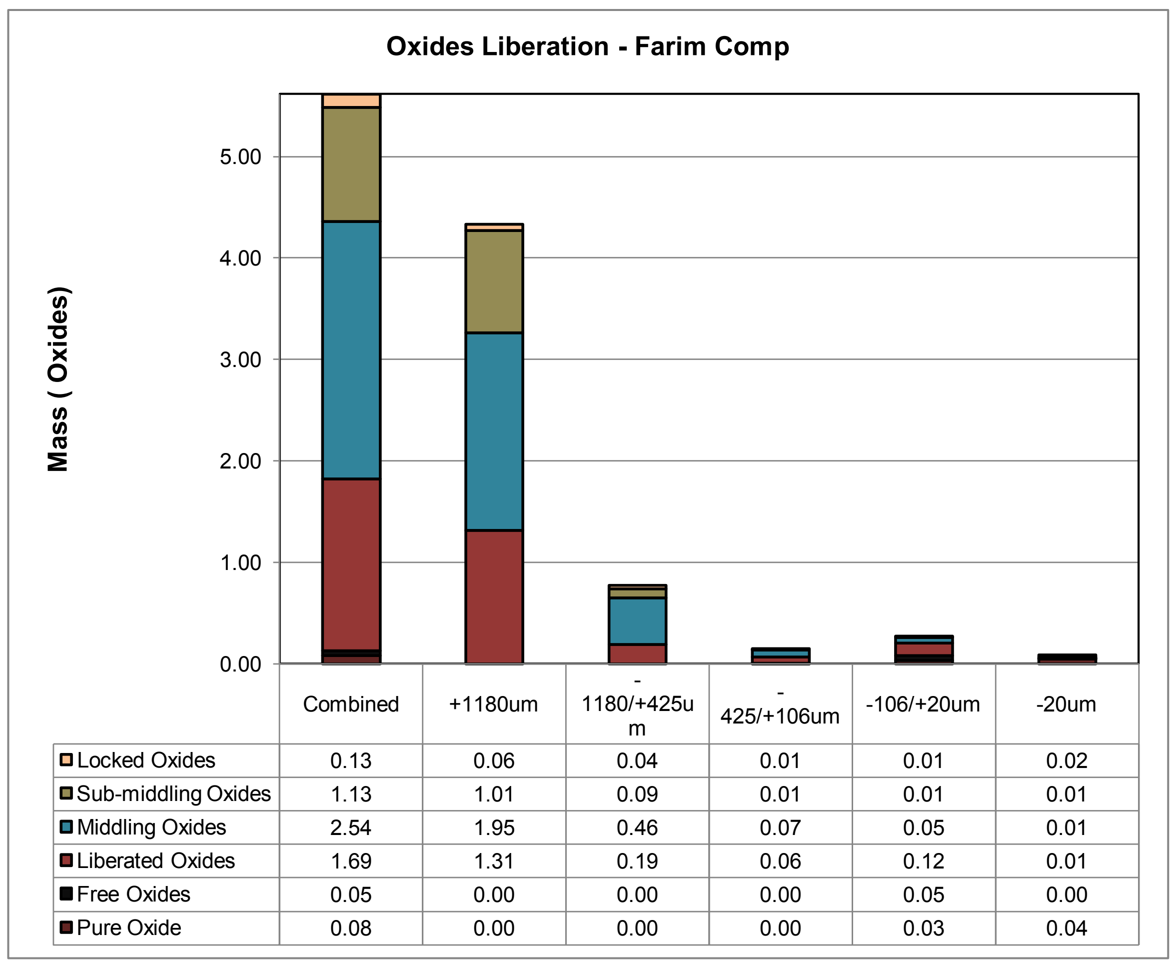
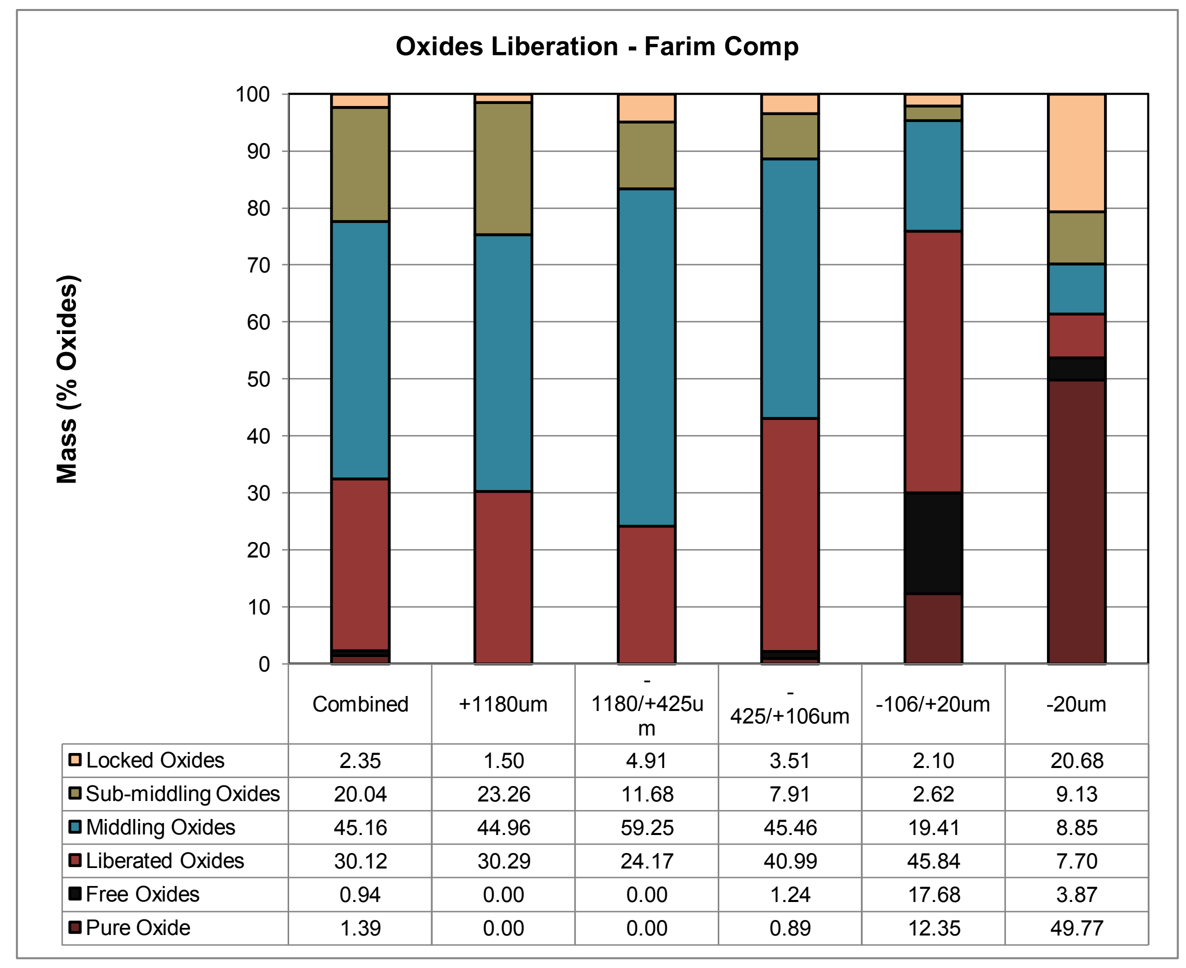
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Oxides Liberation





Absolute Mass of Oxides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Pure Oxide	0.08	0.00	0.00	0.00	0.03	0.04
Free Oxides	0.05	0.00	0.00	0.00	0.05	0.00
Liberated Oxides	1.69	1.31	0.19	0.06	0.12	0.01
Middling Oxides	2.54	1.95	0.46	0.07	0.05	0.01
Sub-middling Oxides	1.13	1.01	0.09	0.01	0.01	0.01
Locked Oxides	0.13	0.06	0.04	0.01	0.01	0.02
Total	5.62	4.33	0.78	0.15	0.27	0.09
Total (% in fraction)	100.0	77.1	13.8	2.7	4.8	1.6

Normalized Mass of Oxides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Pure Oxide	1.39	0.00	0.00	0.89	12.35	49.77
Free Oxides	0.94	0.00	0.00	1.24	17.68	3.87
Liberated Oxides	30.12	30.29	24.17	40.99	45.84	7.70
Middling Oxides	45.16	44.96	59.25	45.46	19.41	8.85
Sub-middling Oxides	20.04	23.26	11.68	7.91	2.62	9.13
Locked Oxides	2.35	1.50	4.91	3.51	2.10	20.68
Total	100.0	100.0	100.0	100.0	100.0	100.0

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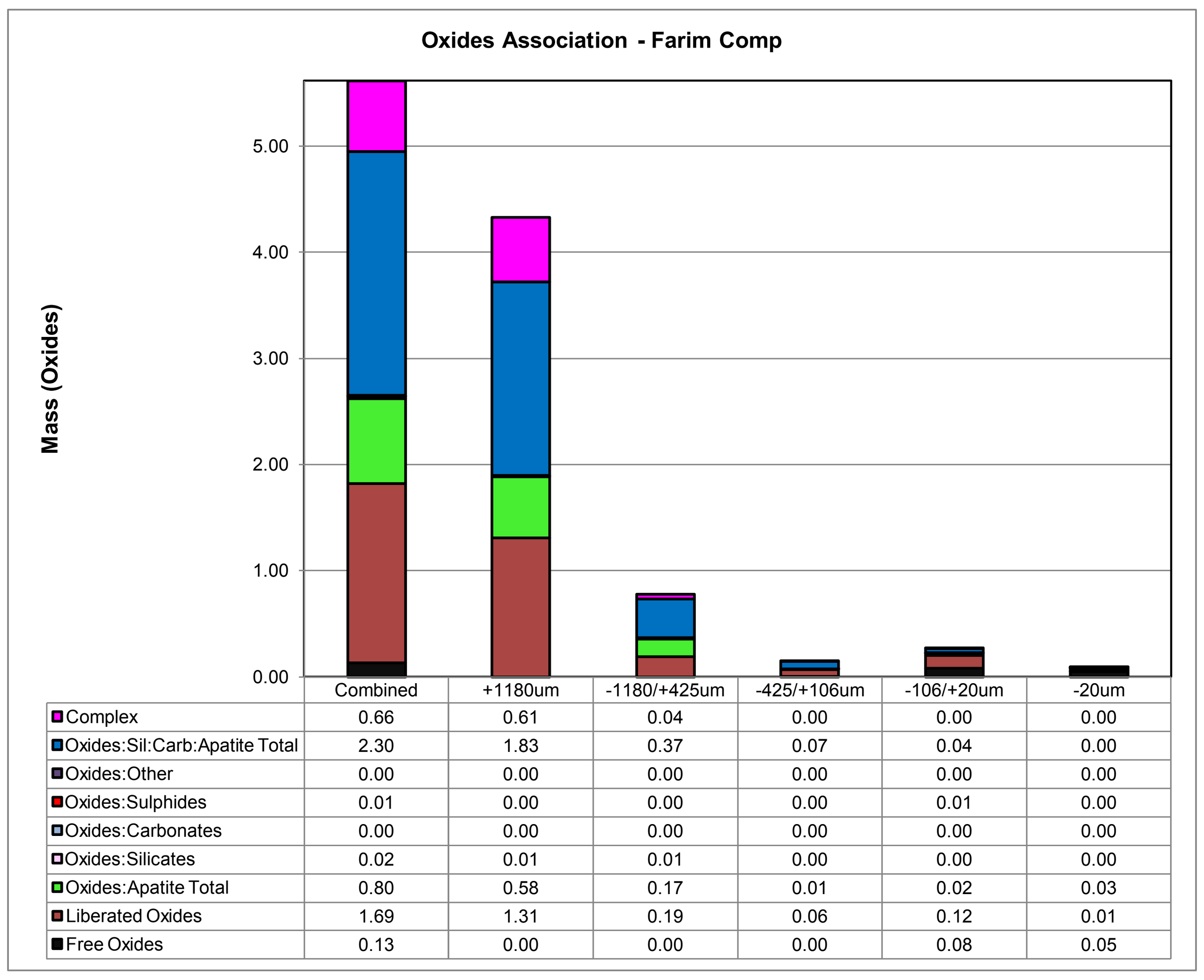
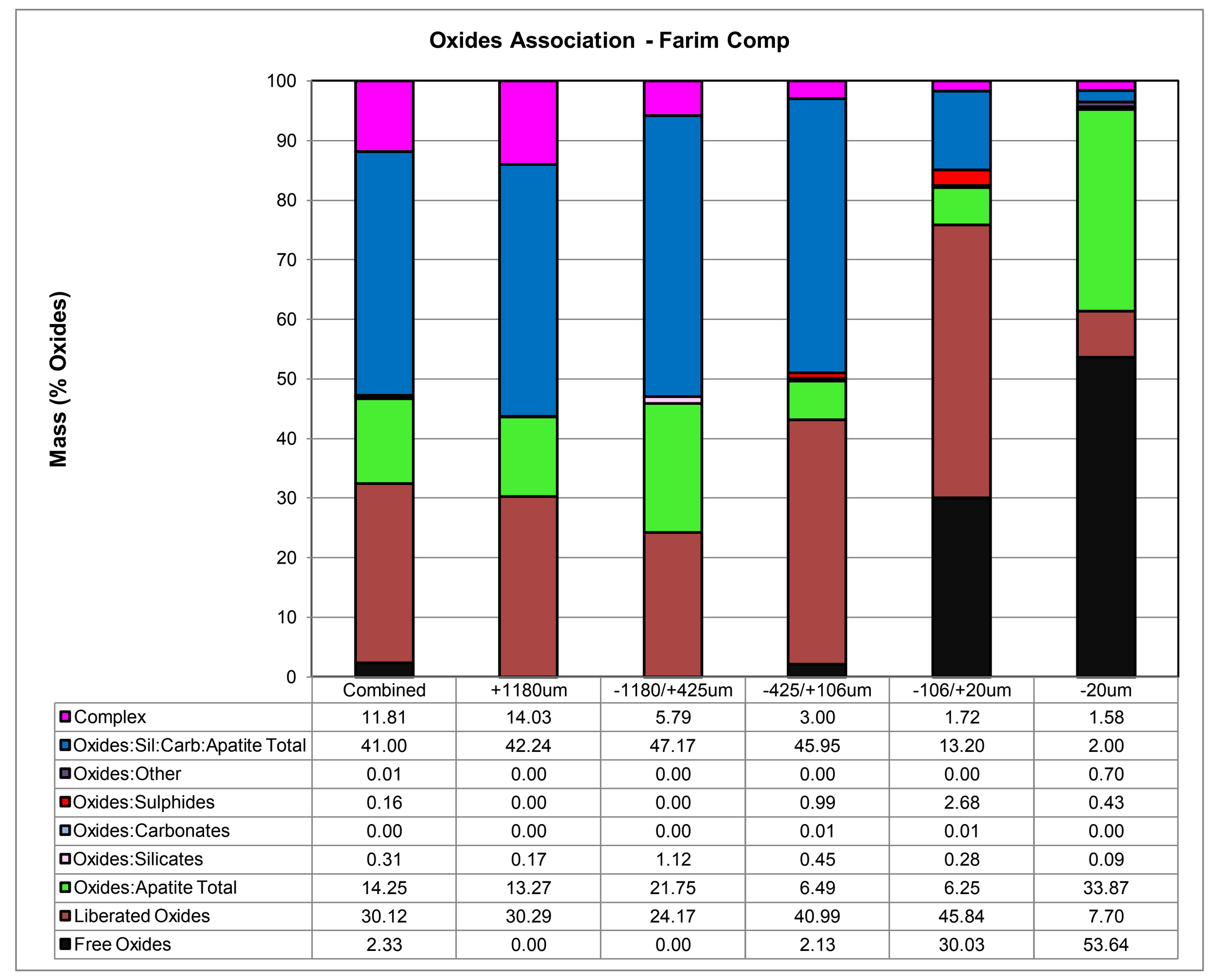
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Oxides Association





