Farim Phosphate Project
NI 43-101 Technical Report and Feasibility Study

Guinea-Bissau, West Africa

Effective Date: May 17, 2023

Prepared for:

Itafos Inc.
109 North Post Oak Lane, Suite 405
Houston, Texas, USA, 77024

Prepared by:

Ausenco Engineering Canada Inc.
11 King St. West, 15th Floor
Toronto, Ontario, Canada, M5H 4C7

List of Qualified Persons:

Tommaso Roberto Raponi, P.Eng., Ausenco Engineering Canada Inc.
Jerry DeWolfe, P.Geo., WSP Golder
Terry L. Kremmel, P.E., WSP Golder
Alexander Duggan, P.Eng., Kristal Font Inc.
Edward Adam Liegel, P.E., WF Baird & Associates Ltd.
Richard Michael Elmer, MIMMM, MCSM, C.Eng., Knight Piésold Ltd.
Richard Alonzo Cook, P.Geo., Knight Piésold Ltd.
Francisco J. Sotillo, P.E., PhD, KEMWorks Technology

CERTIFICATE OF QUALIFIED PERSON
Tommaso Roberto Raponi, P.Eng.

I, Tommaso Roberto Raponi, P.Eng., certify that:

1. I am employed as a Principal Metallurgist with Ausenco Engineering Canada Inc., (Ausenco), with an office address at Suite 1550 - 11 King St West, Toronto, ON M5H 4C7.
2. This certificate applies to the technical report titled "Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study" with an effective date of May 17, 2023 (the "Technical Report").
3. I graduated from the University of Toronto with a Bachelor of Applied Science degree in Geological Engineering with specialization in Mineral Processing in 1984.
4. I am a Professional Engineer registered with the Professional Engineers of Ontario (license No. 90225970), the Engineers and Geoscientists of British Columbia (license No. 23536), the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (license No. L4508), and with the Professional Engineers and Geoscientists of Newfoundland and Labrador (license No.10968).
5. I have practiced my profession continuously for over 38 years with experience in the development, design, operation and commissioning of mineral processing plants and associated infrastructure, focusing on gold projects, both domestic and internationally, including design and commissioning of a phosphate operation. My project design and development experience include the generation of capital and operating costs for mineral processing plants and associated infrastructure.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the Farim Phosphate Project property.
8. I am responsible for Sections 1.1 to 1.4, 1.12, 1.13, 1.15, 1.16, 1.17, 1.19, 1.20, 1.21.1, 1.21.4, 1.21.5, 2.1, 2.2, 2.4, 2.5, 3.1, 3.2, 4, 5, 17, 18.1 to 18.4, 18.6, 18.10, 18.11, 19, 21.1.2, 21.1.3.2, 21.1.3.3, 21.2.1, 21.2.3, 21.2.6, 21.3, 23, 24, 25.4, 25.6, 26.1, 26.4, and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with Farim Phosphate Project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: June 23, 2023

"Signed and Sealed"

Tommaso Roberto Raponi, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Jerry DeWolfe, P.Geo.

I, Jerry DeWolfe, P.Geo., certify that:

1. I am employed as Director of Mining Engineering and Stability and a Senior Principal Resource Geologist with WSP Canada Inc. (formerly WSP Golder) with an office address of 237, 4th Ave. SW, Suite 3300, Calgary, Alberta, Canada, T2P 4K3.
2. This certificate applies to the technical report titled "Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study" with an effective date of May 17, 2023 (the "Technical Report").
3. I graduated from Laurentian University, Sudbury, Ontario (M.Sc. Geology, 2006) and Saint Mary's University, Halifax, Nova Scotia (B.Sc. with honors in Geology, 2000).
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta (APEGA; Registration No. 101287), Engineers and Geoscientists British Columbia (EGBC; Registration No. 38237), and the Professional Geoscientists Ontario (PGO; Registration No. 1240).
5. I have practiced my profession for 22 years. My relevant experience for the purpose of the Technical Report includes 14 years of direct mineral exploration program design and oversight, geological data interpretation and management, geological modelling and mineral resource estimation and mine geology of phosphate, potash, evaporites, coal and other stratigraphically controlled deposits.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Farim Phosphate Project site between April 5th and April 8th, 2015, for the purpose of completing a Qualified Person personal inspection site visit.
8. I am responsible for Sections 1.5, 1.6, 1.7, 1.9, 1.20, 1.21.2, 2.3, 6 to 12, 14, 25.3.1.1, and 26.2 of the Technical Report.
9. I am independent of Itafos, Inc. as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the Farim Phosphate Project since 2015 as an independent Qualified Person for Mineral Resources.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: June 23, 2023

"Signed and Sealed"

Jerry DeWolfe, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Terry L. Kremmel, P.E.

I, Terry L. Kremmel, P.E., certify that:

1. I am employed as a Technical Director, Mining Engineering with WSP USA, Inc. with an office address of 701 Emerson Road, Suite 250, Creve Coeur, Missouri, USA 63141.
2. This certificate applies to the technical report titled “Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study” with an effective date of May 17, 2023 (the “Technical Report”).
3. I graduated from the University of Missouri – Rolla in 1975 with a Bachelor of Science – Mining Engineering.
4. I am a Registered Member of the Society of Mining, Metallurgy & Exploration Inc. (SME) Registration Member Number 1791760, Registered Professional Engineer with the Missouri Board for Architects Professional Engineers and Land Surveyors Registration Number 022340, and Registered Professional Engineer with the North Carolina Board of Examiners for Engineers and Surveyors Registered Number 030597.
5. I have practiced my profession for 44 years. My relevant experience for the purpose of the Technical Report includes 38 years of direct mining engineering surface mine pit optimizations and designs, mine planning and oversight, mine data interpretation and management, mine cost modelling and mineral reserve estimation of phosphate, potash, coal, lithium and other stratigraphically controlled deposits.
6. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Farim Phosphate Project site in 2012 for one week.
8. I am responsible for Sections 1.10 to 1.11, 1.20, 1.21.2, 2.3, 15, 16.1, 16.2, 16.5 to 16.9, 21.1.1, 21.2.2, 25.3.1.2, 25.3.2, and 26.2 of the Technical Report.
9. I am independent of Itafos, Inc. as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the Farim Phosphate Project since 2012 in a mining engineering capacity.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: June 23, 2023

“Signed and Sealed”

Terry L. Kremmel, P.E.

CERTIFICATE OF QUALIFIED PERSON
Alexander Duggan, P.Eng.

I, Alexander Duggan, P.Eng., certify that:

1. I am employed as a Director of Project Services with Kristal Font Inc., with an office address of 8045 Wyandotte Street, East, Windsor, Ontario.
2. This certificate applies to the technical report titled "Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study" with an effective date of May 17, 2023 (the "Technical Report").
3. I graduated from University of Aston in Birmingham, England in 1982 with a B.Sc. in Civil Engineering and from University of Salford, England in 1984 with a M.Sc. in Engineering Planning.
4. I am a Professional Engineer (P. Eng) of Professional Engineers of Ontario.
5. I have practiced my profession for 38 years. I have been directly involved in preparing economic analysis of this Technical Report. My relevant years for the purpose of this technical report includes 18 years of experience producing economic analyses for numerous gold, silver, zinc, copper, copper-moly, iron ore, lithium, rare earth, and phosphorus mining projects. Additionally, I have been responsible for producing capital, sustaining and operating cost analyses for the listed mining projects.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the Farim Phosphate Project site.
8. I am responsible for Sections 1.18, 1.20, 3.2, 3.3, 22, and 25.10 of the Technical Report.
9. I am independent of Itafos Inc. as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the Farim Phosphate Guinea Bissau filed on August 2015.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: June 23, 2023

"Signed and Sealed"

Alexander Duggan, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Edward Adam Liegel, P.E.

I, Edward Adam Liegel, P.E., certify that:

1. I am employed as a Principal with W.F. Baird & Associates Ltd. (Baird), with an office address of 2924 Marketplace Drive, Suite 200, Madison, WI 53719.
2. This certificate applies to the technical report titled "Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study" with an effective date of May 17, 2023 (the "Technical Report").
3. I graduated from the University of Wisconsin-Madison in 2006 with a Bachelor of Science in Civil Engineering (Structural Emphasis).
4. I am a professional engineer licensed by the state of Louisiana (PE.0039383).
5. I have practiced my profession for 17 years. I have been directly involved in the development of prefeasibility and bankable feasibility studies for marine export terminals, design of marine structures, assessment of shipping methodologies, navigation channel design, geotechnical analysis, capital and operational cost estimating, contract preparation, and project delivery planning.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the Farim Phosphate Project site.
8. I am responsible for Sections 1.20, 1.21.618.13, 21.1.4, 21.2.4, 25.7, and 26.5 of the Technical Report.
9. I am independent of Itafos Inc. as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the following phases of work for the Farim Phosphate Project:
    - 2012 – Initial pre-feasibility level study of port infrastructure at Ponte Chugue in the Geba River. Study included conceptual design of the marine terminal, CAPEX and OPEX generally having battery limits from the haul road / Port interface through direct shiploading of vessels up to handymax size. The study was inclusive of limited marine field data gathering related to the Geba River. (Baird, 2012a)
    - 2012 – A conceptual level investigation including CAPEX development of barging material down the Cacheu River utilizing river push tugs and barge flotillas followed by transhipping of material to ocean-going vessels (OGVs). The study's battery limits were from the land/water abutment at Binta through transhipping to OGVs in the mouth of the Cacheu River. The study was not inclusive of any land-based port infrastructure or any marine-based mechanical, electrical, piping, plumbing, material handling, or utility engineering (undertaken by others). The study was inclusive of limited marine field data gathering related to the Cacheu River. (Baird, 2012b)
    - 2015 – An abbreviated investigation into transhipping on the Geba River. (Baird, 2015)
    - 2016 – A conceptual level update of the marine infrastructure originally developed for the Geba River in 2012, with the intent of reducing marine infrastructure and associated CAPEX.
    - 2017 – Management of field data acquisition suitable for design including: bathymetric/hydrographic, topographic, geotechnical, geophysical, hydrodynamic, and tidal data, as well as aerial imagery associated with the planned marine infrastructure within Baird's battery limits on the Geba River.
    - 2018 – Marine Studies to define the marine environment and baseline anticipated operations, including hydrodynamic and sedimentation modelling, moored ship response modelling and navigation simulations. (Baird, 2019a) (Baird, 2019b).

- 2019 – Engineering and Procurement Support Services – Advancement of portions of the marine terminal to a state suitable for tender, the development of tender documents for the supply of long lead fabricated items, the shiploader, and construction services. (Baird, 2019c)
- 2019 – Cacheu River Flood Study. A study to estimate extreme water levels on the Cacheu River near the mine site, in support of pre-feasibility level work to protect the mine from flood events (by others).

11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: June 23, 2023

"Signed and Sealed"

Edward Adam Liegel, P.E.

CERTIFICATE OF QUALIFIED PERSON
Richard Michael Elmer, C.Eng., MIMMM, MCSM

I, Richard Michael Elmer C.Eng., MIMMM, MCSM, certify that:

1. I am employed as a Principal Geotechnical Engineer and Director with Knight Piésold Limited ("Knight Piésold" or "KP"), with an office address of St Magnus House, 3 Lower Thames Street, London, EC3R 6HD.
2. This certificate applies to the technical report titled "Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study" with an effective date of May 17, 2023 (the "Technical Report").
3. I graduated from the University of Southampton with a B.Sc. in Geology in 1987 and from Camborne School of Mines with an M.Sc. in Mining Geology in 1988.
4. I am a Member of the Institute of Materials, Minerals & Mining, membership no. 0049205.
5. I have practiced my profession for 34 years. I am a geotechnical engineer that has been directly involved in feasibility studies and detailed designs and construction supervision for mine waste facilities (tailings facilities and waste rock facilities) around the world including West Africa, for multiple commodities, as well as completing audits on operational facilities.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Farim Phosphate Project site multiple times in 2010 - 2011 for 1- to 2-week visits to undertake site walkovers and drilling supervision. I have not visited the Farim Phosphate Project while employed by Knight Piésold.
8. I am responsible for Sections 1.20, 16.3, 16.4, 18.5, 18.7 to 18.9, 18.12, 18.13, 21.1.3.1, 21.2.5, 25.1, 25.5, 25.8, and 26.6 of the Technical Report.
9. I am independent of Itafos Inc. as independence is defined in Section 1.5 of NI 43-101.
10. I have been involved with the Farim Phosphate Project in 2010 – 2011 while employed with Golder Associates and was QP for the tailings aspects of the previous feasibility study published by Golder Associates in 2011.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: June 23, 2023

"Signed and Sealed"

Richard Michael Elmer C.Eng., MIMMM, MCSM

CERTIFICATE OF QUALIFIED PERSON
Richard Alonzo Cook, P.Geo.

I, Richard Alonzo Cook, P.Geo. (Limited), certify that:

1. I am employed as a Specialist Environmental Scientist | Associate with Knight Piésold Ltd. ("Knight Piésold"), with an office address of #200 – 1164 Devonshire Ave, North Bay, Ontario, Canada.
2. This certificate applies to the technical report titled "Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study" with an effective date of May 17, 2023 (the "Technical Report").
3. I graduated from Queen's University in Kingston, Ontario, Canada, in 1996 with an Honours Bachelor of Science in Environmental Science (Chemistry).
4. I am a Limited Member of Professional Geoscientists Ontario (Membership # 2199).
5. I have practiced my profession for 27 years. I have been directly involved in preparing environmental and social impact assessments (ESIAs) and have been responsible for the environmental studies, permitting and social or community impact section of 43-101 technical reports for mining projects in Canada and in several projects in Africa.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
7. I visited the Farim mine site and Ponta Chugue port site between March 25 and 26, 2015, for a visit duration of two days.
8. I am responsible for Sections 1.14, 1.20, 1.21.7, 2.3 20, 25.9, and 26.7 of the Technical Report.
9. I am independent of Itafos Inc. as independence is defined in Section 1.5 of NI 43-101.
10. I previously led the preparation of the 2015 Environmental and Social Impact Assessment (ESIA) for the Project. I also prepared the Environmental Studies, Permitting, and Social/Community Impact aspects of a feasibility study for the Project led by Lycopodium Minerals in 2015 (Lycopodium Minerals Canada Ltd., 2015. "Technical Report on the Farim Phosphate Project, Guinea-Bissau", Effective Date: 9 September 2015).
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: June 23, 2023

"Signed and Sealed"

Richard Alonzo Cook, P.Geo. (Limited)

CERTIFICATE OF QUALIFIED PERSON
Dr. Francisco J. Sotillo, P.E., PhD

I, Dr. Francisco J. Sotillo, P.E., PhD, state that:

(a) I am Senior Metallurgist at KEMWorks Technology, Inc., 1017 South Blvd., Lakeland, Florida 33803, USA

And President at PerUsa EnviroMet, Inc., 228 Pinellas Street, Lakeland, Florida 33803-4832, USA

(b) This certificate applies to the technical report "Farim Phosphate Project NI 43-101 Technical Report and Feasibility Study" with an effective date of May 17, 2023 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows. I am a graduate of the National University of Engineering, Lima, Peru, with a BS in Metallurgy (1977); the University of California at Berkeley with a MS of Engineering in Materials Science and Mineral Engineering (1985), and a PhD. of Engineering in Materials Science and Mineral Engineering (1995); and member of the Professional Institute of Engineers of Peru, CIP No.: 23688 (1983); Member of the Society of Mining, Metallurgy and Exploration, Inc., Member No.: 03037370; Member of the Mining and Metallurgical Society of America as a QP Member in Metallurgy/Processing, Member No.: 01473QP (2014). My relevant experience after graduation and over 45 years for the purpose of the Technical Report includes starting as a foreman at a mining company, and being now a recognized expert in the beneficiation of phosphate ores. I have worked as a Metallurgical Consultant for KEMWorks Technology, Inc., Pegasus TSI, Inc., Mosaic Company, Agrium Company, JR Simplot Phosphate Ltd., PCS Phosphates, CF Industries, Cargill Fertilizer, among many other clients. I am author of more than 40 publications on phosphate, sulfides, coal, and metallic ores. Since 2008, I am President of PerUsa EnviroMet, Inc.

(d) I have not conducted a personal inspection of the property described in the Technical Report.

(e) I am responsible for Sections 1.8, 1.21.3, 13, 25.2, and 26.3 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of the Instrument.

(g) I have prior involvement with the property that is the subject of the Technical Report preparing the metallurgical aspects for the NI 43-101 Technical Report of September 2015.

(f) I have read National Instrument 43-101. The parts of the Technical Report for which I am responsible have been prepared in compliance with this Instrument; and

(g) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Lakeland, Florida, USA this 23 of June of 2023.

"Signed and Sealed"

Dr. Francisco J. Sotillo, P.E., PhD

KEMWorks Technology, Inc.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Itafos Inc. (Itafos) by Ausenco Engineering Canada Inc. (Ausenco), WSP Golder (WSP), Kristal Font Inc. (Kristal), WF Baird & Associates Ltd. (Baird), Knight Piésold Ltd. (KP), and KEMWorks Technology (KEMWorks), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Itafos subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

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1 SUMMARY

1.1 Overview

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for Itafos Inc. (Itafos) to summarize the results of the NI 43-101 Technical Report and Feasibility Study on the Farim Phosphate Project and consolidate all project de-risk work between 2015 and 2022. The report was prepared in accordance with the Canadian disclosure requirements of National Instruments 43-101 (NI 43-101) and with the requirements of Form 43-101 F1.

The NI 43-101 responsibilities of the engineering consultants are as follows:

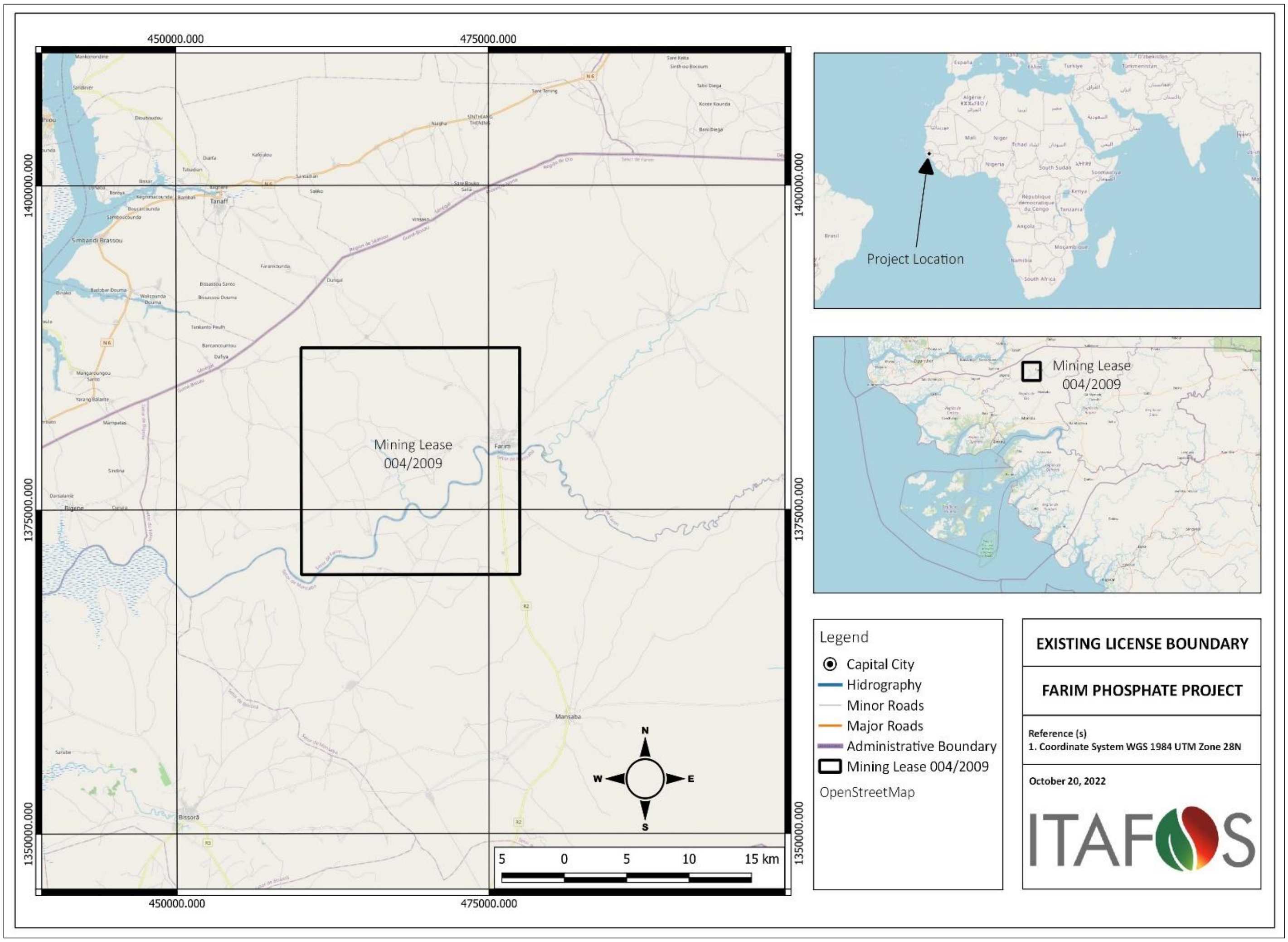
- Ausenco was commissioned by Itafos to manage and coordinate the work related to the NI 43-101, which included the following:
  - consolidate and review the process plant design/engineering deliverables and update core feasibility study documents required to support an AACE Class 3 estimate in 2022 US dollars
  - update equipment and contractor pricing by seeking supply pricing for major equipment and supporting minor equipment pricing with recent historical pricing from similar ongoing studies
  - develop feasibility-level capital and operating cost estimates for the process plant and mineral terminal
  - consolidate capital and operating cost estimates from the various qualified persons (QPs)
  - compile the technical report.
- KEMWorks Technology Inc. (KEMWorks) was commissioned to update the mineral process and metallurgical testing.
- WSP/Golder was commissioned to support the analysis of the previous drilling, exploration, design the open pit mine plan, mine production schedule, and mine capital and operating costs.
- Knight Piésold Consulting (KP) was commissioned to support the feasibility-level tailings management facility design, the geotechnical and hydrology design, and the environmental permitting and social/community impact including the bulk material estimate and operating costs.
- WF Baird (Baird) was commissioned to support design of the marine and mineral terminal loadout feasibility-level design and the bulk material estimate and operating costs.
- Kristal Font Inc. (Kristal) was commissioned to compile the capital cost estimate, operating cost estimate and the financial model for all disciplines.

All measurement units used in this report are metric unless otherwise noted. Currency is expressed in United States dollars (currency: USD; symbol: US$). The report uses American English.

1.2 Property Description and Location

The project is in the northern part of central Guinea-Bissau, West Africa, approximately 25 kilometers (km) south of the Senegal border; approximately 5 km west of the town of Farim; and approximately 120 km northeast of Bissau, the capital of Guinea-Bissau (see Figure 1-1).

Figure 1-1: Location of the Farim Phosphate Project



Source: Itafos, 2022

The Farim Phosphate Project lies within Mining Lease License No. 004/2009 ("Mining Lease 004/2009"), covering 30,625 hectares (ha), granted by the Government of Guinea-Bissau on May 28, 2009 to GB Minerals AG (GBMAG). GBMAG is registered in Switzerland and is a wholly owned subsidiary of Itafos Farim Holdings, which is registered in the Cayman Islands. Itafos Farim Holdings is 100% owned by Itafos Guinea-Bissau Holdings, also registered in the Cayman Islands. Itafos Guinea-Bissau Holdings is 100% owned by Itafos Inc., a corporation headquartered in Delaware.

A Mining Agreement was negotiated and signed between the Ministry of Energy and Natural Resources and GBMAG on May 1, 2009. The Mining Agreement allowed for the subsequent issuance of the following:

- 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.
- Since the initial mining license term of 25 years is from 2009, Itafos is in the process of filing a request with the Minister of Natural Resources of Guinea-Bissau for a 25-year mining license term extension which effectively provides a 25-year term from the issue of the request. A mining license and a mining lease may be renewed repeatedly by the holder according to the 2000 Mining Law.

GBMAG is in good standing on both the mining lease and mining license.

1.3 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The project site 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 River Cacheu. The River Cacheu at the ferry crossing is approximately 300 m wide. The Ministry of Public Works, which is part of a joint task force with the African Development Bank, has plans to build a bridge over the River Cacheu which would be utilized by the Project in the future. From the town of Farim, the property can be accessed via a 5 km unpaved road.

Ponta Chugue, which is the mineral terminal (MT) location in the Geba River estuary, is approximately 18 km east of Bissau and approximately 75 km south of Farim (Figure 5-1). The MT is accessible via public road.

1.4 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 project was explored by the French Bureau of Geological and Mining Research (BRGM) from 1981 to 1983.

In 1997, a Canadian exploration company, Champion Resources Inc. acquired ownership and carried out drilling from 1998 to 1999.

In 2006, GB Phosphate Mining Ltd. was granted rights over the phosphate deposit by the Government of Guinea-Bissau (GoGB). They performed drilling in 2007.

In 2009, GB Minerals AG executed a Mining Agreement with the Government of Guinea Bissau for the development of the Farim Project. This agreement included Mining Lease 004/2009 and Production License 001/2009.

The period 2010 to 2012 was characterized by ongoing field work in Guinea Bissau and in 2011 GB Minerals Ltd. acquired a 50.1% interest in GB Minerals AG and was appointed as operator of the Farim Project.

In 2013, GB Minerals Ltd. acquired full ownership of the Farim project by purchasing the remaining 49.9% interest in GB Mineral AG. In addition, a NI 43-101 technical report for the feasibility study on the Farim Project was filed and submitted to the GoGB for review.

In 2014, an Environmental and Social Impact Assessment (ESIA) on the mine component was prepared by Golder Associates (U.K.) Ltd. (Golder). During this time, GB Minerals Ltd. decided to change the project configuration and embarked on a new feasibility study for the project. The revised feasibility study, followed by a revised ESIA which corresponds to the revised project configuration, was submitted to the GoGB in 2015. Field work continued at the mine and mineral terminal sites during 2016. Bathymetry and geotechnical studies were conducted for the proposed MT in 2017. At the same time, a Resettlement Action Plan was submitted to the GoGB and various feasibility and ESIA workshops were held with various Government ministries.

In 2018, Itafos Inc. acquired 100% of GB Minerals Ltd and commenced with detailed engineering and the construction of a Mining Camp. A Resettlement Action Plan (RAP) and ESIA for the proposed Buredanfa resettlement village were prepared and approved in 2018.

During 2019, the project development was idled while Itafos awaited the conclusion of the presidential elections, and subsequently the coronavirus pandemic. Since 2020, site activities included environmental monitoring, technical trade-off studies, and Mining Agreement negotiations with the GoGB which are still in progress.

1.5 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 the world's current rock phosphate production.

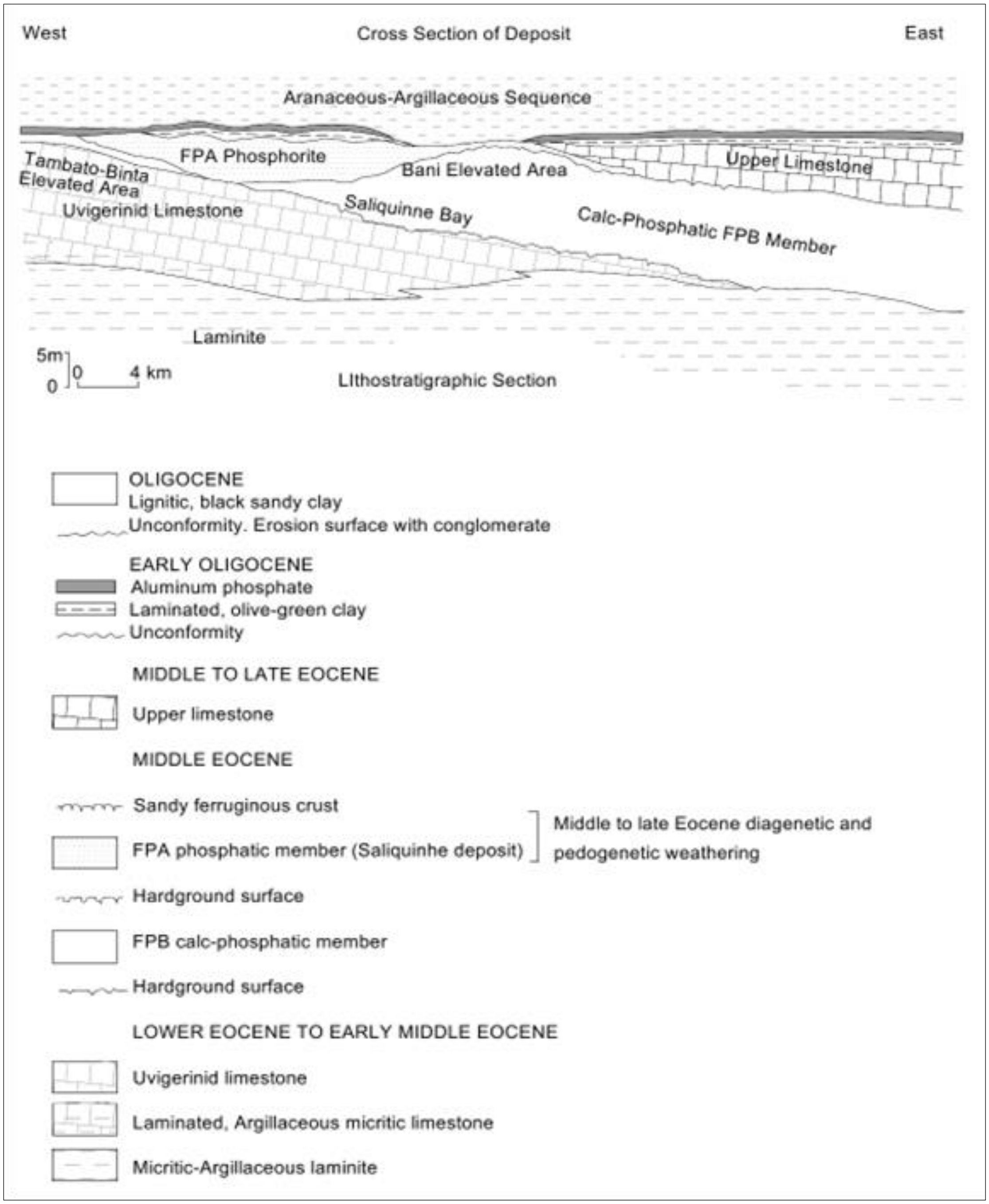
The Farim area forms part of the southern margin of the former Casamance Gulf and is located 60 km northwest of the southern edge of the Senegal-Mauritania-Guinea sedimentary basin in which the Maastrichtian strata unconformably overlies the Devonian pelite sequence (Prian, 1987).

The Farim phosphate deposit is a flat-lying sedimentary phosphatic bed, which underlies an area greater than 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)
- 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



Source: Reproduced from Prian, 1989.

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

- 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
- Lower grade FPB layer – Highly carbonated phosphate, generally containing 5 to 15% P_2O_5 (mean 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.

1.6 Exploration and Drilling

Historical exploration activity in the Farim project area has focused entirely on drilling campaigns. There are no documented non-drilling-related exploration activities aside from the implementation of mine grid in the 1980s and recent topographical LiDAR survey and drill hole collar surveys.

Exploration drilling in and around the Farim property area has been carried out by several companies since discovery of the deposit. The current database contains 291 drillholes comprising 14,724 m of drilling using a combination of percussion and core drilling techniques. Itafos has not conducted any exploration on the property since acquiring it in 2018.

Drilling prior to 2015 was primarily in support of mineral resource estimation. The bulk of the drilling completed since 2015 was in support of geotechnical characterization, metallurgical sampling, overburden characterization, and water monitoring programs unrelated to the mineral resource update.

Since the layers of phosphate are horizontal, all the drillholes were drilled vertically; therefore, the thicknesses intercepted are believed to be true thicknesses. The average depth of the drillholes is 51 m.

1.7 Sample Preparation, Analyses and Security

Sampling has been undertaken on the Farim property since 1983, including chemical assays and density determination. Different techniques, typical for the time in which they were analyzed, were employed.

Quality assurance and quality control (QA/QC) programs have been implemented comprising pulp duplicates, field duplicates and standards. Internal laboratory QA/QC programs were also implemented by the laboratories used for each program since the 1980s. Review of the various QA/QC results indicated limited sample bias, particularly in the high-grade FPA unit.

1.8 Mineral Processing and Metallurgical Testwork

1.8.1 Ore Characterization

Core samples from the Farim South pit phosphate deposit were received at KEMWorks on December 26, 2014. This sample consisted of four subsamples that corresponded to the block model and assay model data for the deposit, representing the first seven years of production (South pit). The samples showed that the main contaminants were acid insoluble (A.I.) 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 to be representative of the deposit. A South pit weighted composite was prepared for characterization studies, horizontal scrubbing (drum), attrition scrubbing, and reverse amine flotation tests.

For the Farim North pit phosphate ore, the objective of the bench-scale beneficiation testwork was to characterize phosphate samples, and to determine the validity of the beneficiation process which was developed for the South pit to the

North pit. The North pit samples were significantly more clayish and wet than those of the South pit. A single North pit composite sample was prepared for three exploratory tests. The North pit phosphate ore was significantly finer than the South pit ore, with a mean particle size (d_{50}) of 115 µm. The mode particle size was at 212 µm (unimodal).

The results of the head sample chemical analysis showed that the South pit composite P_2O_5 grade was 33.0% ± 0.7% with a 2.1% error, resulting in a P_2O_5 grade between 31.5% to 34.5% range. The characterization studies, head chemical analysis, screen analyses, and screen assays, showed that the North pit composite sample has a lower P_2O_5 grade, higher A.I., and high organic carbon ($C_{organic}$), reporting 30.92% P_2O_5.

The metallurgical parameters for the South and North pits are shown in Table 1-1.

Table 1-1: Metallurgical Parameters for Farim ore

Parameter	South Pit	North Pit
CaO/ P_2O_5 Ratio	1.4	1.4
Minor Element Ratio (MER)	0.141	0.152
Adjusted Minor Element Ratio (MER*)	0.079	0.094
P_2O_5 Grade Potential	36.5%	33.3%

The particle size distribution (PSD) of the Farim South pit composite reported a mean particle size (d50) of140 µm 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 some of the Fe_2O_3 seemed to constitute part of the aluminum silicates. The acid insoluble (A.I.) is evenly distributed throughout all size fractions coarser than 106 µm and decreasing for particles smaller than 106 µm. The A.I. is the most critical impurity to be rejected. QEMSCAN results confirmed the interpretation and conclusions of the screen assays.

1.8.2 Horizontal Scrubbing

Tests were conducted on Farim South pit phosphate ore under standard conditions as a baseline first, and at six different conditions to evaluate two solids contents (35% and 50%) at three scrubbing times: 150 seconds, 300 seconds, and 600 seconds. These tests showed that A.I., Al_2O_3, Fe_2O_3, S_{total}, S, and MgO decreased in the product size range (1,180 x 20 µm). 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%) were obtained. The metallurgical parameters were as follows:

- CaO/ P_2O_5 Ratio.. 1.4
- MER..0.103
- MER*..0.034

Confirmation tests validated these results. These tests considered the +6,300 µm and 6,300 x 1,180 µm size fractions as rejects and the -20 µm fraction as slimes.

1.8.3 Attrition Scrubbing

Tests were designed to release significant amounts of quartz, clay, and iron-bearing minerals attached to the francolite surfaces in the 6,300 x 75 µm size fraction obtained after horizontal scrubbing (drum) of the Farim South pit phosphate ore. Nine tests were carried out at three solids contents (45%, 55%, and 60% by mass) for three different scrubbing times, 150 seconds, 300 seconds, and 600 seconds. The South pit metallurgical results shown in Table 1-2 were the best results obtained at 55% solids content and scrubbing for 150 seconds (2.5 minutes).

Three tests on North pit composite were conducted applying the beneficiation process designed for the South pit ore. The North pit metallurgical results in Table 1-2 showed that the beneficiation process was suitable for the North pit and the results were reproducible. However, the process was not able to achieve a 34% P_2O_5 grade in the concentrate due to high A.I. content; the average combined product (1,180 x 20 µm size fraction) was 32.3% P_2O_5.

Table 1-2: Attrition Scrubbing Metallurgical Results for Farim Ore

Parameter	South Pit	North Pit
Mass Yield	73.9%	74.3%
P_2O_5 Recovery	77.2%	76.8%
CaO/ P_2O_5 Ratio	1.5	1.4
MER	0.075	0.116
MER*	0.070	0.078
P_2O_5 Grade	33.8%	32.3%

1.8.4 Pilot Plant Results

Four pilot plant tests were carried out. Based on the average data obtained from the pilot plant tests, a metallurgical balance most likely to be obtained in the industrial plant was estimated. A yield (mass recovery) of 77.5%, and P_2O_5 recoveries between 81.4% and 84.3% were estimated; the most likely P_2O_5 recovery was 81.8%. In the case of the P_2O_5 grade of the combined concentrate, the results were between 33.6% and 34.7%, with the most likely P_2O_5 grade being 33.6%. Thus, the most likely material balance and parameters were as follows:

- Mass Yield........77.5%
- P_2O_5 Recovery........81.8%
- CaO/ P_2O_5 Ratio........1.4
- MER........0.108
- MER*........0.078
- P_2O_5 Grade........33.6%

1.9 Mineral Resource Estimation

This section contains forward-looking information related to mineral resource estimates for the project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this subsection including geological and grade interpretations and controls and assumptions and forecasts associated with establishing reasonable prospects for eventual economic extraction.

The Farim deposit has been delineated over an area of approximately 40 km² and is divided by the River Cacheu. The deposit consists of both a decarbonized phosphate unit (FPA) and a calcareous phosphate member (FPB). The mean depth of the FPA is approximately 40 m and the mean thickness is approximately 3 m. This mineral resource estimate concerns the FPA unit only, as the FPB unit is currently deemed uneconomic based on in-situ P_2O_5 grade. No additional mineralization outside the deposit modelled was considered in the mineral resource estimate.

The QP modelled the Farim resource based on a 2D grid of 125 m x 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 drillholes. 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 QP considers the mineralization contained within the Farim deposit to fulfil the criteria of "'Reasonable Prospects for Eventual Economic Extraction" to be reported as a mineral resource. A 20% P_2O_5 cut-off grade and a minimum FPA thickness of 1 m was applied by the QP to establish reasonable prospects for eventual economic extraction. The 20% P_2O_5 cut-off grade was applied to target the in-situ mineral resource grade requirements that would subsequently meet the plant feed and product grade requirements with the application of mine design and mineral processing considerations.

Table 1-3 summarizes the results of the mineral resource estimate based on a 20% P_2O_5 cut-off grade minimum FPA thickness of 1.0 m and a constant density of 1.4 t/m^3. Estimated mineral resources within the extents of the revenue factor (RF) 1.2 resource pit design are provided in Table 1-3, which summarizes the global mineral resource estimate. This assumes the mineral resource would be exploitable using open pit mining methods.

Table 1-3: Global Mineral Resource Statement, Farim Phosphate Deposit, September 30, 2022

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	102.5	2.91	28.53	2.69	39.71	5.65	11.28	1,162.30	11.34
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	**102.5**	**2.91**	**28.53**	**2.69**	**39.71**	**5.65**	**11.28**	**1,162.30**	**11.34**
Indicated	North of River	-	-	-	-	-	-	-	-	-
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-
Measured + Indicated	North of River	102.5	2.91	28.53	2.69	39.71	5.65	11.28	1,162.30	11.34
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	**102.5**	**2.91**	**28.53**	**2.69**	**39.71**	**5.65**	**11.28**	**1,162.30**	**11.34**
Inferred	North of River	6.8	2.30	25.17	2.99	39.08	4.86	10.46	119.62	17.63
	South of River	24.4	2.21	29.06	5.32	36.21	4.97	11.62	236.18	9.70
	Subtotal	**31.1**	**2.23**	**28.08**	**4.73**	**36.94**	**4.94**	**11.32**	**355.80**	**11.42**

Notes: 1. Mineral resources are reported on a dry in-situ basis and are inclusive of mineral reserves. 2. The statement of estimates of mineral resources has been compiled by Mr. Jerry DeWolfe, who is a full-time employee of WSP Canada Inc. (formerly WSP Golder) and a professional geologist (P.Geo.) with the Association of Professional Engineers and Geoscientists of Alberta (APEGA). Mr. DeWolfe has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a QP as defined in NI 43-101. 3. All mineral resources figures reported in the table above represent estimates at September 30, 2022. Mineral resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies. 4. Mineral resources are reported in accordance with NI 43-101 and CIM Definition Standards for Mineral Resource and Mineral Reserves (2014) and CIM Estimation of Mineral Resource and Mineral Reserve Best Practices (2019).. 5. The reported mineral resource estimate was constrained by a conceptual mineral resource optimized pit shell for the purpose of establishing reasonable prospects of economic extraction based on potential mining, metallurgical and processing grade parameters identified by mining, metallurgical and processing studies performed to date on the project. Key inputs in developing the mineral resource pit shell included a mining cost of US$1.69/tonne for ore and US1.41/tonne for waste, plus processing costs of US$31.72/ ROM tonne, phosphate recovery of 76%, pit slope angle of 20°, and a concentrate selling price of US$147/tonne. In addition, a minimum FPA P_2O_5 grade of 20%, a minimum FPA thickness of 1 m as well as a restriction on any FPA within 50 m of River Cacheu was applied.

The global mineral resource estimate, with a date of September 30, 2022, defines a measured resource of 102.5 Mt at a mean grade of 28.5% P_2O_5 and an inferred resource of 31.1 Mt at a mean grade of 28.0% 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, CIM Definition Standards for Mineral Resource and Mineral Reserves (2014) and CIM Estimation of Mineral Resource and Mineral Reserve Best Practices (2019).

Note to readers: The mineral resources presented in this section are not mineral reserves and do not reflect demonstrated economic viability. The reported inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that all or any part of this mineral resource will be converted into mineral reserves. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.

1.10 Mineral Reserve Estimation

This subsection contains forward-looking information related to mineral reserve estimates for the project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this subsection including mineral resource model tonnes and product quality, modifying factors including mining and recovery factors, production rate and schedule, contractor mining equipment productivity, commodity market and prices and projected operating and capital costs.

This mineral reserve estimate concerns the FPA unit only, as the FPB unit was previously deemed to be uneconomic. No additional mineralization outside the modelled deposit was considered in the mineral resource and reserve estimates.

The reserve estimation was undertaken in Datamine's MineScape™ software (version 2021). The mineral reserve statement has a date of September 30, 2022.

The assessment of 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 43.75 Mt of ROM material to feed a 25-year plan at a rate of 1.75 Mt/a on a dry basis.

As per the mineral resource estimation methodology, a 20% P_2O_5 technical cut-off grade was applied to target the in-situ mineral resource grade requirements that would subsequently meet the plant feed and product grade requirements. This technical cut-off grade did not change in the reserve estimation.

Estimated ROM phosphate matrix reserves and phosphate rock reserves for the proposed 25-year, 1.75 Mt/a pit are listed in Table 1-4. The QP considers the criteria used to define the 25-year mineral inventory to be reasonable for public reporting.

For the Farim phosphate deposit beneficiation circuit, the total estimated proven and probable reserves are 43.75 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 10.09 bank cubic meters (bcm) per tonne of ROM phosphate matrix (17 tonnes overburden per tonne of ROM phosphate matrix), requiring the removal of approximately 441.3 Mbcm of overburden over the life of the mine.

Table 1-4: Proven and Probable Reserves

Category	ROM (Plant Feed) FPA Tonnes, Dry Basis (Mt)	Mean ROM P_2O_5, Dry Basis (%)	Mean ROM Al_2O_3, Dry Basis (%)	Mean ROM CaO, Dry Basis (%)	Mean ROM Fe_2O_3, Dry Basis (%)	Mean ROM SiO_2, Dry Basis (%)
Proven	43.8	30.0	2.6	41.1	4.8	10.6
Probable	-	-	-	-	-	-
Total	**43.8**	**30.0**	**2.6**	**41.1**	**4.8**	**10.6**

Notes: 1. Mineral reserves are reported on a dry in-situ basis. 2. The statement of estimates of mineral reserves has been compiled by Mr. Terry L. Kremmel, who is a full-time employee of WSP USA Inc. (formerly WSP Golder) and a professional engineer (P.E.) and registered member with the Society for Mining, Metallurgy, and Exploration. Mr. Kremmel has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a QP as defined in NI 43-101. 3. All mineral reserves figures reported in the table above represent estimates at September 30,, 2022. Mineral reserve estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies. 4. Mineral reserves are reported in accordance with NI 43-101. 5. The reported mineral reserve estimate was constrained by The River Cacheu, the Rio de Bunja, and surface encumbrances including the two ex-pit waste dumps, tailings storage facility, and processing plant.

1.11 Mining Methods

The FPA matrix is mined by a free-dig, multiple-bench, open-pit, haul-back mine using excavators and trucks. The QP selected the excavator/truck mining method based on lower initial capital, lower investment risk, increased grade control, and limited power supply requirements.

For the 1.75 million tonnes per annum (Mt/a) (dry basis) open pit, it is planned that overburden will be stripped and removed with 12 cubic meter (m^3) front-end loaders (FEL) 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 blended into a ROM bin by front-end loaders with 12 m^3 buckets to achieve the desired plant feed P_2O_5 grade. The plant feed hopper will be installed so that haul trucks can directly feed matrix to the plant.

Overburden excavation will advance ahead of the matrix extraction in maximum 10 m height production benches. The overburden thickness ranges from 26 to 68m within the 25-year pit, multiple overburden stripping benches will be developed and maintained in advance of the matrix extraction. The matrix thickness ranges between 1.5 m and 6.25 m within the 25-year pit.

The most critical design element of the proposed mining plan is water management. All mining areas must be dewatered in advance of mining activities to allow sufficient depressurization and dissipation of pore water pressure and 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 for 65° temporary dig face angles. The proximity of the mine site to the River Cacheu will require the construction of a protective water control berm (bund) to prevent in-pit flooding. Sufficient overburden material will be diverted to begin construction of 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 advanced 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. Mine plan parameters and factors are summarized in Table 1-5.

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 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.

Annual mine plan production statistics are provided in Table 1-6. The mine plan production scenario was targeted to produce approximately 2.19 Mt/a of ROM phosphate matrix on an as-received basis (at approximately 20% moisture) or 1.75 Mt/a ROM phosphate matrix on a dry basis.

North and South pit cross-sections are shown in Figures 1-3 and 1-4, respectively.

Table 1-5: Mine Plan Parameters and Factors

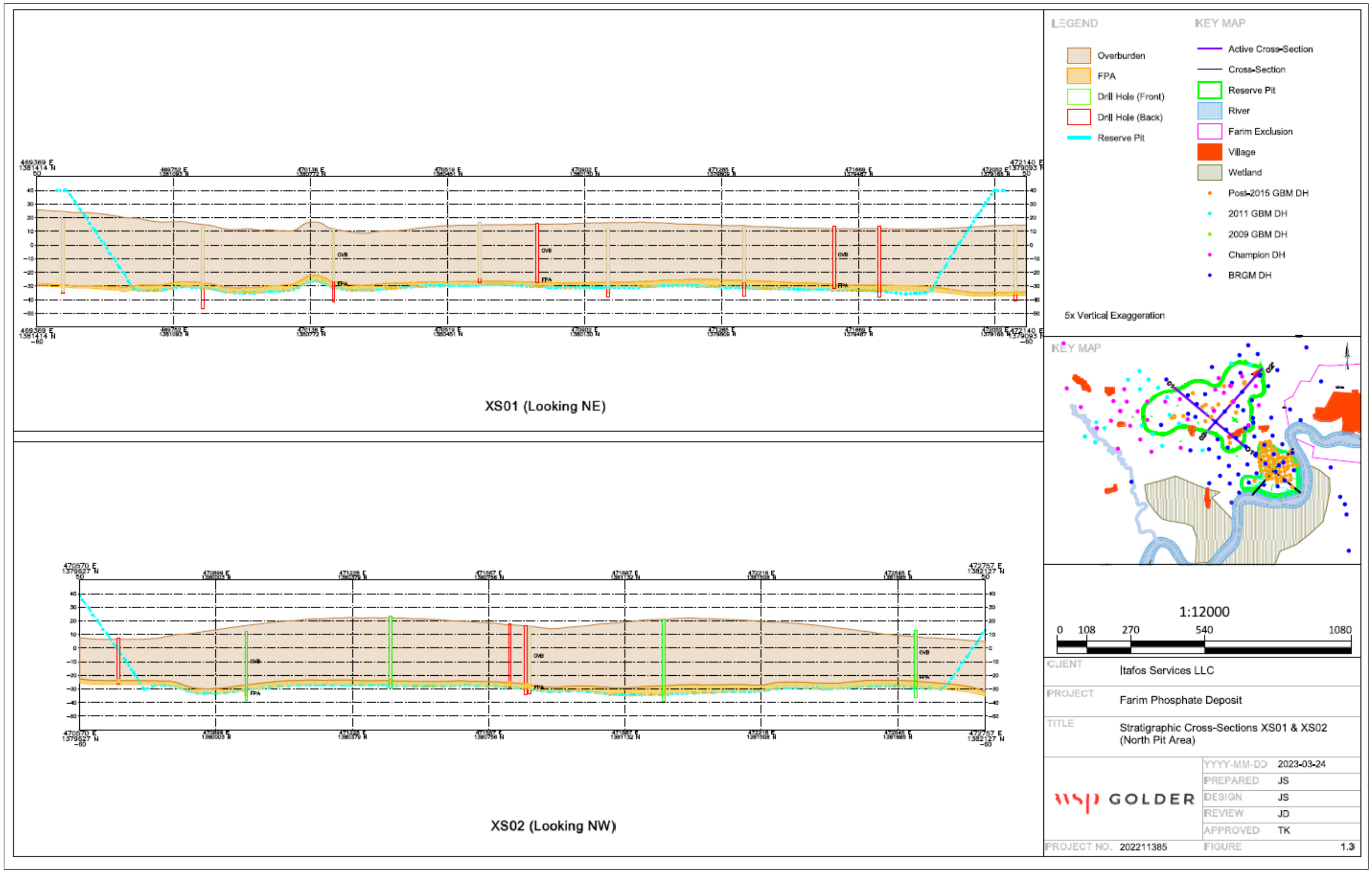
Description	Value
Permanent Wall Angle	20°
Permanent Wall Operational FOS	>1.3
Bench Height	10 m
Short-Term Bench Face (Batter) Angle	65°
Short-Term Berm Width	14.9 m
Long-Term Bench Face (Batter) Angle (After Sloughing)	25°
Long-Term Berm Width (After Sloughing)	6.5 m
Overburden Angle of Repose WD/IOB/SOS	1V:5H / 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

Table 1-6: Annual Mine Plan Production Statistics

Production Statistics	Units	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	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Total
ROM Waste Stripping	kbcm	5,812	8,661	10,081	14,892	12,969	16,800	13,510	17,348	17,847	15,017	18,005	18,375	17,083	17,768	16,525	19,356	23,981	23,006	19,457	17,438	16,609	15,228	18,335	23,654	21,300	22,470	441,513
Total Prime Material (Waste + Ore) Moved	kbcm	5,812	10,050	11,456	16,275	14,350	18,181	14,893	24,448	12,942	16,267	19,255	19,625	18,333	19,018	17,775	20,606	25,231	24,256	20,707	18,688	17,859	16,478	19,585	24,904	22,550	23,641	473,170
Total ROM Tonnes, Wet Basis (20% moisture)	kt	0	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190
Total ROM Tonnes, Dry Basis	kt	0	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	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Total Product Tonnes, Dry Basis	kt	0	1,356	1,356	1,356	1,356	1,356	1,356	1,345	1,300	1,300	1,300	1,301	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,312	32,899
Product Grade	%P_2O_5		33.6	33.6	33.6	33.6	33.6	33.6	33.6	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	32.3	
Total Tailings Tonnes, Dry Basis	kt	0	280	280	280	280	280	280	299	380	380	380	380	380	380	380	380	380	380	380	380	380	380	380	380	380	383	8,818
Prime ROM Strip Ratio, Dry Basis	bcm / ROM tonne	n/a	4.95	5.76	8.51	7.41	9.60	7.72	9.91	10.20	8.58	10.29	10.49	9.76	10.15	9.44	11.06	13.70	13.15	11.12	9.96	9.49	8.70	10.48	13.52	12.17	12.72	10.09
Effective Product Strip Ratio, Dry Basis	bcm / product tonne	n/a	6.39	7.43	10.98	9.56	12.39	9.96	12.89	13.73	11.55	13.85	14.12	13.14	13.66	12.71	14.89	18.44	17.69	14.96	13.41	12.77	11.71	14.10	18.19	16.38	17.12	13.42
Effective Waste Haulage Volumes	kbcm	5,812	8,661	10,081	14,892	12,969	16,800	13,510	17,348	17,847	15,017	18,005	18,375	17,083	17,768	16,525	19,356	23,981	23,006	19,457	17,438	16,609	15,228	18,335	23,654	21,300	22,262	441,319
In-Pit Overburden Backfilling (IOB)	kbcm	0	2,305	8,162	12,271	9,904	16,725	13,296	14,522	9,560	12,035	12,433	11,030	9,426	10,992	11,297	10,672	21,291	13,738	18,009	16,891	15,123	15,228	16,879	18,668	21,300	22,262	344,931
Surcharge Overburden Stockpiling (SOS)	kbcm	0	0	0	0	504	0	0	0	4,628	2,982	5,572	7,344	7,658	6,282	5,228	7,050	2,690	9,268	0	547	1,486	0	765	4,986	0	0	66,991
Construction Material Haulage	kbcm	914	175	1,919	224	100	75	214	2,826	0	0	0	0	0	494	0	1,634	0	0	1,448	0	0	0	691	0	0	0	10,713
Ex-Pit	kbcm	4,898	6,181	0	2,398	2,462	0	0	0	3,659	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	19,597

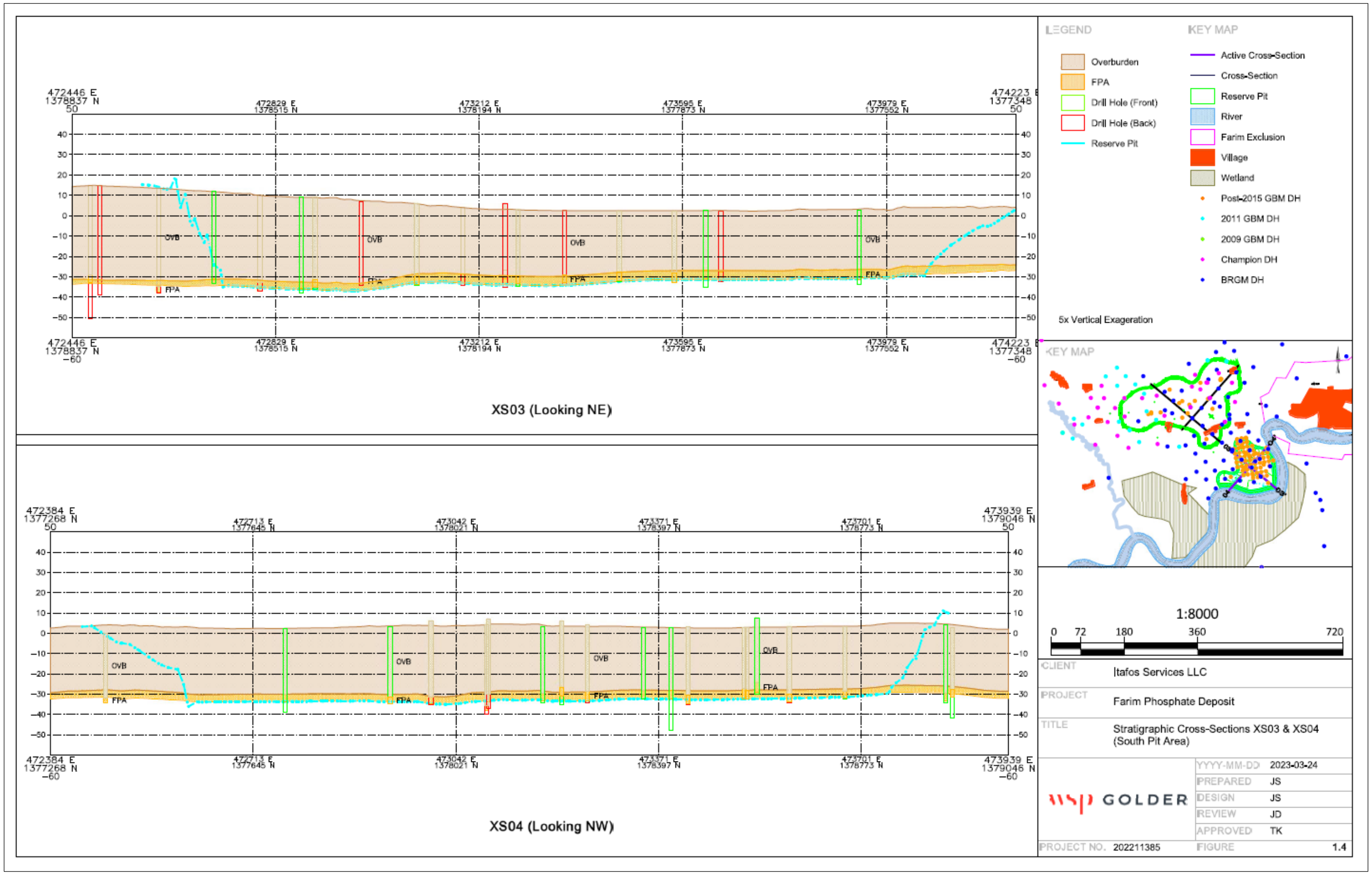
Notes: Expected product tonnages are based on an average 77.5% plant mass yield for the South pit and a 74.3% plant mass yield for the North pit. Effective ROM Strip Ratio includes waste rehandling.

Figure 1-3: North Pit Cross-Sections



Source: WSP Golder, 2023

Figure 1-4: South Pit Cross-Sections



Source: WSP Golder, 2023

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 25 m x 25 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.

The key factors driving the progression of the sequence were annual ROM production, delay of handling potentially acid-generating (PAG) overburden material until there was sufficient mined-out room within the pit to properly store the material, dewatering and surface water management, and backfill opportunities. The mine sequence includes six months of pre-stripping in "Year 0" to allow for immediate matrix production in Year 1.

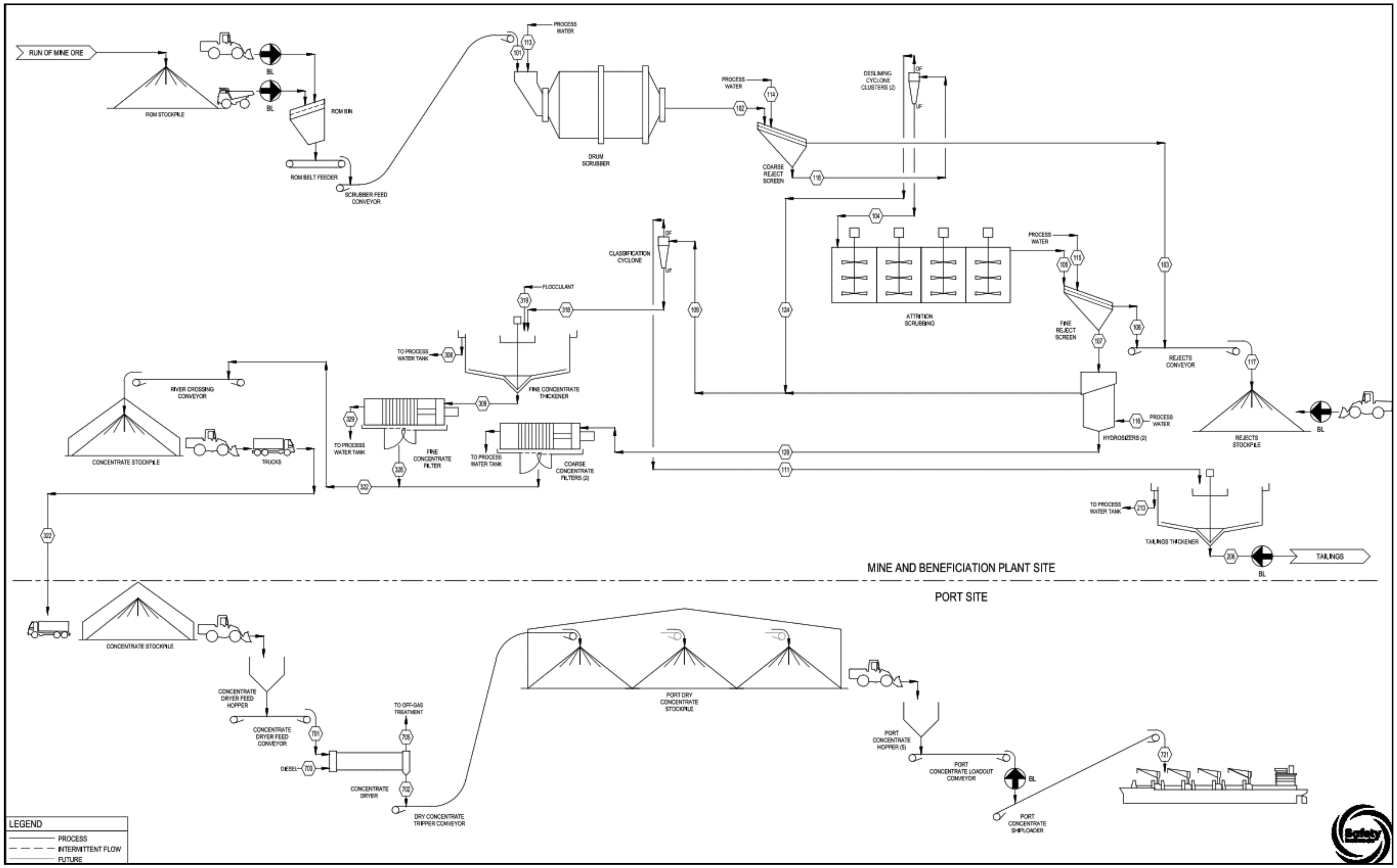
1.12 Recovery Methods

The proposed process design includes the unit operations required to receive run-of-mine ore from mining operations, reject coarse material to waste, deliver tailings to the tailings storage facility (TSF), and deliver dry concentrate to ships. The design considers two distinct feed sources, from the South pit in Years 1 to 7 and from the North pit in Years 8 to 25. The process plant is designed to achieve an annual throughput of 1.75 Mt/a. The material from the South and North pits are expected to produce 1.36 Mt/a and 1.30 Mt/a of concentrate annually, respectively. The design is divided geographically between the beneficiation plant, near the open pit mining operations, and the Mineral Terminal (MT) site. The overall process flow diagram is presented in Figure 1-5.

At the beneficiation plant, the process objective is to remove impurities to achieve the required minor element ratio and phosphate grade in the concentrate. Impurity removal is achieved by concentrating the -20 µm to +1,180 µm particle size fraction of the ROM ore and rejecting the remainder. ROM ore is processed through drum and attrition scrubbing stages, and classified by cyclones, vibrating screens, and hydro separators. The -20 µm size fraction is thickened and pumped to the TSF. The +1,180 µm material is rejected and trucked to a waste stockpile. The resulting fine concentrate stream is thickened and filtered in a horizontal plate and frame filter press. The coarse concentrate stream does not require thickening and is sent directly to two vertical plate-and-frame filter presses operating in parallel. The concentrate filter cakes are combined and conveyed to a covered wet concentrate stockpile. The wet concentrate is then reclaimed and trucked to the MT.

The process objective at the MT site is to dry the concentrate to a moisture content suitable for transport after which it is loaded on to ships. Wet concentrate is received at the MT site in a covered stockpile building. The material is then reclaimed and dried in a diesel fired rotary dryer. Dry concentrate is then stockpiled in a covered building, prior to reclamation and shiploading.

Figure 1-5: Overall Process Flow Diagram



Source: Ausenco, 2023

1.13 Project Infrastructure

Local mining infrastructure is limited and must be upgraded, or in some cases, designed and built as part of the initial construction plan. Although the government of Guinea-Bissau is advancing infrastructure improvements across the country, this study assumes the following key infrastructure works.

- Truck loading facility on the south side of River Cacheu. Concentrate will be transported from the plant (north side of the River), via a conveyor over the River Cacheu. This conveyor will also house a pipe to transfer diesel fuel to storage tanks at the mine site. These facilities are all within the mining lease.
- Upgraded access road from Ponta Chugue to Mansoa (remainder of road to the truck loading site is approved and acceptable for truck haulage and access).
- Mineral Terminal at Ponta Chugue to load and ship the dried concentrate. Ponta Chugue will also be used to accept diesel fuel into holding tanks for delivery to Farim.
- Hybrid power plants (solar and diesel generator) are located at Ponta Chugue and north-east of the Farim process plant. These power plants will be mobilized under a Build, Own, Operate, and Maintain, (BOOM model) by a supplier/contractor already working in the region. Power pricing reflects this BOOM model.
- Tailings storage facility (TSF) adjacent to the beneficiation plant to store fines generated from the process facility. This TSF will be developed in stages as individual cells over the life of mine. In addition to tailings, the TSF is designed to handle rainfall, supernatant water released from the tailings, environmental design flood (EDF) volume, and wave run-up. Water will be reclaimed from the TSF for reuse in processing by pumping to a return water pond (RWP) adjacent to the TSF before pumping to the process plant.
- Waste overburden storage piles for permanent storage of overburden. A cell within one of the waste storage piles will be designed to store potentially acid generating (PAG) material based on the mining sequence and expected PAG volumes.
- Temporary topsoil storage piles sufficient to manage development of waste piles, roads, TSF cell construction, and for use in closure plans.
- Water management system including supply wells, dewatering wells, water diversion channels, flood prevention berms, and settlement ponds. The site will continuously discharge water throughout the operation. Quantities and quality of the water, as well as how it's handled, will vary according to the source.
- Camp facilities already built will be supported by local contractors and be secure.

Details regarding these key infrastructure elements are found in Section 18. All associated infrastructure costs are captured in the construction and operating plans. All future infrastructure development by the Government of Guinea-Bissau is considered as opportunities to enhance the Farim project.

1.14 Environmental Considerations

Comprehensive environmental and social baseline studies were conducted for the project from 2011 through 2015, supporting an ESIA published by Knight Piésold (KP) in September 2015 (KP, 2015a). The 2015 ESIA for the project, as well as a subsequent ESIA for the Buredanfa Resettlement Village, was approved by the Government of Guinea-Bissau (see Permitting Considerations).

Additional baseline studies were conducted from 2016 to 2019 in the areas of meteorology, air quality, noise, groundwater resources, and groundwater and surface water quality to establish an additional and contemporary pre-development baseline record that can be used for comparison in future monitoring programs.

Environmental management plans were advanced for air quality, noise, and water in 2015 and 2016 (KP, 2015b, 2016a, 2016b, and 2016c), and KP provided training to Itafos environmental technicians for these programs and assisted in their oversight and data management, including uploading all the monitoring records into KP's web-based data management tool. Itafos environmental technicians have implemented these monitoring programs since 2016, with some interruptions. In 2021, KP provided a status report to Itafos on these environmental monitoring programs, including a summary of the data collected to date, a data quality review, and recommendations for monitoring in future years in the absence of a construction decision on the project (KP, 2021). These programs will need to be resumed to refresh the pre-development baseline once a construction decision is made.

Updated biodiversity baseline studies are recommended, along with an update to the Biodiversity Management Plan, to account for changes in the conservation status or abundance and distribution of Red Listed species that may have occurred since 2015.

The tailings solids that will be produced by the mine contain high levels of element enrichment with antimony, bismuth, phosphorous, selenium and uranium. However, both short-term leach testing and humidity cell testing demonstrated that only cadmium and nickel were prone to leaching when screened against the River Cacheu Target Receiving Water Quality Standards. The tailings supernatant also demonstrated elevated cadmium and nickel concentrations, which suggest that these parameters may be a result of ore processing and not metal leaching of the tailings solids. Radionuclide testing on the tailings materials indicated that both the lead-210 and radium-226 concentrations from all samples were above Health Canada release limits; however, this was not demonstrated in the tailings supernatant. Given that no exceedances were noted within the tailings liquid, it is likely that the near-neutral pH of the tailings leachate/supernatant does not allow for the isotopes to be mobilized in water.

With respect to public exposure, based on the available data for Farim, exposure to the tailings and ore could theoretically result in doses that exceed Health Canada's (2011) dose constraint of 0.3 mSv/y and the incremental dose of 1 mSv/y which are used for the protection of the public (NECA, 2015). The public would have to be exposed for a full year, which is unlikely as the operating mine will not be accessible to the public. At closure, if local residents seek to inhabit the area again, long-term exposure can be managed by constructing a suitably designed cover on the tailings storage facility that adequately shields exposure.

Limiting exposure to radium-226 can also be achieved by constructing open-walled structures in proximity to the TSF. Radium-226 can build up within walled structures over time, which could result in a higher dose to anyone entering those buildings. Open-walled structures allow for the passage of air, which alleviates the risk of radium-226 buildup.

Based on geochemical testing of waste overburden conducted to date, approximately 33% and 13% of the waste from the South and North pits, respectively, are potentially acid generating (PAG). Nearly all of the PAG waste can be placed back into the pit, except for some PAG waste generated in Years 1 to 2 of mining, which will be placed in a lined section of waste dump WD-1. A basal low permeability soil liner will be formed beneath the footprint of WD-1 and seepage/runoff flows at the base of WD-1 will report to the environmental control dam. Any contaminated water collected in the environmental control dam will be returned to the process plant for re-use via the RWP.

Key environmental effects requiring management include the following:

- Waste overburden and tailings will require management to protect groundwater and surface water.
- Pit dewatering will likely affect household and community wells, so a monitoring program will be required to verify any claims of effects, and the company will need to establish plans to provide water of at least equivalent quantity and quality to affected groundwater users in the area.
- The project will result in ecological impacts, including the loss of mangroves, salt marsh and freshwater areas, as well as secondary forest. Lost mangrove habitat may in turn contribute to riverbank instability and erosion coupled with a loss of crocodile habitat. Establishment of the Buredanfa Resettlement Village and associated livelihood restoration area will result in the loss of indigenous forest. A decrease in forest habitats represents a loss of habitat for primates. Updated ecological studies should be completed before significant construction activities begin to

confirm current inventories and status of species of conservation concern, and to implement an updated Biodiversity Management Plan, potentially with biodiversity offsets.

1.14.1 Closure and Reclamation Considerations

A preliminary Mine Reclamation and Closure Plan (MRCP) and closure cost estimate has been prepared that meets the requirements under Guinea-Bissau's Mining and Minerals Law 1/2000. The MRCP adopts the International Finance Corporation's closure objectives in terms of protecting future public health and safety; ensuring the after-use of the site is beneficial, sustainable, and appropriate for the affected communities in the long-term; minimizing adverse socioeconomic impacts; and maximizing benefits (IFC, 2012).

The MRCP contemplates the progressive rehabilitation of several facilities at the mine, including the overburden waste dumps and the North and South open pits. The South pit and most of the North pit will be backfilled with waste overburden as part of operations. A void in the North pit, representing the final few years of mining, will be backfilled at closure with waste overburden taken from a nearby surface waste dump. The channel of the Rio de Cavaras Marinhos will be re-established, this time connecting to the North pit pond that will form over approximately four to five years. The diversion dam for the Rio de Cavaras Marinhos will be decommissioned.

The TSF will be progressively covered with 1.5 m of NAG waste overburden. The first six cells will be covered during mine operations by diverting NAG waste overburden. Cell 7 will be closed out during the two-year active closure phase. Because the tailings are low density and are expected to be slow to consolidate, a reclamation research program is proposed on Cell 1 to refine the closure cover approach.

The 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 Ponte Chugue Mineral Terminal, buildings, machinery and equipment will be decommissioned and removed from the site. Remediation will be undertaken, as required, so that the MT site will be compatible with future commercial or industrial land use. 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 15 years 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 socioeconomic impacts of closure as part of the company's Community Development Plan.

1.14.2 Permitting Considerations

The Mining Agreement is considered the global agreement aggregating and coordinating the above licenses and any other agreements or conditions relative to the project. The Mining Agreement in its entirety includes:

- Environmental Plan, or the Environmental and Social Management Plan, which is the Environmental and Social Impact Assessments (ESIAs) completed in 2015 and 2018 (KP, 2015a and 2019)
- mining lease (granted)
- mining license (granted)
- annex on incentives (pending).

Further, a Mining Operations Plan must be filed to fulfil the Mining Agreement.

An ESIA for the project was published in September 2015 and was shared with the Government of Guinea-Bissau and prospective lenders (KP, 2015a). An ESIA was subsequently prepared for the Buredanfa Resettlement Village (KP, 2019).

The project and the Buredanfa Resettlement Village were approved by the Government of Guinea-Bissau, according to a Declaração de Conformidade Ambiental (Declaration of Environmental Compliance) issued to Itafos on September 14, 2018 (Secretaria de Estado do Ambiente, 2018).

While the approval (declaration) expired in 2019, the Autoridade da Avaliação Ambiental Competente (Competent Environmental Assessment Authority) notified Itafos in March 2020 that the Authority had almost completely suspended its internal operations, and that the process of renewal of the environmental license will resume when the pandemic is over (Autoridade da Avaliação Ambiental Competente, 2020). To date, no further notification from the Autoridade da Avaliação Ambiental Competente has been received indicating the resumption of operations and the envisaged renewal of the environmental license. The relevant authority has been supportive of the project and Itafos does not anticipate any issues with this notification.

1.14.3 Social Considerations

Key social impacts that require management include:

- Community health, safety, and security – The project will interrupt the current flow of mostly pedestrian and bicycle traffic between the regional service center of Farim and villages to the west and north of the mine. In addition, the presence of the mine 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 MT site.
- Risk of influx and associated impacts – The presence of the mine may result in an influx of people into the region, which will require management in conjunction with the regional and national governments. The effects can be far-reaching in terms of social unrest, overloading of available public services and infrastructure, and causing increased pressures on ecological resources. A Community Health, Safety and Security Management Plan identifies these issues and proposes preliminary mitigation measures that can be discussed with the appropriate authorities.
- Involuntary resettlement – The project will require the acquisition of approximately 3,000 ha of land resulting in the physical and/or economic displacement of an estimated 175 households in villages in the mine area. Candidate host sites were identified, and a preferred site was selected at Buredanfa, immediately northwest of the mine. A livelihoods baseline and restoration strategy and resettlement action plan (RAP) was also prepared in 2017. Because time has passed since this work was completed, the communities that require resettlement may have grown, and it will be necessary to conduct another land and asset survey to update the RAP.
- Livelihood restoration – Other mine project components, such as the truck loadout facility, highway bypass around the town of Mansoa, and Mineral Terminal facility and associated access road, will be positioned on lands held by others. Compensation is planned as part of securing land tenure for these areas, although no household resettlement is required.
- Cultural Heritage – Development of the project will result in direct and unavoidable physical impacts on the following cultural heritage resources:
  - three cemeteries (one of high and two of low sensitivity)
  - two mosques (both of high sensitivity)
  - three sacred sites (one of high and two of low sensitivity)
  - six archaeological sites (two of medium and four of low sensitivity).

Mitigation of these resources will involve the development of a grave relocation plan, a sacred site relocation plan, and a mosque relocation plan. The re-interments will be conducted in consultation with local communities, with the involvement and agreement of the local community and relatives of the deceased. Sacred forests and mosques should also be relocated in consultation with affected communities. Relocation of sacred forests or sites usually refers to moving spirits as well as any ritual huts or offering jars from one tree or grove to another through the appropriate ritual ceremony and conditions. The archaeological sites will require data recovery/rescue excavations prior to construction.

1.15 Markets and Contracts

The medium-term (2023-2027) and long-term (2028-2040) phosphate outlook was assessed. The medium-term outlook was based on five-year demand and supply projections, and the long-term outlook was based on demand forecasts and estimates of capacity and capital requirements needed to meet projected demand. The results are summarized below and discussed in Section 19.

In the medium term, new capacity and higher operating rates are required to meet projected demand during the next five years. Annual phosphoric acid demand, calculated from demand forecasts for downstream products, is projected to increase by 6.36 Mt from 47.51 Mt in 2020 to 53.87 Mt in 2027.

Effective phosphoric acid capacity is expected to increase 4.46 Mt P_2O_5, and the global operating rate is projected to increase from 86% to 92%to meet projected demand during the next five years. Assuming a 92% recovery rate, an additional 21.6 Mt of K-10 phosphate rock will be required to produce an additional 6.36 Mt P_2O_5 of phosphoric acid.

In the long term, demand is projected to increase 1.4% per year or 10.98 Mt P_2O_5 from 53.87 Mt in 2027 to 64.84 Mt in 2040. Fertilizer demand is forecast to increase 1.3% per year or 8.11 Mt P_2O_5. Non-fertilizer demand is forecast to increase at a faster pace of 2.4% per year or 2.86 Mt. Non-fertilizer demand is expected to grow at a faster pace largely due to rapid growth in purified phosphoric acid demand to produce technical MAP (tMAP). How battery technology will evolve still is uncertain, but the growth of LFP battery demand is a positive if not necessarily a game-changing demand development.

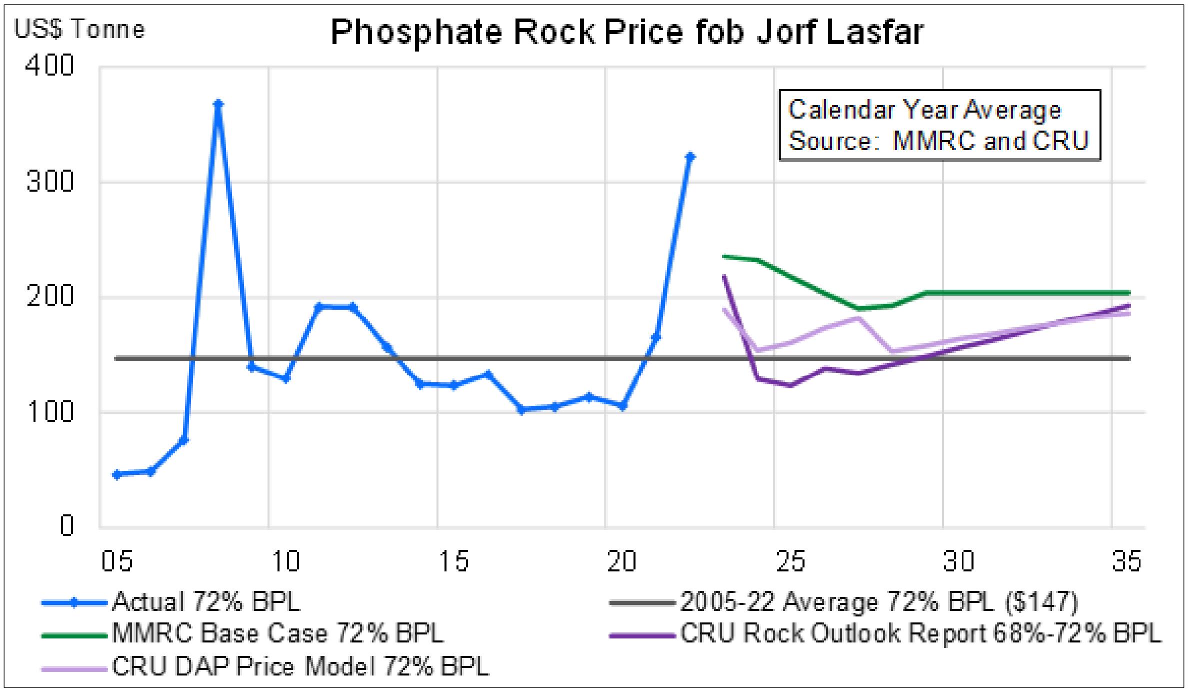
No attempt is made to speculate about what projects will get developed to meet projected demand growth during the 2028 to 2040 forecast period. Morocco will likely develop significant new capacity during this period given its comparative rock advantage and strategy to capture one-half of projected global demand growth. Saudi Arabia likely will continue to build additional capacity as part of an industrial policy to diversify its economy away from petroleum. Other greenfield projects in Algeria, other African countries, Peru, Brazil and eastern Canada are also potential candidates.

Significant new phosphate capacity and capital investment are required to meet projected demand growth during this forecast period. Where new capacity gets built likely will turn increasingly on the availably of economically viable phosphate rock reserves. Additional demand of 10.98 Mt P_2O_5 of phosphoric acid will require more than 37 Mt of K-10 quality rock, (32.04% P_2O_5).

These estimates do not include the capacity or capital needed to replace phosphate rock mines that exhaust reserves during the forecast period. For example, several other U.S. mines are expected to exhaust reserves during the 2028 to 2040 forecast period. In some cases, no viable reserves exist. In other cases, the permitting of viable reserves is uncertain.

Based on these factors and analysis of the long-term DAP price forecasts and statistical relationship between DAP and rock price. the Farim project has forecasted a phosphate rock price for 72% BPL rock FOB Jorf Lasfar Morocco, (MRRC rock price). The average DAP price forecast from 2023 to 2035 is $668 per tonne FOB Jorf, which translates into an average rock price of $207 per tonne for this period. Figure 1-6 shows this relationship along with projected prices used in the cash flow calculations.

Figure 1-6: Phosphate Rock Price FOB Jorf Lasfar



Source: MMRC and CRU, 2022

1.16 Capital Cost Estimate

Table 1-7 provides a summary of the project capital cost estimate, with costs grouped into major scope areas, expressed in Q4 2022 US Dollars. The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International).

The estimate is based on an EPCM execution approach. The following parameters and qualifications were considered:

- No allowance has been made for exchange rate fluctuations.
- No escalation has been added to the estimate.
- A weighted contingency has been included.
- Foreign exchange conversion rates were included for any items not priced in US dollars.
- Data for the estimate have been obtained from numerous sources, including the following:
  - mine schedules
  - feasibility-level design
  - topographical information obtained from the site survey
  - geotechnical investigations
  - firm and budgetary quotes from international suppliers
  - data from similar recently completed studies and projects.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and analyzed. A percentage of contingency was allocated to each of these categories on a line-item basis based on the accuracy of the data. An overall weighted contingency amount was derived in this fashion.

Table 1-7: Project Capital Cost Estimate

Description	Initial Capital (US$M)	Sustaining Capital (US$M)	Total Capital (US$M)
Mining	32.243	265.348	297.591
Process Plant and Infrastructure	68.934	-	68.934
Ponte Chugue Infrastructure (Mineral Terminal & Drying)	99.728	12.050	111.778
Tailings Storage Facility & Water Management	14.049	57.722	71.771
South Pit Dewatering	4.420	12.737	17.157
North Pit Dewatering	-	20.995	20.995
Resettlement and Livelihood Restitution	11.985	5.635	17.620
EPCM	27.452	-	27.452
Indirects	6.057	-	6.057
Owners' Cost	11.637	-	11.637
Contingency	31.765	-	31.765
Progressive Closure and Rehabilitation (TSF)	-	58.817	58.817
Progressive Closure and Rehabilitation (Pits & WDs)		21.169	21.169
Total Site Closure		33.997	33.997
Salvage Value – Mine		-12.893	-12.893
Salvage Value – Port	-	-8.433	-8.433
Total	308.270	467.142	775.413

1.17 Operating Cost Estimates

The operating cost estimate includes mining, processing, shiploading, environmental, fuel, and general and administration (G&A) costs. The total life-of-mine operating cost is $2,270.5 million over 25 years. Of this total, mining accounts for $661.4 million, processing, G&A, and environmental accounts for $545.5 million, shiploading accounts for $111.3 million, and fuel accounts for $952.3 million. A summary of the average annual operating costs is presented in Table 1-8. The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International).

Table 1-8: Operating Cost Estimate Summary – Average Costs Per Pit

Description	Life-of-Mine Operating Cost			South Pit			North Pit		
	US$M	US$/t Feed	US$/t Conc.	US$M/a	US$/t Feed	US$/t Conc.	US$M/a	US$/t Feed	US$/t Conc.
Mining	661.4	15.1	20.1	31.3	17.9	23.1	24.6	14.0	18.9
Process	343.0	7.8	10.4	13.9	7.9	10.3	13.6	7.8	10.5
Shiploading	111.3	2.5	3.4	4.5	2.5	3.3	4.5	2.5	3.4
Tailings, Environment, Water	15.7	0.4	0.5	0.6	0.4	0.5	0.6	0.4	0.5
G&A	186.8	4.3	5.7	7.5	4.3	5.5	7.5	4.3	5.7
Fuel	952.3	21.8	28.9	35.4	20.2	26.1	39.1	22.4	30.1
Total	2,270.5	51.9	69.0	93.2	53.2	68.7	89.9	51.4	69.1

Note: Fuel is itemized separately and is not included in mining, processing, shiploading or G&A costs.

The operating cost estimate is based on the following assumptions:

- Q4 2022 US dollars without allowance for escalation.
- relevant exchange rates to convert to US dollars for equipment sourced in non-US dollars
- annual throughput of 1.75 dry Mt/a ROM feed rate
- mass yield concentrate of 77.5% w/w for the South pit and 74.3% w/w for the North pit
- diesel cost of US$0.86/L
- power cost of US$0.229/kWh for the process plant and US$0.257/kWh for the Mineral Terminal.

1.18 Economic Analysis

The results of the economic analyses in this report represent forward-looking information as defined under Canadian securities law. The results are subject to several known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Forward-looking information includes the following:

- mineral resource and mineral reserve estimates
- assumed commodity price and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- sustaining costs and proposed operating costs
- interpretations and assumptions regarding contract mining terms
- assumptions as to closure costs and closure requirements
- assumptions about environmental, permitting, and social risks.

The economic analysis was performed assuming a 10% discount rate as shown in Tables 1-9 to 1-11.

Income tax holiday is the subject of mining agreement negotiations between Itafos and the Government of Guinea-Bissau. If successful, the incentive annex would be approved and result in the NPV and IRR stated in Table 1.10. If an agreement cannot be reached regarding the tax holiday, Farim would be subject to an additional tax equivalent to 7.9% of the NPV. Results of this outcome are shown in Table 1.11.

Table 1-9: Financial Data (USD, Millions)

Description	Life-of-Mine (US$M)
Revenue	6,497.2
Total Preproduction Capital	308.3
Total All-in LOM Operating Costs (see below)	2,332.1
Total Sustaining Capital (including Progressive Closure and Final Closure Costs – See Below)	467.1
Operating Margin Ratio (Operating Revenue / Operating Cost)	2.8
Royalties	129.9
Income Taxes	714.8
Pre-Tax Cumulative Cash Flow	3,259.8
After-Tax Cumulative Cash Flow	2,545.0
Detail of Expenditures	
Total Operating Costs	2,270.5
Total Other Costs (Corporate Overhead)	61.7
Total All-in LOM Operating Costs	2,332.1
Sustaining Capital Cost	374.5
Sustaining Capital Cost – Progressive Closure	80.0
Closure Capital Cost	12.7
Total Sustaining Capital (including Progressive Closure and Final Closure Costs)	467.1

Table 1-10: Financial Statistics

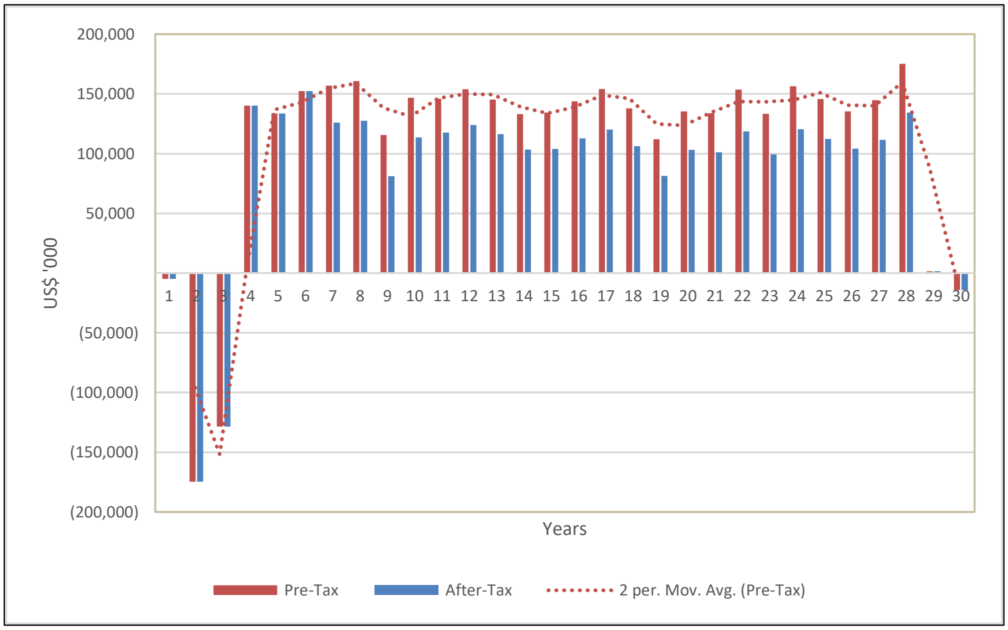
Description	Unit	After-Tax	Pre-tax
Cumulative Net Cash Flow			
Undiscounted (Base Year 2024)	US$k	2,544,960	3,259,753
Net Present Value			
Discounted at 5%	US$k	1,148,827	1,464,435
Discounted at 8%	US$k	749,268	954,843
Discounted at 10%	US$k	572,028	729,998
Discounted at 15%	US$k	300,575	387,968
Internal Rate of Return	%	34.9	37.8
Payback Period	Years	4.2	4.2

Table 1-11: Income Tax Holiday Impact on After-Tax Financial Statistics

Description	Unit	With 3 Years Tax Holiday	Without Tax Holiday
Net Present Value Discounted at 10%	US$k	572,028	526,660
Internal Rate of Return	%	34.9	31.5
Income Tax Payable	US$k	714,793	788,150
Payback Period	Years	4.2	4.2

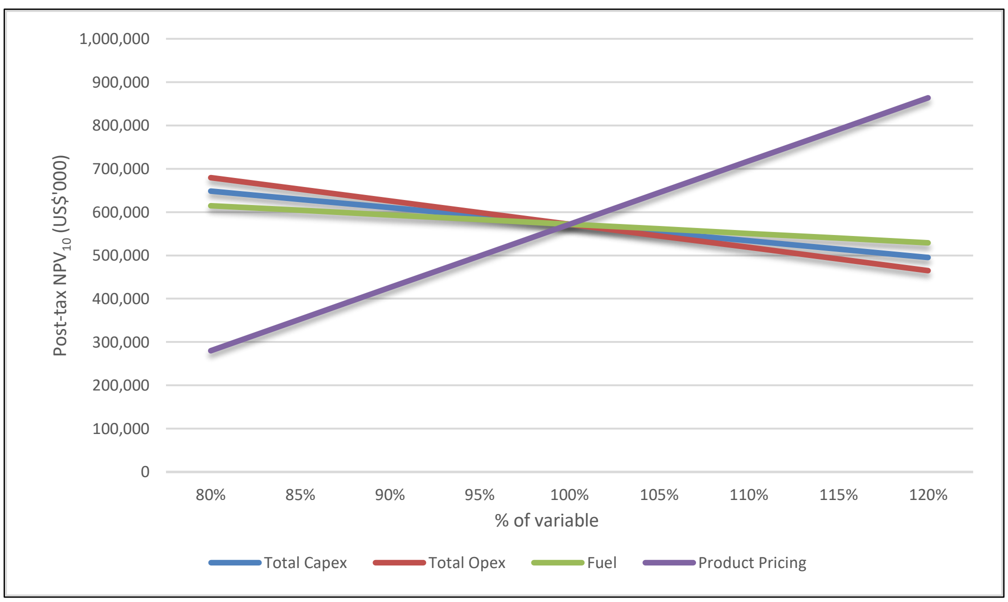
Annual cash flow is summarized in Figure 1-7. A sensitivity analysis was conducted on the post-tax NPV and IRR of the project using the following variables: revenue (P_2O_5 rock price), operating cost, total capital cost, and fuel. Post-tax sensitivity results are shown in Figure 1-8. The analysis revealed that the project is most sensitive to changes in P_2O_5 rock price.

Figure 1-7: Annual Cash Flow Profile



Source: Kristal Font, 2023

Figure 1-8: Post-Tax NPV10 Sensitivity Graph (US$k)



Source: Kristal Font, 2023

1.19 Project Execution Plan

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 will be engaged to provide EPCM services for the overall development of the project, including the process plant and the associated infrastructure. Specialist consultants will be contracted on an EPCM basis to address specific elements of the project outside the core competency of the engineering firm. These elements include mining, geotechnical, resettlement, environmental, marine construction of the Mineral Terminal, surface and sub-surface water management and the tailings storage facility (TSF). Specialist consultants will form an integrated project team under the overarching leadership of the engineer, who will be responsible for the overall project management and coordination between the various parties. Cost estimates assumed in the cash flow model are based on this approach.

The broader socio-economic context underpins the development of the project implementation plan, which is discussed in Section 24. The implementation plan focuses on the supply of construction power, transport and logistics services, project labor, consolidated construction packages, QA/QC management, prioritization of long-lead items, access control, security management and the resettlement program.

The overall schedule duration from the start of detailed engineering to the end of commissioning is 31 months. The ramp-up period will commence in month 32, and the date of first commercial production is expected to be achieved within 6 months following commissioning. Project schedules will be updated during the detailed engineering stage to mitigate risks associated with critical path activities. Specific areas that will be managed include the following:

- Detailed engineering associated with the pre-mining dewatering, the installation and equipping of the pre-mining dewatering wells, pumping the pre-mine wells for at least six months prior to the commencement of pre-stripping and completing the pre-stripping before the ramp-up period of the plant.
- Design, procurement, fabrication, delivery, and commissioning of construction power, which needs to be available for dewatering pumping to start.
- Updating the RAP and biodiversity management plan, followed by detailed engineering associated with the RAP village, the mining bypass road, the fence, and the bulk earthworks for the plant. Phase 1 and Phase 2 of the relocation, as well as the construction of the bypass road and fence must be complete prior to the commencement of infrastructure construction by the mining contractor.
- Detailed engineering, procurement, fabrication, delivery, installation and commissioning of long-lead items, including the concentrate dryer, horizontal scrubber and vertical plate-and-frame filters.