Absolute Mass of Oxides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Oxides	0.13	0.00	0.00	0.00	0.08	0.05
Liberated Oxides	1.69	1.31	0.19	0.06	0.12	0.01
Oxides:Apatite Total	0.80	0.58	0.17	0.01	0.02	0.03
Oxides:Silicates	0.02	0.01	0.01	0.00	0.00	0.00
Oxides:Carbonates	0.00	0.00	0.00	0.00	0.00	0.00
Oxides:Sulphides	0.01	0.00	0.00	0.00	0.01	0.00
Oxides:Other	0.00	0.00	0.00	0.00	0.00	0.00
Oxides:Sil:Carb:Apatite Total	2.30	1.83	0.37	0.07	0.04	0.00
Complex	0.66	0.61	0.04	0.00	0.00	0.00
Total	5.62	4.33	0.78	0.15	0.27	0.09
Total (% in fraction)	100.0	77.1	13.8	2.7	4.8	1.6

Normalized Mass of Oxides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Oxides	2.33	0.00	0.00	2.13	30.03	53.64
Liberated Oxides	30.12	30.29	24.17	40.99	45.84	7.70
Oxides:Apatite Total	14.25	13.27	21.75	6.49	6.25	33.87
Oxides:Silicates	0.31	0.17	1.12	0.45	0.28	0.09
Oxides:Carbonates	0.00	0.00	0.00	0.01	0.01	0.00
Oxides:Sulphides	0.16	0.00	0.00	0.99	2.68	0.43
Oxides:Other	0.01	0.00	0.00	0.00	0.00	0.70
Oxides:Sil:Carb:Apatite Total	41.00	42.24	47.17	45.95	13.20	2.00
Complex	11.81	14.03	5.79	3.00	1.72	1.58
Total	100.0	100.0	100.0	100.0	100.0	100.0
Liberated	2.333736742	0	0	2.131806358	30.03439037	53.64030244

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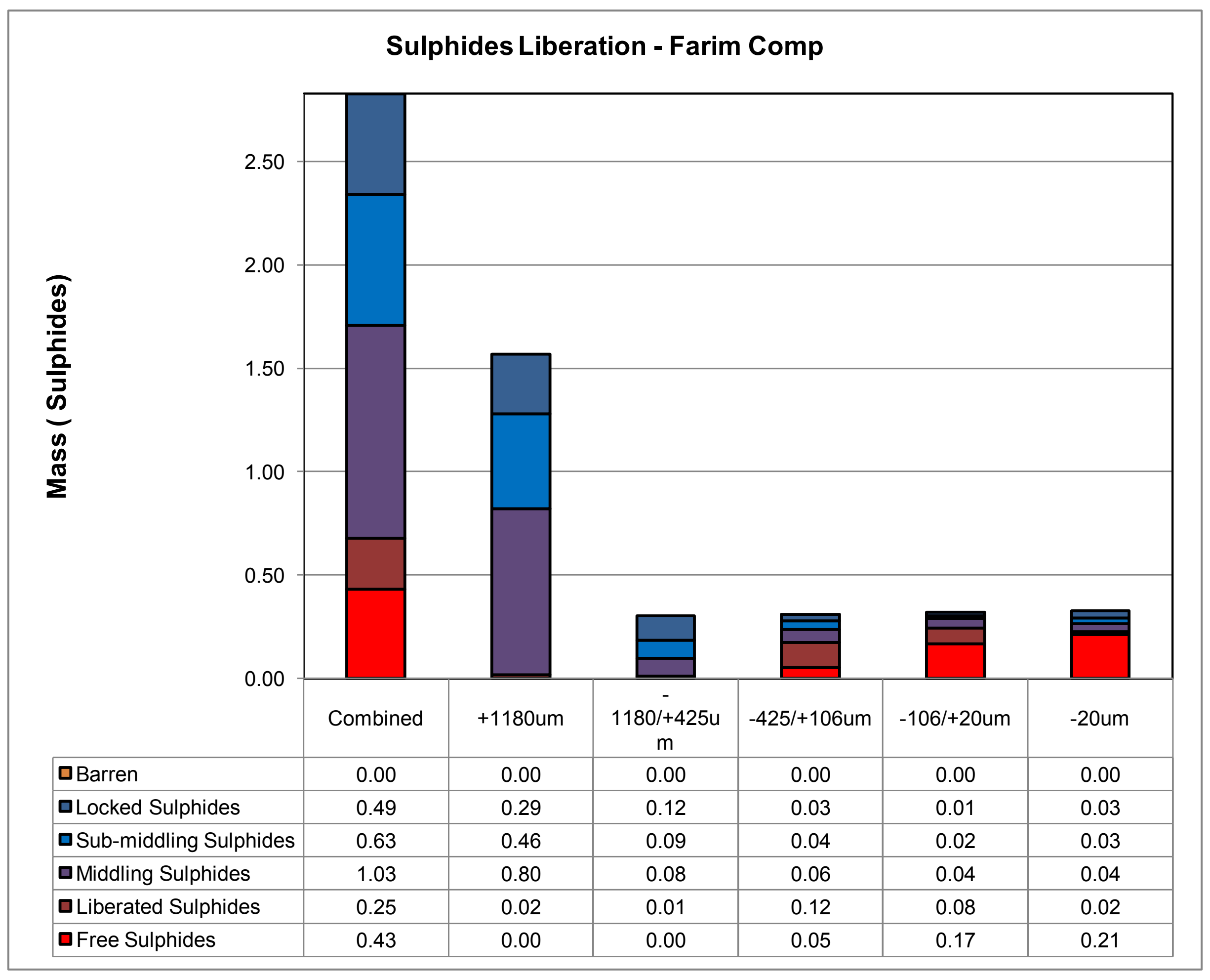
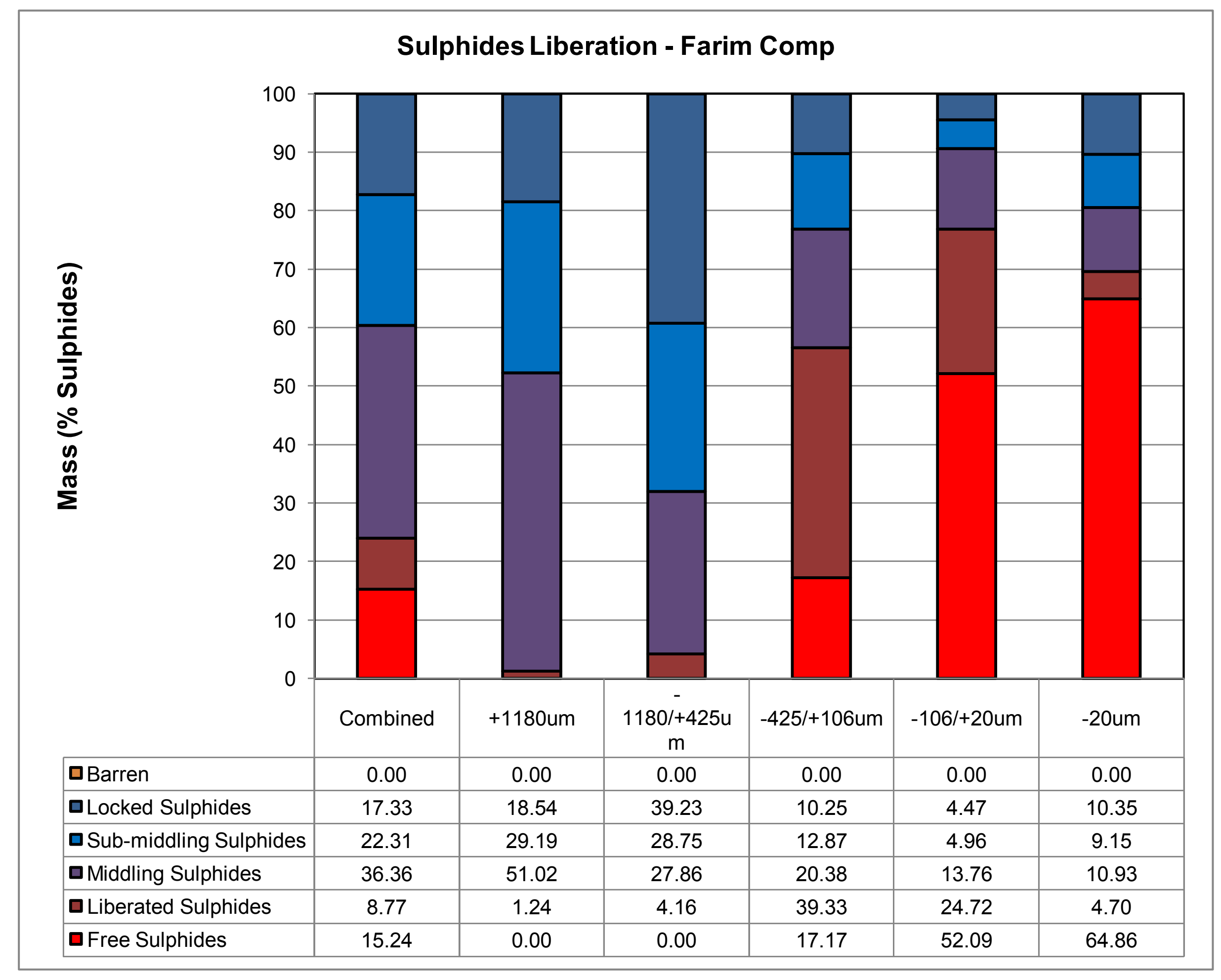
Sulphides Liberation





Absolute Mass of Sulphides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Sulphides	0.43	0.00	0.00	0.05	0.17	0.21
Liberated Sulphides	0.25	0.02	0.01	0.12	0.08	0.02
Middling Sulphides	1.03	0.80	0.08	0.06	0.04	0.04
Sub-middling Sulphides	0.63	0.46	0.09	0.04	0.02	0.03
Locked Sulphides	0.49	0.29	0.12	0.03	0.01	0.03
Barren	0.00	0.00	0.00	0.00	0.00	0.00
Total	2.83	1.57	0.30	0.31	0.32	0.33
Total (% in fraction)	100.0	55.5	10.7	10.9	11.3	11.6

Normalized Mass of Sulphides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Sulphides	15.24	0.00	0.00	17.17	52.09	64.86
Liberated Sulphides	8.77	1.24	4.16	39.33	24.72	4.70
Middling Sulphides	36.36	51.02	27.86	20.38	13.76	10.93
Sub-middling Sulphides	22.31	29.19	28.75	12.87	4.96	9.15
Locked Sulphides	17.33	18.54	39.23	10.25	4.47	10.35
Barren	0.00	0.00	0.00	0.00	0.00	0.00
Total	100.0	100.0	100.0	100.0	100.0	100.0

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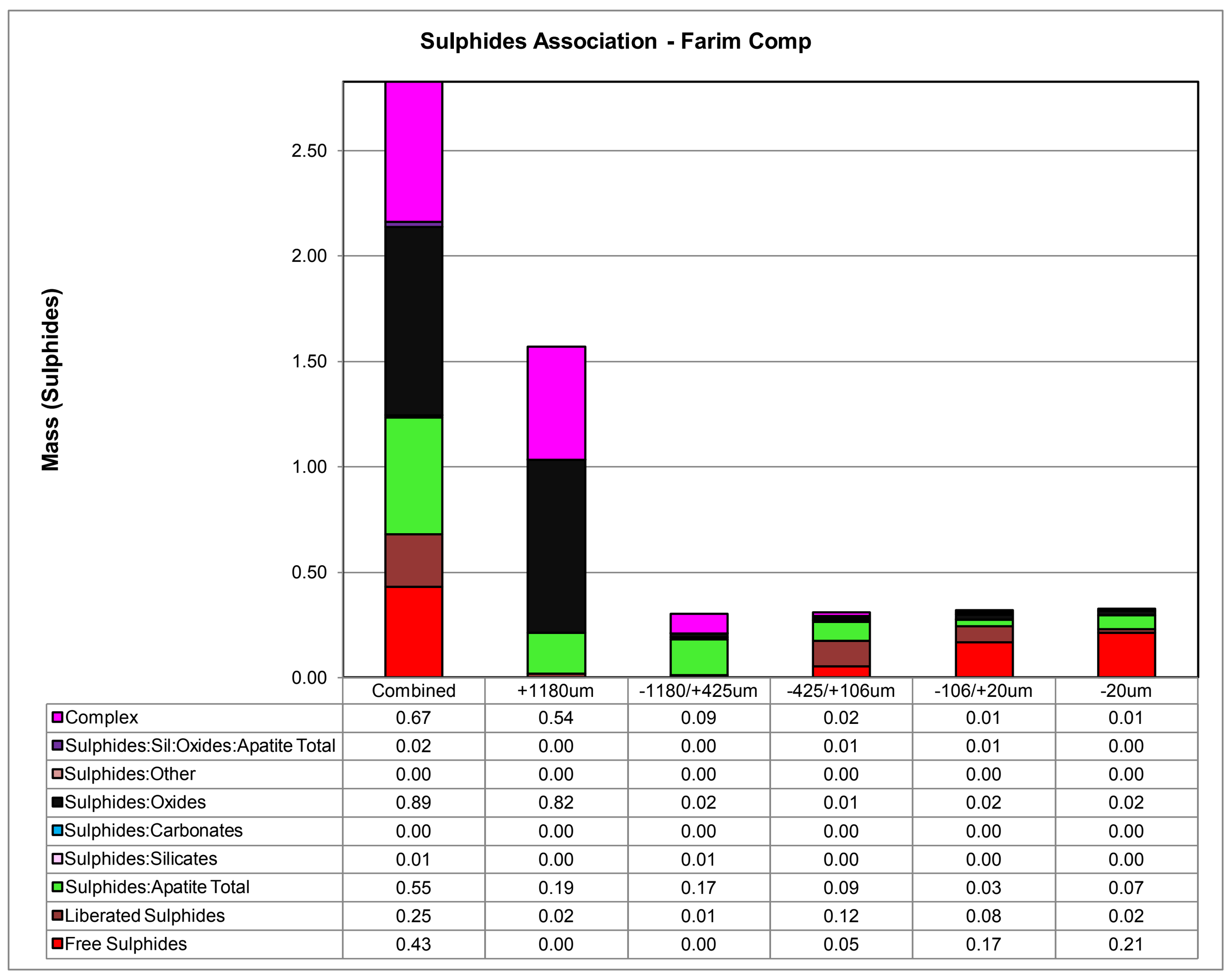
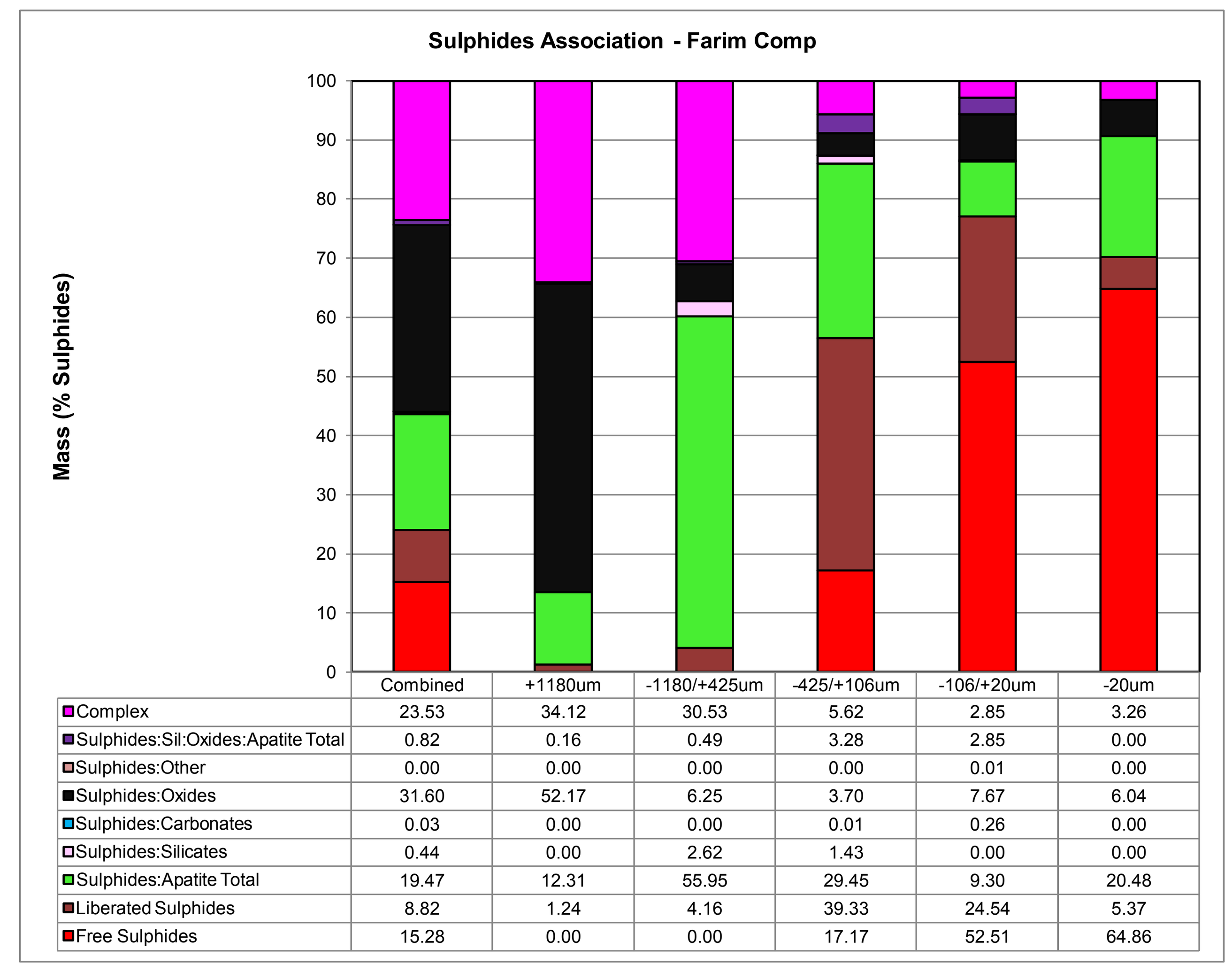
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Sulphides Association





Absolute Mass of Sulphides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Sulphides	0.43	0.00	0.00	0.05	0.17	0.21
Liberated Sulphides	0.25	0.02	0.01	0.12	0.08	0.02
Sulphides:Apatite Total	0.55	0.19	0.17	0.09	0.03	0.07
Sulphides:Silicates	0.01	0.00	0.01	0.00	0.00	0.00
Sulphides:Carbonates	0.00	0.00	0.00	0.00	0.00	0.00
Sulphides:Oxides	0.89	0.82	0.02	0.01	0.02	0.02
Sulphides:Other	0.00	0.00	0.00	0.00	0.00	0.00
Sulphides:Sil:Oxides:Apatite Total	0.02	0.00	0.00	0.01	0.01	0.00
Complex	0.67	0.54	0.09	0.02	0.01	0.01
Total	2.83	1.57	0.30	0.31	0.32	0.33
Total (% in fraction)	100.0	55.5	10.7	10.9	11.3	11.6

Normalized Mass of Sulphides Across Fraction Farim Comp

Mineral Name	Combined	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Free Sulphides	15.28	0.00	0.00	17.17	52.51	64.86
Liberated Sulphides	8.82	1.24	4.16	39.33	24.54	5.37
Sulphides:Apatite Total	19.47	12.31	55.95	29.45	9.30	20.48
Sulphides:Silicates	0.44	0.00	2.62	1.43	0.00	0.00
Sulphides:Carbonates	0.03	0.00	0.00	0.01	0.26	0.00
Sulphides:Oxides	31.60	52.17	6.25	3.70	7.67	6.04
Sulphides:Other	0.00	0.00	0.00	0.00	0.01	0.00
Sulphides:Sil:Oxides:Apatite Total	0.82	0.16	0.49	3.28	2.85	0.00
Complex	23.53	34.12	30.53	5.62	2.85	3.26
Total	100.0	100.0	100.0	100.0	100.0	100.0
Liberated	24.1072913	1.24116727	4.15515527	56.5036699	77.0475425	70.228158