The major project milestones are summarized in Table 1-12.

Table 1-12: Major Project Milestones

Major Milestone	Month
Start of Detailed Engineering for the Plant (Section 17), Concentrate Handling & Drying (Section 17) and Project Infrastructure (Sections 18.7, 18.8, 18.11)	1
Start Update to Biodiversity Management Plan and the Land and Asset Survey of the Relocation Action Plan (RAP) (see Section 20.2)	2
Start detailed engineering for Mining (Section 16)	3
Issue Procurement Packages for Long-Lead Sections, including Power Generation (Section 18.9), Concentrate Dryer (Section 17.4.1), Vertical Plate-and-Frame Filters (Section 17.3.3) and Horizontal Scrubber (Section 17.3.2)	4
Award Contract for Construction of Pre-Mining Pit Dewatering System (Section 16.4.3)	6
Award Combined Contract for PLANT BULK EARTHWORKS, NORTH PIT BYPASS ROAD (Section 18.2), Mine Fence (Section 18.2), Ponta Chugue Access Road (18.3.1) and Buredanfa Resettlement Village Construction (Section 20.2)	7
Issue Year 0 to Year 5 Contractor Mining RFQ (see Section 21.1.1.1) into the Market	7
Start Drilling of Pre-mine Dewatering Boreholes (Pre-mining Pit Dewatering System)	8
Start Detailed Engineering for Marine Terminal (Section 18.13)	8
Publish Updated Biodiversity Management Plan and Resettlement Action Plan (RAP)	9
Start Construction, including Plant Bulk Earthworks, North Pit Bypass Road, Fence around Mining Area, Ponta Chugue Access Road and Buredanfa Resettlement Village	10
Issue Procurement Packages for Marine Works Package (Section 18.13.4) and Shiploader Long-Lead Section (Section 18.13.4.11)	10
Award Year 0 to Year 5 Contractor Mining Contract	13
Commissioning Complete of Phase 1 (Construction Power) of Diesel-Hybrid Power Generation Systems	13
Complete Construction of Pre-mining Pit Dewatering System, incl. Piping and Overhead Line	13
Commence Pre-mine Pit Dewatering (Six Months Prior to Pre-stripping)	14
Complete Phase 1 Relocation (Saliquenhe Porto & Ponto Zeca), as per RAP	14
Start Process Plant, Outloading and Drying Concrete Works	15
Complete North Pit Bypass Road	16
Complete Mine Fence and Establish Access Control to Construction Site	16
Complete Phase 2 Relocation (Canico) as per RAP	16
Complete Year 0 to Year 5 Contractor Mining Contract Mobilization and Site Establishment	16
Mining Contractor to Start Construction of Infrastructure, including Flood Protection Bund, BD1, TSF1, WD1, ECD and the SCD1	17
Start Process Plant, Outloading and Drying SMPP Installation	17
Start Piling and Other Marine Works Construction	20
Start Mining Pre-strip	20
Start Process Plant, Outloading and Drying E&I INSTALLATION	21
Start Commissioning of Process Plant, Outloading and Drying	29
Start Commissioning of Mineral Terminal	31
Commissioning Complete	31
Start of Ramp-up Period	32

1.20 Interpretation and Conclusions

Key outcomes from the feasibility study include:

- The data provided through various exploration and sampling programs, combined with a detailed processing analysis, infrastructure, and cost analysis, is sufficient to support the feasibility study and associated mineral reserves. A global mineral resource estimate defines a measured resource of 102.5 Mt at a mean grade of 28.5% P_2O_5 and an inferred resource of 31.1 Mt at a mean grade of 28.0% P_2O_5.
- Mineral reserves outlined in the study are based on a targeted mine life of 25 years at a rate of 1.75 Mt/a (dry basis) for a total of 43.75 Mt at a life-of-mine mean grade of 30% P_2O_5. Mining will be accomplished using conventional loader, excavator and truck materials handling with an average strip ratio of 10 bcm/t of ROM phosphate matrix.
- The mining agreement from 2009 is valid and renewable for 25-year extensions based on the 2000 Mining Law. Based on the work to date, there have been no environmental or social issues that are expected to prevent Itafos from developing the project. Itafos was issued a Declaration of Environmental Compliance from the Competent Environmental Assessment Authority of the Government of Guinea-Bissau in 2018 for the mine ESIA and the Buredanfa resettlement village ESIA. Because biodiversity and social conditions may have changed since the 2015 ESIA and 2018 Resettlement Action Plan were completed, additional work in these areas is recommended ahead of construction, as described in Section 26. The application process to renew the mining agreement term of 25-years is in progress and can be done up to 1-year prior to its expiration date.
- The process developed for the beneficiation of Farim phosphate ore is robust, continuous, and reliable, rendering reproducible metallurgical results. The flowsheet is based upon unit operations that are proven in industry. Continuous pilot plant tests indicate most likely results of yield (mass recovery) of 77.5%, P_2O_5 recovery of 81.8%, and likely P_2O_5 grade of 33.6% for the South pit. The phosphate rock produced is a high-grade, high-quality product that will attract a premium price.
- Filtered concentrate will be hauled to the Mineral Terminal at Ponta Chugue where it will be dried, loaded onto ships and sent to international buyers. The channel design has been assessed against PIANC channel design guidelines and with desktop and real-time navigation simulations. The channel alignment, including through the Bernafel section, is suitable for the water depths, design depths and prevailing currents. The navigation fairway surrounding the Ponte Chugue Marine Terminal is suitable and provides a generous maneuvering area for inbound and departing vessels.
- Initial capital costs are $308 million and sustaining capital cost (including progressive and final rehabilitation and closure costs) for the 25-year life of mine is $467 million for a total capital cost of $775 million.
- Life-of-mine project revenue is $6,497 million with all-in operating costs at $2,332 million, royalties of $130 million, and taxes of $715 million. The net result is an after-tax cumulative cash flow of $2,545 million (undiscounted).
- Project economics indicate an NPV_{10} of $572 million with an internal rate of return of 34.9% and payback of 4.2 years. The project is most sensitive to phosphate rock sales price.
- Project infrastructure can be constructed as part of the Mining Agreement with key elements including waste rock dumps, tailings storage facilities, Marine Terminal, filtered concentrate loadout, and roads to support access and haulage.
- Mining risks include dewatering ahead of pit excavation to reduce pore pressures in the formation and overall water management during the rainy season. Hydrogeology and groundwater findings are summarized in Section 18.

Opportunities exist to de-risk the project or improve economics, which will be investigated further during the detailed design stage. This includes connecting to the planned Guinea-Bissau electrical grid and the option to trans-ship dried concentrate using barges to offshore ships.

A complete list of project risks and opportunities are described in Section 25.

1.21 Recommendations

1.21.1 Overall

The financial analysis of this feasibility study demonstrates that the Farim Project has robust economics, and it is recommended to continue developing the project through detailed engineering and de-risking, to support a construction decision.

Conclusions from each major area of investigation completed as part of this feasibility study suggest numerous recommendations for further investigations to mitigate risks and/or improve the base case designs. Those details are described in Chapter 26. Costs to address future recommendations are captured in the capital or operating costs.

The following is a summary of recommended work for the next phase based on the feasibility study. Associated costs listed in Table 1.13 are excluded from capital or operating costs summarized in this report. Each recommendation is not contingent to a subsequent one.

Table 1-13: Recommended Work and Budget for Next Phase

Area	Estimated Cost (US$)
Mineral Resources and Mineral Reserves	55,000
Mineral Processing & Metallurgical Testing	150,000
Recovery Methods – Tailings Thickening Testing	10,000
Marine – Evaluation of Transshipping Option	50,000
Tailings Characterization and Settling Testwork	35,000
Updated South Pit Ground Investigation for Pit Dewatering (includes Drilling, CPT Program, Vibrating Wire Piezometers Supply and Install, and Supervision)	625,000
Ongoing community engagement including renewal of its Declaration of Environmental Compliance from the Competent Environmental Assessment Authority to ensure continuity of project approvals	N/A
Total	925,000

1.21.2 Mineral Resources and Mineral Reserves

Work programs related to resources and reserves for the next phase include the following:

- Confirm that the dry density values used in Section 14 and Section 15.6, are representative for future resource and reserve estimations. Additional density measurements should be taken to verify these values. The QP anticipates that this would cost approximately US$5,000.
- As noted in Section 16.7.4.6.1, a lack of samples in Area 4 of the pit (as designated by Figure 16-7) has prevented a thorough evaluation of the liquefaction susceptibility in this Area. Samples in Area 4 should be collected and screened prior to excavation to evaluate the soil's liquefaction susceptibility. The QP anticipates that this would cost approximately US$20,000.

- As stated in Section 16.7.5.2, an important component of the slope development will be to monitor the degree of pore pressure reduction that has been achieved in the bench face that is being excavated. This can be achieved by installation of piezometers or pushed probes with pressure transducers into critical areas along the pit slopes. Supplemental pumping wells or horizontal drains will be needed where isolated pressurized zones are encountered. Further studies should be done to advise the precise locations of these piezometers for optimized performance. The QP anticipates that this would cost approximately US$30,000.

Additional recommendations to de-risk the project are outlined in Section 26.2.

1.21.3 Metallurgical Testwork and Mineral Processing

metallurgical testwork and mineral processing work programs for the next phase include the following:

- Conduct continuous phosphoric acid plant tests to assess likely performance in an industrial plant. Conduct bench-scale phosphoric acid concentration and clarification tests, and bench scale fertilizer testwork for both the South pit and North pit concentrates of the Farim phosphate deposit. The QP anticipates that this would cost approximately US$150,000. Results from this testwork will be used in product off-take negotiations and is independent of the investment decision therefore does not have to be complete prior to detailed design stage.

Several other recommendations to de-risk the project relating to mineral processing and metallurgical testwork are detailed in Section 26.3.

1.21.4 Recovery Methods

The following work is recommended to advance to the next stage:

- Further evaluate tailings thickening and dewatering to maximize achievable underflow density and optimize thickener sizing. The QP anticipates that this would cost approximately US$10,000.

Further recommendations to de-risk the project or enhance project economics are listed in 26.5.

1.21.5 Site Infrastructure

The feasibility study has outlined infrastructure recommendations to de-risk the project or enhance project economics. These recommendations will be addressed during the detailed design stage or during operations and are described in Section 26 in their main areas including tailings storage facility, dewatering, and geotechnical. See those sections for detailed future recommendations.

To advance to the next stage, the following tailing storage facility work program is recommended.

- Complete additional tailings characterization and settling testwork to improve TSF design including tailings settled dry density and tailings entrainment among other design parameters. The QP estimates this work will cost $35,000.

To improve the understanding of the dewatering needs and the potential hydrogeological impacts the following is recommended:

- Additional closer spaced drilling and testing of boreholes (including CPT survey) to determine the depth to bedrock, continuity of clay and sandstone lenses with installation of more vibrating wire piezometers (VWP) to monitor pressure heads in different units, particularly in the vicinity of the pit walls closest to planned infrastructure (TSF, overburden dumps). Updated South pit ground investigation for pit dewatering (includes Drilling, CPT Program, Vibrating Wire Piezometers Supply and Install, and Supervision). The QP estimates this work will cost $625,000 and must be completed prior to detailed design.

Additional dewatering recommendations to de-risk the project is outlined in Section 26.5.2.

1.21.6 Marine

The marine field data acquisition activities and engineering studies have largely concluded. To advance to the next stage, the following marine work program is recommended.

- Update the transshipping trade-off study to evaluate barge loading rather than hauling filtered concentrate to the planned Mineral Terminal at Ponte Chugue. This includes updating the costs from the previously performed work, re-evaluating barge, vessel requirements and throughput, updating the social impacts, and overall project benefits. This trade-off update is estimated to cost US$50,000.

Recommendations for future work are outlined in Section 26.6.

1.21.7 Environmental Studies and Permitting

The project has been de-risked from an environmental perspective to the extent that an environmental baseline has already been established. Nonetheless, ongoing community engagement should continue through the normal care and maintenance activities currently happening on site, including seeking the renewal of the Declaration of Environmental Compliance from the Competent Environmental Assessment Authority. Costs associated with this ongoing work are part of the Itafos project development budget.

For typical mining projects of this nature, the next phase would be used to develop an environmental baseline and a Resettlement Action Plan. However, as discussed in Section 24, the update of the Resettlement Action Plan (RAP) is linked to the overall project development schedule. Although the RAP work could begin early, it is linked to the investment decision to avoid community tensions and fatigue around the resettlement process, as well as avoiding the expiration of the RAP and asset data before the resettlement process can be implemented.

An integral part of updating the land and asset survey and the RAP is the restart of the stakeholder engagement process to get the affected communities back up to speed on the RAP process, while attempting to alleviate concerns related to project delays. KP has assumed that this initial environmental work program includes tasks that should be done ahead of project implementation (i.e., during detailed engineering).

Details regarding these work programs are described in Section 26.8 and the costs are summarized in Table 26.4. All costs are included in the pre-production capital cost estimate. These are tasks that are recommended for completion ahead of construction.

2 INTRODUCTION

2.1 Introduction

This technical report was prepared for Itafos Inc. (Itafos), a vertically integrated phosphate fertilizers and specialty products company registered in Delaware, that is focused on developing the Farim Phosphate Project. Itafos, through a series of wholly owned subsidiaries, owns GB Minerals AG (GBMAG), which is registered in Switzerland. GBMAG holds mining lease No. 004/2009 which was granted by the Government of Guinea-Bissau where the Farim Phosphate Project is located.

All measurement units used in this report are metric unless otherwise noted. Currency is expressed in United States dollars (currency: USD; symbol: US$). The report uses American English.

Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019; the 2019 CIM Best Practice Guidelines).

As ownership has changed hands numerous times during the exploration history (refer to Section 6), the report uses the term "previous operator" to refer to work done prior to 2006. The term "legacy" is used for data generated by the previous operator.

Up to January 2020, Itafos Farim has used the same contractors that were involved with the project during the feasibility study. This approach de-risked the project and saved money as there was not a requirement for knowledge transfer. The contractors were as follows:

- Lycopodium Canada (Toronto office, work began in May 2019) – main EPCM effort
- WF Baird and Associates (Madison office, work began in September 2018) – Mineral Terminal
- Golder (work on-going since FS) – mine design and in pit dewatering, RFP / evaluation of mine contractor and assistance with mining contract terms and conditions
- Knight Piésold (North Bay office, work ongoing since FS) – ex-pit geotechnology, hydrogeology, tailings design and QC/QA of earthworks
- Halyard (Toronto office, work began in 2018) – resettlement village and 150-person worker's camp
- ERM – architecture for resettlement.

Lycopodium advanced the detailed engineering (approximately 25% complete) and received firm pricing on the main process equipment, which is ready for purchase. The process design criteria and process flow diagrams have been finalized. A 3D model review of plant and Mineral Terminal facilities was completed. A third party, independent reviewer has been chosen.

On January 24, 2020, Itafos provided an update on the engineering and construction of Itafos Farim and announced the termination of the EPCM agreement with Lycopodium. The EPCM agreement with Lycopodium contemplated two phases: phase one considered preparation of a definitive cost estimate and schedule and phase two considered additional detailed engineering, equipment procurement and construction. Given the revised project financing timeline of expected lender approval during the H2 2020, Itafos Farim suspended the EPCM agreement with Lycopodium following completion of phase one. Since 2020 and through the COVID-19 related restrictions, the Farim camp facilities and mine site have remained on care and maintenance. In addition, further de-risking studies have continued. In 2022, Itafos commissioned Ausenco to update the feasibility study which incorporates all relevant studies completed since 2015, as well as updated market pricing and project costs.

2.2 Qualified Persons

The individuals listed in Table 2-1 serve as QP for this technical report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in accordance with Form 43-101F1.

2.3 Site Visits and Scope of Personal Inspection

The following QPs have visited the Farim property:

- Tommaso Roberto Raponi of Ausenco is the qualified person for the infrastructure, capital and operating cost compilation, and the process design for the study. He has not visited the project site.
- Jerry DeWolfe of WSP 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. During the site visit, Jerry reviewed the site layout including planned locations for the pits, ex-pit structures, stockpile locations, the test pit, and the proposed Mineral Terminal site. The visit also verified the location of selected drill hole collars via handheld GPS, viewed the core logging and storage facility and reviewed logging and sampling protocol with the project team.
- Terry L. Kremmel of WSP Golder completed the mineral reserve estimation and is responsible for the mining contribution. Terry visited the site in 2012. Site visit activities included reviewing core, reviewing overall site conditions, and potential locations of the mine pit and infrastructure.
- Alexander Duggan of Kristal Font completed the compilation of the capital cost estimate and the financial model for the study. He has not visited the project site.
- Edward Adam 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.
- Richard Michael Elmer of Knight Piésold Limited completed sections related to pit dewatering and project infrastructure, including waste management (stockpiles, tailings storage facility, water storage facility) and surface water management. Richard visited the Farim project site multiple times in 2010 - 2011 for one- to two-week visits to undertake site walkovers and drilling supervision. Richard has not visited the Farim Phosphate Project while employed by Knight Piésold.
- Richard Alonzo Cook of Knight-Piésold completed the sections related to environmental studies, permitting and social impact. Richard visited the site for two days on March 25 and 26, 2015. This included visiting the Farim townsite, the mine area, several of the villages that lie inside the proposed mine area, the product transport route, and the port area.

Table 2-1: Qualified Persons and Section Responsibilities

Qualified Person	Professional Designation	Position	Employer	Independent of Itafos	Report Section
Tommaso Roberto Raponi	P.Eng. (ON)	Senior Mineral Processing Specialist	Ausenco Engineering Canada Inc.	Yes	1.1 to 1.4, 1.12, 1.13, 1.15, 1.16, 1.17, 1.19, 1.20, 1.21.1, 1.21.4, 1.21.5, 2.1, 2.2, 2.4, 2.5, 3.1, 3.2, 4, 5, 17, 18.1 to 18.4, 18.6, 18.10, 18.11, 19, 21.1.2, 21.1.3.2, 21.1.3.3, 21.2.1, 21.2.3, 21.2.6, 21.3, 23, 24, 25.4, 25.6, 26.1 26.4, and 27
Jerry DeWolfe	P.Geo.	Geology & Mineral Resource QP	WSP Golder	Yes	1.5, 1.6, 1.7, 1.9, 1.20, 1.21.2, 2.3, 6 to 12, 14, 25.3.1.1, and 26.2
Terry L. Kremmel	P.E.	Mineral Reserve QP	WSP Golder	Yes	1.10 to 1.11, 1.20, 1.21.2, 2.3 15, 16.1, 16.2, 16.5 to 16.9, 21.1.1, 21.2.2, 25.3.1.2, and 25.3.2, 26.2
Alexander Duggan	P.Eng.	Director	Kristal Font	Yes	1.18, 1.20, 3.2, 3.3, 22, and 25.10
Edward Adam Liegel	P.E.	Senior Marine Engineer	Baird	Yes	1.20, 1.21.6, 18.13, 21.1.4, 21.2.4, 25.7, and 26.5
Richard Michael Elmer	C.Eng. MIMMM MCSM	Principal Geotechnical Engineer and Director	Knight Piésold Ltd.	Yes	1.20, 16.3, 16.4, 18.5, 18.7 to 18.9, 18.12, 18.13, 21.1.3.1, 21.2.5, 25.1, 25.5, 25.8, and 26.6
Richard Alonzo Cook	P.Geo.	Specialist Environmental Scientist / Associate	Knight Piésold Ltd.	Yes	1.14, 1.20, 1.21.7, 2.3, 20, 25.9, and 26.7
Francisco J. Sotillo	P.E.., PhD	Senior Metallurgist	KEMWorks Technology	Yes	1.8, 1.21.3, 13, 25.2, and 26.3

2.4 Effective Dates

The overall effective date of this report is the date of the press release, which is May 17, 2023.

2.5 Source Information

The authors of this report have assumed and relied on the fact that all the information and technical documents listed in Section 27, References, are accurate and complete in all material aspects.

2.6 Abbreviations and Units of Measure

Tables 2-2 and 2-3 define the abbreviations and units of measure used in this report.

Table 2-2: Report Abbreviations

Abbreviation	Meaning
ALS	ALS Metallurgy
amsl	Above Mean Sea Level
CC	Contact Clean Water
CD	Contact Dirty Water
CDA	Canadian Dam Association
CW	Clean Water
DWT	Deadweight Tonnage
ECD	Environmental Control Dam
EDF	Environmental Design Flood
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Plan
EPI	Employer Procured Items
GISTM	Global Industry Standard on Tailings Management
HDPE	High Density Polyethylene
IBC	International Building Code
IDF	Inflow Design Flood
ISO	International Organization for Standardization
ISPS	International Ship and Port Facility Security
KEMWorks	KEMWorks Technology Inc.
LBP	Length Between Perpendiculars
LOA	Length Over All
MAC	Mining Association of Canada
MDE	Maximum Design Earthquake
MER	Minor Element Ratio = [Fe2O3% + Al2O3% + MgO%] / P_2O_5%
MRCP	Mine Reclamation and Closure Plan
MWC	Marine Works Contractor
NAG	Non-Acid Generating
OBE	Operational Basis Earthquake
P	Process Water
PAG	Potentially Acid Generating
PGA	Peak Ground Acceleration
RAP	Resettlement Action Plan
RWP	Return water pond
SCD	Sediment Control Dam
SGS	SGS Mineral Services
TSF	Tailings Storage Facility
UW	Undisturbed Water
WD-1	Waste Dump 1
WD-2	Waste Dump 2

Table 2-3: Units of Measure

Units of Measure	Description
3D	Three-Dimensional
°C	degrees Celsius
C$	Canadian dollars
US$	United States dollars
cm	centimeter
%	percent
%w/w	dry weight concentration of a solution
µ	micro
µm	micrometer
g	gram
g/cm^3	Grams per centimeter cubed
g/t	grams per tonne
ha	hectare
hp	horsepower
h	hour
kg	kilogram
km	kilometer
koz	thousand ounces
kt/d	thousand tonnes per day
kV	kilovolt
kWh	Kilowatt hour
L/s	liter per second
M	million
m	meter
m^2	square meter
m^3	cubic meter
masl	meters above sea level
mamsl	meters above mean sea level
mg/L	milligrams per liter
mm	millimeters
Mt	million tonnes
Mt/a	million tonnes per annum
mV/V	millivolts per volt
MW	Megawatt
MWh	Megawatt hour
oz	ounce
P_{80}	Passing grind size
ppm	parts per million
ppb	parts per billion
t	metric tonne
t/d	tonnes per day
t/m^2/h	tonnes per meter squared per hour
X	times

3 RELIANCE ON OTHER EXPERTS

While the authors have carefully reviewed, within the scope of their technical expertise, all the available information presented to them, they cannot guarantee its accuracy and completeness. The authors reserve the right, but will not be obligated to, revise the technical report and its conclusions if additional information becomes known to them after the effective date of this report.

3.1 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Itafos and legal experts retained by Itafos for this information through the following documents: ·

- Ministerio Dos Recursos Naturais E Energia, Government of Guinea Bissau, May 30, 2022. Confirmation of Payment, Prepared for the Director of Enterprise at Itafos Farim, SARL, 2 pages.
- Ministerio Dos Recursos Naturais E Energia, Government of Guinea Bissau, May 28, 2009. Licenca de Arrendamento de Mineraco, Nº 004/2009. Prepared for GB Minerals. AG. 2 pages.

This information is used in Section 4 of the report.

3.2 Market Studies and Contracts

The QPs have not independently reviewed the marketing or pricing information. The QPs have fully relied upon, and disclaim responsibility for, information derived from Itafos and experts retained by Itafos for this information through the following documents:

- Michael R Rahm Consulting LLC, 2023. Medium-Term and Long-Term Phosphate Outlook, January 2023.

Michael R Rahm has a Master's and PHD in Economics from Iowa State University from 1978 and 1980, respectively. He has worked in the fertilizer market for 38 years. This information is used in Section 19 of the report. The information is also used to support the financial analysis in Section 22.

3.3 Taxes

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Itafos, for this information through the following documents:

- Draft Third Addendum to the Mining Agreement between the Republic of Guinea-Bissau and GB Minerals AG on the Execution of the Mining License for the Farim Phosphate Project, with an effective date of 28 May 2009.

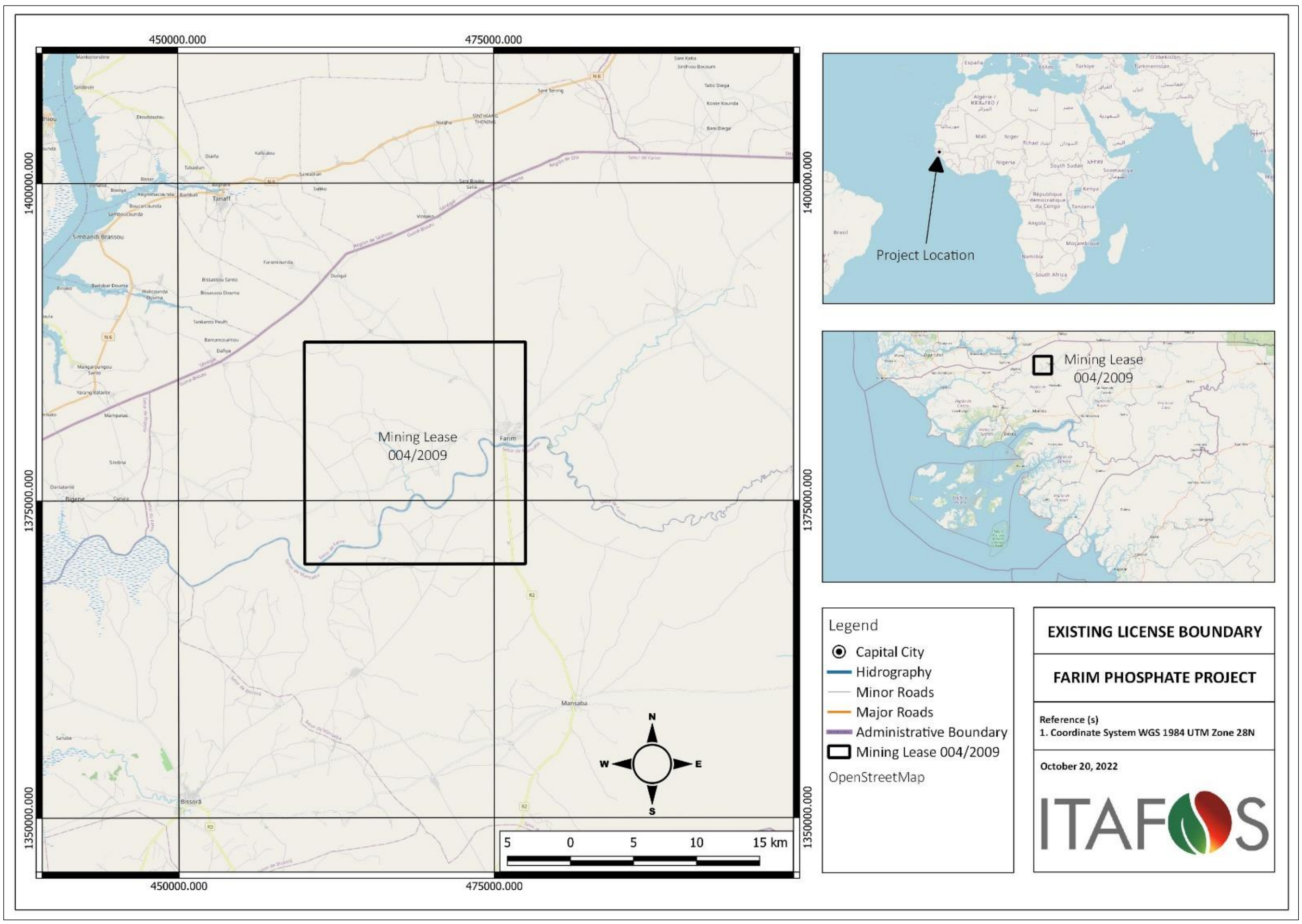
This information is used in Section 1.2 and 4 of the report. The information is also used in support of the tax information in Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Farim Phosphate Project is located in the northern part of central Guinea-Bissau, West Africa, approximately 25 kilometers (km) south of the Senegal border, approximately 5 km west of the town of Farim, and 120 km northeast of Bissau, the capital of Guinea-Bissau (Figure 4-1).

Figure 4-1: Location of the Farim Phosphate Project



Source: Itafos, 2022

4.2 Ownership, Title, Licensing and Permitting

The Farim Phosphate Project lies within Mining Lease License No. 004/2009, which covers 30,625 hectares (ha), granted by the Government of Guinea-Bissau on May 28, 2009, to GB Minerals AG (GBMAG). GBMAG is registered in Switzerland and is a wholly owned subsidiary of Itafos Farim Holdings, which is registered in the Cayman Islands.

Table 4-1 lists the coordinates of Mining Lease 004/2009 ("License Area"). The License Area is shown on Figure 4-1. A map of the site location can be found in Figure 6-1.

Table 4-1: Border Limits of Mining Lease License 004/2009 (Bissau UTM, Zone 28N)

Corner Post Identification	Northing	Easting
#14 Binta	1,387,500	460,000
#15 Farim	1,387,500	477,500
#1 Guidaje	1,370,000	477,500
@3 Jumbembem	1,370,000	460,000

The Mining Agreement is considered the global agreement aggregating and coordinating the above licenses and any other agreements or conditions relative to the Project. The Mining Agreement is valid for an initial period of 25 years. Based on the 2000 Mining Law, this period shall be extended by the Competent Authority for successive periods of 25 years, provided the Licensee so requests in the form and with the deadlines established under articles 109 to 112 of the Law no. 1/2000, of 24th July 2000. The renewal of the Mining License and of the Mining Lease shall imply the automatic and correspondent renewal of the Mining Agreement. The Mining Agreement provides the Company the right to construct and develop a mine to exploit the Farim phosphate deposit, and to construct and operate a Mineral Terminal facility and any bridges, roads, or transportation pipeline infrastructure required to connect the mine to the Mineral Terminal Site. The Government commits within the agreement to make immediately available the lands required for the Mineral Terminal infrastructure at the Ponta Chugue area.

In turn, the Mining Agreement requires the Company to:

- exploit the resource as per good international industry practices and in accordance with a mining operation program
- 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.

In addition to the obligations outlined above, the company must notify in appropriate terms of any circumstances that it becomes aware of and that may affect, impede, or make the execution of the aforementioned programs/plans excessively difficult, and must provide a written and detailed report of these situations and indicate the measures to be taken or implemented to minimize the effects of these situations.

Post-production, there are also obligations of disclosure and due notice to the suspension of production for any reason.

Finally, the company is obligated to disclose any information that is reasonably requested by the government.

4.3 Incentives Annex, Royalties and Other Financial Agreements

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

The Mining Agreement comprises an "Incentives Annex" that defines the financial terms associated with the Farim Phosphate Project and provides the Company certain guarantees and financial incentives. The terms of the Incentives Annex to the Mining Agreement have been negotiated and, as of the date of this report, remain subject to the final approval of the Government of Guinea-Bissau.

4.4 Environmental Regulatory Framework

The Farim Phosphate Project's concession and Mineral Terminal areas consist of both undisturbed land and farmland. There are no known environmental liabilities associated with these areas.

4.5 Surface Rights

The Mining Agreement provides the Holder with the right to construct and develop a mine to exploit the Farim phosphate deposit, and to construct and operate a Mineral Terminal facility and any bridges, roads, and infrastructure required to connect the mine to the Ponta Chugue Mineral Terminal site. Within the agreement the government commits to make immediately available the lands required for Mineral Terminal infrastructure at the Ponta Chugue area.

Land is state-owned property in Guinea-Bissau but is administered at the local level by customary (traditional) authorities. Surface rights must be negotiated with local landowners in compliance with the Land Law (No. 5/98). If negotiations with a landowner fail, the company may appeal to the Minister of Natural Resources, who can order the parties to submit the matter to arbitration pursuant to Article 87 of Law No. 3/2014.

Under the terms of Paragraph 2 of Article 4 of the same law, the State may grant rights for the private use of land under the terms set out therein. Paragraph 3 of Article 4 of the Land Law states: "Said private use rights will be granted through: a) Customary use; or b) Concession."

Law No. 3/2014 (law on prospecting, research, exploration and marketing of mineral substances in the soil) states, "[t]he holder of any right that requires the exclusive use or other form of use, in whole or in part, of the land corresponding to the geographic limits of his license, may, in accordance with the laws relating to such acquisition, purchase, lease, or acquire the right to the land through a legal instrument, for their use, in accordance with the conditions that may be agreed between the holder of the mining rights and the holder of the exclusive land rights, or the competent authorities of the Republic of Guinea-Bissau." It further states, "[i]f the agreement referred to in the previous number is not possible, the holder may appeal to the Minister, who may order the parties to submit the matter to arbitration under the terms of article 87 of this law."

The land in the mine area where Itafos has its camp and office has already been acquired and is owned by Itafos Farim SARL. The company was in discussion with local landowners to obtain ownership of land for the mine area and the Mineral Terminal at Ponta Chugue before being halted due to the COVID-19 pandemic. Based on Itafos' experience acquiring the land for the camp, the time required to acquire land is relatively quick, and land valuations are relatively low.

4.6 Project Risks and Uncertainties

A Resettlement Action Plan was prepared for the Project in 2018. Conditions within the area being acquired for the mine, Mineral Terminal, or mine area resettlement village may have materially changed since this time due to the establishment of new households or the conversion of undeveloped land into agricultural land. This has the potential to increase the cost and complexity of resettlement.

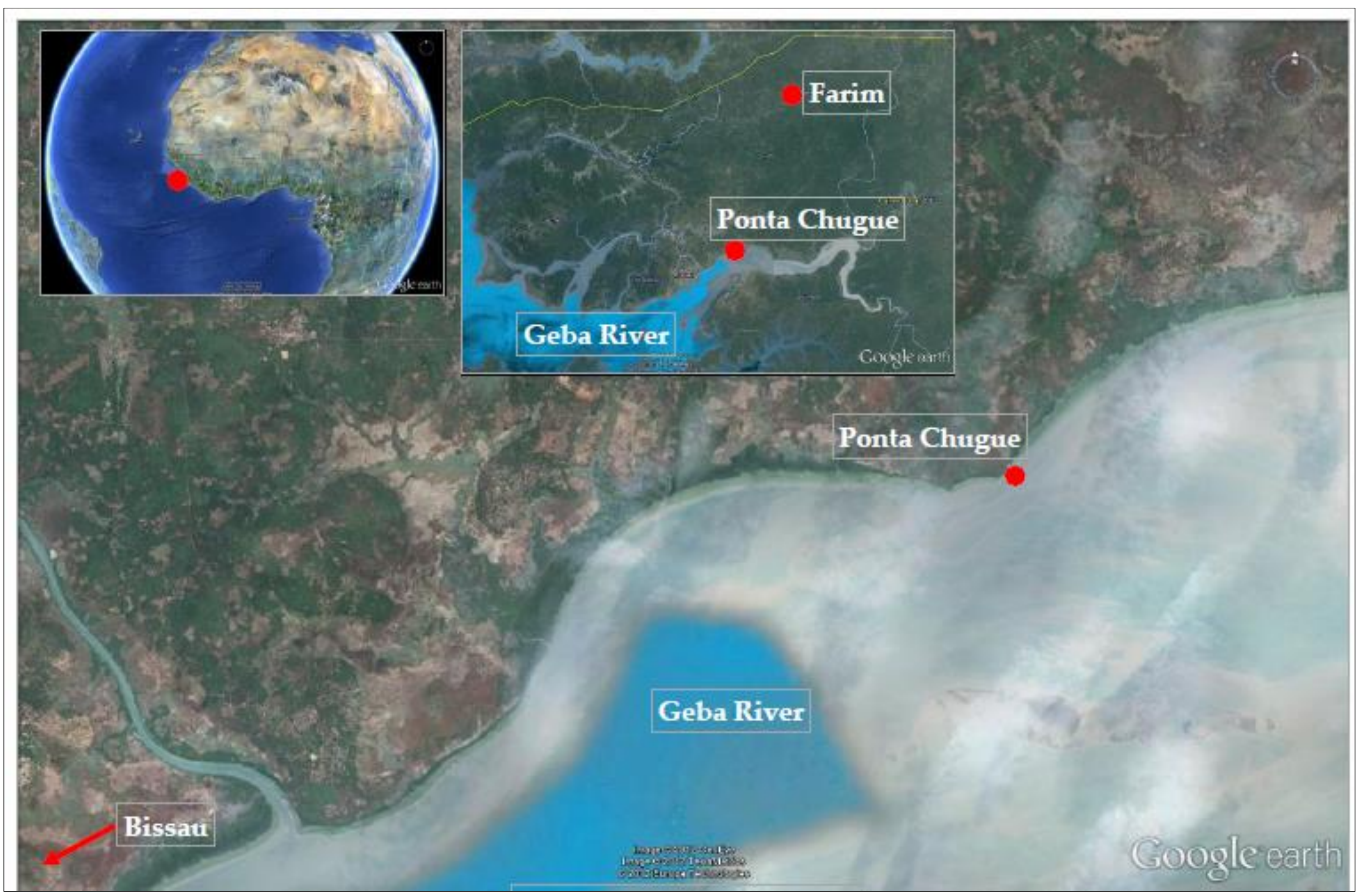
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

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, 5 km west of the town of Farim, and 120 km northeast of Bissau, the capital of Guinea-Bissau (Figure 5-1). 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 River Cacheu. The River Cacheu 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.

Ponta Chugue, which is the Mineral Terminal location in the Geba River estuary, is approximately 18 km east of Bissau and approximately 75 km south of Farim (Figure 5-1). Beneficiated phosphate rock will be trucked from Farim to Ponta Chugue via an existing highway which is in excellent condition. The phosphate rock will be dried, stored, and loaded directly onto 35,000 dead weight tonne (DWT) ships.

Figure 5-1: Farim and Ponta Chugue Locations



Source: Baird, 2019

5.2 Physiography

The project area is a flat, open, semi-arid savannah woodland, and the Farim mine site, where the phosphate is mined, varies from approximately 5 to 10 m above mean sea level. The project area is drained by several tributaries of the River Cacheu, which borders the south of the mine. The River Cacheu and the lower reaches of these tributaries are tidally influenced. The area where the process plant is located is approximately 4 to 5 m above mean sea level. See Figure 5-2 for a typical view of the plant site.

Figure 5-2: Process Plant Site at Farim

Source: WSP Golder 2015

A total of 341 plant species were recorded during various project surveys undertaken between 2011 and 2015 (Hudson Ecology, 2015). Floral species diversity in the area is moderate to high, but not as high as many regions of West Africa, such as the Upper Guinea Forest zone. A large proportion of the species recorded are indigenous with few exotic species occurring in the area although, in areas of higher anthropogenic disturbances, some exotic species are more prevalent.

Based on physiognomy, moisture regime, rockiness, slope and soil properties, seven main communities were recognized, namely:

- *Rhizophora-avicennia* mangrove community (mine and Mineral Terminal site study areas)
- Natural forest vegetation community (mine site study area only)

- Secondary forest community (mine site study area only)
- *Elias-cyperus* floodplain community (mine and Mineral Terminal site study areas)
- *Oryza* paddy vegetation community (mine and Mineral Terminal site study areas)
- *Dialium-sterculia* coastal woodland vegetation community (port site study area only)
- *Anadelphia afzeliana* seasonally wet grassland community.

The *Oryza* Paddy vegetation community occurs in areas of freshwater wetlands which are not affected by tidal ebbs and flows throughout the country, these areas of freshwater wetlands have been modified to facilitate the planting of rice, so alterations to the flow of freshwater systems create large, inundated areas where rice is planted. The only species found which is known to be listed by the IUCN Red Data list is *Raphia palma-pinus,* which is found along rivers and is listed as Data Deficient. The possible presence of two Red Data species *Floscopa axillaris* and *Digitaria patagiata* will be confirmed during follow up ecological surveys. A total of 103 flora species were recorded in this vegetation community.

Large sections of natural forests have been cleared to grow cashew nuts and other crops. This vegetation community encompasses areas that have been cleared and that have been replanted with cashew trees. The cashew plantations vary from areas which are dominated by cashew trees (cashew monoculture) to areas of mixed cashews and secondary forest. A total of 145 flora species were recorded in this vegetation community.

Another halophytic community recorded in the study area is the salt water *lala*, a grassland. This vegetation community is found on fluvisols in the floodplain areas adjacent to the larger rivers which are tidally influenced. The salinity of the water which floods these areas has resulted in the dominance of salt-tolerant species. The denuded areas of these areas are widely utilized by local communities for the gathering of salt during the dry season and this vegetation community appears to be an important dry season grazing area. A total of 76 flora species were recorded in this vegetation community.

The natural forest community occupies large areas of the northern part of the study area, with some variation in structure and composition. This vegetation community is currently under threat due mainly to slash and burn agricultural practices for the cultivation of food crops or cashew nuts. Although only one Red Data species was recorded in this vegetation community, the likelihood of occurrence of Red Data species in this community is high. A total of 209 flora species were recorded in this vegetation community.

Mangrove forests line all the larger rivers in the region. No Red Data species were recorded in this vegetation community and, due to the specialization required for plants to survive in the tidal saline conditions, it is unlikely that any of the Red Data species known to occur in the area occur in this vegetation community. Though species diversity is low, the species occurring are highly specialized, and therefore this vegetation community is characterized as unique. This vegetation community is integral in the functioning of the estuarine nature of the larger rivers in the area. A total of 29 highly specialized flora species were recorded in this vegetation community.

The *Dialium-sterculia* coastal woodland vegetation community occurs in the transition zone between the terrestrial and the halophytic communities in the coastal regions. Much of the substrata of the transitional zone in the vicinity of Ponta Chugue has been severely transformed due to cropping of mainly millet and peanuts.

The most extensive wetland vegetation in the country is a wet grass savannah that is locally called *lala*. This vegetation community is prevalent on gleysols, which are fine textured soils, deep, grey-colored, from alluvial origin, with the upper layers often rich in organic matter. Furthermore, this vegetation community prevails in the inner lowland plains flooded by rainwater during the wet season, located mostly in the lower zones of the mainland. Noticeable in the area of Ponta Chugue, is that this vegetation community is particularly homogenous. It is likely that this is the result of human intervention through the use of fire or harvesting of this grass species for thatching.

Figure 5-3: Conveyor Crossing Location at River Cacheu

Source: WSP Golder 2015

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

Source: WSP Golder, 2015

Figure 5-5: Bay at Ponta Chugue

Source: WSP Golder, 2015

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 a population of approximately 7,000 people. The area surrounding the Mineral Terminal site at Ponta Chugue is agricultural and sparsely populated—the nearest village is Chugue, with a population of approximately 100 people. The capital city, Bissau, has a population of 407,000 people, and is approximately 18 km from Ponta Chugue by partially paved road.

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

Water is available from wells. There is no local power supply for Farim or Ponta Chugue. Power for the project will be provided by hybrid-diesel and solar generating sets at both locations. All working areas of the project will be accessible by well-maintained, dual-lane gravel roads. The Government of Guinea-Bissau is currently planning and constructing an electrical grid infrastructure system which could be a cost saving opportunity in the future for Farim. However, the current Farim project design assumes that all power will be delivered through hybrid power generation.

The town of Farim has limited infrastructure for a mining operation the size of the proposed project. It will be necessary to construct housing, medical facilities and other infrastructure to accommodate project personnel and reduce the project's impact on the town of Farim. At the future Mineral Terminal, the staffing requirements are significantly smaller, and workers will most likely reside in the capital of Bissau or nearby villages around Ponta Chugue.

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°C. The minimum temperature recorded during the same period was 8.1°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 at the project area. Without proper surfacing, travel via road can be more difficult during the rainy season.

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

5.5 Regional Seismicity

A literature review has been conducted of seismicity in Guinea-Bissau and West Africa, and a probabilistic and deterministic seismic hazard assessment has been carried out for the project. Available information and historical data, including earthquake catalogues and technical publications on 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 = 0.15 g
- Seismic coefficient, S1 = 0.04 g
- Peak ground acceleration = 0.06 g.

6 HISTORY

6.1 Exploration 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. To assist with the BRGM drilling programs and mineral resource estimation, an exploration grid was implemented. A 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. A ground survey was completed in April 1983, and a general survey of Guinea-Bissau was carried out by the IGN (French National Geographical Institute).

This enabled BRGM 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 results of the prefeasibility study were positive, but market conditions and political considerations precluded development at that time and the French agencies withdrew from the project.

In 1997 a Canadian exploration company, Champion Resources Inc. (Champion), acquired ownership of the Farim phosphate deposit and carried out diamond drilling campaigns in 1998 and 1999 totaling 1,810 m in 34 holes. In May 2000 Champion filed an NI 43-101 technical report entitled “Farim Project – Resource Estimate, Mine Plan and Cost Estimate” prepared by MRDI Canada with an effective date of June 2000.

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 and 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)., 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 Resource Hunter Capital Corp. (RHC). On February 22, 2011, RHC filed an 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.

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 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 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. Furthermore, in 2013 GB Minerals Ltd. acquired full ownership of the Farim project by purchasing the remaining 49.9% interest in GB Mineral AG. In 2014 an Environmental and Social Impact Assessment (ESIA) was prepared for the mine component by Golder.

During this time GB Minerals Ltd. decided to change the project configuration and embarked on a new feasibility study for the project. The revised feasibility study, followed by a revised ESIA that corresponds to the revised project configuration, was submitted to the GoGB in 2015. In September 2015 GB Minerals Ltd. filed a NI 43-101 technical report for the feasibility study on the Farim Project entitled "NI43-101 Technical Report on the Farim Phosphate Project, Guinea-Bissau", dated effective September 14, 2015.

In 2018, Itafos Inc. acquired 100% of GB Minerals Ltd and commenced with detailed engineering and the construction of a Mining Camp. A Resettlement Action Plan (RAP) and ESIA for the proposed Buredanfa resettlement village were prepared and approved in 2018.

During 2019, the project development was idled while Itafos awaited the conclusion of the presidential elections, and subsequently the coronavirus pandemic. Since 2020, site activities included environmental monitoring, technical trade-off studies, and Mining Agreement negotiations with the GoGB which are still in progress.

Golder has completed an in-pit geotechnical investigation and recommendations in 2020. The report confirmed the previous geotechnical assumptions and did not recommend any changes to the mine plan:

- Golder Years 1 and 2 Pit Area Supplemental Site Investigation and Pit Stability Assessment, Project No. 1775736, April 2, 2020.

A summary of relevant historical mineral resource and mineral reserve estimates is presented in Figure 6-1.

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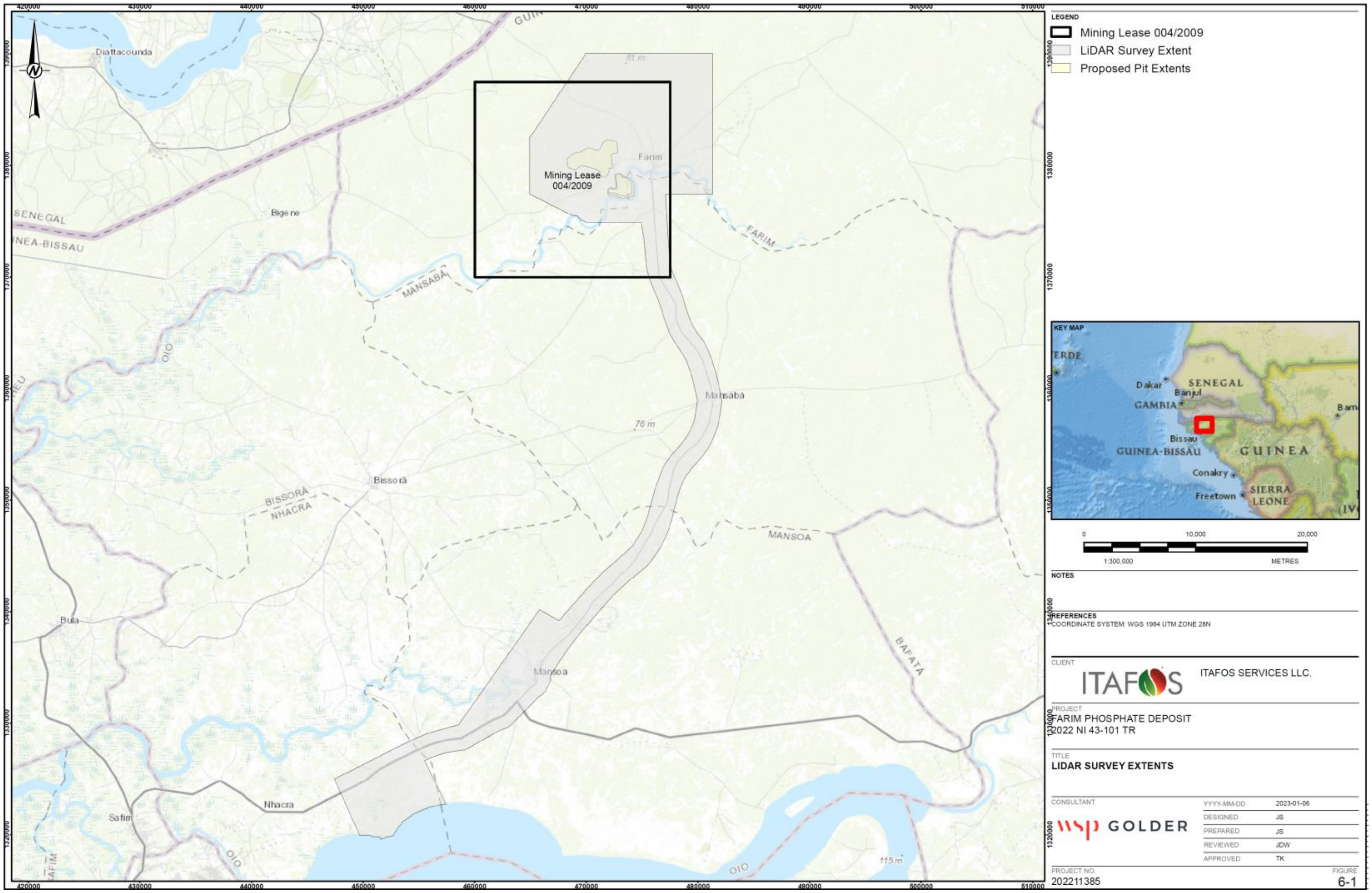
Figure 6-1: LiDAR Coverage Map



Source: WSP Golder, 2023

6.2 Historical Mineral Resource and Mineral Reserve Estimates

Several historical mineral resource and historical mineral reserve estimates have been prepared during the life of the project. Each historical mineral resource and mineral reserve estimate presented below are provided for information purposes only and should not be relied upon as the qualified person has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves; and the Company is not treating the historical estimates as current mineral resources or mineral reserves.

The earliest documented historical mineral resource estimate for the project was prepared by Sofremines in 1986 as part of the BRGM prefeasibility study The historical mineral resource was estimated using polygonal methods. No details are available on the key assumptions and parameters used in the estimate. The historical mineral resource estimate predates the definitions used for current mineral resource estimates and the qualified person is unable to confirm if there are any material differences in the definitions. The historical estimate is presented for information purposes only and as the historical estimate has been superseded by several more recent estimates, the qualified person has not assessed potential work required to upgrade or verify the historical estimate.

Between 2000 and 2015, a series of NI 43-101 mineral resource and mineral reserve estimates were prepared by various operators as listed below:

- May 2000 - Champion: NI 43-101 technical report entitled “Farim Project – Resource Estimate, Mine Plan and Cost Estimate” prepared by MRDI Canada with an effective date of June 2000.
- February 2011 - RHC: 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.
- September 2012 - GB Minerals Ltd.: 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.
- January 2013 - GB Minerals Ltd.: 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
- September 2015 - GB Minerals Ltd.: NI 43-101 technical report for the feasibility study on the Farim Project entitled “NI43-101 Technical Report on the Farim Phosphate Project, Guinea-Bissau”, dated effective September 14, 2015.

The May 2000 through September 2015 historical mineral resource and mineral reserve estimates were all prepared as NI 43-101 technical reports, and all appear to have been prepared in accordance with the relevant CIM definition standards and best practice guidelines for mineral resources and mineral reserves that would have been in place at the time of the disclosure. The mineral resource qualified person for this current technical report was also the qualified person for the mineral resource estimate disclosed in the September 2015 technical report.

The technical reports document the data used, modelling methods and other inputs and assumptions that support the historical estimates. The definitions of mineral resource and mineral reserve categories used in the historical estimates are consistent with the current CIM definitions used in this technical report. The May 2000 through September 2015 historical estimates are presented for information purposes only and as the historical estimates have been superseded by each other in turn and ultimately by the current mineral resource and mineral reserve estimates presented in this technical report, the qualified person has not assessed potential work required to upgrade or verify the historical estimates.

The results of the historical mineral resource and mineral reserve estimates are presented in Tables 6-1 and 6-2, respectively. As stated previously, each historical mineral resource and historical mineral reserve estimate presented in

Table 6-1 and Table 6-2 are provided for information purposes only and should not be relied upon as the qualified person has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves; and the Company is not treating the historical estimates as current mineral resources or mineral reserves.

Table 6-1: Summary of Historical Mineral Resource Estimates

Date	Owner	Disclosure Standard	Definition & Best Practice Standards	Historical Mineral Resource Categories							
				Measured		Indicated		Inferred		Non-Categorized	
				Tonnage (Mt)	P_2O_5 Grade (%)	Tonnage (Mt)	P_2O_5 Grade (%)	Tonnage (Mt)	P_2O_5 Grade (%)	Tonnage (Mt)	P_2O_5 Grade (%)
1986	BRGM	Unknown	Unknown	-	-	-	-	-	-	113	30.00
May 2000	Champion	NI 43-101	CIM	53.16	29.79	112.82	28.69	3.1	24.98	-	-
February 2011	RHC	NI 43-101	CIM	68.76	29.91	15.1	30.06	43.7	29.60	-	-
September 2012	GB Minerals	NI 43-101	CIM	64.6	29.11	28.1	27.68	18.3	28.66	-	-
January 2013	GB Minerals	NI 43-101	CIM	64.6	29.11	28.1	27.68	18.3	28.66	-	-
September 2015	GB Minerals	NI 43-101	CIM	105.6	28.41	-	-	37.6	27.74	-	-

Table 6-2: Summary of Historical Mineral Reserves

Date	Owner	Disclosure Standard	Definition & Best Practice Standards	Historical Mineral Reserve Categories			
				Proven		Probable	
				Tonnage (Mt)	P_2O_5 Grade (%)	Tonnage (Mt)	P_2O_5 Grade (%)
May 2000	Champion	NI 43-101	CIM	18.1	31.31	19.2	30.63
September 2012	GB Minerals	NI 43-101	CIM	29.5	30.40	3.5	29.60
January 2013	GB Minerals	NI 43-101	CIM	29.5	30.40	3.5	29.60
September 2015	GB Minerals	NI 43-101	CIM	44.0	30.00	-	-

6.3 Production History

The Farim project is in the development stage, with no historical production of phosphate or other mineral commodities for the property.

7 GEOLOGICAL SETTING AND MINERALIZATION

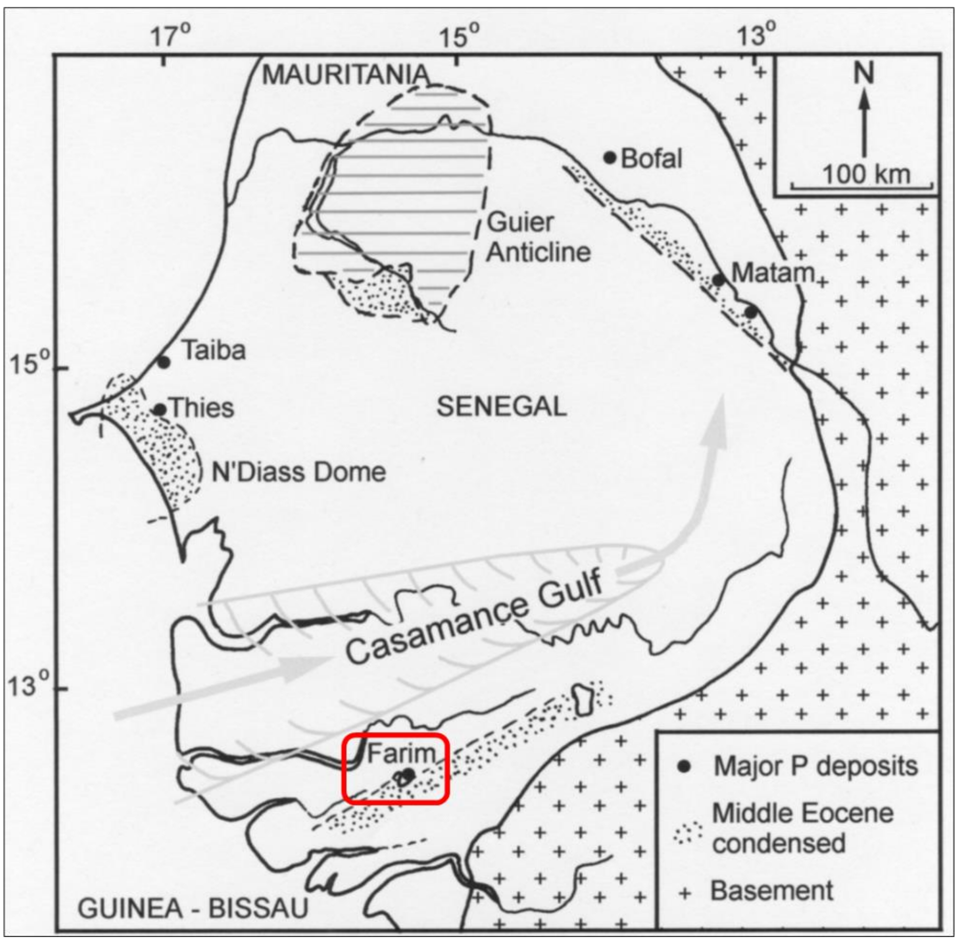
7.1 Regional Geology

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 several 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 paleo-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 in Figure 7-1.

Figure 7-1: Regional Geology and Setting of Farim

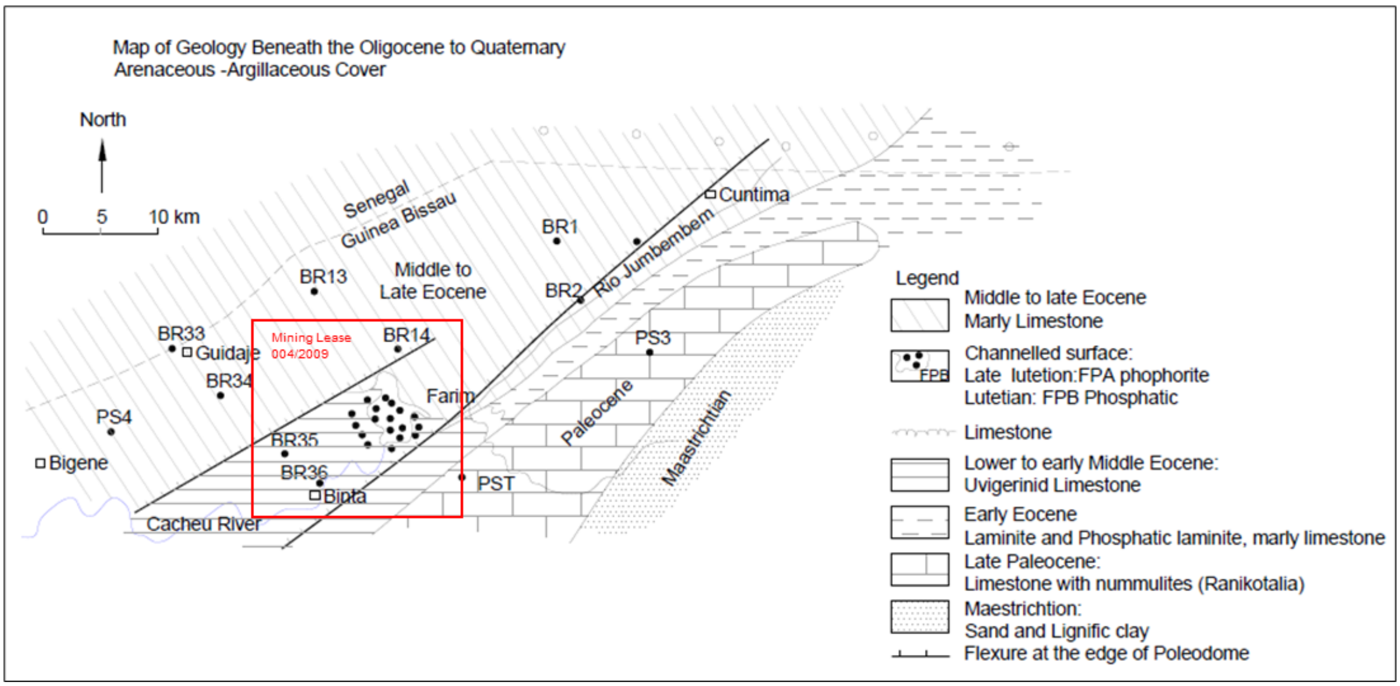


Note: Reproduced from Prian, 1987.

7.2 Local Geology

The local geology beneath the overburden is shown in Figure 7-2. 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 Paleocene and Eocene sedimentary units.

Figure 7-2: Local Geology Beneath the Overburden



Note: Reproduced from Prian, 1987.

The late Paleocene 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 Paleocene 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, Paleocene 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.

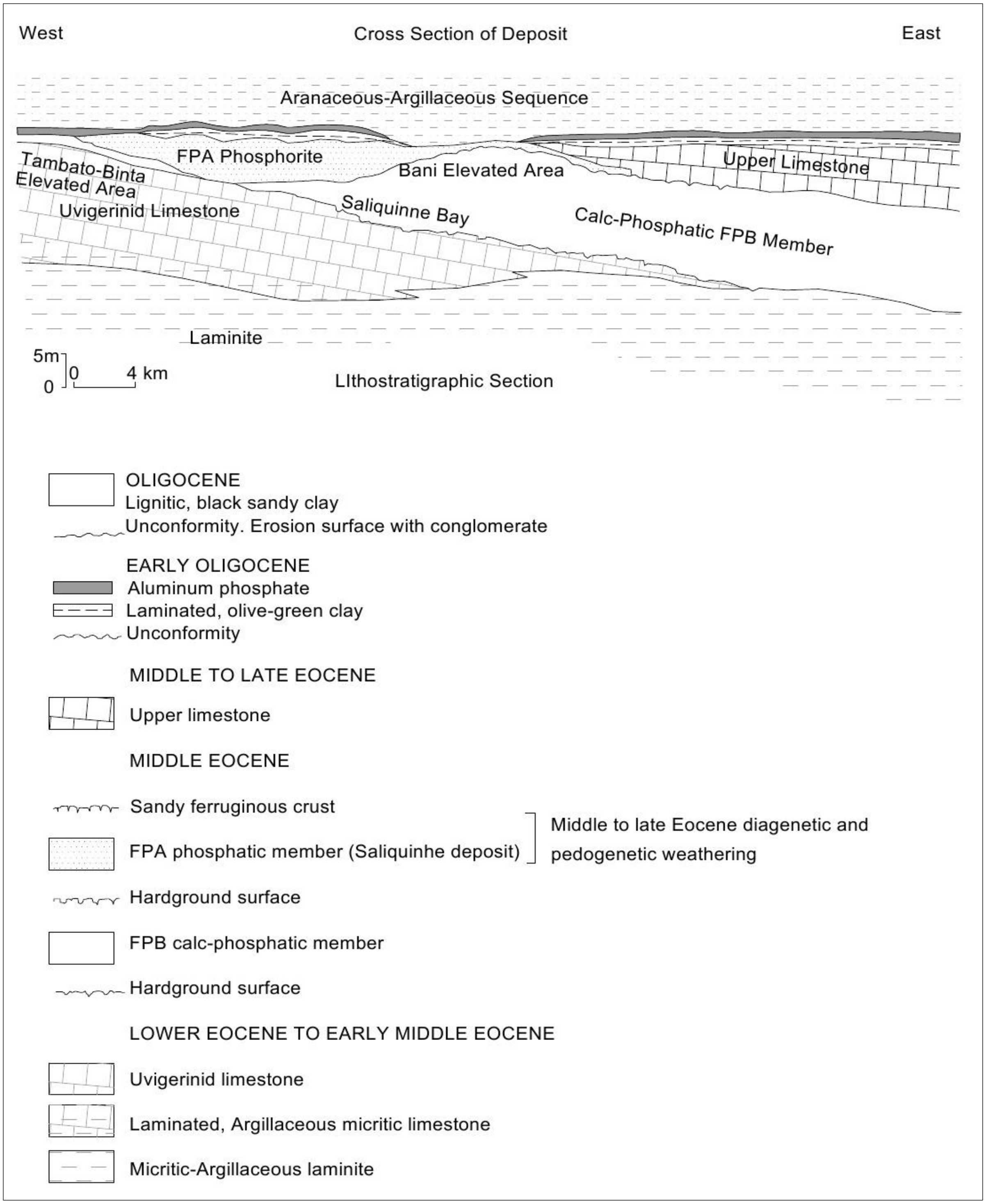
7.3 Property Geology

The Farim phosphate deposit is a flat-lying sedimentary phosphatic bed, which underlies an area larger than 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)
- 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



Note: Reproduced from Prian, 1989.

7.4 Deposit Geology and Mineralization

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

7.4.1 Overburden

The overburden waste at Farim typically consists of a layer of reddish-brown laterite gravel, followed by cream-colored 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 ranges from 26 m to 70 m with a mean of 41 m in the mining areas, whereas the phosphate matrix layer which is also a sedimentary deposit ranges from 1.0 m to 6.2 m with a mean of 2.7 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.

7.4.2 FPO

The FPO is a clayey dolomitic limestone that is weakly phosphatic and has limited economic potential and is sparsely distributed across the deposit. 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. The FPO is not included in the mineral resource estimate at Farim.

7.4.3 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 phosphatized shell and bone material, teeth, fecal pellets and crustacean coprolites. There is no calcareous cement, and it contains little silica and clay. It is mildly indurated and includes siliceous or pyritized layers 5 cm 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 localized 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 northwestern 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 limits of the FPA unit but this has not been demonstrated by drilling.

The limits of the FPA unit, the hanging wall and footwall, 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 cm to 60 cm thick unconsolidated sand. The hanging wall rocks are oxidized 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 oxidized interval. This is important because iron oxide, which is soluble in sulfuric acid, is a contaminant in phosphate deposits whereas iron sulfide, which is insoluble in sulfuric acid, is not (Champion, 2000).

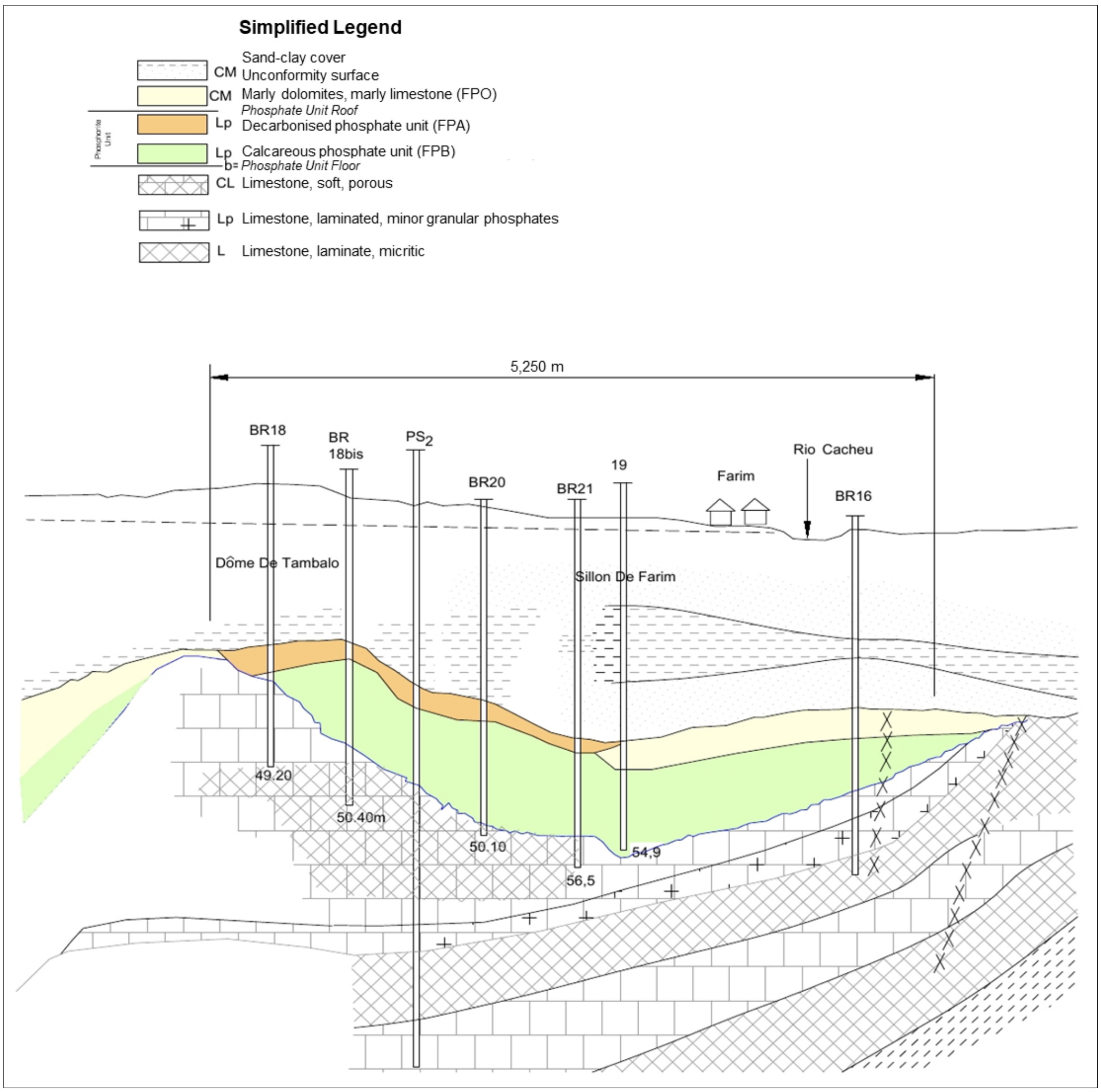
The FPA is very regular, sub-horizontal and continuous. The FPA unit has an average width of about 3 m (in the resource area) and underlies an area of about 60 km². In the northern part of the basin, north of the village of Saliquinhé, a northeasterly trending area about 5.5 km long and 1.5 km wide has FPA thickness typically greater than 3.0 m and up to 6.0 m. A smaller area to the south of Saliquinhé, near the River Cacheu, also exceeds 3.0 m in thickness.

7.4.4 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).

Figure 7-4: Representative Cross-Section through Farim Deposit



Source: WSP Golder, 2023

8 DEPOSIT TYPES

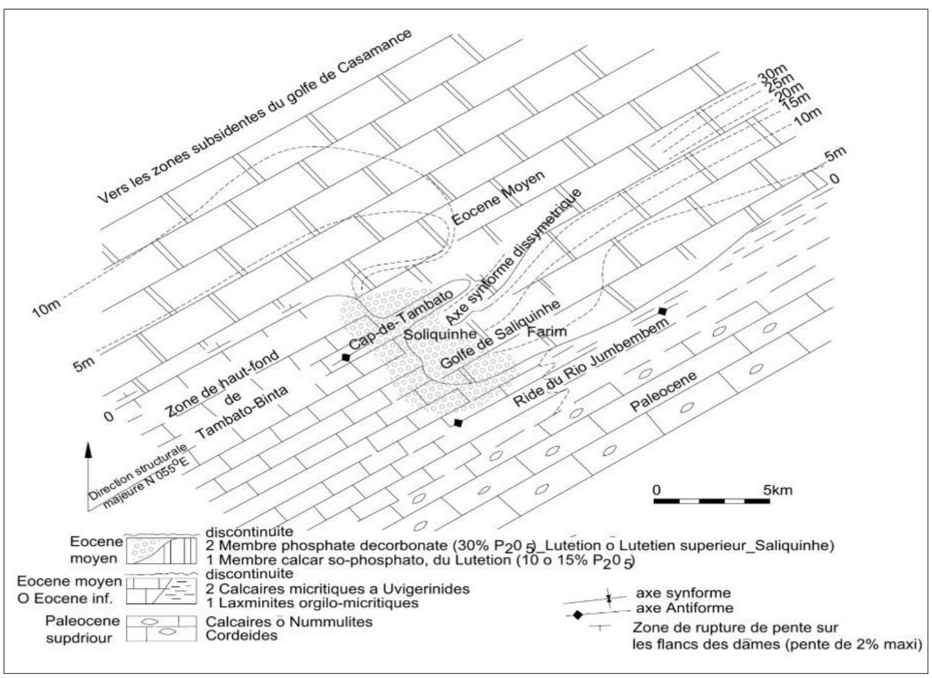
The following two main types of phosphate, differentiated by their petrography and chemical composition, have been identified on the Farim property:

- 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.
- Lower grade FPB layer – A 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 onto the ocean. The first phosphate deposit, FPB, was thick at the entry of the gulf and formed a bar (the “bar of Bani”) that 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 show a paleostructure 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).

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



Source: WSP Golder, 2023

9 EXPLORATION

Historical and recent exploration activity in the Farim project area has focused entirely on drilling campaigns; there are no documented non-drilling related exploration activities. Details of historical and recent drilling campaigns are discussed in Section 10. Itafos has not completed any exploration activities on the Farim Project. Historical activities are summarized in Section 6.

10 DRILLING

Itafos has not done any drilling since taking ownership of the Farim Property. The following is a summary of the drilling done by the previous owners of the property until 2019.

Phosphate was first discovered in the Farim area in one geotechnical drillhole as part of a water survey in 1950 and again in one oil drillhole 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, which were reported in 1980, 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 drillhole intersected 4.9 m of phosphate at 25% P_2O_5 under 40 m of sand-clay overburden.

Drilling in and around the Farim project area has been carried out by several companies since the discovery of the deposit. The current database contains 291 drillholes comprising 14,724 m of drilling using a combination of percussion and core drilling techniques. The drillhole map is shown in Figure 10-1. Drillholes drilled prior to 2015 are shown in black and those drilled since 2015 in orange. Since the layers of phosphate are horizontal, all the drillholes were drilled vertically and therefore the thicknesses intercepted are believed to be true thicknesses. The mean depth of the drillholes at Farim is 51 m, the mean overburden thickness is 41 m and the mean PFA thickness is 2.7 m.

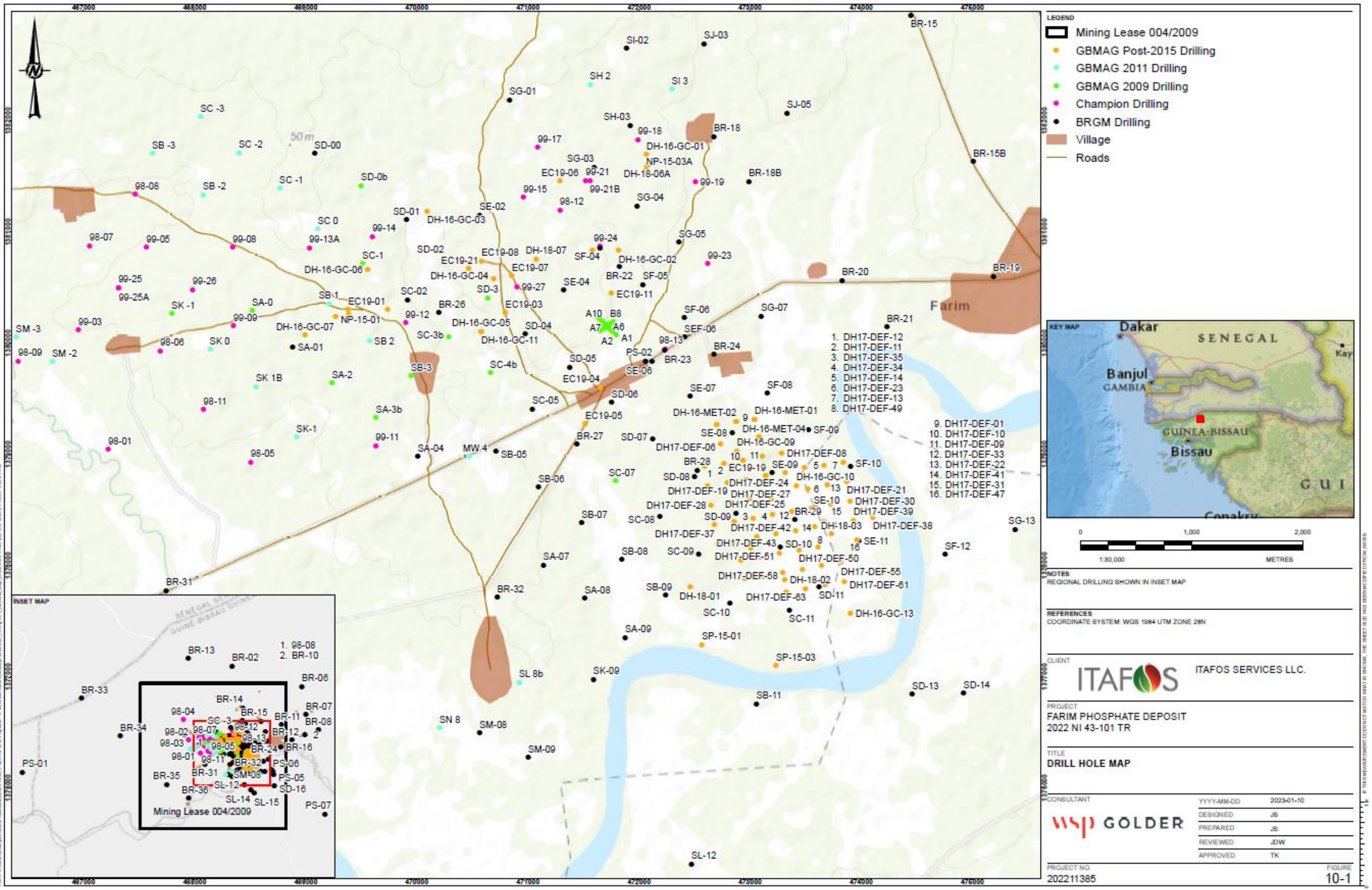
A summary of the historical and recent drilling by operator at Farim is presented in Table 10-1.

Table 10-1: Farim Drilling Summary by Operator

Program	No. of Drill Holes	Total Depth (m)
BRGM	101	5,672.0
Champion	34	1,810.0
GBMAG 2009	30	1,564.0
GBMAG 2011	25	1,280.5
GBMAG 2015-2019	101	4,289.0
Total	291	14,615.5

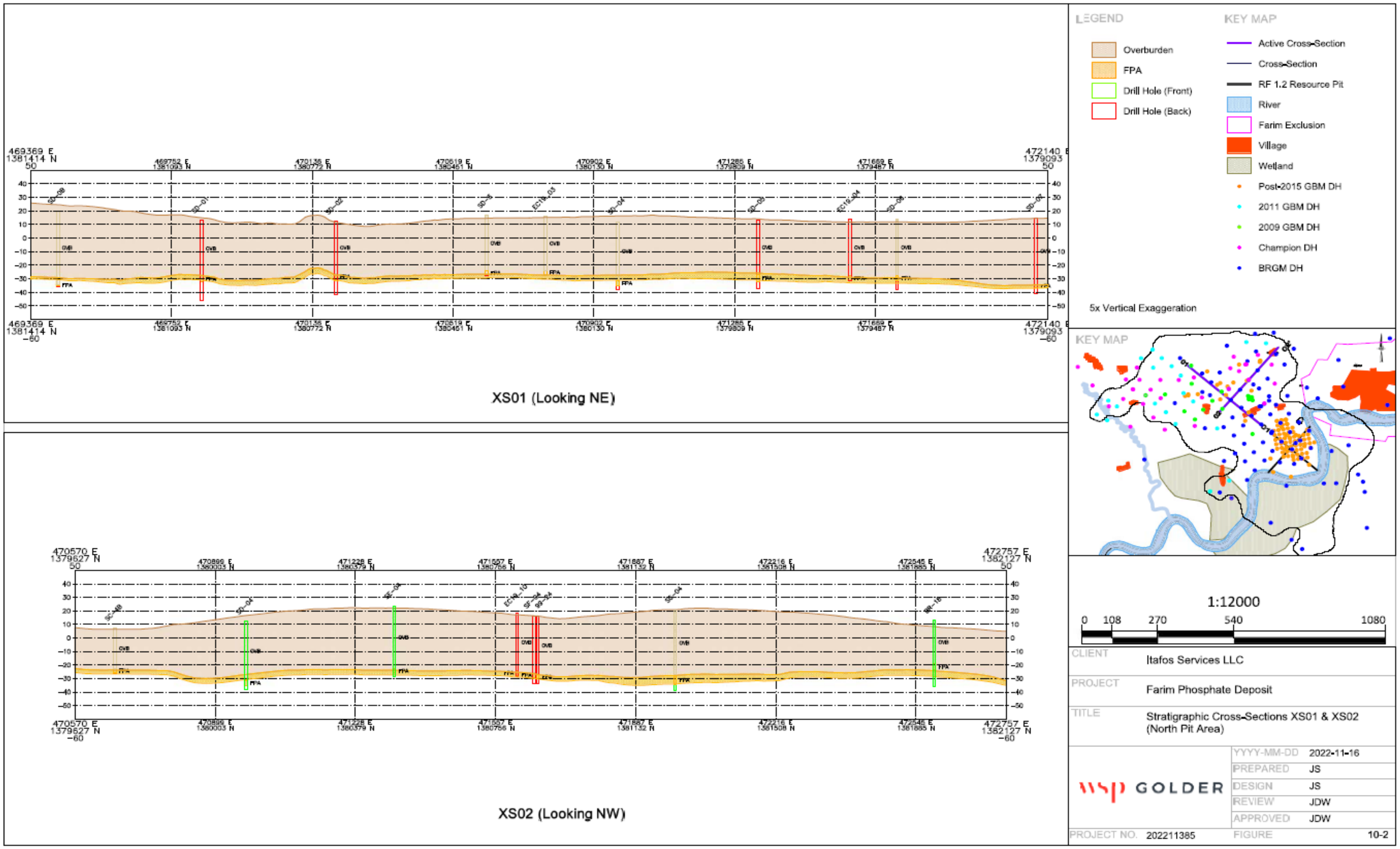
Examples of stratigraphic cross-sections through the north and South pit areas are illustrated in Figure 10-2 and Figure 10-3.

Figure 10-1: Drillhole Map



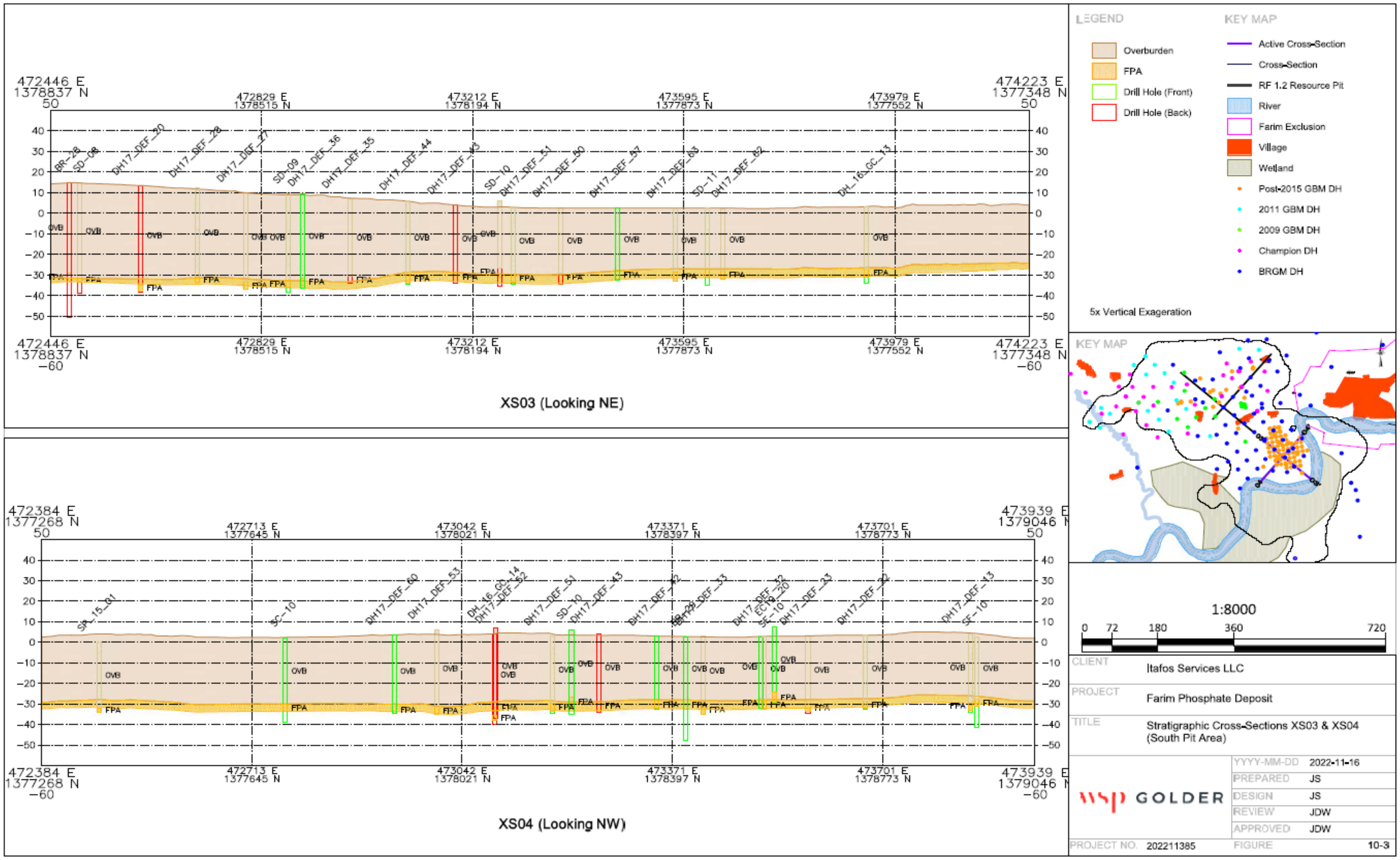
Source: WSP Golder, 2023

Figure 10-2: Stratigraphic Cross-Section XS01 and XS02 (North Pit Area)



Source: WSP Golder, 2023

Figure 10-3: Stratigraphic Cross-Section XS03 and XS04 (South Pit Area)



Source: WSP Golder, 2023

10.1 Historical Drilling

10.1.1 BRGM

The BRGM drilling program was carried out in three phases between 1981 and 1985. The program consisted of 101 drillholes totaling 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 the FPA roof was possibly 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 expert driller, the recovery can vary, and low recoveries reported in the BRGM and following reports should not be attributed to bad practice or negligence. The phosphatic clasts have a porous texture and very low density. Although the crystallized 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.

An 18 kg composite sample of FPA was taken from four drillholes 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 drillholes but did not produce good results. Phase 2 was a local exploration campaign carried out from 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.

Representative samples totaling 470 kg were taken from 30 drillholes 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 material 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 micropaleontology; these were examined by BRGM specialists
- 90 samples of micropaleontology of vertebrates, teeth of Selacians and Betides; these were examined by the Faculty of Science of Montpellier
- examination of invertebrates (ostracized) in seven surveys by the Faculty of Science of Lyon
- X-ray diffraction of 47 samples 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 pyritized 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.

The following equipment was used:

- Longyear 34 drill on a truck and a tanker of 7,000 liters
- Trepans tri-cone of 160 mm for overburden drilling, casing of diameter 135 mm to 145 mm
- 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
- casing before introduction of the probe gamma ray (Probe Mount Sopris).

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

Most of the drillhole collars were marked in the field with strong concrete beacons (Figure 10-4) and were located effectively by GBMAG geologists. It is unknown how the drill collars were originally surveyed.

Figure 10-4: BRGM Collar Marked in the Field with Concrete Beacon

Source: WSP Golder, 2015

10.1.2 Champion

During 1998 and 1999, Champion carried out 34 core drillholes totaling 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 how the collar locations were surveyed, or if the holes were gamma ray logged. The drillhole collars were marked by smaller, flat, concrete plugs which have been difficult to locate during recent resurvey efforts.

10.2 Recent Drilling

10.2.1 GBMAG/GEEEM Drilling

Between 2008 and 2009, GBMAG drilled 30 holes totaling 1,564 m, of which 423 m was core drilling. In 2011, 25 holes were drilled totaling 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 mineralization is horizontal, the vertical hole intersections are representative of the true thickness of the mineralization.

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 drillholes and give better definition of the variability of the mineralization. The second phase of drilling was planned to further extend the known mineralization towards the north and west and to infill to an approximate 500 m grid spacing.

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.

This work was managed and supervised by Geologie Exploration Environment Expertise Mine (GEEEM), 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.

10.2.2 Post-2015 GBMAG/GEEEM

Between 2015 and 2019, GBMAG completed a further 101 drillholes totaling 4,289 m within the existing measured mineral resource area. These holes were completed primarily for definition drilling in support of geotechnical characterization, metallurgical sampling, overburden characterization, and water monitoring programs unrelated to the mineral resource estimate. Most of the drillholes (83) were drilled in the South pit area, with the remaining 18 in the North pit area (Figure 10-1). The metallurgical drillholes were destructively drilled from surface to 2 m above the estimated roof of the FPA, and then cored through the FPA, FPB or into the limestone below. The geotechnical, overburden and water monitoring drillholes were cored as needed to obtain the required information.

To evaluate the reliability of the historical drilling data, the QP reviewed the post-2015 drilling against the previous drilling data and the existing model and found that there was no material difference between them and no influence on the global mineral resource. Given that the post-2015 drilling was completed within the existing measured resource area, the QP chose not to include the drilling in the 2022 mineral resource update. The QP recommends that any future model updates include all available drilling and sampling information to aid in short-term mine planning for preproduction or production activities.

10.3 Drill Core Recovery

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

The mean core recovery for the 2016 and 2017 metallurgical drilling programs was 83%, with more than 63% of the holes having greater than 80% recovery. The QP was not provided with core recovery rates for the other drilling completed on the Project since 2015.

10.4 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 the most recent work by 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 in the project area.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Itafos has not undertaken any sampling or analysis since taking ownership of the Farim property. The following is a summary of work done by the previous operators, but which has been incorporated into the geological modelling and resultant mineral resource and mineral reserve estimates.

The sample preparation, analyses and sample security procedures and results are summarized for BRGM and Champion (historical operators) and GBMAG (current operators) in the following sections. A summary of the total assay sample counts by operator is presented in Table 11-1.

Table 11-1: Summary of Assays by Operator

Program	No. of Drill Holes	Total Depth (m)	No. of Assay Samples
BRGM	101	5,672.0	63
Champion	34	1,810.0	17
GBMAG 2009	30	1,564.0	30
GBMAG 2011	25	1,280.5	19
GBMAG 2015-2019	101	4,289.0	48
Total	291	14,615.5	177

11.1 Historical Sampling Programs

11.1.1 BRGM Drilling 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. While certificates do not exist for review, the BRGM data are supported by several extensive duplicate analyses programs conducted at various stages in the project history, including several iterations of pulp duplicate analyses performed by BRGM at various laboratories as well as field duplicate analyses performed by Sofremines in December 1985 on 43% of the BRGM sample population using reference core. Additionally, as discussed in Section 10, the results from the historical drill holes were also compared to proximal drill holes from the post-2015 GBMAG drilling programs and no material differences in the results were identified.

11.1.1.1 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, the QP was unable to view the historic cores and validate any of the geological logging.

11.1.1.2 Density

Dry density measurements were made by BRGM in 1983. BRGM took 31 samples from 14 drillholes 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 color phosphate and the high-density values relate to a dark color 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.

11.1.1.3 Sample Preparation Procedures

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

- The drill core was split in increments as received along the 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 meter length. The remaining quarter core was retained as a control sample. Drying was carried out in an oven or by natural drying and weighing and stage crushing of quarter core samples down to about 8 mm using jaw crushers.
- The grinding jaw crusher product was ground down to about 0.5 mm to 2 mm using either a roll crusher or a disc mill.
- Ground material less than 2 mm was split 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 G's equations were used to estimate sampling errors made in primary sample splitting. Typically, drawing a 100 g subsample 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.
- The samples were dried in an oven at 105°C and then weighed. One of the subsamples was milled 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 pulverized material was split, subsampled and spare samples were kept in sealed plastic tubes to be dispatched to laboratories in charge of analysis and check analysis.

11.1.1.4 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 analyzed 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 analyzed for a further 10 elements.

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

No information is available on the size of these samples.

Core samples collected from 60 drillholes of the 1982 to 1983 campaign were analyzed 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 color development reagent
- CO_2 from carbonates and phosphate particles: CO_2 volume measurement following acid dissolution
- U: uranium content of selected finely ground samples were determined by the CEA (Commissariat à l'Energie Atomique) in France, which specializes in uranium analyses.

At BRGM in Orleans, P_2O_5 contents of samples were determined from solutions obtained after acid dissolution with sulfuric and nitric acids, using the following:

- spectrophotometric method based upon the yellow color of the ammonium phosphor-vanadomolybdate complex
- 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 fertilizers. Spectrophotometric determinations were validated against gravimetric determinations at BRGM since at that time the gravimetric method was the reference method in the phosphate fertilizer industry in France.

Routine analyses 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.