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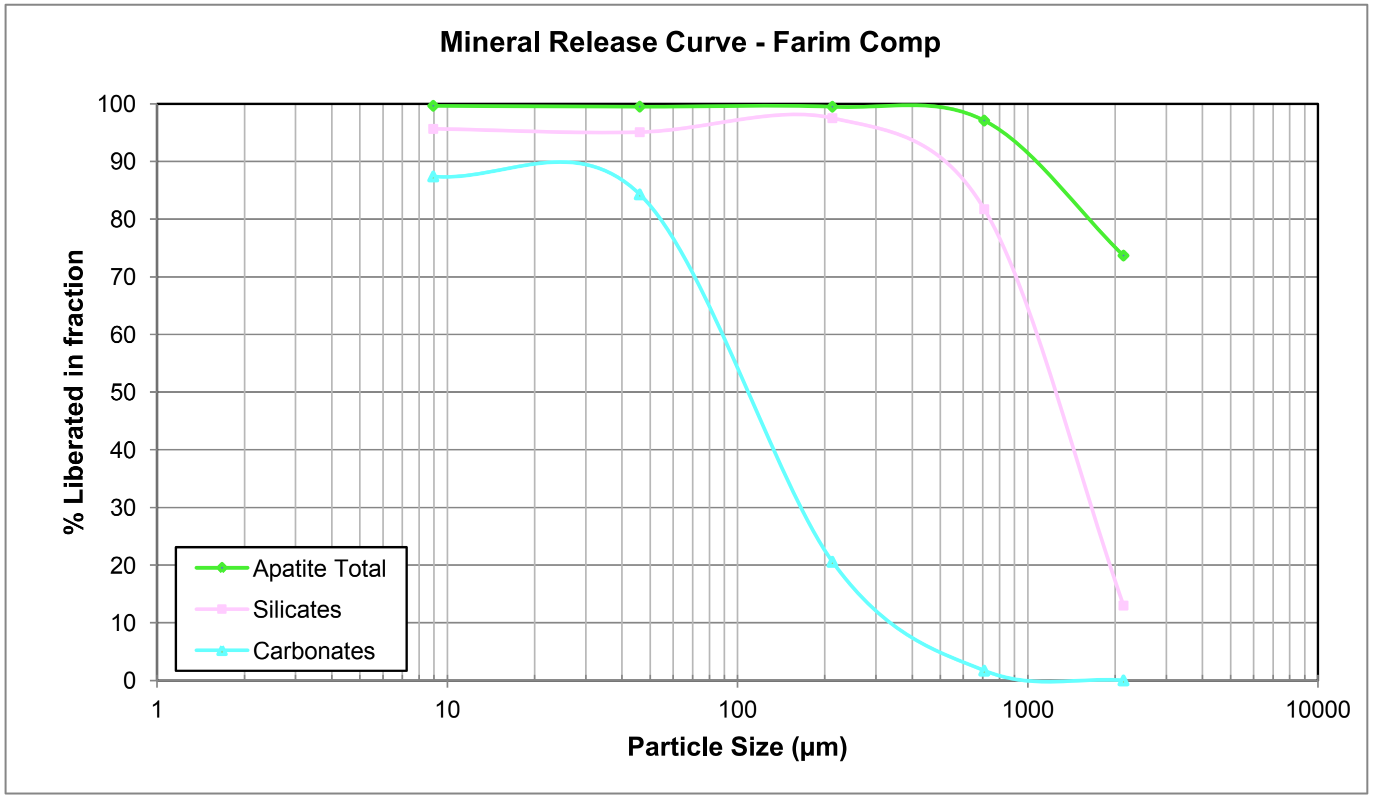
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Mineral Release Curves



Sample	Farim Comp				
Fraction	+1180um	-1180/+425um	-425/+106um	-106/+20um	-20um
Average Particle Size (µm)	2137.5	708.2	212.2	46.0	8.94
Mineral Mass % 80% Lib					
Apatite Total	73.7	97.1	99.5	99.5	99.6
Silicates	13.0	81.7	97.5	95.1	95.6
Carbonates	0.00	1.70	20.6	84.3	87.4

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13478-003
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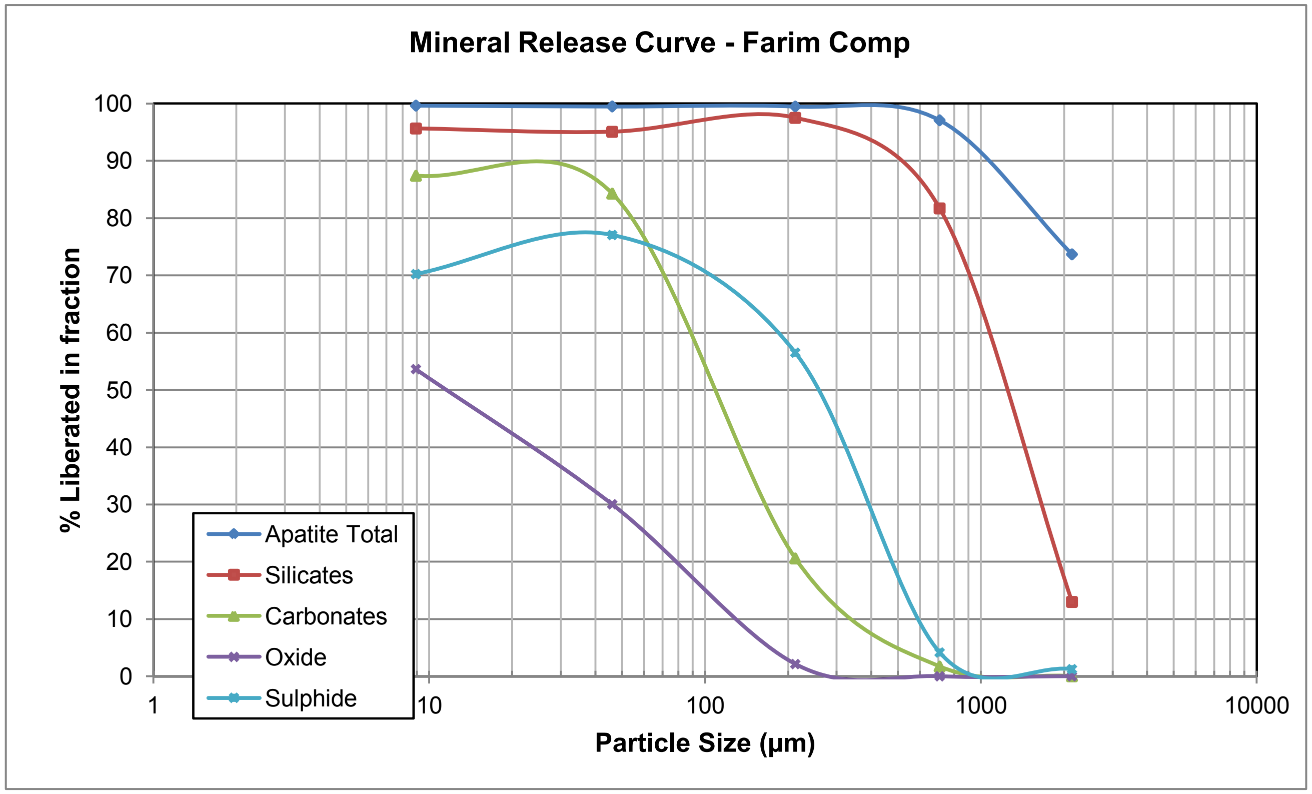
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Mineral Release Curves



Sample	Farim Comp				
Fraction					
Average Particle Size (µm)	2137.51	708.17	212.25	46.04	8.94
Mineral Mass % 80% Lib					
Apatite Total	73.67	97.08	99.48	99.50	99.64
Silicates	12.99	81.69	97.50	95.07	95.65
Carbonates	0.00	1.70	20.58	84.27	87.39
Oxide	0.00	0.00	2.13	30.03	53.64
Sulphide	1.24	4.16	56.50	77.05	70.23

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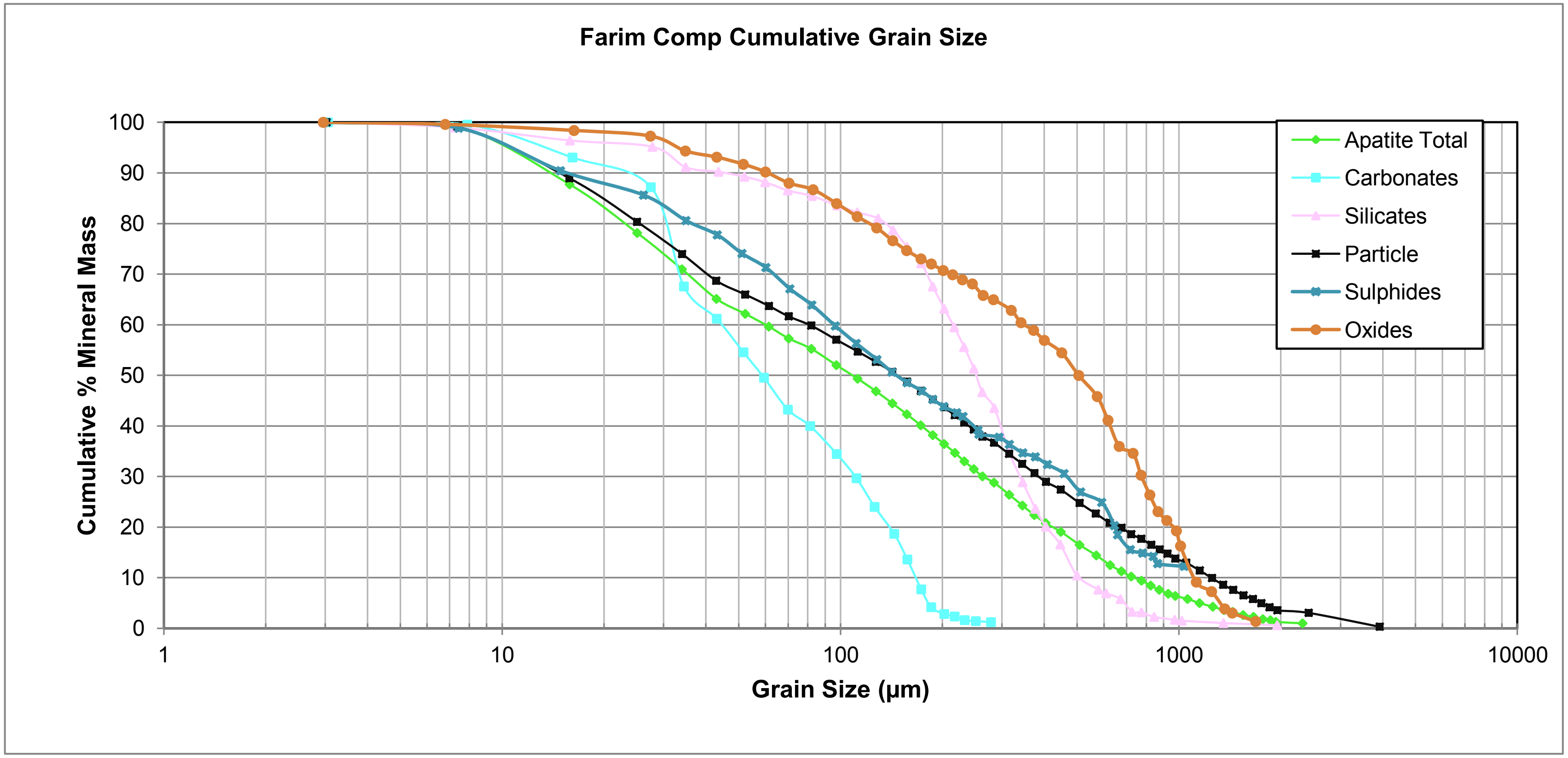
Cumulative Grain Size Distribution



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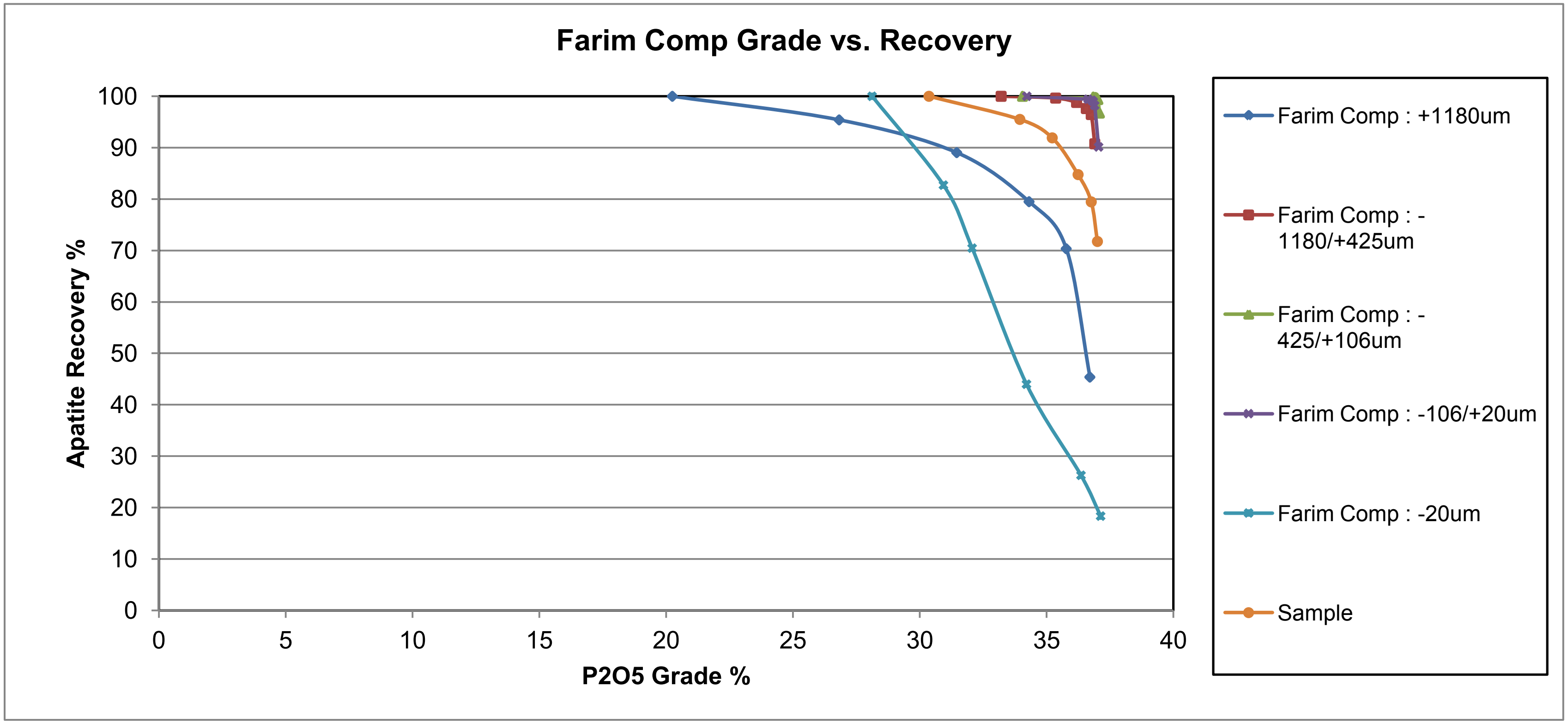
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

Apatite Grade vs. Recovery: Farim Comp



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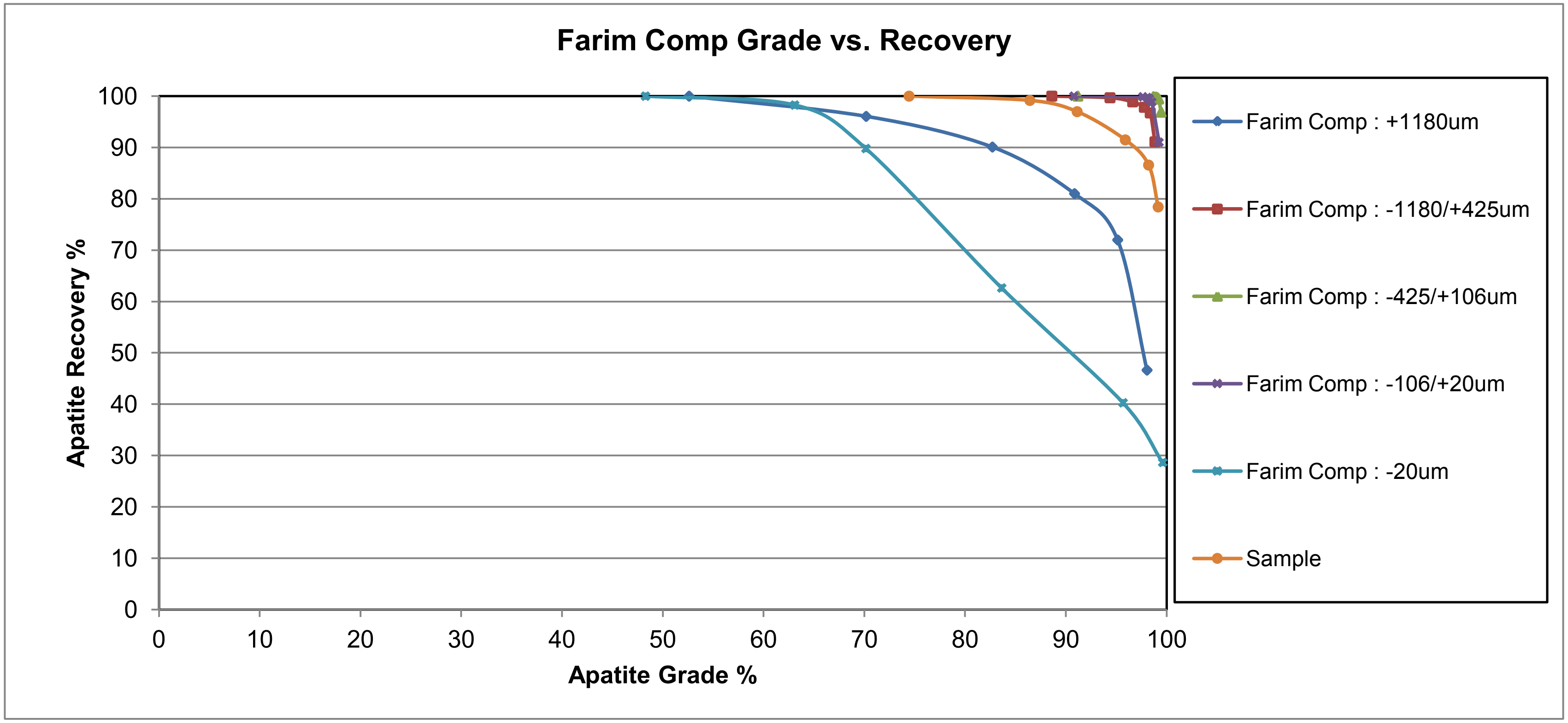
High Definition Mineralogical Analysis using QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy)

<u>Apatite Grade vs. Recovery: Farim Comp</u>



Appendix B – X-Ray Diffraction Analysis

Qualitative X-Ray Diffraction

Report Prepared for:	*Lycopodium Minerals Canada Ltd*
Project Number/ LIMS No.	*13478-003/MI5021-MAR15*
Sample Receipt:	*March 23, 2015*
Sample Analysis:	*April 10, 2015*
Reporting Date:	*April 16, 2015*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Summary of Mineral Asemblages
3) XRD Pattern(s)

"*Kathryn Sheridan*" (signed by)

Kathryn Sheridan, H.B.Sc., P.Geo.
Mineralogist

"*Huyun Zhou*" (signed by)

Huyun Zhou, Ph.D., P.Geo.
Senior Mineralogist

ACCREDITATION: SGS Minerals Services Lakefield is accredited to the requirements of ISO/IEC 17025 for specific tests as listed on our scope of accreditation, including geochemical, mineralogical and trade mineral tests. To view a list of the accredited methods, please visit the following website and search SGS Canada - Minerals Services - Lakefield: http://palcan.scc.ca/SpecsSearch/GLSearchForm.do.

SGS Minerals
a division of SGS Canada Inc.
P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0
Tel: (705) 652-2000 Fax: (705) 652-6365 www.sgs.com www.sgs.com/met
Member of the SGS Group (SGS SA)

Method Summary

The Qualitative Mineral Identification By XRD (ME-LR-MIN-MET-MN-D01) method used by SGS Minerals Services is accredited to the requirements of ISO/IEC 17025.

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown test sample to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) and released on software as a database of Powder Diffraction Files (PDF).

Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds. Mineral proportions are based on relative peak heights and may be strongly influenced by crystallinity, structural group or preferred orientations. Interpretations and relative proportions should be accompanied by supporting petrographic and geochemical data (Whole Rock Analysis, Inductively Coupled Plasma - Optical Emission Spectroscopy, etc.).

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Member of the SGS Group (SGS SA)

Lycopodium Minerals Canada Ltd
13478-003/MI5021-MAR15
04/16/2015

Summary of Qualitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample ID	Major	Moderate	Minor	Trace
(13) Farim Comp -1180/+425um	apatite	-	quartz	-
(27) Farim Comp -20um	apatite	-	quartz	-

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Apatite	$Ca_5(PO_4)_3(F,Cl,OH)$
Quartz	SiO_2

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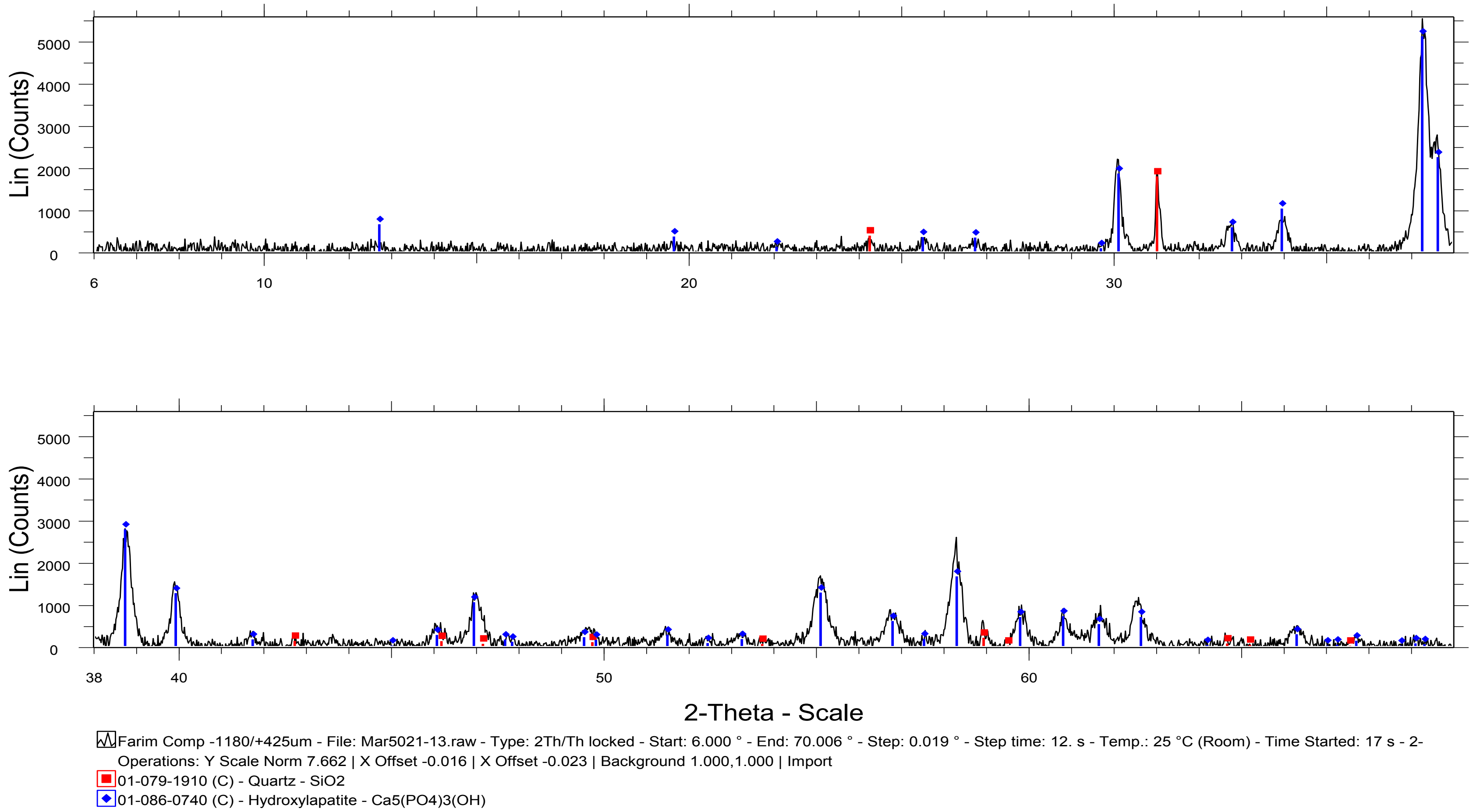
04/16/2015