11.1.1.5 Sample Storage and Dispatch

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

11.1.1.6 QA/QC

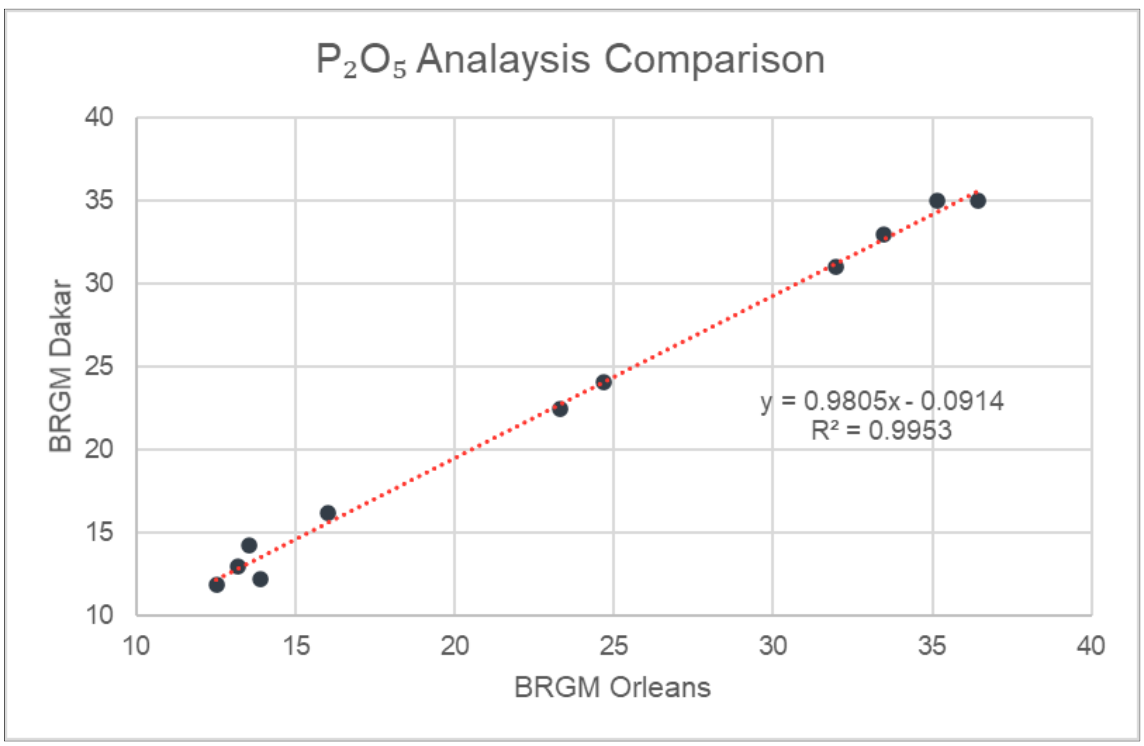
11.1.1.6.1 Pulp Duplicates

Quality assurance and quality control (QA/QC) check analyses 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). The results are shown in Table 11-2 and Figures 11-1 and 11-2.

Table 11-2: BRGM-DGM Drilling Campaign (1981 to 1983)

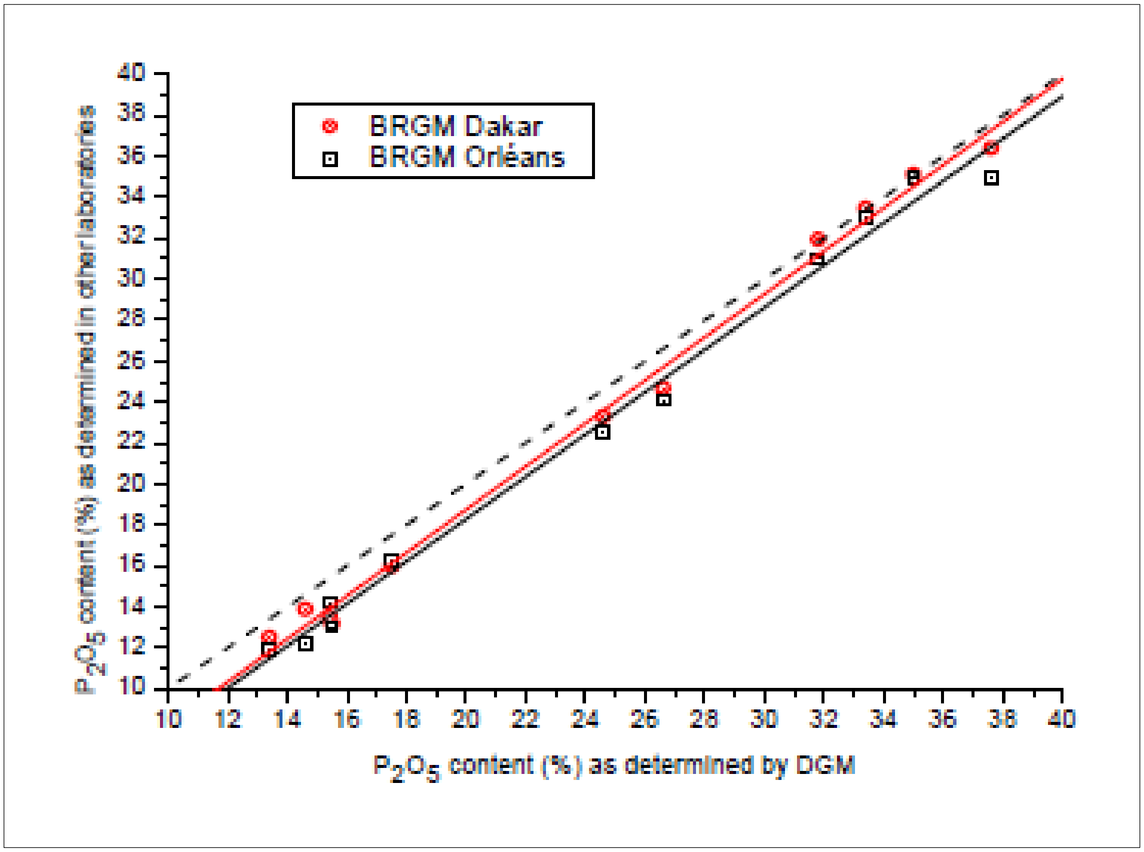
Drillhole	Drill Core Distance from surface (m)	Drill Core Length (m)	P_2O_5 Content (%) by DGM Dakar	P_2O_5 Content (%) by BRGM Dakar	P_2O_5 Content (%) by BRGM Orléans
BR20	34.50 to 35.50	1.00	13.4	12.51	11.9
	35.50 to 37.10	1.60	14.6	13.89	12.2
	45.50 to 47.30	1.80	15.5	13.20	13.0
BR21	37.50 to 38.70	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)



Source: IMC, 2011

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



Source: IMC, 2011

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^2 = 0.9973; mean absolute error SD = 0.759% P_2O_5 with 95% confidence limits on parameters of 1.0519 ± 0.052 and -2.28314 ± 1.3356

- % P_2O_5 Orleans = 1.03238% P_2O_5 DGM – 2.35389 Correlation coefficient, R^2 = 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.

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% P_2O_5 Orléans + 0.20055.

Correlation coefficient, R^2 = 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.

11.1.1.6.2 Internal QA/QC

It is reported that BRGM was well-qualified and has significant experience in P_2O_5 analyses. BRGM oversaw 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.

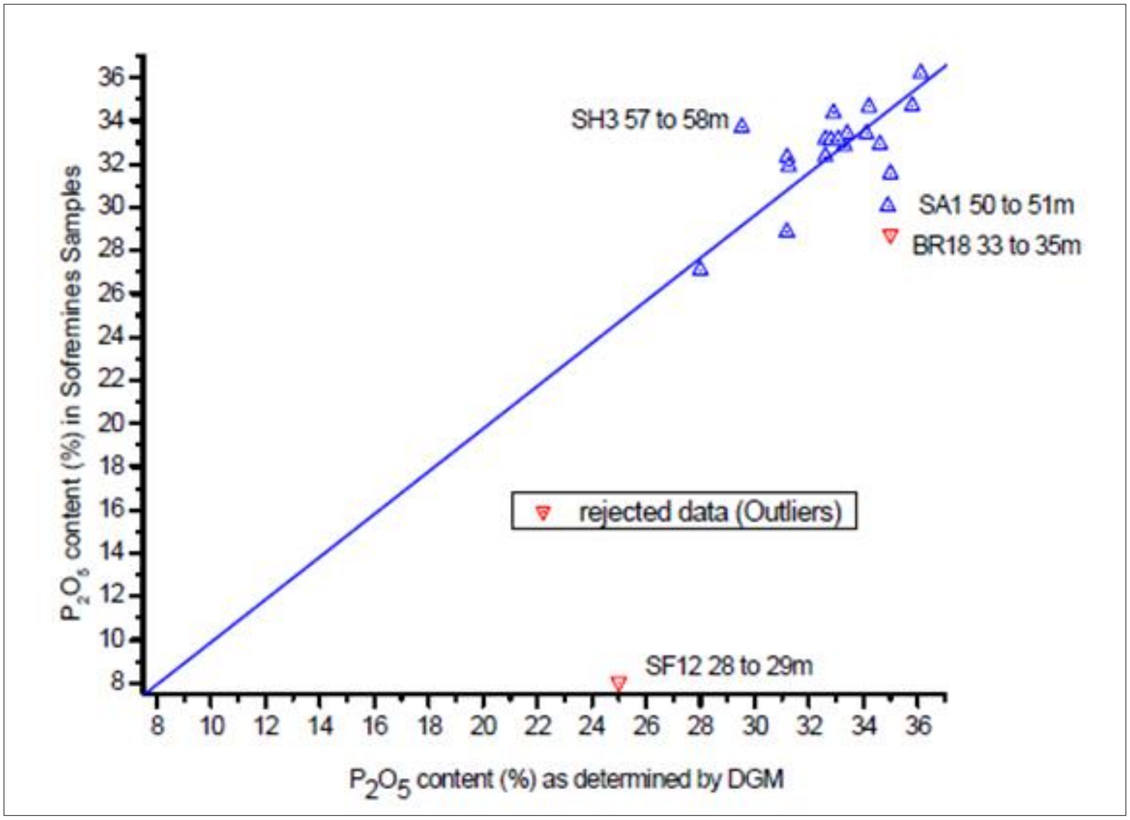
11.1.1.6.3 Field Duplicates

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. In total, 43% of the primary samples were submitted for duplicate analyses as part of the program. The principal purpose of these tests was to validate the BRGM sampling and analyses.

Twenty-one Sofremines samples analyzed 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 analyzed 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. 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 because of physical reconstitution of core intervals for Sofremines samples and reconstitution of phosphate grades of DGM samples by calculation.

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



Source: IMC, 2011

Seven Sofremines samples were analyzed for Fe_2O_3 by XRF on glass pellets and six of them were also analyzed by wet chemical methods for S as sulfides with the objective to estimate proportions of total iron occurring as iron sulfides (FeS_2 as pyrite and marcasite in accordance with BRGM mineralogical data). Data including comparisons of Fe_2O_3contents in Sofremines samples and Fe_2O_3 contents calculated from DGM analysis are shown in Table 11-3. From the limited data available, the proportion of iron occurring as sulfides in FPA samples assaying 28.85% to 36.20% P_2O_5 appears to be highly variable, ranging from 43% to 95%.

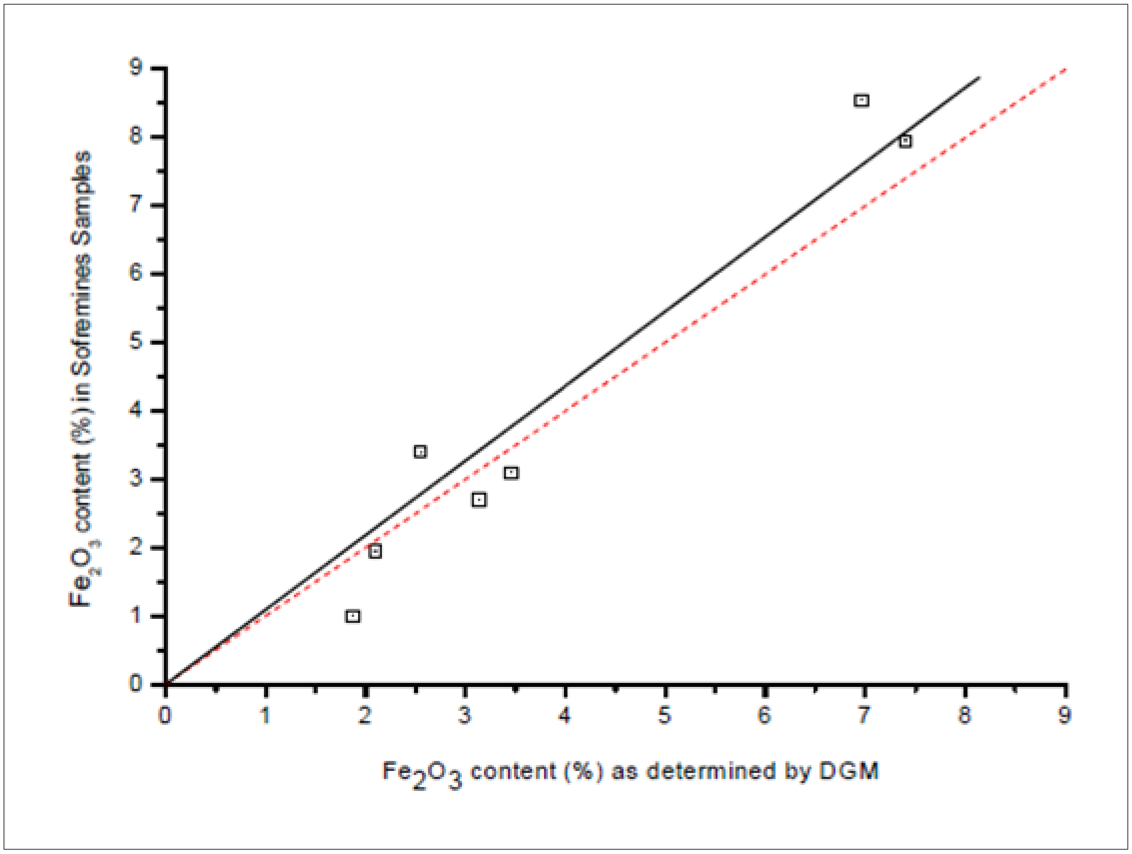
Figure 11-4 shows correlation between Fe_2O_3 contents as analyzed in Sofremines' samples and Fe_2O_3 contents as calculated from DGM analysis. As mentioned for phosphate analysis, differences with DGM results account for both sampling and analytical errors.

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.

Table 11-3: 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 Analyzed or Reconstituted	Drill Hole	Drill Core Distance from Surface (m)		Drill Core Length (m)	P_2O_5 Content (%) in Core Length
	From	To					From	To		
SC 2	41.3	42.3	1.1	32.3	31.2	SC 2	41.4	42.4	1.0	31.2
	42.3	43.3	1.0	33.2	32.6		42.4	43.4	1.0	32.6
	43.3	44.4	1.1	34.7	34.2		43.4	44.9	1.5	34.2
	44.4	46.0	1.1	32.4	32.6		44.9	46.0	0.6	32.6
	46.0	47.3	0.9	33.4	34.1		46.0	47.2	0.9	34.1
SB11	28.5	30.5	2.0	31.9	31.3	SB11	28.5	28.8	0.3	294.0
	30.5	32.1	1.6	28.9	31.2		28.8	29.5	0.7	32.4
							29.5	30.4	0.9	31.0
							30.4	31.2	0.8	28.0
							31.2	32.1	0.9	34.0
SE2	56.3	58.0	<1.00	27.1	28.0	SE2	56.3	58.5	1.0	28.0
	58.0	60.6	About 1.50	33.7	29.5		58.5	59.3	0.5	32.6
							59.3	60.6	1.0	28.0
BR23	36.0	38.0	2.0	32.9	33.3	BR23	35.7	37.6	1.9	35.2
							37.6	38.8	1.2	25.7
BR28	46.0	47.0	1.0	36.2	36.1	BR28	45.5	46.4	0.9	36.3
							46.4	46.8	0.4	37.9
							46.8	47.9	1.2	33.3
BR29	32.0	34.0	2.0	33.1	32.8	BR29	31.7	32.4	0.7	28.0
							32.4	33.4	1.0	35.5
							33.4	34.4	1.0	31.5
SH3	54.0	55.0	1.0	34.4	32.9	SH3	53.5	54.5	1.0	33.2
	55.0	56.0	1.0	33.4	33.4		54.5	55.5	1.0	32.6
	56.0	57.0	1.0	32.9	34.6		55.5	56.5	1.0	34.2
	57.0	58.0	1.0	31.6	35.0		56.5	57.5	1.0	35.0
	54.0	58.0	4.0	33.1	34.0		57.5	58.5	1.0	35.0
SH3	54.0	55.0	1.0	34.4	32.9	SH3	53.5	54.5	1.0	33.2
	55.0	56.0	1.0	33.4	33.4		54.5	55.5	1.0	32.6
	56.0	57.0	1.0	32.9	34.6		55.5	56.5	1.0	34.2
	57.0	58.0	1.0	31.6	35.0		56.5	57.5	1.0	35.0
	54.0	58.0	4.0	33.1	34.0		57.5	58.5	1.0	35.0
SF12	28.0	29.0	chert pebbles inter waste	8.1	25.0	SF12	27.7	29.5	1.8	25.0
SA1	48.0	49.0	1.0	35.2	33.1	SA1	47.7	48.3	0.6	30.2
	49.0	50.0	1.0	34.7	35.8		48.3	49.0	0.7	31.8
	50.0	51.0	1.0	30.1	34.9		49.0	49.1	0.1	35.0
							49.1	50.0	1.0	35.8
							50.0	50.5	0.5	33.2
							50.5	51.7	1.2	36.6
BR18	33.0	35.0	2.0	28.8	35.0	BR18	33.0	34.0	1.0	35.8
							34.0	35.2	1.2	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



Source: IMC, 2011

The QP 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 prevailing international standards.

11.1.2 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).

11.1.2.1 Lithological Logging

Refer to Section 11.1.1.

11.1.2.2 Sample Preparation Procedures

There is no information on how Champion processed the core.

11.1.2.3 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/m³ after excluding the highest abnormal values. The lowest value is 1.18 t/m³ and the highest value of 1.98 t/m³. There is no apparent evidence of a relationship between density and phosphate grade.

11.1.2.4 Analytical Procedures

The following discussion and observations result only from review of reports. No original analytical procedural documents 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), drillhole core samples collected during the Champion exploration campaigns were analyzed for the following:

- 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
- 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 provided, and the analytical methods used are not fully described in the reports made available.

11.1.2.5 Sample Storage and Dispatch

There is no information on how Champion processed the core.

11.1.2.6 QA/QC

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

- 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, R^2, between pairs of determinations were about 1 for the first set and 0.998 for the second set. These high R^2 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 regarding the reference assay.

- 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.

11.1.2.6.1 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^2 = 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.

11.1.2.6.2 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 inductively coupled plasma (ICP) 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^2 = 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 organization specialized in chemical characterization of phosphate substances. For this reason, results from AAS determinations should be preferred.

11.1.2.6.3 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^2 = 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_3analysis within the appraisal of the deposit.

11.2 Recent Sampling Programs

11.2.1 GBMAG/GEEEM Program

11.2.1.1 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 QP, assisted by Geologist Guy Voglet, who oversaw the GBMAG drilling program. The QP 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 the QP's site visit, several recommendations were made to improve the housekeeping within the core shed. The QP recommended the use of a logging table, rather than core boxes being placed on the floor for logging and sampling. The QP 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, color, 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.

11.2.1.2 Density

GBMAG carried out no density measurements. It is recommended that future Itafos drilling and sampling programs include collection of samples for density analyses to further supplement the historical density data.

11.2.1.3 Sample Preparation Procedures

The procedure used for core sampling and sample preparation is as follows:

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

11.2.1.4 Sample Storage and Dispatch

The collection and processing of all samples prior to dispatch to a laboratory was conducted by GEEEM and GBMAG employees. All sampling was sent as a single batch once drilling was complete. Samples were stored in a locked room at the GBMAG office, to which only GBMAG and GEEEM employees had access. Samples were dispatched in wooden crates, submitted using a standardized laboratory submission form which listed 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.

11.2.1.5 Analytical Procedures

From the holes drilled by GBMAG pre-2015, 156 intervals from 55 holes were sampled and analyzed. Since 2015, an additional 56 intervals were sampled and analyzed. 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 set up an onsite 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 Mineral Resource estimate.

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

- Samples were pulverized using disc mills with steel bowls until 90% of the sample passes a 75 µm screen.
- A subsample was taken and placed in a Kraft envelope for dispatch to the analytic lab. The amount of material in the envelope was weighed and recorded in the system.
- The preparatory laboratory stored the pulps for GBMAG.
- Samples were sent to ALS Chemex in Vancouver, Canada for analysis.

The analytical procedures were as follows:

- Samples were 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 sulfur using the Leco method.
- The laboratory stored the pulps rejects for GBMAG.
- GBMAG received the assay results from the laboratory via email as Microsoft spreadsheets and PDF scan of the original certificate.

WSP Golder personnel visited both 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 QP 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.

11.2.1.6 QA/QC

The GBMAG QA/QC 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 and 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 QP was not provided with details on any QA/QC programs for the post-2015 drilling.

There were no failures related to the GBMAG QA/QC samples and the sample results are considered as reliable and fit for purpose for preparing geological models and mineral resource estimates. However, both the overall sample counts and the associated QA/QC sample counts were low, providing limited data to allow for a more robust analysis of GBMAG analytical QA/QC. It is recommended that future drilling and sampling programs include a more robust QA/QC sampling and analyses program.

The QP 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.

11.3 Laboratory Accreditation

Formal accreditations for the various analytical laboratories used throughout the Farim Phosphate Project history are as follows:

- 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.

All laboratories were independent of the project owner at the time of the analyses described in the previous sections.

11.4 Qualified Person's Statement

It is the QP's opinion that the sample preparation, security, and analytical procedures used by GBMAG, Champion, BRGM and their representatives followed appropriate industry standard best practices and procedures, and the results are fit for the purpose of preparing geological models and estimates of mineral resources as well as for the development of associated mine plans and estimates of mineral reserves for the project. It is recommended that future programs continue to collect and review QA/QC data to current standards. Additional density data should also be collected to supplement the historical density data collected during the BRGM and Champion campaigns.

12 DATA VERIFICATION

12.1 Independent Sampling

The QP did not collect any samples during the May 2015 site visit. During the May 2011 site visit, however, the QP 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 μm 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.

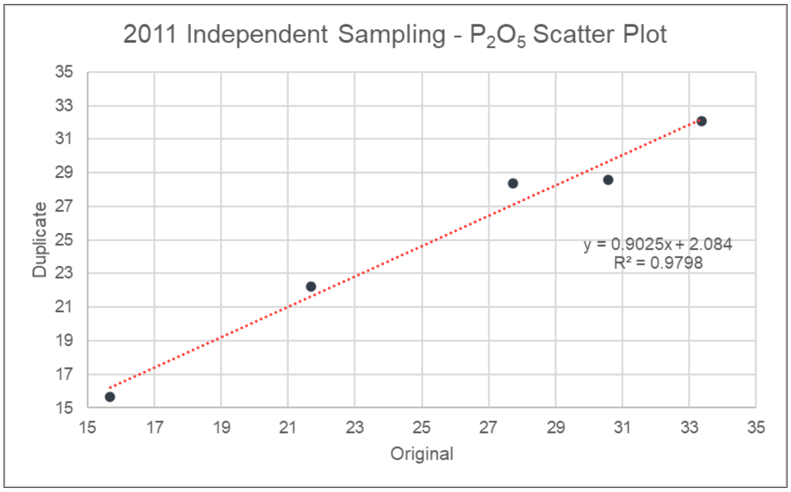
The comparison between the original assay values and the values obtained by the independent assaying carried out by the QP is presented in Table 12-1 and the scatter plot in Figure 12-1. The largest differences in P_2O_5 grade were observed in the two high grade samples, 070 (1.97% difference in P_2O_5 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 laboratory 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



Source: WSP Golder 2011

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 2011 and May 2015 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 QP site visit and the QP's 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 the QP supervised drilling of both resource and metallurgical drillholes. The QP is satisfied that the procedures being used are adequate for the style of mineralization (Figure 12-2).

Figure 12-2: Drilling at Farim

Source: Knight Piesold, 2015

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

12.3 Drillhole Collar Survey Check

It is not known what method was used to survey the BRGM and Champion drillhole collars. The GBMAG drillhole collars were initially surveyed using a handheld GPS, except for the set for the variogram, which was surveyed and levelled locally by a consulting surveyor. In 2011, all drillhole 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 the QP inspected a random subset of collars and took measurements of the locations using a handheld GPS (GARMIN GPSmap76CSx and GARMIN GPSmap60CSx). The QP visited a total of 12 drill sites for the purpose of verifying collar coordinates (five drill sites during the 2011 site visit and 12 drill sites during the 2015 site visit). Except for one drillhole (KP-SGW-BH01) where differences were in excess of 20 m, no material differences were found between the original and the QP GPS coordinates. All collars plotted within 12 m of the recorded position, which is within the expected accuracy of such equipment.

Table 12-2 summarizes the validation work undertaken by the QP by area.

Table 12-2: Drillhole Collar Survey Check

Drillhole	QP 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 the following information for each hole:

- X coordinate
- Y coordinate
- Z coordinate
- total depth
- lithology interval/unit names
- lithology interval from and to depths and thickness
- core recovery percentage
- sample interval from and to depths and thickness
- weight percent P_2O_5 – available for all sample intervals
- weight percent Al_2O_3, CaO, Cr_2O_3, Fe_2O_3, K_2O, MgO, MnO, SiO_2, TiO_2, loss on ignition [LOI], F, and C – not available for all drillholes/sample intervals.

The P_2O_5 grades reported in the database were only the length weighted average per drillhole. The individual assay results for each sample were not detailed. No separate lithology, collar or survey files were supplied. Analyses for additional grade parameters (Al_2O_3, CaO, Cr_2O_3, Fe_2O_3, K_2O, MgO, MnO, SiO_2, TiO_2, LOI, F, and C) are available variably across the different drilling campaigns.

The QP 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 drillholes 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
- P_2O_5 drillhole composite grade.

During the 2011 project, the QP 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) drillholes. 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 drillhole, the QP digitized the individual sample assay values as written on the geological logging sheet and recalculated a length weighted average per drillhole. 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, the QP 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 drillholes, 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, the QP 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).

The QP could not validate or check any of the Champion data against assay certificates; the checks were performed using duplicate assays as well as via comparison against recent drill hole assay results.

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

The QP did not actively participate in implementing the exploration drilling and sampling programs, and site visits were performed outside of the implementation phases of the various drilling programs. The QP therefore cannot speak to the exploration drilling, logging, sampling and analytical procedures and methodologies implemented during all phases of exploration work on the Farim Phosphate Project. However, it is the QP's opinion that the verified data and observations are consistent with data and observations collected using the exploration procedures and methodologies provided by Itafos for work that was performed under GBMAG 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 the QP's opinion that the exploration data and observations collected from the drillholes and analytical samples that comprise the geological database of the Farim Phosphate Project have been appropriately verified for the purpose of completing a geological model, estimating mineral resources, and preparing an NI 43-101 Technical Report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The objective of the testwork was to quantify the metallurgical response of ore from the Farim Phosphate deposit and to develop the basic engineering for the metallurgical processing of this phosphate ore. The program was designed to develop the parameters for process design criteria for ore washing/scrubbing, desliming, flotation, and dewatering in the processing plant including pilot plant tests.

The metallurgical program was organized by KEMWorks Technology Inc. (KEMWorks), with bench-scale testing at KEMWorks' lab in Lakeland, Florida, and the pilot plant tests at ALS – Metallurgy (ALS) in Kamloops, Canada, and SGS Mineral Services (SGS) in Lakefield, Canada.

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. Later, selected samples of the North pit were submitted to preliminary characterization, and metallurgical testwork using the designed beneficiation process developed for the South pit to extend the years of operation of the Farim deposit.

Five size fractions of the Farim composite sample of the South pit 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. In addition, beneficiation tests for the North pit samples, and four pilot plant studies were conducted on September 2016 and between 2017 and 2019, respectively. Historical testing has been summarized in Table 13-1.

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. For a map of drillhole locations, refer to Section 10, Figure 10-1. 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 were preserved, while the remaining half of each cut was used to prepare the composites.

Table 13-1: Summary of Historical Testing

No.	Document Tittle	Deposit	Technical Content	Date
1	Evaluation of Farim Phosphate Beneficiation Test Report and Metallurgical Tests and	Farim	Sample Preparation, Characterization Studies,	2014
	Process Development		Horizontal Scrubbing, Attrition Scrubbing, Desliming,	
			Flotation Test	
2	Evaluation of Stored Farim Samples Qualification Report	Farim	Physical and Chemical Properties	2014
3	Completion of Additional Work and Updated Feasibility Study Result in Significant	Farim	Technical News Release for Feasibility Study	2015
	Operational and Financial Improvements			
4	Pilot Scale Processing of a Phosphate Ore	Farim	Flowsheet Test, Samples for Sedimentation and	2015
			Tailings Tests, Sample for Phosphoric Acid testing	
5	Knight Piesold-Tailings Physical Testing	Farim	Tailings Characterization and Storage Studies	2015
6	Metallurgical Test and Process development	Farim	Chapter 13 NI 43-101 Technical Report	2015
7	Outotec-Thickening Test Report	Farim	Thickening of Phosphate Tailings	2015
8	Beneficiation Technology	Farim	Chapter 17 NI 43-101 Technical Report	2015
9	SGS-Mineralogical Characterization of Ore Phosphate Composite Sample	Farim	Mineralogy, QEMscan	2015
10	Recommendations	Farim	Section 26 NI 43-101 Technical Report	2015
11	Interpretation and Conclusions	Farim	Section 25 NI 43-101 Technical Report	2015
12	GB Mineral Limited-Farim Phosphate Project	Farim	Mass Balance	2015
13	GB Minerals Limited-Farim Phosphate Project-Process design Criteria	Farim	Feasibility Study	2015
14	GB Minerals Limited-Farim Phosphate Project	Farim	Water Balance	2015
15	NI 43-101 Technical Report	Farim	Feasibility Study	2015
16	Farim Phosphate Project-North Pit Beneficiation Studies	Farim	Bench Scale Tests + Process Flowsheet Evaluation	2016
17	An Investigation into The Beneficiation Characteristics of Material from The Farim	Farim	Samples Preparation, Characterization Studies,	2017
	Phosphate Project		Bench Scale Tests, Pilot Plant Tests, Solids Liquid	
			Separation and Rheology, Filtration	
18	SGS Second Pilot Plant test	Farim	Pilot Plant results, Solid-Liquid Separation, Rheology	2017
19	ALF Third Pilot Plant Test	Farim	Pilot Plant Results, Concentrate Quality,	2017
			Environmental Testing on Selected Tailings Products	
20	ALF Fourth Pilot Plant Test	Farim	Pilot Plant Results and Concnetrate Samples	2019
21	Solids-Liquid Separation Testing Report	Farim	Settling, flocculation and Rheology	

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 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 13-2 summarizes the information received and the weights of each cut received along with the proportional weight used for each drill hole. Shading represents the top, middle, and bottom cuts selected for preliminary analysis before the hole subsamples were blended.

Table 13-2: Sample Reception and Composite Recipe

SB 9					SC 10				
Section	kg	Percent of Hole	Hole Composite, g	Reserve, g	Section	kg	Percent of Hole Total	Hole Composite, g	Reserve, g
32,15-32,35	3.0	8.3%	1500	1287	32,24-32,56	3.8	10.6%	1900	1809
32,35-32,65	4.2	11.7%	2100	2006	32,56-32,86	2.7	7.5%	1350	1406
32,86-33,08	3.3	9.2%	1650	1653	32,86-33,26	3.8	10.6%	1900	1936
33,08-33,46	5.4	15.0%	2700	2546	33,26-33,51	2.9	8.1%	1450	1436
33,46-33,79	4.8	13.3%	2400	2394	33,51-33,73	2.7	7.5%	1350	1280
33,79-34,09	4.2	11.7%	2100	1990	34,00-34,31	3.6	10.0%	1800	1741
34,09-34,27	3.0	8.3%	1500	1540	34,31-34,61	2.9	8.1%	1450	1472
34,50-34,82	4.2	11.7%	2100	2126	34,61-34,91	2.9	8.1%	1450	1511
34,82-35,12	3.9	10.8%	1950	1924	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	36.0		18000	17466	Total	35.9		17950	17992

SC 11					SE10				
Section	kg	Percent of Hole Total	Hole Composite, g	Reserve, g	Section	kg	Percent of Hole Total	Hole Composite, g	Reserve, g
30,47-30,82	0.42	9.4%	210	202	30,63-31,11	3.0	16.5%	1500	1360
30,82-31,17	0.47	10.5%	235	235	31,11-31,41	3.0	16.5%	1500	1535
31,17-31,52	0.47	10.5%	235	252	31,42-31,87	2.3	12.7%	1150	1180
31,52-31,64	0.16	3.6%	80	82	31,87-32,20	2.2	11.8%	1075	1106
31,93-32,28	0.39	8.7%	195	240	32,10-32,56	2.6	14.3%	1300	1244
32,28-32,58	0.37	8.3%	185	89	33,46-33,60	0.7	3.9%	350	376
32,58-32,93	0.39	8.7%	195	245	34,33-34,61	1.9	10.5%	950	986
32,93-33,20	0.34	7.6%	170	154	34,61-34,92	1.9	10.5%	950	949
33,20-33,60	0.47	10.5%	235	219	34,90-35,30	0.6	3.3%	300	243
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	Total	18.2		9075	8980

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.

In the case of the North pit samples, on September 14, 2016, the samples in Table 13-3 from the North pit were received from Guinea Bissau at KEMWorks Laboratory in Lakeland Florida.

Table 13-3: North Pit Samples Received

Box	Sample ID	Weight, kg	Tare, kg	Net Weight, kg	Colour
Blue	DH-16-MET-05	10.00	0.00	10.00	Dark Green
	DH-16-GC-03	10.00	0.00	10.00	Dark Green
Sub Total 1		20.00	0.00	20.00	
Red	NP-15-1	10.00	0.00	10.00	Dark Brown
	NP-15-03A	10.00	0.00	10.00	Brown
	NP-15-4	10.00	0.00	10.00	Beige
Sub Total 2		30.00	0.00	30.00	
Total		50.00	0.00	50.00	

The five samples were processed to obtain the following from each one:

- homogenized-blended sample
- head chemical analysis sample
- sample for determination of moisture content and dry density (bulk and in situ)
- sample for determination of wet density (bulk and in situ).

After the processing of each of these samples, the preparation of a weighed composite sample was carried out and equal amount of each of the five samples were blended to make the composite. From this composite sample, the following subsamples were obtained:

- moisture content subsample
- head chemical analysis subsample
- reserve of head chemical analysis subsample
- screen assays subsamples (duplicates)
- screen assays reserve subsample
- tests samples split of 500 g each subsample
- general reserve tests material.

Then, these composite subsamples were processed to perform the characterization studies, such as head chemical analysis, moisture content, and screen assays; and the beneficiation tests in triplicates for the composite North pit phosphate ore using the designed beneficiation process for the South pit phosphate ore.

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 +1,180 µm, 1,180 x 425 mm, 425 x 106 µm, 106 x 20 µm and -20 µm size fractions. The QEMSCAN results agree with the interpretation and conclusions of the screen assays results.

To demonstrate that the South pit 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 13-3 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 13-4.

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

Table 13-4: Chemical Analysis of Selected Cuts

Sample Identification	SB 9			SC 10			SC 11			SE 10		
	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 – % Fe_2O_3	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 Insoluble	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

Table 13-5: 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

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:

$$\% \, Fe_2O_3 \text{ pyritic} = \% \, S \text{ pyritic} \times (160 / 128)$$

Then the MER* is calculated by:

$$(\% \, Al_2O_3 + (\% \, Fe_2O_3 - \% \, Fe_2O_3 \text{ pyritic}) + \% \, MgO) / \% \, P_2O_5$$

The results of the chemical analysis for the North pit samples are shown in Table 13-5 for five North pit samples.

Table 13-6: Chemical Analysis for 5 North Pit Samples

Composite Sample	P_2O_5	CaO	Acid Insol	Al_2O_3	Fe_2O_3	MgO	Spyritic	Corganic	Moisture	CaO/P_2O_5	MER	Grade Potential	Fe_2O_3 *	Adjusted
	%	%	%	%	%	%	%	%	%	Ratio		P_2O_5, %	Pyritic, %	MER
DH-16-MET-05	31.02	43.48	8.31	2.59	3.33	0.28	1.36	0.86	26.83	1.402	0.2	33.83	1.69	0.145
DH-16-GC-03	23.29	41.48	8.85	2.79	6.75	0.13	3.74	1.32	22.96	1.416	0.33	32.13	4.66	0.171
NP-15-1	34.2	46.53	5.37	2.05	2.93	0.11	0.58	0.79	23.42	1.361	0.149	36.14	0.72	0.128
NP-15-03A	34.23	48.52	4.02	1.19	2.36	0.33	0.18	0.77	28.21	1.417	0.113	35.66	0.22	0.107
NP-15-4	34.91	41.39	4.77	2.19	1.57	0.12	0.13	0.4	27.06	1.186	0.111	36.66	0.16	0.107

The chemical analysis results indicated that the North pit phosphate ore was lower in P_2O_5 grade with higher impurities than the South pit phosphate ore. The main impurity for the North pit material was acid insoluble (A.I.), followed by Fe_2O_3, Al_2O_3, $S_{pyritic}$, and $C_{organic}$ in that order.

13.3.1 Head Sample Chemical Analysis

Three different South pit Farim composite samples were prepared and sent for chemical analysis throughout the testwork process.

Table 13-7 presents the results and the parameters of interest, such as CaO/P_2O_5 ratio, MER, adjusted MER (MER*), and P_2O_5 grade potential. 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 13-7: 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, %	$S_{pyritic}$ / 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				

The composite for the North pit phosphate ore was prepared and submitted to head chemical analysis. The characterization studies clearly showed that the North pit composite reported lower P_2O_5 grade (30.92%), higher acid insoluble, A.I. (7.26%), and high organic carbon, $C_{organic}$ (1.43%); the Al_2O_3 (1.75%), Fe_2O_3 (2.82%), MgO (0.14%), and pyritic sulfur, $S_{pyritic}$ (1.46%) being similar to those of the South pit. This is shown in Table 13-8. With respect to the metallurgical parameters for the North pit phosphate ore, the CaO/P_2O_5 ratio was low indicating that the presence of little amount of dolomite and calcium carbonates, the MER being similar to that of the South pit phosphate ore at 0.152. Since the content of $S_{pyritic}$ of North pit ore was higher than that of the South pit ore, the MER* (MER calculated with the subtraction of the

Fe_2O_3 coming from pyrite) reported was lower (0.094). The P_2O_5 grade potential (33.34%) was also low since it was calculated taking into consideration only the A.I. indicating that other contaminants, such as Al_2O_3, Fe_2O_3, and $S_{pyritic}$ may also need to be removed to upgrade the P_2O_5 to 34%.

Table 13-8: Composite Head Chemical Analysis[1]

P_2O_5	CaO	Acid Insol %	Al_2O_3	Fe_2O_3	MgO	Stotal	Spyritic	Moisture	CaO/P_2O_5	MER	Grade Potential	Fe_2O_3 *	Adjusted*
%	%		%	%	%	%	%	%	Ratio		P_2O_5, %	Pyritic, %	MER
30.92	43.24	7.26	1.75	2.82	0.14	1.46	1.43	2.8	1.398	0.152	33.34	1.82	0.094

13.3.2 Screen Analysis

Table 13-9 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 South pit 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.

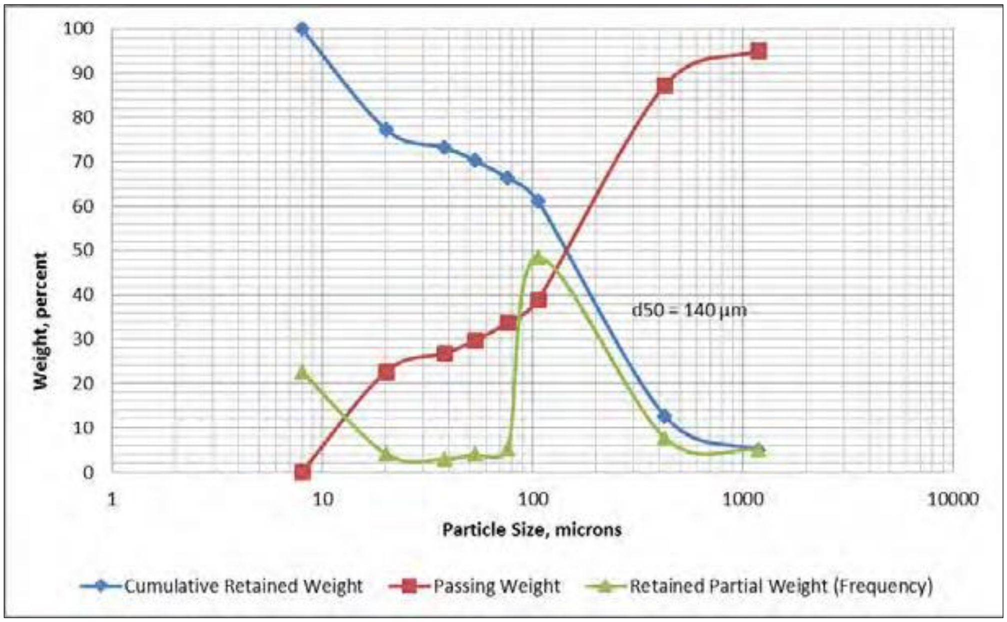
It was found that the North pit phosphate ore was significantly finer than the South pit phosphate ore, the mean particle size (d_{50}) being 115 µm for the North pit material, whereas that of the South pit was 140 µm. For the North pit phosphate ore, the mode particle size was at 212 µm (accumulation of particles in a size fraction), which was coarser than that presented by the South pit ore since the North pit weight retained in the 420 x 212 µm size fraction was significantly higher (17.52%). This indicated that the North pit contained more weight percentage of particles smaller than 212 µm than the South pit ore. However, similar weight percent of particles was observed on the -20 µm size fraction (22.91%). Thus, the size fraction of 212 x 20 µm may play an important role in the beneficiation behavior of the North pit phosphate ore, the 212 x 150 µm, 150 x 106 µm, and 53 x 20 µm containing significant weight.

Table 13-9: 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		

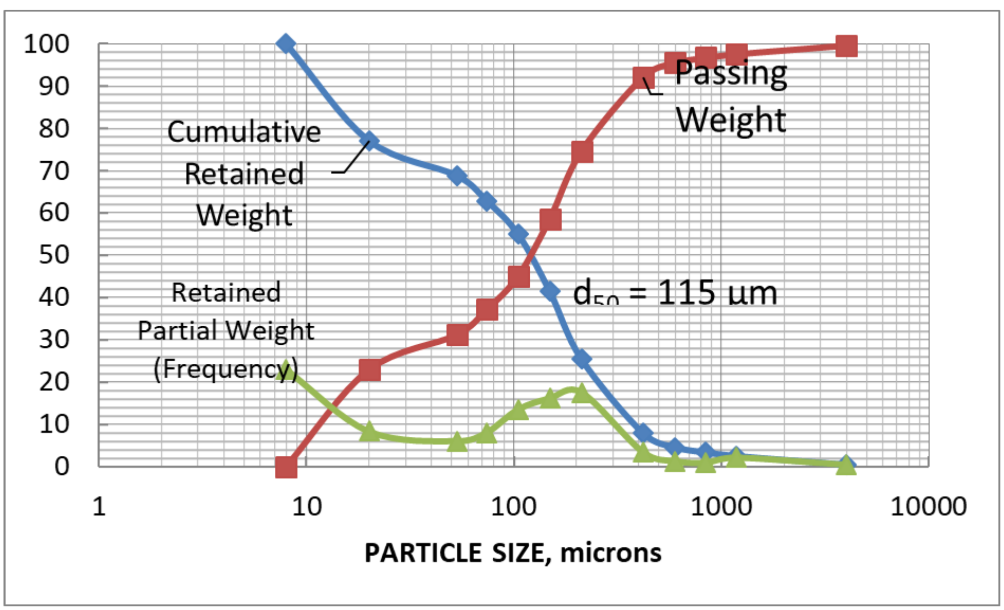
[1] Since sample was not dried, it is assumed that all sulfur comes from pyrite/marcasite, and sulfates are oxidation of the sulfur from sulfides. Tests performed on this sample may render only trend or indication of the potential improvement of the beneficiation process.

Figure 13-1: Cumulative Retained and Passing Particle Size Distribution for the Farim South Pit



Source: KEMWorks, 2019

Figure 13-2: Cumulative Retained and Passing Particle Size Distribution for the Farim North Pit



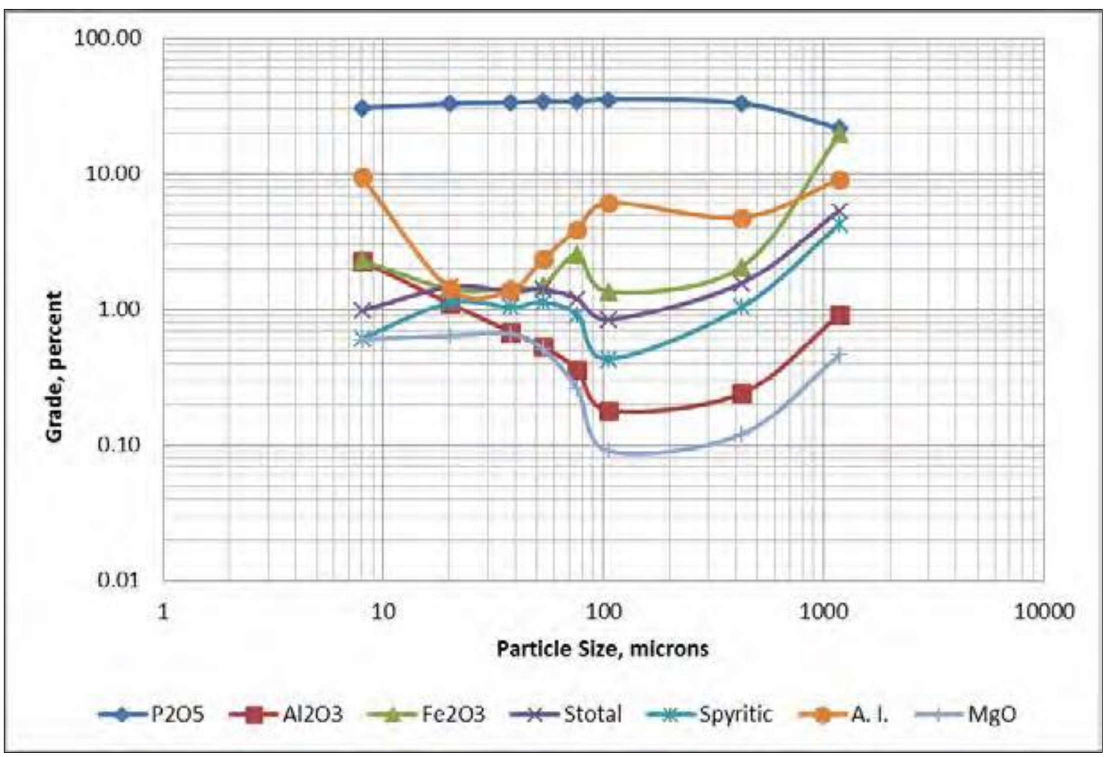
Source: KEMWorks, 2019

13.3.3 Screen Assays

The results of the screen assays for the Farim South pit phosphate ore are shown in Figure 13-3 to Figure 13-6. Figure 13-3 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 53 µm. A.I. is almost flat for particles larger than 106 µm and decreasing for particles smaller than 106 µm. The cumulative grades are presented in Figure 13-4 that shows this trend.

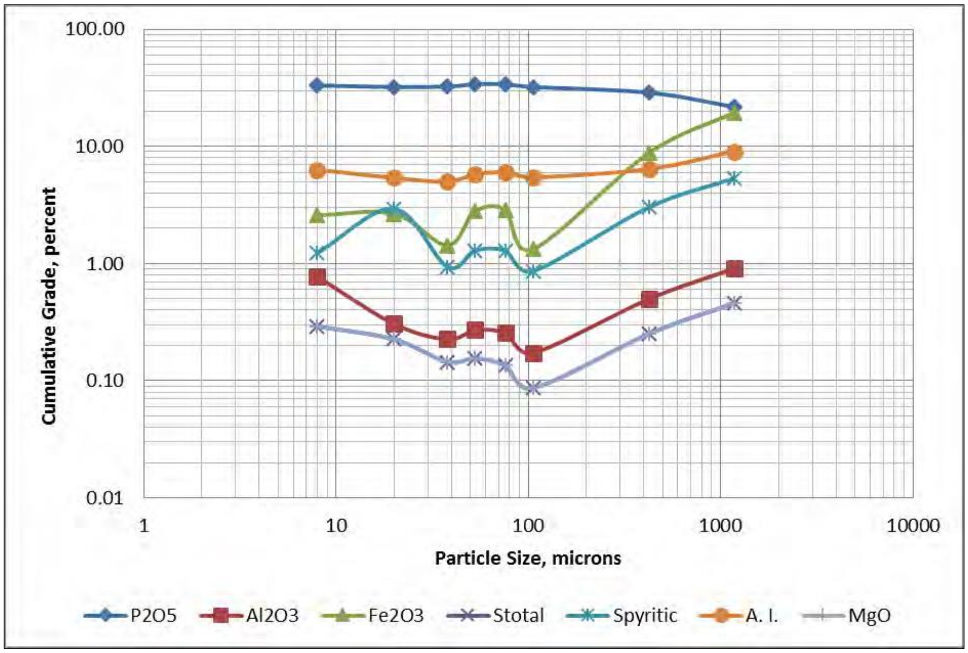
Figure 13-5 shows the frequency distribution for P_2O_5, A.I., Al_2O_3, Fe_2O_3, MgO, S_{total}, and $S_{pyritic}$ as a function of particle size for the South pit ore. This figure indicates that the Cumulative Weight Retained Distribution dominates this system; the variation in grades of the different compounds not being enough to modify the weight frequency distribution (Figure 13-1) significantly. Figure 13-6 shows the cumulative distribution for all compounds studied as a function of particle size for the South pit ore. The cumulative distribution of Fe_2O_3, S_{total}, and $S_{pyritic}$ shows that they accumulated in the +106 µm 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 13-3: Grades as a Function of Particle Size for Farim South Pit



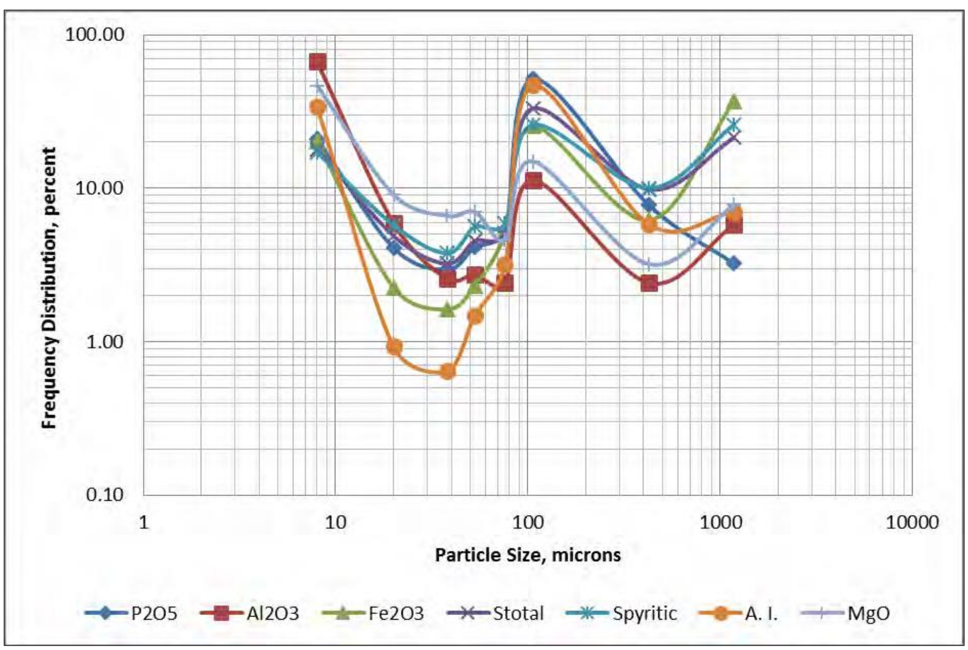
Source: KEMWorks, 2019

Figure 13-4: Cumulative Grades as a Function of Particle Size for Farim South Pit



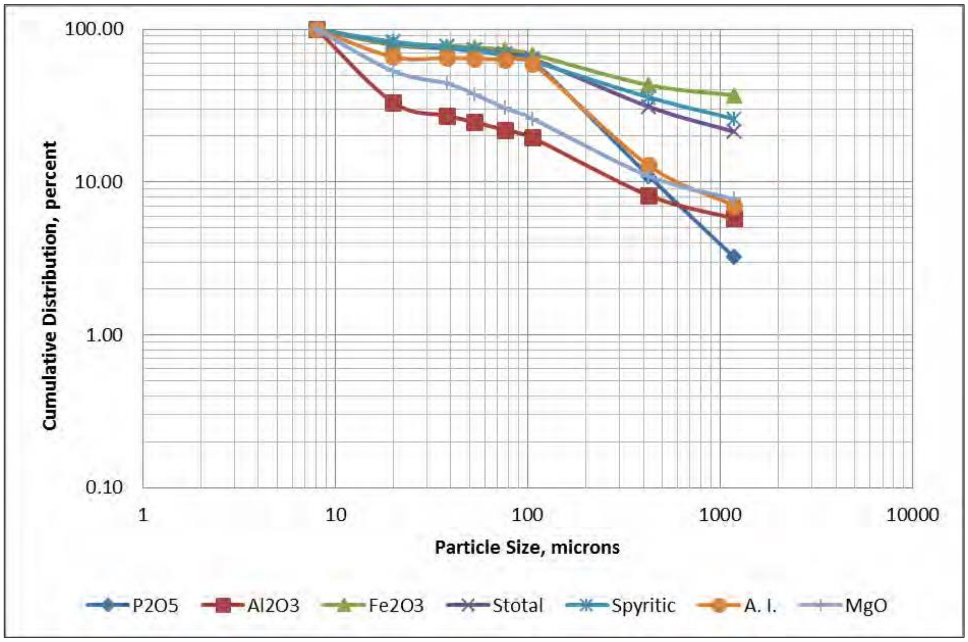
Source: KEMWorks, 2019

Figure 13-5: Frequency Distribution as a Function of Particle Size for Farim South Pit



Source: KEMWorks, 2019

Figure 13-6: Cumulative Distribution of as a Function of Particle Size for Farim South Pit



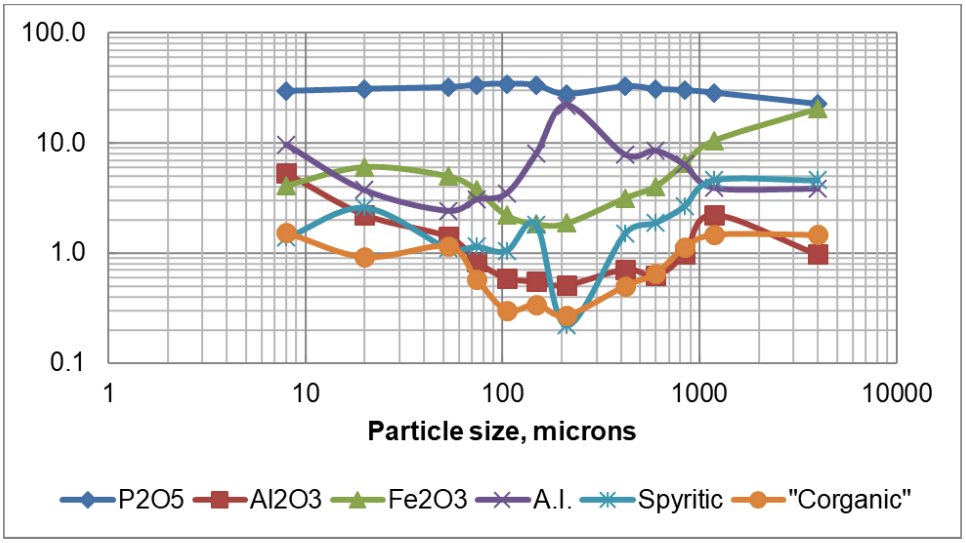
Source: KEMWorks, 2019

The screen assays for the Farim North pit are presented in Figures 13-7 to 13-10. Figure 13-7 shows the grade of P_2O_5, Al_2O_3, Fe_2O_3, A.I., $S_{pyritic}$, and $C_{organic}$ as a function of particle size for the North pit ore. The locus of the Grade curve for P_2O_5 clearly shows that at 212 μm a decrease in the P_2O_5 grade occurs. This is related to a maximum in the locus of the curve for the A.I. at this size fraction even though all other impurities decrease. The locus of the $S_{pyritic}$ grade follows that of the Fe_2O_3, showing a much shaper decrease in content at the 212 μm size fraction, probably an effect of chemical analysis deviation. The loci of the Al_2O_3 and $C_{organic}$ follow the same trend and to a lesser extent to that of the locus of the Fe_2O_3 indicating that $C_{organic}$ seems to be tied to clays and aluminum-iron silicates. Thus, the rejection of more clays and silicates by attrition scrubbing may result in lower $C_{organic}$ in the beneficiated product.

Figure 13-8 shows the cumulative grade of P_2O_5, Al_2O_3, Fe_2O_3, A.I., $S_{pyritic}$, and $C_{organic}$ as a function of particle size for the North pit ore. The trends described in Figure 13-7 are shown in Figure 13-8, but the loci of the curves are smoothed for all compounds analyzed. However, this figure clearly shows that the Al_2O_3, Fe_2O_3, and $S_{pyritic}$, follow the same trend, indicating that $C_{organic}$ may be trapped in the clays, silicates, and iron bearing minerals during the weathering and oxidation of the deposit.

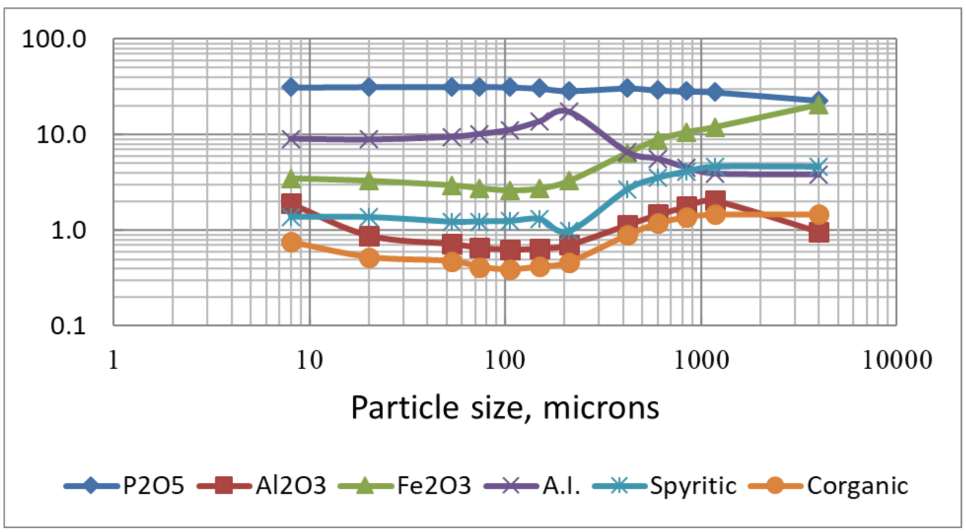
The frequency distribution for P_2O_5, Al_2O_3, Fe_2O_3, A.I., $S_{pyritic}$, and $C_{organic}$ as a function of particle size is presented in Figure 13-9. This figure shows that the distribution of values as a function of particle size is dominated by the retained weight distribution at each size fraction considered. This is clearly observed by comparing the locus of the frequency distribution of the material with the loci of the curves of the compounds studied in Figure 13-2. The cumulative distribution of P_2O_5, Al_2O_3, Fe_2O_3, A.I., $S_{pyritic}$, and $C_{organic}$ as a function of particle size is depicted in Figure 13-10. This figure shows that the cumulative distribution of the impurities followed the same trend, whereas, the cumulative distributions of P_2O_5 and A.I. present a different pattern, the locus of the A.I. Cumulative distribution emphasizing the high A.I. content in the 212 μm size fraction.

Figure 13-7: Grades as a Function of Particle Size for Farim North Pit



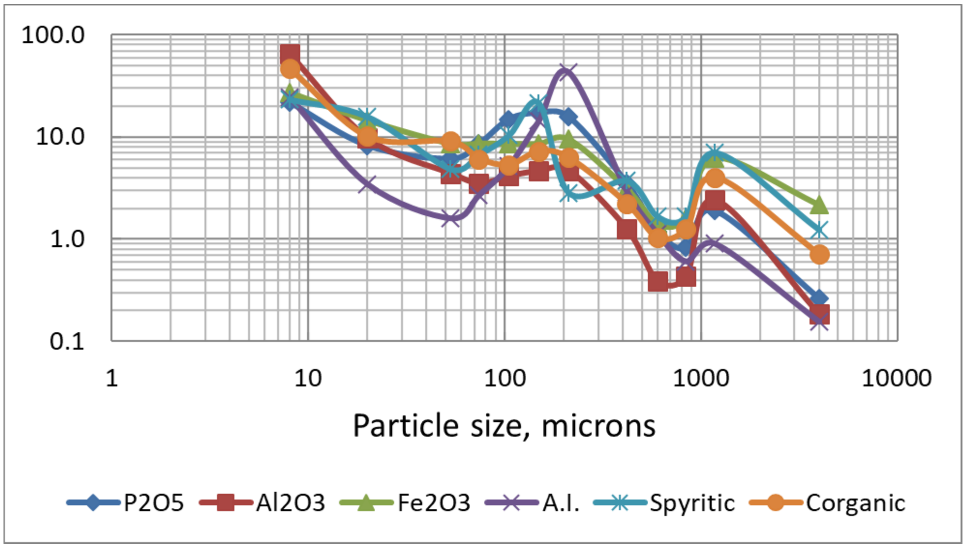
Source: KEMWorks, 2019

Figure 13-8: Cumulative Grades as a Function of Particle Size for Farim North Pit



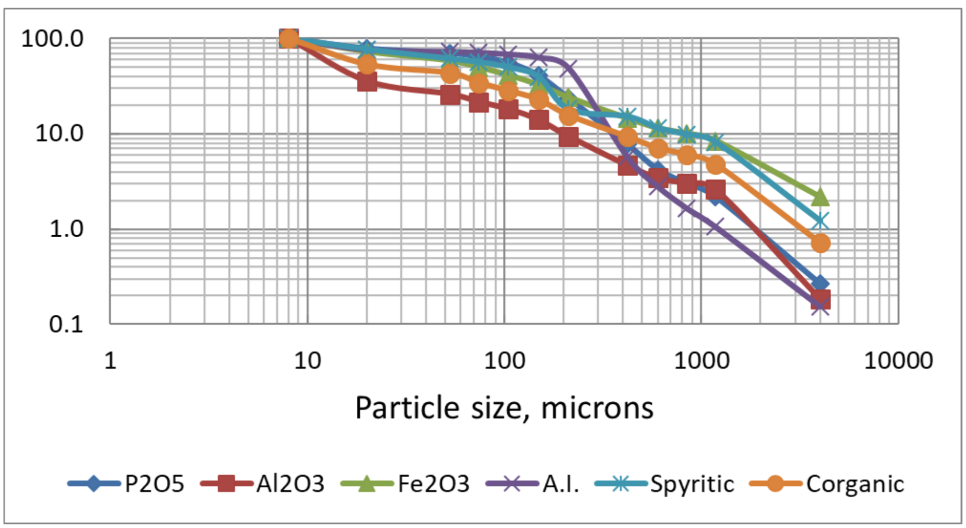
Source: KEMWorks, 2019

Figure 13-9: Frequency Distribution as a Function of Particle Size for Farim North Pit



Source: KEMWorks, 2019

Figure 13-10: Cumulative Distribution of as a Function of Particle Size for Farim North Pit



Source: KEMWorks, 2019

In summary, the screen assay results for the North pit phosphate ore showed similar trends as those showed by the South pit phosphate ore. In general, North pit material presented much higher A.I. grade than the South pit ore, A.I. being the main contaminant. It was observed that the +1180 µm size fraction, which is rejected, contained low A.I., but high Fe_2O_3, $S_{pyritic}$, Al_2O_3, and $C_{organic}$ resulting in lower P_2O_5 grade. The 1180 x 106 µm size fraction was seriously affected by the presence of large concentration of A.I. in the 420 x 212 µm size fraction (42.64% of the total A.I. content in the sample), but low $S_{pyritic}$, and $C_{organic}$. This resulted in high A.I. grade (21.96%) and low P_2O_5 grade (27.77%). Furthermore, it should be taken into consideration that the retained weight in this size fraction was 17.52% of the total material fed to the screen assays. Consequently, the 1180 x 106 µm size fraction (equivalent to the coarse concentrate) only achieved 32.87% P_2O_5 with 11.48% A.I., 0.65% Al_2O_3, 2.16% Fe2O3, 1.08% $S_{pyritic}$, and 0.34% $C_{organic}$. In the case of the fine fraction, 106 x 20 µm, (equivalent to the fine concentrate) the screen assays showed that the main contaminants were Fe_2O_3, Al_2O_3, $S_{pyritic}$, and $C_{organic}$, analyzing 1.51% Al_2O_3, 4.95% Fe_2O_3, 1.68% $S_{pyritic}$, and 0.80% $C_{organic}$, and rendering a 106 x 20 µm size fraction P_2O_5 grade of 32.15%, the average product (1180 x 20 µm size fraction) reporting 32.65% P_2O_5, which is lower than the required 34% P_2O_5.

13.3.4 South Pit 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 sulfides.

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

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

13.3.4.1 Mass Distributions and Elemental Chemical Data

The mass distributions and elemental chemical data by size fraction are summarized in Table 13-10. 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 13-10: Size Fractions for Analysis and Mass Distribution (%) of the Farim Composite

Fraction	Combined	+1180 µm	- 1180 / +425 µm	-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

13.3.4.2 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. These correlate well with the higher iron assay in this fraction.
  - Pyrite content is also highest in the +1,180 µm fraction and correlate well with the sulfur 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, but it can also contain low levels of potassium and magnesium. This phase mainly occurs in the -20 µm fraction accounting for 48.9%.

13.3.4.3 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 revealed that apatite contains significant SO_2 and fluorine at ~0.65% and 4.72%, respectively.

13.3.4.4 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 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 sulfide are most abundant in the +1,180 µm size fraction and show poor liberation.

13.3.4.5 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 for Farim South Pit Phosphate Ore

Based on the interpretation of the characterization studies results for the Farim South pit 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. The horizontal scrubbing tests were then 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 +6,300 µm size fraction is screened out (reject), dried and weighed; and the -6,300 µm 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 1,180 µm, 425 µm, 106 µm and 20 µm to obtain the 6,300 x 1,180 µm, 1,180 x 425 µm, 425 x 106 µm, 106 x 20 µm, and -20 µm size fractions.

Even though the "standard baseline scrubbing test" outlined above proved not ideal for the Farim composite, the results indicated that an adjustment to the standard test would be suitable for the Farim composite. Using the screen assay of the new scrubbed product, it was clear that by rejecting the +1,180 µm material and the -20 µm size fraction, the highest P_2O_5 and CaO grades were obtained with the lowest level of impurities except for the A.I. (see Figure 13-11 and Figure 13-12).

Figure 13-13 and Figure 13-14 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

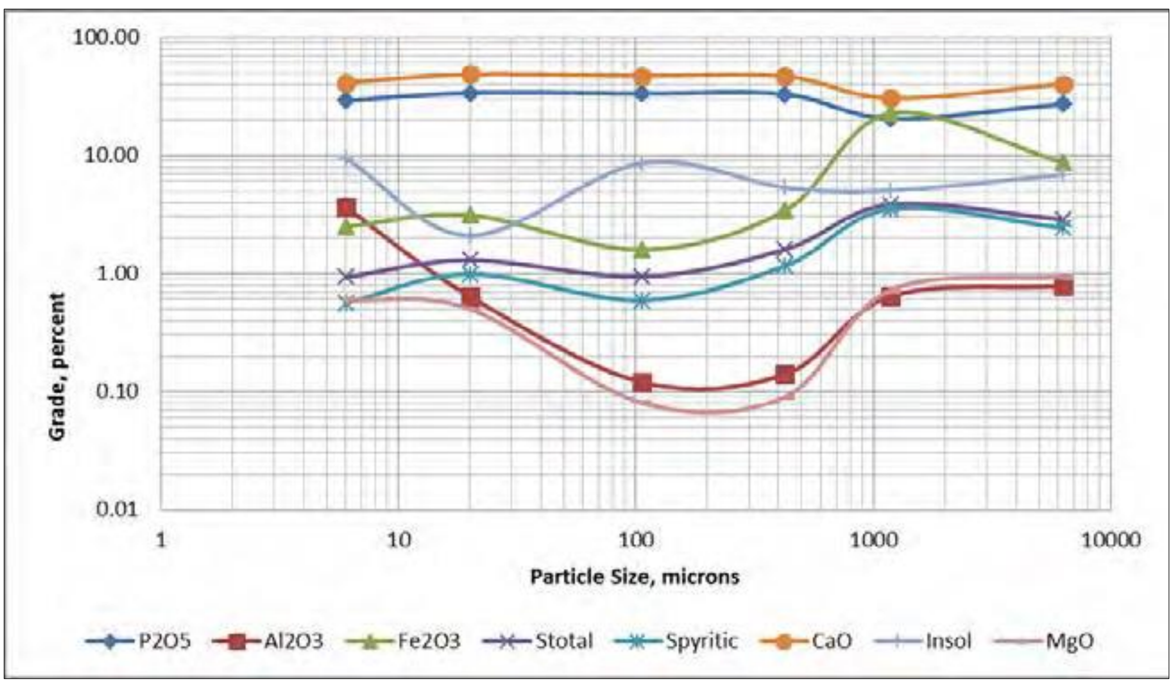
1,180 µm and below the -20 µm size fractions. Figure 13-14 shows lower cumulative recoveries of Al_2O_3, P_2O_5, and A.I. above 1,180 µm (reject), but higher recoveries between 1,180 µm and 20 µm.

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%

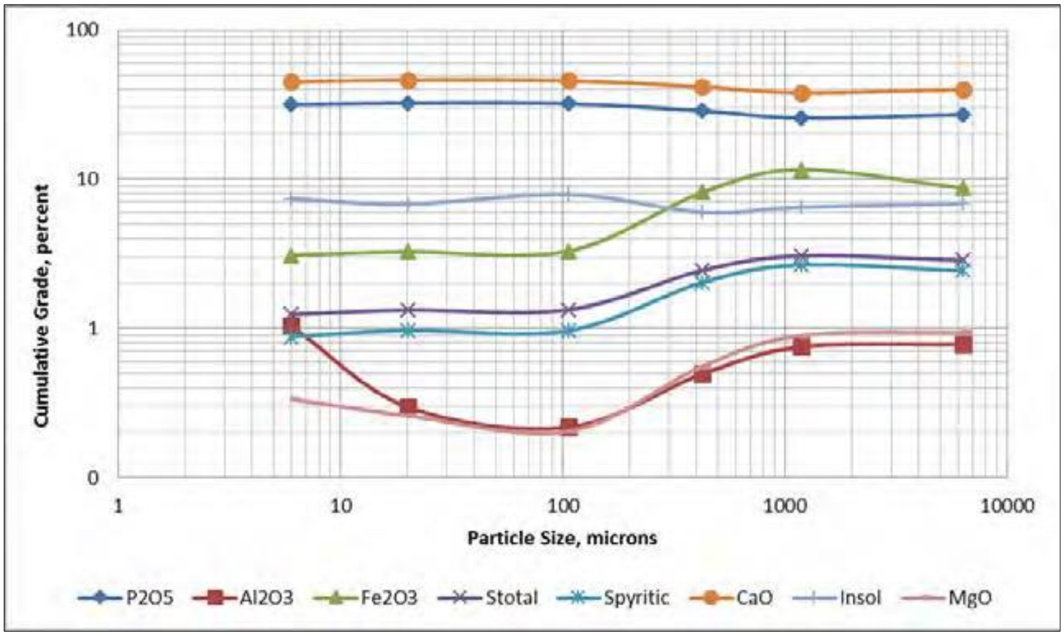
The presence of large amount of clay material in the ore results in a cushioning effect and a high viscosity of slurry in the scrubbing stages. It was cautiously inferred that by horizontal scrubbing under the right conditions, then desliming at 75 µm followed by attrition scrubbing, the 1,180x75 µm size fraction would result in a higher P_2O_5 grade and recovery. However, the presence of high A.I. in the 425 x 106 µm size fraction was also considered 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 13-11: Grades as a Function of Particle Size after Baseline Horizontal Scrubbing



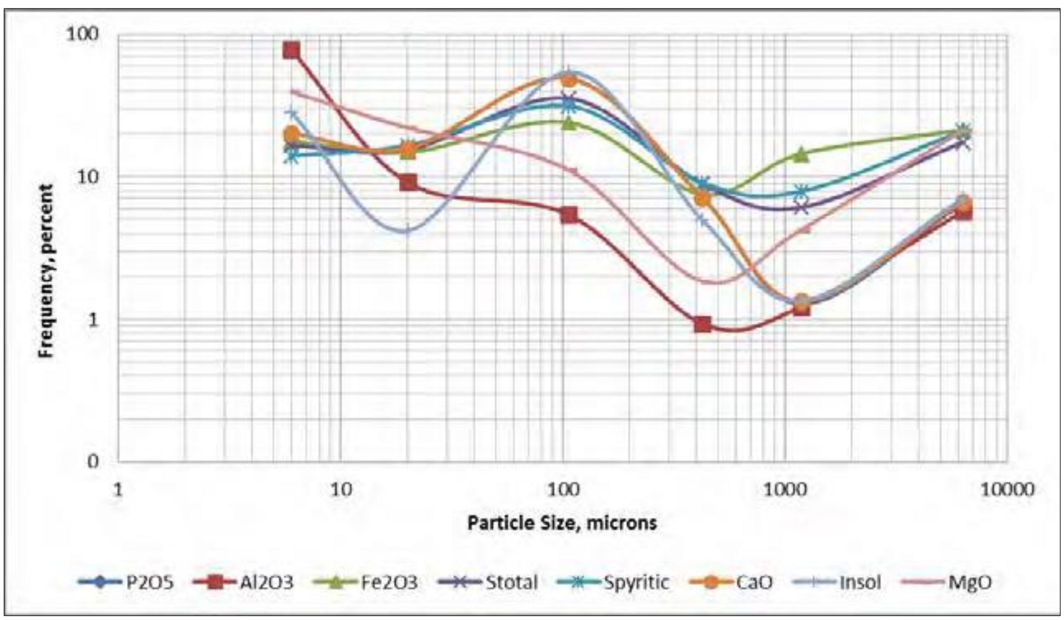
Source: KEMWorks, 2019

Figure 13-12: Cumulative Grades as a Function of Particle Size after Baseline Horizontal Scrubbing



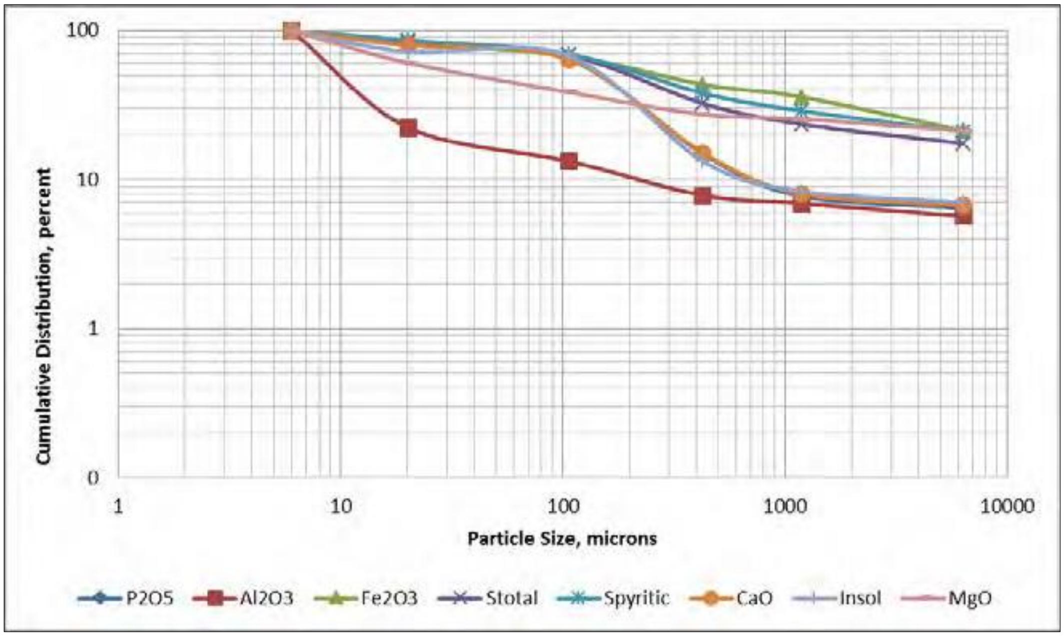
Source: KEMWorks, 2019

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



Source: KEMWorks, 2019

Figure 13-14: Cumulative Distribution as a Function of Particle Size after Baseline Horizontal Scrubbing



Source: KEMWorks, 2019

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 1,180 x 20 µm 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 +6,000 µm and +1,180 µm 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 -20 µm 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.

Figure 13-15 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).

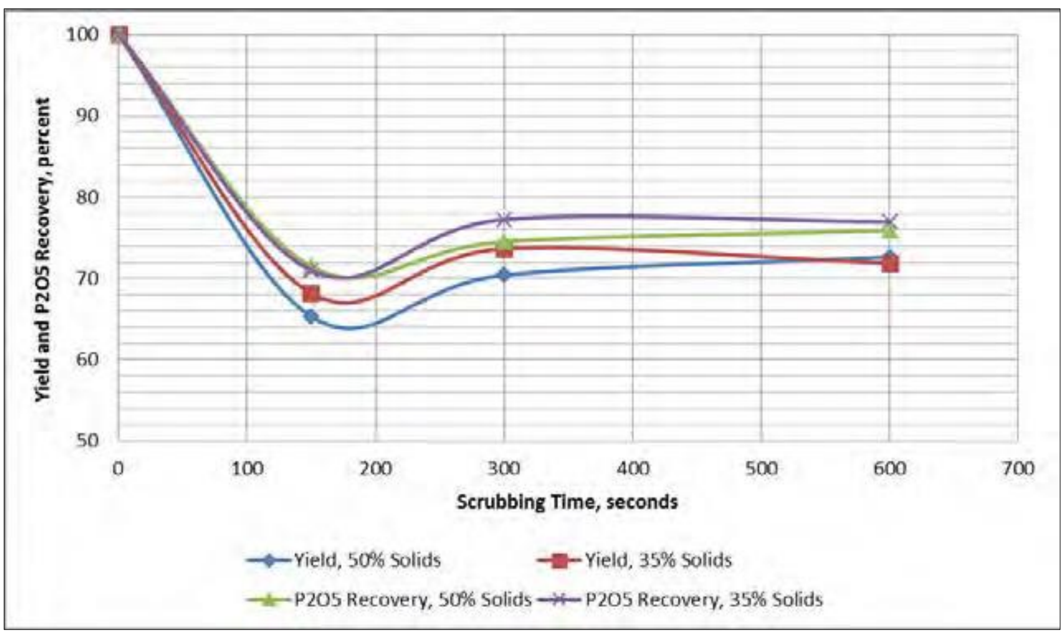
The P_2O_5 grade, grade potential and the A.I. grade as a function of scrubbing time is presented in Figure 13-16 for 35% and 50% solids content. Again, the results show that high P_2O_5 grade and grade potential are obtained at 35% solids content and 300 seconds (5 minutes) of horizontal scrubbing time. 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 13-17 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 13-18 and Figure 13-19.

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 13-18). However, Figure 13-19 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.

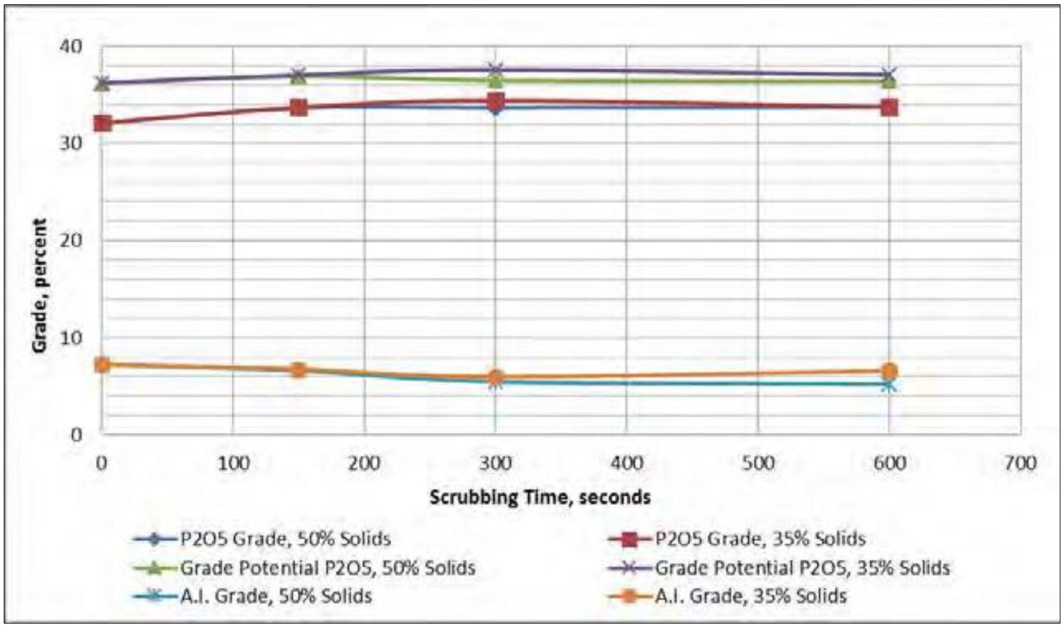
In summary, the results show that horizontal scrubbing at 35% solids content for 300 seconds (5 minutes) renders the highest mass yield of 73.7%, the highest P_2O_5 grade of 34.4% and the highest P_2O_5 recovery of 77.3%. As a result, the operating conditions for the horizontal scrubbing stage in the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

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



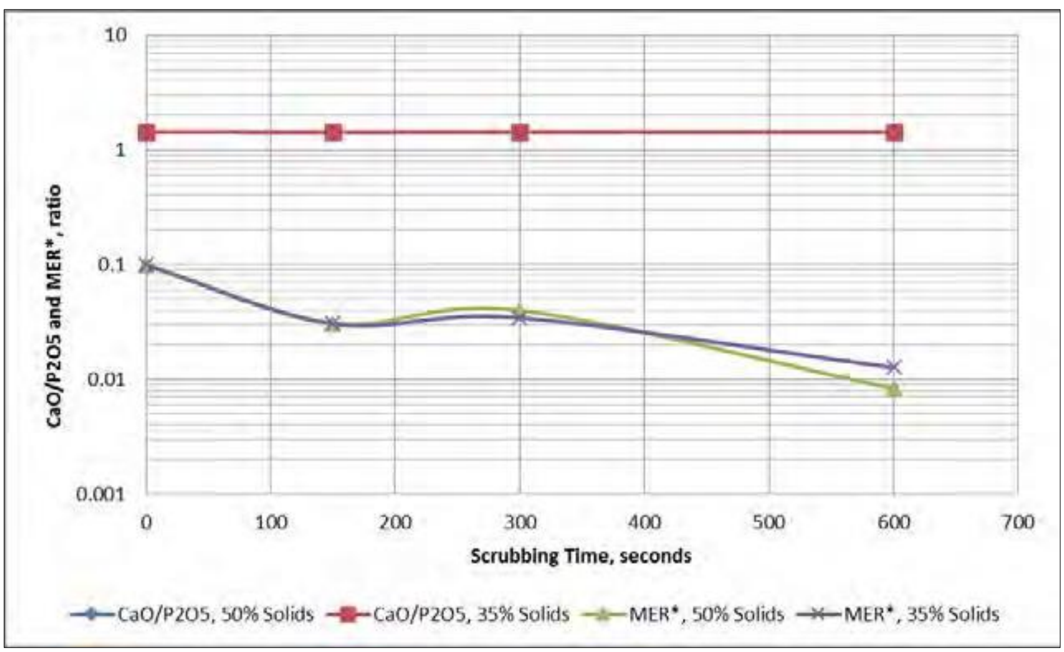
Source: KEMWorks, 2019

Figure 13-16: Grades as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content



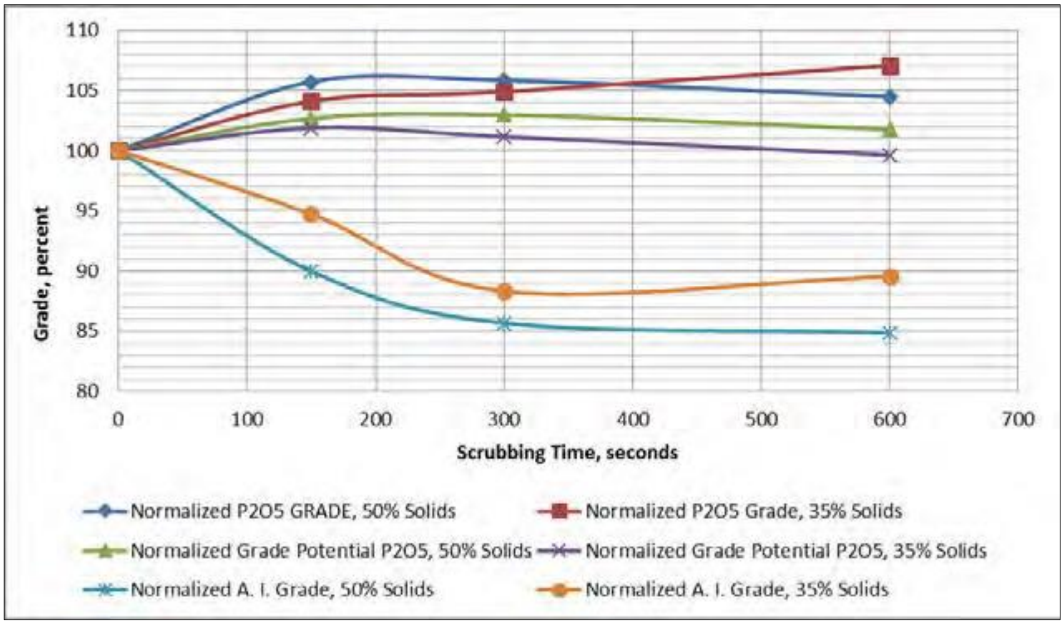
Source: KEMWorks, 2019

Figure 13-17: CaO/ P2O5 Ratio and MER* as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content



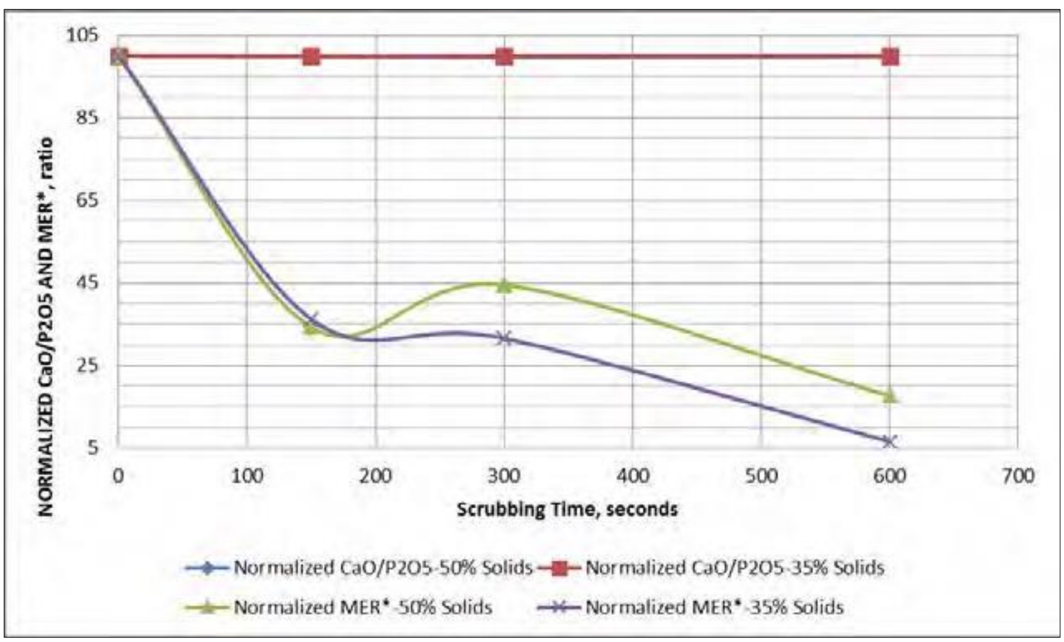
Source: KEMWorks, 2019

Figure 13-18: Normalized Grades as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content



Source: KEMWorks, 2019

Figure 13-19: Normalized CaO/ P_2O_5 Ratio and MER* as a Function of Horizontal Scrubbing Time at 35% and 50% Solids Content



Source: KEMWorks, 2019

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).

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 1,180 x 20 µm size fraction was product, the 6,300 x1180 µm was considered reject, the 75 x 20 µm size fraction part of the fine product, and the material finer than 20 µm was considered slimes (tailings). The results obtained from Test HS #7 Confirmation Test for the 1,180 x 20 µm size fraction is summarized in Table 13-11.

Test HS #5 is also included in this table for comparison. The data in Table 13-11 and Table 13-12 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 1,180 x 20 µm product.

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

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.

Table 13-11: Wet Horizontal Scrubbing Confirmation Test 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, %	Stotal, %	Spyritic, %	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 13-12: Normalized Wet Horizontal Confirmation Test 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, %	Stotal, %	Spyritic, %	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

13.5 Attrition Scrubbing Studies for Farim South Pit Phosphate Ore

13.5.1 Introduction

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 6,300 x 75 µm 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 6,300 x 75 µm 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 -20 µm size fraction
- selectively release the ultra-fine quartz particles into the -20 µm size fraction
- reduce the quartz content (A.I.) in the 1,180 x 106 µm and 106 x 20 µm size fractions.

According to the QEMSCAN and mineralogical analyses, the quartz rejection into the -20 µm 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 1,180 x 106 µm and 106 x 020 µm 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 -20 µm 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.

13.5.2 Effect of Attrition Scrubbing Time for Three Different Solids Contents

Nine attrition scrubbing tests were carried out to investigate the effect of on the 6,300 x 75 µm 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 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:

- +1,180 µm is rejected as oversize
- 1,180 x 106 µm becomes flotation feed

- 106 x 20 µm becomes fine concentrate
- -20 µm is rejected slimes.

The results show that depending on the attrition scrubbing conditions, Al_2O_3, Fe_2O_3, S_{total}, $S_{pyritic}$, and MgO decreased in the 1,180 x 20 µm size range, but the A.I. increased in the 1,180 x 106 µm range and decreased in the 106 x 20 µm 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 (except for 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 13-13 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 confirmed these conclusions.

Tests carried out at 55% solids content demonstrated the same trend in impurities, and a similar recovery of P_2O_5 and yield was 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 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 13-13: Effect of % Solids in Attrition Scrubbing

Description	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.

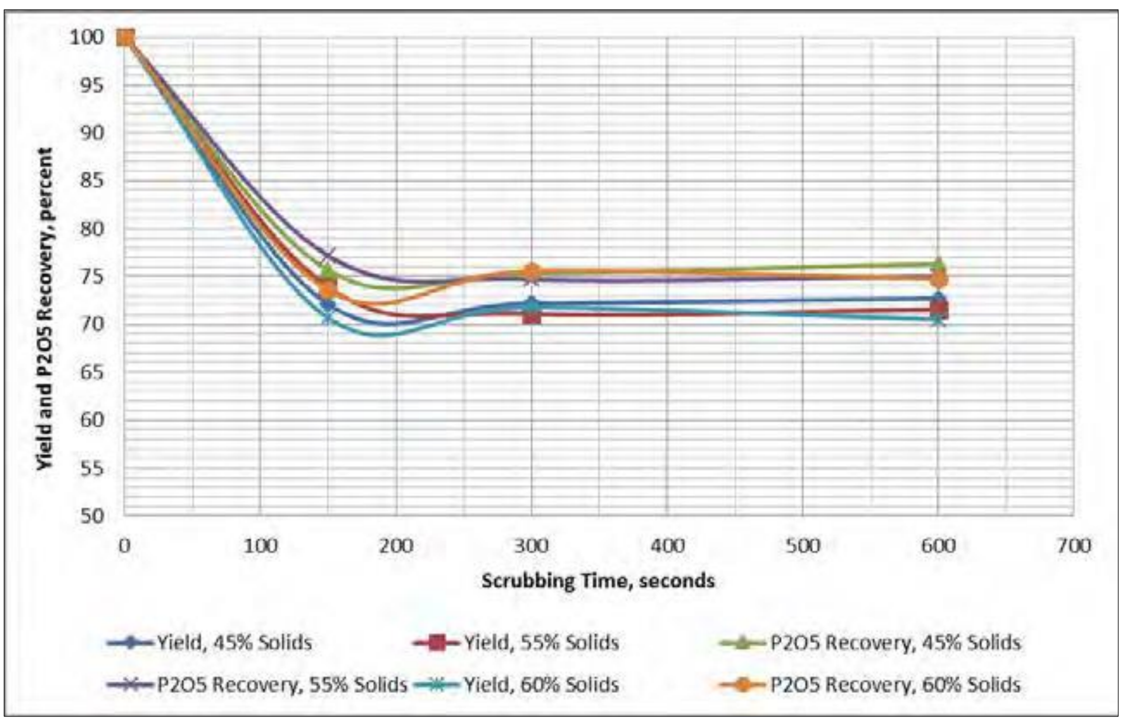
Figure 13-20 presents the mass yield and P_2O_5 recovery as a function of scrubbing time for the three solids contents evaluated: 45%, 55%, and 60%. This plot clearly shows that the highest yield and P_2O_5 recovery is obtained after scrubbing for only 150 seconds (2.5 minutes) at 55% solids content. The mass yield and P_2O_5 recovery levels off as the scrubbing time is increased for all solids content studied.