Farim Comp -1180/+425um



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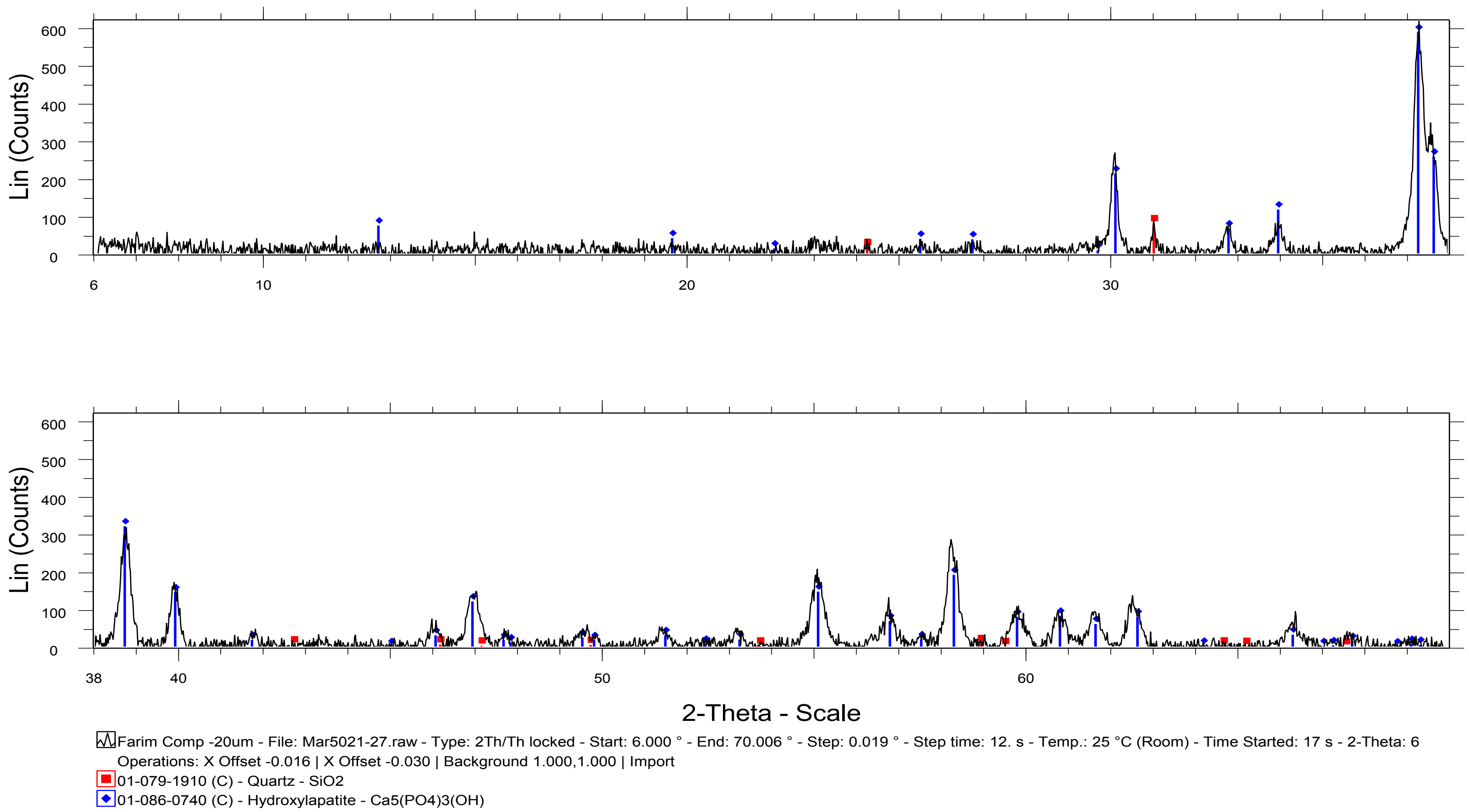
04/16/2015

Farim Comp -20um



Appendix C – Chemical Certificate of Analysis

SGS Canada Inc.
P.O. Box 4300 - 185 Concession St.
Lakefield - Ontario - KOL 2HO
Phone: 705-652-2000 FAX: 705-652-6365

LR Internal Dept 14
Attn : Chris Gunning / Elaine Glover

---, ---
---, ---
Phone: ---, Fax:---

27-April-2015

Date Rec. : 26 March 2015
LR Report : **CA03058-MAR15**
Project : CALR-13478-003
Client Ref : MI5021-MAR15

CERTIFICATE OF ANALYSIS

Final Report

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	Cr2O3 %	V2O5 %	LOI %	S %	Sum %
1: Farim Comp +1180µm	6.70	0.74	31.5	0.93	23.6	0.10	0.03	0.05	15.1	0.78	0.04	0.04	16.5	2.17	96.0
2: Farim Comp -1180/+425µm	4.72	0.25	4.12	0.12	48.4	0.18	< 0.01	0.01	33.4	0.10	0.02	0.02	4.46	1.46	95.8
3: Farim Comp -425/+106µm	7.98	0.22	1.33	0.08	48.9	0.17	< 0.01	0.01	33.8	0.03	0.03	0.03	3.80	0.76	96.3
4: Farim Comp -106/+20µm	3.79	0.76	2.67	0.44	49.4	0.19	0.03	0.05	33.7	0.04	0.05	0.03	4.91	1.21	96.0
5: Farim Comp -20µm	9.98	4.16	2.24	0.64	42.6	0.17	0.13	0.17	29.4	0.02	0.14	0.07	8.11	0.82	97.8

Control Quality Assay
Not Suitable for Commercial Exchange

"*Tom Watt*" (signed by)

Tom Watt
Project Coordinator

Online LIMS

0000395015

Appendix D – Terminology for Liberation and Association

13478-003 Terminology and Definitions
Lycopodium Minerals Canada Ltd

Modal Mineralogy

Note: The size of the minerals as shown in the modal mineralogy tables is calculated statistically from the length of all the horizontal intercepts through each particle. It uses an assumption of random sectioning of spherical particles having uniform size, to obtain an estimate of the stereologically-corrected grain size in microns. The size calculation is a statistical property, which means that it is only valid when applied to a population of particles, and its accuracy increases as the population size increases. The accuracy of the size calculation is extremely low if applied to just a single cross-section.

Liberation and Association

For the purposes of this analysis, particle liberation is defined based on 2D particle area percent. Particles are classified in the following groups (in descending order) based on mineral-of-interest area percent: free (=100% of the total particle area) and liberated (≥97%). The non-liberated grains have been classified according to association characteristics, where binary association groups refer to particle area percent greater than or equal to 95% of the two minerals or mineral groups. The complex groups refer to particles with ternary, quaternary, and greater mineral associations including the mineral of interest.

The liberation and association characteristics of these minerals for each sample are given below. Note that when minerals are present in trace amounts, roughly <0.2 wt%, statistical data might not be adequate to calculate the liberation and association. Thus, results must be interpreted with caution.

Terminology developed for liberation and association are presented in below.

Liberation classes were defined as the following;

- **Pure:** A mineral with =100% Area Percent
- **Free:** A mineral with ≥95% Area Percent
- **Liberated:** A mineral with ≥90% but <95% Area Percent
- **Midds:** A mineral with ≥50% but <90% Area Percent
- **Sub-Midds:** A mineral with ≥20% but <50% Area Percent
- **Locked:** A mineral with <20% Area Percent

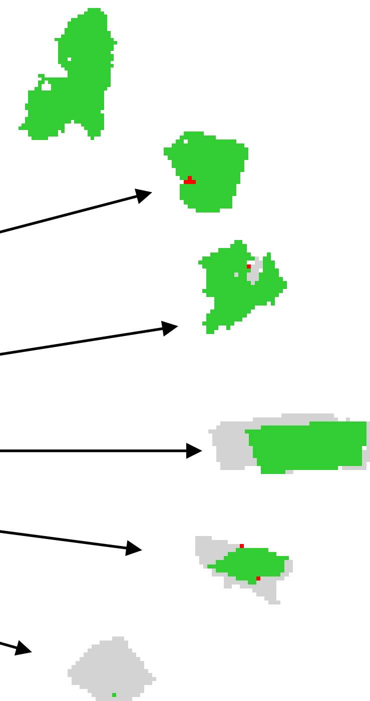


Figure 1. Legend for liberation classes.

Association classes were defined as the following;

- **Free Apatite -** A particle that has >=95% of Apatite
- **Liberated Apatite**: A particle that has ≥80 but <95 area% of Apatite
- **Apatite**: **Silicates -** A particle that has ≥95 area% of Apatite + Silicates
- **Apatite: Oxides -** A particle that has ≥95 area% of Apatite + Oxides
- **Apatite**: **Carbonates**- A particle that has ≥95 area% of Apatite + Carbonates
- **Apatite**: **Sulphides -** A particle that has ≥95 area% of Apatite + Sulphides
- **Apatite**: **Other -** A particle that has ≥95 area% of Apatite + Any Other Minerals
- **Apatite**: **Sil:Carb -** A particle that has ≥95 area% of Apatite + Any Other Minerals
- **Complex**: Any combination of the above definitions has been defined as a complex particle.

Note: similar associations were created for Silicates, Carbonates, Fe-Oxides and Sulphides

Mineral Release Curves

Mineral release curves are used to predict the amount of liberated mineral of interest at varying size distributions. This can be an indicator of optimum grind targets for metallurgical processes in order to achieve the most liberation for the least amount of grind energy. The variation between value and gangue mineral release curves may sometimes be used to enhance separation.

Note: The size used for the mineral release is the mid-point screen size, which is calculated by the following: Midpoint = square root (top size) x square root (bottom size). For the top size, (e.g., +200 µm) the top size particle (e.g., 340 µm) is identified, then 340 µm will be the top size and 200 the bottom size. Thus, the point for the mineral release at this liberation would be calculated as: square root (340) x square root (200) = 18.4390 x 14.1421 = 260.76. For any mid-size, the size fraction µm is used for this calculation. However, for the bottom size, 3 µm is used because that is approximately the beam diameter limitation for the QEMSCAN.

Grade – Recovery Curves

Another, more functional, method of presenting liberation is the mineralogically limiting grade-recovery curves, as are shown below. They are based on the calculated mass of minerals and the total mass in each liberation category. Thus, the highest grade (>80% sulphides) is contained in the >80% liberated sulphides particles. Then the next category (60-80% liberation) is added and the combined grade is calculated. This is repeated until all sulphides are accounted for. Mineralogically limited grade-recovery analyses provide an indication of the theoretical maximum achievable elemental or mineral grade by recovery, based on individual particle liberation and grade. These results, of course, do not reflect any other recovery factors that could occur in the actual metallurgical process.

Appendix E – Electron Microprobe Analysis (EMPA)

Apatite

	P	Si	S	Al	La	Ce	Mg	Ca	Mn	Fe	Na
Average	15.86	0.40	0.33	0.21	0.02	0.02	0.05	36.29	0.02	0.25	0.13
Max	17.27	5.40	0.71	3.22	0.07	0.08	0.46	40.57	0.17	1.12	0.35
Min	9.57	0.00	0.18	0.00	0.00	0.00	0.00	21.44	0.00	0.06	0.05
Std Dev	1.71	1.11	0.11	0.64	0.02	0.02	0.08	3.95	0.03	0.18	0.07

Apatite (Impure)

	P	Si	S	Al	La	Ce	Mg	Ca	Mn	Fe	Na
Average	11.25	4.20	0.29	2.41	0.01	0.00	0.30	25.14	0.00	0.77	0.10
Max	14.69	5.40	0.34	3.22	0.01	0.02	0.46	32.27	0.00	1.12	0.10
Min	9.57	1.68	0.25	0.92	0.01	0.00	0.15	21.44	0.00	0.42	0.09
Std Dev	2.33	1.73	0.04	1.02	0.00	0.01	0.15	4.85	0.00	0.32	0.01

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EMPA Data - Conducted At Laval University

Apatite		P	Si	S	Al	La	Ce	Mg	Ca	Mn	Fe	Na
	Conversion	2.2914	2.1394	1.9997	1.8895	1.172	1.1713	1.6583	1.3992	1.2912	1.2865	1.3481
	12A	16.98	0.29	0.28	0.14	0.02	0.00	0.05	39.24	0.03	0.29	0.08
	12A	17.24	0.10	0.28	0.04	0.03	0.01	0.02	39.65	0.01	0.35	0.09
	12A core	16.93	0.06	0.36	0.02	0.00	0.02	0.03	38.35	0.00	0.21	0.09
	12A rim	15.99	0.03	0.36	0.01	0.04	0.00	0.01	36.37	0.03	0.06	0.07
	12A	16.91	0.01	0.31	0.01	0.00	0.00	0.02	39.09	0.02	0.14	0.11
	12A	15.93	0.01	0.59	0.01	0.00	0.05	0.02	35.26	0.01	0.20	0.16
	12A	15.04	0.01	0.25	0.01	0.00	0.00	0.00	37.13	0.00	0.58	0.05
	12A	16.97	0.01	0.22	0.01	0.01	0.01	0.01	40.37	0.00	0.14	0.11
	12A	16.26	0.20	0.40	0.07	0.00	0.00	0.03	36.14	0.17	0.34	0.11
	12A	17.15	0.11	0.28	0.02	0.05	0.00	0.02	39.35	0.00	0.21	0.10
	12A	16.54	0.06	0.42	0.02	0.05	0.00	0.06	39.26	0.00	0.30	0.24
	12A rim	16.26	0.39	0.34	0.14	0.05	0.01	0.06	36.89	0.02	0.24	0.09
	12A core	16.50	0.22	0.26	0.09	0.01	0.00	0.05	38.91	0.00	0.17	0.09
	12A core	16.97	0.01	0.30	0.01	0.01	0.00	0.04	38.56	0.00	0.16	0.14
	12A rim	16.33	0.01	0.48	0.00	0.01	0.00	0.02	36.62	0.03	0.08	0.12
	13A	16.94	0.00	0.30	0.00	0.02	0.04	0.01	38.87	0.00	0.17	0.10
	13A	16.97	0.01	0.22	0.01	0.02	0.00	0.02	38.89	0.00	0.23	0.09
	13A	17.19	0.03	0.29	0.03	0.02	0.01	0.02	39.10	0.03	0.25	0.13
	13A	16.86	0.08	0.26	0.01	0.02	0.04	0.01	38.20	0.00	0.16	0.07
	13A	16.80	0.17	0.21	0.09	0.00	0.03	0.03	38.48	0.03	0.21	0.10
	13A	16.88	0.22	0.28	0.09	0.00	0.00	0.05	38.07	0.00	0.35	0.17
	13A	16.93	0.00	0.28	0.00	0.07	0.00	0.02	39.17	0.00	0.20	0.08
	13A	17.14	0.14	0.23	0.07	0.00	0.04	0.03	38.00	0.03	0.23	0.10
	13A	16.93	0.01	0.31	0.01	0.03	0.05	0.01	37.59	0.04	0.15	0.08
	13A	17.01	0.01	0.21	0.01	0.02	0.00	0.02	38.75	0.01	0.23	0.11
	13A	16.62	0.18	0.30	0.04	0.00	0.02	0.02	37.13	0.00	0.16	0.07
	13A	17.14	0.00	0.21	0.00	0.00	0.00	0.01	39.20	0.00	0.29	0.09
	13A	14.94	0.08	0.54	0.01	0.00	0.07	0.08	35.62	0.00	0.10	0.34
	13A	16.37	0.03	0.43	0.03	0.02	0.07	0.02	36.60	0.02	0.25	0.12
	13A	17.23	0.01	0.29	0.02	0.00	0.00	0.03	39.41	0.06	0.16	0.11
	14A	15.55	0.06	0.26	0.01	0.03	0.00	0.03	35.39	0.01	0.25	0.08
	14A	16.00	0.11	0.33	0.04	0.01	0.01	0.04	36.19	0.00	0.21	0.09
	14A	16.03	0.11	0.25	0.07	0.02	0.03	0.01	36.52	0.02	0.20	0.10
	14A	14.85	0.39	0.41	0.13	0.00	0.08	0.04	33.52	0.00	0.16	0.08
	14A	16.13	0.50	0.30	0.20	0.03	0.00	0.05	36.33	0.00	0.30	0.11
	14A	17.00	0.05	0.18	0.02	0.05	0.02	0.02	40.14	0.06	0.17	0.09
	14A	17.22	0.06	0.29	0.01	0.01	0.02	0.04	39.15	0.03	0.25	0.16
	14A	16.75	0.03	0.28	0.06	0.01	0.00	0.03	37.69	0.00	0.22	0.13
	14A	13.25	0.62	0.50	0.19	0.01	0.05	0.11	31.39	0.00	0.24	0.32
	14A	16.91	0.01	0.23	0.00	0.00	0.05	0.02	40.57	0.00	0.17	0.11
	14A	17.25	0.00	0.21	0.02	0.03	0.02	0.02	39.35	0.00	0.21	0.09
	14A	15.78	0.07	0.30	0.03	0.03	0.02	0.04	35.78	0.03	0.13	0.08
	14A	17.27	0.03	0.22	0.12	0.05	0.00	0.01	39.30	0.00	0.31	0.09
	14A	16.16	0.07	0.20	0.04	0.00	0.01	0.02	38.53	0.02	0.20	0.09
	14A	16.43	0.49	0.30	0.19	0.00	0.04	0.07	36.74	0.00	0.31	0.12
	15A	15.04	0.15	0.49	0.03	0.01	0.00	0.07	35.87	0.00	0.07	0.34
	15A	14.06	0.02	0.43	0.00	0.00	0.00	0.04	32.50	0.00	0.08	0.13
	15A	14.78	0.06	0.48	0.00	0.00	0.00	0.06	34.80	0.00	0.08	0.31
	15A	12.69	0.04	0.71	0.00	0.02	0.00	0.05	29.99	0.04	0.10	0.32
	15A	16.33	0.26	0.30	0.09	0.03	0.00	0.04	36.55	0.02	0.13	0.11
	15A	15.85	0.21	0.36	0.07	0.00	0.00	0.05	35.51	0.00	0.14	0.11
	15A	13.83	0.04	0.45	0.00	0.00	0.03	0.05	31.09	0.07	0.19	0.16
	15A	13.51	1.16	0.65	0.36	0.03	0.02	0.21	32.56	0.01	0.24	0.35
	15A	16.11	0.06	0.33	0.02	0.00	0.00	0.03	35.61	0.01	0.26	0.10
	15A	16.27	0.31	0.38	0.14	0.03	0.03	0.04	36.33	0.05	0.27	0.08
	15A	15.70	0.06	0.38	0.02	0.00	0.00	0.03	34.84	0.01	0.31	0.09
	Average	15.86	0.40	0.33	0.21	0.02	0.02	0.05	36.29	0.02	0.25	0.13
	Max	17.27	5.40	0.71	3.22	0.07	0.08	0.46	40.57	0.17	1.12	0.35
	Min	9.57	0.00	0.18	0.00	0.00	0.00	0.00	21.44	0.00	0.06	0.05
	Std Dev	1.71	1.11	0.11	0.64	0.02	0.02	0.08	3.95	0.03	0.18	0.07

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EMPA Data - Conducted At Laval University

Apatite (Impure)

15A	14.69	1.68	0.27	0.92	0.01	0.00	0.15	32.27	0.00	0.42	0.09
15A	10.59	4.42	0.29	3.22	0.01	0.00	0.18	23.80	0.00	0.60	0.09
15A	9.57	5.29	0.34	2.64	0.01	0.02	0.46	21.44	0.00	0.94	0.10
15A	10.14	5.40	0.25	2.84	0.01	0.00	0.39	23.07	0.00	1.12	0.10
Average	11.25	4.20	0.29	2.41	0.01	0.00	0.30	25.14	0.00	0.77	0.10
Max	14.69	5.40	0.34	3.22	0.01	0.02	0.46	32.27	0.00	1.12	0.10
Min	9.57	1.68	0.25	0.92	0.01	0.00	0.15	21.44	0.00	0.42	0.09
Std Dev	2.33	1.73	0.04	1.02	0.00	0.01	0.15	4.85	0.00	0.32	0.01

Carbonates

	Mg	Ca	Mn	Fe	Sr
	1.6583	1.3992	1.2912	1.2865	1.1826
12A	10.49	24.07	0.33	1.10	0.00
12A	10.23	23.89	0.38	1.00	0.00
12A	0.69	1.03	5.65	38.87	0.00
12A	0.65	0.89	4.36	40.03	0.00
12A	10.21	23.87	0.35	1.01	0.00
12A	10.31	24.23	0.30	1.11	0.00
12A	1.44	2.05	0.62	40.91	0.00
12A	1.44	1.70	0.60	40.56	0.00
12A	0.26	2.26	2.73	40.00	0.00
12A	0.62	0.61	0.69	43.27	0.00
12A	0.92	1.75	4.04	38.42	0.01
12A	0.74	1.32	2.79	41.12	0.00
Average	4.00	8.97	1.90	27.28	0.00
Max	10.49	24.23	5.65	43.27	0.01
Min	0.26	0.61	0.30	1.00	0.00
Std Dev	4.67	11.12	1.93	19.41	0.00

Certificate of Author

Dan Markovic

To accompany the Technical Report titled, *Farim Phosphate Project*, dated 14 September 2015, on the Farim Property (the "Report"), I, Dan Markovic, P.Eng. do hereby certify that:

1. I am currently employed as Project/Study Manager in the consulting firm Lycopodium Minerals Canada Ltd., located at 5060 Spectrum Way, Mississauga, Ontario, Canada, L4W 5N5.
2. I graduated with a Bachelor of Applied Science degree in Materials and Metallurgical Engineering from Queen's University, Kingston, Ontario, Canada, in 1993.
3. I am in good standing as a member of the Professional Engineers of Ontario (#90426818).
4. I have practiced my profession continuously since my graduation. My relevant experience includes consulting and managing mining projects in various stages of the execution cycle from scoping, pre-feasibility and feasibility studies through detailed design. I have been involved in phosphate, gold, nickel, iron ore, copper, and uranium mining projects in North America, South America, Eurasia, and West Africa.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am independent of the Issuer and related companies applying all of the tests in section 1.5 of National Instrument 43-101.
7. I have had no prior involvement with the property that is the subject of the Technical Report.
8. I am responsible for Sections 1, 2, 3, 4, 5, 6, 18 (except 18.18 and 18.19), 19, 21.5, 21.7, 23, 24, 27, with contributions to 25.3, 25.5, 26.3 and 26.5, and am co-author of Section 17, of the Technical Report.
9. I have visited the Farim project site on October 5th through 8th, 2014 and on July 11th, 2015.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains the necessary scientific and technical information to make the Technical Report not misleading.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated September 14th, 2015.

"Signed and Sealed"

"Dan Markovic" (signed by)

Dan Markovic, P. Eng.

PerUsa EnviroMet Inc., 228 Pinellas Street, Lakeland, FL 33803-4832, USA Phone/Fax: (863) 646-5096 E-mail: fsotillo@msn.com

Certificate of Author

Francisco J. Sotillo

To accompany the Technical Report titled, *Farim Phosphate Project*, dated 14 September 2015, on the Farim Property (the "Report"), I, Dr. Francisco J. Sotillo, CIP, MSc, PhD, and MMSA QP do heraby certify that:

1. I am a Principal Consultant (Metallurgy/Processing) of PerUsa EnviroMet, Inc., 228 Pinellas Street, Lakeland, FL 33803-4832, USA, subcontracted to KEMWorks, with a business address at 231 N. Kentucky Ave., Lakeland, FL, 33801-4977, USA.

2. I graduated with a degree in Metallurgical Engineering from the National University of Engineering, Peru in 1977. In addition, I have a Master of Science and Philosophy Doctor in Engineering degrees in Materials Science and Mineral Engineering from the University of California at Berkeley in 1985 and 1995, respectively. I am a member of the Society of Mining, Metallurgy & Exploration, SME, (Membership Number 3037370); Member of the Mining & Metallurgical Society of America, MMSA, (Membership Number 01473QP); and I am a Professional Engineer (PE) from the Board of Engineers of Peru, CIP, (CIP Number 23688).

3. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualify Person" for the purpose of NI 43-101.

4. I have personally carried out and inspected the subject project from 14 November, 2013 to 23 July, 2015.

5. I am co-author of this report and responsible for the preparation of database and compilations of the metallurgical information. I am responsible for Section 13, with contributions to 25.3, 26.3 and co-author for Section 17 of the Technical Report.

6. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43101.

7. I have had no prior involvement with the property that is the subject of the Technical Report.

8. I have read NI 43-101 and Form 43-101F1 and the Chapters of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

9. As of the date of this Certificate, to the best of my knowledge, information and belief, the Chapters of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th Day of September, 2015.

"Signed and Sealed"

"Dr. Francisco J. Sotillo" (signed by)

Francisco J Sotillo
01473QP
Mining & Metallurgical Society of America

Dr. Francisco J. Sotillo CIP, MSc, PhD, MMSA
Principal Consultant (Metallurgy/Processing)

Certificate of Author

Edward H. Minnes

To accompany the Technical Report titled, *Farim Phosphate Project*, dated 14 September 2015, on the Farim Property (the "Report"), I do hereby certify that:

- My name is Edward Hood Minnes, and my title is Mining Practice Leader with the firm of Golder Associates Inc., with a business address at 44 Union Boulevard, Suite 300, Lakewood, CO 80228, USA. My residential address is 1921 Denver West Court, Apt. 2011 Lakewood, CO 80401, USA.
- My formal education qualifications include B.S. (Bachelor of Science) in Mining Engineering from Queens University – Kingston (1984).
- I am a practicing Mining Engineer, Registered Member with the Society for Mining, Metallurgy and Exploration, Inc., and a Registered Professional Engineer in the State of Missouri, USA.
- I have practiced my profession continuously since 1985.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purpose of NI 43-101.
- My relevant experience with respect to the Farim Phosphate Deposit includes 30 years of open pit mining experience in mineral resource, mine planning, and including reserve estimation of phosphate deposits in the United States and Peru. My related experience also includes geological modelling and evaluation of mining projects.
- My prior involvement with the Property is limited to work performed in the preparation of previous Technical Reports prepared in 2012.
- I have supervised and reviewed the work carried out by Golder professionals for the mine planning for the Farim Phosphate Deposit. I am responsible for the preparation of Sections 15, 16 (except 16.5), 21.6, with contributions to 25.2, and 26.2 of the Technical Report. I have personally visited the property on April 5 through 8, 2015.
- As of the date of this Certificate, to my knowledge, information and belief, the section of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101, and the sections in this Technical Report have been prepared in compliance with National Instrument 43-101, Companion Policy 43-101CP and Form 43-101F1.