Figure 13-21 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 were 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 increases 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 13-22. 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.

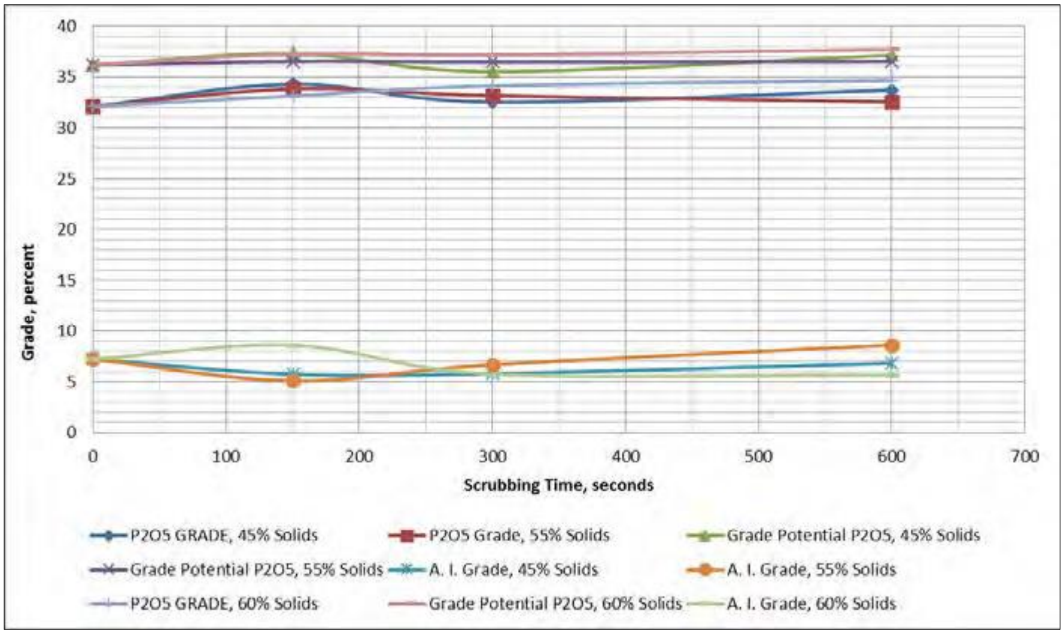
The normalized data as a function of scrubbing time for 45%, 55%, and 60% solids content are presented in Figure 13-23 and Figure 13-24. These figures further show the same trends observed for the actual results, indicating that the P_2O_5 feed grade variations are not significant for these tests.

Figure 13-20: Yield and P_2O_5 Recovery as a Function of Attrition Scrubbing Time



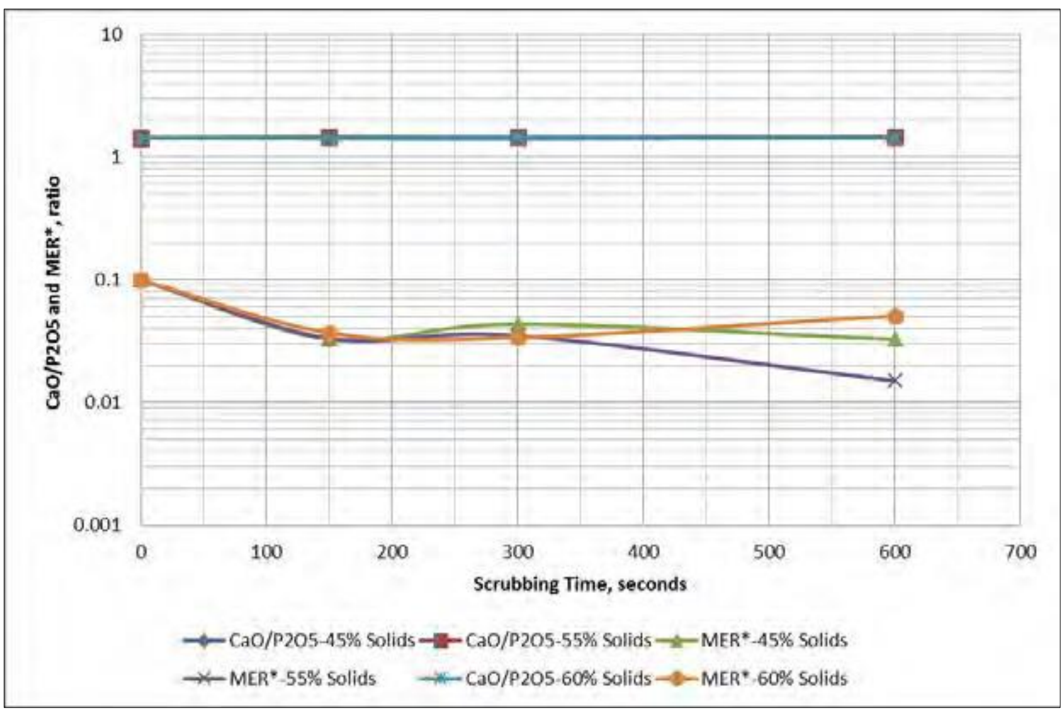
Source: KEMWorks, 2019

Figure 13-21: P_2O_5 Grade and Potential Grade, and A.I. Grade as a Function of Attrition Scrubbing Time



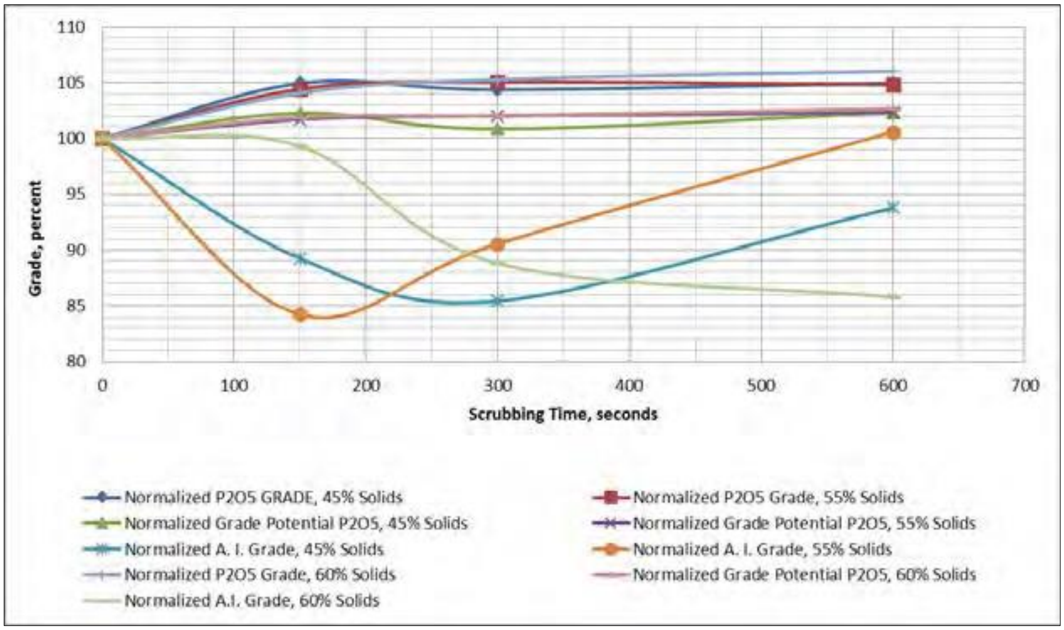
Source: KEMWorks, 2019

Figure 13-22: CaO/ P_2O_5 Ratio and MER* Parameters as a Function of Attrition Scrubbing Time



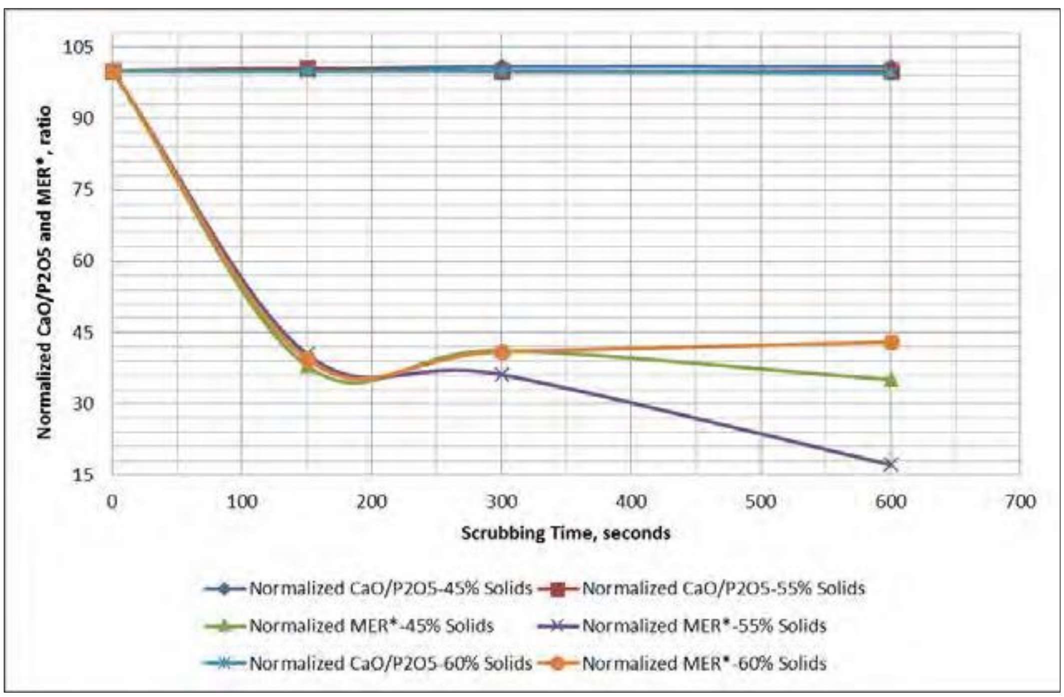
Source: KEMWorks, 2019

Figure 13-23: Normalized Grade as a Function of Attrition Scrubbing Time



Source: KEMWorks, 2019

Figure 13-24: Normalized CaO/ P_2O_5 Ratio and MER* as a Function of Attrition Scrubbing Time



Source: KEMWorks, 2019

13.6 South Pit Reverse Amine Flotation Studies on the 1.18 x 0.106-mm Size Fraction

Seven tests were carried out to determine the flotation operating conditions for the 1,180 x 75 µm 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 +6,300- µm size fraction that was considered reject. The remaining material was screened at 75 µm to remove fines and clays before attrition scrubbing. Then, the 6,300 x 75 µm size fraction was submitted to attrition scrubbing at 55% solids content for 150 seconds (2.5 minutes). This scrubbed material was then screened at 1,180 µm where the 6,300 x 1,180 µm size fraction was considered reject. The remaining 1,180 x 75 µm size fraction was then screened at 106 µm. Two products were obtained here, the 1,180 x 106 µm size fraction, which constitutes the amine reverse flotation feed, and the -106 µm size fraction. This -106 µm size fraction and the -75 µm size fraction removed after horizontal scrubbing were combined and deslimed again at 20 µm to produce the 106 x 20 µm concentrate product which considered the fine concentrate. The -20 µm size fraction was rejected as slimes.

13.6.2 Flotation Results

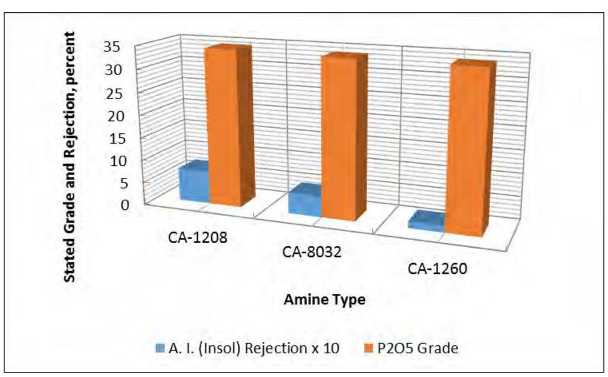
The individual flotation test data and the metallurgical balance of the process developed for the Farim composite phosphate ore are presented in this section. Flotation tests of the prepared feed were carried out to select the most efficient of three condensate amines provided by ArrMaz Custom Chemicals 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 1,180 x 106 µm concentrate.

13.6.3 Amine Selection

To determine which of the three condensate amines was best suitable for the 1,180 x 106 µm 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 13-25 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 13-25: Effect of Amine Type at 0.23 kg/ton of Amine Addition



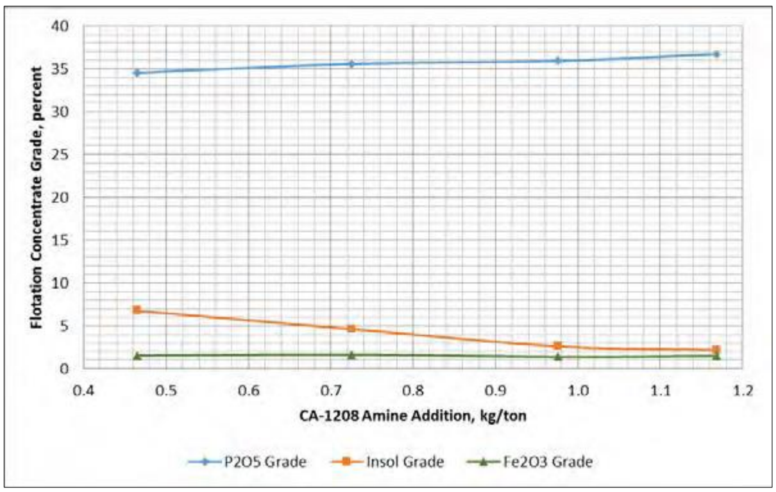
Source: KEMWorks, 2019

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.

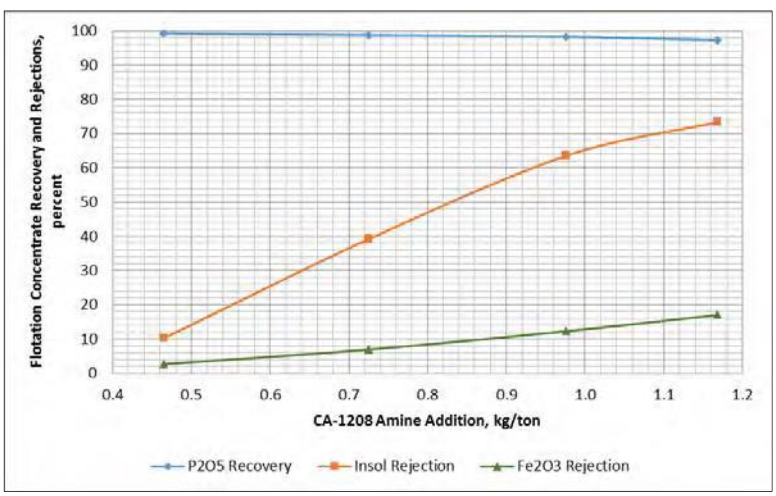
The P_2O_5 grade, A.I. grade, and Fe_2O_3 grade in the concentrate as a function of CA-1208 amine addition are presented in Figure 13-26. The best results are obtained with the addition of 1.168 kg CA-1208 amine per ton of flotation feed. The concentrate reports 36.7% P_2O_5 with 2.2% A.I. and 1.5% Fe_2O_3. Figure 13-27 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 of the flotation feed in the concentrate while rejecting 73.4% of the A.I. and 17.0% of the Fe_2O_3.

Figure 13-26: Grades as a Function of CA-1208 Amine Addition



Source: KEMWorks, 2019

Figure 13-27: P_2O_5 Recovery, A.I. and Fe2O3 Rejections as a Function of CA-1208 Amine Addition



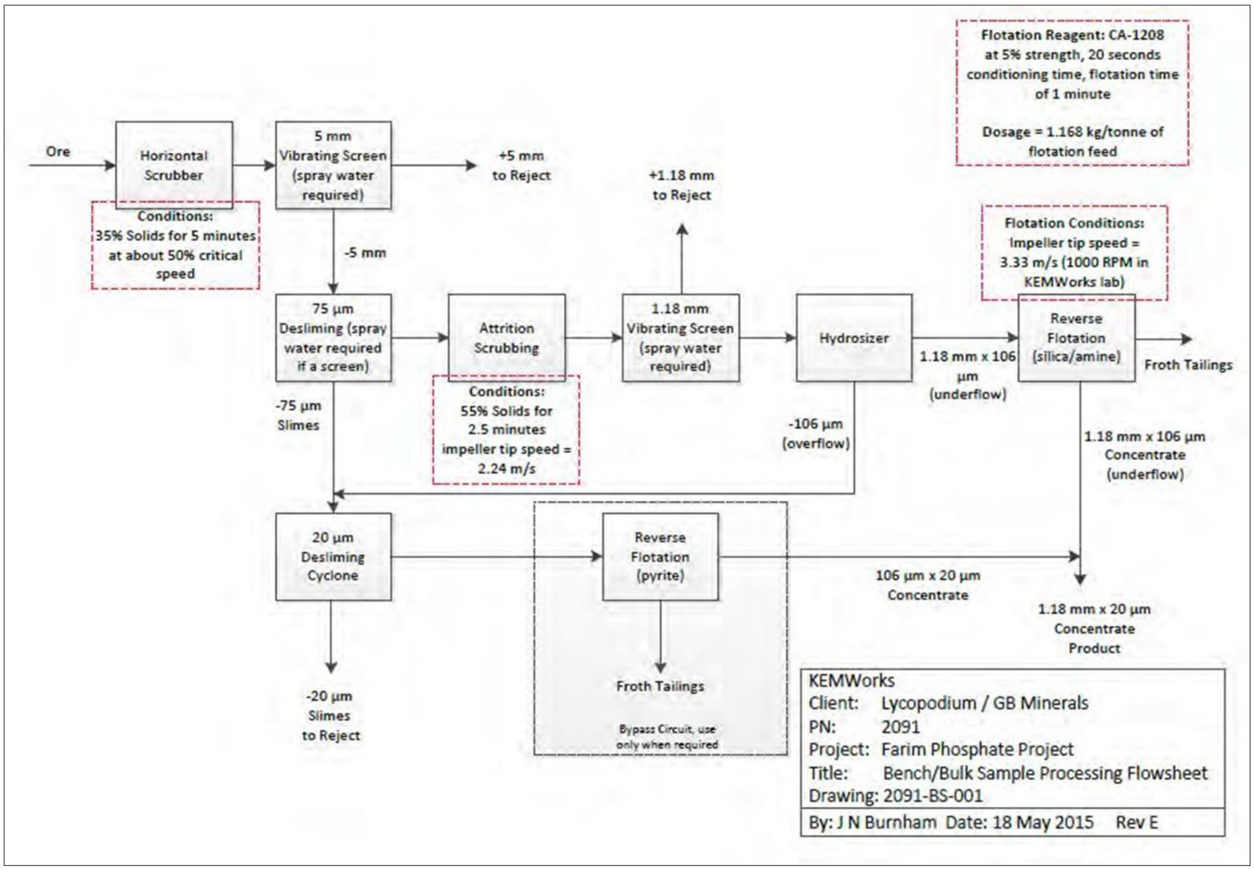
Source: KEMWorks, 2019

13.7 Metallurgical Balance from KEMWorks Bench Scale Testwork

13.7.1 Scrubbing and Flotation for Farim South Pit Phosphate Ore

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 13-28. This diagram summarizes the experimental procedure delineated above and required process conditions, and it is the basis for the process flowsheet.

Figure 13-28: Process Block Flow Diagram for the Farim Composite Sample



Source: KEMWorks, 2015

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 13-14. This table shows that 5.8% of the feed is rejected in the +6,300 µm size fraction and 2.1% of the feed is rejected in the 6,300 x 1,180 µm-size fraction. The total slimes (-20 µm material) reported were 21.6% of the feed. Table 13-13 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.

Table 13-14: Metallurgical Balance for the Farim Composite Sample

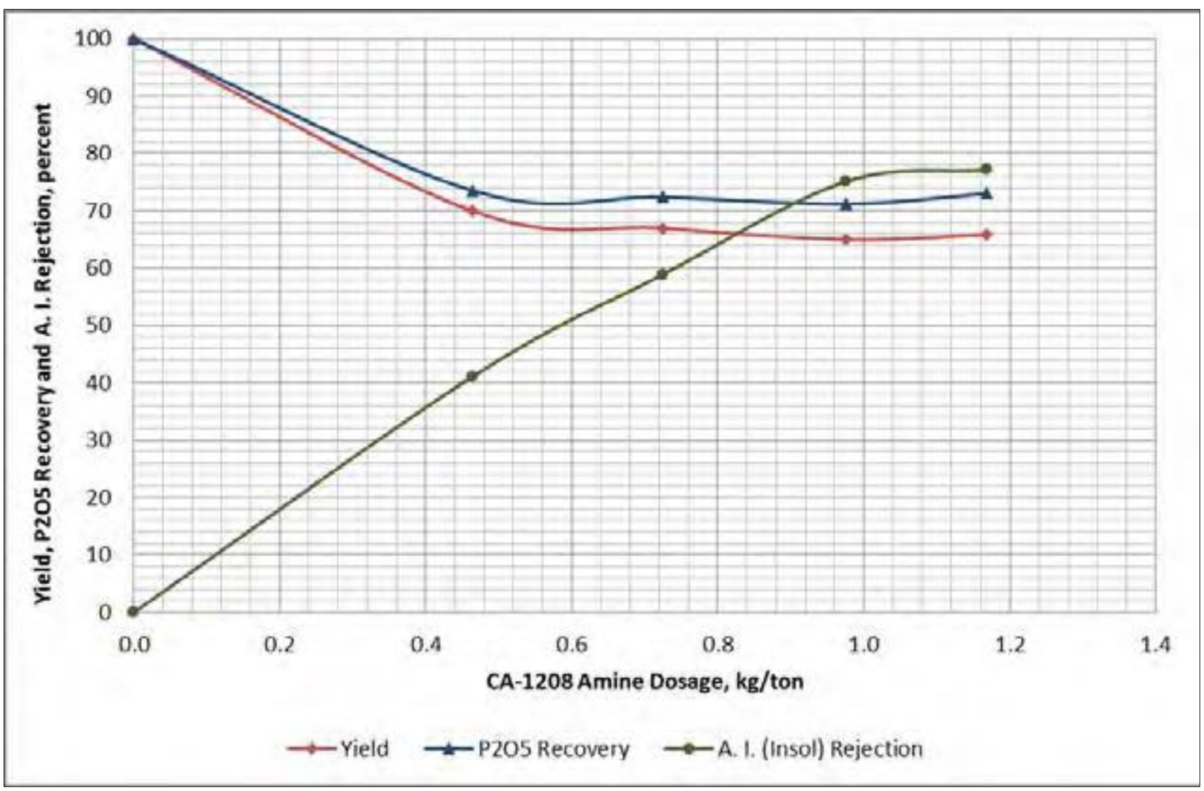
Products	Opening, µm	Retained Wt., g	Retained Wt., %	Cum. Reta. Wt., %	Passing Wt., %	Grades					Cumulative Grades					Distribution					Cumulative Distribution-Products					Products
						P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %	P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %	P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %	P_2O_5, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Insol, %	
Rejects	6300	27.6	5.81	5.81	94.19	26.82	0.42	1.13	9.87	7.73	26.82	0.42	1.13	9.87	7.73	4.82	8.81	5.67	19.69	6.01	4.82	8.81	5.67	19.69	6.01	
Rejects	1180	9.8	2.06	7.87	92.13	18.85	0.49	0.66	20.52	5.37	24.73	0.44	1.01	12.66	7.11	1.20	3.65	1.17	14.54	1.48	6.02	12.46	6.84	34.23	7.49	Rejects
Flot Con	106	234.3	49.32	57.19	42.81	36.7	0.076	0.164	1.48	2.2	35.05	0.13	0.28	3.02	2.88	55.94	13.53	6.98	25.07	14.52	55.94	13.53	6.98	25.07	14.52	Flot Con
Flot Tails	106	22.5	4.74	61.92	38.08	10.6	0.097	0.199	3.16	63.1	33.18	0.12	0.27	3.03	7.48	1.55	1.66	0.81	5.14	40.00	1.55	1.66	0.81	5.14	40.00	Flot Tails
Fine Con	20	78.4	16.50	78.43	21.57	33.45	0.43	0.83	3.11	3.76	33.24	0.19	0.39	3.05	6.70	17.06	25.62	11.82	17.63	8.30	17.06	25.62	11.82	17.63	8.30	Fine Con
Slimes	6	102.5	21.57	100.00	0.00	29.14	0.6	3.95	2.42	10.28	32.35	0.28	1.16	2.91	7.47	19.43	46.73	73.55	17.93	29.68	19.43	46.73	73.55	17.93	29.68	Slimes
Total		475.10	100.00			32.35	0.28	1.16	2.91	7.47						100.00	100.00	100.00	100.00	100.00						

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. Figure 13-29 presents the yield, P_2O_5 recovery and A.I. rejection as a function of CA-1208 amine addition. 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 13-30. 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 13-28 results in an average feed mass distribution of:

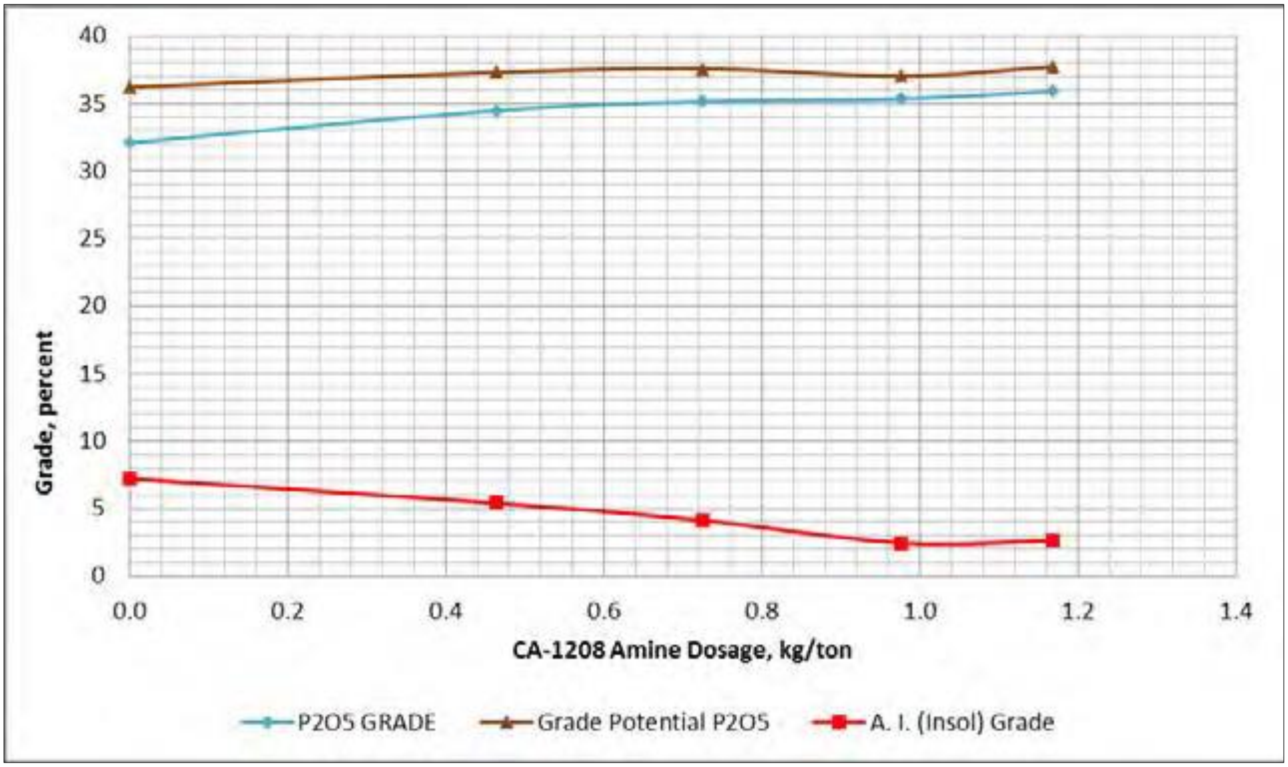
- 6,300 µm rejection.. 5.2% ± 1.9%
- 6,300 x 1,180 µm rejection.. 2.2% ± 0.2%
- 1,180 x 106 µm flotation concentrate.. 49.3% ± 2.8%
- reverse flotation tailings.. 4.7% ± 1.7%
- 0.10 6x 020 µm fine concentrate.. 16.6% ± 0.5%
- 20 µm slimes rejection.. 21.9% ± 0.3%

Figure 13-29: Yield, P_2O_5 Recovery, and A.I. Rejection as a Function of CA-1208 Amine Addition



Source: KEMWorks, 2019

Figure 13-30: P_2O_5 Grade and Grade Potential, and A.I. Grade as a Function of CA-1208 Amine Addition



Source: KEMWorks, 2019

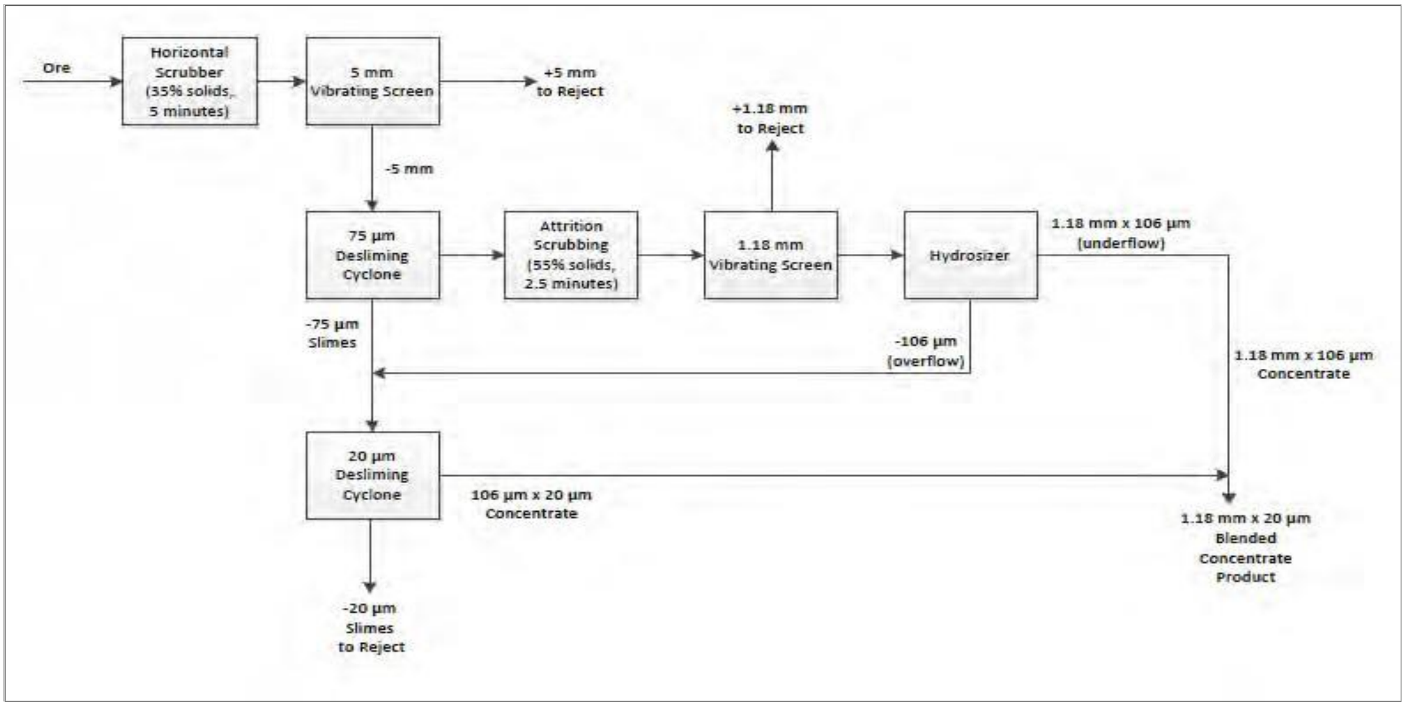
13.7.2 Only Scrubbing for Farim South Pit Phosphate Ore

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 13-31. This diagram summarizes the experimental procedure delineated above and required process conditions and it is the basis for the process flowsheet.

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 13-15. This table shows that 2.0% of the feed is rejected in the +6300 µm size fraction and 2.2% of the feed is rejected in the 6,300 x 1,180 µm size fraction. The total slimes (-20 µm material) reported were 21.9% of the feed. Table 13-15 also 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.

Figure 13-31: Process Block Flow Diagram for the Farim Composite Sample



Source: KEMWorks, 2015

Table 13-15: Bench Scale Metallurgical Balance for the Farim Composite Sample

Product Designation	Opening	Weight %	P2O5, %	Insol, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Pyritic 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

The summary of the bench scale metallurgical results are as follows:

- Mass yield.. 73.9%
- P_2O_5 recovery... 77.2%
- CaO/ P_2O_5 Ratio.. 1.45
- MER... 0.075
- MER*... 0.070
- P_2O_5 grade... 33.8%

13.7.3 Only Scrubbing for Farim North Pit Phosphate Ore

The application of the beneficiation process designed for the South pit was carried out in triplicate to determine the reproducibility, and average values of the product to be obtained for the North pit composite. In general, all three tests were <1% error of each other, the results of these tests being presented in this section. As an example, Test 2 results were shown in Figures 13-32 to Figure 13-35.

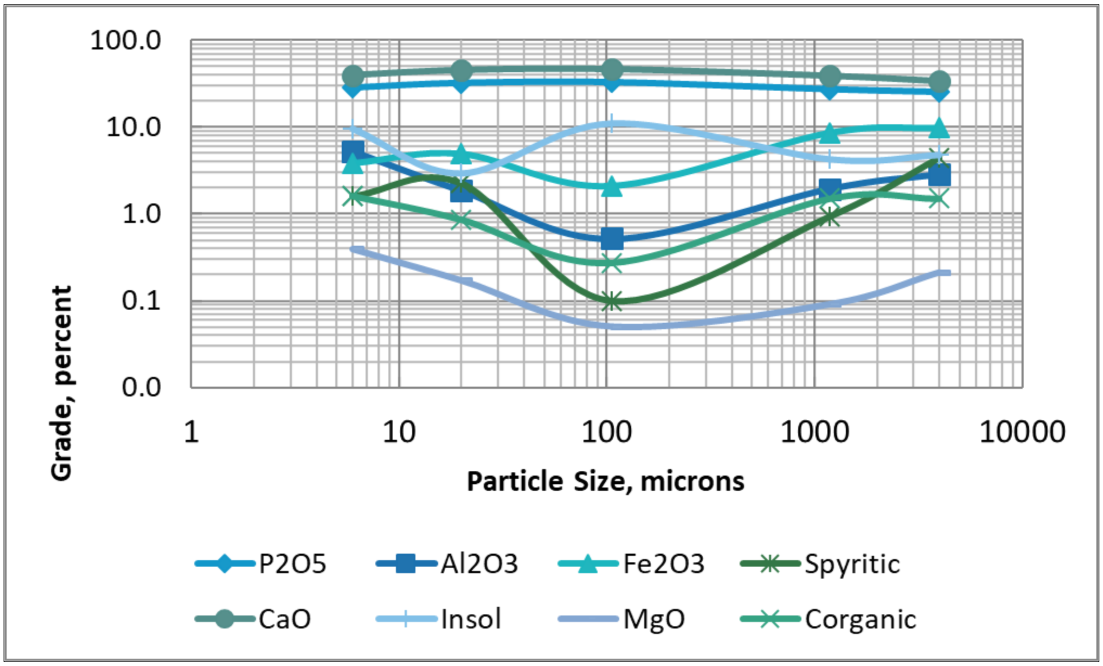
Figure 13-32 depicts the grades of P_2O_5, Al_2O_3, Fe_2O_3, A.I., CaO, MgO, $S_{pyritic}$, and $C_{organic}$ as a function of particle size. As predicted by the screen assays results, this chart shows that the P_2O_5 and CaO grades increased in the particle size range of 1180 x 20 µm along with the increase in A.I. because of the A.I. concentration in 420 x 212 µm size fraction. As expected, the grade of impurities, Al_2O_3, Fe_2O_3, MgO, $S_{pyritic}$, and $C_{organic}$ decreased in this size fraction range. Thus, the composite concentrate (1,180 x 20 µm size fraction) resulted in higher P_2O_5, but did not reach the 34% P_2O_5 required due to the presence of A.I. Figure 13-33 presented the cumulative grade of P_2O_5, Al_2O_3, Fe_2O_3, A.I., CaO, MgO, $S_{pyritic}$, and $C_{organic}$ as a function of particle size. This figure showed that the P_2O_5, CaO, and A.I. were lower in the +1180 µm size fraction and at the -20 µm size fraction; whereas, the impurities (Al_2O_3, Fe_2O_3, MgO, $S_{pyritic}$, and $C_{organic}$) increased in these size fractions.

The frequency and cumulative distributions for P_2O_5, Al_2O_3, Fe_2O_3, A.I., CaO, MgO, $S_{pyritic}$, and $C_{organic}$ are presented in Figure 13-34 and 13-35, respectively. Again, the frequency distributions are dominated by the retain weight distribution as seen in Figure 13-2. However, the effect of the horizontal, and attrition scrubbing unit operations resulted in the increase of impurities in the minus 20 µm size fraction, the impurities decreasing in the coarser size fractions. Figure 13-35 shows that the P_2O_5, CaO, and A.I. follow the same pattern increasing in grade in the 1,180 x 20 µm size fraction; whereas, the loci of all impurities present similar cumulative distributions decreasing at that size fraction, the $S_{pyritic}$ decreasing significantly. This may be due to chemical analysis.

The composite concentrate average for all three tests is presented in Table 13-15 showing the results of wet horizontal and attrition scrubbing apply to the North pit composite, and the normalized values. Tests were carried out at 35% solids content and 5.0 minutes, and 55% solids with 2.5 minutes' residence time for the horizontal and attrition scrubbing, respectively.

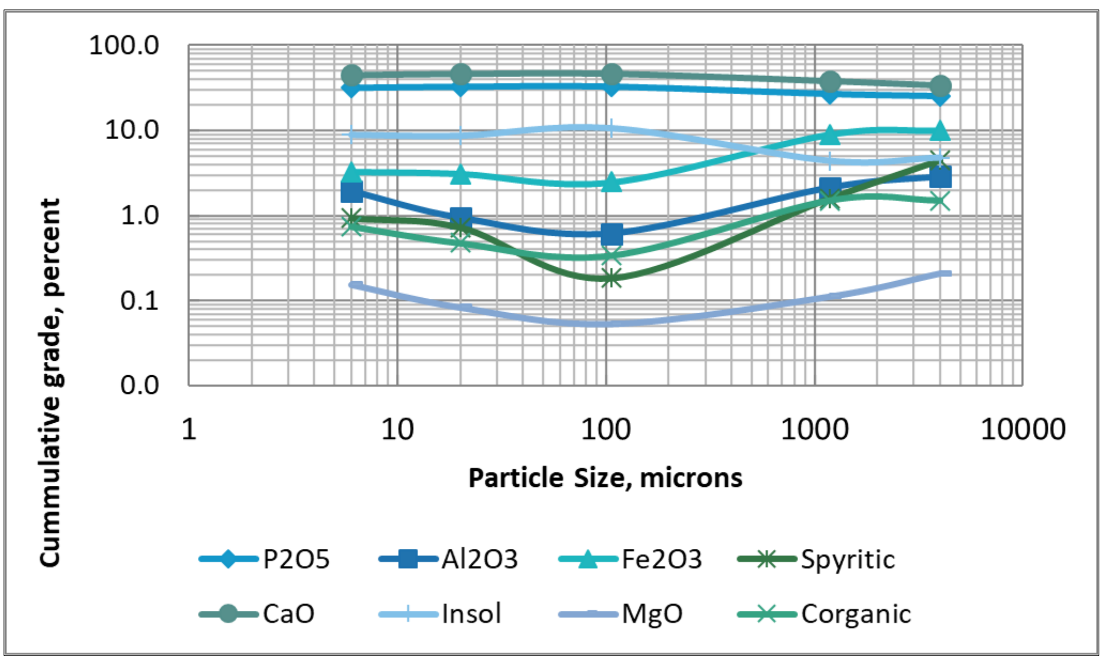
The average mass yield is 74.31%, the average P_2O_5 grade is 32.29% with 9.21% A.I., 0.89% Al_2O_3, 2.76% Fe_2O_3, 0.97% $S_{pyritic}$, and 0.44% $C_{organic}$, and the average P_2O_5 recovery is 76.75%, the product reporting a CaO/P_2O_5 ratio of 1.394, MER* of 0.078, and a P_2O_5 grade potential of 37.06% P_2O_5.

Figure 13-32: Grades as a Function of Particle Size for Farim North Pit



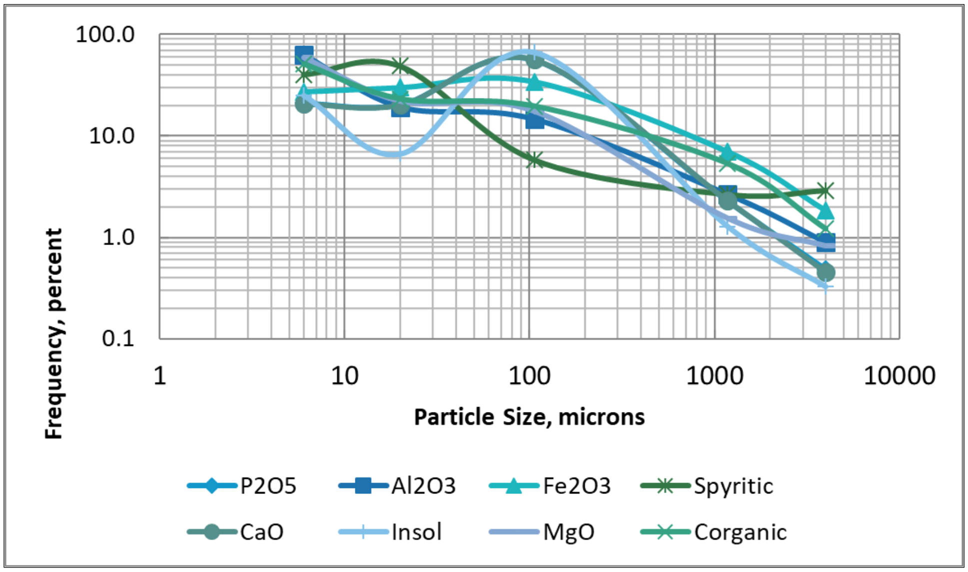
Source: KEMWorks, 2019

Figure 13-33: Cumulative Grades as a function of Particle Size for Farim North Pit



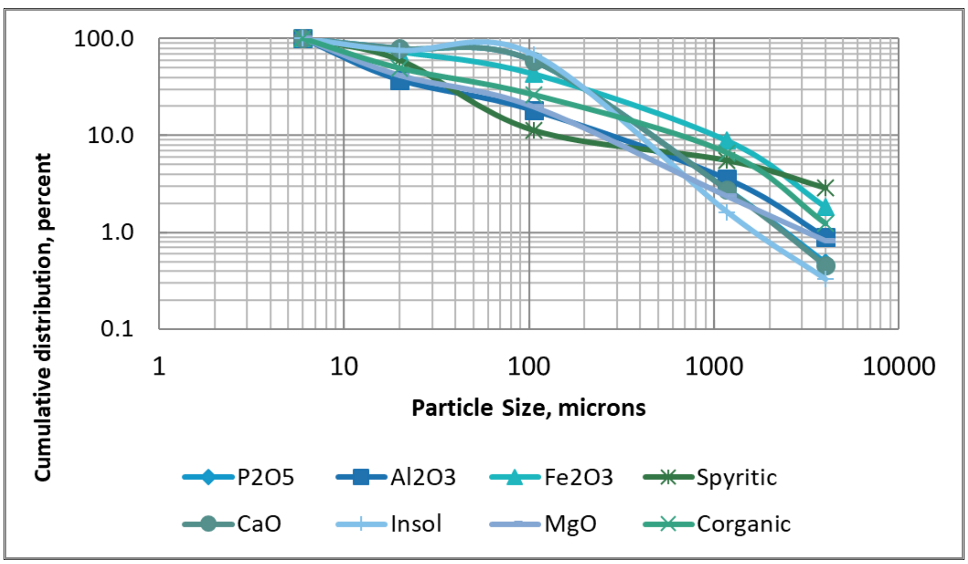
Source: KEMWorks, 2019

Figure 13-34: Frequency Distribution as a Function of Particle Size for Farim North Pit



Source: KEMWorks, 2019

Figure 13-35: Cumulative Distribution as a Function of Particle Size for Farim North Pit



Source: KEMWorks, 2019

For the coarse concentrate (1,180 x 106 µm), the mass yield (weight recovery) ranged from 53.72% to 54.55%., the P_2O_5 grade from 32.12% to 32.78% analyzing an average (composite) of 11.37% A.I., 0.48% Al_2O_3, 2.00% Fe_2O_3, 0.61% $S_{pyritic}$, 0.29% $C_{organic}$, and the P_2O_5 recovery from 55.28% to 56.46%. In the case of the fine concentrate (106 x 20 µm) the mass yield ranged from 19.6% to 21.25%, the P_2O_5 grade from 31.48% to 32.48% with an average (composite) of 3.45% A.I., 2.00% Al_2O_3, 4.79% Fe_2O_3, 2.21% $S_{pyritic}$, and 0.84% $C_{organic}$, and P_2O_5 recovery ranged from 19.88% to 21.93%. The composite product (coarse and fine concentrate combined), 1,180 x 20 µm size fraction reported mass yield between 73.57% to 75.21%, P_2O_5 grade between 31.95% to 32.60%, and P_2O_5 recovery between 76.32% and 77.21%. The summary of the bench scale metallurgical results of the combined three tests carried out for the North pit phosphate composite were:

- Mass yield..74.31%
- P_2O_5 recovery..76.75%
- CaO/ P_2O_5 Ratio..1.39
- MER..0.116
- MER*..0.078
- P_2O_5 grade..32.3%

Table 13-16: Composite Result for the Farim North Pit Phosphate Ore

Product	Time	Opening	Retained	Retained	Cumulative Retained	Passing
	Seconds	µm	Wt., g	Wt., %	Wt., %	Wt., %
Composite	150	1180x20	369.33	74.31	74.31	25.69

Cumulative Grade							
P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Corganic, %	Spyritic, %	Insol, %
32.29	45.01	0.09	0.89	2.76	0.44	0.97	9.21
Cumulative Grade							
P_2O_5, %	CaO, %	MgO, %	Al2O3, %	Fe_2O_3, %	Corganic, %	Spyritic, %	Insol, %
76.75	76.74	41.17	35.2	64.24	44.41	69.40	75.22

CaO/P2O5	MER*	Grade Potential P2O5, %
1.394	0.078	37.06

Composite Results – Normalized

Product	Time	Opening	Retained	Retained	Cumulative Retained	Passing
	Seconds	µm	Wt., g	Wt., %	Wt., %	Wt., %
Composite	150	1180x20	369.33	74.31	74.31	25.69

Cumulative Grade							
P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Corganic, %	Spyritic, %	Insol, %
103.29	103.28	55.41	47.37	89.45	59.77	93.39	101.23
Cumulative Grade							
P_2O_5, %	CaO, %	MgO, %	Al_2O_3, %	Fe_2O_3, %	Corganic, %	Spyritic, %	Insol, %
76.75	76.74	41.17	35.2	65.16	44.41	69.40	75.22

CaO/P_2O_5	MER*	Grade Potential P_2O_5, %
99.988	62.205	101.73

13.8 Metallurgical Balances from Pilot Plant Testwork – Scrubbing Only

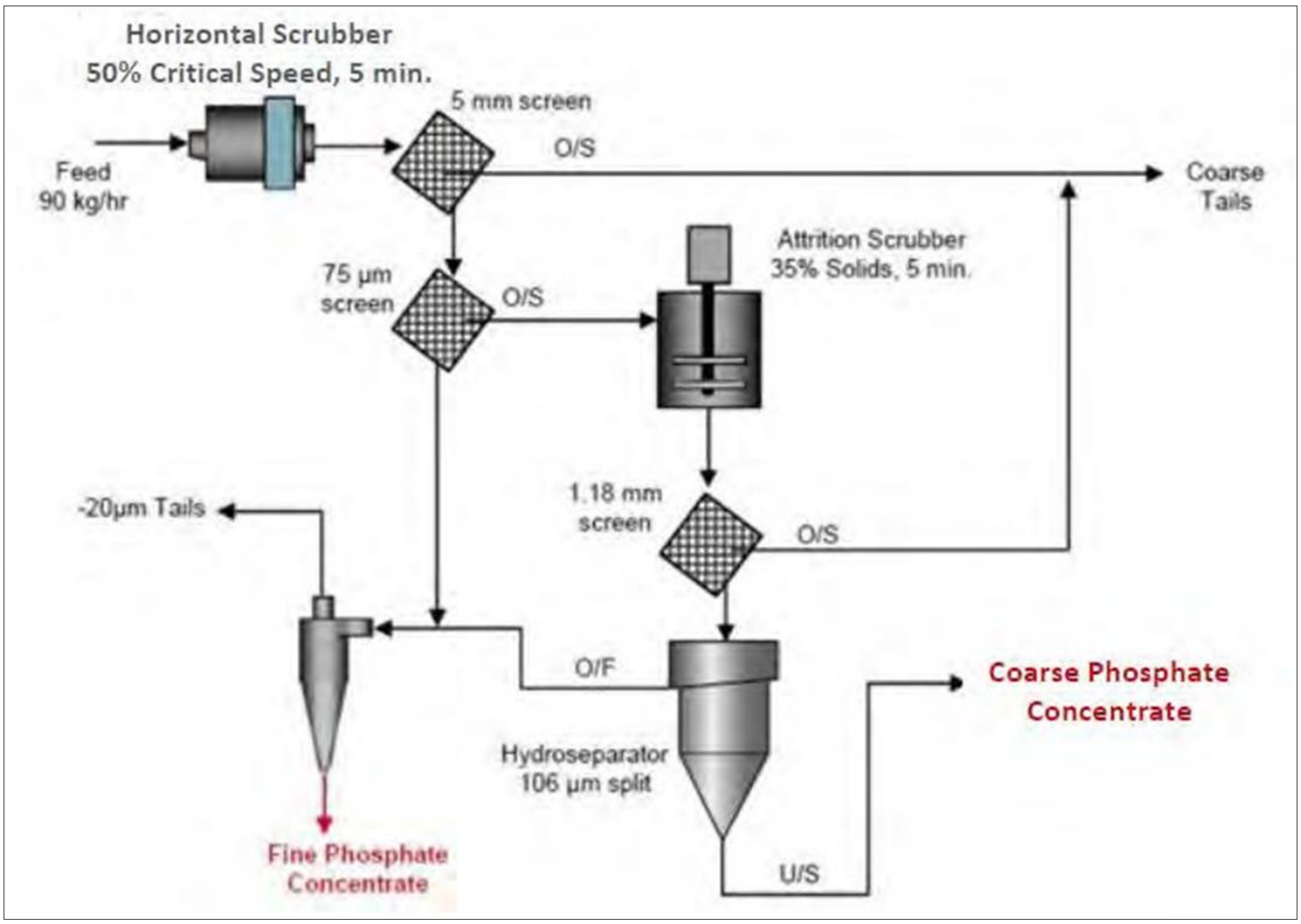
Four pilot plant tests were conducted, three at ALS Metallurgy Kamloops, and one at SGS Mineral Services. The purpose of the first, third and fourth pilot plant tests at ALS Metallurgy Kamloops were to demonstrate and improve the metallurgical response of the designed beneficiation process, and to produce enough concentrate. The concentrate was supplied as samples to potential clients for characterization analyses and for phosphoric acid processes testing. The second pilot plant test was aimed at obtaining enough material on different stages of the beneficiation process to perform rheological and characterization studies for the proper design of the different unit operations of this metallurgical process.

13.8.1 ALS First Pilot Plant Testwork

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 13-36 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 13-36.

Figure 13-36: Pilot Flowsheet Developed by ALS



Source: ALS, 2015

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 13-17.

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, a hydroseparator unit, and different conditions in the attrition scrubbing unit operation in the pilot plant. The pilot testing better represents the behavior of the Farim ore in the proposed process plant.

Table 13-17: 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, %	Pyritic 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, %	Combine d 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.

13.8.2 SGS Second Pilot Plant Testwork

The beneficiation pilot-scale test was conducted on a sample from Farim Phosphate Project with the objective of generating concentrate for downstream testing, and to provide a preliminary indication of the metallurgy in a semi-continuous processing environment. Thirteen samples were received as wet core sections weighing 737 kg and were used to prepare a pilot plant feed composite. Assay from each type of core section was carried out before the composite was prepared according to the client's instructions. An estimated 548 kg dry equivalent solid sample was prepared as testwork material. The blended feed head assay summary is shown in Table 13-18. The metallurgical parameters for the blended feed were calculated from the full analysis results of the bore hole samples, which included MgO = 0.18% and Pyrite = 1.02%: CaO/P_2O_5 ratio = 1.483, MER = 0.187, MER* = 0.163, and P_2O_5 grade potential = 36.66%.

As part of the pilot-scale testwork, 40 kg of the feed was used to run four bench scale tests of 10 kg each to demonstrate the beneficiation process. These bench scale tests resulted in a coarse concentrate (hydrosizer underflow) of 33.4% P_2O_5, this stream recovering 66.5% of P_2O_5 for a mass yield of 62.8%. The cyclone underflow considered the fine concentrate of 31.2% P_2O_5 grade recovering 13.2% of P_2O_5 for a mass yield of 13.4%. The +4 mesh (4,760 µm) and 16 mesh (1,180 µm) oversize products were 2.88% and 2.53% of the mass yield rejected, respectively. Thus, the combined concentrate analyzed 33.01% for a total mass yield of 76.2% with a total P_2O_5 recovery of 79.7%. An estimated 474 kg dry equivalent solids were processed through small pilot-scale equipment. Several pieces of equipment were changed due to problems with their performance, such as the horizontal scrubber (a grated grinding mill instead of an overflow grinding mill), and the hydrosizer (replaced by a Sweco 150 mesh vibrating screen). In addition, some operating parameters were changed, such as the solids content of the attrition scrubbing (40%) and the speed at the tip of propeller.

Table 13-18: Head Assay Summary-SGS Pilot Plant Feed Sample

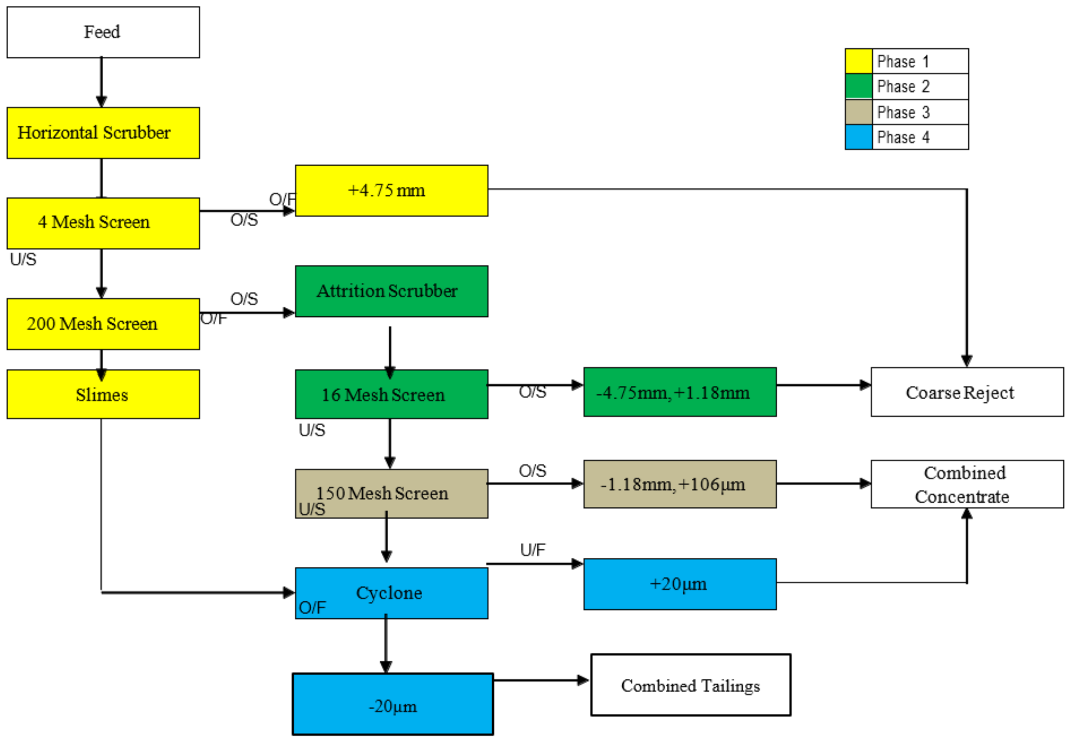
Sample ID	% Used in PP Feed	Assays, %					
		SiO_2	Al_2O_3	Fe_2O_3	CaO	P_2O_5	LOI
SP-15-01	2.0	6.08	1.19	2.94	48.4	30.1	6.92
SP-15-02	6.5	8.00	1.42	3.35	46.5	32.6	5.16
SP-15-03	1.1	11.3	3.55	6.48	39.1	26.9	8.40
DH 16 GC-09	6.8	11.6	2.26	2.80	44.7	30.4	5.97
DH 16 GC-10	11.3	8.60	0.81	2.32	47.7	31.8	5.90
DH 16 GC-11	7.0	5.24	1.42	1.82	49.0	34.5	4.43
DH 16 GC-12	0.4	13.6	1.62	2.76	43.1	30.3	5.23
DH 16 GC-13	15.0	11.6	2.48	4.22	42.9	29.4	5.99
DH 16 GC-14	8.7	6.56	1.31	3.35	47.4	31.1	6.92
DH 16 MET-01	9.1	9.29	1.08	2.16	47.5	32.8	4.32
DH 16 MET-02	4.6	8.34	1.68	3.82	45.6	30.3	5.12
DH 16 MET-03	14.0	9.70	1.26	10.9	39.4	26.3	7.48
DH 16 MET-04	13.5	4.46	0.98	1.22	51.0	35.4	3.88
PP Feed (direct)	-	8.19	1.35	4.36	46.7	31.5	4.97

The pilot plant test was carried out in four phases (see Figure 13-37) for two weeks from January 5 to 13, 2017. The overall metallurgical balance is presented in Table 13-19.

The results of the pilot plant feed and out product analyses were used to calculate the metallurgical parameters for the combined concentrate. This data showed a pyrite content of 0.74% and MgO of 0.13%. The CaO/P_2O_5 ratio was 1.483, the MER was 0.126, and the MER* was 0.110. The metallurgical balance was difficult to calculate due to the large amount of sample that was collected from seven streams, 4,750 x 75 µm, -75 µm (200 mesh undersize), 1,180 µm mesh feed (16 mesh feed), -1,180 µm (16 mesh undersize), +106 µm (150 mesh oversize), -106 µm (150 mesh undersize), and -20 µm (cyclone underflow) for solid-liquid separation and rheology testing even though only rheology testing was needed. This accounts for about 16% of the total feed but corresponding to different streams with different phosphate contents. Consequently, the mass yield of the Combine Concentrate only accounts for 62.1%, and the P_2O_5 recovery for 64.7%. If collected material is added to the material balance it is estimated that the mass yield will be about 73.9% with a P_2O_5 recovery of about 77.0%, this data being closer than those of the bench scale tests.

The subsamples of combined tailings were provided for a series of solid-liquid separation and rheology tests. BASF Magnafloc 10 Flocculant, which is a very high molecular weight, slightly anionic polyacrylamide flocculant was selected as a suitable flocculant for cyclone overflow thickening tests. Cyclone overflow dynamic thickening tests indicate that the underflow achieved 10.8%wt solids content. Higher underflow is expected to be achieved in industrial thickeners based on extended underflow solid density and CSD results obtained from rheology tests.

Figure 13-37: Pilot Plant Processing Block Diagram



Source: KEMWorks, 2019

Table 13-19: Pilot Plant Products Reconciled Assay Summary

Streams	Reconciled Assays, %					Reconciled Distribution, %					
	SiO_2	Al_2O_3	Fe_2O_3	CaO	P_2O_5	Weight	SiO_2	Al_2O_3	Fe_2O_3	CaO	P_2O_5
Combined Concentrate	8.13	0.27	3.69	48.1	32.4	62.1	60.5	11.4	52.5	64.6	64.7
Combined Tails	9.58	3.94	2.77	44.5	30.0	30.0	34.4	81.2	19.0	28.8	28.9
Coarse Reject	3.84	0.42	22.2	34.6	22.6	5.1	2.3	1.5	25.8	3.8	3.7
Feed	8.34	1.45	4.37	46.3	31.1	100.0	100.0	100.0	100.0	100.0	100.0

A subsample of combined concentrate was provided for vacuum filtration and rheology testing. Direct discharge filtration of the combined concentrate resulted in high throughput and high residual moisture.

13.8.3 ALS Third Pilot Plant Testwork

The pilot plant testwork was carried out to process 3,600 kg of a bulk sample of ore prepared from drill cores, and it was reported by ALS on June 20, 2017. The testwork was aimed at applying the developed flowsheet to process the sample, to demonstrate the metallurgical performance of the sample to the developed process, to produce a bulk quantity of concentrate for marketing purposes, and to conduct environmental testing on selected tailings products. The pilot plant testwork was carried out during April 2017. The head chemical analyses are presented in Table 13-20, and head subsamples were submitted to screen assay after attrition scrubbing of the head sample. Results indicated that high levels of iron were present in the fractions coarser than 1180 µm, while the material finer than 20 µm contained elevated levels of alumina and silica.

Table 13-20: Head Assay Data

Sample	Assay – Percent								Moisture Percent
	P_2O_5	CaO	Fe_2O_3	Al_2O_3	MgO	SiO_2	Insol	F	
Head 1	31.2	45.4	2.8	1.27	0.20	7.4	5.3	-	23.9
Head 2	32.3	46.2	3.3	1.13	0.22	6.7	4.8	-	24.6
Average	31.7	45.8	3.0	1.20	0.21	7.0	5.1	3.35	24.3

Notes: a) Oxide analyses determined by Whole Rock methods: lithium borate fusion followed by ICP. b) Fluorine analysis conducted by ALS Minerals.

The proposed beneficiation process follows the flowsheet shown in Figure 13-36, the pilot plant testwork being conducted at a feed rate of 90 kg/h on a dry basis. It was produced coarse concentrate (1180 x 106 µm) and a fine concentrate (106 x 20 µm), the screen assays indicating that than ideal separation would produce a mass yield of about 76% with a P_2O_5 grade of 33.7%. Both concentrates were filtered, air dried, and weighed. In the case of the fine concentrate, it was thickened without the use of flocculant, whereas the slimes (fine hydrocyclone overflow stream) were thickened using an anionic flocculant. The average process flow data is presented in Table 13-21.

Table 13-21: Average Process Flow Data

Stream	Mass %	kg/h	% Solids	Slurry L/min	Water Addition L/min
Horizontal Scrubber Feed	100	90.7	76.0	0.98	2.30
Horizontal Scrubber Discharge	100	90.7	35.2	3.28	
+5 mm Screen Oversize	0.9	0.8	70.3	0.01	2.81
5mm Screen Undersize	99.1	89.8	21.1	6.10	
+75 µm Screen Oversize	79.3	71.9	35.0	2.63	
-75 µm Screen Undersize	19.7	17.9	8.1	3.47	
Attrition Feed	79.3	71.9	35.0	2.63	
+1.18 mm Screen Oversize	3.6	3.3	69.7	0.04	
-1.18 mm Screen Undersize	75.7	68.7	28.8	3.21	0.62
Hydroseparator Underflow (Coarse Concentrate)	48.0	43.5	63.7	0.65	
Hydroseparator Overflow	27.7	25.1	9.0	4.36	1.80
Deslime Feed	47.5	43.0	8.6	7.8	
Cyclone Feed	47.5	43.0	3.8	18.3	10.5
Cyclone Underflow (Fine Concentrate)	29.0	26.3	18.0	2.14	
Cyclone Overflow	18.5	16.7	1.7	16.2	

Table 13-22: Bulk Processing Metallurgical Balance

Product	kg	Mass Percent	Assay – Percent					Distribution – Percent				
			P_2O_5	Fe_2O_3	Al_2O_3	MgO	SiO_2	P_2O_5	Fe_2O_3	Al_2O_3	MgO	SiO_2
Scrubber Feed	2259	100	31.7	3.1	1.3	0.2	8.6	100	100	100	100	100
+5 mm Tail	21	0.9	15.6	28.2	1.2	0.4	7.6	0.5	8.3	0.8	2.0	0.8
+1.18 mm Tail	79	3.5	27.2	15.1	0.5	0.2	3.9	3.0	16.9	1.4	4.3	1.6
Cyclone O/F – Slime Tails	420	18.6	28.4	2.2	5.18	0.50	11.5	16.7	13.4	72.4	45.8	24.8
Hydrosep UF – Coarse Concentrate	1085	48.0	32.4	2.6	0.20	0.09	9.52	49.2	39.5	7.1	22.4	53.0
Cyclone U/F – Fine Concentrate	655	29.0	33.6	2.3	0.84	0.18	5.88	30.7	21.9	18.4	25.5	19.8
Combined Concentrate	1739	77.0	32.8	2.5	0.44	0.13	8.15	79.9	61.4	25.5	47.9	72.8

Notes: a) Based on bulk sample processed during P2 – P4 run days. b) Summation of mass balanced survey sample data for each run day. C) Analyses by whole rock method – lithium borate fusion, digested solutions read by ICP.

Product	kg	Mass Percent	Assay – Percent					Distribution – Percent				
			P_2O_5	Fe_2O_3	Al_2O_3	MgO	SiO_2	P_2O_5	Fe_2O_3	Al_2O_3	MgO	SiO_2
Bulk Feed	2738	100	31.6	3.6	1.4	0.2	8.4	100	100	100	100	100
+5 mm Tail	75	2.7	17.3	25.2	1.3	0.4	7.9	1.5	19.2	2.6	5.5	2.6
+1.18 mm Tail	84	3.1	27.3	14.7	0.5	0.2	3.8	2.6	12.6	1.1	3.6	1.4
Cyclone O/F – Slime Tails	487	17.8	28.4	2.2	5.18	0.50	11.5	15.9	11.1	68.0	41.8	24.4
Hydrosep UF – Coarse Concentrate	1180	43.1	33.1	2.7	0.20	0.10	9.14	45.1	32.7	6.2	20.3	47.0
Cyclone U/F – Fine Concentrate	912	33.3	33.1	2.6	0.90	0.18	6.22	34.8	24.4	22.0	28.8	24.7
Combined Concentrate	2092	76.4	33.1	2.7	0.50	0.14	7.87	79.9	57.0	28.3	49.1	71.7

Notes: a) Based on total bulk sample processed, including +50 mm coarse rock removed during preparation. b) Concentrate data from final dried bulk concentrate produced. c) Analyses by whole rock method – lithium borate fusion, digested solutions read by ICP.

The concentrate analyses are presented in Table 13-23.

Table 13-23: Concentrate Chemical Analyses

Product	kg	P_2O_5 (ICP)	P_2O_5 (Gravimetric)	S (Total)	S (Pyritic)	CaO	Fe_2O_3	Al_2O_3	MgO
P2-P4 Coarse Con.	1061	33.0	32.3	0.87	0.52	47.73	2.74	0.20	0.10
P2-P4 Fine Con.	778	33.1	33.0	1.20	0.73	48.13	2.63	0.86	0.18
P2-P4 Total Con.	1839	33.0	32.6	1.01	0.61	47.9	2.69	0.48	0.13

Product	kg	BaO	K_2O	Cr_2O_3	MnO	Na_2O	TiO_2	SiO_2	Insol
P2-P4 Coarse Con.	1061	<0.01	<0.01	0.03	0.05	0.22	0.02	9.22	8.42
P2-P4 Fine Con.	778	<0.01	<0.01	0.04	0.04	0.22	0.05	6.26	5.14
P2-P4 Total Con.	1839	<0.01	<0.01	0.03	0.05	0.22	0.03	7.97	7.03

Based on these results, it was possible to calculate the metallurgical parameters for both concentrates. The coarse concentrate reported a CaO/P_2O_5 ratio of 1.446, the fine concentrate 1.454, and the combined concentrate 1.451. The MER for the coarse concentrate was 0.119, that of the fine concentrate 0.111, and 0.100 for the combined concentrate. The adjusted MER or MER* reported 0.072 for the coarse concentrate, 0.083 for the fine concentrate, and 0.077 for the combined concentrate. Finally, the P_2O_5 grade potential for the coarse concentrate was 37.27%, that of the fine concentrate 36.29%, and the P_2O_5 grade potential for the combined concentrate was 36.80%.

13.8.4 ALS Fourth Pilot Plant Testwork

Pilot plant testwork on 3,000 kg of a bulk composite of 54 drill hole samples was reported on November 28, 2019. The objectives of this test program were to process the bulk composite in a pilot-scale circuit using the developed flowsheet (see Figure 13-36) to confirm the metallurgical performance, produce a bulk quantity of concentrate, and collect sufficient quantities of selected process streams to obtain dewatering testing.

The head average P_2O_5, Fe_2O_3, and SiO_2 grades were about 32.9%, 3.8% and 7.2%, respectively. It was observed that the lower the P_2O_5 grade the higher the iron, silica and loss on ignition (LOI), this data suggesting that volatile elements, such as carbon, sulfur, and hydrated compounds may be the source of grade dilution. Table 13-24 presented the head assay results. From this table, the metallurgical parameters obtained were CaO/P_2O_5 ratio = 1.430, MER = 0.151, MER* = 0.129, and P_2O_5 grade potential = 36.94%. Screen assay of the bulk composite showed elevated iron in the fraction coarser than 1180 µm, while material finer than 20 µm contained elevated levels of alumina and silica. The screen assay data suggested that the ideal separation using the developed process would result in a 1,180 x 20 µm phosphate concentrate of about 33% P_2O_5, with a mass yield of 78%.

Table 13-24: Head Assay Results

Sample	Assay – Percent								
	P_2O_5	CaO	Fe_2O_3	Al_2O_3	MgO	SiO_2	Insol	S(t)	S(pyr)
Pilot Feed Head 1	32.5	46.5	3.20	1.57	0.23	9.3	6.4	1.30	0.58
Pilot Feed Head 2	33.4	47.8	3.32	1.38	0.22	8.0	5.1	1.17	0.58
Average	33.0	47.2	3.26	1.48	0.23	8.7	5.7	1.24	0.58

Notes: a) Oxide analyses determined by lithium borate fusion followed by ICP-OES. b) Total sulfur was analyzed by LECO and pyritic sulfur by standard method (ASTM D2492).

The flowsheet of the developed process is depicted in Figure 13-36 and the testwork was conducted with a feed rate of 84 kg/h that was maintained during the four day of pilot plant testwork. For this pilot-scale test, the attrition scrubber conditions and equipment were modified to adjust to the design parameters after the first operating day. Sample from selected streams were collected over a steady-state two-hour period of each operating day. All concentrate was filtered, air dried, and weighed.

The process flow data is presented in Table 13-25, and the metallurgical mas balance was shown in Table 13-26.

The results obtained were 76.2% mass yield as combined concentrate recovering 80.3% of P_2O_5 with a grade of 32.4% P_2O_5. However, the bulk concentrate analysis presented in Table 13-276 showed for the coarse and fine concentrates chemical analysis 33.7% P_2O_5 and 34.0% P_2O_5, respectively for a combined concentrate of 33.7% P_2O_5. The coarse tailings was 6.8% of the mass containing 41% of the iron, 18% of the magnesium, and 4.7% of the phosphate. Additionally, 16.8% of the feed mass was rejected in the hydrocyclone overflow (slimes) resulting in approximately 15% of P_2O_5 of the feed, 60% aluminum, 36% magnesium, and 25% silica.

Table 13-25: Process Flow Data

Stream	Mass %	kg/h	% Solids	Slurry L/min	Water Addition L/min
Total Feed	100	88.8	82.8		
+12 mm O/S	4.08	3.6	3.4		
Horizontal Scrubber Feed	95.9	85.2	79.4	0.84	2.30
Horizontal Scrubber Discharge	95.9	85.2	34.7	3.14	
+5 mm O/S	0.39	0.3	71.8	0.00	3.36
5mm U/S	95.5	84.8	19.0	6.51	
+75 µm O/S	65.4	58.1	73.8	0.67	
-75 µm U/S	30.1	26.7	7.3	5.84	
Attrition Feed	65.4	58.1	54.7	1.12	0.46
+1.18 mm O/S	2.58	2.3	65.3	0.03	
-1.18 mm U/S	62.8	55.8	26.8	2.85	1.76
Hydrosizer U/F (Coarse Phosphate Con)	52.3	46.4	64.0	0.69	
Hydrosizer O/F	10.6	9.4	3.3	4.66	2.50
Deslime Feed	40.7	36.1	5.5	10.5	
Cyclone Feed	40.7	36.1	4.3	13.7	3.22
Cyclone U/F (Fine Phosphate Con)	23.9	21.3	12.6	2.58	
Cyclone O/F	16.7	14.9	2.2	11.1	

Table 13-26: Bulk Sample Metallurgical Mass Balance

Product	kg	Mass %	Assay – Percent							Distribution – Percent						
			Fe_2O_3	Al_2O_3	CaO	MgO	P_2O_5	SiO_2	Insol	Fe_2O_3	Al_2O_3	CaO	MgO	P_2O_5	SiO_2	Insol
Total Feed	2425	100	4.31	1.46	45.5	0.24	30.7	7.61	6.78	100	100	100	100	100	100	100
+12 mm Tail	99	4.1	31.5	0.70	24.8	0.83	16.7	4.02	3.33	29.9	2.0	2.2	14.2	2.2	2.2	2.0
Scrubber Feed	2326	95.9	3.2	1.49	46.4	0.21	31.3	7.76	6.92	70.1	98.0	97.8	85.8	97.8	97.8	98.0
+5 mm Tail	7	0.30	26.8	0.76	27.5	0.36	18.4	5.34	3.91	1.9	0.2	0.2	0.5	0.2	0.2	0.2
+1.18 mm Tail	65	2.7	14.8	0.96	38.9	0.25	26.5	4.08	2.44	9.2	1.8	2.3	2.8	2.3	1.4	1.0
Cyclone O/F	406	16.8	2.78	5.21	40.5	0.52	27.5	11.3	11.2	10.8	59.9	14.9	36.4	15.0	24.9	27.7
Hydrosizer UF – Coarse Concentrate	1267	52.3	2.62	0.29	47.7	0.10	**32.2**	7.87	7.07	31.8	10.5	54.7	21.1	**54.7**	54.0	54.5
Cyclone U/F – Fine Concentrate	581	23.9	2.97	1.56	48.9	0.25	**32.9**	5.49	4.15	16.5	25.7	25.7	25.1	**25.6**	17.3	14.7
Combined Concentrate	1848	**76.2**	2.73	0.69	48.1	0.14	**32.4**	7.12	6.15	48.3	36.2	80.4	46.1	**80.3**	71.3	69.1

Notes: a) Samples were analyzed by Whole Rock and Insoluble Acid analyses. B) The +12mm material was removed during bulk preparation, the remaining 2326 kg of scrubber feed was processed over 4 operating periods in the Pilot Plant. The balance reflects the summation of all processing data.

Table 13-27: Bulk Concentrate Analyses

Sample	Assay – Percent								
	P_2O_5	CaO	Al_2O_3	Fe_2O_3	MgO	SiO_2	S(t)	S(pyr)	C
Fine Concentrate	34.0	49.1	1.57	3.18	0.24	5.44	1.08	0.57	1.09
Coarse Concentrate	33.7	47.6	0.30	2.69	0.10	8.04	0.90	0.43	0.96

Table 13-27 data allowed the calculation of the metallurgical parameters for each concentrate. The coarse concentrate CaO/P_2O_5 ratio of 1.412, MER of 0.092, MER* of 0.057, and a P_2O_5 grade potential of 37.92% were obtained. The fine concentrate metallurgical parameters were CaO/P_2O_5 ratio = 1.444, MER = 0.147, MER* = 0.126. and P_2O_5 grade potential = 37.96%.

Special characterization studies were carried out, such as feed, coarse, and fine specific gravity determination, resulting in an average of 3.1. The angle of repose on a blend dried concentrate was 34% at a moisture content of 8%. During processing, buckets of coarse concentrate exiting the hydroseparator were collected at 65% solids content, and semi-thickened fine concentrate at 35% solids content without using flocculant were collected for dewatering testwork to be performed at Pocock Industrial. The fine concentrate was decanted, filtered, air dried to 3% moisture.

The hydrocyclone overflow (slimes) density was about 2% solids content but could be thickened to 20% solids content in the thickener underflow without flocculant, and a clean water overflow. This product was stored in drums where it reached 30% solids content in two days.

Based on this Fourth Pilot Plant test results and flowsheet considering the modifications required in the different unit operations to match original processing conditions of the KEMWorks' bench-scale laboratory test, control of the chemical analyses of the different products by a competent Florida laboratory, and the average data obtained from the previous pilot plant tests, a metallurgical balance most likely to be obtained in the industrial plant was estimated.

The modifications to the operating conditions of the Fourth Pilot Plant test were the following:

- The percentage of the critical speed (Cs) of the horizontal scrubber (a rod mill adapted as a scrubber) was increased to 60% to approach a cataract type of scrubbing of the material in the horizontal scrubber.
- Reduce the feed rate to the horizontal scrubber to 85 kg/h instead of 90 kg/h to obtain the necessary 5 minutes of retention time since the Cs was increased.
- Increase the solids content of the attrition scrubbing from 35% to 55%.
- Reduce the tank size to maintain the retention time at 2.5 minutes.
- Slightly increase the differential pressure in the hydrosizer to avoid the presence of fines in the coarse concentrate (1180 x 106 µm size fraction).
- Increase the operating pressure at the hydrocyclone from 9 psi to 10 psi to increase quality of the fine concentrate (106 x 20 µm size fraction).
- Eliminate the addition of flocculant in both the fine concentrate thickener and the hydrocyclone overflow (slimes or tailings) thickening.

Observations under the microscope showed quite clean concentrates, the grade being expected to be between 33% and 34% P_2O_5, after modifications were made the pilot plant running smoothly, and in continuous mode for all the working shifts. However, chemical analysis results were not in agreement with the observations under the microscope. Thus, it was clear that chemical analyses were not accurate, or samples were contaminated. So, it was decided to take grab samples of the feed, coarse concentrate, and fine concentrate to be sent to Thornton Laboratories, Testing and Inspection Services, Inc.

for chemical analyses since this laboratory was used by Florida Phosphate Industry for the certification and arbitration on phosphate analyses. Even though these are grab samples, the results from Thornton Laboratory indicated that the observations under the microscope were accurate resulting in 33.6% P_2O_5 for the both the coarse and fine concentrate, whereas ALS reported 31.5% P_2O_5 and 33.1% P_2O_5 for the coarse and fine concentrates, respectively.

The analyses reported by ALS on the final dry concentrates were more in agreement with the P_2O_5 results from Thornton Laboratory on the grab samples even though the fine concentrate reported higher P_2O_5 grade (33.9%) than that of the coarse concentrate (33.5%).

These estimates are using the values of the known weights of the different products: +5- mm Rejects, Coarse concentrate (1,180 x 106 µm), Fine Concentrate (106 x 20 µm), and Tailings (Slimes). It was considered that the weight of the coarse concentrate was known data, and the ratio of the coarse concentrate to the fine concentrate was 71:29. The grades of the rejects and tailings are based on the data of analysis assuming similar analyses as those of previous Pilot Plant tests. The calculations were done to match the head results of 33% P_2O_5 and 32% P_2O_5. In addition, the P_2O_5 grades of the different products were estimated except for the last two material balances that used the grades reported for the concentrates by Thornton laboratory and ALS, respectively.

Estimated material balances, listed in Table 13-28, showed a yield (mass recovery) of 77.5%, and P_2O_5 recoveries between 81.4% and 84.3%, the most likely P_2O_5 recovery being 81.8%. In the case of the P2O5 grade of the combined concentrate, the results were between 33.6% and 34.7%, the most likely P_2O_5 Grade being 33.6%. Consequently, it was concluded that:

- The process developed for the beneficiation of Farim Phosphate Ore is robust, continuous, and reliable.
- The combined concentrate analyzed over 33.5% P_2O_5 with a yield of 77.5% and a P_2O_5 recovery of over 81%.
- The phosphate parameters were all within specs: CaO/ P_2O_5 ratio of about 1.40; MER about 0.100; and MER* about 0.080.
- It was demonstrated that no flocculant was required for both the fine concentrate and tailings thickening. Since no reagents are used in the process it is likely that the concentrate could be certified as “organic”.

Table 13-28: Estimated Material Balances for Pilot Plant Tests Based on 4th Pilot Plant Test (March 19-22, 2019)

Head about 33% P_2O_5

Product	Weight, kg	Weight, %	Grade	Content	Recovery
			P_2O_5, %	P_2O_5	P_2O_5, %
Rejects	171.13	6.89	24.50	168.73	5.16
(1,180 x 106)	1371.00	55.17	34.50	1903.48	58.24
(106 x 20)	554.00	22.29	34.00	758.02	23.19
Slimes	388.76	15.64	28.00	438.06	13.40
	2484.89	100.00	32.68	3268.29	100.00

Yield = 77.47%
P_2O_5 Recovery, % = 81.43
Combined Con Grade = 34.36 % P_2O_5

Head about 32% P_2O_5

Product	Weight, kg	Weight, %	Grade	Content	Recovery
			P_2O_5, %	P_2O_5	P_2O_5, %
Rejects	171.13	6.89	20.50	141.18	4.47
(1,180 x 106)	1371.00	55.17	34.00	1875.90	59.43
(106 x 20)	554.00	22.29	33.50	746.87	23.66
Slimes	388.76	15.64	25.10	392.69	12.44
	2484.89	100.00	31.57	3156.64	100.00

Yield = 77.47%
P_2O_5 Recovery, % = 83.09
Combined Con Grade = 33.86 % P_2O_5

Head about 33% P_2O_5

Product	Weight, kg	Weight, %	Grade	Content	Recovery
			P_2O_5, %	P_2O_5	P_2O_5, %
Rejects	171.13	6.89	24.50	168.73	5.12
(1,180 x 106)	1371.00	55.17	35.00	1931.07	58.59
(106 x 20)	554.00	22.29	34.00	758.02	23.00
Slimes	388.76	15.64	28.00	438.06	13.29
	2484.89	100.00	32.96	3295.88	100.00

Yield = 77.47%
P_2O_5 Recovery, % = 81.59
Combined Con Grade = 34.71 % P_2O_5

Head about 32% P_2O_5

Product	Weight, kg	Weight, %	Grade	Content	Recovery
			P_2O_5, %	P_2O_5	P_2O_5, %
Rejects	171.13	6.89	20.50	141.18	4.47
(1,180 x 106)	1371.00	55.17	34.70	1914.52	60.65
(106 x 20)	554.00	22.29	33.50	746.87	23.66
Slimes	388.76	15.64	25.10	392.69	12.44
	2484.89	100.00	31.95	3195.26	100.00

Yield = 77.47%
P_2O_5 Recovery, % = 84.31
Combined Con Grade = 34.35 % P_2O_5

Head about 31.7% P_2O_5

Product	Weight, kg	Weight, %	Grade	Content	Recovery
			P_2O_5, %	P_2O_5	P_2O_5, %
Rejects	171.13	6.89	21.50	148.07	4.67
(1,180 x 106)	1371.00	55.17	33.58	1852.73	58.41
(106 x 20)	554.00	22.29	33.59	748.88	23.61
Slimes	388.76	15.64	27.00	422.41	13.32
	2484.89	100.00	31.72	3172.09	100.00

Yield = 77.47%
P_2O_5 Recovery, % = 82.02
Combined Con Grade = 33.58 % P_2O_5

Calculated Head, P_2O_5 = 31.82%

Product	Weight, kg	Weight, %	Grade	Content	Recovery
			P_2O_5, %	P_2O_5	P_2O_5, %
Rejects	171.13	6.89	21.63	148.96	4.68
(1.180x0.106)	1371.00	55.17	33.58	1852.73	58.22
(0.106x0.020)	554.00	22.29	33.59	748.88	23.53
Slimes	388.76	15.64	27.60	431.80	13.57
	2484.89	100.00	31.82	3182.37	100.00

Yield = 77.47%
P_2O_5 Recovery, % = 81.75
Combined Con Grade = 33.58 % P_2O_5

Calculated Head, P_2O_5 = 31.85%

Product	Weight, kg	Weight, %	Grade	Content	Recovery
			P_2O_5, %	P_2O_5	P_2O_5, %
Rejects	171.13	6.89	21.63	148.96	4.68
(1,180 x 106)	1371.00	55.17	33.50	1848.31	58.03
(106 x 20)	554.00	22.29	33.90	755.79	23.73
Slimes	388.76	15.64	27.60	431.80	13.56
	2484.89	100.00	31.85	3184.87	100.00

Yield = 77.47%
P_2O_5 Recovery, % = 81.76
Combined Con Grade = 33.62 % P_2O_5

Note: The following calculations are using data that was obtained in the pilot plant that are final and combined to some that are estimated: 1. The weight of the coarse concentrate is a known data. 2. The ratio of the coarse to fine concentrate is about 71:29. 3. The grades of the rejects and slimes are based on the data of analysis assuming similar analyses as those of previous pilot plant tests. 4. The calculations were done to match the reported head results of 33% P_2O_5 and 32% P_2O_5. 5. The total weight processed was 2,484.89 kg.

13.8.4.1 Metallurgical Variability

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. Later, selected samples of the North pit were submitted for preliminary characterization and metallurgical testwork using the designed beneficiation process developed for the South pit to extend the years of operation of the Farim deposit.

13.8.4.2 Deleterious Elements

The concentrates produced for Farim South pit did not contain deleterious elements and compounds. Table 13-29 presents a comparison of the phosphate rock chemical assays carried out on the pilot plant tests conducted: ALS 2015, SGS 2017, ALS 2017, and ALS 2019. Chemical analyses were performed by ALS, SGS, and included the results of Thornton Laboratory and Moroccan K10 Standard for comparison.

The data shows that the Farim South pit phosphate rock contained lower contaminants than those shown by Moroccan K10 Standard, with Cl and $C_{organic}$ being the only potential contaminants that may be deleterious elements. However, Cl is below the accepted content for phosphate rock. $C_{organic}$ is present in both the Farim North pit and South pit phosphate rock. This could be a deleterious element if the total $C_{organic}$ analyzed over 0.4%, depending on downstream product to be obtained. This is not the case for the Farim South pit. For the Farim North pit, the exploratory results show higher values than 0.4%, but they are produced by applying the metallurgical process designed for the Farim South pit. Since $C_{organic}$ is tied to alumina and iron-bearing minerals, rejection of Al_2O_3 and Fe_3O_3 during the optimized metallurgical process for the North pit may result in values below 0.4%.

Table 13-29: Farim Rock Pilot Plant Tests Results

Thornton Assay Comparison	2017 ALS Coarse	2017 ALS Fines	ALS 2017 Blended Mathematical Assay	ALS 2017 Pilot Plant Data Actual Blended Assay Result	Delta (Actual – Mathematical)	Moroccan K10 Standard	SGS 2017 Pilot Plant Data	ALS 2015 Pilot Plant Data	ALS 2019 Pilot Plant Data
P_2O_5 (%)	33.78	32.49	33.295	33.21	-0.085		33.98	34	33.94
Al_2O_3 (%)	0.23	0.82	0.452	0.45	-0.002		0.36	0.68	0.67
CaO(%)	48.5	47.4	48.086	47.98	-0.106		49.03	46.9	46.34
Fe_2O_3 (%)	2.28	2.49	2.359	2.36	0.001		3.58	2.15	2.84
MgO (%)	0.1	0.19	0.134	0.14	0.006		0.15	0.12	0.13
K_2O (%)	0.01	0.03	0.018	0.02	0.002		0.024		0.02
Na_2O (%)	0.21	0.16	0.191	0.19	-0.001		0.16		0.31
Cl (%)	299	290	295.613	314	18.387		470		66
CO_2 (%)	2.76	3.05	2.869	3.1	0.231		2.4		2.57
F (%)	3.18	3.6	3.338	3.52	0.182		3.69	3.6	2.22
S_{total} (%)	0.49	1.06	0.704	1.19	0.486		1.25	1.21	0.75
$S_{pyritic}$ (%)	0.05	0.58	0.249	0.73	0.481		0.45	0.06	0.44
$S_{sulfate}$ (%)	0.44	0.48	0.455	0.46	0.005				0.59
Acid Insol (%)	4.56	3.8	4.274	4.27	-0.004		2.59	6.4	2.97
Organic Carbon (%)	0.2	0.52	0.320	0.34	0.020		0.45		0.37
Cd (%)	5.8	8.8	6.929	6.9	-0.029		9.9		10.1
MER	0.0773	0.1077	0.0885	0.0888	0.000		0.1204	0.0868	0.1072
MER* taking Pyritic S into Account	0.0754	0.0854	0.0791	0.0614	-0.018		0.1038	0.0846	0.0910
CaO/P_2O_5 Ratio	1.436	1.459	1.444	1.445			1.443	1.379	1.365
Grade Potential, %	36.26	34.7	35.673	35.57	-0.103				

13.9 Summary and Conclusions

13.9.1 Ore Characterization

One hundred kilograms of core samples from the Farim South pit deposit was sent to 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. (acid insoluble) 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 (SB9, SC10, SC11, and SE10 as described in Section 13-4) were homogenized and 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 13-3. 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 140 µm with a single mode in the distribution (unimodal), the mode located at 106 µm (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 106 µm and decreasing for particles smaller than 106 µm. 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.

In the case of the Farim North pit phosphate ore, the objective of the bench-scale beneficiation testwork was to characterize phosphate samples from the North pit at Farim and to determine the validity of the beneficiation process developed for the South pit to the North pit.

Five separately identified 10 kg samples were received on September 14, 2016. The samples were processed to obtain the following for each one:

- A homogenized-blended sample
- A head chemical analysis sample

- A sample for determination of moisture content and dry density (bulk and in situ)
- A sample for determination of wet density (bulk and in situ)

These samples were significantly more clayish and wet than those from the South pit studied in 2015. Sample DH-16-MET-05 was wet and clayish but did not have any medium or large particles (pieces of rock). Sample DH-16-GC-03 had medium-size particles (rocks). Sample NP-15-1 had large pieces of rock, NP-15-03A was the most clayish and wet. NP-15-4 besides being clayish and wet showed some pieces of iron oxide, probably hematite (Fe_2O_3). The chemical analysis of the head sample was shown in Table 13-6.

A single composite sample was prepared from an equal amount of each 10 kg sample and the following activities were performed:

- Characterization studies.
- Splitting and bagging of 500-gram composite samples
- Application of the beneficiation process developed for the South pit in triplicate to the North pit composite.

The North pit phosphate ore was significantly finer than the South pit ore, the mean particle size (d_{50}) being 115 µm compared to 140 µm for the South pit. For the North pit ore, the mode particle size was at 212 µm (accumulation of particles in a size fraction) – the same as the South pit ore (unimodal). This indicates that the North pit contained more weight percentage of particles smaller than 212 µm than the South pit ore. However, similar weight percent of particles was observed on the -20 µm size fraction.

The characterization studies, head chemical analysis, screen analyses, and screen assays, showed that the Farim North pit composite sample has a lower P_2O_5 grade, higher acid insoluble (A.I.), and high organic carbon ($C_{organic}$). The minor elements (Al_2O_3, Fe_2O_3) and $S_{pyritic}$ were higher, and the MgO was lower than in the South pit. The results of the head sample chemical analysis were showed in Table 13-8 reporting 30.92% P_2O_5 with the following metallurgical parameters:

- CaO/ P_2O_5 ratio 1.4
- MER 0.152
- MER* 0.094
- P_2O_5 Grade Potential .... 33.3%.

13.9.2 Horizontal Scrubbing

Tests were conducted on Farim South pit phosphate ore under standard conditions as a baseline first and 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 (1,180 x 20 µm). 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%) were 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 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 +6,300 µm and 6,300 x 1,180 µm size fractions as rejects and the -20 µm material as slimes.

13.9.3 Attrition Scrubbing

Tests were designed to release significant amounts of quartz, clay, and iron bearing minerals attached to the francolite surfaces in the 6,300 x 75 µm size fraction obtained after horizontal scrubbing (drum) on the Farim South pit phosphate ore. However, A.I. rejection was limited to the -20 µm 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 (45%, 55%, and 60%) 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.

The North pit was submitted to the horizontal and attrition scrubbing unit operations designed. The results indicated that beneficiation process designed for the South pit ore was suitable for the North pit. The beneficiation process designed was carried out in triplicate to determine the reproducibility, and average values of the product to be obtained for the North pit composite. In general, all three tests were within 1% error from each other.

However, the presence of higher SiO_2 in the North pit ore resulted in not being able to achieve 34% P_2O_5 grade in the concentrate. The coarse fraction, 1180 x 106 µm, achieved a grade of 32.4% P_2O_5 and the fine fraction, 106 x 20 µm, achieved a grade of 32.15% P2O5, the average combined product (1180 x 20 µm size fraction) being 32.3% P_2O_5, which was lower than the target 34% P_2O_5.

- Mass yield........ 74.3%
- P_2O_5 recovery........ 76.8%
- CaO/ P_2O_5 Ratio........ 1.4
- MER........ 0.116
- MER*........ 0.078
- P_2O_5 grade........ 32.3%

13.9.4 Reverse Amine Flotation

Studies of the 1,180 x 106 µm size fraction of the Farim South pit phosphate ore 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 1,180 x 75 µm 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 13-28 which resulted in the following products:

- +6,30 rejection.. 5.2% ± 1.9%
- 6,300 x 1,180 µm rejection.. 2.2% ± 0.2%
- 1,180 x 106 µm flotation concentrate.. 49.3% ± 2.8%
- reverse flotation tailings.. 4.7% ± 1.7%
- 106 x 20 µm fine concentrate.. 16.6% ± 0.5%
- -20 µm slimes rejection.. 21.9% ± 0.3%

13.9.5 Pilot Plant Results

The results of the first 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 Wet Acid Process (WAP) testing for phosphoric acid production.

- 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%

The second pilot-scale testwork objectives were to generate concentrate for downstream testing, and to provide a preliminary indication of the metallurgy in a semi-continuous processing environment. The downstream testing consisted in solid-liquid separation and rheology of the combined concentrate and overflow hydrocyclone (slimes). The combined concentrate analyzed 32.4% P_2O_5 for a total of 297 kg of concentrate produced and shipped to different potential clients. The data reported corresponded to the metallurgical mass balance calculated and data estimated from the bench-scale results of the tests carried out by SGS and total material considered.

- Mass yield.. 62.15%, estimated 73.9%
- P_2O_5 recovery.. 64.7%, estimated 77.0%
- CaO/ P_2O_5 Ratio.. 1.5
- MER.. 0.126
- MER*.. 0.110
- P_2O_5 grade.. 32.4%, estimated 33.0%

The third pilot plant testwork was aimed at applying the developed flowsheet to process the sample, to demonstrate the metallurgical performance of the sample to the developed process, to produce a bulk quantity of concentrate for marketing purposes, and to conduct environmental testing on selected tailings products. The total combined concentrate produced was 2,092 kg for marketing purposes at 33.1% P_2O_5, and the difference being used for different characterization studies carried out. The metallurgical mass balance and the parameters were:

- Mass yield.. 76.4%
- P_2O_5 recovery.. 79.9%
- CaO/ P_2O_5 Ratio.. 1.4
- MER.. 0.110
- MER*.. 0.077
- P_2O_5 grade 33.1%

A fourth pilot-scale testwork was conducted with the objectives to process the bulk composite in a pilot-scale circuit using the developed flowsheet to confirm the metallurgical performance, produce a bulk quantity of concentrate, and collect sufficient quantities of selected process streams to obtain dewatering testing. The combined concentrate obtained was 1,868 kg with a P_2O_5 grade of 33.7%. Special characterization studies were carried out, such as feed, coarse, and fine specific gravity determination, resulting in an average of 3.1. The angle of repose on a blend dried concentrate was 34% at a moisture content of 8%. The hydrocyclone overflow (slimes) density was about 2% solids content but could be thickened to 20% solids content in the thickener underflow without flocculant, and a clean water overflow. This product was stored in drums where it reached 30% solids content in two days. The metallurgical mass balance and parameters were similar to those reported.

- Mass yield.. 76.2%
- P_2O_5 recovery.. 80.3%
- CaO/ P_2O_5 Ratio.. 1.4
- MER.. 0.109
- MER*.. 0.078
- P_2O_5 grade 33.7%

Based on the fourth pilot plant test results and flowsheet considering the modifications required in the different unit operations to match original processing conditions on the KEMWorks' bench-scale laboratory test, control of the chemical analyses of the different products by a competent Florida laboratory, and the average data obtained from the previous pilot plant tests, a metallurgical balance most likely to be obtained in the industrial plant was estimated.

Estimated material balances showed a yield (mass recovery) of 77.5%, and P_2O_5 recoveries between 81.4% and 84.3%, the most likely P_2O_5 recovery being 81.8%. In the case of the P_2O_5 grade of the combined concentrate, the results were between 33.6% and 34.7%, the most likely P_2O_5 grade being 33.6%. The most likely material balance and parameters were:

- Mass yield.. 77.5%
- P_2O_5 recovery.. 81.8%
- CaO/ P_2O_5 Ratio.. 1.4
- MER.. 0.108
- MER*.. 0.078
- P_2O_5 grade 33.6%

14 MINERAL RESOURCE ESTIMATES

14.1 Mineral Resource Definition

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

Under CIM's definition, 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 km² and is divided by the River Cacheu. 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.

The QP modelled the Farim resource based on a 2D grid of 125 m x 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 drillholes. 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 mineral resources estimate for the Farim deposit was 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.) in 2015. The mineral resource was updated in 2022 by Jennifer Simper of Golder, again under the supervision of QP, Jerry DeWolfe. The QP is independent of the Issuer as defined by Section 1.5 of the National Instrument. The mineral resource statement has a date of September 30, 2022.

The initial 2012 estimation was undertaken in Isatis™ (Version 2011.3) and Vulcan™ (Version 8.1.3), while the updated 2015 and 2022 estimates were performed in MineScape™ (Version 2021) and Vulcan™ (Version 9.1.3).

14.3 Data Provided

14.3.1 Drillhole Data

The mineral resource estimate is based on diamond drillhole data. A total 10,326 m were drilled in 190 diamond core holes on the Farim deposit between 1981 and 2011, as follows:

- BRGM 1981 – 2,100 m from 32 diamond core holes were drilled over a 25 km² 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 km² 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 km² 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 was 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 were 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 were available for inspection.

The QP chose not to include the post-2015 additional and/or infill drilling in the model or for the updated mineral resource estimate as it was determined to not be material to the global mineral resource estimate. The QP reviewed the drilling against the 2015 model surfaces and found no impact to the global estimate as all drilling occurred within areas previously classified as measured resources. The QP recommends that any future model updates incorporate all available drilling and sampling to aid short-term mine planning ahead of any preproduction or production activities.

All drillholes 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 mineralization is intersected by 148 drillholes with the majority on 500 m grid spacing. Several holes either had low or no recovery and were therefore excluded from the database (or fell outside the Farim deposit). Holes that were close to the Farim deposit and did not intersect FPA were 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 the QP through thorough validation checks against digital data. The QP acknowledges the limitations of the assay verification for the historical BRGM and Champion drilling due to the lack of assay certificates, however, the QP has reviewed the historical data against available QA/QC data as well as results from proximal drillholes. Based on these results the QP believes there is sufficient recent drilling to support the inclusion of the historical data in the mineral resources estimate. 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 QA/QC program and the compiled drillhole 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 center. 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 drillholes as a guide to where the unit thins out. This outline and the resource drillhole 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 behavior of the elements before estimation is carried out and 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:

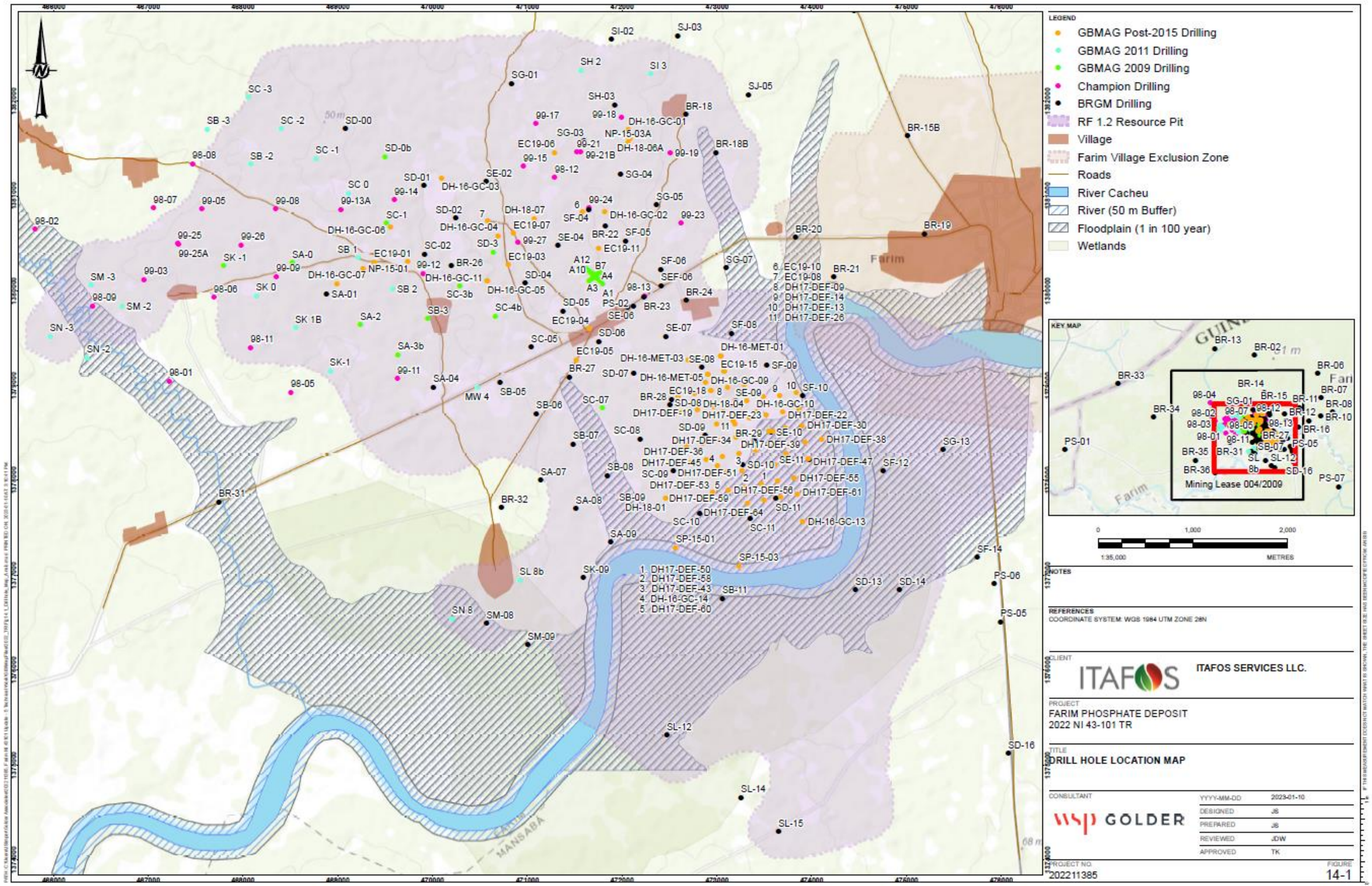
- domaining
- declustering
- top-cutting or treatment of high grades or outliers
- compositing
- parameters to be used during variography such as lag distance
- block size for the resource model.

The results of these analyses are described in the following chapters where appropriate.

14.5.1 Data Capture

Domains were 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 defines the extent of the FPA. The wireframes were used to select all the composites lying inside, which were flagged with a numeric code.

Figure 14-1: Farim, Drillhole Location Map



Source: WSP Golder, 2023

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. Individual assay results were not entered into the digital database; instead, length weighted averages per drillhole were entered by GBMAG. This is in effect lithological compositing, where drillholes 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 mineral resource estimate.

14.5.3 Statistical Analysis

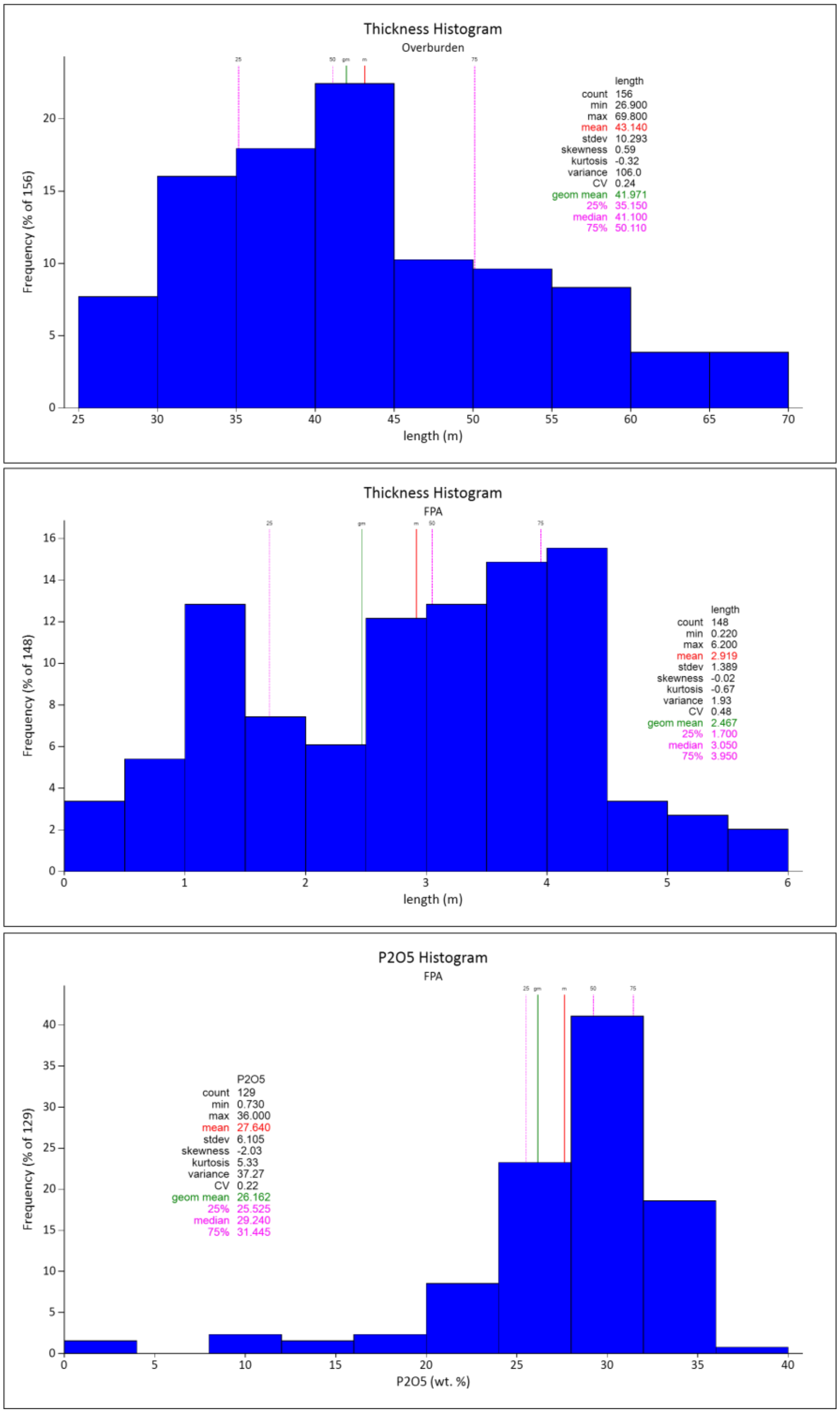
Figure 14-1 above shows the location of the drillholes contained within the current resource database for Farim. Most of the drillholes are in the north and central parts of the deposit where the spacing is approximately 500 m. On the periphery, especially to the south of the River Cacheu, the drillholes are sparser.

Univariate statistics and histograms of grade and thickness variables were generated and are summarized in Table 14-1 and Figure 14-2.

Table 14-1: Farim, Univariate Statistics

Variable	Count	Minimum	Maximum	Mean	Median	25%	75%
Al_2O_3	104	0.51	29.86	3.27	2.15	1.27	3.79
CaO	104	7.15	50.13	3.70	40.63	35.35	43.26
Fe_2O_3	104	0.49	40.98	5.90	4.31	2.99	6.76
P_2O_5	129	0.73	36.00	27.64	29.24	25.53	31.45
SiO_2	104	4.36	35.50	11.65	10.31	8.85	12.57
Overburden Thickness (m)	156	26.90	69.80	43.14	41.10	35.15	50.11
FPA Thickness (m)	148	0.22	6.20	2.92	3.05	1.70	3.95

Figure 14-2: Farim, Histograms



Source: WSP Golder, 2023

14.6 Variography

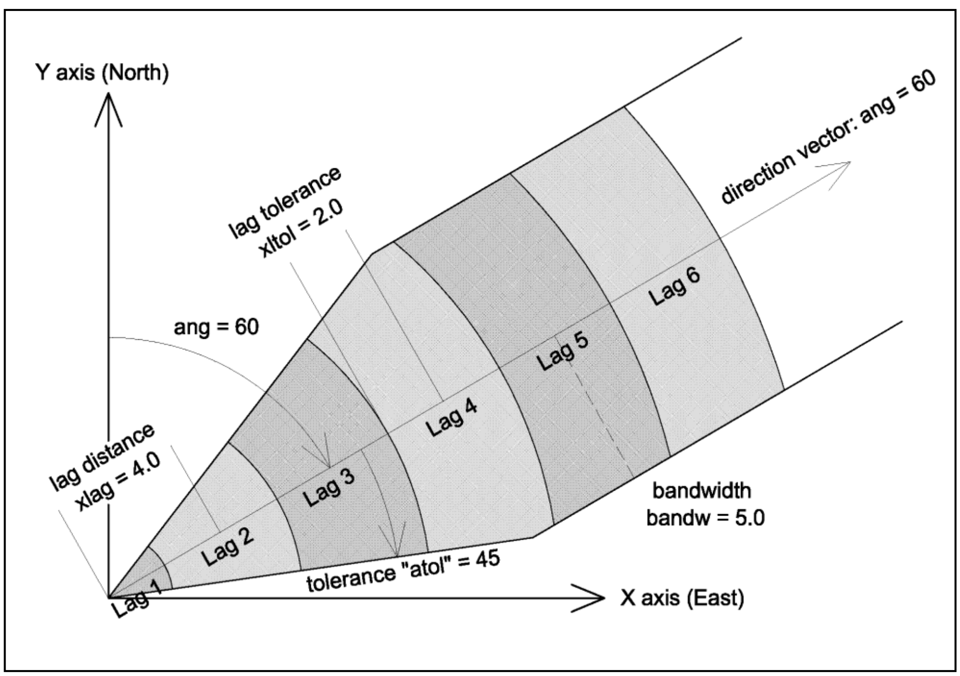
Variography is used to model the continuity of spatial phenomena such as the distribution of grade in a mineralized body. The objectives of the variography were to establish the directions of major and semi-major continuity for both P_2O_5 grade and thickness of the FPA phosphate horizon.

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 14-3 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 14-3: Conventions for Variogram Search Parameters



Source: Deutsch, C.V. and Journel, A.G., (1997). GSLIB Geostatistical Software Library and User's Guide, Oxford University Press, New York, second edition. 369 pages. (variogram conventions)

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

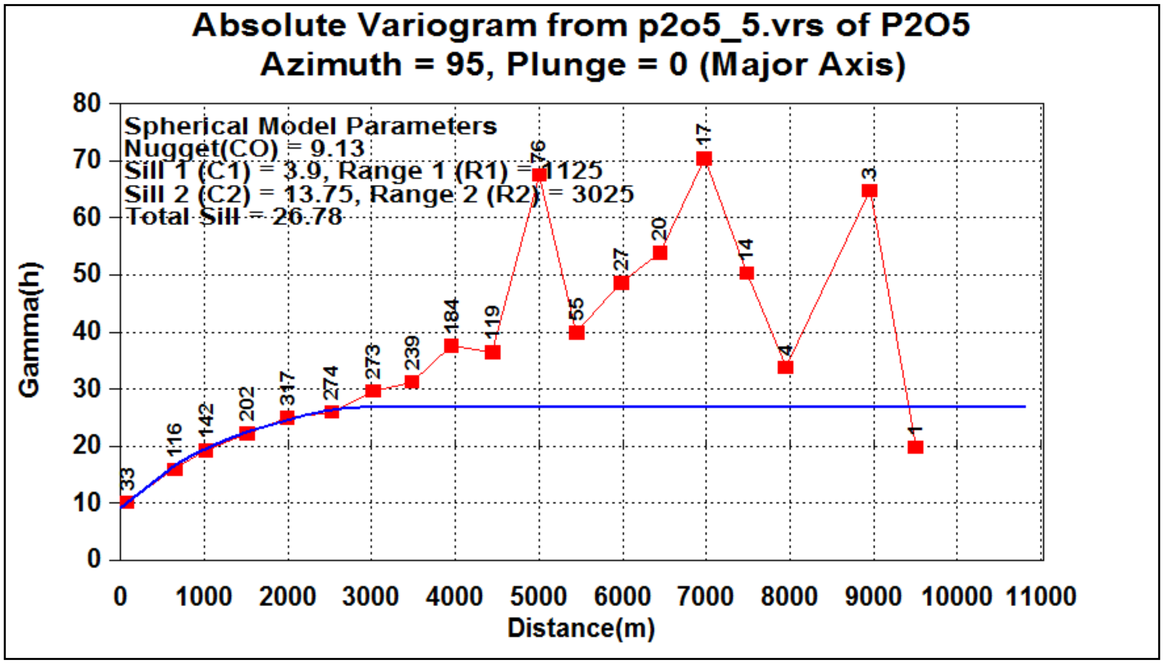
14.6.1 Phosphate

Directional variography shows a direction of greatest continuity in the major direction of N95 in Figure 14-4 and in the semi-major direction of N05 in Figure 14-5. Maximum continuities in the order of 3,000 and 2,500 m, respectively, are observed.

14.6.2 Thickness

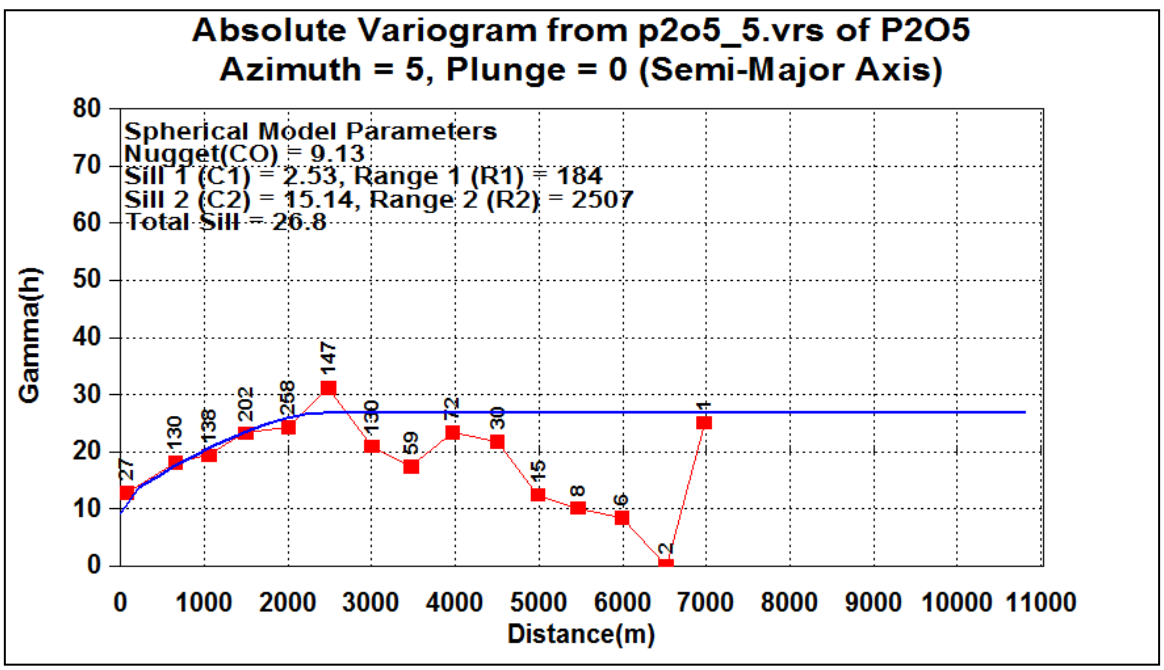
Directional variography shows a direction of greatest continuity in the major direction of N10 in Figure 14-6 and in the semi-major direction of N01 in Figure 14-7. Maximum continuities in the order of 3,000 and 2,000 m respectively are observed. No cutoff was used for thickness.

Figure 14-4: Directional Variogram in the Major Direction (N95) For P_2O_5 Showing Approximately 3,000 m Maximum Continuity



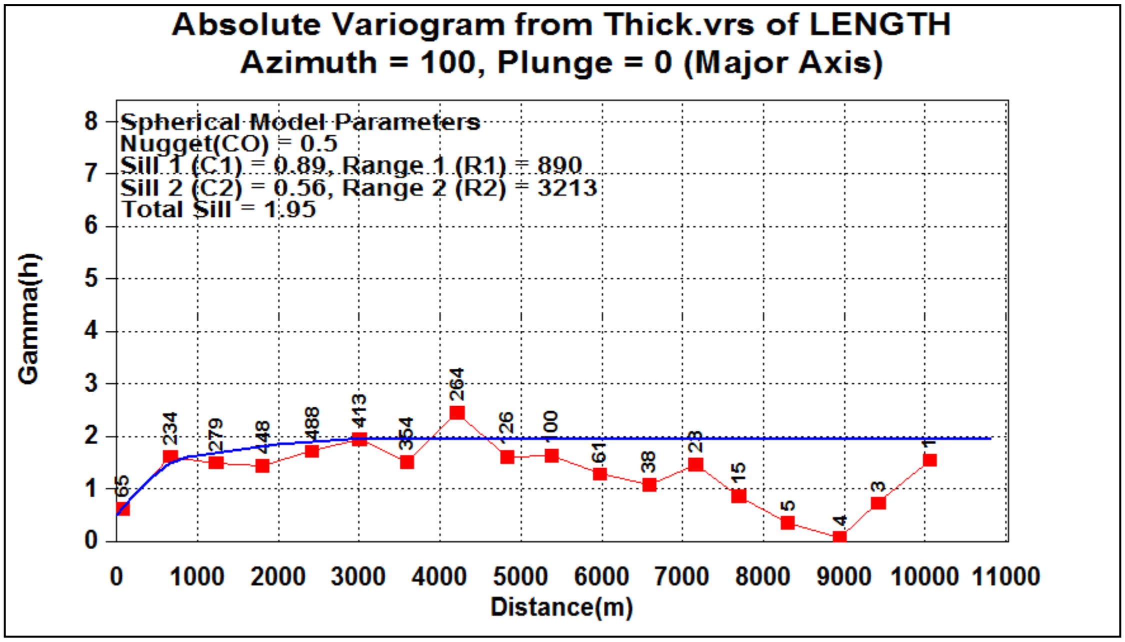
Source: WSP (formerly Golder) 2015

Figure 14-5: Directional Variogram in the Semi-Major Direction (N05) for P_2O_5 Showing Approximately 2,500 m Maximum Continuity



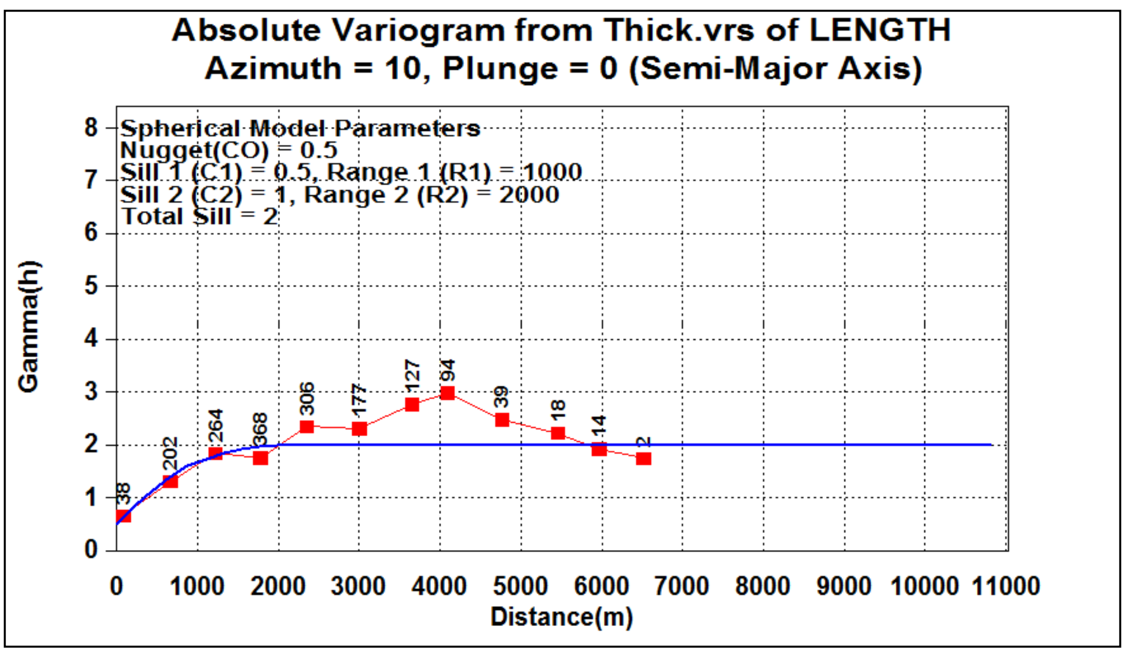
Source: WSP (formerly Golder) 2015

Figure 14-6: Directional Variogram in the Major Direction (N10) for Thickness Showing Approximately 3,000 m Maximum Continuity



Source: WSP (formerly Golder) 2015

Figure 14-7: Directional Variogram in the Semi-Major Direction (N01) for Thickness Showing Approximately 2,000 m Maximum Continuity



Source: WSP (formerly Golder) 2015

14.7 Summary of Statistical and Geostatistical Assessment

Sections 14.7.1 and 14.7.2 provide 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 east-west direction with the semi-major being in the north-south) direction. Therefore, variography supports the geological observations that the FPA is very regular, sub-horizontal and continuous.

The exploration drill pattern utilized has had a marked effect on variography results. The direction of greatest continuity (major direction) of mineralization appears to be different than the northeast-southwest and northwest-southeast 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 y-intercept, otherwise known as the nugget, is approximately 33%, and represents the small-scale variability in the grade. 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.

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 affected 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 was used in mineral 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 x 125 m reflecting a quarter of the average drillhole 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

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

Neighborhood 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 neighborhood 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 (OK) and inverse distance weighted (IDW) methods for the following variables:

- P_2O_5 (OK)
- Al_2O_3 (IDW^2)
- CaO (OK)
- Fe_2O_3 (OK)
- SiO_2 (IDW^2)
- FPA thickness, m (OK)
- overburden thickness, m (IDW^2).

Table 14-4 summarizes the final estimation parameters chosen following neighborhood 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
Minimum Samples Total	3	2	1
Number of Sectors	4	4	4
Minimum Samples per Sector	2	2	2
Discretization	5 x 5 x 1	5 x 5 x 1	5 x 5 x 1
Minimum 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. In light of overall density data availability and vintage, a slightly conservative density 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 drillhole 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.

Table 14-5: Block Model Validation, Statistical Comparison

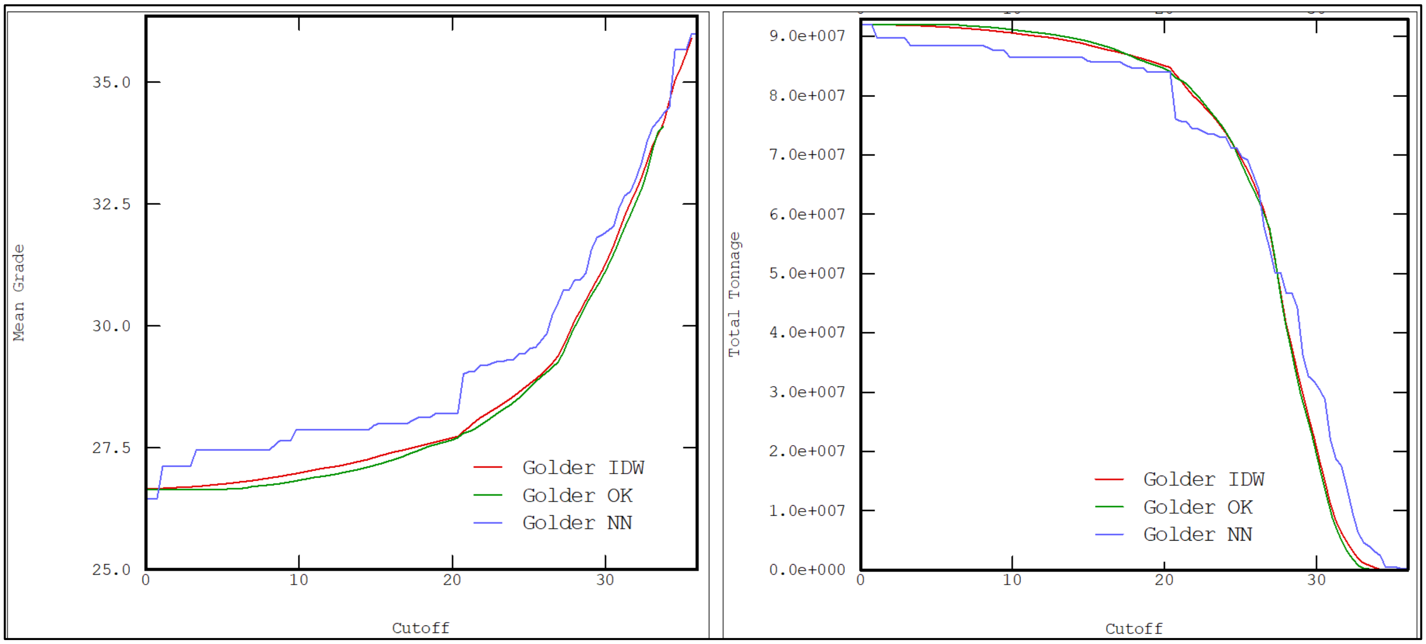
Univariate Statistics of Composite Values and Block Estimates – First + Second Passes Only									
Variable	Drillhole Composites		Block Estimates		EST/ CMP[1]	f [2]	f [3]	f diff[4]	Smoothing
	Mean	Variance	Mean	Variance	(%)				(%)
P_2O_5	28.69	27.50	27.32	20.14	95.22	0.732	0.923	0.190	19.0
FPA Thickness (m)	2.70	2.35	2.52	1.50	93.26	0.639	0.939	0.301	30.1
Overburden Thickness (m)	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

14.11.1 Grade-Tonnage Curves

Graphs showing grade, tonnage, and metal values versus cut off grades were plotted. These compare the OK and IDW estimated block model curves, Figure 14-8 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.

Figure 14-8: Farim, Grade-Tonnage Curves, P_2O_5

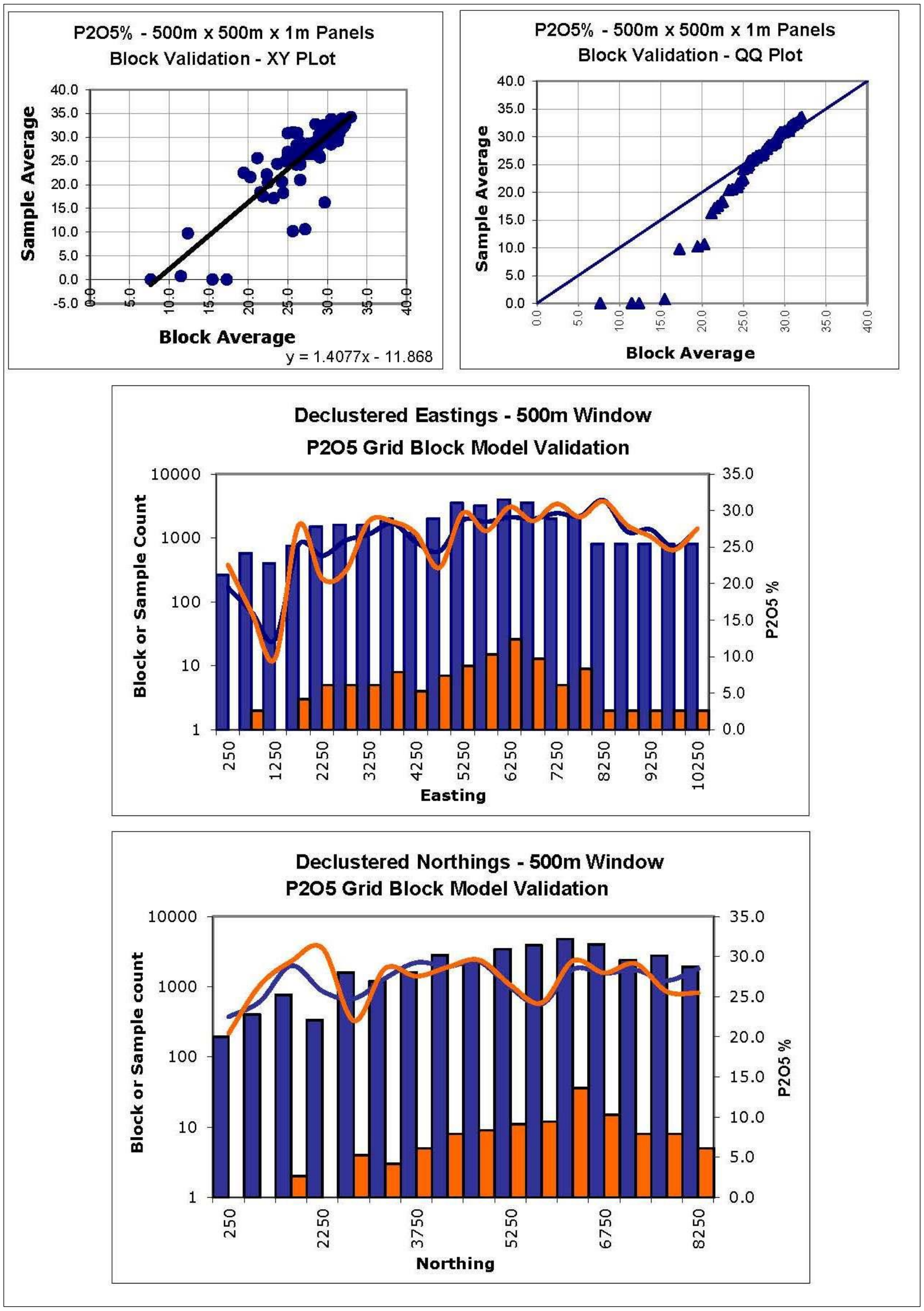


Source: WSP (formerly Golder) 2015

14.11.2 Swath Plots

Swath plots comparing local mean grades in broad “swaths” of the block model and corresponding composites were generated. This allows an analysis of local reproduction of composite grades by the block model. Figure 14-9 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.

Figure 14-9: Farim, Swath Plots P_2O_5



Source: WSP (formerly Golder) 2015

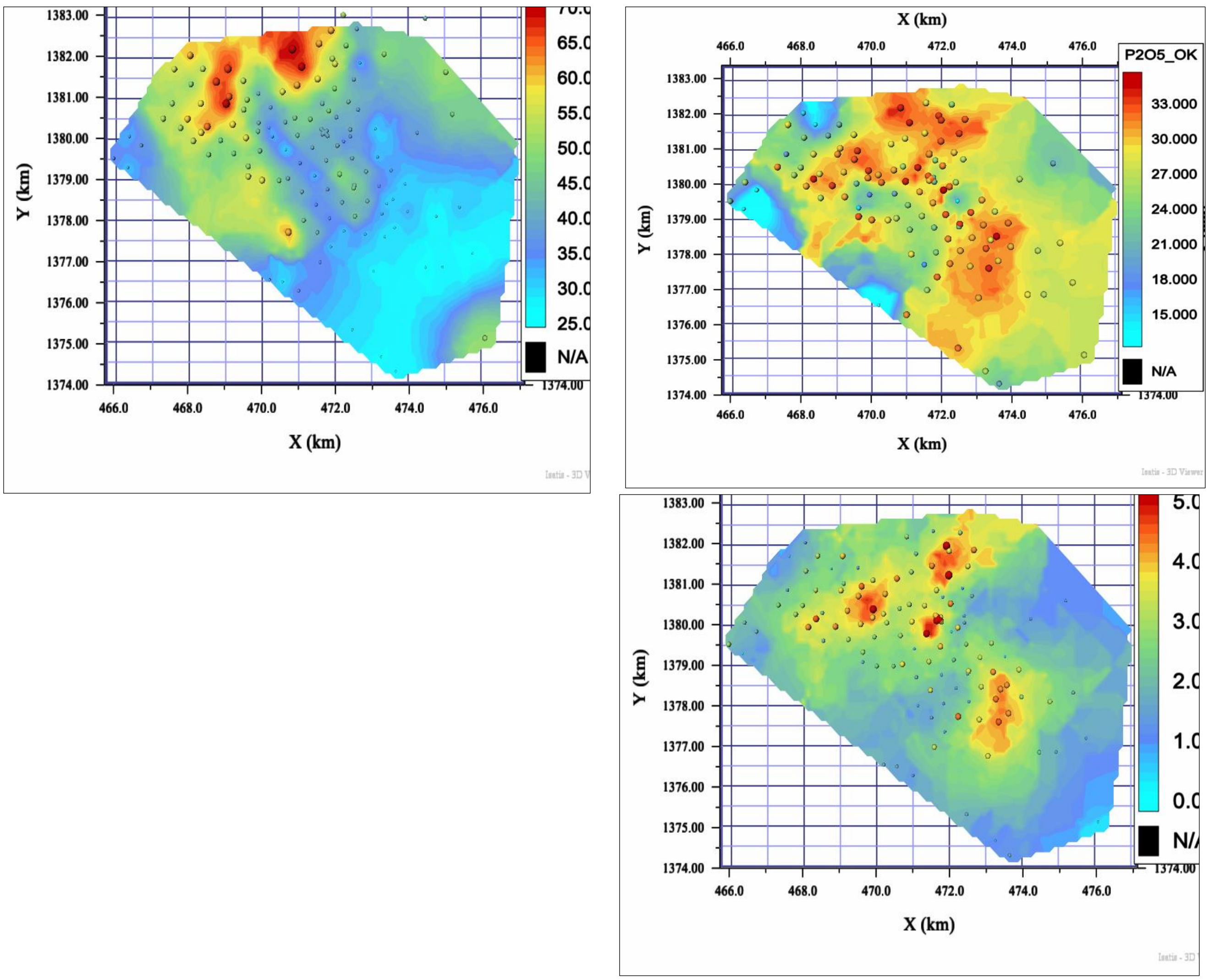
This is not a significant issue as there are very few low grades within the FPA layer. The block model shows good local reproduction of composite P_2O_5 grades. Similar plots for FPA thickness showed very good reproduction of composite values on a global and local scale.

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 drillhole 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 14-10 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.

Figure 14-10: Farim, Visual Validation – P_2O_5, FPA Thickness and Overburden Thickness



Source: WSP (formerly Golder) 2015

14.12 Mineral Resource Classification

The mineral resource classification for the project is based on the QP's assessment of confidence in the sampling data, geological knowledge, and geostatistical estimation.

The QP 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 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 mineral resources 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 River Cacheu 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.

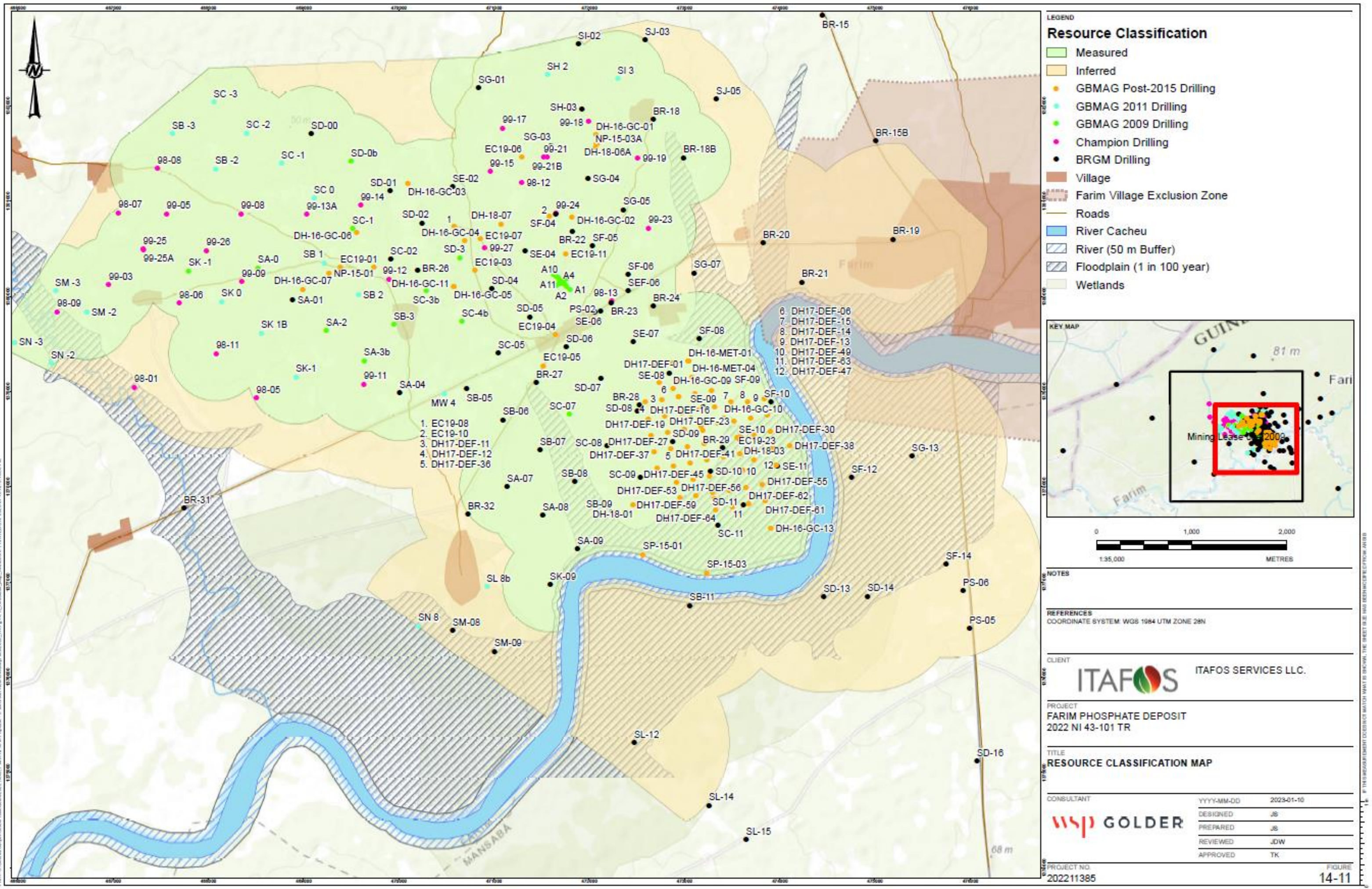
The resulting resource classification is shown in Figure 14-11. The resource estimate has also been divided into FPA corresponding to location relative to River Cacheu (i.e., "north" or "south").

14.13 Mineral Resource Statement

This section contains forward-looking information related to mineral resource estimates for the project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this subsection including geological and grade interpretations and controls and assumptions and forecasts associated with establishing reasonable prospects for eventual economic extraction.

Note to readers: The mineral resources presented in this section are not mineral reserves and do not reflect demonstrated economic viability. The reported inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that all or any part of this mineral resource will be converted into mineral reserve. All figures are rounded to reflect the relative accuracy of the estimates and totals may not total correctly.

Figure 14-11: Farim Resource Classification



Source: WSP Golder, 2023

The QP considers the mineralization contained within the Farim deposit to fulfil the criteria of "'Reasonable Prospects for Eventual Economic Extraction" to be reported as a mineral resource. A 20% P_2O_5 cut-off grade and a minimum FPA thickness of 1 m was applied by the QP to establish reasonable prospects for eventual economic extraction for the mineral resource estimate. The 20% P_2O_5 cut-off grade was applied to target the in-situ mineral resource grade requirements that would subsequently meet the plant feed and product grade requirements with the application of mine design and mineral processing considerations.

This differs from the 2012 Mineral Resource Estimate, 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 the QP'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 cutoff has been applied as the Lerchs-Grossman (LG) optimizations used to define the revenue factor (RF) 1.2 pit 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 Section 16.

Table 14-6 summarizes the assumptions used to develop the 2022 Mineral Resource Estimate. Table 14-7 summarizes the results of the September 30, 2022 mineral resource estimate based on a 20% P_2O_5 cut-off FPA thickness of 1.0 m and a constant density of 1.4 t/m^3; estimated mineral resources within the extents of the RF 1.2 pit design are provided in. Additional information regarding the RF 1.2 pit design is provided in Sections 15 and 16. The QP considers the criteria used to define the mineral inventory to be reasonable for public reporting. This assumes the mineral resource would be exploitable using open pit mining methods.

Table 14-6: Mineral Resource Pit Shell Parameters

Cutoff Parameter	Value
P_2O_5 Cutoff	20%
Overall Pit Slope Angle	20°
Mining Cost	US$1.69/tonne (ore); US$1.41/tonne (waste)
P_2O_5 Recovery	76%
Concentrate Selling Price	US$147/tonne
Matrix Density, dry basis	1.4 t/m^3
Overburden Density, dry basis	1.68 t/m^3

The global mineral resource estimate, dated September 30, 2022, defines a measured mineral resource of 102.5 Mt at a mean grade of 28.5% P_2O_5 and an inferred mineral resource of 31.1 Mt at a mean grade of 28.0% 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.

Table 14-7: Global Mineral Resource Statement, Farim Phosphate Deposit, September 30, 2022

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	102.5	2.91	28.53	2.69	39.71	5.65	11.28	1,162.30	11.34
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	**102.5**	**2.91**	**28.53**	**2.69**	**39.71**	**5.65**	**11.28**	**1,162.30**	**11.34**
Indicated	North of River	-	-	-	-	-	-	-	-	-
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	-	-	-	-	-	-	-	-	-
Measured + Indicated	North of River	102.5	2.91	28.53	2.69	39.71	5.65	11.28	1,162.30	11.34
	South of River	-	-	-	-	-	-	-	-	-
	Subtotal	**102.5**	**2.91**	**28.53**	**2.69**	**39.71**	**5.65**	**11.28**	**1,162.30**	**11.34**
Inferred	North of River	6.8	2.30	25.17	2.99	39.08	4.86	10.46	119.62	17.63
	South of River	24.4	2.21	29.06	5.32	36.21	4.97	11.62	236.18	9.70
	Subtotal	**31.1**	**2.23**	**28.08**	**4.73**	**36.94**	**4.94**	**11.32**	**355.80**	**11.42**

Notes: 1. Mineral resources are reported on a dry in-situ basis and are inclusive of mineral reserves. 2. The statement of estimates of mineral resources has been compiled by Mr. Jerry DeWolfe, who is a full-time employee of WSP Canada Inc. (formerly WSP Golder) and a professional geologist (P.Geo.) with the Association of Professional Engineers and Geoscientists of Alberta (APEGA). Mr. DeWolfe has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a QP as defined in NI 43-101. 3. All mineral resources figures reported in the table above represent estimates at September 30, 2022. Mineral resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies. 4. Mineral resources are reported in accordance with NI 43-101 and CIM Definition Standards for Mineral Resource and Mineral Reserves (2014) and CIM Estimation of Mineral Resource and Mineral Reserve Best Practices (2019). 5. The reported mineral resource estimate was constrained by a conceptual mineral resource optimized pit shell for the purpose of establishing reasonable prospects of economic extraction based on potential mining, metallurgical and processing grade parameters identified by mining, metallurgical and processing studies performed to date on the project. Key inputs in developing the mineral resource pit shell included a mining cost of US$1.69/tonne for ore and US1.41/tonne for waste, plus processing costs of US$31.72/ ROM tonne, phosphate recovery of 76%, pit slope angle of 20°, and a concentrate selling price of US$147/tonne. In addition, a 20% P_2O_5 cut-off grade, a minimum FPA thickness of 1 m as well as a restriction on any FPA within 50 m of River Cacheu was applied.

14.14 Mineral Resource Uncertainty

Mineral resources are not mineral reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this mineral resource will be converted into mineral reserve.

Inferred mineral resources are too speculative geologically to have economic considerations applied to them to enable them to be categorized as mineral reserves.

Mineral resource estimates may be materially affected by the quality of data, natural geological variability of mineralization and/or metallurgical recovery and the accuracy of the economic assumptions supporting reasonable prospects for economic extraction including metal prices, and mining and processing costs.

Mineral resources may also be affected by the estimation methodology and parameters and assumptions used in the grade estimation process including top-cutting (capping) of data or search and estimation strategies although it is the QP's opinion that there is a low likelihood of this having a material impact on the Mineral Resource estimate.

As the Farim project is quite advanced with a long history of study, the likelihood of unforeseen environmental, permitting, legal, title, taxation, socio-economic, marketing, and political risks to the project are low. However, the QP notes that every effort should be made to maintain a good relationship with the local communities and to ensure adherence to all local and federal environmental regulations.

15 MINERAL RESERVE ESTIMATES

The QP has applied the definitions of “mineral resource” and “mineral reserve” as set forth in the updated “CIM Definition Standards for Mineral Resources and Reserves” (May 10, 2014) by the Canadian Institute of Mining, Metallurgy, and Petroleum Council (CIM).

A “mineral reserve” is defined as the economically mineable part of a measured and/or indicated mineral resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include application of modifying factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The public disclosure of a mineral reserve must be demonstrated by a pre-feasibility study or feasibility study.”

CIM defines modifying factors as considerations used to convert mineral resources into mineral reserves (including, but not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, and governmental factors). The modifying factors used to convert mineral resources to mineral reserves are described in Section 15.3.

Mineral reserves are subdivided into two classes—probable mineral reserves and proven mineral reserves—that correspond to indicated and measured mineral resources, respectively, with the level of confidence increasing with each class. CIM has defined mineral reserves as follows:

- Probable Mineral Reserve –The economically mineable part of an indicated, and in some circumstances, a measured mineral resource. The confidence in the modifying factors applying to a probable mineral reserve is lower than that applying to a proven mineral reserve.
- Proven Mineral Reserve –The economically mineable part of a measured mineral resource. A proven mineral reserve implies a high degree of confidence in the modifying factors.

Except as stated herein, the QP 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 km² and is divided by the River Cacheu. The deposit consists of both FPA and FPB mineralized units. This mineral reserve estimate concerns FPA only, as the FPB unit was previously deemed to be uneconomic. No additional mineralization outside the modelled deposit was considered in the mineral resource and reserve estimates.

The reserve estimation was undertaken in Datamine’s MineScape™ software. The mineral reserve statement has a date of September 30, 2022.

15.2 Key Assumptions, Parameters, and Methods

The following key assumptions, parameters, and methods describe how the QP converted the mineral resources to mineral reserves. Open pit mining methods were considered for extracting the phosphate matrix at Farim. The primary mining equipment fleet includes use of front-end loaders, backhoes, and rigid-frame rear-dump haul trucks.

Ore will be extracted and hauled by truck to the process plant for concentration. Overburden will be used to develop water control infrastructure or stored in external overburden storage facilities or backfilled into the pit mined out areas. To estimate mineral reserves, a study was prepared under the supervision of the QP including open pit mine design and mining plans for Farim. The mining plans included annual stripping and ore production quantities. Annual production costs were estimated based on the mine plan quantities, open pit mining methods, proposed equipment fleets, prices bid by a mining contractor, and costs estimated from first principles. The open pit mine designs, mining plans, and production schedules are summarized in Section 16 of this report.

15.3 Modifying Factors

This subsection contains forward-looking information related to the modifying factors for the mineral reserve estimates for the project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts, or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this subsection including modifying factors such as dilution, mining recovery factors, beneficiation assumptions, property limits, commodity price, cutoff grades, pit optimization assumptions, and the ultimate pit design.

Modifying factors are applied to mineralized material within the measured and indicated resource classifications to establish the economic viability of mineral reserves. A summary of modifying factors applied to the 2022 Mineral Reserve Estimate is provided below.

15.3.1 Criteria for Determination of ROM Phosphate Matrix

Dilution in mining can be defined as the addition of waste material to the ore during the mining process. Dilution can be internal (caused by intrinsic deposit factors) or external (caused by operational factors). Dilution cannot be fully eliminated as it is impossible to have the exact accuracy of the mining limits; however, it can be estimated and considered, thus minimizing the differences between the mine plan and the actual operations.

Mining loss is the loss of in situ ore during the mining process. It is caused when ore is inadvertently mixed with roof or floor waste and sent to one of the waste storage locations (i.e., not sent to the plant). Like dilution, mining losses of in situ ore cannot be fully eliminated due to inherent inaccuracies of modeled vs actual ore zone limits but can be controlled during the mining process to minimize effects of recovered ore.

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 backhoes. 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.

15.3.2 Beneficiation Plant Yield and Product Quality Model

The effects of beneficiation on 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 Section 13 of this report. The results show a mass recovery of 74.3% for the North pit, 77.5% for the South pit, and a P_2O_5 product grade of 34%.

15.3.3 Development of the 3D Block Model for Pit Optimization

After developing the ROM surfaces, the grid-based MineScape model was blocked into 3D blocks of 25 m x 25 m x 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 14.6 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/m³ (dry basis) per the resource estimation methodology. The 3D block model was checked against both the 25 m x 25 m 2D Vulcan block model and MineScape reserves to confirm that data were honored, and that no volumes, tonnages, or assay data were altered. After review, the MineScape block model was compiled and exported for optimization purposes.

15.3.4 Cutoff Grade

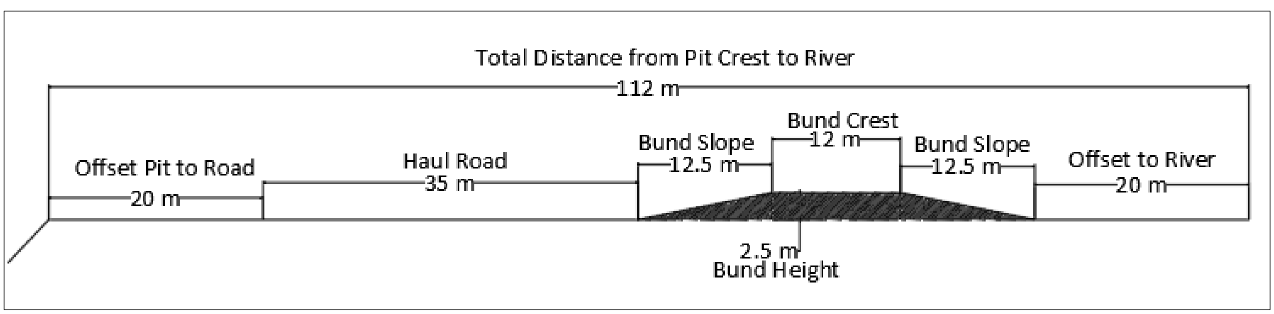
As per the mineral resource estimation methodology, a 20% P_2O_5 technical cut-off grade was applied to target the in-situ mineral resource grade requirements that would subsequently meet the plant feed and product grade requirements. This technical cut-off grade did not change in the reserve estimation. A minimum FPA thickness of 1 m as well as constraints as per section 15.3.5, were applied. A block was considered to be ore if the revenue generated by the block was greater than or equal to the total cost of mining that block.

15.3.5 Constraints Affecting the Mineral Reserve

The location of planned surface infrastructure limited the extent of the ultimate pit and correspondingly constrained the mineral reserve estimate. These constraints included:

- The River Cacheu, which divides the deposit north to south, was considered as a constraint in the mineral reserve estimate. Only that portion of the deposit lying north of the River Cacheu was included in the mineral reserve estimate.
- A 300 m wide exclusion zone adjacent was applied to the eastern ephemeral stream (Rio de Bunja) that flows near the current plant site. This exclusion zone was used to better manage surface water issues that would occur when mining through the ephemeral stream.
- Flood control infrastructure was also considered as a surface constraint, specifically the permanent flood control bund to be installed between the South pit and the River Cacheu. The conceptual measurements of the flood control bund are shown in Figure 15-1, indicating that the pit crest must be offset at least 112 m from the river's edge.
- The processing plant, tailings storage facility, and two ex-pit waste dumps (WD1 and WD2) served as surface encumbrances limiting the extent of the ultimate pit designs.

Figure 15-1: Conceptual Flood Control Bund Measurements



Source: WSP Golder, 2023

15.4 Pit Design

15.4.1 Pit Optimization

The economic limits of the open pits were determined using Datamine Studio NPVS software applied to the geological block models described in Section 14. The Datamine Studio NPVS software uses the industry-standard Lerchs-Grossman (LG) algorithm to assign an economic value to each block based on user-defined unit costs and other relevant input parameters and constraints such as dilution and mining recovery assumptions for mineral resource blocks. For a given revenue, the pit optimization process produces a pit shell that includes all economic mineral resource blocks within the limits of the pit shell. Economic mineral resource blocks are those blocks with a positive value at the assumed revenue parameters.

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, the 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.

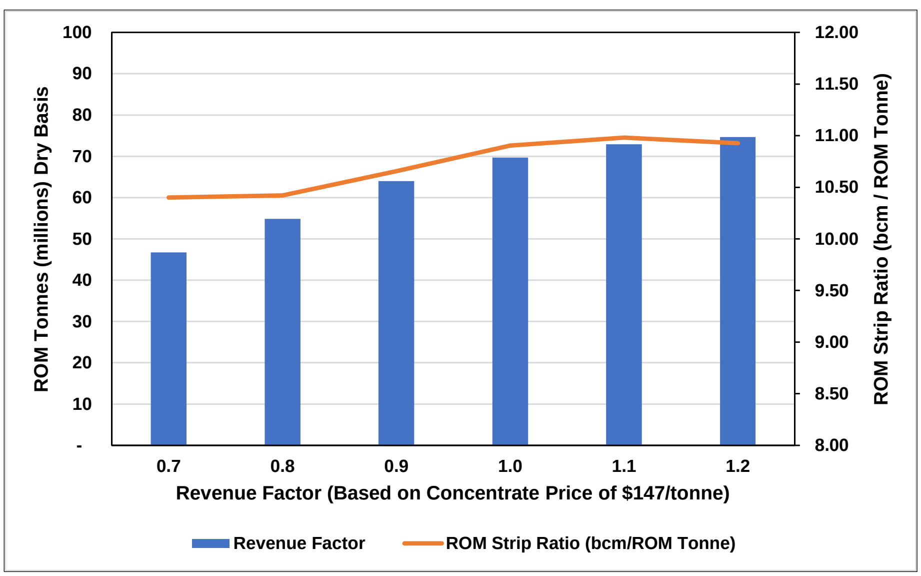
Optimization was conducted on measured and indicated resources only; inferred resources were treated as waste. The constraints described in Section 15.3.5 were also used for the pit optimization. The pit optimization resulted in two distinct pits. The optimized resource was defined as the matrix with the best available P_2O_5 grade and lowest resultant strip ratio. The resulting pit shell limits for these incremental pits were used as the basis for pit designs, mine planning, and reserve estimation.

Table 15-1: Summary of the Unit Costs Used in the Pit Optimization Analysis

Description	Value (US$ / Unit)	
Total Overburden Stripping Cost[1]	$2.37	per bcm
Total Matrix Mining Cost[2]	$1.69	per ROM tonne
Beneficiation	$31.72	per ROM tonne
Concentrate Price	$147.00	per tonne produced
Metallurgical Recovery	76%	-
Overall Pit Slope	20	degrees

Notes: 1. Cost includes overburden stripping and haulage, operations support, and mine maintenance. Cost assumes a diesel price of $0.86/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 $0.86/liter.

Figure 15-2: Incremental Value Pit Reserves Comparison



Source: WSP Golder, 2023

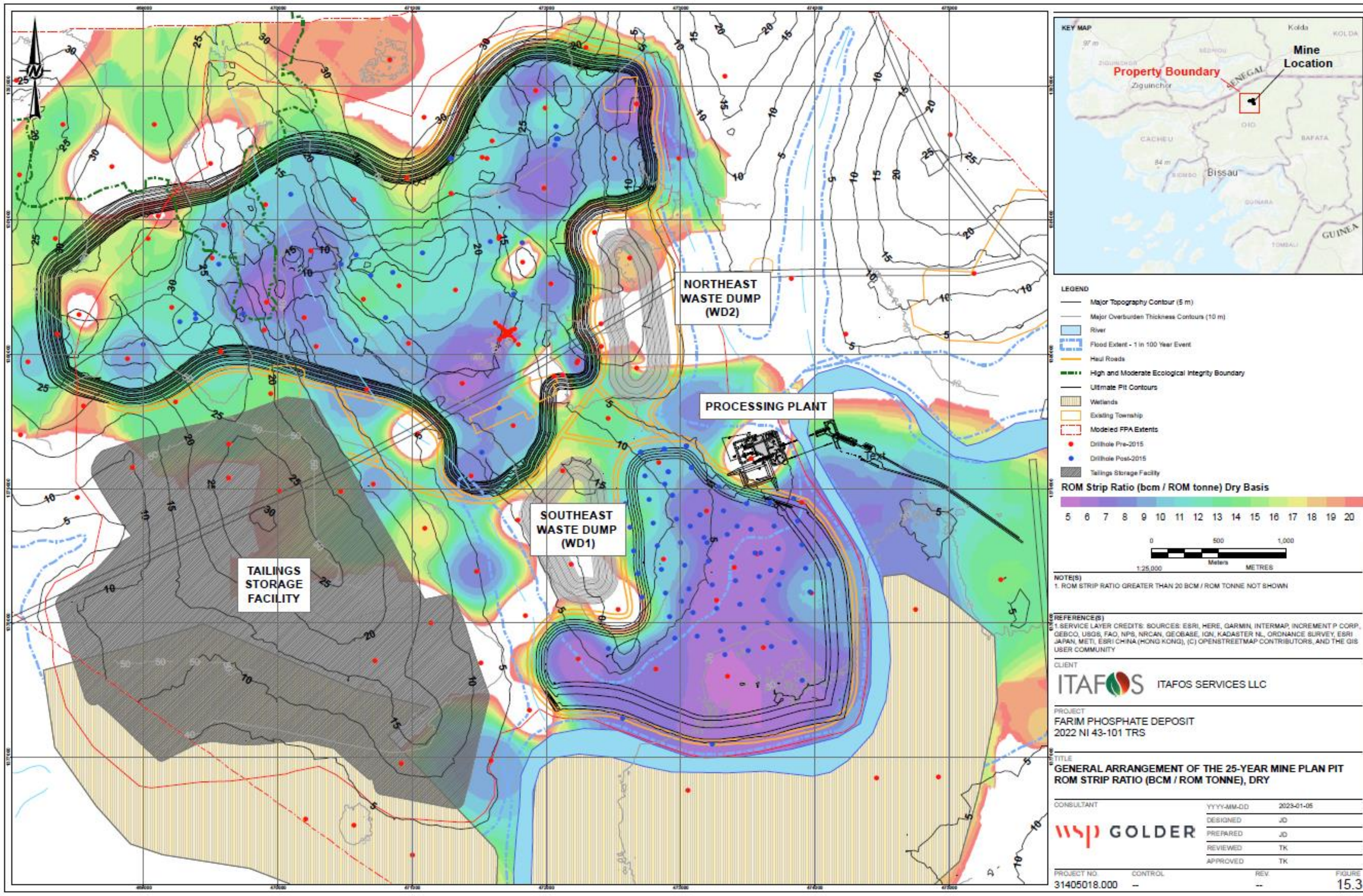
15.4.2 Ultimate Pit Design

The design criteria for the final pit configuration are shown in Table 15-2. The ultimate pit layout for the deposit was limited to a 25-year mine plan at 1.75 Mt ROM FPA per year. This yields a LOM FPA tonnage of approximately 43.75 Mt, which reflects the revenue factor 0.7 pit shown in the graph in Figure 15-2. The 25-year mine plan duration was chosen to limit the impact market uncertainty over the long term. Figures 15-3 and 15-4 provide general overview figures of the mine site with stripping ratio contours and P_2O_5 contours, respectively. The mine site is also shown in Figure 15-5 with aerial imagery.

Table 15-2: Summary Table of Mine Design Parameters

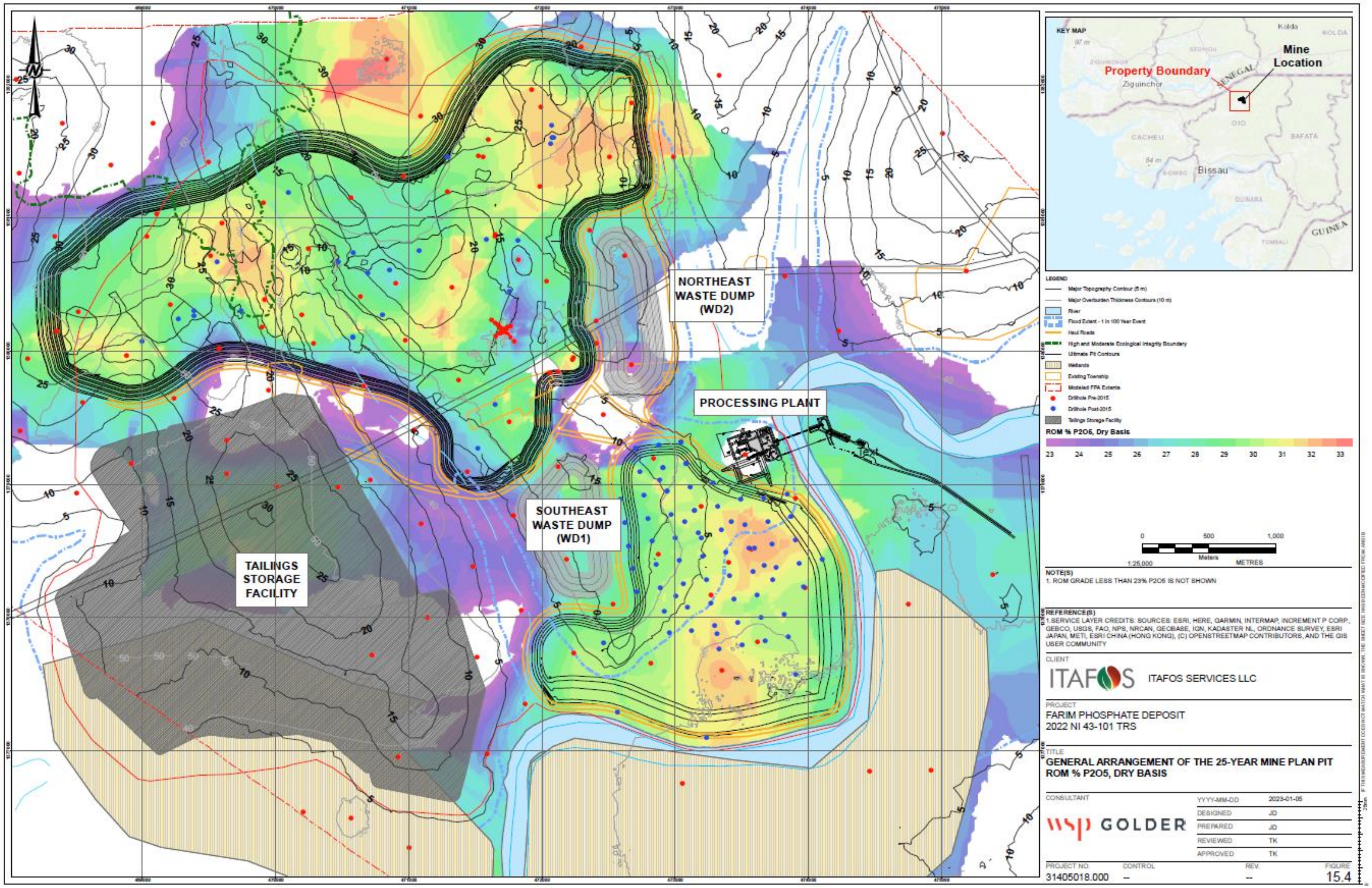
Description	Value
Permanent Wall Angle	20°
Permanent Wall Operational FOS	>1.3
Bench Height	10 m
Short-term Bench Face (Batter) Angle	65°
Short-term Berm Width	14.9 m
Long-term Bench Face (Batter) Angle (After Sloughing)	25°
Long-term Berm Width (After Sloughing)	6.5 m
Overburden Angle of Repose WD/IOB/SOS	1V:5H / 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

Figure 15-3: General Arrangement of the 25-Year Mine Plan – ROM Strip Ratio (BCM / ROM Tonne), Dry Basis



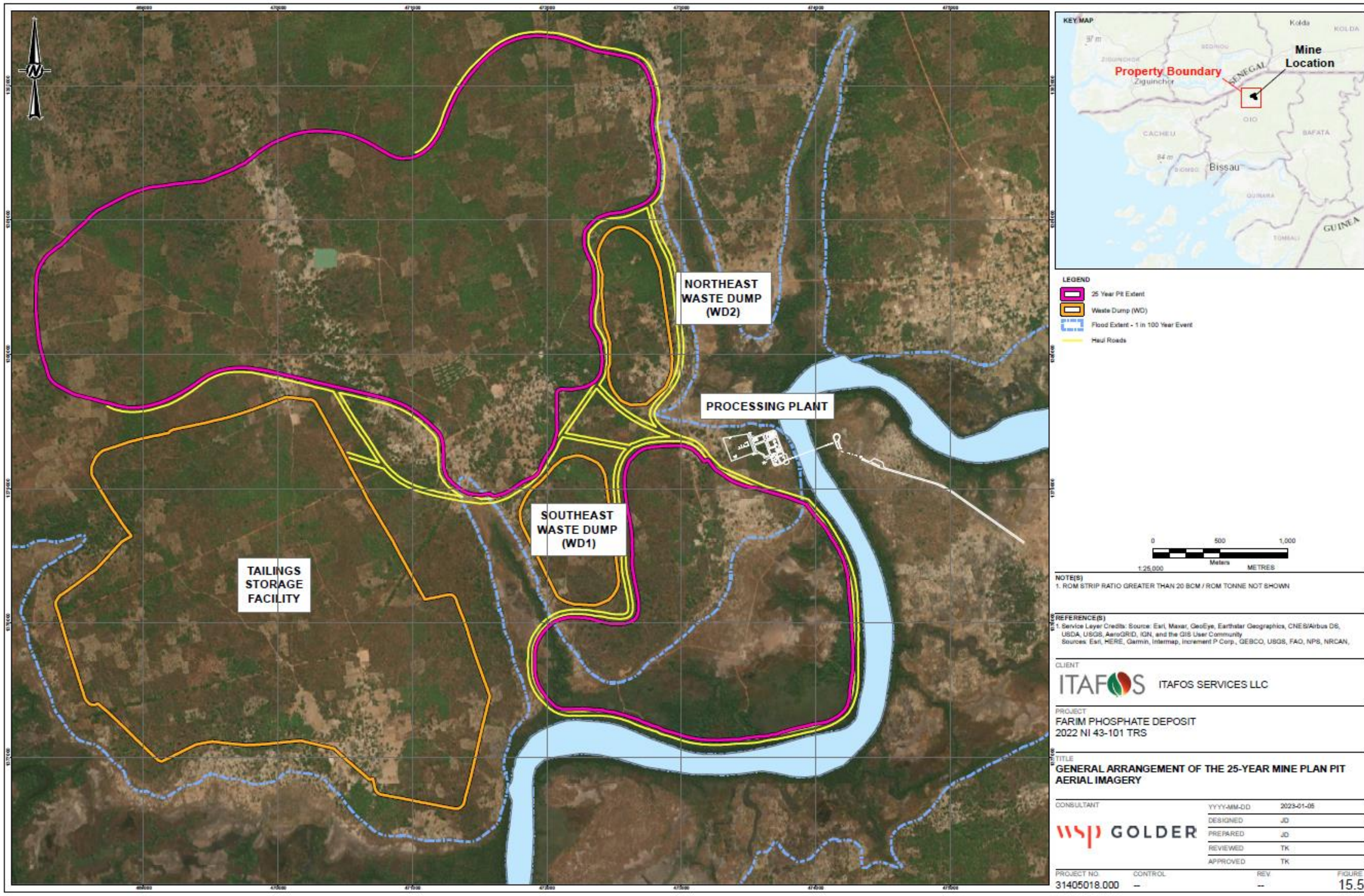
Source: WSP Golder, 2023

Figure 15-4: General Arrangement of the 25-Year Mine Plan – ROM %P_2O_5, Dry Basis



Source: WSP Golder, 2023

Figure 15-5: General Arrangement of the 25-Year Mine Plan – Aerial Imagery



Source: WSP Golder, 2023

15.5 Mineral Reserve Estimation Methodology and Statement

This subsection contains forward-looking information related to mineral reserve estimates for the project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this subsection including mineral resource model tonnes and product quality, modifying factors including mining and recovery factors, production rate and schedule, contractor mining equipment productivity, commodity market and prices and projected operating and capital costs.

The mineral reserve estimate for the project was prepared in compliance with industry accepted best practices.

The economic pit shells from the pit optimization were defined and used to limit the simulated mining sequences planned within each pit shell. Prior to sequencing, the QP applied a mining loss of 5% to the designated mineral resource blocks within the pit shell, based on the discussion in Section 15.3. Based on the mining sequence, overburden and mineral resource blocks were aggregated to produce estimated annual overburden and ore quantities and average ore grades. Based on the pit advance and blocks sequenced each year, production costs were estimated for the mining operations. The mining plan sequence and associated cost estimates are discussed in Section 16.8 and Section 21, respectively.

Using the geological model, modifying factors, and methods discussed in Section 15, the QP converted measured and indicated mineral resources described in Section 14 into the estimated mineral reserves shown in Table 15-3. The mineral resources stated in Table 14-7 are inclusive of the mineral reserve estimates shown in Table 15-3. The QP 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	ROM (Plant Feed) FPA Tonnes, Dry Basis (Mt)	Mean ROM P_2O_5, Dry Basis (%)	Mean ROM Al_2O_3, Dry Basis (%)	Mean ROM CaO, Dry Basis (%)	Mean ROM Fe_2O_3, Dry Basis (%)	Mean ROM SiO_2, Dry Basis (%)
Proven	43.8	30.0	2.6	41.1	4.8	10.6
Probable	-	-	-	-	-	-
Total	**43.8**	**30.0**	**2.6**	**41.1**	**4.8**	**10.6**

Notes: 1. Mineral reserves are reported on a dry in-situ basis. 2. The statement of estimates of mineral reserves has been compiled by Mr. Terry L. Kremmel, who is a full-time employee of WSP USA Inc. (formerly WSP Golder) and a professional engineer (P.E.) with the Society for Mining, Metallurgy, and Exploration (SME). Mr. Kremmel has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a QP as defined in NI 43-101. 3. All mineral reserves figures reported in the table above represent estimates at September 30, 2022. Mineral reserve estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies. 4. Mineral reserves are reported in accordance with NI 43-101. 5. The reported mineral reserve estimate was constrained by The River Cacheu, the Rio de Bunja, and surface encumbrances including the two ex-pit waste dumps, tailings storage facility, and processing plant.

The total estimated proven and probable reserves are 43.8 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 10.1 bank cubic meters (bcm) per tonne of ROM phosphate matrix, (17 tonnes overburden (dry) per tonne of ROM phosphate matrix (dry)), requiring the removal of approximately 441.5 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 15-6 on the following page.

The QP 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 loader and truck methods.

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

As stated in Section 14, the QP used a dry density of 1.4 t/m³ for the resource and reserve estimates. However, a previous resource estimate by others used a value of 1.43 t/m³ and 1.50 t/m³ 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.

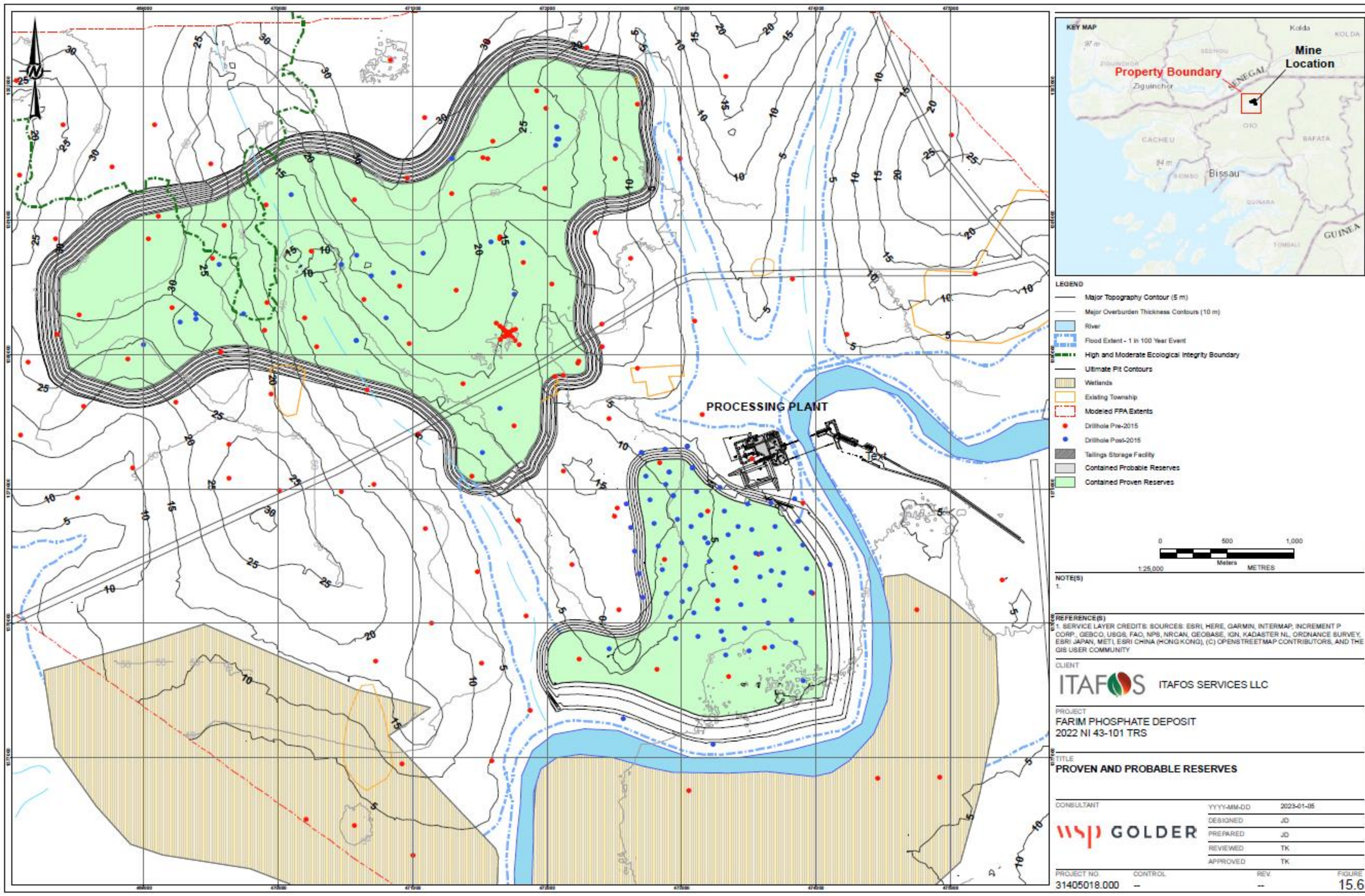
Failure to achieve reasonable outcomes in the following areas could result in significant changes to the resources and reserve estimates presented in this report.

1. Cost Estimates – Significant deviations to the operating and capital cost estimates described in Section 21 could result in a decrease in the amount of matrix which is economical to mine.
2. Extraction of the ore under physical conditions – Mining and hauling during the rainy season discussed in Section 16.2 will be challenging. Efforts were made to mitigate and reduce the impact of the heavy rainfall including haul road reinforcement and derating the equipment productivity during the rainy months.
3. Process metallurgical recovery – Bench-scale testing indicated a processing mass recovery of 74.3% for the North pit and 77.5% for the South pit. Failure to achieve a reasonable mass recovery during actual production may result in decreased mineral reserve quantities.
4. Equipment and labor productivity.

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 backhoes. Additionally, due to the lack of sampling, 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.

Market conditions and pricing for phosphate rock must remain favorable to support project development at Farim.

Figure 15-6: Proven and Probable Reserves



Source: WSP Golder, 2023

15.7 Potential for Future Reserve Expansion

As stated in Section 15.5, the mineral reserves estimate from September 30, 2022 is based solely on the 25-year mine plan open-pit design with highwall laybacks and a production rate of 1.75 Mt/a (dry basis). Resources outside of the 25-year pit extents were not considered in the mineral reserve estimate. There is a strong indication that future reserve expansion is possible through further economic evaluation. Future studies should investigate expanding reserves to include current resources outside of the 25-year pit (Figure 15-6).

16 MINING METHODS

16.1 Extraction Methodology

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.

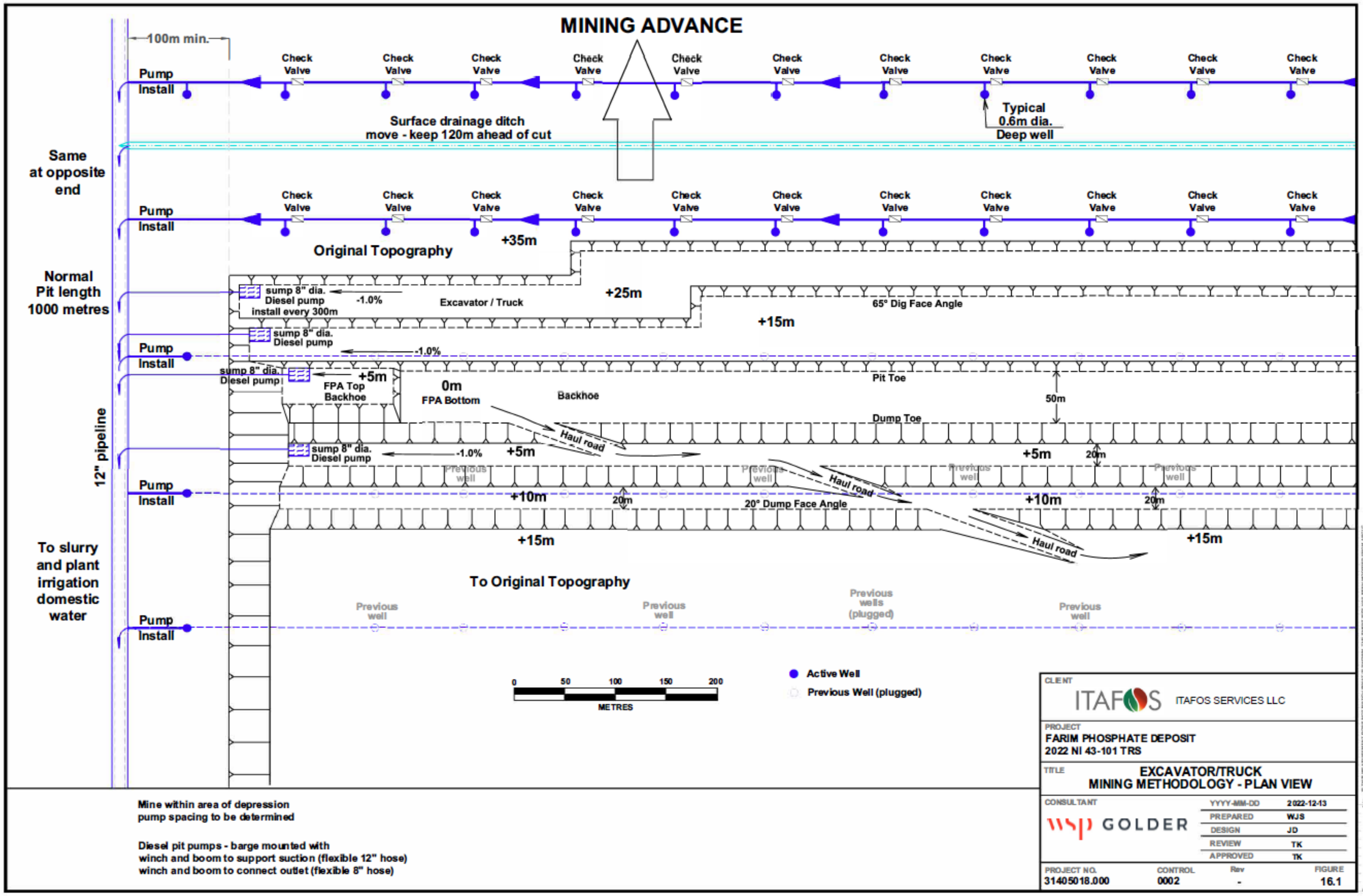
The FPA matrix is mined by a multiple-bench, open-pit, haul-back mine using backhoes and trucks. Initial pit development will consist of a boxcut that requires storage of overburden outside the pit. Once a sufficient volume has been excavated, the overburden is back-hauled into the mined-out area. Based on in-pit overburden backfill (IOB) design slopes and the size of the mined-out area required to allow overburden to 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 the pit will either be sent to an ex-pit waste dump (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 Mt/a (dry basis) open pit, it is planned that overburden will be stripped and removed with 12 m³ front-end loaders (FEL) matched with 97 t capacity haul trucks. The matrix will be mined with 5 m³ 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 blended into a plant feed hopper by front-end wheel loaders with 12 m³ buckets to achieve the desired plant feed 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 16-1 shows a typical pit configuration for this method of mining.

Figure 16-1: Base Case Loader/Truck Mining Methodology – Plan View

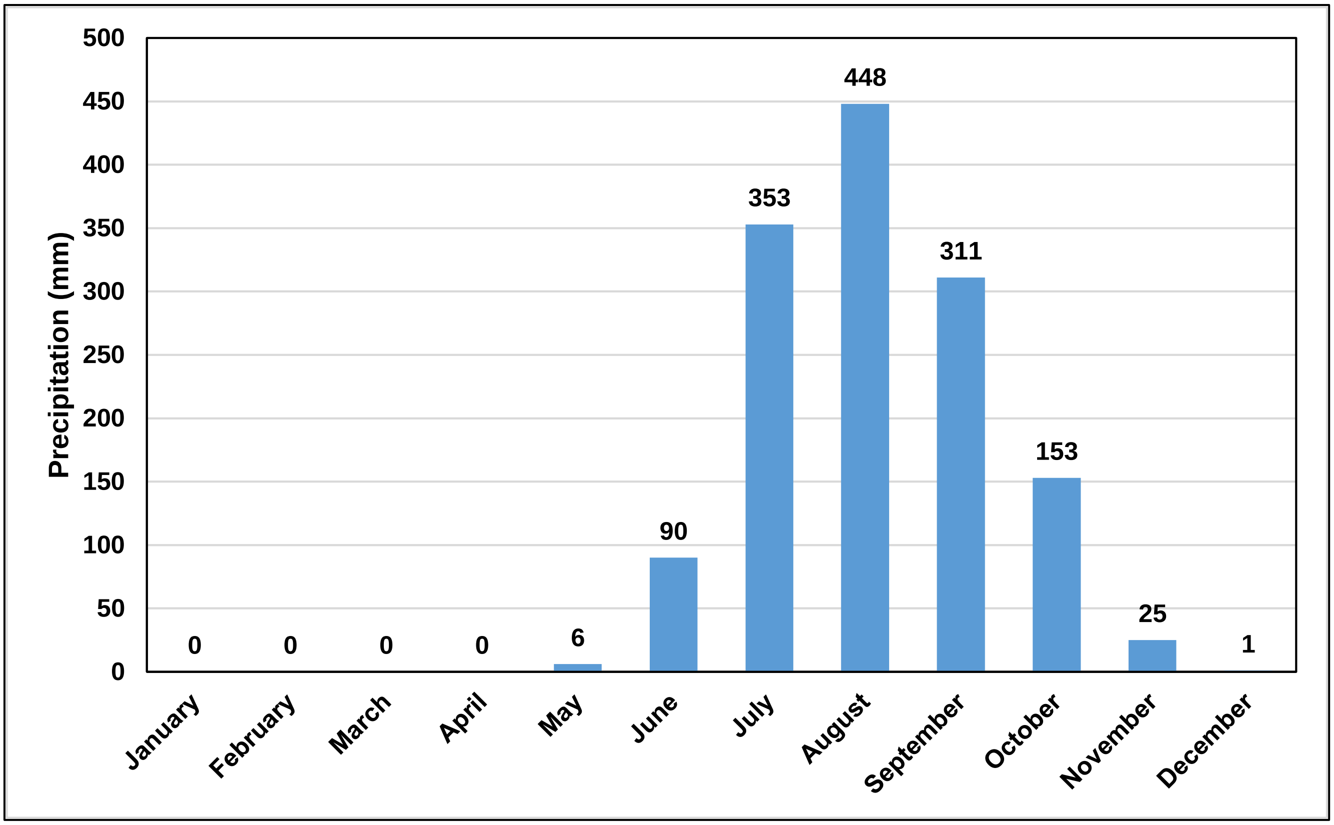


Source: WSP Golder, 2023

16.2 Surface and Groundwater Constraints

The project area experiences a five-month rainy season from June to October, with most rainfall occurring in mid-July to mid-September. From December through April, the country experiences drought with no significant rainfall. Rainfall data from 1991 to 2021 indicates almost 1,400 mm of rainfall per year with July and August experiencing the heaviest rainfall. Accumulated data from 1991 to 2021 indicates that approximately 25% of the rainfall occurs in July and 32% of the rainfall occurs in August. The average monthly precipitation data is provided in Figure 16-2.

Figure 16-2: Average Monthly Precipitation Data (1991 – 2021)



Source: en.climate-data.org/Africa/Guinea-Bissau185fricau/bissau-3095/

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 meters above mean sea level (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.

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 is planned for construction in stages along the south border of the pit and the northern bank of the river. The material required for construction of the flood protection bund will be sourced from the pit (i.e., overburden).

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 advanced 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 the surface water from entering the mining areas. Roads must be well-graded and crowned with a thick layer of pervious crushed rock. In-pit armored 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. These pumps must be able to handle at least 456 m^3 per hour. Mining will continue at decreased productivities during the wettest three 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. Road maintenance must be updated before each rainy season and maintained as much as possible during the heavy rain season. Failure to do so will result in operational inefficiencies and delays.

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

16.3 Hydrogeological Considerations

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

- 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
- 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 River Cacheu in the southeast. The groundwater elevations recorded between August 2009 and February 2012 ranged between -1.13 meters above mean sea level (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 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 River Cacheu 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 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 x 10^{-4} and 2.5 x 10^{-3} m²/s and a storativity range of between 1 x 10^{-5} and 1 x 10^{-3} are reported. For the limestone aquifer a transmissivity range of between 4 x 10^{-5} and 7 x 10^{-4} m²/s and a storativity range of between 2 x 10^{-4} and 4 x 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} m²/s) than the upper overburden aquifer (ranging between 3.1 x 10^{-5} and 1.8 x 10^{-3} m²/s). 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 River Cacheu) compared to the northern part of the proposed open pit area.