Signed and dated this 14th day of September 2015 at Lakewood, Colorado.

Original Document signed by:
Edward H. Minnes, PE (MO), SME (QP)
Mining Practice Leader

"Edward H. Minnes" (signed by)

____

Signature

Certificate of Author

Jerry Christopher DeWolfe

To accompany the Technical Report titled, *Farim Phosphate Project*, dated 14 September 2015, on the Farim Property (the "Report"), I do hereby certify that:

- My name is Jerry Christopher DeWolfe, and my title is Associate and Senior Geological Consultant with the firm of Golder Associates Ltd., with a business address at 102, 2535 3rd Avenue S.E., Calgary, Alberta, Canada T2A 7W5.
- My formal education qualifications include M.Sc. (Masters of Science) in Geology from Laurentian University – Sudbury (2006) and B.Sc. (Bachelor of Science) in Geology from Saint Mary's University – Halifax (2000).
- I am a practicing Resource Geologist, and a Registered Professional Geologist in the provinces of Alberta (APEGA), British Columbia (APEGBC) and Ontario (APGO), Canada.
- I have practiced my profession continuously since 2000.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purpose of NI 43-101.
- My relevant experience with respect to the Farim Phosphate Deposit includes 15 years of open pit mining experience in exploration, mine geology and resource estimation of phosphate, coal, oil shale and other stratigraphically controlled deposits, including phosphate deposits in West Africa and Peru.
- I have had no prior involvement with the Property.
- I have supervised and reviewed the work carried out by Golder professionals for the geological modelling and phosphate Mineral Resource estimation for the Farim Phosphate Deposit. I am responsible for the preparation of sections 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.0, with contributions to 25.2, and 26.2 of the Technical Report. I have personally visited the property on April 5 through 8, 2015.
- As of the date of this Certificate, to my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101, and the sections in this Technical Report have been prepared in compliance with National Instrument 43-101, Companion Policy 43-101CP and Form 43-101F1.

Signed and dated this 14th day of September 2015 at Calgary, Alberta.

Original Document signed and sealed by:
Jerry DeWolfe, P. Geo. (AB, BC, ON), M.Sc.
Associate, Senior Geological Consultant

"*Jerry DeWolfe*" (signed by)

____

Signature

Certificate of Author

George Lightwood

To accompany the Technical Report titled, *Farim Phosphate Project,* dated 14 September 2015, on the Farim Property (the "Report"), I do hereby certify that:

- My name is George Lightwood, and my title is Senior Engineer with the firm of Golder Associates Ltd., with a business address at 595 Double Eagle Court, Suite 1000, Reno, Nevada, USA, 89521.
- My formal education qualifications include M.Eng. Civil Engineering (Geotechnical), University of California, Berkeley (1988), M.S. Civil Engineering (Geotechnical), Stanford University (1987), and B.S. Mining Engineering, Colorado School of Mines (1979).
- I am a practicing Registered Professional Mining Engineer, Licensed Professional Mining Engineer, Nevada (12212), and a Registered Member, Society for Mining, Metallurgy, and Exploration
- I have practiced my profession since 1980.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purpose of NI 43-101.
- My relevant experience with respect to the Farim Phosphate Deposit includes 25 years of experience in the geotechnical analysis and design of open pit mine slopes, highway and railroad embankments and cuts, and water resource facilities. Further qualifying experience includes subsurface exploration in rock and soil, geophysical methods for characterization or rock, laboratory testing of rock and soils, and pit slope and waste dump design for open pit mines in sedimentary and residual soil deposits.
- I have had no prior involvement with the Property.
- I have supervised and reviewed the work carried out by Golder professionals for the pit slope design for the Farim Phosphate Deposit. I am responsible for the preparation of Section 16.5 of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101, and the sections in this Technical Report have been prepared in compliance with National Instrument 43-101, Companion Policy 43-101CP and Form 43-101F1.

Signed and dated this 14th day of September 2015 at Reno, Nevada

Original Document signed and sealed by:
George Lightwood, PE (NV)
Senior Engineer

"*George Lightwood*" (signed by)

____
Signature

Knight Piésold
CONSULTING

Certificate of Author
David John Toomey Morgan

To accompany the Technical Report titled, *Farim Phosphate Project*, dated 14 September 2015, on the Farim Property (the "Report"), I, David John Toomey Morgan, MAusImm (CP), do hereby certify that:

1) I am a civil engineer with Knight Piésold Pty Ltd. My office address is Level 1, 184 Adelaide Terrace, East Perth, Western Australia 6004.
2) I am a graduate of the University of Manchester, (BSc, Civil Engineering, 1980) and the University of Southampton (MSc, Irrigation Engineering, 1981).
3) I am a Member of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and registered as a Chartered Professional. I have worked as a civil engineer for a total of 34 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - review and report as a consultant on numerous tailings storage facilities and mining projects around the world for due diligence and regulatory requirements.
   - Project director on a number of feasibility studies and detailed designs in the gold industry in Africa, Australia and Asia.
   - Consulting engineer at a number of gold mines in Africa, Australia and Asia.
4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.
5) I have visited the Farim Phosphate Project site on 21 – 22 February 2015.
6) I am responsible for all preparation of Section 18.19, with contributions to Sections 25.5, 25.6, 26.5, and 26.6 of the Technical Report.
7) I am independent of the Issuer applying the test set out in Section 1.5. (4) of NI 43-101.
8) I have not been involved in any previous Technical Report on the Farim Phosphate Project.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 14th day of September 2015
"David John Toomey Morgan" (signed by)

The Institution of Engineers, Australia
David J T Morgan
MIEAust CPEng
Chartered Professional Engineer
Membership No. 974219

David John Toomey Morgan, MAusImm (CP)

Certificate of Author

Richard A. Cook

To accompany the Technical Report titled, *Farim Phosphate Project*, dated 14 September 2015, on the Farim Property (the "Report"), I, Richard A. Cook, P.Geo. (Ltd.), do hereby certify that:

1. I am currently employed as a Senior Environmental Scientist with Knight Piésold Ltd., with an office at 1650 Main Street West, North Bay, ON P1B 8G5;

2. I am a professional geoscientist (limited designation; Reg. No. 2199) with the Association of Professional Geoscientists of Ontario (APGO).

   I am a graduate of Queen's University with a B.Sc. (Hon.) in Environmental Science (Chemistry). I have practiced environmental consulting to the mining industry for 19 years, have been involved at a senior level in the preparation of over a dozen environmental and social impact assessments (ESIAs) within Canada and abroad, and have been involved in various environmental assignments on more than two dozen mining projects or sites.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43- 101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

3. I visited the Farim Phosphate Project site on March 25-26, 2015;

4. I am responsible for Section 20, 25.7, and 26.7 of the Technical Report;

5. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have had no prior involvement with the property that is the subject of the Technical Report;

7. I have read NI 43-101, and Section 20 of the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

8. As of the date of this Certificate, to the best of my knowledge, information and belief, Section 20 of this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

"*Richard Cook*" (signed by)

Signing and Dated: September 14th, 2015

Original signed and sealed Richard Cook, P.Geo. (Ltd.)

Certificate of Author

Edward Liegel

To accompany the Technical Report titled, *Farim Phosphate Project*, dated 14 September 2015, on the Farim Property (the "Report"), I, Ed Liegel, P.E., do hereby certify that:

1. I am currently employed as an Associate in the consulting firm W.F. Baird & Associates, located at 2981 Yarmouth Greenway, Madison, Wisconsin, United States of America, 53711.

2. I graduated with a Bachelor of Science degree in Civil Engineering from The University of Wisconsin-Madison in 2006.

3. I am in good standing as a registered Professional Engineer in the States of Wisconsin (42524-6), and Louisiana (39383).

4. I have practiced Marine Engineering continuously since my graduation. My relevant experience includes participation in the development of marine terminals at various stages including scoping, feasibility, and detailed design. I have been involved in the development of phosphate, iron ore, illmenite, petroleum, and coal terminals in North America, Africa and Australia.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am independent of the Issuer and related companies applying all of the tests in section 1.5 of National Instrument 43-101.

7. I am responsible for Sections 18.18, 25.4, & 26.4 of the Technical Report.

8. I have not visited the Farim project site. However, a Baird Representative did visit the site in 2012.

9. My prior involvement with the Property is limited to work performed in the preparation of previous Technical Reports prepared in 2012.

10. As of the date of the certificate, to the best of my knowledge, information and belief, Sections 18.18, 25.4, & 26.4, of the Technical Report contain the necessary scientific and technical information to make the Technical Report not misleading.

11. I have read National Instrument 43-101 and Form 43-101F1, and Sections 18.18, 25.4, & 26.4 the Technical Report have been prepared in compliance with that instrument and form.

Dated September 14th, 2015.
"Signed and Sealed"
"*Ed Liegel*" (signed by)

Ed Liegel, P.E.

Certificate of Author
Alexander O. Duggan

To accompany the Technical Report titled, Farim Phosphate Project, dated 14 September 2015, on the Farim Property (the "Report"), I, Alexander O. Duggan, M. Sc., P.Eng., do hereby certify that:

- I am employed as a Contractor in the capacity as Estimating and Planning Consultant with Lycopodium Minerals Canada Ltd., located at 5060 Spectrum Way, Mississauga, Ontario, Canada, L4W 5N5.
- I graduated with a Bachelor of Science, Honours degree in Civil Engineering from the University of Aston in Birmingham, England in 1982 and Masters degree Planning and Transportation from the University of Salford in Manchester, England in 1984. I have worked as an estimator for a total of 30 years since obtaining my M.sc. degree. I am a P. Eng., registered in the Province of Ontario (PEO No. 100103898).
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- My relevant experience for the purpose of the Technical Report is:
    - Manager, Estimating and Planning, Amec Saskatoon –KPSC Potash Project. 2014 – 2015
    - Manager, Estimating and Planning, Jacobs Minerals Canada Inc. 2011 – 2013
    - Manager, Estimating and Planning, Aker Metals, a division of Aker Solutions Inc. 2005 -2007, 2009– 2011
    - Senior Estimator, AMEC Americas 2007 – 2008
    - Senior Estimator, SNC Lavalin 2004 - 2005
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for authoring Section 22 of this Technical Report, all related to the economic analysis of the project.
- I am independent of the Issuer applying all of the tests in section 1.5 of National Instrument 43-101.
- I have not had prior involvement with the project that is the subject of this Technical Report.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
- As of the date of this Certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated September 14th, 2015.

Signed and Sealed"

"Alexander O. Duggan" (signed by)

LICENSED PROFESSIONAL ENGINEER
A. O. DUGGAN
100103898
PROVINCE OF ONTARIO

Alexander O. Duggan, B.Sc(Hons), M.Sc, P. Eng.

Certificate of Author
Alexander O. Duggan

To accompany the Technical Report titled, Farim Phosphate Project, dated 14 September 2015, on the Farim Property (the "Report"), I, Alexander O. Duggan, M. Sc., P.Eng., do hereby certify that:

- I am employed as a Contractor in the capacity as Director of Estimating and Planning with Kristal Font Inc., located at 8045 Wyandotte Street, East Windsor, Ontario, Canada, N8S 1T2.
- I graduated with a Bachelor of Science, Honours degree in Civil Engineering from the University of Aston in Birmingham, England in 1982 and Masters degree Planning and Transportation from the University of Salford in Manchester, England in 1984. I have worked as an estimator for a total of 30 years since obtaining my M.sc. degree. I am a P. Eng., registered in the Province of Ontario (PEO No. 100103898).
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- My relevant experience for the purpose of the Technical Report is:
  - Manager, Estimating and Planning, Amec Saskatoon –KPSC Potash Project. 2014 – 2015
  - Manager, Estimating and Planning, Jacobs Minerals Canada Inc. 2011 – 2013
  - Manager, Estimating and Planning, Aker Metals, a division of Aker Solutions Inc. 2005 -2007, 2009– 2011
  - Senior Estimator, AMEC Americas 2007 – 2008
  - Senior Estimator, SNC Lavalin 2004 - 2005
- I have not visited the Property that is the subject of this Technical Report.
- I am responsible for Sections 21.1 to 21.4, 25.1 and 26.1 of this Technical Report, all related to the capital cost estimate of the process plant and port facilities.
- I am independent of the Issuer applying all of the tests in section 1.5 of National Instrument 43-101.
- I have not had prior involvement with the project that is the subject of this Technical Report.
- I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated September 14th, 2015.

"Signed and Sealed"
"*Alexander O. Duggan*" (signed by)

LICENSED PROFESSIONAL ENGINEER
A. O. DUGGAN
100103898

Alexander O. Duggan, B.Sc(Hons), M.Sc, P. Eng.