The pit slope stability models assume that the groundwater 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 during mining (Golder, 2015). The phosphate deposit lies above the limestone bedrock, and the pits will advance laterally throughout the mine life to a depth of -30 mamsl. The South pit will be mined first in a series of phases from north to south.

A series of studies has been conducted at the project to determine the hydrogeological properties of the various geological units. The data provided indicates a large variability in dewatering conditions due to the range of hydraulic properties of the overburden and bedrock. Dewatering targets identified include the sandy units of the overburden aquifer, the phosphate-bearing sandstones, and the weathered bedrock aquifer comprised of sandy clayey limestone underlying the ore deposit.

The objective of the dewatering strategy is to reduce the groundwater levels to below the working elevation of the operational areas according to the mining plan and schedule. Test pumping has confirmed that dewatering boreholes are a viable method to dewater the open pits in a staged approach.

16.4 Dewatering in Advance of Mining

16.4.1 Dewatering Strategy

The pit slope stability models assume that the groundwater 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 pits during mining (Golder, 2015).

The objective of the dewatering strategy is to reduce the groundwater levels to below the working elevation of the operational areas according to the mining plan and schedule. The phosphate bedrock lies above the bedrock, and the pits will advance laterally throughout the mine life to a depth of -30 m above mean sea level. The South pit will be mined first in a series of phases from north to south.

Pumping tests by Golder (2012) and KP (2018) indicate that dewatering of the pits using a series of dewatering boreholes is a feasible option, provided that the following criteria are met:

- The superficial Lodo clay aquitard provides a low permeability barrier between the water in the Rio Cacheu and the potentiometric head in the sandy aquifer unit, although it is not known if this pinches out in places.
- The aquifer unit is dominated by permeable sands.
- The bedrock and overburden geology are reasonably well understood.

The dewatering targets are the sandy units in the overburden aquifer as well as the phosphate layers and sandy limestone bedrock aquifer.

16.4.2 Groundwater Modelling

The groundwater numerical flow model (KP, 2015) was developed for the open pits based on the proposed mining plan at that time. The model was used to predict groundwater inflows during the various stages of development of the open pits (North and South) and the associated drawdown extents.

Modelled inflows throughout the simulation period that would need to be managed are summarized as follows:

- South pit average daily pit inflow is approximately 13,000 m^3/d, ranging from 9,800 m^3/d to 16,700 m^3/d.
- North pit average inflows are 6,500 m^3/d, peaking at 8,900 m^3/d and decreasing to 5,100 m^3/d at the end of mining (Year 26).

Based on the 2015 numerical flow model, dewatering simulations indicated the following:

- The assumed yield per dewatering borehole is 10.5 m^3/h (3 L/s).
- The initial number of bores required is 40, drilled to -70 m above mean sea level (about 70 m deep).
- As the pit is developed in phases, some boreholes will be decommissioned and new boreholes constructed.
- Over the operating life of mine, approximately 15 to 90 boreholes will operate at a time.
- A total of 418 borehole locations will be developed, of which 238 will be decommissioned during mining of the South pit and 18 following mine closure.
- Pit sumps will be required.
- The drawdown resulting from pit dewatering has the potential to impact a significant number of nearby water users.

The 2015 numerical flow model was not updated with the most recent field data collected in 2016 and 2017.

16.4.3 Dewatering Plan

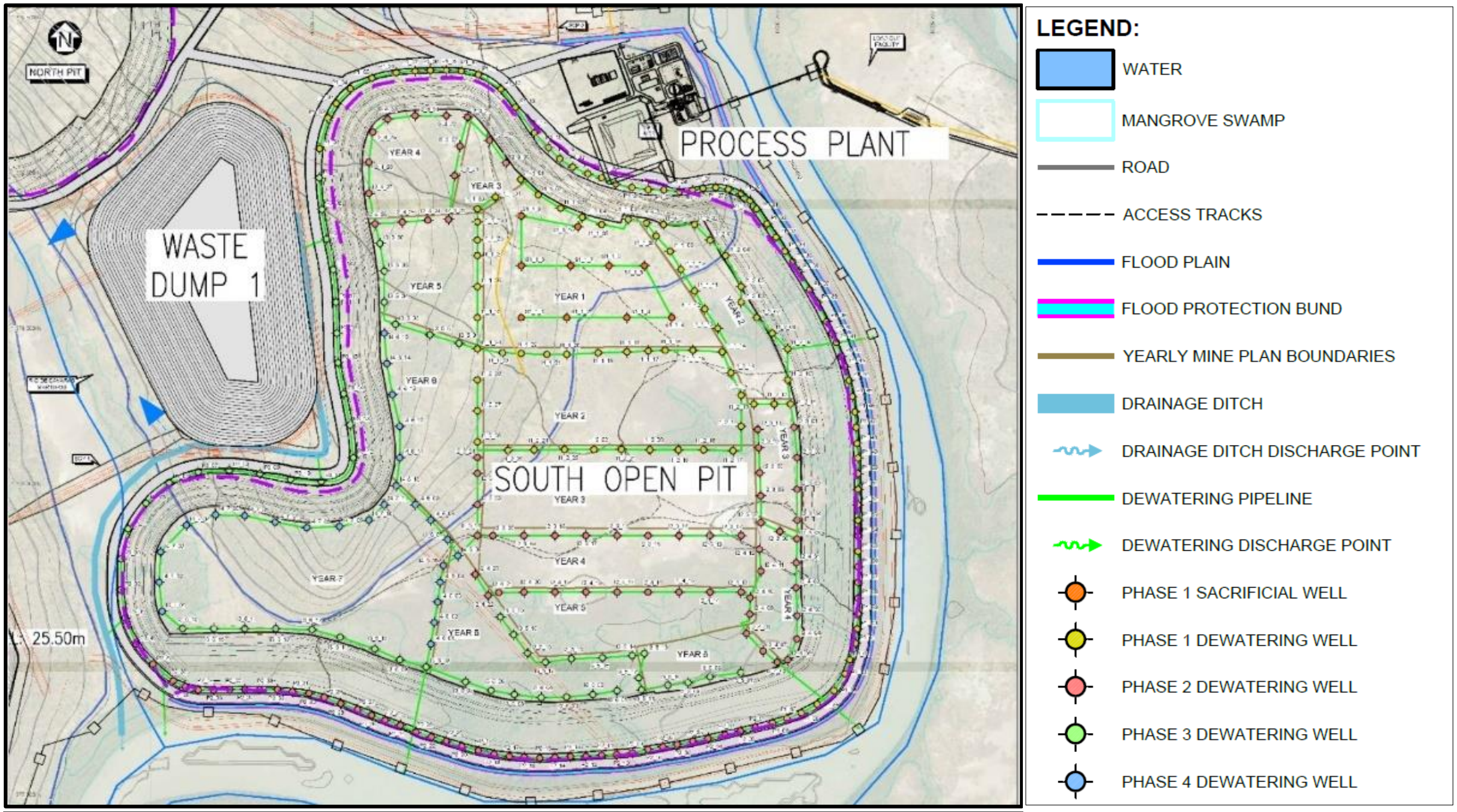
The improved dewatering plan consists of a phased installation schedule of dewatering boreholes, wellpoints, and pit sumps designed to prepare each area for pre-stripping and mining six months prior to the start of operations.

The most recent dewatering plan (KP, 2020) was developed using the revised overburden stratigraphy model and test pumping drawdown results from the South pit area. Pumping rates were estimated from 2016 to 2017 pumping tests at 1 L/s for the limestone bedrock aquifer and 14 L/s for the overburden aquifer.

The South pit will be mined first, and the proposed dewatering plan (see Figure 16-3) is as follows:

- Deep dewatering boreholes (60 m) will be placed around the perimeter of South pit and are designed to penetrate approximately 9 m into the limestone bedrock aquifer as well as draining the overburden aquifer. The spacing will be 50 m along the eastern and southern sectors of the pit that bound the River Cacheu, increasing to 100 m along the western and northern perimeters of the South pit.
- Shallow dewatering boreholes will focus on draining the shallow sand units of the overburden unit and are assumed to be 40 m deep. Dewatering wells will be installed along the outside pit perimeter of each interim yearly pit at 100 m spacing.
- Wellpoints spaced 1 to 3 m apart and up to 6 m in depth will be placed locally along the toe of slopes to dewater mining areas. These costs are assumed to be included in the mining contractor costs.
- The dewatering boreholes should be screened with stainless steel casing due to the moderate to high corrosivity of the groundwater.
- In-pit sumps will capture surface water runoff during the wet season and from the wellpoint arrays.

Figure 16-3: South Pit Dewatering Plan Layout



Source: Knight Piesold, 2023

- Groundwater monitoring boreholes will be added to the existing monitoring network to measure the performance of the dewatering system.
- The dewatering plan for South pit includes a total of approximately 313 deep dewatering boreholes, 55 shallow dewatering boreholes, and 50 monitoring boreholes.
- By staging the dewatering program, the upfront dewatering infrastructure can be reduced to 40 boreholes and thereafter additional boreholes added. This will provide the opportunity to update the groundwater model to define the future dewatering requirements more accurately.

16.4.4 Verification of Dewatering Plan

A sample section of the Phase 1 boreholes was modelled using EPANET hydraulic modelling software to create a snapshot view of the dewatering plan (KP,2020) to identify any shortcomings in the proposed dewatering plan.

The model contained the following:

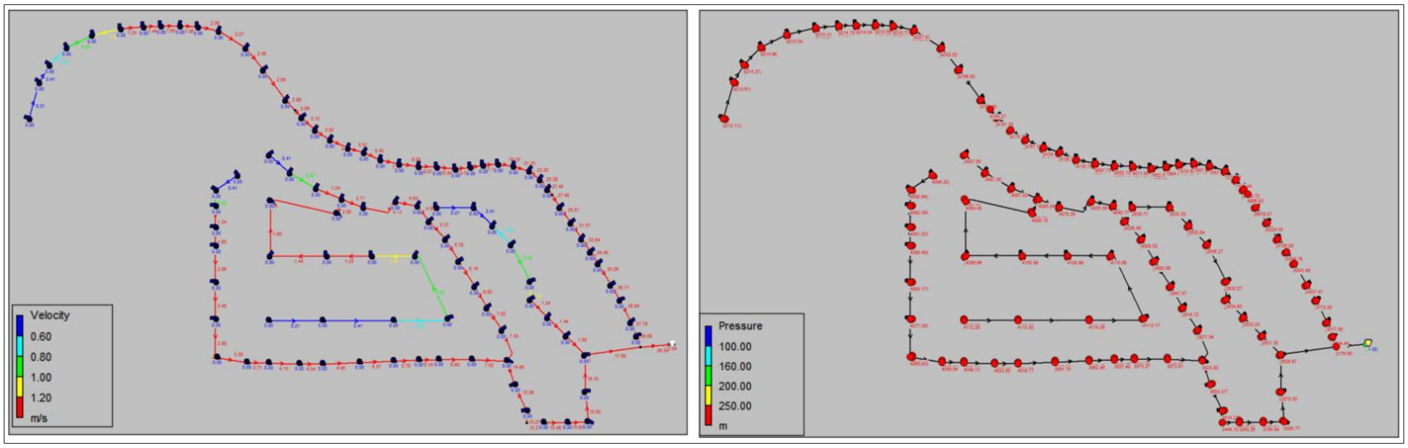
- perimeter wells from P1_01 to P1_41 along the north crest of the pit
- sacrificial wells S1_1_1 to S1_1_10
- Year 1 interim wells I1_1_01 to I1_1_31
- Year 2 interim wells I2_2_01 to I2_2_14
- surface pipes as ND90 SDR17 (PN10) PE100 HDPE pipes with an inner diameter (ID) of 78.55 mm
- pumps with the following duty points:
    - static head = 50 m
    - total head = 60 m
    - maximum flow = 17 L/s
    - minimum flow = 1 L/s
- typical pump curve for an 8-inch 60 Hz three-phase deep well submersible pump
- seven drainage points around the perimeter of the South pit in accordance with the January 2020 South Pit Dewatering Plan (KP, 2020)
- surface infrastructure elevations based on the surface data received.

The following key parameters were analyzed in the resultant hydraulic model:

- Velocity – Ideal velocities for an efficient pipe network are between 1.0 and 1.2 m/s. These velocities result in acceptable unit head losses between 2 and 5 m/km.
- Pressure – The model was assessed for pressures under 100 m (PN10) in accordance with the pressure rating of the pipe proposed in the original 2020 concept.

The results from the model showed that the velocities in the pipes far exceed the ideal velocity of 1.2 m/s for a 90 mm pipe. Pressures in the system were also in excess of 100 m due to the restricted flows in the pipe network. Figure 16-4 shows the velocities and pressures in the hydraulic model.

Figure 16-4: Velocity and Pressures – Sample of Phase 1 Dewatering System – 2020 Proposal



Source: KP, 2020

The following amendments were made to the network to achieve the desired velocities and pressures:

- An additional discharge point was installed between perimeter wells P1_31 and P1_32. This is a region of high flow, so the removal of water from the system at frequent intervals in this section eliminates the need for long lengths of large diameter pipes.
- Pipe diameters were increased incrementally in the direction of flow toward the discharge points. Pipe sizes ranged between ND90 and DN450 at the discharge points.

Figure 16-5 depicts the increases in pipe diameters and the position of the additional discharge point in the EPANET model. After increasing the pipe diameters and adding the discharge point, the resultant velocities and pressures are within the acceptable ranges (Figure 16-6).

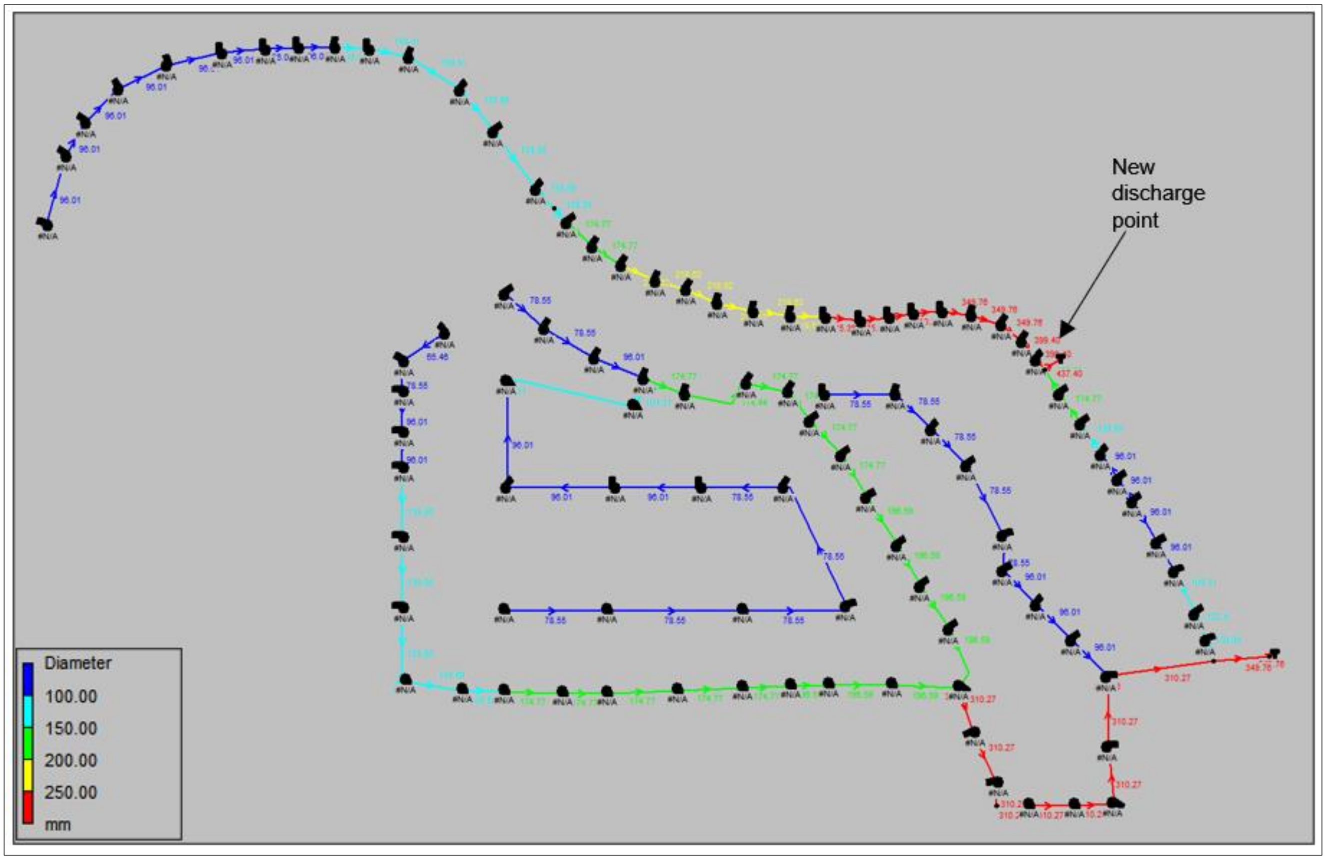
This verification exercise was limited to a portion of the Phase 1 proposed dewatering plan for the South pit to identify any shortcomings in the dewatering plan.

It has been assumed that the amendments applied to this sample of the dewatering system will be undertaken for the remainder of the system. The cost estimates have been updated by applying a percentage increase in pipe diameters implemented in this section to the remainder of the South pit dewatering system.

It should be noted that the system that has been modelled does not contain any updated borehole data. Expansion of the mining operational areas and additional flows have not been accounted for in this model.

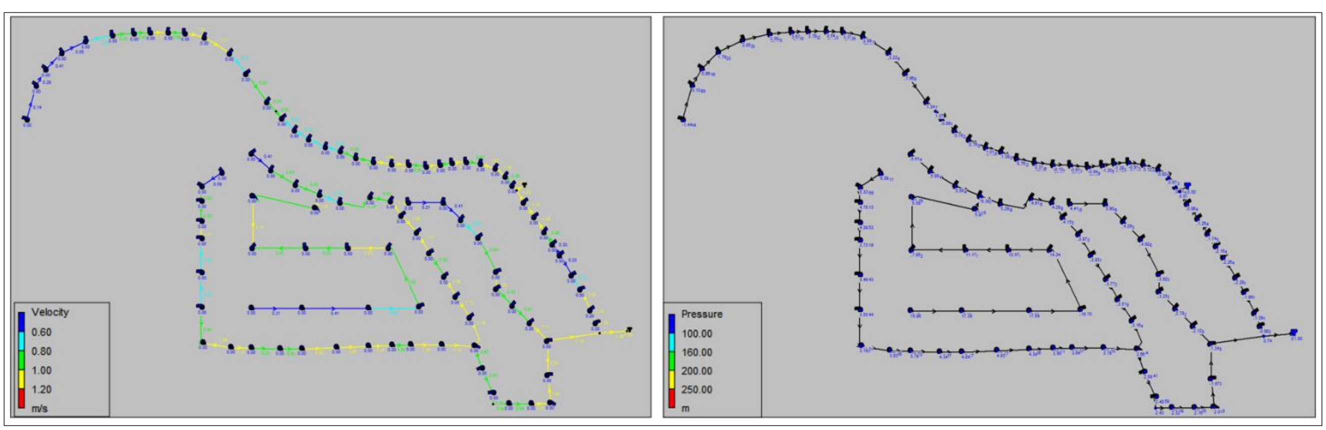
It is recommended that a complete hydraulic model be prepared to analyze pipe sizes and optimize the dewatering design of the South and North pits during detailed engineering.

Figure 16-5: Increased Pipe Diameters and New Discharge Point – Sample of Phase 1 Dewatering System



Source: KP, 2023

Figure 16-6: Velocity and Pressures – Sample of Phase 1 Dewatering System – 2022 Update



Source: KP 2023

16.5 Mine Design Criteria

The QP has performed an update to the pit slope design geotechnical study detailed in Section 16.7. In the study, the QP 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. The QP 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, the QP recommends an average overburden swell factor of 27%. This swell factor is applied to the ex-pit WDs, 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 WDs 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 vertical-to-horizontal slope of 1:4, and SOS and IOB are designed to an overall vertical-to-horizontal slope of 1:6. WDs will be built in lifts and compacted with a dozer and compactor. Sections 16.8.2 and 18.8 detail 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% 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 deposit will require mine haulage trucks. Loader/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 loader/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 is covered in Sections 16.7.1 and 16.8.3.

The proximity of the mine site to the River Cacheu 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 begin construction of 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. Construction of the flood bund for the South pit will progress in stages from Year 0 through Year 7, with construction of a flood bund for the North pit to follow. 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 112 m in width to allow construction of the bund to an elevation of 4 mamsl with vertical-to-horizontal slopes of 1:5, a crest width of 12 m, a river buffer of 20 m, and a pit side buffer of 55 m to allow for a 35 m wide haul road and a 20 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. Additional discussion on the design of the flood control bund is found in Section 18.11.2.2.

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.

The mine plan parameters and factors were previously summarized in Table 15-2.

16.6 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 x 25 m grid. Triangulation surfaces for the FPA roof and floor were also provided. The 2D geological model data was imported into Datamine's MineScape™ software 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 honored 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/m³ (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 16-1.

Table 16-1: 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
0.5	0.1	0.075	5%	0.475	90%	30	25.3	76%
1.0	0.1	0.075	5%	0.975	93%	30	27.7	86%
1.5	0.1	0.075	5%	1.475	93%	30	28.5	89%
2.0	0.1	0.075	5%	1.975	94%	30	28.9	90%
2.5	0.1	0.075	5%	2.475	94%	30	29.1	91%
3.0	0.1	0.075	5%	2.975	94%	30	29.2	92%
3.5	0.1	0.075	5%	3.475	94%	30	29.4	92%
4.0	0.1	0.075	5%	3.975	94%	30	29.4	93%
4.5	0.1	0.075	5%	4.475	94%	30	29.5	93%
5.0	0.1	0.075	5%	4.975	95%	30	29.5	93%
5.5	0.1	0.075	5%	5.475	95%	30	29.6	93%
6.0	0.1	0.075	5%	5.975	95%	30	29.6	93%

Notes: In-situ P_2O_5 grade for demonstration purposes only

As demonstrated in Table 16-1, 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 16-2.

After developing the ROM surfaces, a 3D block model with blocks measuring 25 m x 25 m x 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 14.6 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/m³ (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 honored, and that no volumes, tonnages, or assay data had changed.

Table 16-2: Effects of Mining Methodology and Beneficiation on FPA Matrix Recoveries and Grades

Parameter	Value
FPA Thickness (m)	3.5
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	5.0%
ROM (Recovered) Thickness (m)	3.30
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
Mass Recovery (%) – North Pit	74.30%
Mass Recovery (%) – South Pit	77.50%
Product %P_2O_5 (Dry Basis) [1]	34.00%

16.7 Geotechnical Parameters

The mine site is bounded by the River Cacheu to the east and south of the Farim South pit. The plant area is located to the northeast of the south open pit, between the pit and the River Cacheu.

The current mine plan applies 8H:1V slope angles to the upper 15 m along the southern and eastern perimeter of the South pit where the pit is adjacent to the River Cacheu and the presence of very soft organic clays (OPA-1A), also referred to as Lodo clays, has been documented. The subsurface investigation information from the 2019 field investigation (Golder, 2020a) appears to indicate the OPA-1A unit in the Years 1 and 2 pit area is thinner than in the southeast and south (Years 3 through 7) pit areas. Golder evaluated the lateral stratigraphy along the river (east and south) using cone penetration tests (CPT) and boring data, and the geological profiles are presented in Figures 16-4 and 16-5, respectively. The 2019 investigation (Golder 2020a) data generally confirms the 15 m thickness of the OPA-1A unit at the final pit crest in the eastern and southern pit areas (Years 3 through 7). However, the current spacing of borehole and CPT information is not adequate to define a detailed variability of the thickness of the OPA-1A unit in the Years 1 to 7 pit areas to permit the development of a reliable revision to the mine plan.

Based on the limited data available, the lateral variability of the materials in the slopes, Golder conservatively selected six geological profiles extending to bedrock for slope stability analysis. These sections are presented in Figures 16-9 and 16-10 and stability modelling is discussed in greater detail in Section 16.7.4.

The current mine plan has two waste dumps (WD 1 and WD 2) located between the north and the South pits. WD 1 is approximately 85 m away from the pit crest that includes 35 m wide haul road and 20 m offset to the pit crest at east walls of the South pit.

16.7.1 Geotechnical Field Investigations

16.7.1.1 Summary

Table 16-3 lists the summary of previous geotechnical investigation programs carried out for the open pit area (OPA) at the Farim phosphate project.

Table 16-3: Previous Geotechnical Investigation Programs

Geotechnical Investigations	Number of BHs & CPTs	Investigation Area
2011-2012 – Golder Associates UK (Golder, 2011), (Golder, 2012)	22 BHs	North and south OPAs.
2016 – Knight Piésold (KP, 2016)	9 BHs	South and east perimeter of the South pit along the River Cacheu.
2017 – Golder Associates (Golder, 2017)	11 BHs	Along two fence lines perpendicular to the pit slope.
2019 – Golder Associates (Golder, 2020a)	7 BHs 52 CPTs	Close to the proposed plant site and along the pit crest; and adjacent to the river course on the proposed east slope.

16.7.1.2 Standard Penetration Testing

16.7.1.2.1 Field Vane Testing

A total of 37 field vane shear tests were completed from 18 borings during KP's (2016) and Golder's (2017 and 2019) field investigations to target the soft clay layers (OPA-1A and OPA-1). Mobilized shear strength of soils was also calculated. Details of field vane test data were compiled and reported in Golder's 2020 stability evaluation report (Golder, 2020b), and developed values are reported in Section 16.7.5.1. Based on field vane data, Golder estimated undrained shear strengths, S_u, for organic clay (OPA-1A) for soft clay (OPA-1). These field vane test results were also used to the calibrate the CPT cone factor, N_{kt}, when estimating undrained shear strength from CPT.

16.7.1.2.2 Cone Penetration Testing

A total of 52 CPTs were advanced through the South pit areas during the 2019 geotechnical investigation (Golder 2020a). The cones were hydraulically advanced using the CPT rig at a controlled rate to provide continuous data, including measured tip resistance, sleeve friction, and pore pressure. All CPTs were advanced until refusal and ranged in depth from approximately 2 to 36 meters below ground surface (bgs). Between 4 and 10 pore pressure dissipation tests were conducted in select soundings to estimate static water pressure levels in the subsurface.

16.7.2 Geotechnical Characterizations

16.7.2.1 Geotechnical Units

Previous studies identified several lithologic layers in the South pit area that are considered as geotechnical units for this pit slope stability study. These layers are shown on cross-sections on Figure 16-7 through Figure 16-11. These units are summarized and referred to by the following names in this report:

- Topsoils: Generally brown or grey, low plasticity clays and silts, with few to some sand and none too few gravels. Rootlets, organic material, and duricrust nodules are prevalent. Topsoils vary in composition, density, and thickness across the site, ranging from very soft to firm and 0.5 to 4 m deep.
- Soft Clays (OPA-1): This clay unit was identified in the 2012 investigation. These materials consist largely of grey or mottled orange, medium to high plasticity clays with some sand. These clays are typically found at depths between 4 and 22 m deep across the site and range from soft to firm. The material is thickest on east pit slope along cross-section EW2.
- Organic Clays (OPA-1A): This material consists generally of grey, high to very high plasticity organic clays with trace sand. Occasional root material is present at depths of 0 to 6 m. These clays are typically found at shallow depths between 2 and 13 m across the site and overly the soft clays (OPA-1) unit or the upper sands (OPA-2). The material is thickest (10 to 20 m) nearest to the river (e.g., along cross-sections NS1 on south slopes and EW1 on east slopes) on both south and east pit slopes and taper out with distance away from the river. It has limited thickness on north slopes of the South pit (e.g., along cross-sections NS2 and EW3). This unit is very soft such that a standard penetration sampler can be pushed through the material with only the weight of the hammer.
- Stiff Clays (OPA-1B): This material consists generally of yellow to reddish brown medium to high plasticity sandy clays. These clays are typically found at depths between 2 and 22 m in the north area of the pit and in the area of the proposed future plant and waste dump (e.g., along cross-sections EW5 on west slopes and cross-sections NS2 and EW3 on north slopes). The material is typically found away from the river and on west slopes. This unit is classified as hard, such that a cone penetrometer often met refusal. OPA-1B applied in this report corresponds to the unit that was referred to as OPA-1 in the 2012 and 2015 feasibility reports.
- Upper Clayey Sands (OPA-2): Generally orange, yellow, or brown, clayey and silty sands. This unit is typically found at depths between 4 and 22 m across the site underlying the upper clay units. The sands were found to be medium dense to very dense. The material is partly expected to form the middle bench slopes of north, south and east walls of South pit.
- Middle Clays (OPA-3): Generally grey, orange, yellow, or brown silty clays with trace to some sand. These sands are typically found at depths between 12 to 30 m across the site and range from firm to stiff. The material is expected to form the lower bench slopes of north and south walls of South pit.
- Lower Clayey Sands (OPA-4): Generally orange, yellow, or brown, clayey and silty sands. This unit is typically found at depths between 4 and 22 m across the site and ranges from dense to very dense. The material is expected to form the lower bench slopes of east walls of South pit.
- Phosphate Sands: This unit contains phosphates and ranges from brown-grey sandy clays and silts to high plasticity clayey sands. This unit is consistently found in a 3 to 5 m thick layer at depths between 29 to 35 m across the South pit site. The unit is very dense.

- Weathered Limestone/Bedrock: The weathered limestone bedrock unit immediately underlies the phosphate sands consistently throughout the site at depths between 30 to 35 m.
- Fill/Bund Material: Source currently unknown, but material is assumed to be sourced from imported silty sands and gravels. Bund material will be used to create a flood barrier between the river and the pit along the crest.

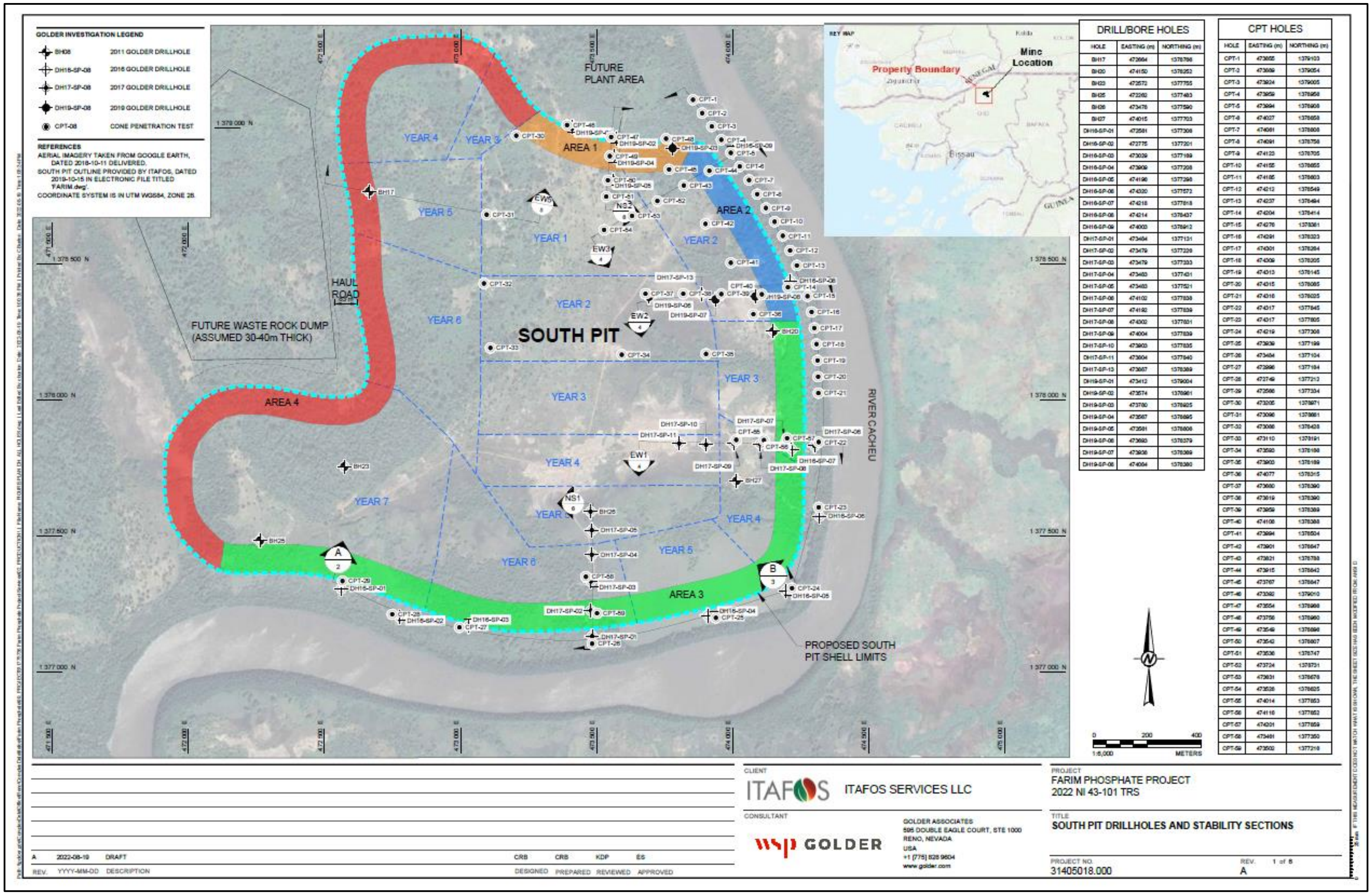
16.7.2.2 Geotechnical Domains

For purpose of slope stability analysis and evaluations, Golder divided the South pit area into four areas (i.e., Area 1 to 4) based on the following considerations as presented in Figure 16-6:

- The thickness and presence of organic clays (OPA-1A) interpreted along cross-sections based on available site investigations. Organic clays were identified to have weakest shear strength among other soil layers in South pit area, and flatter slopes for the pit walls were considered when organic clays form them.
- Proximity of pit slopes to River Cacheu. The flood protection bund design and offset distances to pit slopes provided by KP were evaluated along south and east walls of South pit.
- Mine facilities adjacent to the pit crest. Loading from the facilities and safe setback distances to the pit crest were evaluated from a slope stability perspective.

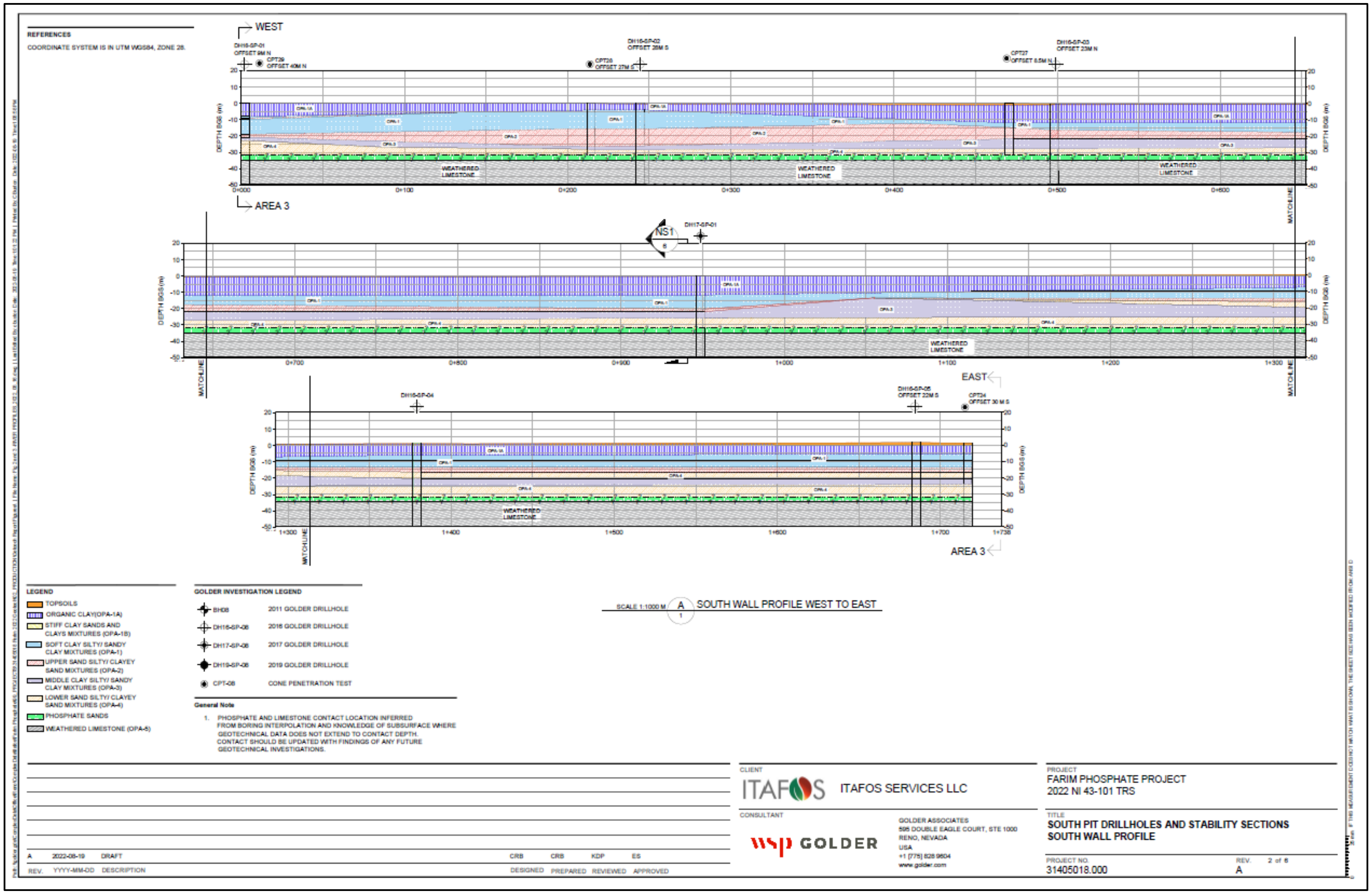
Golder prepared and updated soil profiles along south and east walls of South pit as presented in Figure 16-6. The domain Areas 1-4 were developed based on where the flood protection bund is proposed, available geotechnical subsurface data, and horizontal and subsurface variability of soil layers. The domains are used for evaluating representative critical sections for geotechnical slope stability analysis. Representative profiles are presented in Figure 16-5 through Figure 16-11 for the South pit area.

Figure 16-7: South Pit Drillholes and Stability Sections



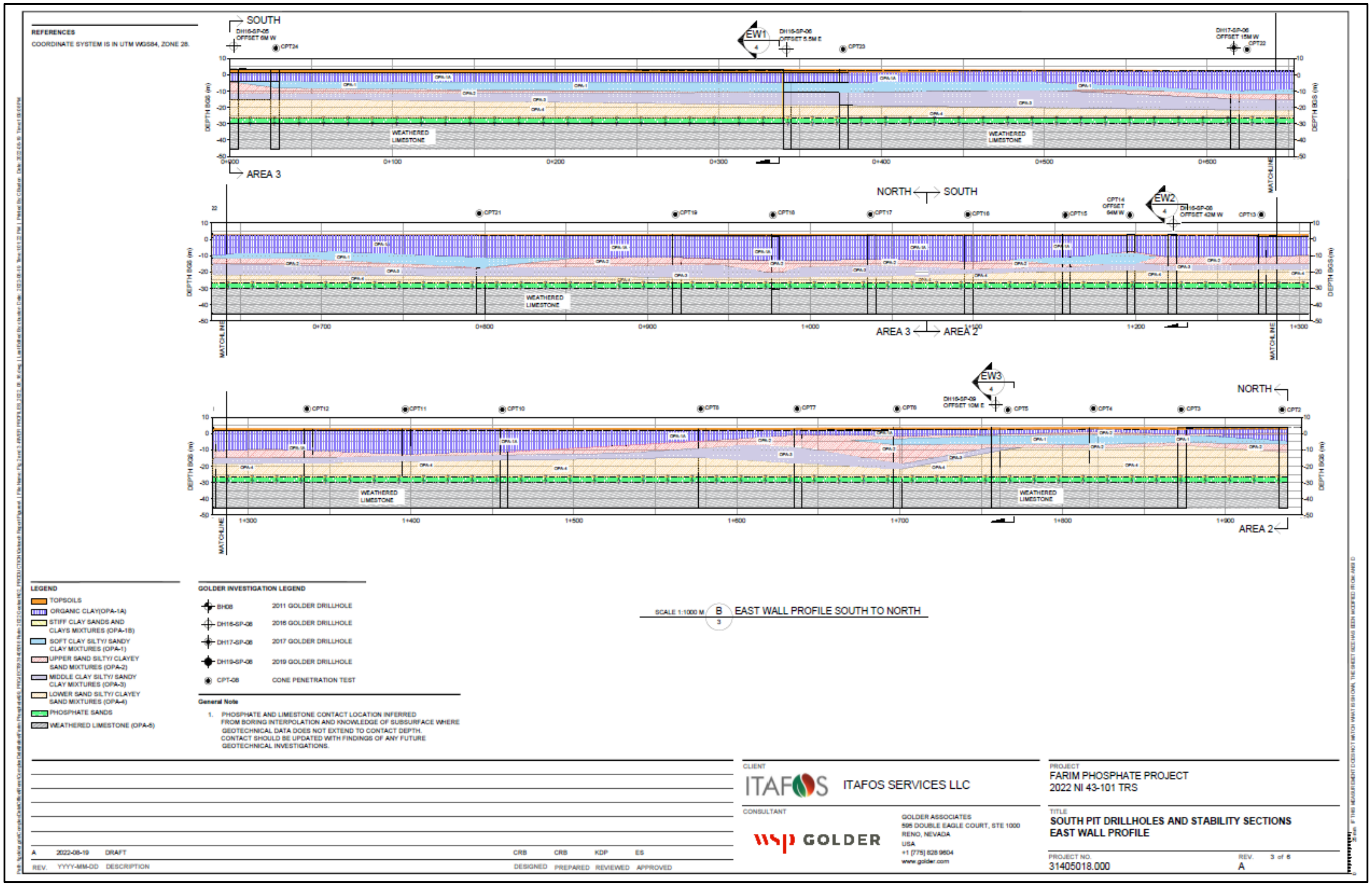
Source: WSP Golder, 2022

Figure 16-8: South Pit Drillholes and Stability Sections South Wall Profile



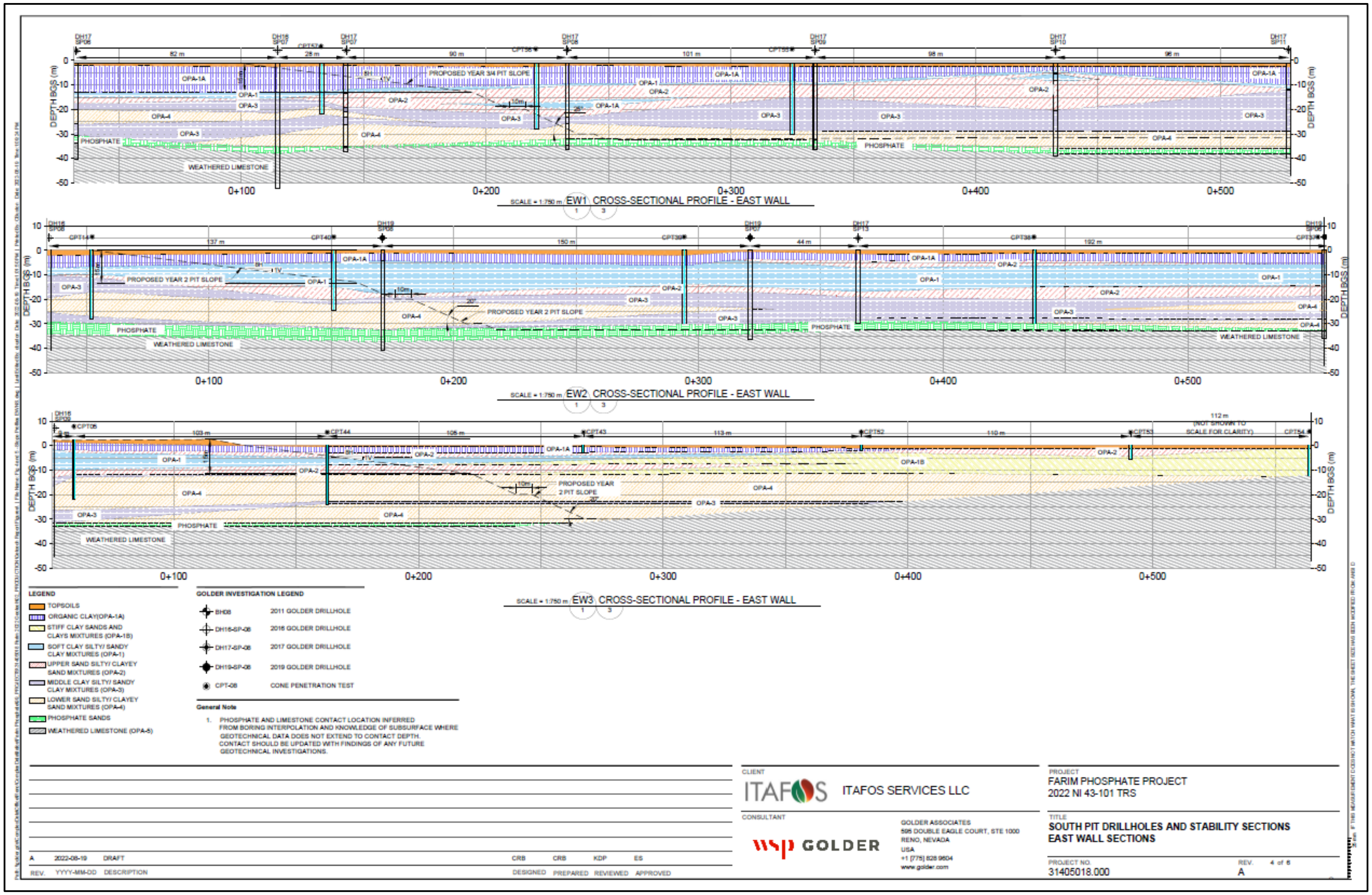
Source: WSP Golder, 2022

Figure 16-9: South Pit Drillholes and Stability Sections East Wall Profile



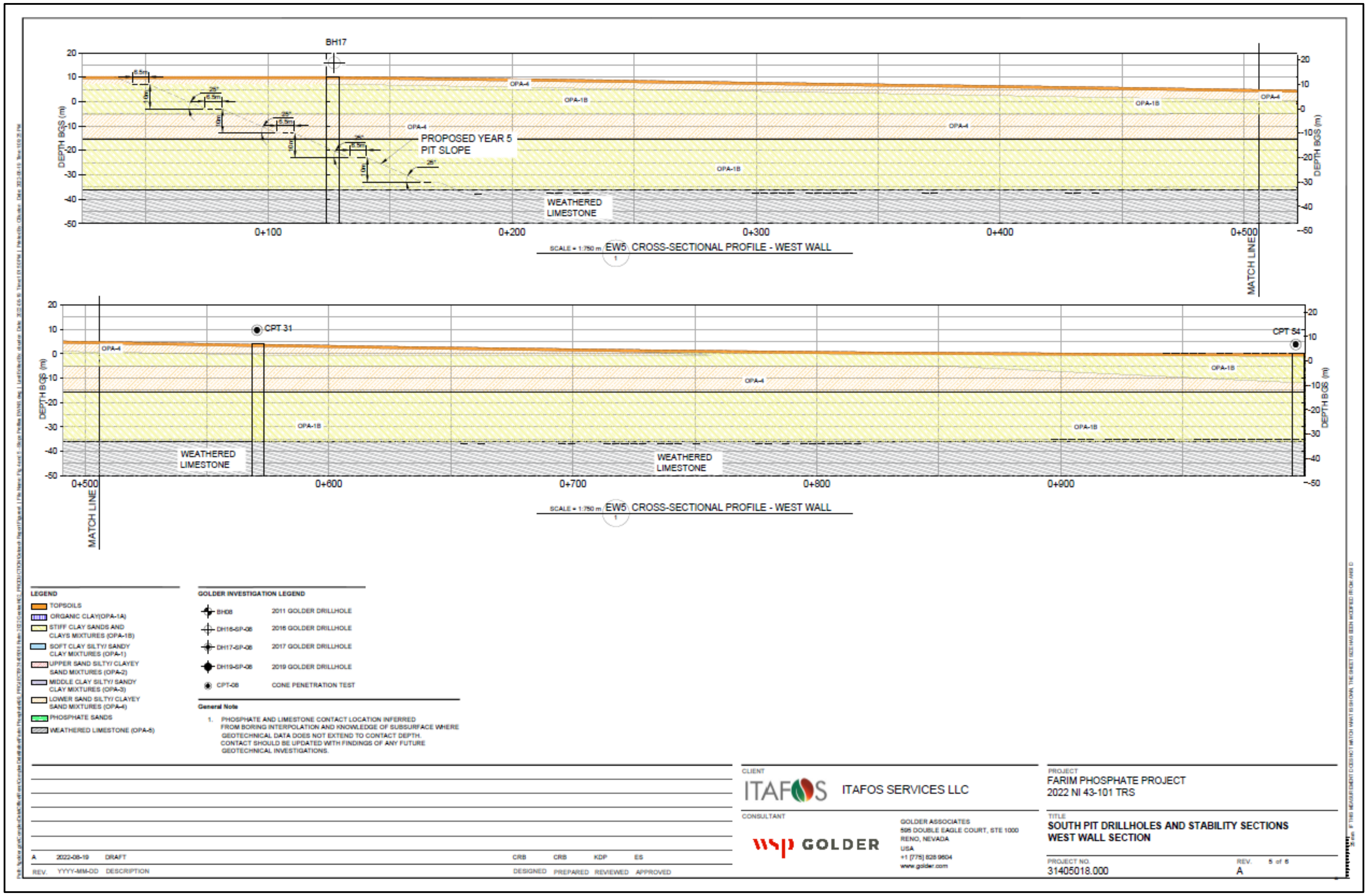
Source: WSP Golder, 2022

Figure 16-10: South Pit Drillholes and Stability Sections East Wall Sections



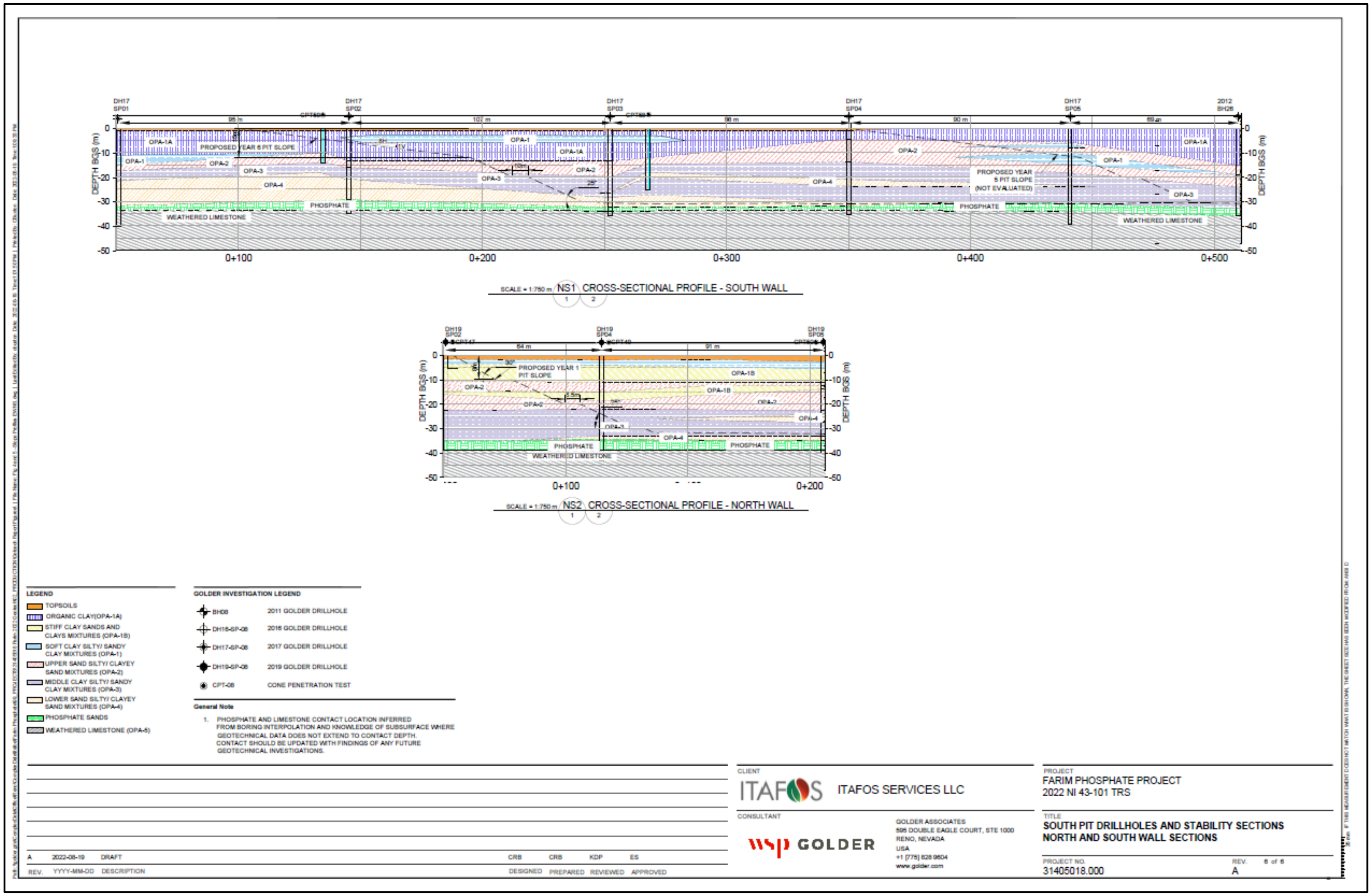
Source: WSP Golder, 2022

Figure 16-11: South Pit Drillholes and Stability Sections West Wall Section



Source: WSP Golder, 2022

Figure 16-12: South Pit Drillholes and Stability Sections North and South Wall Sections



Source: WSP Golder, 2022

16.7.2.3 Pit Excavation Dewatering Plan

Hydrogeological characterization studies have been completed in support of the 2012 and 2015 feasibility reports as well as various subsequent studies. The most recent report (KP, 2020) provides a feasibility-level dewatering strategy along with associated dewatering costs for mining of the South pit (i.e., the first seven years of mining). The objective of the dewatering plan is to reduce the phreatic surface to below the working elevation of the operational areas of the South pit. The most recent hydrogeologic model applied for the South pit included the upper very soft (Lodo) clay, the underlying overburden aquifer, and the bedrock below the base of the deposit.

The most recent proposed dewatering strategy includes vertical dewatering wells around the perimeter of the pit crest and around the interim yearly pit shells proposed by KP in 2017. These wells are proposed to be screened through the overburden aquifer unit and approximately 9 m into the bedrock. The dewatering wells around the perimeter of the South pit would have a spacing of 50 m along the eastern and southern sectors of the pit that bound the River Cacheu. The perimeter well spacing is increased to 100 m along the western and northern perimeters of the South pit away from the river. Dewatering wells around the perimeters of the interim yearly phases would be spaced 100 m apart. The dewatering would begin six months before mining of each interim pit begins. Vertical pumping wells are anticipated to be effective in dewatering more permeable, sandy units for the planned distances between wells and the well points planned on active benches. However, there may be less permeable silt and clay zones or clay-encapsulated sand zones that cannot be effectively dewatered with the well spacing used.

Shallow dewatering wells are planned within each interim pit and are planned to target draining shallow sand units in the overburden that may perch on low permeability subunits. Ten to 20 shallow dewatering wells are proposed in each annual pit area. Additional shallow, 6 m deep well points spaced 1 to 3 m apart are proposed at the toe of the slopes on the active mining benches in areas where groundwater requires localized management.

Monitoring wells are also planned to measure the performance of the dewatering wells. In-pit sumps are planned to capture surface water runoff and water pumped from the shallow well points.

Expected impact on stability and limitations of this currently proposed dewatering plan include the following:

- Slope dewatering is expected to significantly increase slope stability along the pit slopes adjacent to the river.
- The effectiveness of dewatering is currently unknown without a desktop and field dewatering study. The rate of pore water reduction for many of the finer units (clays) is unknown to the large lateral variability of the sandy and silty/clayey lenses. Additionally, the finer lenses have the potential to act as aquitards and prevent effective drainage of the sandier, more permeable layers.
- It will be important to monitor the pore pressure development in the slopes during excavation and construction for slope performance management

16.7.2.4 Regional Seismicity

Golder completed site-specific probabilistic and deterministic seismic hazard assessments in West Africa over the last seven years (i.e., Guinea, Ivory Coast, and Ghana). Table 16-4 lists the studies considered for this assessment for the Farim open pit site.

Table 16-4: Site-Specific PSHA Studies Prepared by Golder in West Africa Since 2015

Year	Country	Site	Site Ground Conditions Shear wave velocity (V_{S30}, ASCE 7)1	Ground Motion Models (GMMs) Used
2015	Guinea	Simandou Port Site	V_{S30} = 800 m/s	Abrahamson, Silva and Kamai (ASK 2014)
				Boore, Stewart, Seyhan and Atkinson (BSSA 2014)
				Campbell and Bozorgnia (CB 2014)
				Chiou and Youngs (CY 2014)
				Pezeshk et al. (2011)
		Simandou Mine Site		Atkinson and Boore (2006) Eastern North America with magnitude dependent stress drop
2016	Côte d'Ivoire	Bonikro Mine	V_{S30} = 760 m/s	Atkinson and Boore (2006, 2011)
				Atkinson (2008) modified with Atkinson and Boore (2011)
				Pezeshk et al. (2011)
				Silva et al. (2002)
				Campbell (2003)
2020	Ghana	Damang Mine	V_{S30} = 760 m/s	Suite of seventeen (17) GMMs developed for the Next Generation Attenuation-East (NGA-East) project for eastern and central North America (Goulet et al. 2018).
2021	Ghana	Ahafo North	V_{S30} = 493 m/s (Average value of V_{S30} determined during MASW survey at single facility applied to all sites)	
		Ahafo South		

Notes: Site ground conditions estimated. General soil/rock conditions anticipated. Average vs. data obtained from MASW survey at one of the project sites and applied to all sites.

Golder developed the PGA estimates for the Farim site by taking the results for five sites with ground conditions at V_{S30} = 800 or 760 m/s. Golder excluded the values for the Afhao north and south sites because they were calculated for a V_{S30} = 493 m/s for an outcropping stiff soil site.

Golder compared the average mean PGA value to that developed for West Africa by the Global Earthquake Model (GEM) (Western Africa (WAF) | GEM Global Mosaic of Hazard Models (openquake.org)). The GEM model has 1/475 annual exceedance probability (AEP) values that are up to two to three times less than the values developed in the Golder's site. Golder also downloaded the West Africa GEM OpenQuake source files to calculate mean PGA at a 1/2,475 and 1/10,000 AEPs. These values are similarly two to three times less than the values recommended in this assessment. The review of the GEM model suggests that the relatively low GEM PGAs for the range of AEPs is likely caused by the following:

- earthquake recurrence rates based only on gridded-area historical earthquake epicenters and magnitude assessments, rather than the uniform-area sources used in the Golder-WSP site-specific studies
- a zero-activity background earthquake rate outside of the regions of historical earthquake activity rather than a low background rate used in the site-specific studies
- an assumed b-value of 1.04 rather than 0.8 used in the site-specific studies
- application of older ground motion models (GMM)
- very limited incorporation of uncertainties.

The limited level of analysis developed by GEM for West Africa provides only minimum PGA estimates that are insufficient for the seismic analysis of a high failure consequence facility. In the QP's opinion, the mean PGA evaluated by averaging the PGA estimates from the five site-specific studies provide reasonable estimates of the PGA expected at the Farim site at the AEPs indicated. Table 16-5 lists the recommended PGAs assuming a V_{S30} = 760 m/s (a very stiff soil/weak outcropping rock site ground condition). The highlighted average mean PGA value is used for this simplified analysis for the Guinea-Bissau Farim site.

Table 16-5: Mean PGA Values Selected AEPs for Site-Specific Studies by Golder in West Africa

Location	1/475 AEP (g)	1/975 ½ (g)	1/2,475 AEP (g)	1/5,000 AEP (g)	1/10,000 AEP (g)
Simandou Port	0.051	0.081	0.140	0.206	0.288
Simandou Mine	0.044	0.073	0.135	0.209	0.310
Bonikro TSF	0.023	0.037	0.065	0.100	0.152
Damang TSF	0.017	0.033	0.064	0.000	0.150
Average (PGA)	0.034	0.045	0.080	0.130	0.180

Note: PGA estimates are not based on a site-specific analysis of earthquake sources and site ground conditions. These results should not be used for development of any further studies outside of this evaluation and are not intended to the take the place of a site-specific seismic hazard assessment. Any site-specific analysis should focus on the incorporation of the significant epistemic uncertainties in this region of historically low earthquake occurrence.

16.7.3 Slope Stability Analysis

Slope stability analyses were performed to evaluate the stability of the pit slopes in the Years 1 through 7 pits for the South pit using available current mine plan pit geometry, geotechnical, and piezometric data from previous design and stability reports (Golder, 2020b) and refined findings from the 2019 field and laboratory investigations (Golder, 2020a). Additionally, the pit slope design was optimized using the findings from these analyses to advise construction of adjacent infrastructure construction such as the waste rock dump and plant. This section summarizes the assumptions, analysis methods, geotechnical material properties, and results of these stability analyses.

Golder used a 2D limit-equilibrium (LEM) stability model implemented in the software program SLIDE2 (Version 9.002 2020 by RocScience) to evaluate the stability of the slopes. This software program allows for both circular and non-circular potential sliding surfaces to be either pre-selected or automatically generated. Each potential sliding surface is analyzed, and the potential sliding surface with the lowest factor-of-safety (FOS) against failure is considered the critical sliding surface. SLIDE allows the use of multiple analysis methods which vary depending on the assumptions used for equilibrium in the model. Golder selected Morgenstern-Price's Method of Slices as appropriate to analyze the sliding surfaces as this method satisfies conditions of static horizontal and vertical force equilibrium, as well as moment equilibrium.

For each stability section described below, Golder considered the following circular and non-circular sliding geometries:

- Circular Failures – Sliding surfaces that enter at the crest of the slope and pass through midslope.
- Non-Circular Failure – An auto-refined sliding surface search routine and Cuckoo-search that applies a sophisticated algorithm search for the most critical sliding surfaces in layered materials.
- Factors of Safety (FOS) – Must be commensurate with the level of understanding of the site conditions and geotechnical parameters. Given the newly obtained data from the 2019 investigation (Golder, 2020a) and based on the current state of practices for stability review of temporary pit slopes, the minimum required static FOS was selected as 1.1 for temporary pit slopes and 1.3 for pit slopes with adjacent infrastructure which warrant a higher failure consequence. The minimum required pseudo static (seismic) FOS was selected as 1.0 for all slopes.

16.7.3.1 Material Properties

The geotechnical units used in the slope stability modelling have been described in Section 16.7.2.1. The selected engineering properties for native materials were developed and discussed in detail in the previous stability evaluation (Golder, 2020b). With no new geotechnical information available to date, these properties were considered appropriate and valid for use in this study. Material properties are summarized in Table 16-6, and account for the following property changes or additions to select units:

- OPA-1A: The engineering properties developed through the current study for the various lithologic units are comparable to the values that have been applied in the previous stability assessments. The estimated strength of 9 kPa for organic clay (OPA-1A) unit was applied in the 2020 evaluation (Golder 2020b) throughout the thickness of the OPA-1A that is based on the vane shear tests and CPT data in the thin, shallow layers of the OPA-1A in the Year 1 to 2 areas. To be more representative and better fit the data for deeper and thicker organic clays interpreted over the pit area in the Year 3 to 7 areas, a vertical stress ration (Su/σ'_v) was also applied in addition to the 9 kPa strength value.
- Bund Fill: A bund fill is also included in the development of the slope stability models which represents imported material for haul road base and for river bund material. This material is assumed to be from the waste stripping, so generic properties have been assumed for this material. The material properties developed by KP for the bund fill were determined to be appropriate for the generalized expected material that will be used to construct the bund.
- Pseudo-static Undrained Strengths: These stability analyses considered a seismic loading scenario. It is typical for undrained materials to experience a strength reduction in cyclic loading as experienced in an earthquake. Thus, a conservative reduction of 20% strength loss is applied to all undrained materials in pseudo-static loading conditions.

Table 16-6: Interpreted Engineering Material Properties of Lithologic Units

Material	Modelled Strength Condition	Unit Weight	Static Loading – Intact Strength			Pseudostatic Loading – Undrained Materials Experience 80% Strength		
		(kN/m^3)	c' (kPa)	φ' (°)	Su (kPa)	c' (kPa)	φ' (°)	Su (kPa)
Random Fill	Drained	20	0	30	-	0	30	-
Bund Fill	Drained	20	0	25	-	0	25	-
Organic Clay (OPA-1A)	Undrained	15.3	-		9 + 0.10*σ'_v	-		7.2 +0.08*σ'_v
Stiff Clay (OPA-1B)	Undrained	19	-		160 + 0.42*σ'_v			128 +0.34*σ'_v
Soft Clay (OPA-1)	Undrained	17.5	-		30			24
Upper Sand (OPA-2)	Drained	17.6	10	30	-	10	30	-
Middle Clay (OPA-3)	Undrained	19	-		80 + 0.42*σ'_v			64 + 0.34*σ'_v
Lower Sand (OPA-4)	Drained	20	0	32	-	0	32	-
Phosphate Sands	Drained	19	0	37	-	0	37	-
Limestone (OPA-5)	Drained	20	70	20	-	70	20	-

16.7.3.2 Applied Groundwater Conditions

The stability models assume that slopes will be dewatered prior to mining and dewatering will continue throughout the operation of the exposed slopes. Dewatering is expected to be effective only in the sand units (OPA-2 and OPA-4) and bedrock. The fine-grained units (OPA-1, OPA-1A, OPA-1B, OPA-3) will not be significantly dewatered.

These stability models therefore assume a water table at the ground surface. The sand units are modelled as drained, while the fine-grained units are modelled as undrained for the base scenarios. Undrained conditions are assumed to control the behavior of the fine-grained units as the mine plan calls for backfilling of interim pit areas as the pit is developed further. Golder has assumed that cut slopes will not be exposed for long enough (a few months at the most) for drained conditions to develop in the fine-grained units, or for strength loss to occur due to exposure, opening and softening of fissures, or degradation of the organic materials. The criticality of these assumptions for drainage behavior and water table drawdown is explored in the Sensitivity Analyses in Section 16.7.3.5.1.

16.7.3.3 Slope Geometry and Analysis Sections

Slope stability of South pit walls was evaluated for representative sections of the South pit. The stratigraphy is based on the units identified in the 2012 to 2017 drilling investigations and refined with the insights gained from the 2019 CPT and drilling program. Slopes with similar geology are grouped by domain Areas 1 to 4 as presented in Figure 16-6. Profiles along the river were developed to evaluate lateral variability of the soils and are presented in Figure 16-7 and Figure 16-8.

Six geotechnical profiles were prepared to evaluate the north, east, west, and south slopes (Figure 16-9 through Figure 16-11). The stability models evaluated the current mine plan slopes to determine whether adequate FOS are indicated and if there are opportunities to steepen slopes. The following sections describe the representative sections analyzed in the stability models. The results of the stability analyses are provided in Section 16.7.3.5.

The current mine plan shown in the SLIDE2 models applies the geometric criterion by domain area as described in the subsections below.

16.7.3.3.1 Area 1 – Northeast Slope (Year 1)

The slope immediately adjacent to the process plant area is composed of 10 m high benches, 25° final bench face angle, and 6.5 m safety bench width, and the uppermost 15 m is steepened to 30°. An 85 m wide haul road corridor runs along the pit crest in this area based on the current mine plan. The process plant is adjacent to the slope in this area and slope effects and recommended offsets to the infrastructure were evaluated.

Section NS2 was developed to represent the profile running north-south along a borehole and CPT line to represent a typical section through the northern slope of the pit, below the plant site area, that will be excavated in Year 1 and is representative of geotechnical Area 1. The low strength clay units (OPA-1 and OPA-1A) were not present in this portion of the South pit. The uppermost bench of this section is evaluated at 57° and 30° angles in the stiff clays. The subsurface profile for the upper 36 m profile for this section was developed using three boreholes and five CPT logs.

16.7.3.3.2 Area 2 – Northeast Slopes (Years 1 to 2)

This area is subject to 7.1° inter-ramp slopes in the upper 15 m for the slopes along the east and south perimeter of the South pit where the pit slopes are adjacent to the River Cacheu. Permanent batter (bench face) angle will be 20°, 10 m tall, with a 10 m wide safety bench in the lower half of the slope. Above the pit crest, a 55 m wide haul road runs along the pit crest in this area. The bund is located adjacent to the haul road between the road and the river. The river is approximately 110 m from the pit crest.

Section EW2 was developed to represent the profile running east-west along an alignment of boreholes and CPT soundings to represent a typical section through the eastern slope that will be excavated in Year 2. The subsurface profile for the upper 36 m profile for this section was developed using five boreholes and six CPT logs. The thickness of the OPA-1A unit in this section is approximately 6.5 m thick and is underlain by approximately 11.5 m of unit OPA-1 for a combined maximum thickness of about 18 m.

Section EW3 was developed to represent the profile running northeast-southwest along a borehole and CPT line to represent a typical section through the northeastern corner of the pit that will be excavated in Years 1 and 2. The subsurface profile for the upper 36 m profile for this section was developed using one borehole and six CPT logs. The thickness of the OPA-1A unit in Section EW3 is approximately 4.5 m thick and is underlain by approximately 6 m of unit OPA-1, with a layer of sand (OPA-2) about 1.5 m thick.

16.7.3.3.3 Area 3 – Southeast and South Slopes

This area is subject to 7.1° inter-ramp slopes in the upper 15 m for the slopes along the east and south perimeter of the South pit where the pit slopes are adjacent to the River Cacheu. The permanent batter (bench face) angle will be 25°, 10 m tall, with a 10 m wide safety bench in the lower half of the slope.

Section EW1 was developed to represent the profile running east-west along an alignment of boreholes and CPT soundings to represent a typical section through the eastern slope that will be excavated in Years 3 and 4. The subsurface profile for the upper 36 m profile for this section was developed using seven boreholes and four CPT logs. The thickness of the OPA-1A unit in this section is approximately 13 m thick and is underlain by approximately 1 m of unit OPA-1 for a combined maximum thickness of about 14 m.

Section NS1 was developed to represent the profile running north along a borehole and CPT line to represent a typical section through the south slope of the pit that will be excavated in Years 6. The subsurface profile for the upper 36 m profile for this section was developed using six boreholes and three CPT logs. The thickness of the OPA-1A unit in Section EW3 is approximately 15 m thick and is underlain by approximately 3 m of unit OPA-1, with a layer of sand (OPA-2) about 1.5 m thick.

16.7.3.3.4 Area 4 – West and Northwest Slopes (Year 3 to 7)

The west and northwest slopes were evaluated with overall 20° inter-ramp slope benched with 10 m high benches, 25° final bench face angles, and 6.5 m safety bench width. An 85 m wide haul road runs along the pit crest in this area. Additionally, the waste rock dump is adjacent to the slope in this area and slope effects and recommended offsets to the infrastructure were evaluated.

Section EW5 was developed to represent the profile running east-west through the western slope proposed and includes loading from a 30 to 40 m thick waste rock dump to represent a typical section through western slopes in Years 3 through 7 and is representative of geotechnical Area 4. The subsurface profile for the upper 36 m profile for this section was developed using one borehole and six CPT logs. The thickness of the OPA-1A unit in Section EW3 is approximately 4.5 m thick and is underlain by approximately 6 m of unit OPA-1, with a layer of sand (OPA-2) about 1.5 m thick.

16.7.3.4 Seismic Loading

The regional seismicity assessment discussed in Section 16.7.2.4 included recommended PGAs for varying intensity return interval earthquakes. For these temporary open slopes, the 1-in-975-year return earthquake event was selected as the design criteria (PGA of 0.045 g) and used in these slope stability analyses. Four of the most critical stability sections, namely NS1, NS2, EW2, and EW3 were selected for pseudo-static evaluations based on calculated static FOS values.

For pseudo-static slope stability analyses that model earthquake loading, the calculated pseudo-static coefficient is usually lower than the peak acceleration to account for alternating inertia effects with the slide mass. A seismic coefficient, k, of 0.030 g (two-thirds of the PGA) was utilized for pseudo-static slope stability analysis to model earthquake loading on the slopes. This reduction in the PGA is in line with the commonly accepted state-of-practice by Hynes-Griffin and Franklin (1984). In addition, undrained materials are assumed to experience cyclic shear strength reduction due to increased pores pressures during the shaking event and thus have applied 80% strengths of the static shear strengths.

16.7.3.5 Slope Stability Results

The results of the updated slope stability analyses are presented for each section in Table 16-7.

The analysis results indicate FOS equal and/or greater than the design criteria for both the static and pseudo-static design criteria.

Additionally, it was found that the adjacent infrastructure to the pit is subject to the following proximity restrictions to maintain acceptable criteria levels of stability:

- The process plant shall be no less than 40 m from the crest of the pit on the north wall. The upper slope of the pit in this area shall be set back and no steeper than a 30° angle.
- Per Table 16.7, a minimum offset of 85 m from WD-1 to South pit Area 4 is required to maintain an appropriate factor of safety. If the dump height is outside the range evaluated here or the dump height is required to be 30 or 40 m with a lesser standoff, further stability studies should be performed to determine an appropriate offset from the pit.

Table 16-7: Slope Stability Analyses Results

Section-Area	Pit Slope Excavation Year	Sliding Surface Geometry	Static FOS			Pseudo-static FOS (k=0.03g)
			Upper Slope	Lower Slope	Overall Slope	Overall Slope
		Minimum Required FOS	1.1	1.1	1.3	1.0
NS1 – Area 3	6	Circular	1.7	1.4	-	1.1
		Non-circular	1.6	1.2	-	1.0
NS2 – Area 1	1	Circular	1.4	-	1.4	1.2
		Non-circular	1.3	-	1.3	1.1
EW1 – Area 3	4	Circular	1.4	1.4	-	-
		Non-circular	1.3	1.3	-	-
EW2 – Area 2	2	Circular	1.9	1.3	-	1.1
		Non-circular	1.8	1.1	-	1.0
EW3 – Area 2	2	Circular	2.7	1.2	-	1.1
		Non-circular	2.4	1.2	-	1.0
EW5 – Area 4	5	Circular[1]	-	-	1.4	-
		Non-circular [1]	-	-	1.3	-
		Circular[2]	-	-	1.4	-
		Non-circular [2]	-	-	1.3	-

Notes: 1. Waste rock dump WD1 is evaluated assuming dump is 40 m tall and offset from the crest 120 m. 2. Waste rock dump WD-1 is evaluated assuming dump is 30 m tall and offset from the crest 85 m.

16.7.3.5.1 Sensitivity Analysis

16.7.3.5.1.1 Variable Water Table and Rapid Drawdown

There is inherent uncertainty in saturation levels within slopes due to lack of active monitoring instrumentation and variability of sediments. A variable water table or rapid drawdown due to dewatering conditions can lead to slope instability. This effect was evaluated by modelling a drawdown of water table away from slope face down to top of phosphate sands. Sandy units (OPA-2, -4, and phosphate sands) were evaluated with drained Mohr-Coulomb strength properties. Saturation levels in the clayey units are not expected to be much affected by the drawdown of the water table, and thus a conservative excess pore pressure (Ru = 0.3) is applied in undrained clay units (OPA-1, -1A, and -3).

The results of the water table sensitivity analyses are presented for three critical sections in Table 16-8.

When compared to the base scenario FOS results presented in Table 16-7, the results of sensitivity analyses indicate that dewatering and drawdown of the water table away from the slope face could have a significant impact on increasing slope stability. Dewatering of the slopes will be critical to pit slope performance.

Table 16-8: Sensitivity Analysis Results – Water Table Drawdown

Section-Area	Pit Slope Excavation Year	Sliding Surface Geometry	Static FOS	
			Upper Slope	Lower Slope
NS1 – Area 3	6	Circular	1.4	2.4
		Non-circular	1.3	2.3
EW2 – Area 2	2	Circular	1.8	2.6
		Non-circular	1.8	2.3
EW3 – Area 2	2	Circular	2.9	2.0
		Non-circular	2.5	1.9

16.7.3.5.1.2 Drainage Conditions

There is limited laboratory testing data available to adequately determine the drainage behavior of the in-situ soils; therefore, the effects of assuming drained versus undrained behaviors of the materials were evaluated. Sandy units (OPA-2, -4, and phosphate sands) appear to have a sufficiently high fines content that they may behave in an undrained manner during excavation. Therefore, stability of three critical sections were evaluated applying undrained conservative constant strength values based on CPT and SPT data in these units: namely, a cohesive strength of 180, 200, and 250 kPa for OPA-2, -4, and phosphate sands, respectively.

The results of the undrained sensitivity analyses are presented for three critical sections in Table 16-9.

Table 16-9: Sensitivity Analysis Results – Sand Drainage Behavior

Section-Area	Pit Slope Excavation Year	Sliding Surface Geometry	Static FOS	
			Upper Slope	Lower Slope
NS1 – Area 3	6	Circular	1.7	2.5
		Non-circular	1.6	2.5
EW2 – Area 2	2	Circular	1.9	2.4
		Non-circular	1.8	2.4
EW3 – Area 2	2	Circular	4.1	2.4
		Non-circular	2.7	2.4

When compared the base scenario FOS results presented in Table 16-7, the results of sensitivity analyses indicate that undrained behaviors in the clayey sands could serve to increase the strength of the slopes. Therefore, assuming drained conditions in the sandy units as applied in the slope stability analyses is considered a more conservative approach and appropriate for this study.

16.7.3.5.1.3 Soft and Organic Clays Thickness Variability

There is uncertainty in the maximum thickness of the soft and organic clay layers (OPA-1 and -1A) due to the variability and limited drilling data along the river in Areas 2 and 3. The effects of poor strength material geometric variability is evaluated by conservatively increasing the estimated OPA-1A clay layer thickness and the weaker lower sandy layers (OPA-2 and -4) in three critical sections.

The results of the sensitivity analyses are presented for three critical sections in Table 16-10.

Table 16-10: Sensitivity Analysis Results – Soft and Organic Clay Thickness

Section-Area	Pit Slope Excavation Year	Sliding Surface Geometry	Static FOS	
			Upper Slope	Lower Slope
NS1 – Area 3	6	Circular	1.7	1.0
		Non-circular	1.5	0.9
EW2 – Area 2	2	Circular	1.9	1.2
		Non-circular	1.8	1.1
EW3 – Area 2	2	Circular	2.2	0.9
		Non-circular	1.9	0.9

When compared the base scenario FOS results presented in Table 16-7, the results of sensitivity analyses indicate that the thickness of the soft and organic clays could have a significant impact on the stability of the slopes. Assuming possible thicker OPA-1, -1A, -2, or -4 units are encountered in the field in Area 2 and 3 than what is currently estimated in this study, it is possible to have unstable slopes (FOS < 1.1). It will be critical to monitor the thickness and condition of these units as construction begins, and should significant adverse conditions be encountered, the slope in the area should be re-evaluated to account for the conditions prior to further excavations.

16.7.3.5.1.4 Effectiveness of Dewatering

Dewatering the slopes along the river in Areas 2 and 3 and will be required to maintain a dry pit below the shallow groundwater table. If sufficiently dewatered, this can serve to stabilize the slopes. Effectiveness of dewatering the slopes is evaluated by decreasing the pore pressures in the sandy units (i.e., assuming 10% of the pore pressure in the sands due to the water table is removed due to dewatering efforts).

The results of the dewatering sensitivity analyses are presented for three critical sections in Table 16-11.

Table 16-11: Sensitivity Analysis Results – Dewatering Effectiveness (Hu)

Section-Area	Pit Slope Excavation Year	Sliding Surface Geometry	Static FOS	
			Upper Slope	Lower Slope
NS1 – Area 3	6	Circular	1.7	1.5
		Non-circular	1.6	1.3
EW2 – Area 2	2	Circular	1.8	1.4
		Non-circular	1.8	1.3
EW3 – Area 2	2	Circular	2.7	1.3
		Non-circular	2.7	1.3

When compared the base scenario FOS results presented in Table 16-7, the results of sensitivity analyses indicate that even minor dewatering (Hu reduced be 10%) of the sandy units in the Areas 2 and 3 slopes can lead to significant improvement on slope stability. Continuous dewatering will be critical to slope performance in the areas along the river.

16.7.3.6 Liquefaction Stability Risk Assessment

Evaluation of the liquefaction potential for the materials identified in past investigations in the Years 3 through 7 pit areas was conducted using the same approach as performed for Year 1 and 2 pit areas in the previous Golder study (Golder, 2020b). Namely, the inherent ability for each material to liquefy (known as the material's liquefaction susceptibility) was first evaluated in Years 3 to 7 pit areas based on the index properties (plasticity and water content) of the laboratory samples taken from the borings in the south and west wall areas. Next, for any soils determined to have a low to very high susceptibility to liquefaction, Golder evaluated the potential of the material to liquefy in the design earthquake event (i.e., the material's liquefaction potential) using the available CPT data in the area. The design earthquake event is discussed and developed in Section 16.7.2.4 and selected as the 1-in-975-year earthquake event (PGA = 0.045 g).

16.7.3.6.1 Liquefaction Susceptibility (Screening)

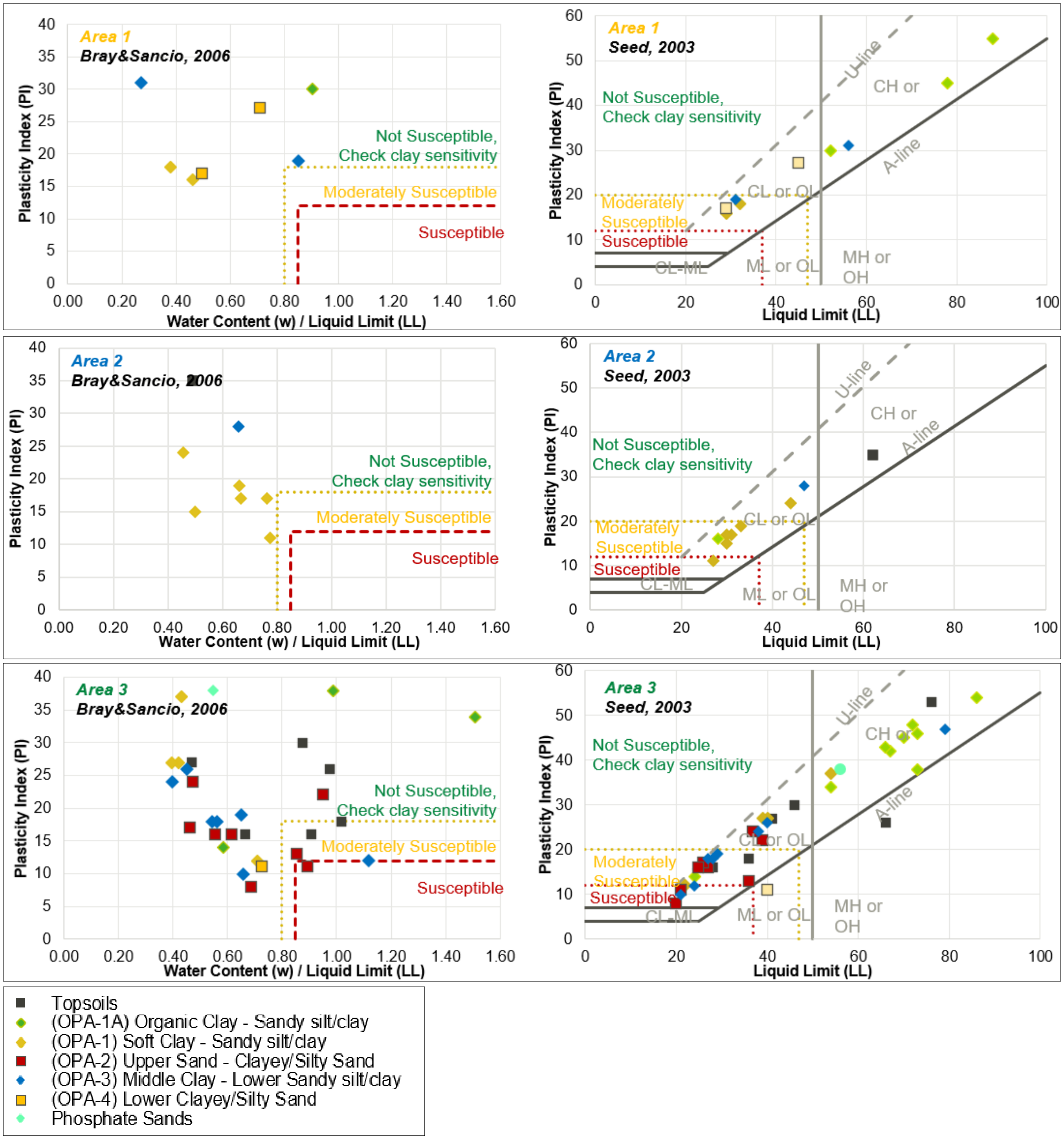
The available laboratory index data from the 2019 investigation (Golder, 2020a) was applied to two index property screening approaches in Golder's previous liquefaction screening assessment (Golder, 2020b) using screening tools by Seed et al. (2003) and Bray and Sancio (2006). Materials from Area 1 through 3 have been previously collected and tested for the index properties required for liquefaction screening (plasticity and water content). It should be noted that no samples have been tested for index properties to date from Area 4 (west and northwest pit walls). Samples in Area 4 should be collected and screened prior to excavation to evaluate soil's liquefaction susceptibility.

The imported fill material for the haul roads and the bund fill materials are assumed to be well-graded with a significant gravel content. Additionally, it is assumed that these materials will be compacted to a high degree. A large gravel content and dense state will preclude liquefaction potential and thus these materials are assumed to be non-liquefiable. However, the selected materials should be screened for liquefaction potential and these assumptions re-visited once laboratory samples can be taken from the imported materials.

The screening assessment of all native material tested specimens is shown in Figure 16-13 by area and material unit. A summary of the liquefaction susceptibility assessment by area is as follows:

- Area 1 – Materials encountered and collected in this area included topsoils, OPA-1, -1A, -3, and -4. The encountered materials are screened to have negligible to moderate liquefaction susceptibility per Seed (2003) method but are insufficiently saturated to liquify. Due to the consistent high plasticity of these materials, it is unlikely that these materials will saturate or become less plastic over time. It is therefore considered that Area 1 has a negligible liquefaction susceptibility and does not require further liquefaction potential evaluation.
- Area 2 – Materials encountered and collected in this area included topsoils, OPA-1, -1A, and -4. Topsoils, OPA-1A, and OPA-3 are screened to have negligible liquefaction susceptibility per all methods. OPA-1 is found to range from susceptible to moderately susceptible per Seed's method but is insufficiently saturated per Bray and Sancio's method to be determined to be not susceptible. Area 2 is considered to present a negligible liquefaction susceptibility and does not require further liquefaction potential evaluation.

Figure 16-13: Liquefaction Screening Assessment of Areas 1 to 3 (No Samples Collected and Tested in Area 4)



Source: Bray & Sancio, 2006

- Area 3 – Materials encountered and collected in this area included topsoils, OPA-1, -1A, -2, -3, and-4 and phosphate sands. OPA-1A, OPA-3, and phosphate are screened to have negligible liquefaction susceptibility per all methods. OPA-1, -2, -4 and topsoils were found to range from susceptible to negligibly susceptible per both methods. Topsoils are sufficiently shallow and are not determined to pose a stability risk to the pit and therefore discounted from this stability analysis. OPA-2 and -4 are sufficiently sandy to have the potential to saturate in adverse conditions. Therefore, Area 3 could potentially present liquefaction susceptibility in the sandy units OPA-2 and -4 and which then required a further liquefaction potential evaluation where these units are encountered.
- Area 4 – No samples have been collected and tested in this area; however, past investigations indicate that native materials in these areas are largely included topsoils, OPA-1B (stiff clays), and OPA-4. OPA-1B is sufficiently fine and plastic to present a negligible liquefaction susceptibility. However, there is insufficient data to determine susceptibility of the OPA-4 unit in this area. Therefore, Area 4 is considered to present liquefaction susceptibility in the sandy unit OPA-4 and thus requires further liquefaction potential evaluation where these units are encountered.

16.7.3.6.2 Liquefaction Potential in Design Earthquake

As previously discussed in the liquefaction screening assessment, the sandy units OPA-2 and -4 may be considered susceptible to liquefaction in Areas 3 and, due to lack of data, Area 4. Golder therefore evaluated the liquefaction potential of these units in Areas 3 and 4 to evaluate whether they would liquefy in the design earthquake (975-year return earthquake, PGA = 0.045 g).

An FOS against liquefaction, $FS_{liq,}$ was calculated for the design event based on the 2019 CPT data. FS_{liq} takes into account both the soils state and the earthquake loading by calculating the FS as the ratio of the soil's critical resistance ratio (CRR) to the critical stress ratio (CSR) of the earthquake. To simplify this concept, the CRR can be thought of as the soil's inherent ability to resist strength reduction to dynamic loading and the CSR can be thought of as the dynamic stresses applied by the earthquake. The CSR was calculated for 16 samples of OPA-2 and 11 samples of OPA-4 using Seed's simplified method (Seed and Idriss, 1971). For each sample, the soil's CRR was calculated based on adjacent paired CPT soundings at the sample depth following Boulanger and Idriss (2011) method. This method utilizes the normalized clean sand tip resistance, q_{c1Ncs}, to calculate the CRR, which is constituted by empirical correlations of the tip resistance and soil behavior type index. The FS_{liq} for OPA-2 and -4 in Area 3 are reported in Table 16-12.

All samples for the OPA-2 units were found to have factors for safety greater than 1.6, and all OPA-4 samples were found to have a FS greater than 3.0. This indicates that there is negligible potential for liquefaction for the design earthquake considered in this study. This conclusion agrees with the previous findings from 2019, which indicated the sand units (OPA-2 and OPA-4) have a low potential for liquefaction due to the high fines content, dense to very dense state of the units, and dilative responses during strength testing. Therefore, liquefaction does not appear to pose a risk to slope stability in Area 3.

However, it should be noted that no samples have been collected and tested to date in Area 4. Area 4 should be sampled and tested to allow for screening and liquefaction potential evaluation prior to excavation in this area.

Table 16-12: Factor of Safety Against Liquefaction – Area 3

Material Type	Critical Stress Ratio (CSR) at Sample Depth [Seed's Simplified Method, 1971]				Critical Resistance Ratio (CRR) from CPT Soundings [Boulanger and Idriss 2011]		FS Against Liquefaction (CRR/CSR)
	Sample Hole	Sample Depth (m)	USCS Soil Type	CSR	Paired CPT No.	CRR	
OPA-2 (not susceptible to susceptible)	DH17-SP01	21	SC	0.03	CPT27	0.09	2.6
	DH17-SP02	12.25	SC	0.05	CPT59	0.08	1.7
	DH17-SP03	15	SM	0.04	CPT58	0.09	2.0
	DH17-SP03	19.2	SC	0.04	CPT58	0.09	2.5
	DH17-SP04	8.4	SC	0.05	CPT58	0.09	1.6
	DH17-SP04	10	SC	0.05	CPT58	0.09	1.7
	DH17-SP04	23.2	SC	0.03	CPT58	0.14	4.4
	DH17-SP05	13.9	SC	0.05	CPT58	0.20	4.4
	DH17-SP06	19.15	SC	0.04	CPT22	0.33	8.8
	DH17-SP08	9.45	SC	0.05	CPT56	0.15	2.9
	DH17-SP08	18.45	SC	0.04	CPT56	0.13	3.3
	DH17-SP09	9	SM	0.05	CPT55	0.14	2.6
	DH17-SP10	9.7	SM	0.05	CPT55	0.14	2.7
	DH17-SP10	14.5	SC	0.04	CPT55	0.09	1.9
	DH17-SP11	12.15	SC	0.05	CPT55	0.15	3.2
	DH17-SP11	19.55	SC	0.04	CPT55	0.15	4.2
OPA-4 (not susceptible to susceptible)	DH17-SP01	27.3	SC	0.03	CPT27	0.10	3.3
	DH17-SP01	27.9	SW	0.03	CPT27	0.10	3.3
	DH17-SP01	31.75	SW	0.03	CPT27	0.10	3.4
	DH17-SP02	22.45	SM	0.03	CPT59	0.08	2.5
	DH17-SP03	25.5	SM	0.03	CPT58	0.14	4.5
	DH17-SP07	21.6	SM	0.03	CPT57	0.14	4.1
	DH17-SP07	25.8	SC	0.03	CPT57	0.14	4.6
	DH17-SP07	29.85	SM	0.03	CPT57	0.14	4.8
	DH17-SP11	30.05	SM	0.03	CPT55	0.12	4.1
	DH17-SP08	28.95	SC	0.03	CPT56	0.11	3.7
	DH17-SP09	30	SC	0.03	CPT55	0.12	4.1

16.7.4 Pit Slope Design Recommendations

This section provides slope design recommendations for bench, inter-ramp, and overall slope design. These recommendations were followed in the final design.

16.7.4.1 Summary of Pit Slope Recommendations

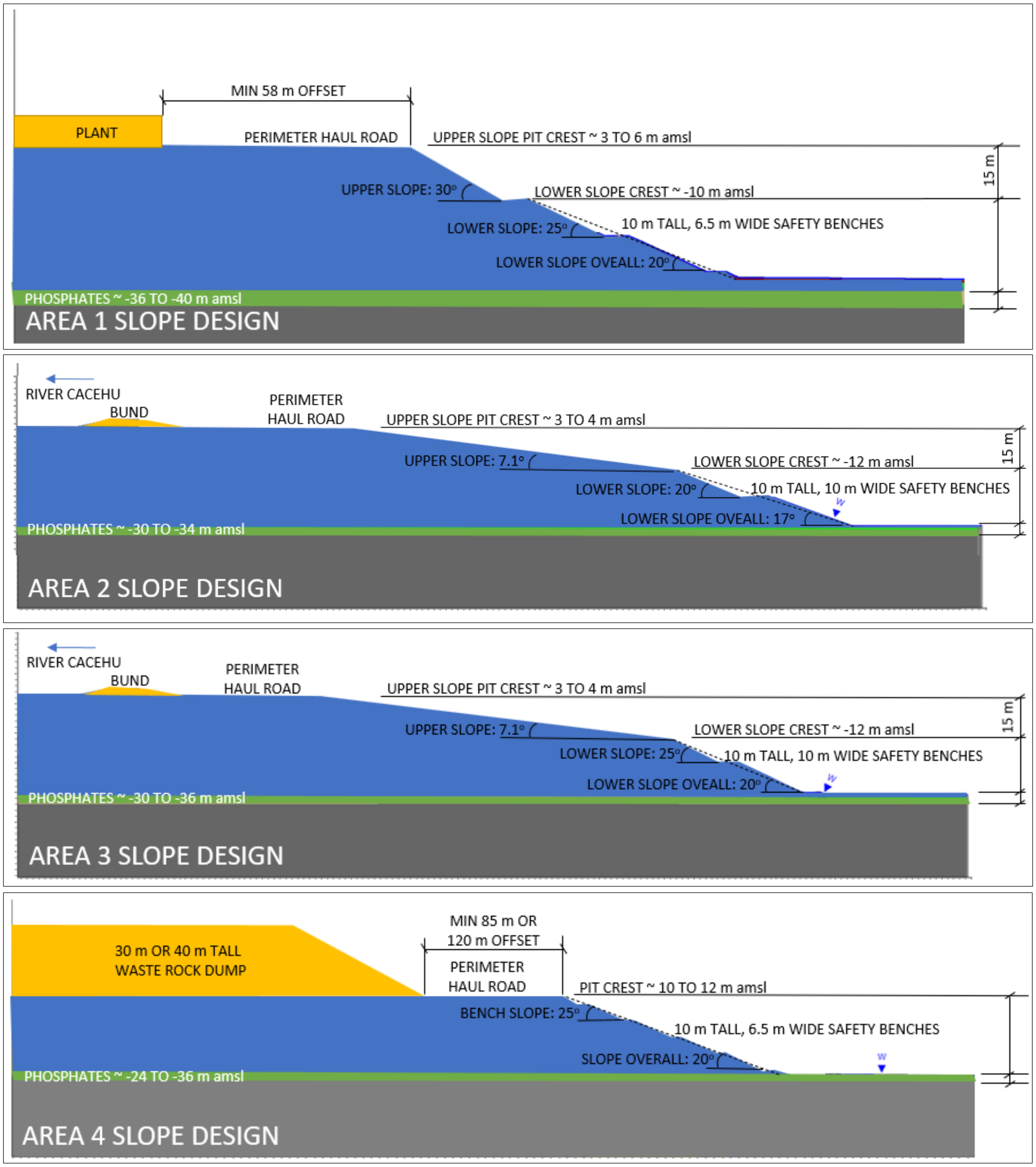
Based on the results of these analyses, the previous design schedule from the 2019 stability evaluation is still being considered, with the following exceptions:

- The lateral variability of the soft and organic clays in the Area 2 slopes were found to be significant. Therefore, it is recommended to decrease the lower slope bench angles from 25° (previous design) to 20° and this recommendation was followed in the final pit design.
- The slope adjacent to the power plant (Area 1) was previously designed to have a steep upper slope from 0 to 15 meters bgs. This over-steepened slope (previously 57°) was found to be sensitive to earthquake loading, and the areas along the crest that has impacted stability levels is large. To improve performance of the upper slope in static and dynamic conditions and to reduce the area impacted by the slope, it is recommended to set back this upper slope angle to 30° in the upper 15 m.

Pit slope design was optimized in the stability modelling and acceptable slope angles and bench configurations were selected by area. A summary of Golder's pit slope geometric design recommendations is presented in Figure 16-14. Operational and construction recommendations are presented in Table 16-13.

Golder recommends that the catch bench width and bench face angle be adjusted based on the conditions exposed at the pit face during mining.

Figure 16-14: Pit Slope Design Recommendations



Source: WSP Golder, 2023

Table 16-13: Pit Slope Design Recommendations for Farim South Pit

Description	South and East Walls of the South Pit	
	Area 2	Area 3
Area Description	Northeast slope adjacent to river	Southeast and south slopes adjacent to river
Design Pit Year	Year 2	Year 3 to Year 7
Dewatering Schedules	Continuous dewatering of upper and lower sands (target maximum Hu of 0.9)	Continuous dewatering of upper and lower sands (target maximum Hu of 0.9)
Description	**Other Slopes of the South Pit**	
	Area 1	**Area 4**
Area Description	North Slope adjacent to plant	West and northwest slopes and adjacent to waste rock dump
Design Pit Year	Year 1 and Year 2	Year 3 to Year 7
Zone of Pit's Influence on Adjacent Infrastructure[1]	40 m	230 m to 40 m high dump 190 m to a 30 m high dump
Minimum Setback Distance from Pit Crest to the Infrastructure[2]	40 m to the plant	120 m to 40 m high dump 85 m to a 30 m high dump
Dewatering Schedules	Continuous dewatering of upper sands (target maximum Hu of 0.9)	Continuous monitoring of upper and lower sands piezometric pressures with dewatering as needed

Notes: The zone of influence shows the area that includes slip surfaces that are with calculated FOS less than 1.5 which includes the loading from the waste dump. The offset distance was selected by moving the loading from the infrastructure back until critical calculated FOS of 1.3 for overall static, and FOS of 1.0 for pseudo-static conditions could be achieved.

16.7.4.2 Pit Water Management

An important component of the slope development will be to monitor the degree of pore pressure reduction that has been achieved in the bench face that is being excavated. This can be achieved by installing piezometers or pushed probes with pressure transducers into critical areas along the pit slopes. Supplemental pumping wells or horizontal drains will be needed where isolated pressurized zones are encountered. Further studies should be done to advise precise locations of these piezometers for optimized performance.

Diversion of the natural drainages, direct precipitation onto the pit slopes, and the runoff from the crest of the pit will be required to address the runoff that may flow into the pit from an area above the crest of the pit. To collect direct precipitation and surface water runoff within the pit area, the water would be collected at the toe of each bench and diverted to in-pit collection channels located along the pit access ramp and the intermediate sump stations. These channels, as well as the intermediate sump stations, will need to be relocated and developed over time as the pit expands and deepens.

16.7.4.3 Slope Displacement Monitoring

Slope displacement monitoring through the installation of inclinometers to detect the slope displacement at depth as well as surface displacements through prism or radar or lidar systems will be important as the slope development advances closer to the River Cacheu. A slope monitoring and action plan should be prepared that defines the details of the pore pressure and slope displacement monitoring activities and defines alert levels in terms of pore pressure or displacement readings and actions that should be taken when alert levels are reached. Alert levels will range from normal operating to low and high levels and will have progressively higher levels of notification and increased frequency of monitoring and restrictions on activities.

16.7.4.4 Geotechnical and Geological Mapping

The distribution of the geotechnical units is highly uncertain and exerts significant controls on the slope design. Therefore, as mining progresses, the exposed pit slopes are recommended to be mapped for the geotechnical and geological units. The results of this mapping should then be used to update the geotechnical model and cross-sections. The results should also be reviewed against design assumptions and be used to adjust the pit slope design, where necessary.

16.7.5 Geotechnical Risks and Opportunities

16.7.5.1 Risks

For this study, geotechnical characterization is based on subsurface data from relatively large-spaced boreholes and CPT soundings concentrated mainly along the east and south walls of South pit. Variability in the subsurface conditions could result in geotechnical conditions different than estimated for this study, which could lead to instabilities. As the slopes of clayey units with estimated weakest shear strengths are exposed, they may begin to deteriorate and may undergo strength loss due to exposure, opening and softening of fissures, or degradation of the organic materials. Slopes should be backfilled in a timely manner and not left open for prolonged periods. Ongoing monitoring of slope conditions should be part of mining best practices to maintain a safe work environment.

There is no site-specific seismic hazard assessment study for the project. For this study, Golder completed site-specific probabilistic and deterministic seismic hazard assessments in West Africa over the last seven years (i.e., Guinea, Ivory Coast, and Ghana). A site-specific seismic hazard assessment should be considered for a better level of analysis. Any site-specific analysis should focus on the incorporation of the significant epistemic uncertainties in this region of historically low earthquake occurrence.

Mining facilities, such as the waste dump and process plant, are designed near the South pit crest in the current mine plan. Golder evaluated the effect of loading from adjacent facilities in slope stability analysis and recommended minimum setback distances from these facilities to the pit crest based on the slope stability analysis results. The analysis results indicated that a minimum setback distance of 120 m for a 40 m high dump, and 85 m for a 30 m high dump is needed to meet acceptable FOS=1.3 based on the assumed unit weight of 20 kN/m^3 for the waste dump material. If dump heights more than 40 m is required, a more sophisticated analysis might be needed. The safe setback distance to the plant should be at least 40 m from the pit crest to avoid any shear surfaces with calculated FOS up to 1.5 based on the stability analysis results. In other words, the minimum 40 m offset places the plant at a point with a lower risk of being impacted by slope instability.

16.7.5.2 Opportunities

Steeper inter-ramp design slope angles in Years 4 to 7 at the south and west walls of South pit may be feasible if geological conditions are more favorable than the assumptions documented in this report. Additional geotechnical data and lessons learned from mining in Years 1 and 2 can also help with this decision.

Golder does not recommend assuming steeper design slope angles without field verification of geotechnical conditions and slope performance during mining.

16.8 Mining Plan Sequence

16.8.1 Pit Progression

The mine plan production scenario was targeted to produce approximately 2.19 Mt/a of ROM phosphate matrix on an as-received basis (at approximately 20% moisture) or 1.75 Mt/a ROM phosphate matrix on a dry basis. The mine production schedule was developed to achieve these targets and to optimize the plan to defer mining as much of the potential acid-generating (PAG) material as possible and the areas adjacent to the River Cacheu until sufficient neutralizing material could be stripped to mitigate the PAG overburden. Approximately 883,000 bcm of PAG material was stripped in Year 0 which cannot be stored in pit until Year 1. As such, the PAG material was temporarily stockpiled in a designated portion of WD-1 lined with a 500 mm thick basal liner.

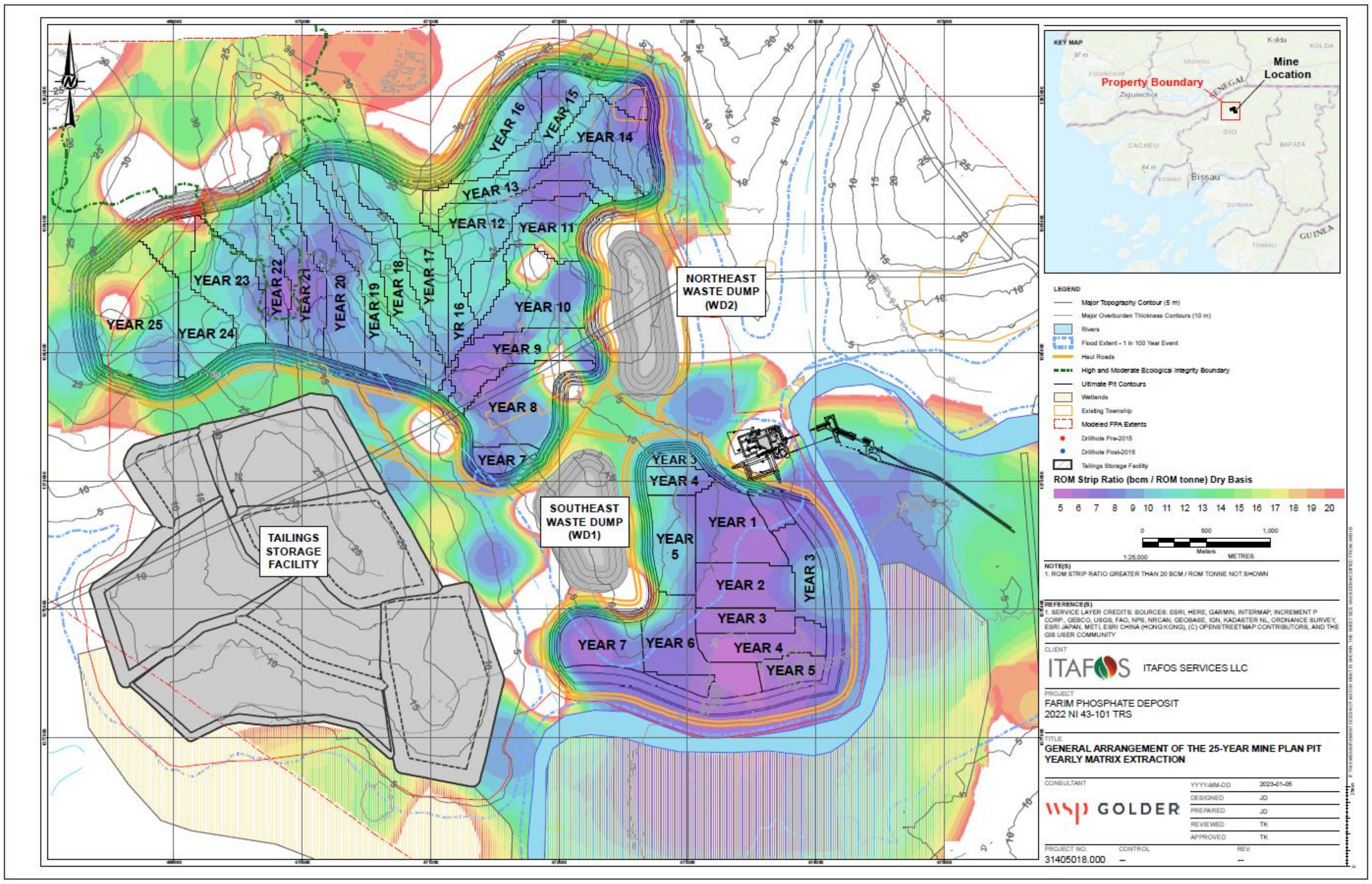
The mine sequence includes six months of pre-stripping in Year 0 to allow for immediate matrix production in Year 1. The North pit is not a preferred starting area due to metallurgical uncertainty and therefore mining commences in the South pit on the north edge adjacent to the plant and continues down the central portion of the pit, avoiding both the east and west highwall for as long as possible. This alternative was selected as preferrable to starting along the eastern highwall as it allows for additional time to construct the bund between the pit and the River Cacheu. This alternative was also preferrable to starting along the Western highwall which has some of the highest stripping ratio ore and has some potentially acid generating material overlying the ore. The Phase 1 Pit avoids most of the PAG material as it advances southward from the North highwall. Mining continues to advance south until Years 2 and 3 when the pit extends to the eastern highwall as the bund construction advances and also opens up in the North lobe of the South pit where the majority of the PAG material is located. This Phase 1 has 1.93 Mt of ROM FPA, or approximately one year and one month of production at 1.75 Mt/a at a ROM strip ratio of 7.0 bcm of waste per ROM FPA tonne (bcm/ROM t). Mining continues advancing to the south and west until the South pit is mined out in Year 7.

After this first pushback is completed midway through Production Year 1 (PY1), opening in-pit backfill opportunities becomes the priority in an effort to reduce truck haulage requirements and permanently store PAG material in pit where it can be covered by sufficient neutral material. This necessitates that the area with PAG concerns be stripped and mine earlier in the sequence than would be preferred. While mining this area could be deferred to advance more quickly into the lower-strip "heart" of the South pit near the river, the appeal of the lower-strip ratio area is lessened by the reduced backfill opportunities and Golder's recommendation of a maximum 8H:1V overall interim slope when advancing the pit to within 500 m of the final pit extents adjacent to the river. The QP, therefore, opted to schedule the area of PAG concurrently with another 100 m pushback toward the river to make sure there is sufficient neutralizing material to blend with the PAG overburden while also reducing strip ratio. These two cuts are scheduled for completion early in PY3.

In-pit backfilling can begin more aggressively in PY3 after completion of the two aforementioned pushbacks. Another 100 m pushback is then made toward the river within 300 m of the final extents of the South pit. After completion of this 100 m pushback at the end of Year 3, a slot cut is made perpendicular to the river to the final extents of the South pit to limit the length of pit wall opened up adjacent to the river at any time. Successive cuts are then made along the river until the South pit is mined out in Year 7.

During the final year of mining in the South pit, the North pit is pre-stripped to allow for immediate ore production. The North pit 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. The 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 17, 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.

Figure 16-15: General Arrangement of the 25-Year Mine Plan – Yearly Matrix Extraction



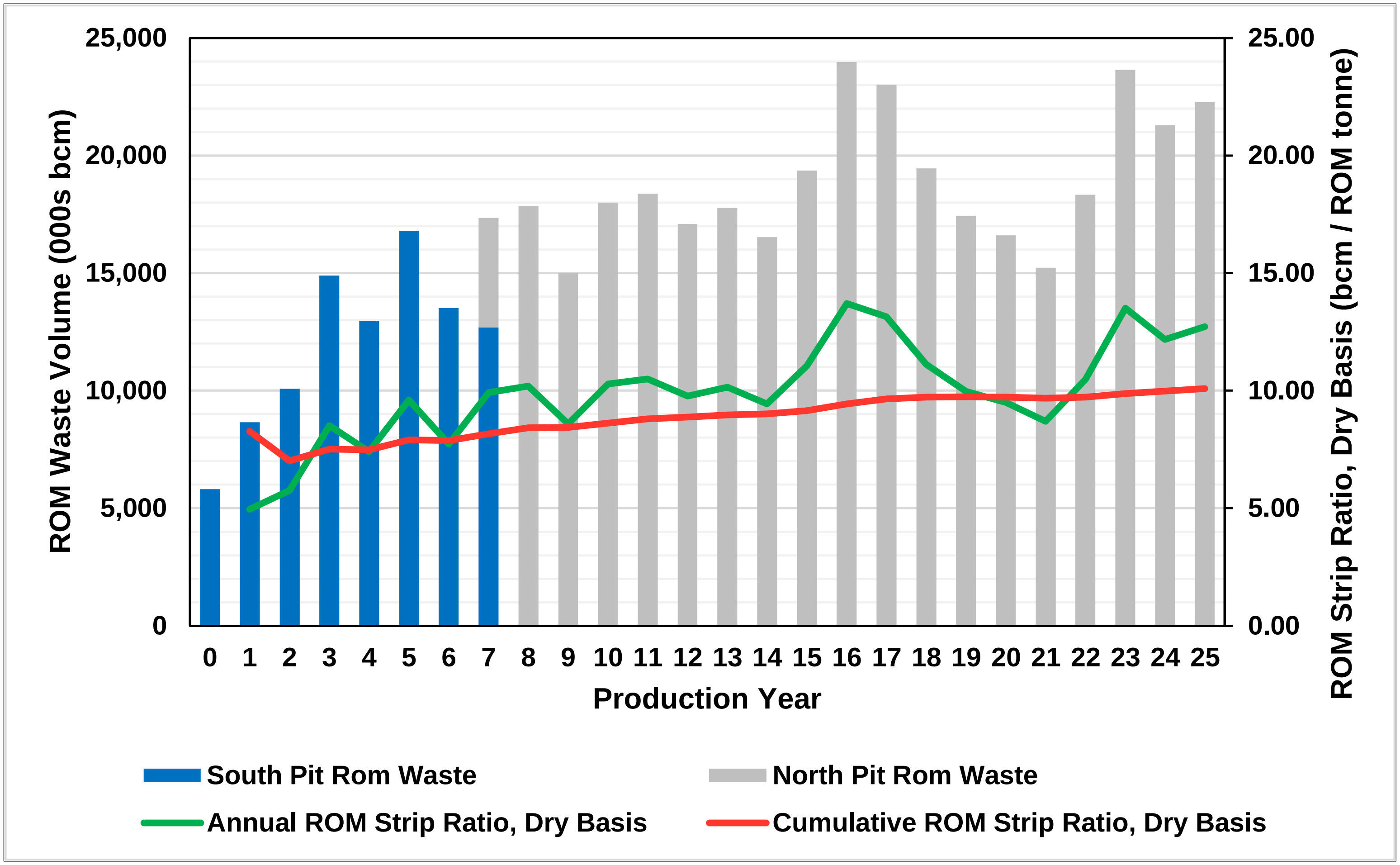
Source: WSP Golder, 2023

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.

The mine plan meets 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 16,000 t over the life of mine. As seen in Figure 16-16, the yearly strip ratio remains under 10 bcm / ROM tonne as mining progresses through the South pit and then increases in Year 7 as mining transitions to the higher strip ratio North pit. A comparison of the yearly ROM (plant feed) grades is provided in Table 16-14.

End-of-period maps showing the mine progression, access, haul road progression, and facilities annually for Years 0, 1, 3, 5, 10, 20, and 25 have been provided as Figure 16-16 through Figure 16-23.

Figure 16-16: Annual Waste Stripping Requirements

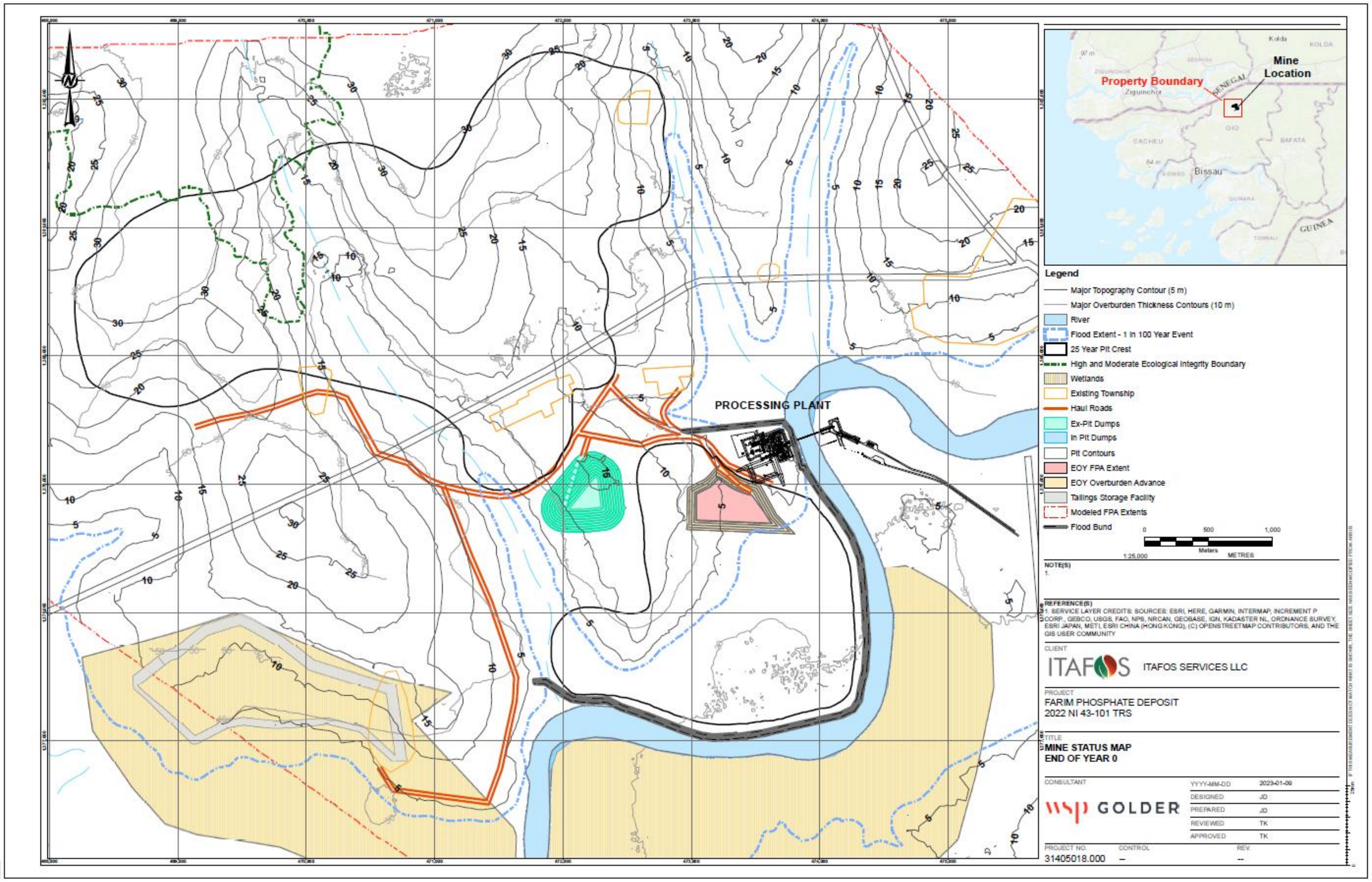


Source: WSP Golder, 2023

Table 16-14: Annual ROM Qualities

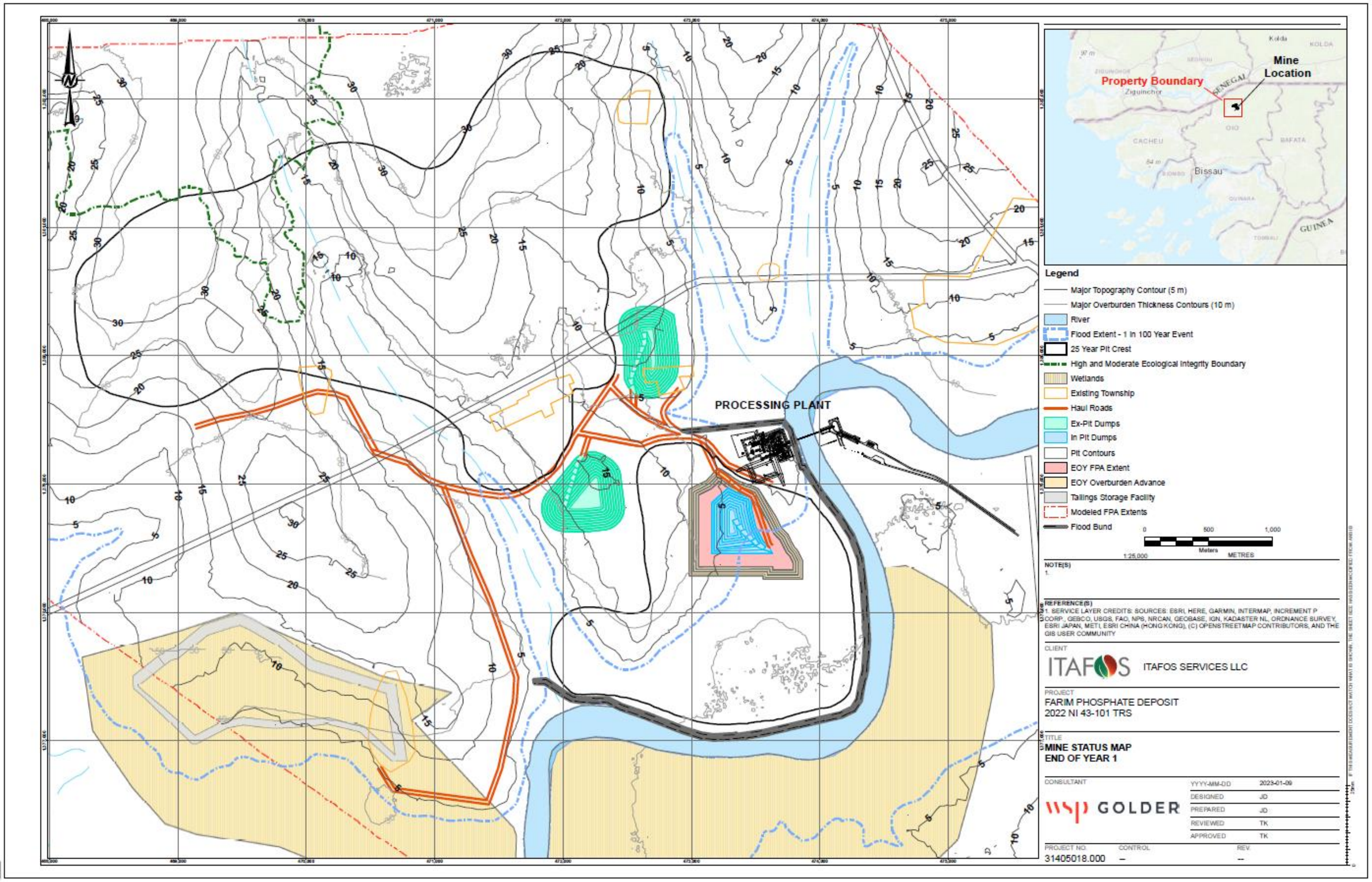
Parameter	Unit	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
Overburden Removal	kbcm	5,812	8,661	10,081	14,892	12,969	16,800	13,510	17,348	17,847	15,017	18,005	18,375	17,083	17,768	16,525	19,356	23,981	23,006	19,457	17,438	16,609	15,228	18,335	23,654	21,300	22,470
ROM Ore (Dry Basis)	kt	0	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	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Total Annual ROM Strip Ratio	bcm/t	NA	4.95	5.76	8.51	7.41	9.60	7.72	9.91	10.20	8.58	10.29	10.49	9.76	10.15	9.44	11.06	13.70	13.15	11.12	9.96	9.49	8.70	10.48	13.52	12.17	12.72
Total Cumulative ROM Strip Ratio	bcm/ t	NA	8.27	7.02	7.51	7.49	7.91	7.88	8.17	8.42	8.44	8.63	8.80	8.88	8.97	9.01	9.14	9.43	9.65	9.73	9.74	9.73	9.68	9.72	9.88	9.98	10.09
ROM P_2O_5, Dry Basis	%	NA	31.11	30.97	30.86	30.99	30.21	30.87	29.17	28.49	28.34	28.19	30.33	30.24	30.57	31.41	30.59	30.47	29.27	28.81	28.68	29.58	30.78	30.95	29.52	29.41	29.66
ROM Al_2O_3, Dry Basis	%	NA	2.59	2.19	2.18	2.21	2.20	2.10	2.66	2.69	2.50	2.68	2.09	1.77	1.61	1.53	1.97	2.53	3.06	3.15	3.36	3.20	3.17	3.34	3.43	3.27	3.67
ROM CaO, Dry Basis	%	NA	40.80	41.61	41.78	40.12	39.45	39.98	39.12	38.84	40.36	39.94	42.63	43.46	43.33	42.20	41.40	42.14	41.68	40.79	39.34	40.03	41.36	41.63	40.48	40.71	41.12
ROM Fe_2O_3, Dry Basis	%	NA	3.86	3.32	3.34	3.97	5.15	3.34	5.16	7.88	5.72	4.91	3.87	3.64	3.97	5.60	5.99	4.63	4.67	5.26	5.55	5.61	5.01	4.40	4.66	5.04	4.99
ROM SiO_2, Dry Basis	%	NA	10.92	11.65	11.14	11.21	11.08	11.33	11.65	10.98	11.73	13.10	10.30	8.92	8.72	8.87	8.42	8.61	9.17	9.50	11.80	11.53	10.14	10.14	12.85	11.73	9.84

Figure 16-17: Mine Status Map — End-of-Year 0



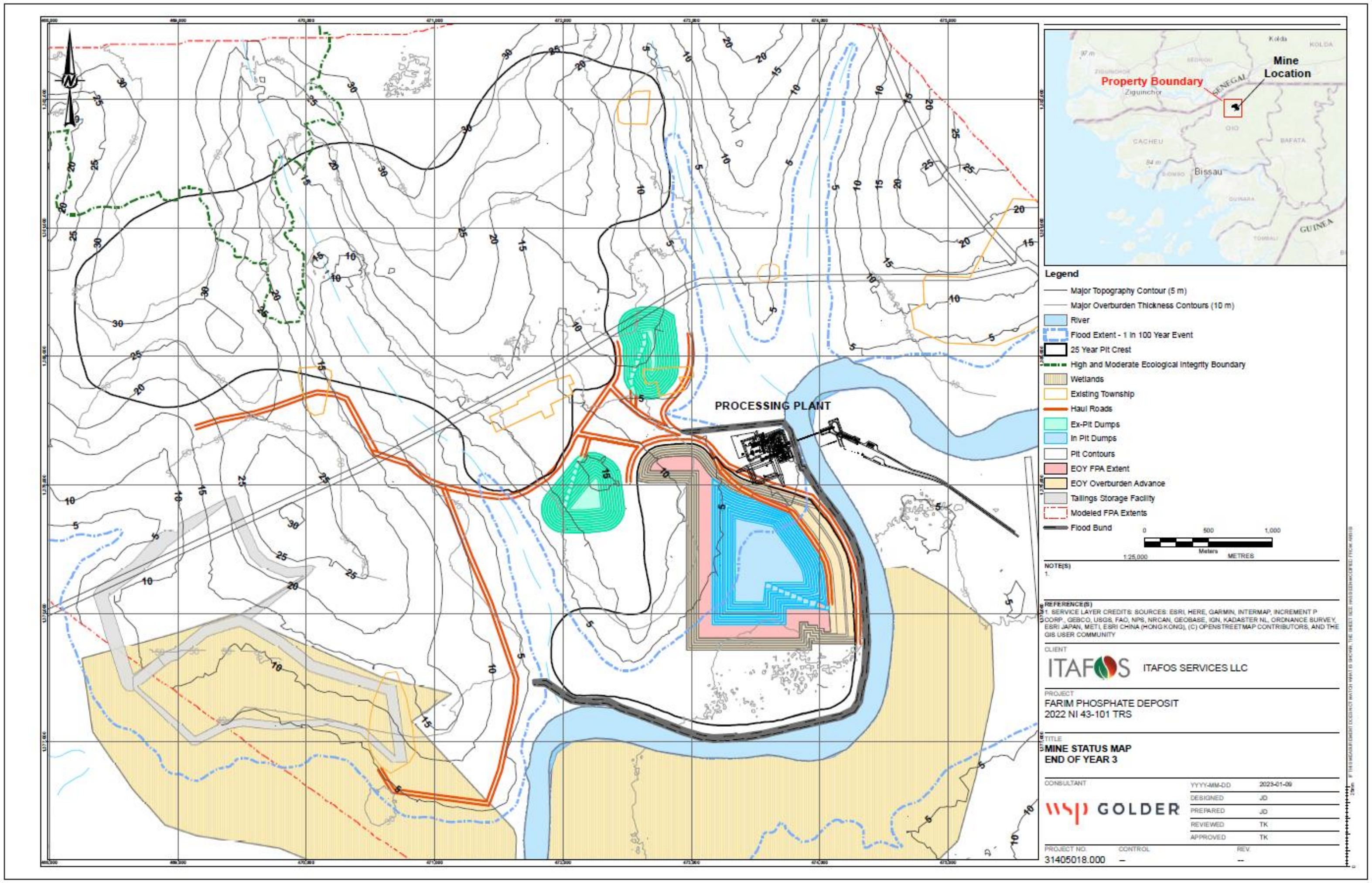
Source: WSP Golder, 2023

Figure 16-18: Mine Status Map — End-of-Year 1



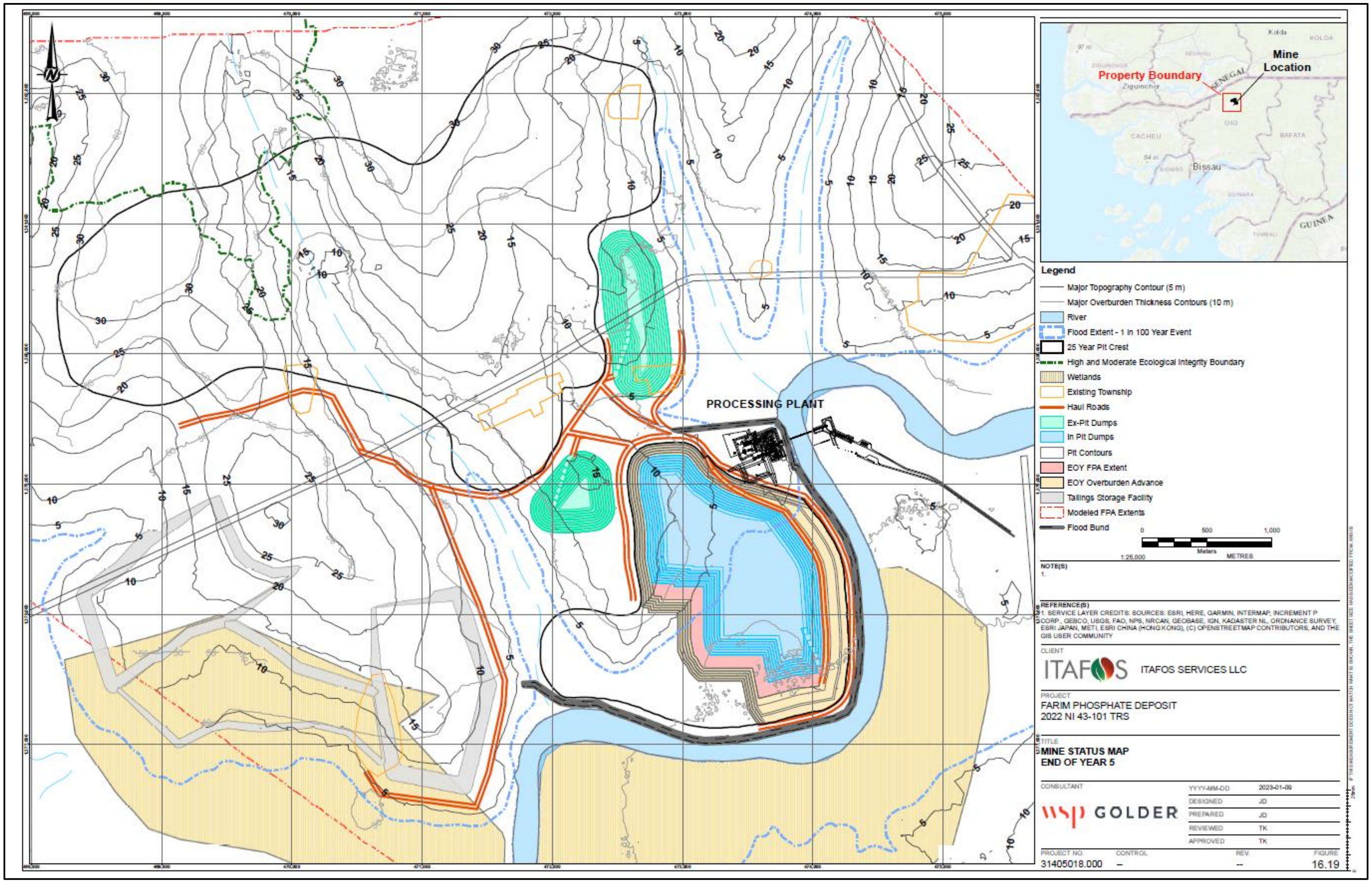
Source: WSP Golder, 2023

Figure 16-19: Mine Status Map — End-of-Year 3



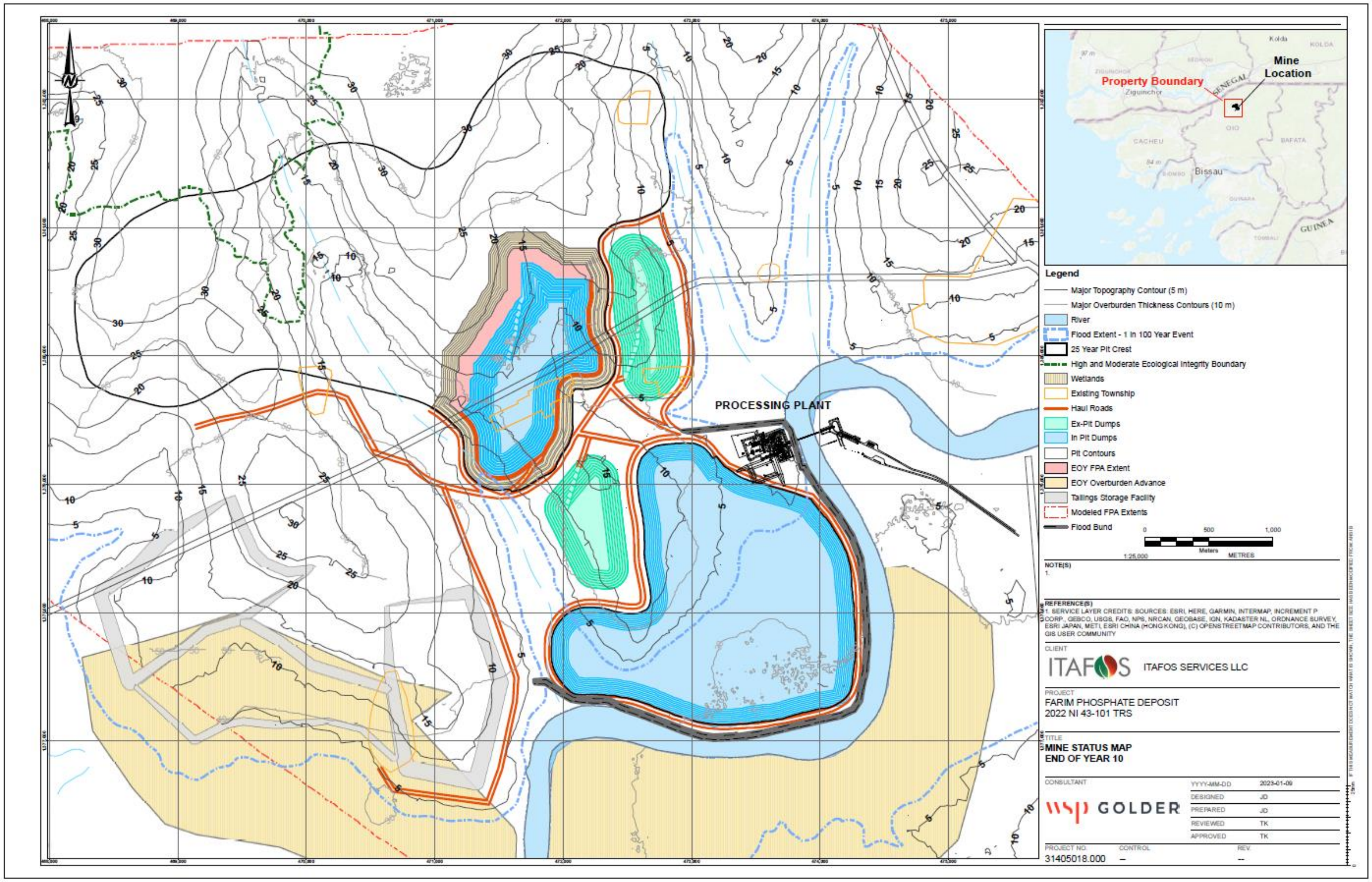
Source: WSP Golder, 2023

Figure 16-20: Mine Status Map — End-of-Year 5



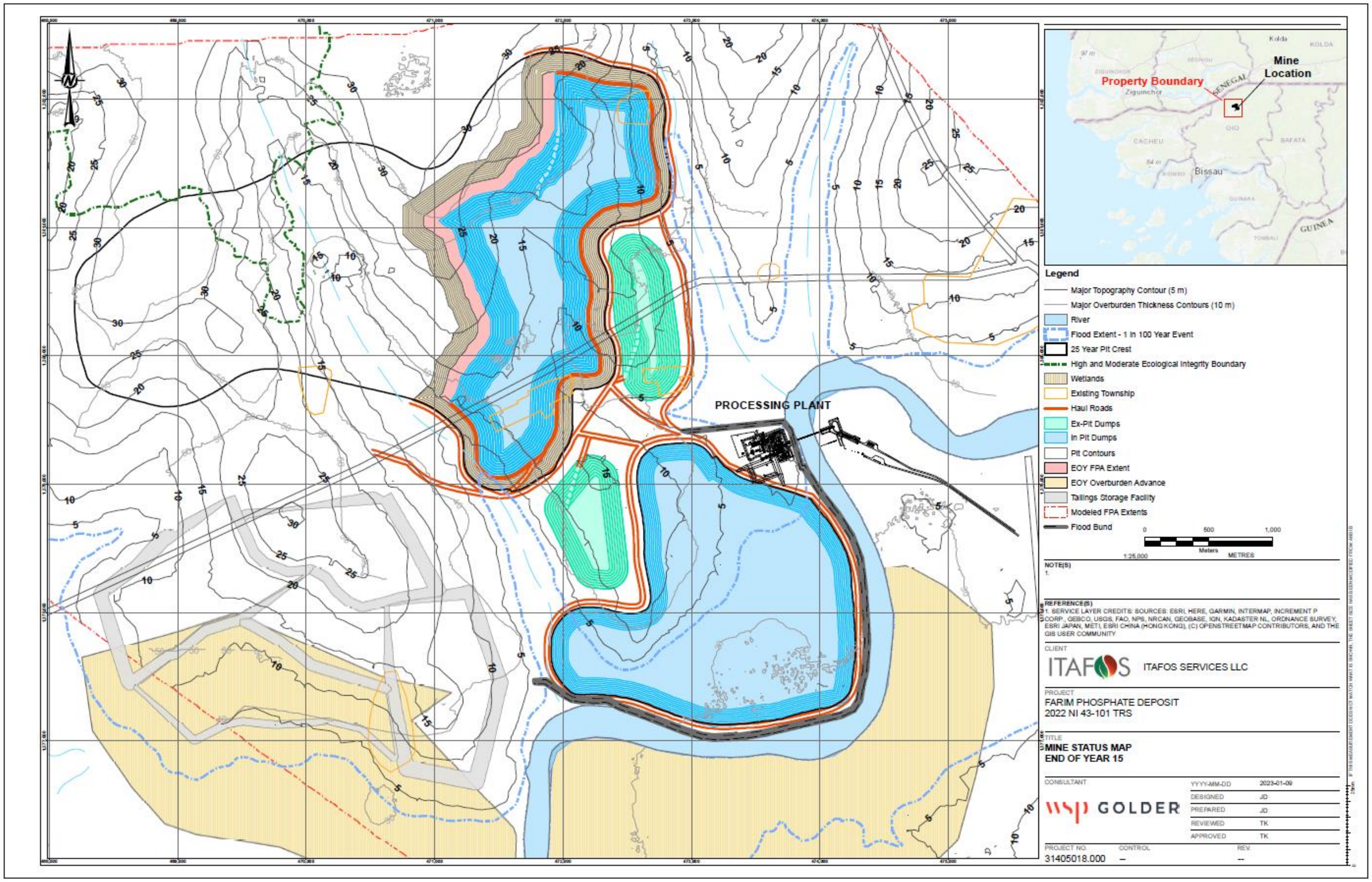
Source: WSP Golder, 2023

Figure 16-21: Mine Status Map — End-of-Year 10



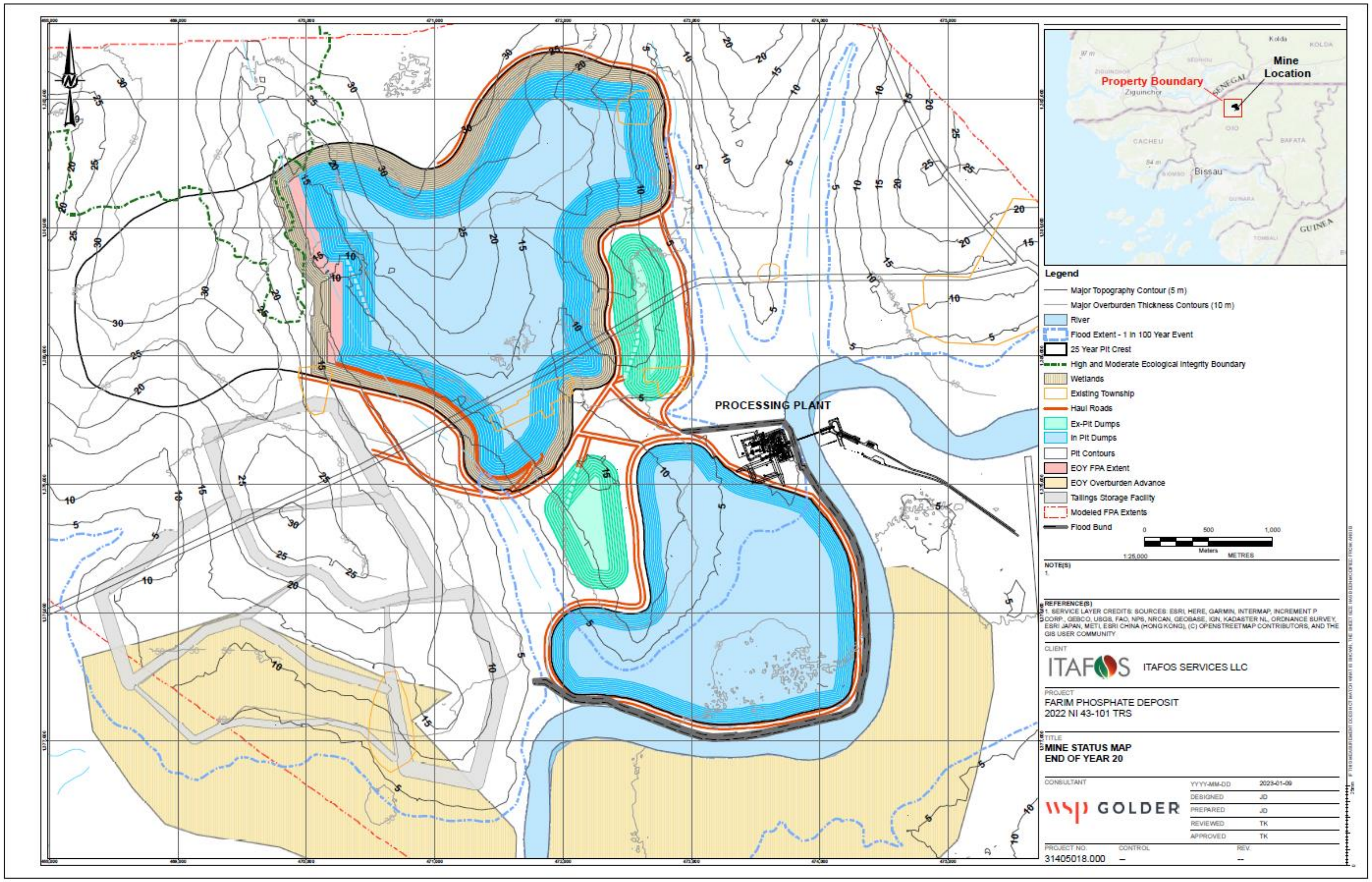
Source: WSP Golder, 2023

Figure 16-22: Mine Status Map — End-of-Year 15



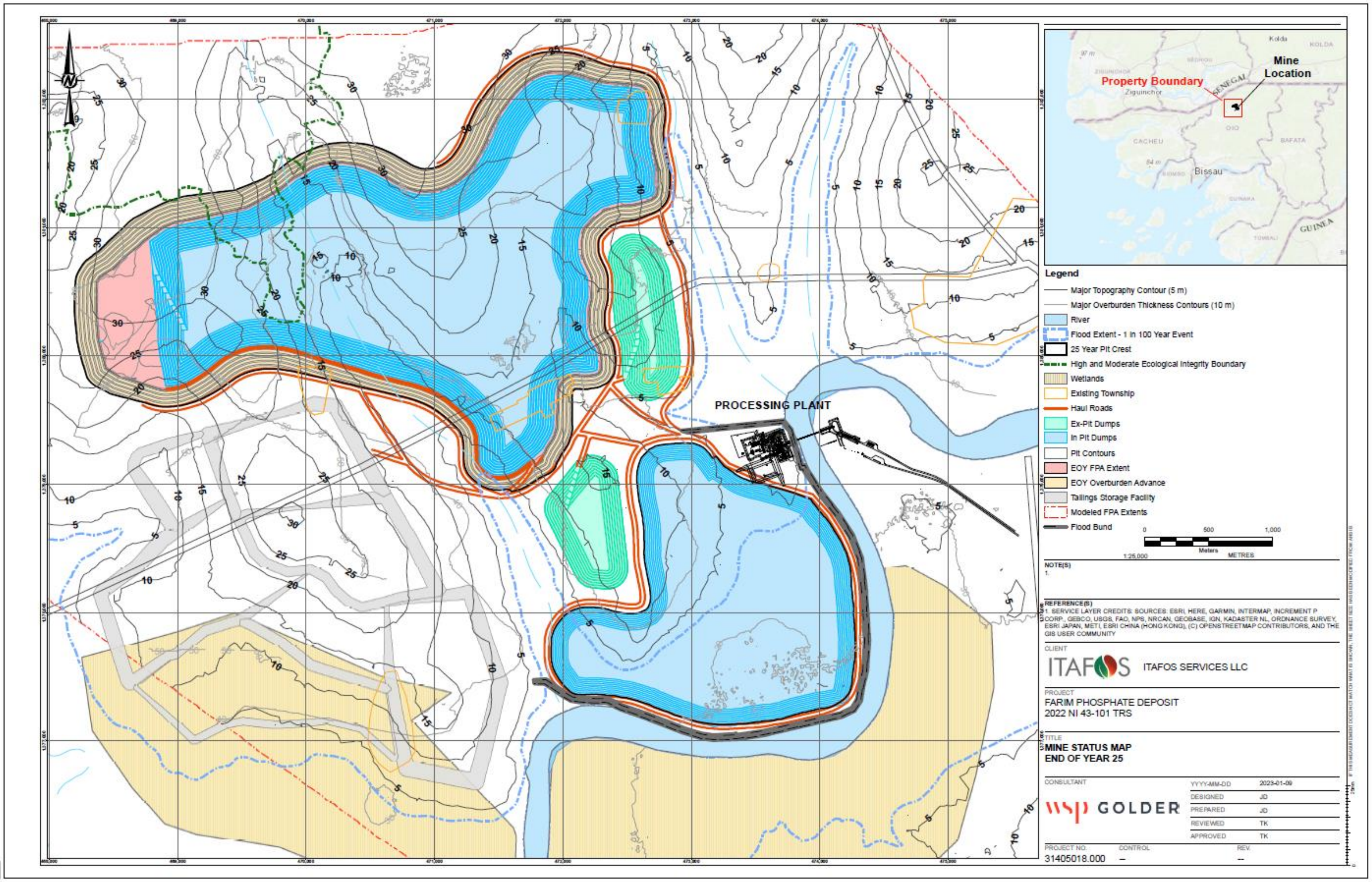
Source: WSP Golder, 2023

Figure 16-23: Mine Status Map — End-of-Year 20



Source: WSP Golder, 2023

Figure 16-24: Mine Status Map — End-of-Year 25



Source: WSP Golder, 2023

16.8.2 Overburden Storage Facilities

The QP developed a dump sequence to meet the waste stripping demands and estimate haulage requirements. Three different types of overburden storage facilities are required to accommodate mine waste: in-pit overburden backfill (IOB), ex-pit waste dump (WD), 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 the QP 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.

In addition to the three types of overburden storage facilities described above, a substantial amount of overburden material was used for construction of the flood protection bund and the embankment walls for the cells of the tailings storage facility. Summaries of the overburden required for the flood protection bund and the embankment walls are provided in Table 16-15 and Table 16-16, respectively.

Table 16-15: Flood Bund Construction Material Requirements

Year	Surface Water Management Material (000s m^3)	Location
0	320	SCD1, SCD2, Flood Bund
1	222	Flood Bund
2	128	Flood Bund
3	111	Flood Bund
4	284	Flood Bund
5	127	Flood Bund
6	95	Flood Bund
7	272	Flood Bund
8	333	Flood Bund – North Pit
Total	1,890	

Table 16-16: TSF Embankment Construction Material Requirements

Cell	Embankment Fill Volume (000s m3)	Embankment Construction Year
1	622	Year 0
2	2,199	Year 2 (Q1/Q2)
3	3,257	Year 7
4	628	Year 13
5	2,075	Year 15
6	1,839	Year 18
7	877	Year 22
Total	11,497	

The external waste dumps at Farim include the short-haul ex-pit dumps WD-1 and WD-2 located between the North pit and South pit extents. The WDs were designed and volumetrically balanced in Vulcan and MineScape using the facility design criteria specified in Section 16.5. 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. The maximum annual stack height of 40 m, which only plays a factor in IOB dump sequence in the North pit, is based on the QP's experience in similar projects and the QP considers it a reasonable operational constraint for the project. A sectional drawing representing this concept has been provided in Figure 16-24. 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. A generic section of an ex-pit WD design is shown in Figure 16-25.

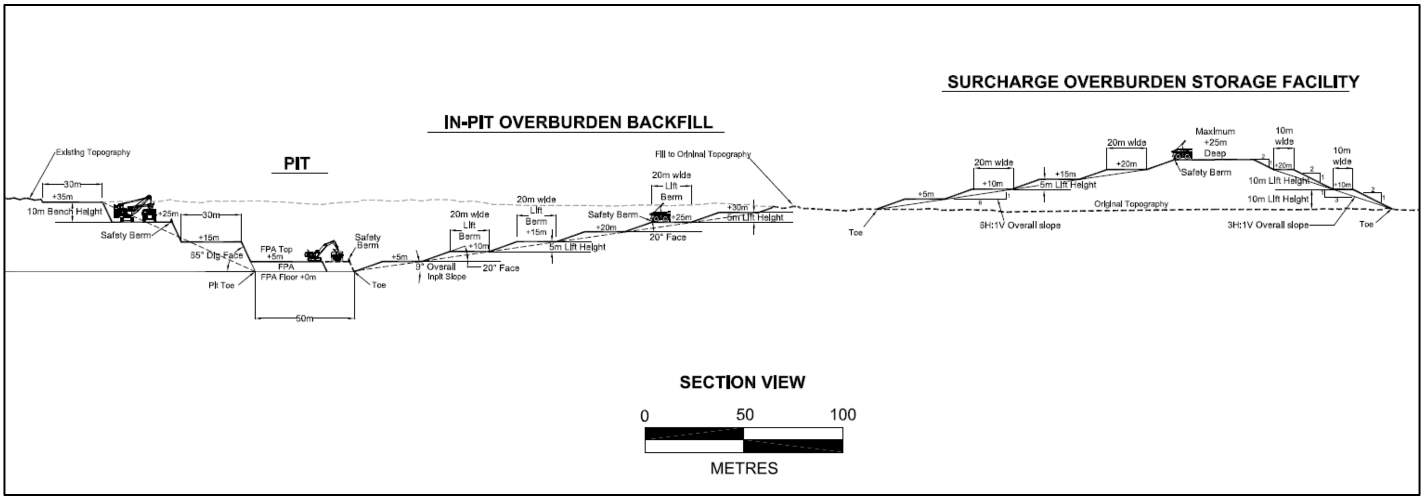
. The WD-1 is designed to handle 0.9M bcm of PAG material within a designed lined area of the facility. WD-2 ex-pit dump can only handle non-potentially acid generating (NAG) material. All other potentially leachable material, including the assumed 7.5 m thick unit of waste directly above the FPA seam and the area with PAG concerns near drillhole DH16-16-09, must be dumped to IOB and covered with sufficient NAG waste. Using an area of influence polygon provided on a map by KP, the QP estimates the amount of potential PAG material near drillhole DH16-GC-09 to be approximately 4-9 Mbcm; this is inclusive of the 7.5 m thick unit of waste assumed to exist directly above the FPA seam. A graphical summary of the waste stripping volumes by PAG and NAG categorization is provided in Figure 16-26.

The QP developed its dump sequence to accommodate the PAG and NAG constraints. Due to geometrical constraints, the majority of waste stripped through the first year of production must be hauled to one of the ex-pit dumps or to the tailings embankment or river protection bund for use in construction, until the area with PAG concerns can be mined out. Although the schedule aims to defer stripping of PAG material, approximately 0.9 Mbcm of PAG material is removed from the pit and dumped to WD-1 in Year 0. Afterwards sufficient backfill volume is available in the South IOB and the PAG material is rehandled and placed in the South IOB.

Beginning in PY4, SOS dump capacity within the South pit becomes available, and all waste stripped through the remainder of the 25-year mine life is deposited in either IOBs or SOSs. Because the North pit mining sequence is not as conducive to in-pit backfilling as the South pit, much of the North pit waste must be hauled to the SOS facility within South pit extents or the smaller SOS facilities within the North pit extents. When the South pit SOS facility is dumped to its designed capacity of 50 Mbcm at 25 m above ground level (aGL) by the end of PY17, all remaining waste can be hauled in-pit due to the transition of mining from the northeastern portion of the North pit to the central and western areas.

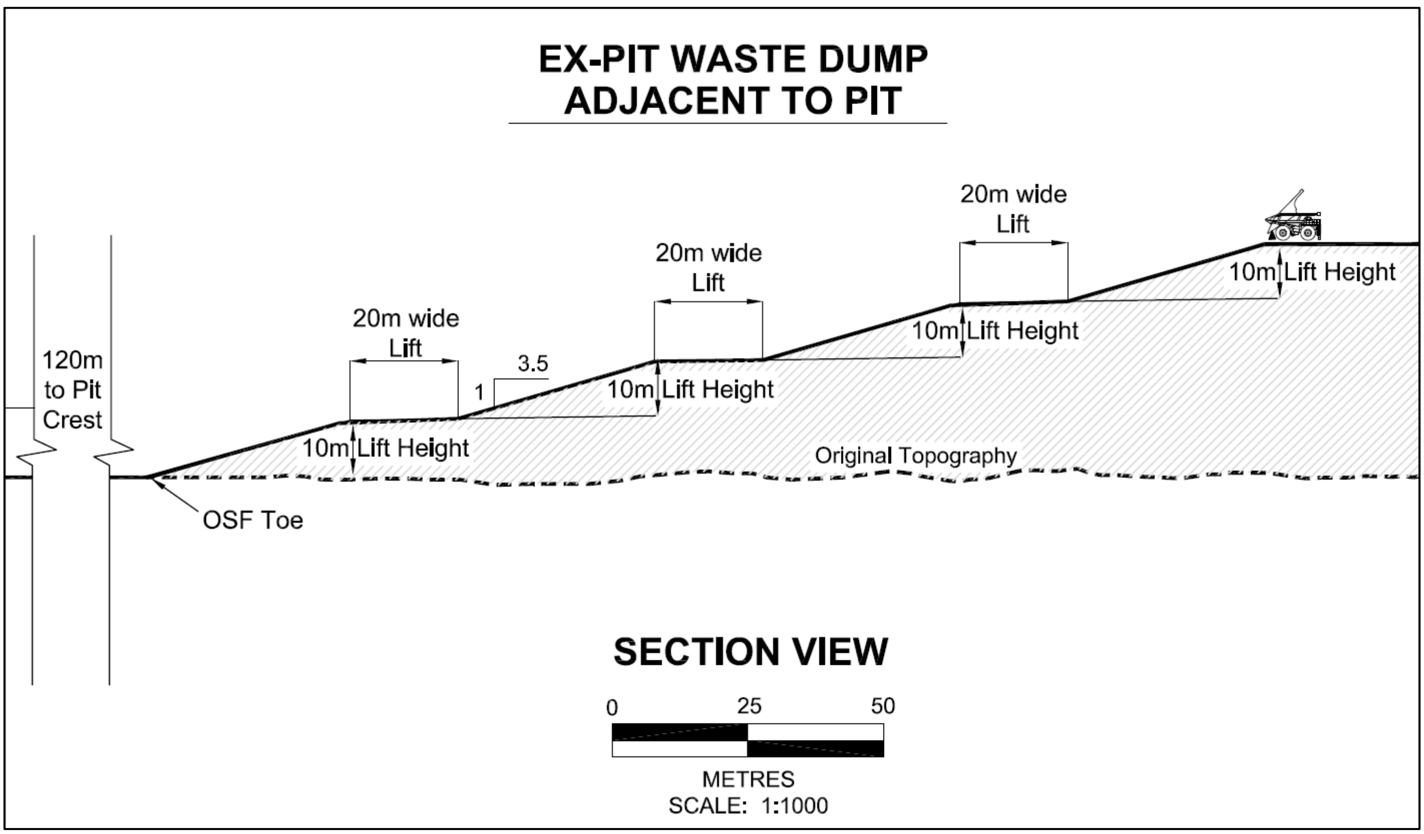
Graphs showing the PAG/NAG characteristics of waste to the various waste dumps and the resultant waste haulage truck requirements and average haul cycle times are in Figure 16-27 and Figure 16-28.

Figure 16-25: Mining Methodology – Profile View



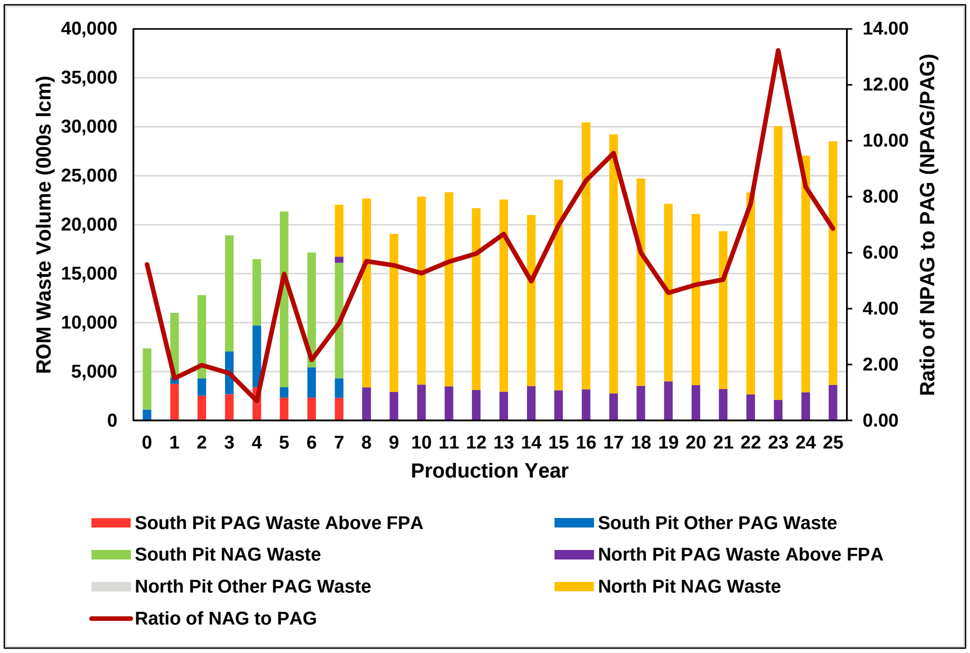
Source: WSP Golder, 2023

Figure 16-26: Ex-Pit Waste Dump – Section View



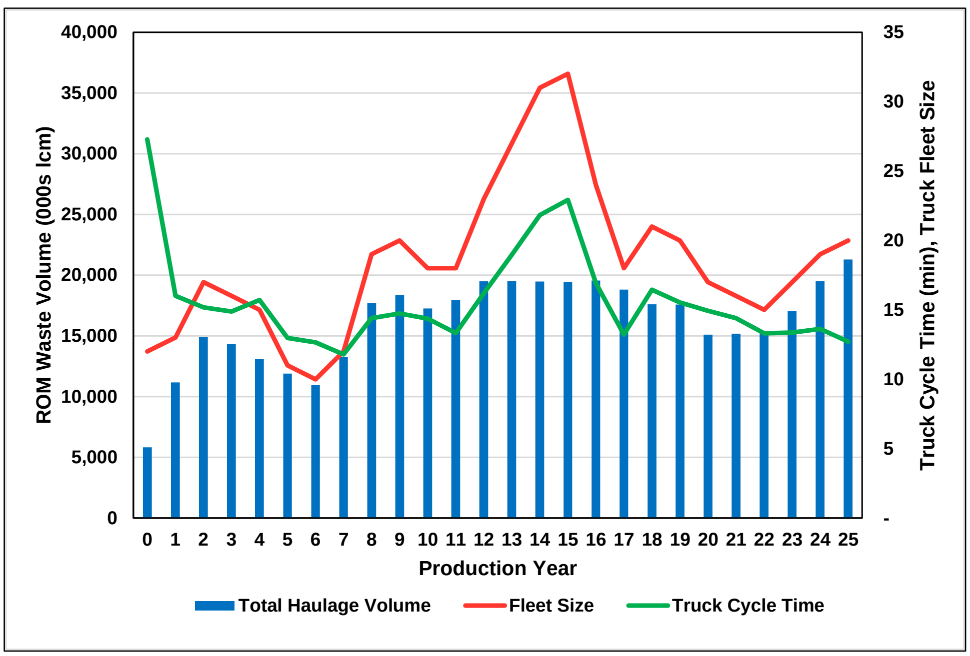
Source: WSP Golder, 2023

Figure 16-27: Annual NPAG / PAG Waste Stripping Characteristics



Source: WSP Golder, 2023

Figure 16-28: Annual Waste Haulage Volumes and Truck Requirements



Source: WSP Golder, 2023

16.8.3 Haul Road Requirements

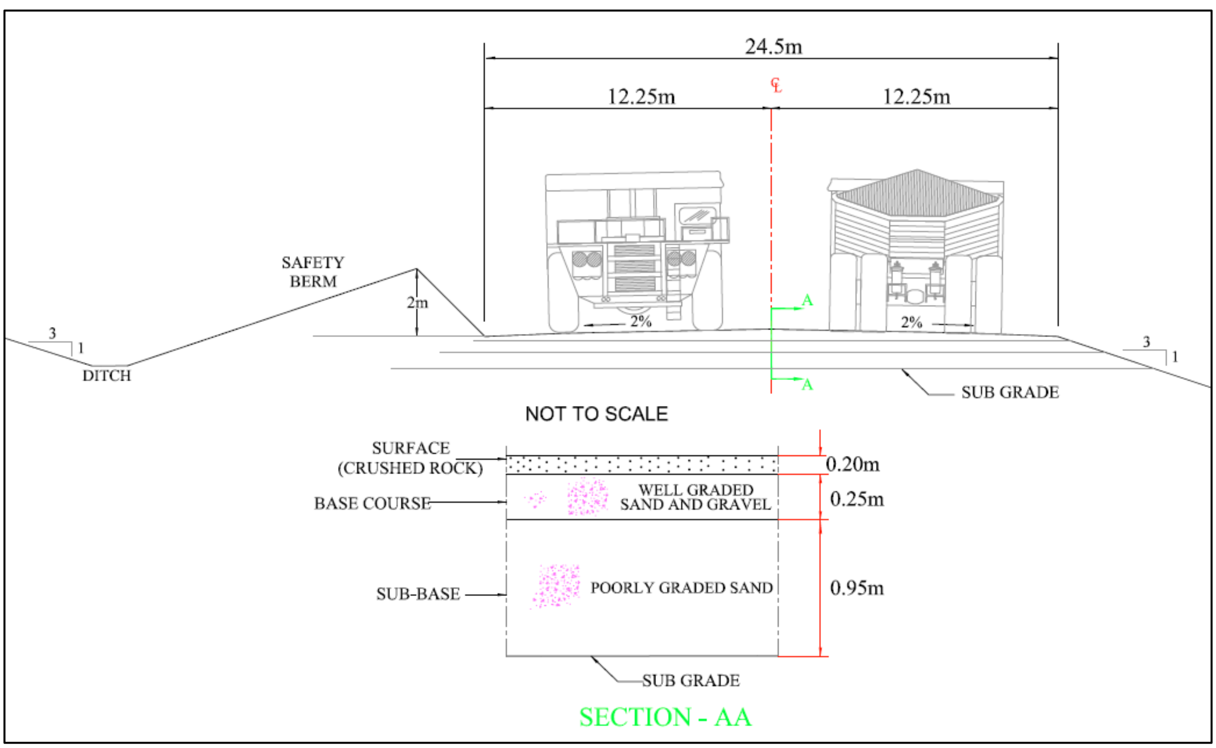
Haul roads will be needed to transport overburden from the open pit face to the in-pit overburden backfill, or ex-pit stockpiles as well as for transporting phosphate to the plant. A stable road base will be important for safety and truck efficiency. Procedures that may be needed to provide a stable road base, such as compaction and import of base course and surface course materials, are important cost considerations.

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 content following dewatering and a CBR value of 10 to 15 was recommended.

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.

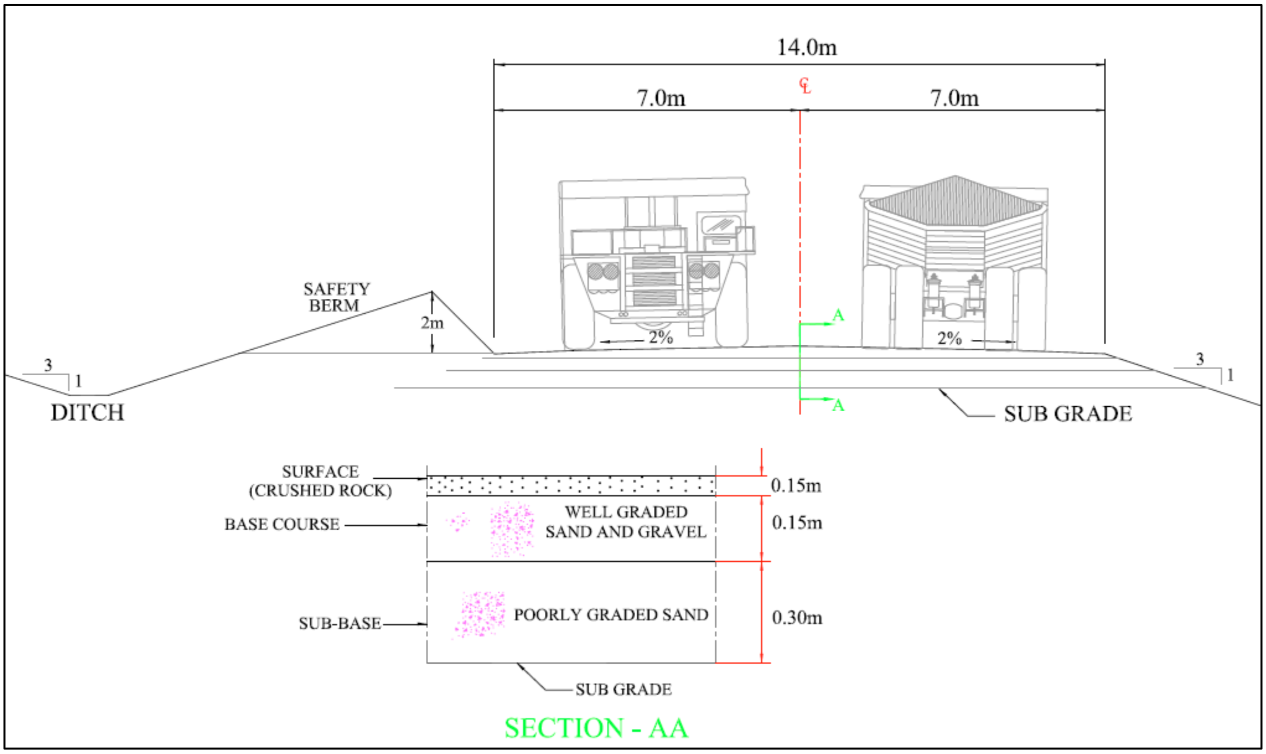
Two different types of haul roads will be constructed for mining activities: overburden haul roads that support a 97-tonne capacity end-dump truck (180 t fully loaded), and matrix haul roads that support a 36-tonne capacity end-dump truck (approximately 72 t fully loaded). Drawings showing the typical design of overburden and matrix haul roads have been provided as Figure 16-29 and Figure 16-30, respectively.

Figure 16-29: Typical Overburden Storage Facility Haul Road Design



Source: WSP Golder, 2023

Figure 16-30: Typical Matrix Haul Road Design



Source: WSP Golder, 2023

Heavy traffic to the TSF in Year 0 will require the construction of a permanent overburden haul road approximately 6.6 km long leading from the pit to the TSF. 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 7 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 7 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 12 through 17 to haul North pit overburden to the SOS above the South pit, the QP 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.

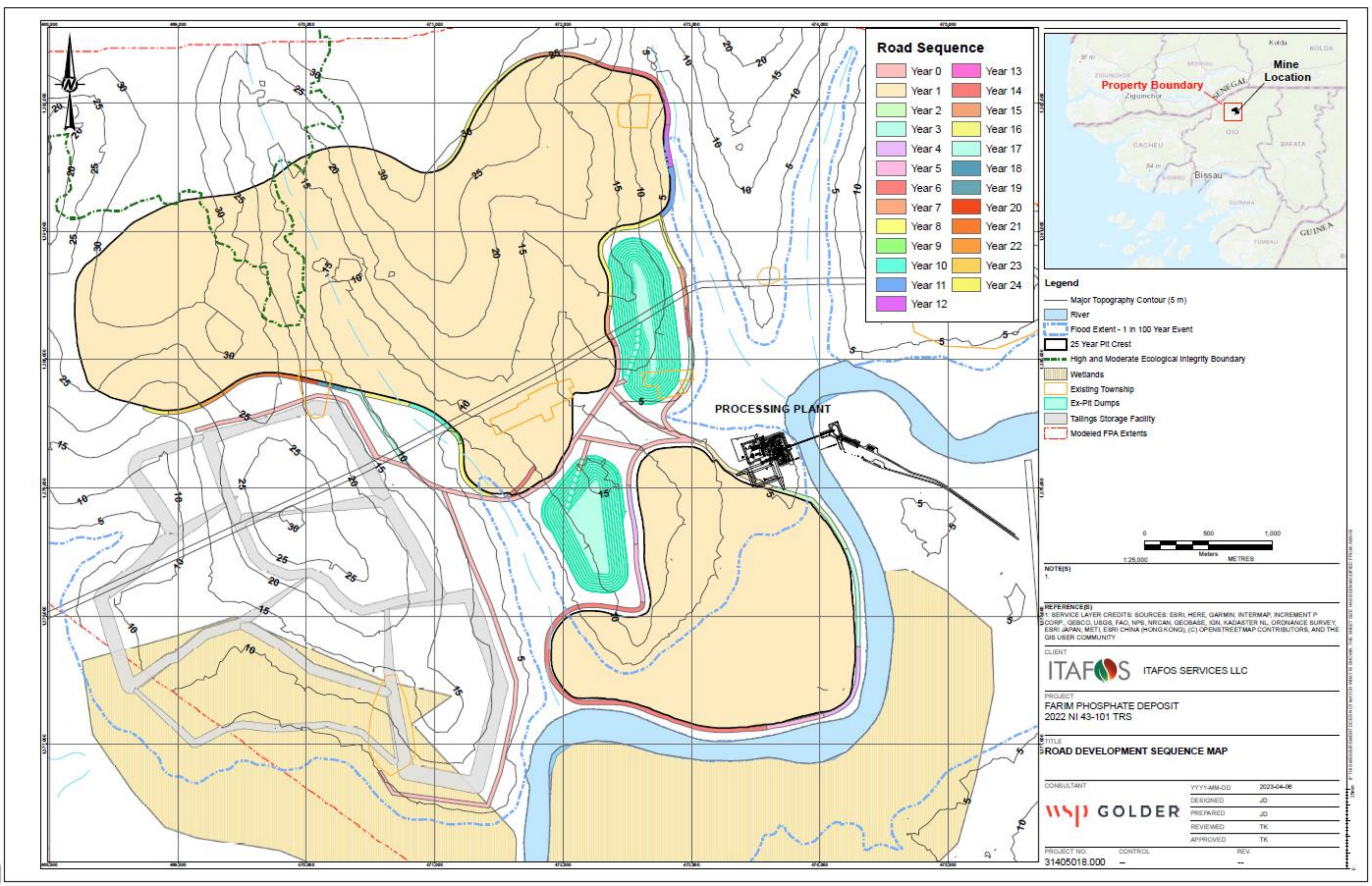
Haul roads providing access to the pit floor will be constructed to provide matrix mining equipment access to the mining face. In-pit backfill ramps/roads and haul roads will be built to overburden haul road specifications and will progress with the pit face for the LOM. In-pit access ramps at a 10% 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 16-29 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, the QP's experience with similar phosphate mines with clayey soil conditions indicates that road rock recovery may be minimal and may not provide meaningful cost savings. For this reason, the QP has assumed no recovery or reuse of road rock for the mining cost estimate.

The haul road development sequence map can be found in Figure 16-31.

Figure 16-31: Road Development Sequence Map



Source: WSP Golder, 2023

16.8.4 Haul Profile Simulations

Pit centroids, IOB centroids, ex-pit WD and TSF 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 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 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.

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 16-17. The results of the FPC haulage simulations for matrix and overburden are provided in Table 16-18, and their effects on haul truck fleet requirements are detailed in Section 21.

Table 16-17: 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 m to 125 m	5%
125 to 225 m	4%
225 to Last 75 m	3%
Last 75 m to Last 25 m	5%
Last 25 m	6%

Table 16-18: FPC Haul Simulation Results

Production Year	Effective One-Way Matrix Haul Distance (km)	Matrix Haul Cycle Time (min)	Effective One-Way Overburden Haul Distance (km)	Overburden Haul Cycle Time (min)
0	0.0	0.0	3.2	27.3
1	1.4	14.1	2.3	16.0
2	2.0	15.9	2.4	15.2
3	1.9	15.7	1.7	14.9
4	2.1	16.2	2.7	15.7
5	1.9	15.6	1.9	13.0
6	2.4	17.3	1.9	12.7
7	2.4	17.3	3.3	11.8
8	2.5	17.4	2.4	14.4
9	3.0	19.5	2.4	14.7
10	2.6	17.8	2.7	14.4
11	2.8	19.2	3.1	13.3
12	3.2	19.9	3.3	16.2
13	3.3	20.6	1.8	19.0
14	3.1	19.5	1.9	21.8
15	3.6	22.3	1.8	22.9
16	3.7	21.9	1.9	16.9
17	3.2	19.8	1.9	13.2
18	3.5	20.6	1.8	16.5
19	3.9	22.3	1.9	15.5
20	4.2	23.3	1.9	14.9
21	4.4	24.7	1.9	14.4
22	4.7	26.0	1.9	13.3
23	5.9	29.0	1.9	13.4
24	5.9	29.5	1.9	13.6
25	6.1	30.3	1.9	12.7

16.8.5 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 15-5 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.

16.9 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-tonne class of trucks selected for the project. A 12.2 m³ 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-tonne haul trucks. These FELs were assigned to excavate full bench height stripping.

Table 16-20 lists the equipment by class and models by manufacturer.

Table 16-19: Summary of Available Equipment Models

Equipment Type	Size Class	Applicable Models
Wheel Loader	12.2 m^3 bucket	Caterpillar 992K, Komatsu WA900-3, Liebherr L850
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, Komtasu HD 325
Water Truck	34-liter tank capacity	Caterpillar 770, Komtasu HD 325
Bulldozer	405 hp	Caterpillar D9R, Komatsu D275AX
Grader	297 hp	Caterpillar 16M
Compactor	147 hp	Caterpillar CS-56

A typical operating configuration for the project is depicted in Figure 16-1. The large (12.2 m³) wheel loaders are used to efficiently expose matrix leaving a temporary face angle of approximately 65°. Dozers in the 405 horsepower (hp) 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 m³ wheel loader can load these trucks with overburden in five passes. Matrix is exposed and mined with the 5 m³ 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 16-21; 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 properly included.

Availability and utilization factors, as shown in were applied to calculate scheduled hours, operating hours, and number of units required.

The rates outlined in Table 16-20 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 backhoe 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 availability, as outlined in Table 16-20 and utilized in this technical report, reflects the QP'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 loader production rates for different mining applications are summarized in Table 16-21.

The bucket fill factors in Table 16-21 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 meters to an equivalent capacity in bank cubic meters 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 are summarized in Table 16-22.

Support equipment for the operations included 405 HP bulldozers assigned to the wheel loader to perform pit cleanup, prepare benches, 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 onsite 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 16-20: Summary of Equipment Delays and Performance Factors of Major Equipment

Delay	DELAY & IDLE TIME (minutes) PER SHIFT											
	Shovel		Backhoe		Wheel Loader			Haul Truck		Dozer		
	Overburden	Matrix	Overburden	Matrix	Overburden	Matrix	Stockpile	Overburden	Matrix	Overburden	Matrix	Support
Operating Delays ("D")	**115**	**125**	**110**	**120**	**65**	**85**	**55**	**60**	**70**	**35**	**35**	**35**
Fuel & Lube ("F&L")	15	15	15	15	15	15	15	15	15	15	15	15
Walking / Moving	30	30	30	30	10	15	10	5	5	10	10	10
Waiting On Trucks ("WOT")	30	30	25	25	10	15	-	-	-	-	-	-
Waiting On Other Equipment	10	10	10	10	5	5	5	5	5	5	5	5
Queuing	-	-	-	-	-	-	-	10	10	-	-	-
Misc. / Other / De-rating Factor for Rainy Season	30	40	30	40	25	35	25	25	35	5	5	5
Idle Time ("I")	**75**	**90**	**75**	**90**	**75**	**90**	**75**	**75**	**90**	**75**	**90**	**75**
Weather ("WTH")	20	30	20	30	20	30	20	20	30	20	30	20
Meal / Break ("M/B")	30	30	30	30	30	30	30	30	30	30	30	30
Shift Change ("SC")	20	20	20	20	20	20	20	20	20	20	20	20
Misc./Other	5	10	5	10	5	10	5	5	10	5	10	5
Total Delays & Idle Time (minutes)	**190**	**215**	**185**	**210**	**140**	**175**	**130**	**135**	**160**	**110**	**125**	**110**
% Of An 8-Hour Shift for Waste/Support or a 12-Hour Shift for Matrix	39.60%	29.90%	38.50%	29.20%	29.20%	24.30%	27.10%	28.10%	22.20%	22.90%	17.40%	22.90%
Mechanical Availability ("MA")	80.00%	80.00%	80.00%	80.00%	82.00%	82.00%	82.00%	90.00%	90.00%	80.00%	80.00%	80.00%
Operational Usage ("OU")	50.50%	62.70%	51.80%	63.50%	64.40%	70.40%	67.00%	68.80%	75.30%	71.40%	78.30%	71.40%
Effective Pit Utilization ("EPU")	**40.40%**	**50.10%**	**41.50%**	**50.80%**	**52.80%**	**57.70%**	**54.90%**	**61.90%**	**67.80%**	**57.10%**	**62.60%**	**57.10%**
Working Hours Per 8-hr. Shift for Waste/Support or Per 12-hr. Shift for Matrix ("W")	3.2	6	3.3	6.1	4.2	6.9	4.4	5	8.1	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	0.3	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	7.9	4.9
Engine Factor ("EF")	1.52	1.3	1.48	1.29	1.2	1.17	1.15	1.15	1.11	1.07	1.04	1.07
Consumption Factor ("CF")	**61.30%**	**65.40%**	**61.30%**	**65.40%**	**63.30%**	**67.40%**	**63.30%**	**71.30%**	**75.40%**	**61.30%**	**65.40%**	**61.30%**
Truck Saturation ("TS")	86.60%	92.30%	88.80%	93.60%	96.20%	96.50%	n/a	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; OU = W/A; EF = EH/W; S = Scheduled hours per shift; EPU = MA x OU; CF = EPU x EF; CD = (D − F&L) + WTH + SC + (0.75 x M/B); W = S x EPU; TS = W / (W + (WOT/60)).

Table 16-21: Summary of Available Equipment Models

Machine	Truck Fleet Capacity (tonnes)	Eff. Truck Payload Capacity (tonnes)	Number of Passes	Rated Bucket Capacity	Bucket Fill Factor (FF)	Swell Factor (S)	Effective Bucket Capacity (1)	
				bcm			bcm	tonnes
				B1			EB1	EB2
Stripping Machines								
Caterpillar 992K – Wheel Loader – Waste	90.5	90.8	5	12.2	0.90	0.787	8.6	18.2
Matrix Loading Machines								
Caterpillar 374DL – Backhoe – Matrix	36.0	29.7	5	5.0	0.95	0.893	4.2	5.9
Caterpillar 992K – Wheel Loader – Stockpile	n/a	n/a	n/a	12.2	0.90	0.893	9.8	13.7
Rock Loading Machines								
Caterpillar 336DL – Backhoe – Support	36.0	34.0	9	2.1	0.90	1.000	1.9	3.8

Machine	Cycle Time (sec) (CT)	Nominal Shift Schedule	Mechanical Availability (2)	Operational Usage (3)	Estimated Prod. Rate Per Shift (4)	Estimated Annual Production Capacity (5)
Stripping Machines						
Caterpillar 992K – Wheel Loader – Waste	48	3 x 8	82.0%	64.4%	2,740 bcm	3,000,500 bcm
Matrix Loading Machines						
Caterpillar 374DL – Backhoe – Matrix	60	1 x 12	80.0%	63.5%	2,175 tonnes	794,000 tonnes
Caterpillar 992K – Wheel Loader – Stockpile	120	3 x 8	82.0%	67%	1,810 tonnes	1,982,000 tonnes
Rock Loading Machines						
Caterpillar 336DL – Backhoe – Support	60	3 x 8	80.0%	65.20%	475 bcm	520,000 bcm

Notes: 1. Effective bucket capacity in bank cubic meter ("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 16-22: Summary of Primary Equipment Requirements

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	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Caterpillar 374DL – Backhoe	0	3	3	3	3	3	3	3	3	3	3	3	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	1	1	1	1	1	1	1	1	1	1	1
Caterpillar 992K – Wheel Loader	3	4	5	7	7	8	7	7	7	6	7	8	7	7	7	8	9	9	8	7	7	6	7	9	8	9
Caterpillar D9R – Dozer	2	3	4	5	5	6	5	5	6	5	6	6	5	6	5	6	7	7	6	5	5	5	6	7	7	7
Caterpillar 777G – End Dump Truck	8	10	12	22	17	18	15	21	15	16	21	23	23	8	17	21	24	23	21	17	15	15	18	14	19	10
Caterpillar 770 – End Dump Truck	1	6	7	7	7	7	7	6	6	8	7	8	8	9	8	9	9	8	8	9	9	10	10	11	12	12
Caterpillar 16M – Motor Grader	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	4	3	3	3	3	3	3	3	3	3
Caterpillar CS-56 – Compactor	3	4	4	6	5	7	5	7	7	6	7	7	7	7	6	7	9	9	7	7	6	6	7	9	8	8
Caterpillar 770 – Water Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Caterpillar 428F – Backhoe Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Screening Plant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Plant	5	7	8	12	11	13	11	12	12	11	12	12	12	12	11	13	16	15	13	12	11	11	12	16	14	15
Fuel/Lube Truck	2	2	2	3	3	4	3	4	4	3	4	4	4	4	3	4	5	4	4	4	3	3	4	5	4	4
Mechanic's Truck	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Pickup Truck	11	11	11	11	11	11	11	11	11	11	11	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	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	1	1	1	1	1	1	1	1	1	1	1
Welding Machine	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

17 RECOVERY METHODS

17.1 Overview

The process design includes the beneficiation plant and the relevant equipment at the Mineral Terminal facility. The beneficiation plant utilizes physical separation processes to classify the feed by size and reject those fractions that contain higher relative concentrations of impurities. The result is a filtered bulk concentrate that achieves product specification targets. The concentrate is then trucked to the Mineral Terminal facility where it is dried prior to ship loadout. The process design was based on testwork managed by KEMWorks and used Ausenco's extensive database of reference projects and in-house modelling programs.

The beneficiation plant is primarily fed from the South pit during Years 1 to 7 of the mine life, and from the North pit during Years 8 to 25. The process is designed to accommodate material from both pits, and the corresponding design criteria are presented in Table 17-1.

Table 17-1: Process Design Criteria

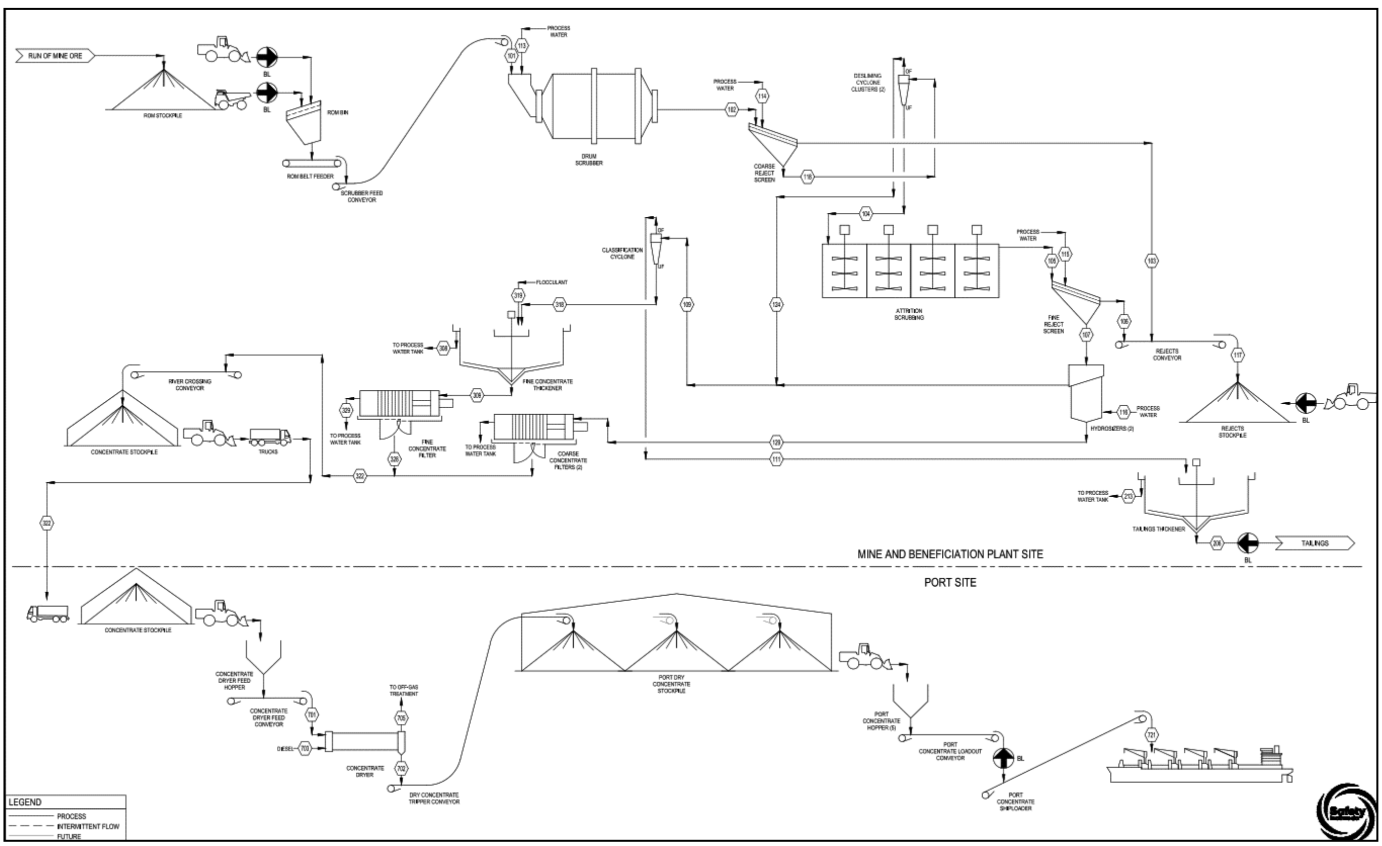
Description	Units	South Pit	North Pit
Annual Throughput	dry t/a	1,750,000	
Concentrate Produced	t/a	1,356,250	1,300,250
Operating Availability	h/a	7,972	
Mass Yield, Concentrate	% w/w	77.5	74.3
Mass Yield, Coarse Rejects	%	6.8	4.0
Recovery, P_2O_5	%	81.8	76.8
ROM Ore Feed Size, 100% passing	mm	75	
Bond Abrasion Index	-	0.08	
Nominal Throughput	dry t/h	219	
Drum Scrubber Retention Time	min	5	
Coarse Rejects Screen, Cut size	mm	5	
Primary Desliming Cyclone – Overflow P_{80}	µm	75	
Secondary Desliming Cyclone – Overflow P_{80}	µm	75	
Attrition Scrubber Retention Time	min	2.5	
Fine Rejects Screen, Cut Size	mm	1.18	
Classification Cyclone – Overflow P_{95}	µm	20	
Fine Concentrate Thickener – Underflow Density	% w/w	40	
Fine Concentrate Filter – Cake Moisture	% w/w	25	
Coarse Concentrate Filter – Cake Moisture	% w/w	11	
Tailings Thickener – Underflow Density	% w/w	15	
Rotary Dryer – Product Moisture	% w/w	3	
Dry Concentrate Stockpile – Residence Time	d	21	
Concentrate Grade – P_2O_5	%	33.9	32.3
Concentrate Grade – CaO	%	46.30	45.00
Concentrate Grade – Al_2O_3	%	0.70	0.89
Concentrate Grade – Fe_2O_3	%	2.84	2.76
Concentrate Grade – MgO	%	0.13	0.09
Concentrate Grade – F	%	2.20	N/A
Concentrate Grade – SiO_2	%	2.23	N/A

A summary of the expected process performance is as follows:

- Average annual throughput of 1.75 Mt/a
- Scrubbing and classification availability of 91%
- Concentrate filtration circuit availability of 82%
- South Pit
    - Mass yield of 77.5% w/w
    - Minor element ratio (MER) of 0.108
    - Concentrate phosphate grade of 33.9%
- North Pit
    - Mass yield of 74.3% w/w
    - MER of 0.116, where MER = [Fe_2O_3% + Al_2O_3% + MgO%] / P_2O_5%
    - Concentrate phosphate grade of 32.3%.

The overall process flow diagram is presented in Figure 17-1.

Figure 17-1: Simplified Process Flowsheet



Source: Ausenco, 2022

17.2 Process Design

17.2.1 Beneficiation Plant

The process design for the beneficiation plant includes the following:

- reclaim and receipt of run-of-mine (ROM) ore
- horizontal drum scrubbing and coarse screening
- two-stage desliming via hydrocyclones
- attrition scrubbing and fine screening
- hydrosizing and hydrocyclone classification
- fine concentrate thickening and filtration
- coarse concentrate filtration
- thickening and overland pumping of tailings to the tailings storage facility (TSF)
- truck loadout facilities including filtered concentrate storage and trucking to the Mineral Terminal site
- water services including raw, fire, potable, gland, process, and reclaim water
- plant services including sewage and compressed air.

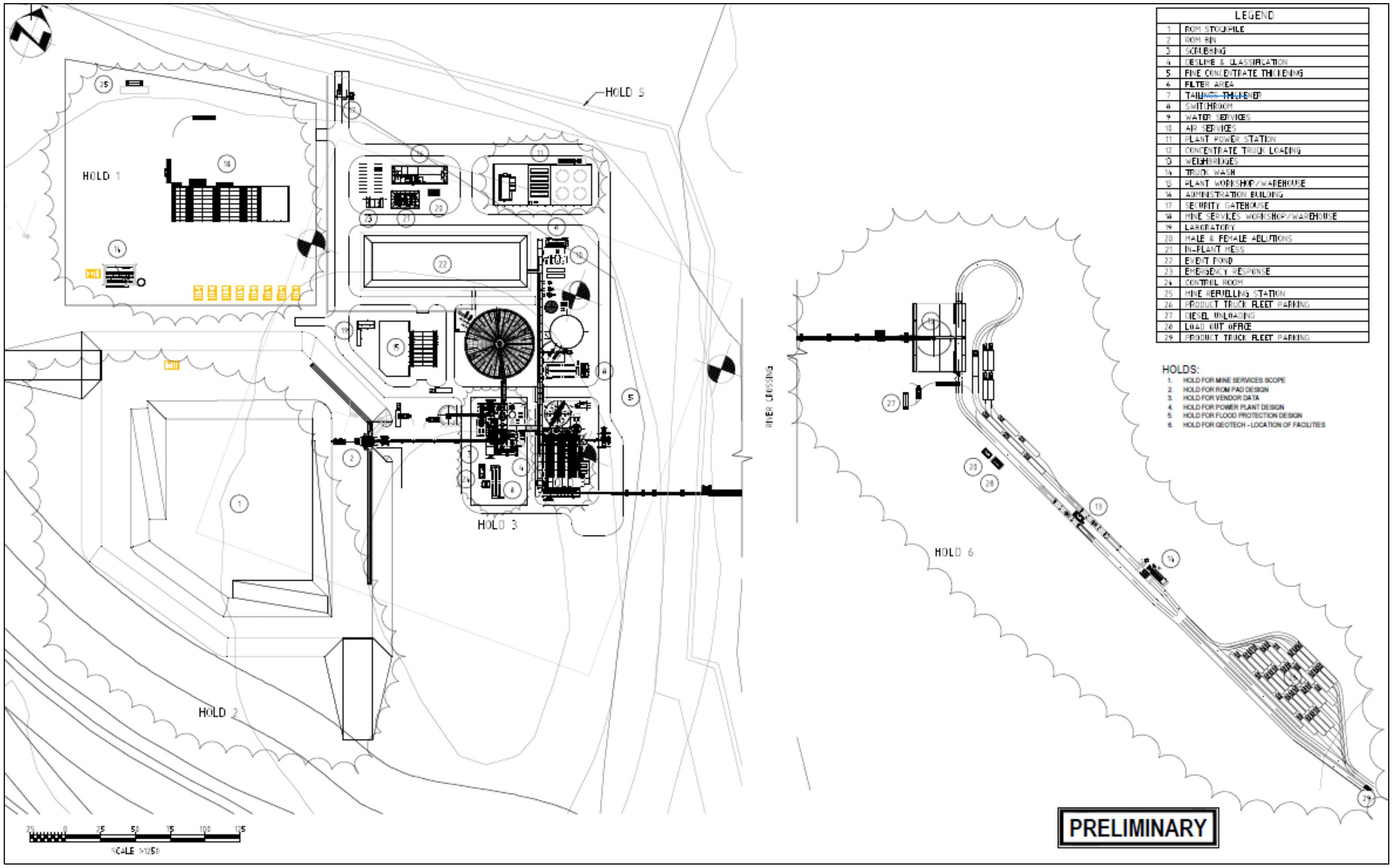
A general arrangement of the plant can be seen in Figures 17-2 and 17-3.

17.2.2 Mineral Terminal Site, Process Equipment

Phosphate concentrate is transported from the beneficiation plant to the Mineral Terminal. A general arrangement of the Mineral Terminal site can be seen in Figure 17-4. The process design includes the following:

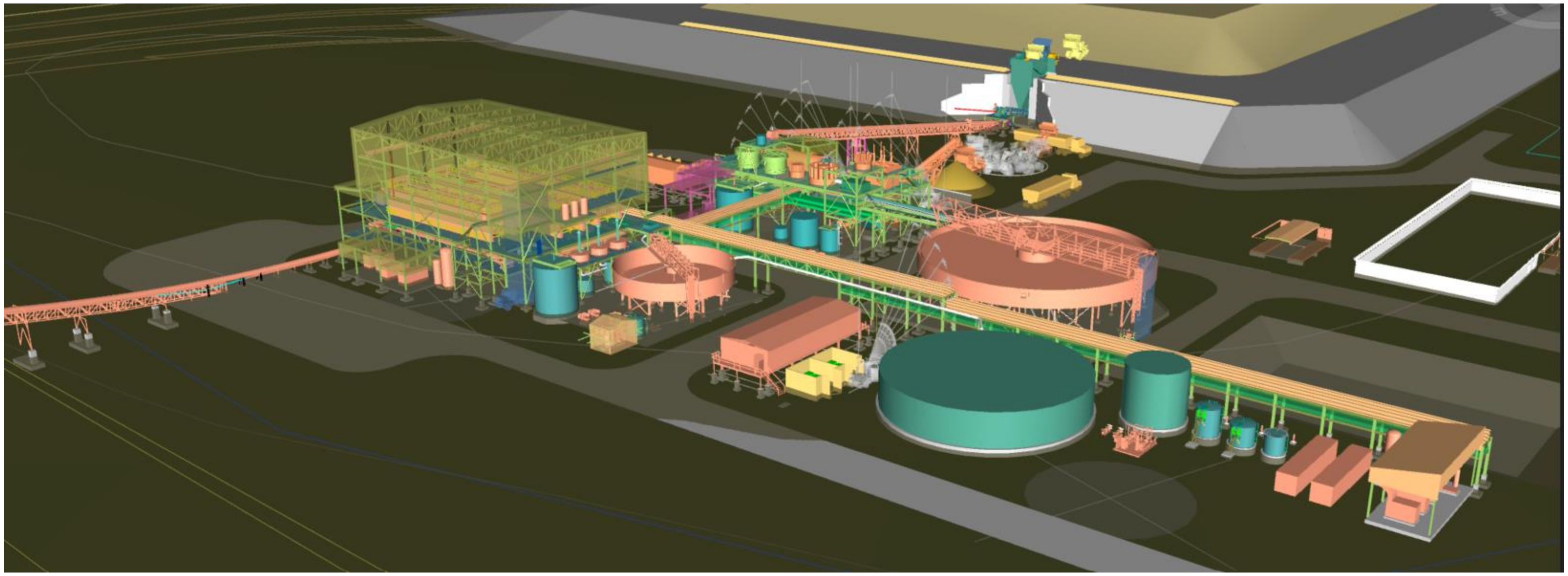
- filtered concentrate storage and reclaim
- concentrate drying and storage
- dry concentrate reclaim and shiploading
- water services (including raw water, fire water, potable water, and effluent treatment)
- plant services including sewage and compressed air.

Figure 17-2: Beneficiation Plant General Arrangement



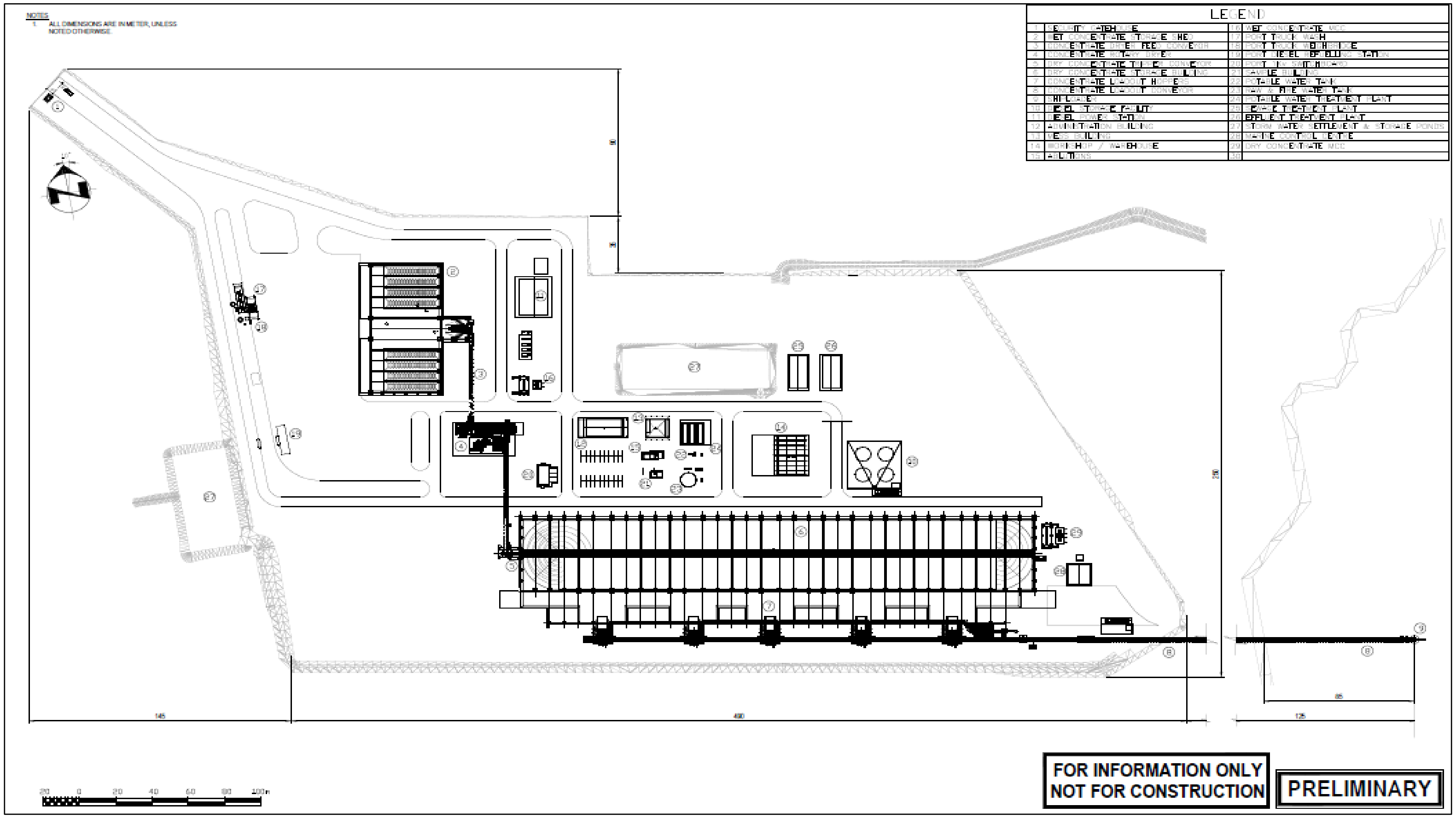
Source: Ausenco, 2023

Figure 17-3: Beneficiation Plant Computer Rendering



Source: Ausenco, 2023

Figure 17-4: Mineral Terminal General Arrangement



Source: Ausenco, 2023

17.3 Plant Design

17.3.1 ROM Material Handling

ROM ore is delivered from the open pit mine to the beneficiation plant by 36 t dump trucks. The trucks deposit ore on the ROM stockpile, which has an operating storage capacity of five weeks, or 175,000 t. Ore from the stockpile is reclaimed to the ROM bin by a front-end loader. The trucks also have the option of direct dumping to the ROM bin, which is sized to handle three truckloads or 108 t. The ROM bin is equipped with a 75 mm aperture static grizzly to prevent oversized material from entering the circuit. A belt feeder extracts ROM ore from the bin to be conveyed to the horizontal scrubber. Oversize material will be reclaimed by a front-end loader.

For metallurgical accounting and plant control purposes, a weightometer and moisture analyzer are installed on the scrubber feed conveyor.

The major equipment and facilities in this area include the following:

- CAT 980H front-end loader
- 75 mm x 75 mm aperture grizzly screen
- ROM bin (61.7 m^3 live volume).

17.3.2 Scrubbing and Classification

ROM ore is fed into the horizontal drum scrubber feed chute at a nominal throughput of 219 dry t/h. Process water is added to maintain a scrubber discharge slurry density of 35% w/w. The 3.6 m x 10 m (D x L) drum scrubber is tire-driven by five 90 kW motors and provides five minutes of slurry retention time.

Product from the horizontal drum scrubber discharges onto a vibrating screen with 5 mm apertures to remove +5 mm material. Water sprays are fitted onto the screen to remove clays and fine slimes from the surface of oversize material. The screen oversize is conveyed to the rejects stockpile. Screen undersize reports to the primary desliming cyclone pump tank and is subsequently processed through two stages of sequential desliming with cyclones at a cut point of 75 µm. Underflow from the secondary cyclone cluster flows into the attrition scrubber. The attrition scrubber has four compartments, each 5.6 m^3 in volume to give a total retention time of five minutes. Process water is added to the underflow of the desliming cyclones as required to maintain a slurry of approximately 55% w/w in the attrition scrubber.

The attrition scrubber discharge reports to a vibrating screen with 1,180 µm slotted openings to remove +1,180 µm material. The screen is fitted with water sprays to clean material surfaces and release agglomerated clay, iron, and phosphate particles. Oversize material from the vibrating screen reports to the rejects stockpile via the same conveyor as the +5,000 µm rejects. A weightometer is installed on the rejects conveyor for accounting purposes. Vibrating screen undersize is pumped to two hydrosizers for classification at 106 µm.

The hydrosizers utilize an upflow current to drive fine particles to the overflow and allow coarse particles to settle and discharge as underflow. Hydrosizer underflow at 70% w/w is diluted to 55% w/w in an agitated tank prior to being pumped to the concentrate filter feed tank. Hydrosizer overflow at -106 µm is combined with overflow from the desliming cyclones in a pump feed tank and pumped to a cyclone cluster for classification at 20 µm.

The classification cyclone cluster utilizes canister style cyclones, and underflow at 45% w/w reports to the fine concentrate pump tank for transfer to the fine concentrate thickener. This material is considered the 106 x 20 µm fine concentrate. The -20 µm cyclone overflow is sent to the tailings thickener.

The major equipment and facilities in this area include the following:

- 3.6 m diameter x 10 m length horizontal drum scrubber

- coarse reject screen (5 mm slotted apertures)
- primary desliming cyclone cluster
- secondary desliming cyclone cluster
- attrition scrubber with four compartments, total 8.0 m length x 2.0 m width x 3.3 m height
- fine reject screen (1,180 mm aperture)
- two vertical-current hydrosizers
- classification cyclone cluster.

17.3.3 Concentrate Dewatering

Classification cyclone cluster underflow is collected in the fine concentrate pump tank and pumped to the high-rate fine concentrate thickener. Filter cloth wash water is also pumped to the fine concentrate pump tank to be re-processed. Flocculant is utilized to aid in the settling process. The slurry is thickened to 40% w/w and pumped to the fine concentrate filter feed tank. Thickener overflow gravity flows to the process water tank.

The concentrate filtration and storage area consider two product streams, the fine concentrate from the fine concentrate thickener and coarse concentrate from the hydrosizer underflow. Fine concentrate is pumped to the vertical plate-and-frame fine concentrate pressure filter from the fine-concentrate filter feed tank. The filter incorporates a membrane squeeze stage and air drying to achieve a target product moisture of 25% w/w. The filter cake discharges to a belt feeder, which includes a weightometer and moisture analyzer for accounting purposes.

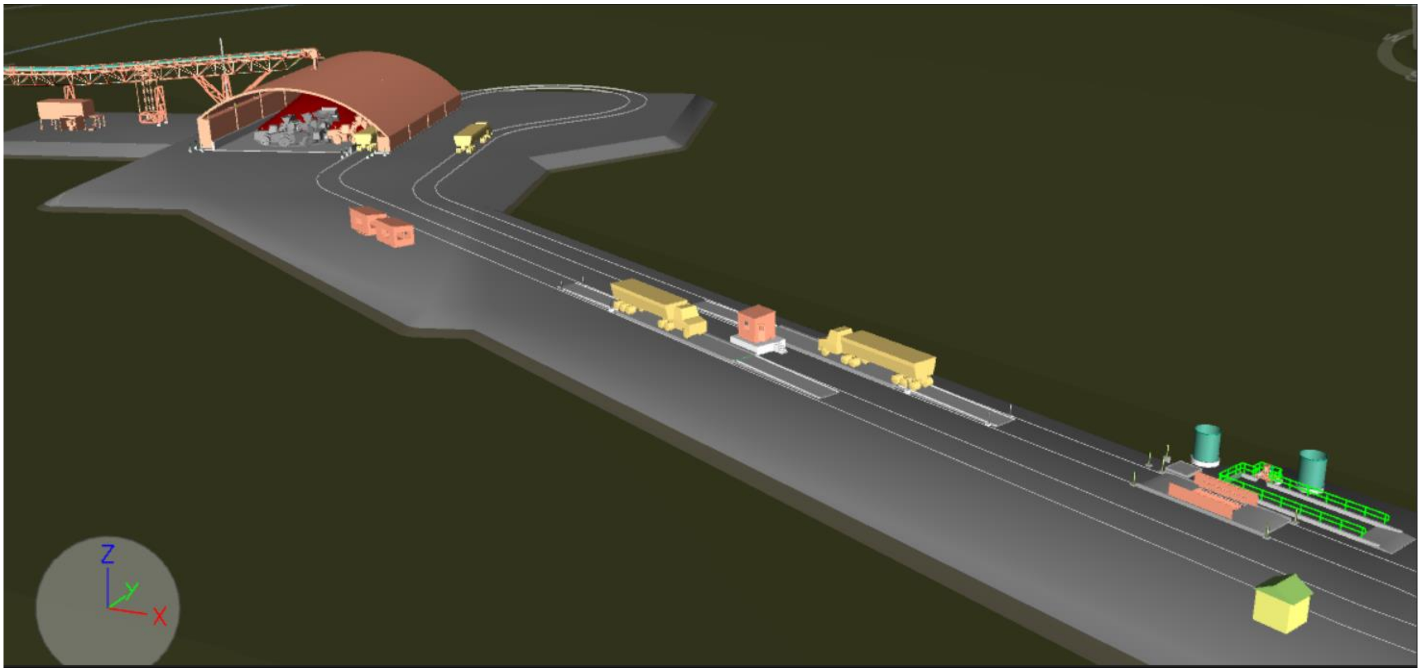
Cloth wash water is collected and returned to the fine concentrate thickener. Filtrate is collected in the filter filtrate tank and is pumped to the process water tank.

Hydrosizer underflow reports to the coarse concentrate filter feed tank. The slurry is pumped to two vertical plate-and-frame coarse concentrate pressure filters which operate in parallel to achieve a target product moisture of 11% w/w. The concentrate from each filter discharges to dedicated coarse concentrate filter cake feeders which have a weightometer and moisture analyzer for accounting purposes.

Cloth wash water is collected and returned to the fine concentrate thickener. Filtrate is collected in the filter filtrate tank and is pumped to the process water tank.

The fine concentrate feeder and coarse concentrate feeders discharge to a river crossing conveyor. The conveyor transports material across the River Cacheu to a filtered concentrate stockpile. The material is subsequently reclaimed with front-end loaders and transported approximately 75 km to the Mineral Terminal facilities with a fleet of 31 t trucks. Figure 17-5 below is a computer rendering of the filtered concentrate shed and truck loadout facility.

Figure 17-5: Filtered Concentrate Shed and Truck Loadout Facility



Source; Ausenco, 2023

The major equipment and facilities in this area include the following:

- 15 m diameter high-rate thickener
- 2.5 m x 2.5 m vertical plate-and-frame fine concentrate pressure filter
- two 2.5 m x 2.5 m vertical plate-and-frame coarse concentrate pressure filters
- three filter cake discharge belt feeders (2 m x 32 m)
- river crossing conveyor (750 mm x 413.4 m)
- ancillary equipment including filter feed tanks, pumps, and compressors.

17.3.4 Tailings Handling

Classification cyclone overflow reports to the tailings thickener. Tailings are thickened to 15% w/w solids and pumped via two-stage underflow pumping to the TSF. Tailings thickener overflow reports to the process water tank. Decant water from the TSF is reclaimed and pumped to the process water tank.

The major equipment and facilities in this area include the following:

- 51 m diameter thickener
- two-stage thickener underflow pumps in duty / standby configuration
- two decant return water pumps
- one process water tank.

17.3.5 Process Plant Sampling

Samplers are located at different points throughout the plant to monitor process conditions and to perform metallurgical accounting. Samplers are located at the following points:

- scrubber feed conveyor
- reject conveyors
- fine concentrate thickener underflow
- hydrosizer underflow (one per unit, for a total of two samplers)
- tailings discharge pipeline.

17.3.6 Flocculant

Dry flocculant is delivered in 25 kg bags and added to the feed hopper. Flocculant is pulled from the hopper by an eductor (jet wet mixing system) using filtered water. The initial flocculant mix strength is 0.5% w/w. Mixed flocculant is aged in the flocculant mixing tank while being stirred at a low intensity. Once ready, the batch is transferred to the flocculant storage tank. Parallel standby and duty metering pumps dose flocculant to the inline mixer where the flocculant is diluted to 0.05% w/w prior to entering the fine concentrate thickener. The nominal flocculant addition rate is 35 g/t.

17.3.7 Water Services

17.3.7.1 Raw/Fire Water

A raw water well pump delivers water to the 894 m^3 combined raw and fire water tank. Raw water is used as makeup water for the process water tank or sent to the water treatment plant.

The fire water pumping system consists of an electric fire water pump, jockey pump, and diesel pump to deliver the flow rate required to protect systems during a fire event.

17.3.7.2 Process Water

The 2022 m^3 process water tank accepts tailings thickener overflow, tailings reclaim return water, fine concentrate thickener overflow, filtrate from fine and coarse concentrate filters, pit dewatering water, and makeup water from the raw water tank. Process water is distributed to the system as necessary to meet process requirements. Process water overflow reports to an onsite event pond to manage excess 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 operating, one on standby) will deliver process water to users distributed throughout the plant. A process water tank 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 reverse osmosis water treatment plant for treatment.

The process plant has a design throughput of 1.75 Mt/a. The plant will produce between 280,000 t/a of tailings and 113,750 t/a of oversized rejects during Years 1 to 7, and 379, 750 t/a of tailings and 70,000 t/a of oversized waste from Years 8 to 25. Ore will be fed into the process at a moisture content of 23% to 25%.

The process design criteria indicate that of the 2,202 m^3/h (Years 1 to 7) to 2,221 m^3/h (Years 8 to 25) of process water demand, 93% (Years 1 to 7) to 90% (Years 18 to 25) will be sourced from tailings and concentrate thickening and filtrate processes. The balance of the process water demand will be sourced, in order of priority, from TSF decant return (RWP), the environmental control dam, and sediment control dam no. 2 (SCD2).

Recommended minimum pumping capacities between specific sources and destinations are summarized in Table 17-2.

Table 17-2: Recommended Maximum Pumping Capacities

Source	Destination	Water Classification	Purpose	Maximum Pumping Capacity (L/s)
TSF	Return Water Pond	Process	Reclaim	65
Return Water Pond	Process Plant	Process	Reclaim return	100
Environmental Control Dam	Return Water Pond	Contact Dirty	Make-up	100
Process Plant (Rainfall Runoff)	Environmental Control Dam	Process	Storage	20
SCD2	Process Plant	Contact Clean	Make-up	100
Treatment Plant	River Cacheu	Clean	Discharge	30
Open Pit Boreholes	SCD2	Clean	Make-up, discharge	3 (per borehole)

17.3.7.3 Gland and Treated Water

The water treatment plant accepts flows from a variety of sources including the event pond, process water tank, and raw water tank. The feed tank has a live volume of 6.3 m^3, and the reverse-osmosis treatment plant has a design capacity of 12 m^3/h. The waste stream from the plant will go to the storm water pond for evaporation. Treated water is stored in the 24 m^3 treated and gland water tank. Treated water is used for flocculant mixing, and gland water is distributed to slurry pumps throughout the beneficiation plant.

17.3.7.4 Potable Water

The potable water treatment plant, which operates by reverse osmosis, receives water from the raw water tank and has a capacity of 0.5 m^3/h. Potable water is stored in a 12 m^3 tank and pumped to potable water users and the safety shower and eyewash system.

17.3.7.5 Sewage System

Plant sewage is collected in a septic tank and transferred to the sewage plant at the contractor's camp by truck as required.

17.3.7.6 Event Pond

An event pond captures all untreated process water and slurry spillage. This spillage is returned to the treatment plant by two event pond return water pumps as required.

17.3.8 Plant Air Distribution

Instrument and plant air are distributed to the process at a design pressure of 750 kPag. Two compressors feed an air-drying system consisting of pre-filters, an air-dryer, and after-filters. The dried air is stored in a receiver and distributed to various users.

17.3.9 Plant Diesel

Diesel will be imported via the Marine Terminal and trucked to the filtered concentrate stockpile area at the truck loadout area. At the wet concentrate stockpile area, diesel is stored in a 68 m^3 storage tank that services the haul trucks and front-end loaders transporting material to the Mineral Terminal site.

At the beneficiation plant, diesel is delivered into two 460 m^3 storage tanks by a pipeline across the river on the conveyor gantry. The storage area includes an oil water separator to collect spillage, as well as a dedicated diesel storage fire suppression system. Diesel is distributed to plant site vehicles, the power plant, and a secondary storage tank which services open pit mining operations.

17.4 Mineral Terminal Site, Process Equipment

17.4.1 Mineral Terminal Concentrate Unloading, Drying, and Storage

Concentrate is delivered from the truck loadout facility by 31 t trucks to the filtered concentrate stockpile. The concentrate stockpile is covered and has a capacity of 2,400 t.

The combined filtered concentrate has a moisture content of 15.7% w/w and is reclaimed from the stockpile to the concentrate dryer feed hopper by a front-end loader. A belt feeder draws concentrate from the hopper to a conveyor equipped with a weightometer. A wet concentrate dust collector has pickup points at the hopper and feeder allowing material to be returned to the conveyor. The conveyor discharges the material to the concentrate dryer screw feeder which feeds the rotary drum concentrate dryer. Hot air is produced by a burner through the combustion of diesel and air and enters the dryer at 600°C. Dried concentrate will exit the dryer at approximate 105ºC with target moisture of 3% w/w to a dry concentrate conveyor. During start-up or upsets, a diverter gate allows concentrate to be discharged to a bunker to prevent off-specification material from entering the dry concentrate stockpile. The dry concentrate conveyor discharges onto a tripper conveyor which distributes material within the dry concentrate storage shed.

Hot dryer off-gas is treated in a dust collector to remove fine entrained concentrate. The collected fines are pneumatically conveyed to a ribbon blender, where they are combined with a wetting agent and report to the dry concentrate stockpile. The air stream reports to a scrubber where raw water is used to further remove entrained material, reduce off-gas temperature, and condense the moisture in the off-gas. The condensate is collected in a scrubber seal tank and pumped to the Mineral Terminal storm water settlement pond. Cooled scrubber off-gas is discharged to the atmosphere.

Major equipment and facilities in this area include the following:

- 2400 t wet concentrate stockpile
- 1.05 GJ rotary drum dryer (3.7 m x 18.3 m, D x L)
- dust collection, off gas treatment, and fines handling system
- port dry concentrate stockpile (100 000 t capacity).

17.4.2 Mineral Terminal Concentrate Loadout

When a ship is berthed, front-end loaders transfer dried concentrate from the storage shed into five concentrate hoppers. Each concentrate hopper is equipped with its own belt feeder for regulated delivery of concentrate onto the Mineral Terminal concentrate loadout conveyor. A belt weightometer is installed on the loadout conveyor to accurately measure the tonnage of concentrate being loaded onto the ships. Reclaimed concentrate from the loadout conveyor is discharged onto the Mineral Terminal concentrate shiploader. The Mineral Terminal concentrate shiploader is a traversing Radial telescoping shiploader with a retractable conveyor discharge spout.

A centralized dust collector is 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 transfers the collected fines to a ribbon blender where the dust is mixed with a wetting agent to increase the bulk density of the fine particles and prevent the fines from becoming airborne. Treated fines are collected in a hopper and a belt feeder under the hopper transports the material to the dried concentrate conveyor. Cleaned air from the dust collector is discharged to the atmosphere. A sampler collects material prior to shiploading for quality accounting purposes.

The shiploading system has a maximum capacity of 1,200 t/h. The average shipment size has a 49,000 DWT concentrate capacity. With an annual mill feed throughput of 1.75 Mt, the South and North pits respectively produce 1.36 Mt and 1.30 Mt of concentrate annually. Over the life of mine, this results in an average concentrate production of 1.32 Mt per annum. Twenty-eight shipments are required to transport this annual concentrate tonnage.

17.4.3 Mineral Terminal Water Services

The major equipment and facilities in the Mineral Terminal water services area include the following:

- Mineral Terminal storm water settlement pond
- Mineral Terminal storm water storage pond
- Mineral Terminal effluent treatment plant
- Mineral Terminal potable water treatment plant
- 10.0 m diameter x 10.8 m height Mineral Terminal raw and fire water tank with a live volume of 801 m^3 live volume
- 1.9 m diameter x 2.1 m height Mineral Terminal potable water tank with a live volume of 5 m^3.

17.4.3.1 Raw and Fire Water

The raw and fire water tank has a live capacity of 576 m^3 and 136 m^3 for fire and raw water, respectively. A vendor package includes an electrical, a jockey and a diesel pump piped in parallel to supply water to fire water users. Raw water is pumped from a standpipe into three streams: the Mineral Terminal potable water treatment plant, the concentrate dryer scrubber, and intermittently to the truck wash station.

17.4.3.2 Potable

The Mineral Terminal potable water reverse-osmosis treatment plant receives raw water and is stored in the Mineral Terminal potable water tank with a capacity of 5 m^3. Back wash-water is sent to the Mineral Terminal storm water settlement pond. Potable water is pumped to potable water users and intermittently to the safety showers and eyewash stations.

17.4.3.3 Effluent

Effluent from the Mineral Terminal area, including site drainage, dirty truck wash water, dryer scrubber water and water treatment plant effluent, are sent to the settlement pond. Overflow from the settlement pond is sent to the storage pond. The Mineral Terminal diesel oil separator discharge also reports to the storage pond. Water from the storage pond is pumped to the effluent treatment plant. Sludge produced by the effluent treatment plant is disposed. Treated effluent from the effluent treatment plant is discharged to the environment. Water will be treated to meet the IFC mine effluent guidelines as per the ESIA. A discharge permit from the local government will be required, and any discharge not meeting the IFC requirements will be subject to penalties.

17.4.4 Mineral Terminal Air Services

Instrument and plant air are distributed to the process at a design pressure of 750 kPag. Two compressors feed an air-drying system consisting of a pre-filter, an air-dryer, and an after-filter. Air is stored in a dry air receiver and distributed to various users.

17.4.5 Mineral Terminal Diesel

Diesel is delivered via barge into one of two Mineral Terminal diesel storage tanks with 520 m^3 capacity each. Diesel is pumped to the following five areas:

- 68 m^3 land side diesel storage tank to a tugboat fueling station at the floating tug berth

- 15 m³ Mineral Terminal power plant diesel day tank
- 68 m³ rotary dryer diesel day tank
- 68 m³ Mineral Terminal vehicle fueling storage tank, which supplies two heavy equipment fueling stations and one light vehicle fueling station
- Mineral Terminal diesel loading pump, allowing diesel transfer to the mine site.

Barge diesel delivery is pumped directly to the dock diesel storage tank where it is distributed to a tugboat fueling station or transported back to the two large diesel storage tanks.

Spillages in the diesel storage area are collected in an oil/water separator. The area also includes a dedicated fire suppression system of fire extinguishers.

17.5 Energy, Water, and Process Materials Requirements

17.5.1 South Pit

The electrical power requirements for the first seven years of operation (mining the South pit) are presented in Table 17-3. Power is generated on site through a mix of solar and diesel generation.

Table 17-4 presents the consumables, raw water, flocculant, and fuel requirements for the first seven years of mine life.

Table 17-3: Power Requirements for First Seven Years of Mine Life (Mining the South Pit)

WBS	Area	Installed (kW)	Nominal Demand (kW)	Operating Hours per Year (h/a)	Consumption (MWh/a)
120	Feed Preparation	37.0	15.9	7,972	126.8
130	Scrubber Feed	56.1	27.8	7,972	221.9
140	Scrubbing/Desliming/Tailings	2,547.0	1,217.7	7,972	9,707.0
150	Fine Concentrate Thickening	100.0	35.7	7,972	284.4
190	Concentrate Filtration & Storage	2,717.0	1,464.4	7,183	10,519.0
210	Reagents	9.6	4.8	7,972	38.1
230	Water Services	1,060.5	393.3	7,972	3,134.6
240	Plant Services	0	0	7,972	0
250	Plant Air Services	125.0	41.7	7,972	332.2
270	Electrical Services	25.0	13.9	7,972	110.7
330	Power Supply	0	0	7,972	0
340	Tailings Storage Facility	110.0	40.9	7,972	325.8
350	Buildings – Plant Site	370.0	205.6	8,760	1,800.7
N/a	Pit dewatering	500.0	331.0	8,760	2,900.0
440	Mining Facilities	186.8	84.9	7,972	676.4
Plant Site Total		7,844.0	3,877.3		30,177.6
730	Mineral Terminal Water Services	107.1	25.6	7,972	204.0
740	Concentrate Drying/Storage/Loadout	1,622.7	1,044.8	7,972	8,328.6
750	Mineral Terminal Utilities & Services	129.0	54.8	7,972	436.7
760	Mineral Terminal Fuels	131.1	47.7	7,972	380.2
770	Mineral Terminal Electrical Services	15.0	8.3	7,972	66.4
780	Buildings – Mineral Terminal	260.0	144.4	8,760	1,265.3
Mineral Terminal Site Total		2,264.9	1,325.6		10,681.2
Grand Total		10,108.9	5,203.0		40,858.8

Table 17-4: Consumables, Water, and Fuel Requirements – South Pit

Input	Rate	Unit	Annual Consumption	Unit
Raw Water	54.6	m^3/h	435,249	m^3/a
Drum Scrubber Liner	1	#/a	1	no. per year
Filter Cloth	5,000	cycles/unit	2 845	no. per year
Pressing Diaphragms	80,000	cycles/unit	89	no. per year
Filter Plates	623,864	cycles/unit	12	no. per year
Flocculant	35	g/t	10,533	kg/a
Gasoline	-	-	120,240	L/a
Diesel				
Dryer	2,088	L/h	18,290,911	L/a
Vehicles	-	-	2,936,044	L/a
Power Plant (Plant)	-	-	4,915,469	L/a
Power Plant (Mineral Terminal)	-	-	1,679,775	L/a
Total Diesel	-	-	27,822,199	L/a

17.5.2 North Pit

The electrical power requirements for year 8 and beyond of the mine life when mining the North pit are presented in Table 17-5. Power is generated on site through a mix of solar and diesel generation. For more details on the solar and diesel generation, refer to Section 18-8.

Table 17-5: North Pit Power Requirements – Year 8 and Beyond (Mining the North Pit)

WBS	Area	Installed (kW)	Nominal Demand (kW)	Operating Hours per Year (h/a)	Consumption (MWh/a)
120	Feed Preparation	37.0	15.9	7,972	126.8
130	Scrubber Feed	56.1	27.8	7,972	221.9
140	Scrubbing/Desliming/Tailings	2,547.0	1,217.7	7,972	9,707.0
150	Fine Concentrate Thickening	100.0	35.7	7,972	284.4
190	Concentrate Filtration & Storage	2,717.0	1,403.9	7,183	10,084.7
210	Reagents	9.6	4.6	7,972	36.5
230	Water Services	1,060.5	393.2	7,972	3,134.6
240	Plant Services	0	0	7,972	0
250	Plant Air Services	125.0	41.7	7,972	332.2
270	Electrical Services	25.0	13.9	7,972	110.7
330	Power Supply	0	0	7,972	0
340	Tailings Storage Facility	110.0	40.9	7,972	325.8
350	Buildings – Plant Site	370.0	205.6	8,760	1,800.7
	Pit Dewatering	350.0	240.0	8,760	1,892.0
440	Mining Facilities	186.8	84.9	7,972	676.4
Plant Site Total		7,694.0	3,725.7		28,733.7
730	Mineral Terminal Water Services	107.1	25.6	7,972	204.0
740	Concentrate Drying/Storage/Loadout	1,622.7	1,001.6	7,972	7,984.7
750	Mineral Terminal Utilities & Services	129.0	54.8	7,972	436.7
760	Mineral Terminal Fuels	131.1	47.7	7,972	380.2
770	Mineral Terminal Electrical Services	15.0	8.3	7,972	66.4
780	Buildings – Mineral Terminal	260.0	144.4	8,760	1,265.3
Mineral Terminal Site Total		2,264.9	1,282.5		10,337.3
Grand Total		9,958.9	5,008.2		39,071.0

Table 17-6 presents the consumables raw water, flocculant, and fuel requirements for Years 8 to 25 of the mine life.

Table 17-6: Consumables, Water, and Fuel Requirements – North Pit

Input	Rate	Unit	Annual Consumption	Unit
Raw Water	69.9	m^3/h	557,215	m^3/a
Drum Scrubber Liner	1	#/a	1	#/a
Filter Cloth	5,000	cycles/unit	2,727	#/a
Pressing Diaphragms	80,000	cycles/unit	85	#/a
Filter Plates	623,864	cycles/unit	12	#/a
Flocculant	35	g/t	11,344	kg/a
Gasoline	-	-	120,240	L/a
Diesel				
Dryer	2,002	L/h	17,535,673	L/a
Vehicles	-	-	2,833,335	L/a
Power Plant (Plant)	-	-	4,915,469	L/a
Power Plant (Mineral Terminal)	-	-	1,679,775	L/a
Total Diesel	-	-	26,964,253	L/a

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The Farim project is located in the northern part of central Guinea-Bissau, West Africa, approximately 25 kilometers (km) south of the Senegal border, approximately 5 km west of the town of Farim, and approximately 120 km northeast of Bissau, the capital of Guinea-Bissau. The project site 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 River Cacheu. From the town of Farim, the property can be accessed via a 5 km unpaved dirt road.

The main project area In Farim will have an open pit mining operation with two individual pits identified as the "South pit" and the "North pit", a tailings storage facility (TSF) with return water pond (RWP) for reclaim to the process plant, a process (beneficiation) plant, two waste overburden dumps, access roads, surface water management infrastructure, a truck loadout facility, and other minor infrastructure.

Once processed in the Farim process plant, the filtered concentrate is trucked 75 km to a second facility located at the Mineral Terminal of Ponta Chugue where it is unloaded, conveyed through a rotary dryer, stockpiled, and directly transferred via shiploader onto 49,000 DMT ships. Both facilities in Farim and Ponta Chugue will produce their own power via hybrid electrical generating sets.

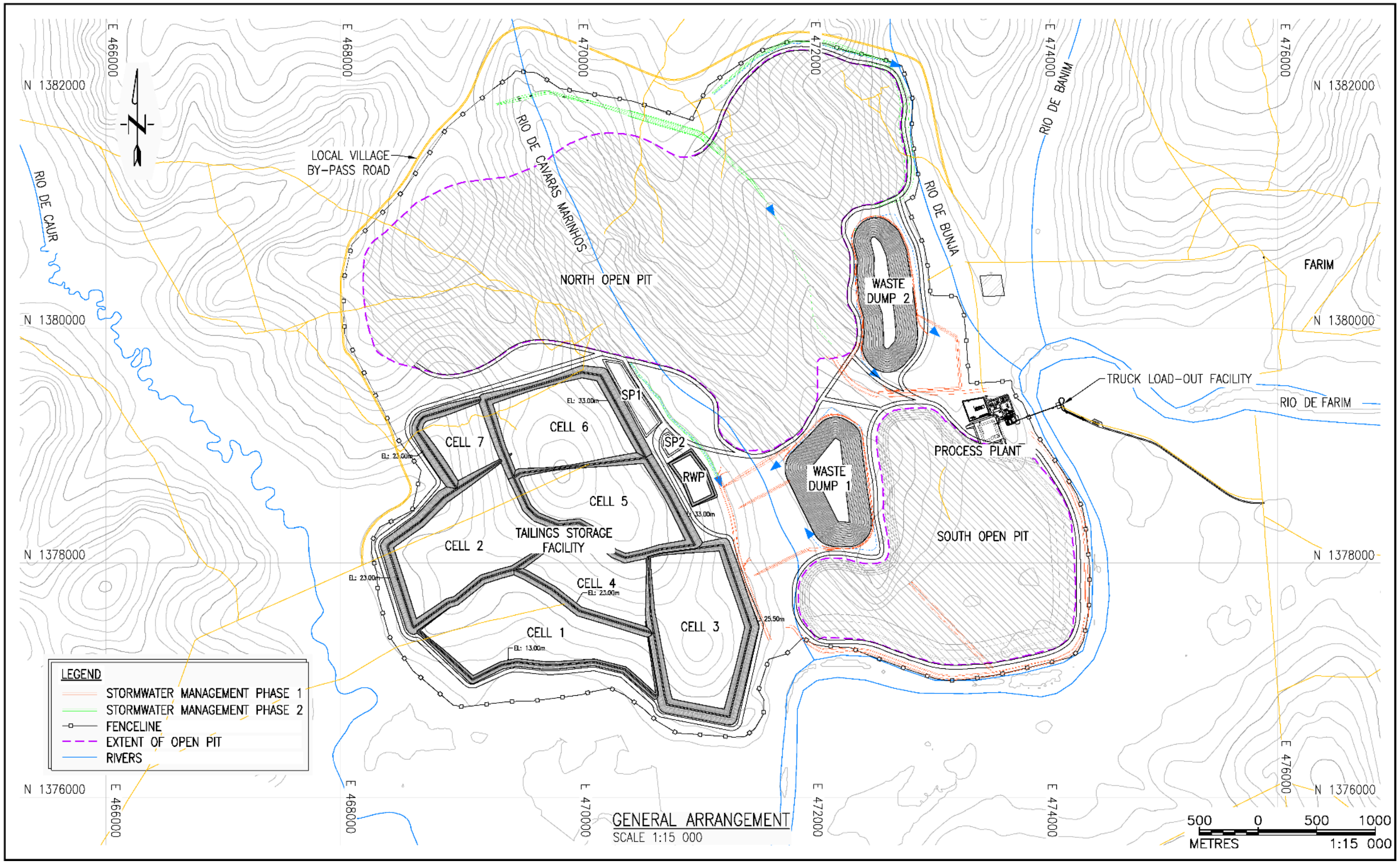
In general, the Farim project is isolated and requires significant infrastructure construction as described in this section.

Figure 18-1 shows the mine site layout plan located near the town of Farim. Figure 18-2 shows the general arrangement of the Farim phosphate process plant. Figures 18-3 and 18-4 provide a site plan and computer rendering of the Ponta Chugue facility.

18.2 Site Access

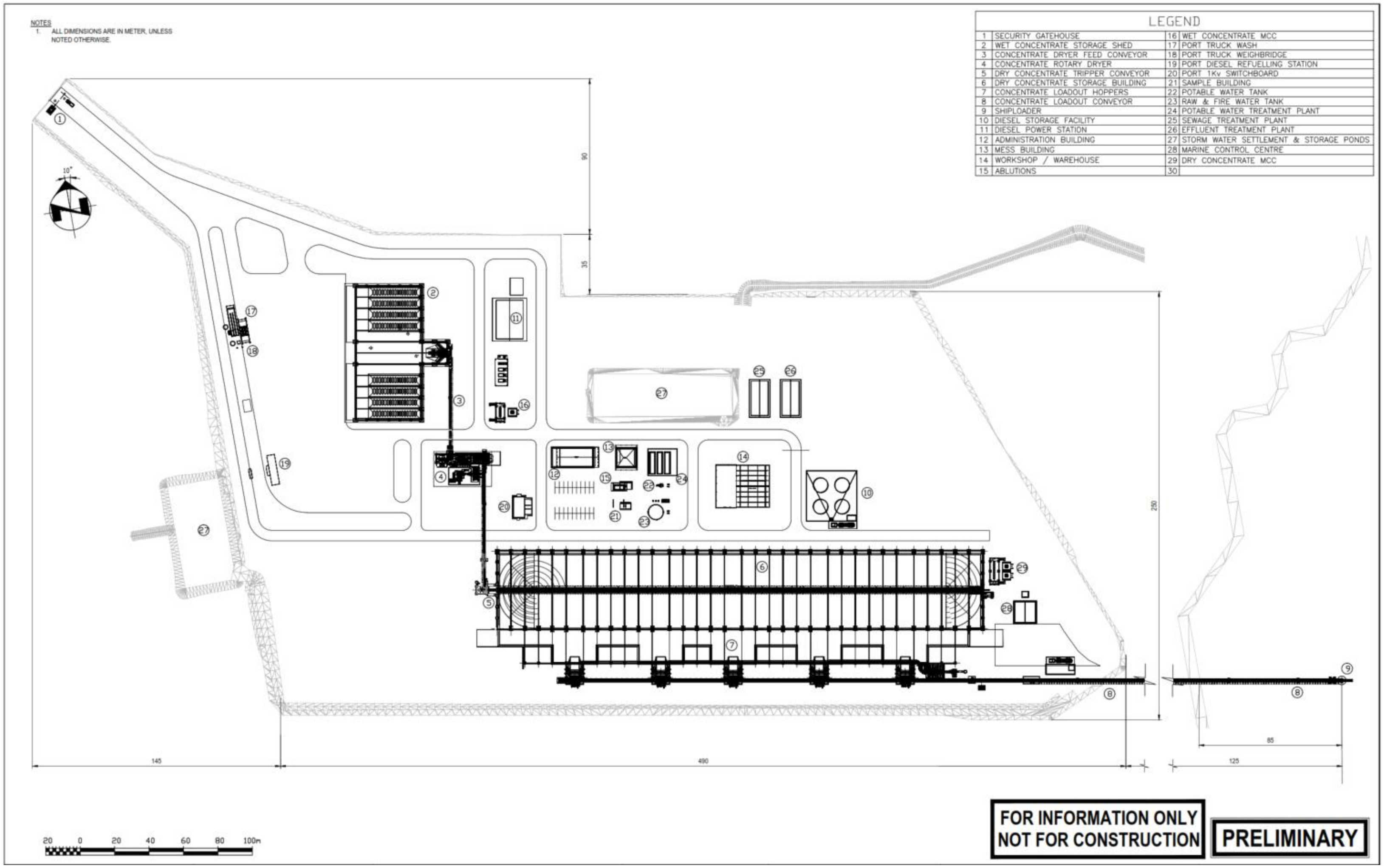
The mine site is located approximately 5 km west of the town of Farim. The mine site is bound by the River Cacheu to the east and south of the open pit. The beneficiation plant has been located between the southern and northern open pits, adjacent to the River Cacheu. The plant area, including site buildings, is approximately 200 m x 200 m. The beneficiation plant is located at the narrowest point of the River Cacheu, where it is approximately 150 m wide, to minimize the cost of the conveyor river crossing. A conveyor is utilized to transfer filtered concentrate into a storage bin on the east side of the river, where trucks are loaded (see Figure 17-2). A 2 km gravel road will be constructed to connect the truck loading facility to the existing paved highway to the east.

Figure 18-1: Mine Site Layout Plan



Source: Knight Piesold, 2023

Figure 18-2: Ponta Chugue Site Plan



Source: Lycopodium, 2019

Figure 18-3: Ponta Chugue Pictorial View

Source: Lycopodium, 2019

A tailings storage facility with embankments constructed from open pit overburden is located approximately 5 km west of the process plant. For overburden storage, two waste dumps (WDs) are located west of the process plant between the south and North pits. The WD locations have been chosen to minimize haul distances, thus reducing capital and operating costs.

The Farim and Ponta Chugue onsite and offsite roads will be constructed with crushed rock from existing offsite quarries in Guinea-Bissau, as there are no naturally available materials on site. South of Farim on the opposite side of the river, an offsite gravel road approximately 2 km in length will connect the truck loadout facility on the eastern side of the River Cacheu to the new paved highway leading to Ponta Chugue. An existing 4 km dirt road provides access to the mine site from the river crossing. The final 1.5 km between Farim and the mine site is expected to be widened and resurfaced to provide year-round access. Maintenance, replacement, or upgrades to the existing river crossings at Rio de Banim and Rio de Bunja are expected over the life of mine.

At the Mineral Terminal in Ponta Chugue, the offsite gravel road is approximately 6 km in length and connects the Mineral Terminal facilities to the paved highway from the highway turnoff at Dugal to the Mineral Terminal site. Additional grading will be required during the rainy season to maintain road safety.

The mine roads have been designed to connect the various infrastructure for operation and maintenance. The mine site roads will be newly constructed, widened and resurfaced (at minimum, the roads will be resurfaced with crushed waste rock from an aggregate source in Saltinho, approximately two hours east of Bissau). Three main types of roads will be constructed within the mine site, as follows:

- overburden haul roads capable of supporting a 97-tonne capacity end-dump truck (180 t fully loaded)
- ore haul roads that support a 36-tonne capacity end-dump truck (approximately 72 t fully loaded)
- general use access roads for non-mining traffic.

For overburden and ore haul roads, a minimum road width of 33 m has been adopted. To avoid trafficability issues during the rainy season, a minimum 200 mm thick wearing course of coarse rock is specified. In low-lying wet areas, a base layer of coarse rock is required. A 1 m thickness has been adopted based on the equipment loads. For general use access roads, a reduced width of 10 m can be adopted along with a 150 mm wearing course. To construct the TSF embankments, embankment fill will be sourced from overburden waste. The perimeter access road around the downstream toe of the TSF footprint includes a minimum 150 mm thick wearing course, since the embankments are intended to be constructed only during the dry season (a reduced road design can be adopted).

A new village road around the northern extent of the mine site is required to replace the existing village road that passes directly through the mine site, as local villages west of the mine site will not have access to Farim once the mine site fence is installed and the existing road is decommissioned. The bypass road will be at least 10 m wide (1 m shoulders and two 4 m wide lanes) and will have a 150 mm wearing course for year-round trafficability. The width of the road will accommodate two-way traffic and provide an adequate shoulder for foot traffic. Following mine closure, the reintroduction of a more direct route for the local villages could be considered.

18.3 Existing Infrastructure

18.3.1 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 process plant and Mineral Terminal facility areas. 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 Mineral Terminal in Ponta Chugue, the offsite gravel road is approximately 4 km in length and connects the Mineral Terminal facilities to the paved highway.

18.3.2 Accommodation

An existing 150-person camp will be used to accommodate the administration staff and upper management during the construction and operation of the process plant. At the time of writing, only the sewage treatment works permit is outstanding. Contractors will provide their own accommodation for offsite work.

During the construction of the Mineral Terminal, a local hotel is intended to be used to accommodate management and staff; any additional staff will be accommodated in camps supplied and built by individual contractors.

18.4 Stockpiles

Two topsoil stockpiles have been identified for the project, one directly east of TSF cell 6 and the other directly north of the RWP. The combined storage volume has been calculated at between 0.35 to 0.4 Mm^3. This is considered sufficient also to temporarily store topsoil from the development of WD-1, WD-2, the mine site access roads, and TSF cells 1 and 2. Should additional storage be required, areas within the future cells (cells 5 and 6) could be used as the design approach assumes progressive closure of the inactive TSF cells. The topsoil thickness is estimated at 150 mm.

Topsoil stockpiles are required to temporarily store stripped soils from the TSF, mine site roads, waste dumps and surface water infrastructure. The individual cells for the TSF will be progressively closed; therefore, the total storage requirements for the topsoil stockpile will be reduced. The top 150 mm will be stripped following clearing and grubbing. With limited space on site, the stockpile area required is approximately 100,000 m^2 (based on a maximum stockpile height of 4 m with slopes of 1V:5H). The estimated capacity for the stockpile is 350,000 m^3.

The main topsoil stockpile will be located along the eastern side of proposed TSF cell 6. With progressive closure of the waste dumps and individual TSF cells, future cell areas (cells 5 and 6) can also be used as temporary stockpile locations.

18.5 Site Buildings

The Farim Phosphate Project has incorporated various types of building in this project such as stick-built modular buildings, fabric buildings, pre-engineered buildings, etc. Table 18-1 provides a summary of general building information. Additional details on the building envelopes are provided in the following subsections.

Table 18-1: Building Description

Description	Location	Building Construction	Length (m)	Width (m)	Height (m)	Area (m^2)
Filtration Building	Plant	Stick Build (Main building + 2 Lean-to)	38.5 26.5 15.8	26.5 6.0 7.0	20.5 10.5 4.5	1,277
Administration Building	Plant	Modular Building	42.6	20.3	2.9	866
Security Gatehouse	Plant	Modular Building	3.0	3.0	2.4	9
Plant Ablutions	Plant	Modular Building	6.0	3.0	3.0	18
In-Plant Mess Building	Plant	Modular Building	22.0	9.2	2.9	203
Laboratory	Plant	Modular Building	24	12	2.6	230
Control Room	Plant	Modular Building	8.0	3.0	2.7	24
Emergency Response Vehicle/Equipment Storage	Plant	Fabric Building	12.2	6.1	3.0	74
Stockpile Cover (Truck Loadout)	Truck Loadout	Fabric Building	48.0	38.0	6.0	1,824
Plant Workshop/Warehouse	Plant	Fabric Building	24.4	19.2	8.3	468
Flocculant Building	Plant	Pre-Eng Building	8.8	5.1	5.7	45
Compressor Building	Plant	Pre-Eng Building	10.5	6.0	4.5	63
Mine Security Gatehouse	Mine	Modular Building	3.0	3.0	2.4	9
Concentrate Loadout Office	Truck Loadout	Modular Building	12.0	3.0	2.7	36
Concentrate Area Ablutions	Truck Loadout	Modular Building	6.0	3.0	3.0	18
Concentrate Weighbridge Office	Truck Loadout	Modular Building	12.0	3.0	2.7	36
Concentrate Area Security Gatehouse	Truck Loadout	Modular Building	3.0	3.0	2.4	9
Concentrate Drying Building.	Port	Stick Build	31.0	5.5	12.5	171
Port Administration Building	Port	Modular Building	51.4	10.2	2.9	522
Port Ablutions	Port	Modular Building	6.0	3.0	3.0	18
Port Security Gatehouse	Port	Modular Building	3.0	3.0	2.4	9
Sample Preparation Building	Port	Modular Building	12.0	3.0	2.6	36
Dry Concentrate Shed	Port	Pre-Eng Building	282.0	45.0	16.0	12,690
Wet Concentrate Shed	Port	Pre-Eng Building	78.0	38.4	17.3	2,995

18.5.1 Filtration Building

The filtration building will be constructed of stick-build design. The main body of this building has a dimension of 26.5 m (wide) x 38.5 m (long) x 20.5 m (high) and is designed to support and cover two 30 m long and one 24 m long filter. Two lean-to structures of 6.0 m (wide) x 26.5 m (long) x 10.3 m (high) and 7.0 m (wide) x 15.8 m (long) x 4.5 m (high) are connected to this building, which is located in the plant area.

18.5.2 Concentrate Drying Building

The concentrate drying building will be constructed of stick-build design. This building is half covered at the top with metal cladding and has a dimension of 5.5 m (wide) x 31.0 m (long) x 12.5 m (high).

18.5.3 Flocculant Building

The flocculant building has a dimension of 5.1 m (wide) x 8.8 m (long) x 5.7 m (high). This building is located in the plant area.

18.5.4 Compressor Building

There are two compressor buildings. One is located at the plant area, and one in Mineral Terminal area. Each compressor building is 6.0 m (wide) x 10.5 m (long) x 4.5 m (high).

18.5.5 Dry Concentrate Shed

The dry concentrate shed is the largest pre-engineered building for the project at 45.0 m (wide) x 282 m (long) x 16 m (high). The building will be located at the Mineral Terminal area. The dry concentrate shed has three areas: the storage area, truck area, and bin area.

18.5.6 Wet Concentrate Shed

The wet concentrate shed is a pre-engineered building with dimensions of 38.4 m (wide) x 78.0 m (long) x 17.3 m (high) located in the Mineral Terminal area. One side of this building is on-covered to enable the access of trucks and site vehicles.

18.5.7 Marine control Center

The marine control center is a pre-engineered building at the Mineral Terminal area with dimensions of 30.0 m (wide) x 30 m (long) x 13 m (high).

18.5.8 Emergency Response Vehicle / Equipment Storage

The emergency response vehicle / equipment storage is a fabric building with dimensions of 6.1 m (wide) x 12.2 m (long) x 3.0 m (high) located in the plant area. This fabric building is supported by one single-level 12 m (40 ft) sea-can container on each side.

18.5.9 Stockpile Cover (Truck Loadout Building)

The stockpile cover (truck loadout building) is a 38.0 m (wide) x 48.0 m (long) x 6.0 m (high) fabric building located at the east side of River Cacheu. This fabric building is supported by eight two-level 12 m (40 ft) sea-can containers on each side.

18.5.10 Workshop/Warehouse

The workshop/warehouse is a fabric building 19.2 m (wide) x 24.4 m (long) x 8.3 m (high) located in the plant area. This fabric building is supported by two single-level 12 m (40 ft) sea-can containers on each side.

18.5.11 Administrative and Offices

The project will construct two administrative buildings, three plant ablution buildings, one in-plant mess building, one loadout office, one weight bridge office of modular building design. All modular buildings will be single-story.

The plant area will include an administrative building with dimensions of 20.3 m (wide) x 42.6 m (long), a plant ablution building with dimensions of 3.0 m (wide) x 6.0 m (long), and an in-plant mess building with dimensions of 9.2 m (wide) x

22.0 m (long). The administrative building will include offices, washrooms, working stations. The plant ablution building includes men's, women's, and unisex/barrier-free washrooms. The in-plant mess building includes a kitchen and dining rooms.

The truck loadout area will include an ablution building with dimensions of 3.0 m (wide) x 6.0 m (long), loadout office with dimensions of 3.0 m (wide) x 12.0 m (long), and weighbridge office with dimensions of 3.0 m (wide) x 12.0 m (long).

The Mineral Terminal area will include an administrative building and ablutions building with dimensions of 10.2 m (wide) x 51.4 m (long), and 3.0 m (wide) x 6.0 m (long), respectively. The administrative building will include offices, washrooms, and working stations. The plant ablution building includes men's, women's, and unisex/barrier-free washrooms. The in-plant mess building includes the kitchen and dining rooms.

18.5.12 Security Gatehouse

Four security gatehouses are included, one for each plant, mine, product loadout, and Mineral Terminal areas. Each security gatehouse will be a small, prefabricated modular building with dimensions of 3.0 m (wide) x 3.0 m (long) single-story building with a single-boom gate. Site inductions for visitors and new employees can be conducted at this point.

18.5.13 Control Room

The control room is a single-story modular building with dimensions of 3.0 m (wide) x 8.0 m (long). This building list is located at the Mineral Terminal area.

18.5.14 Sample Preparation Building

The sample preparation building is a single-story modular building with dimensions of 3.0 m (wide) x 12 m (long). This building list is located at the Mineral Terminal area.

18.6 Geotechnical Assessment

The mine site lies within an extensive sedimentary basin on low-lying ground close to tidal rivers that are bordered with mangroves, mudflats, and salt flood plains. The ground conditions reflect the geological and topographical setting with normal to lightly consolidated and primarily cohesive alluvial deposits encountered close to the rivers and more over-consolidated deposits at a greater distance from the rivers.

Along the southern perimeter of the South pit and the truck loadout facility, the ground consists of very soft alluvial clays to depths up to 14 m. The ground conditions at the sites of the proposed tailings storage facility and process plant comprise more over-consolidated deposits. Near-surface soils are predominantly cohesive and the safe bearing pressures afforded by these soils will be 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 truck loadout facility 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, so piling and subgrade remediation measures will be required.

Selected sedimentary soils will be suitable for low permeability and general fill. Local sources of drainage sand/gravel, road base, and concrete aggregate have not been identified on site. The nearest known hard rock quarry is approximately 150 km from the site.

Foundation parameters for the TSF are summarized in Table 18-2.

Table 18-2: TSF Foundation Parameters

Material	Unit Weight (kN/m^3)	Cohesion (kPa)	Friction Angle (°)	Permeability (m/s)
Clayey Silty Sand (0-5 mbgl)	20	0	28	$1x10^{-7}$
Clayey Silty Sand (5-10 mbgl)	20	0	30	$1x10^{-7}$
Sandy to Silty Clay	20	0	27	$1x10^{-6}$

Note: "mbgl" is meters below ground level.

Site investigations along the boundary for the South pit flood bund were reported as very poor in previous site investigations. The soils comprised very soft to soft saturated clay with low strength and significant settlement potential when loaded. The soils consisted of silty clay, overlying clayey silty sand and sand. The silty clay has high plasticity and recorded standard penetration test (SPT) N values of less than one.

A CPT investigation is recommended at detailed design stage to identify the material behavior and undrained shear strengths. Parameters used in the feasibility study are presented in Table 18-3. The water level was recorded between 0.0 and 6.7 mbgl and will be influenced by the tide.

Table 18-3: Flood Bund Foundation Parameters

Material	Unit Weight (kN/m^3)	Cohesion (kPa)	Friction Angle (°)
Flood Bund Foundation	19	0	25

At waste dump no. 1 (WD-1), the general stratigraphy consists of interbedded silty clay and clayey silty sand on the west side. The east side consists of interbedded silty clayey sand, clayey sandy silt, clay and sand to a depth of 16.5 m with sand and trace silt to 25 mbgl. At waste dump no. 2 (WD-2), the general stratigraphy consists primarily of a silty clayey sand. During prior investigations, bedrock was not encountered at maximum drillhole depths of approximately 25 m below ground level (mbgl). Depths to groundwater were reported at 5.4 and 3.0 mbgl, respectively, for WD-1 and WD-2.

Based on testing completed to date, Table 18-4 summarizes the strength and permeability parameters recommended for the waste dump foundations.

Table 18-4: Waste Dump Foundation Parameters

Material	Unit Weight (kN/m^3)	Cohesion (kPa)	Friction Angle (°)	Permeability (m/s)
Clays and Silts	18	0	27	$1x10^{-7}$
Sands	18	0	30	$1x10^{-6}$

18.6.1 Laboratory

The laboratory consists of three 12 m (40 ft) modules which are compiled into an L-shaped single-story modular building, is used to test metallurgical samples. The laboratory also includes a 12 m x 12 m veranda, which makes the total area of the laboratory 230 m^2.

18.7 Tailings and Storage Facilities

The tailings storage facility (TSF) will be progressively developed over the life of mine utilizing seven individual cells with confining embankments tying into natural ground. Each embankment will be constructed from waste overburden excavated from either the south or North pit and delivered to the embankment footprints by the mining fleet. The seven-cell configuration provides capacity for the 25-year life-of-mine tailings slimes production estimated at 8.85 million tonnes (Mt). 2.07 Mt of coarse rejects will also be produced over the mine life; however, the coarse rejects will be placed within the planned open pits as part of the waste management infill. The TSF storage capacity is sufficient for storage of tailings solids, an operational pond based on average climatic conditions, production conditions, as well as storage of the environmental design flood (EDF).

Supernatant and direct rainfall water will be reclaimed from active TSF cells (cell receiving tailings) as well as inactive cells (previously filled with tailings) for reuse in the process plant. Inactive cells will continue to be dewatered to prepare for and during progressive closure. Progressive closure for each filled cell is assumed to commence one year after filling or during the next dry season months, with each cell closure estimated to take up to two years. The reclaimed water will be pumped from the cells into the RWP located along the eastern side of the TSF Cell 5. The RWP will provide additional treatment (settling of suspended solids) before being pumped to the process plant for reuse. The RWP will be constructed in two phases, Phase 1 in Year 0 has a storage volume of 141,000 m^3 and Phase 2 in Year 1 will increase the total storage volume to 286,000 m^3. The RWP total storage volume is approximately three months of reclaim water at the plant rate of 130 m^3/h. The RWP has been oversized to allow progressive closure of the individual TSF cells as they reach capacity; excess water can be removed via pumping to the RWP.

To limit seepage from the TSF and to avoid construction of an expensive erosion protection barrier along the upstream face of each embankment, a 1.5 mm thick high-density polyethylene (HDPE) geomembrane liner will be installed.

18.7.1 Site Characteristics

The selected location for developing the TSF is situated west of the process plant and South pit and south of the North pit. The topography of the TSF area is broken into three small local drainage zones, with a maximum elevation of 30 mamsl in the north portion of the proposed layout, and a minimum elevation of 5 mamsl along the west, south and east sides. The drainage areas are to the northwest, centrally north to south and to the east.

The groundwater rest water level in this area ranges from 2.8 m to 13.6 m below ground surface based on monitoring boreholes installed at the TSF. There is a shallow aquifer associated with the overburden and deeper confined aquifer in the calcareous sandy limestones at –30 m RL in the TSF area.

18.7.2 Tailings Slimes Characterization

Physical and geochemical tailings testing programs were carried out in 2012 (Golder, 2012), 2015 (KP Perth), 2016 (KP, 2018) and 2017 (KP, 2017a), (KP, 2017b), (KP, 2017c). The physical properties of the tailings slimes, as reported from the 2015 and 2017 testwork, are as follows:

- Historic testwork results indicate the tailings comprise 1% sand, 31.5% silt and 67.5% clay.
- Atterberg limits testing indicates that the tailings slimes are a high plasticity with a liquid limit of 108%, a plastic limit of 19%, and a plastic Index of 89%.
- The tailings slimes have a P_{80} ranging between 980 to 40 µm, according to 2017 testwork.
- The specific gravity (SG) of the tailings solids is 3.3.
- The tailings have an average in-situ density of 0.3 t/m^3.
- The material is a high plasticity silty clay with trace sand and would be classified as “CH” in accordance with the Unified Soil Classification System.

The tailings will settle very slowly and only release a small quantity of water, based on the tested percent solids. Initial densities will therefore be low. Tailings settling testwork was carried out on slurried tailings with percent solids by mass (% m) ranging from 10% m to 25% m, tested under undrained, drained and air-dried conditions. From the 2017 project update, an average in-situ density of 0.3 t/m^3 was selected based on the anticipated operating conditions of the TSF. The results suggest the solid's in-situ dry density increases as the slurry percent solids is increased. In 2017, the tailings slurry percent solids by mass from the process plant was 17% m. For 2022, this was reduced to 15% m; however, the in-situ density of 0.3 t/m^3 was retained.

The geochemical properties of the tailings are reported as follows:

- The tailings slimes are classified as non-acid-generating (NAG) as the alkalinity loading over time is sufficient to buffer any acid potential (KP, 2018).
- Metal leaching testwork suggests the tailings are prone to leaching cadmium and nickel at elevated levels above target receiving water quality standards for the River Cacheu, but below WHO (2017) guidelines for drinking water quality and IFC (2007) mine effluent guidelines.
- Radionuclide leaching potential testwork indicated that the tailings are likely to exhibit leaching of radionuclides from the tailings solids (KP, 2018). Therefore, the TSF must be designed and operated to fully contain the tailings solids and water. The leeching potential indicates an engineered containment and capping system is required with progressive closure to be adopted to reduce exposure once each cell reaches capacity.

18.7.3 Tailings Design Parameters

The design parameters for the tailings storage are summarized in Table 18-5.

Table 18-5: Tailings Design Parameters

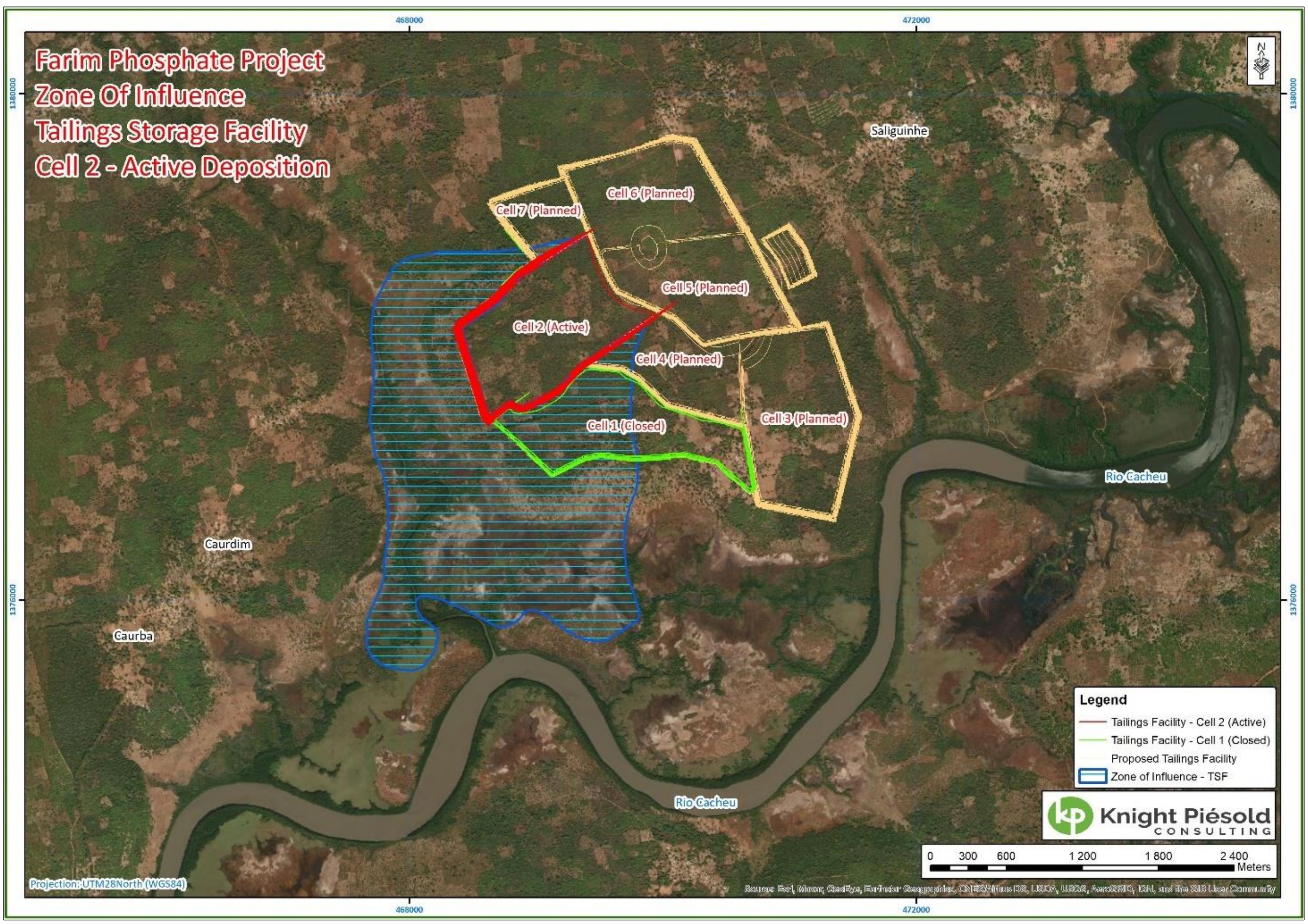
Item	Design Criteria	Reference
Codes and Standards	Canadian Dam Association (CDA) Dam Safety Guidelines and Technical Bulletin	CDA, 2013 and 2019
	Global Industry Standard on Tailings Management	GISTM, 2020
	A Guide to Management of Tailings Facilities	MAC, 2022
Design Tonnage	8.85 Mt	Farim 2022 Mine Schedule
Average In-Situ Density	0.3 t/m^3	SGS & KP, 2017 testwork
Tailings Storage Volume	29.5 Mm^3	KP Calculated
Life of Mine	25 years	Farim 2022 Mine Schedule
Tailings Output	Years 1 to 6: 281,000 dry tonnes per annum Year 7: 291,000 dry tonnes per annum Years 8-25: 382,000 dry tonnes per annum	Farim 2022 Mine Schedule
Tailings Beach Slope	Negligible – assume clays will settle flat	KP, 2017
Configuration	Seven cells	KP, 2022
Tailings Slurry	15% solids by mass	Farim 2022 Mine Schedule
TSF Dam Classification	High	KP Calculated
Embankment Materials	Constructed from pit waste overburden delivered by mining fleet	KP, 2022
	Blanket drain sourced from offsite quarry	
	HDPE geomembrane liner across upstream slope	KP, 2022
Water Management	Non-contact rainfall runoff diverted away from TSF cells Supernatant water pumped to the RWP	
Design Flood Events	EDF: 1-in-100-year, 24-hour storm event – 463 mm IDF: 1/3 between 1:1,000 and PMP – 985 mm	
Freeboard	Tailings maximum level – 2.5 m below crest Maximum operating water level – 1.5 m below crest Environmental design flood – 1.0 m below crest Emergency spillway – 1.0 m below crest Inflow design flood maximum level – 0.5 m below crest	

Return Water Pond	Maximum pumped rate to process plant – 130 m^3/hour	Ausenco, 2022

18.7.4 Tailings Storage Facility Design

The TSF will consist of seven cells located approximately 4,000 m west of the proposed process plant location. To spread construction costs over the life of mine, the TSF will be progressively developed with new cells being constructed approximately one year prior to the active cell reaching capacity. The actual timing of cell construction will be dependent on production and the timing of the filling schedule with the dry season as construction is expected to only occur during the drier months, from November to May. The TSF is designed to accommodate the life-of-mine design tonnage of tailings slimes, the operational pond, and rainfall from the 24-hour EDF. Each cell includes an emergency spillway, 10 m wide and 1 m deep, to safely pass inflows greater than the EDF up to the inflow design flood (IDF) 24-hour event. The EDF and IDF events have been defined according to a dam classification of "high" based on the potential loss of life of 10 or fewer and possible impacts to the environment. The zone of influence map for a theoretical failure of the cell 2 embankment is shown in Figure 18-4.

Figure 18-4: TSF Cell 2 Embankment Failure Zone of Influence



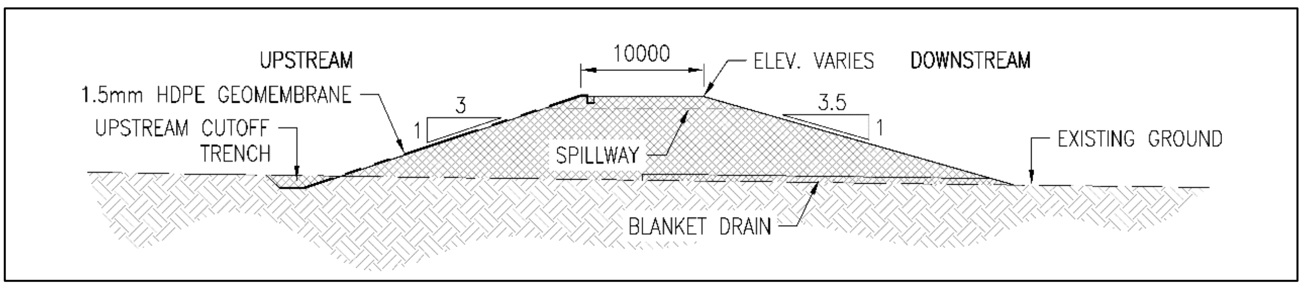
Source: Knight Piesold, 2023

The embankment design incorporates a zoned embankment with an upstream cutoff trench that also acts as the anchor trench for the embankment upstream slope HDPE geomembrane liner. The geometry for the embankments provides appropriate room for the tailings slurry pipeline, reclaim water pipeline, medium-sized construction equipment, and a safety berm along the downstream crest edge while meeting minimum safety requirements for stability. Key design parameters are included in Table 18-5, and the typical embankment section as shown in Figure 18-6.

Table 18-6: Embankment Design Parameters

Item	Design Parameter
Upstream Slope	1V:3H with cutoff trench
Downstream Slope	1V:3.5H – slope vegetated to reduce erosion potential
Crest Width	10 m
Wearing Course	6 m wide, 150 mm thick
HDPE Liner	Upstream slope – cells 1 to 7
Blanket Drain	500 mm thick from crest centerline to downstream toe

Figure 18-5: TSF Cell Typical Embankment Section

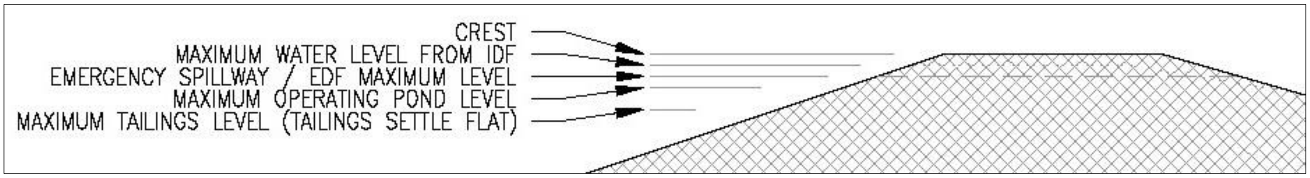


Source: Knight Piesold, 2023

Tailings will be discharged into the active cell by sub-aerial deposition methods through spigots either located along the embankment crest or from specific open-ended discharge points around basin perimeter. The active tailings deposition area will be rotated along the embankment and perimeter of the cells to maximize tailings storage and develop a consistent tailings level within the cell. The tailings slimes are assumed to settle flat with negligible beach slope developed; therefore, the supernatant water will be located above the tailings as each cell fills.

Due to the low-density tailings and negligible beach slope of the tailings slimes, no formal beach will be exposed during operation, except during the dry season where evaporation exceeds water inputs into each cell. Since the tailings are assumed to settle flat, a large freeboard will be required to meet design requirements for managing the operational pond, for managing the EDF (which should be safely stored within the TSF cell), and for either storing or safely passing the inflow design flood. Figure 18-6 identifies the maximum levels for passing the IDF event through the spillway (500 mm below crest), the spillway invert (1000 mm below crest), the maximum operating pond level (1500 mm below crest), and the maximum tailings elevation, assuming a flat surface (2,500 mm below crest).

Figure 18-6: TSF Cell Typical Embankment Freeboard Configuration



Source: Knight Piesold, 2023

Supernatant water will be removed from the active cells via a skid-mounted pump situated either on the embankment crest or on a temporary causeway inside the basin. The portable pump system will allow operations to adjust the intake location to fully utilize available water for reclaim to the process plant via an RWP used for further treatment. The skid-mounted pump will be fitted with a 20 m long linatex intake hose suitable for floating on top of the water with a suction bend to intake near surface water. This pump configuration will allow reclaim to remain active when each cell nears capacity and the pond is shallow, with depths less than 1000 mm. Water pumped from individual cells will be discharged into the RWP for further treatment (settling) before being pumped to the process plant. The pumps have been sized to provide a maximum flow rate of 130 m^3/hour as controlled by the process plant water demand. The routing of the reclaim piping from the TSF cells to the RWP will follow the shortest route between the cell and RWP while avoiding high points that exceed the pump specifications. To meet water pumping demands, a 200 mm, DR17 HDPE pipe has been specified to meet flow, velocity, and pump requirements.

The RWP will operate as a large clarification pond, to allow TSF active and inactive cells to be operated with minimal pond volumes while providing additional clarification time within the RWP before reuse at the plant. The RWP will be constructed in two stages to spread construction costs over several years. Stage 1 will be constructed during Year 0, providing a maximum storage capacity of 141,000 m^3, while Stage 2 will be constructed in Year 1 increasing the total capacity to 286,000 m^3 approximately three months of reclaim process water demand. The RWP has been sized to maximize water removal from the TSF cells while targeting up to 210 m^3/h reclaim back to the process plant when water from the environmental control dam is added to the RWP. The RWP will be lined with a 1.5 mm HDPE geomembrane liner to minimize infiltration losses. Water inputs to the RWP include the TSF reclaim, environmental control dam (Year 3 onwards) and direct rainfall only as the RWP will be constructed with a perimeter berm raised above natural ground. Dewatering the TSF cells will be given priority over the SCD1 when the RWP nears capacity.

The pumping system for the RWP to the process plant includes an operational submersible pump and standby pump. The reclaim water will be pumped through a 200 mm, DR17 HDPE pipe with velocities maintained below 2 m/s. The pipeline will be routed along the tailings and decant return corridor which travels alongside the mine site access roads between the RWP and plant site. The corridor, which acts as a spill containment system, will be excavated in natural ground to form a channel 1 m deep with a 2 m wide base. The excavated material will be used to form a bund on either side of the corridor.

Each TSF cell will include an emergency spillway as mentioned above. For cells, 1, 2, 3, 4 and 5, the spillway will be constructed in natural ground routing any potential flows away from the embankments. Cells 6 and 7 spillways will be constructed within the embankments or an option to route into cells 2 and 5 could be considered depending on the current state of closure for those cells.

The TSF cell embankments will be constructed in stages over the life of the facility to suit the storage capacity requirement. The cell embankment crest levels and design storage capacity at each cell are summarized in Table 18-7.

Table 18-7: TSF Cell Tailings Storage

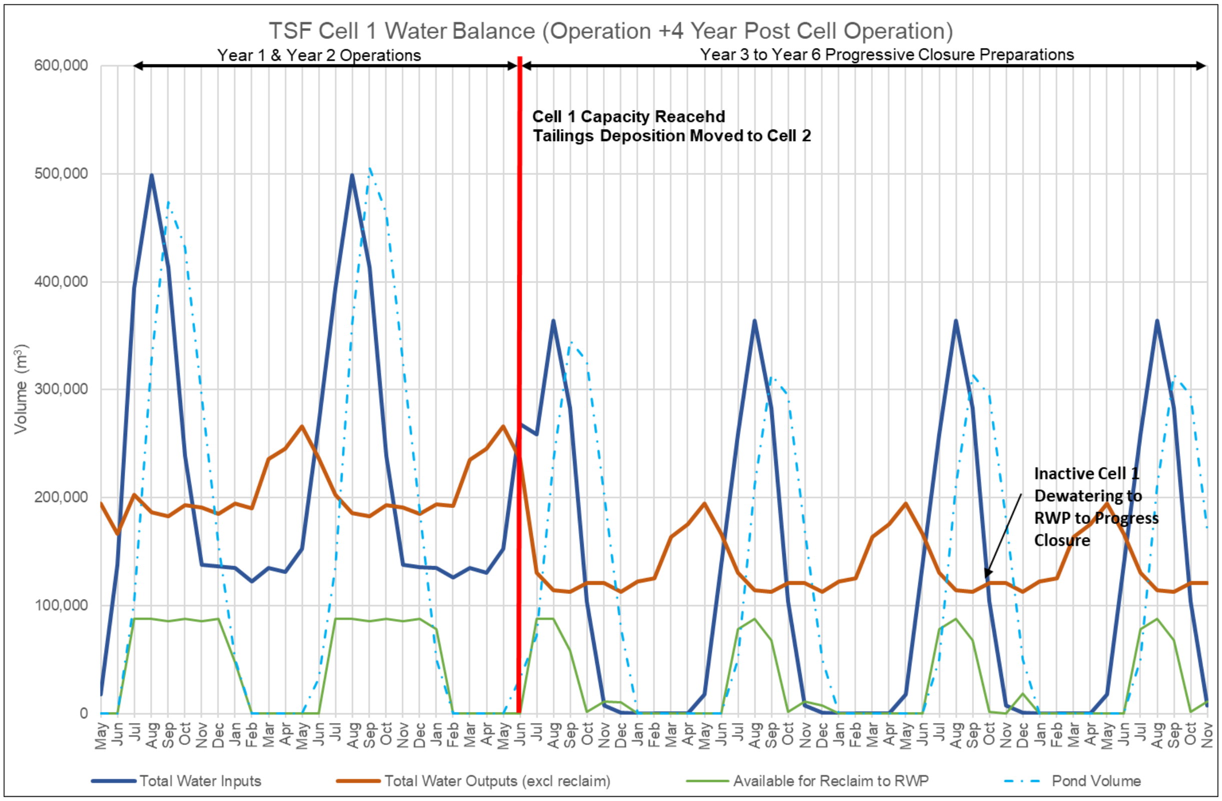
Cell	Embankment Crest (mRL)	Embankment Volume (m^3)	Maximum Tailings Elevation (mRL)	Maximum Operating Pond Elevation (mRL)	Basin Surface Area (ha)	Tailings Storage Volume (Mm^3)	Capacity (Mt)	Years
1	13.0	622,000	10.5	11.5	114.2	1.907	0.572	2.04
2	23.0	2,199,000	20.5	21.5	102.58	6.546	1.964	6.31
3	25.5	3,257,000	23.0	24.0	96.75	8.064	2.419	6.33
4	23.0	628,000	20.5	21.5	30.175	1.836	0.551	1.44
5	33.0	2,075,000	30.5	31.5	77.085	4.419	1.326	3.47
6	33.0	1,839,400	30.5	31.5	77.03	4.570	1.371	3.59
7	23.0	876,000	20.5	21.5	31.33	2.311	0.693	1.82
Total					529.15	29.653	8.896	25.00

The design comprises a multi-zoned embankment with an HDPE geomembrane liner across the upstream slope. Construction materials include earthfill (utilizing waste overburden material delivered from the open pits by the mining fleet) and a fine filter produced from crushed quarry rock, because there is no natural source available on site. Due to the low strength soils on site and lack of available rock for constructing the embankments, the individual cells will not be raised and are limited to a maximum height of 20 m using the current configuration. The fine filter material will be used to construct a horizontal blanket drain. The blanket drain, which lies below the crest centerline and extends to the downstream toe, has been included as a preventative measure in addition to the upstream slope HDPE liner to prevent excess pore pressure buildup within the low permeability embankment.

To minimize external water sources (rainfall runoff) from entering each cell, small diversion channels will be constructed above cells 1, 2, 3, and 4. This will allow the TSF to maintain a neutral water balance, provided water is reclaimed for processing. Once tailings deposition has ceased in a specific cell, the water balance indicates water will pond seasonally from rainfall and will require pumping until the 1.5 m thick closure cover has been fully established. The average water balance for cell 1 is identified in Figure 18-7, which shows reclaim is not sustained year-round, but is dependent on the time of year. During the dry season where evaporation exceeds rainfall, not enough water is released from the tailings for reclaim.

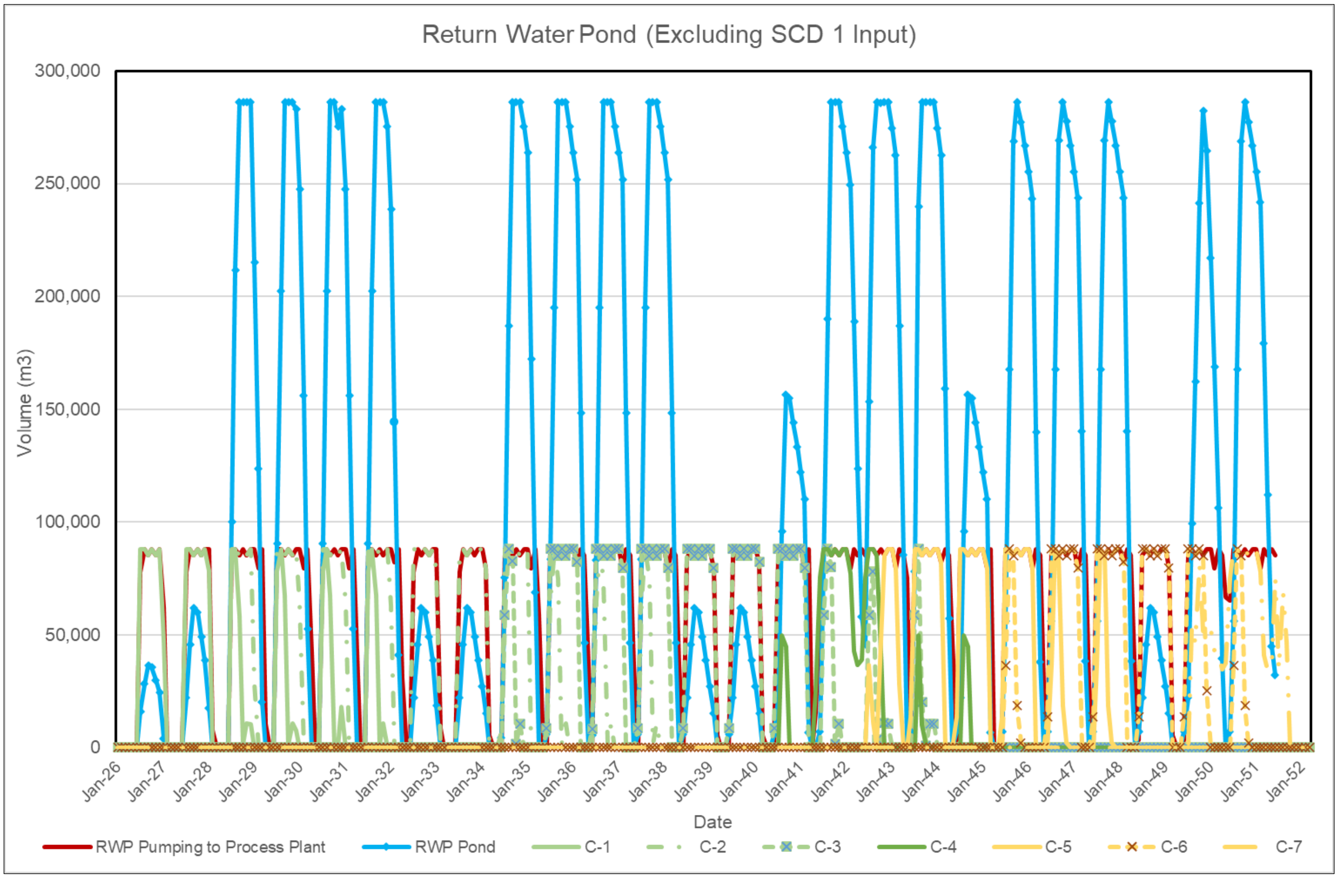
TSF cell dewatering has been optimized to maximize available water for return to the process plant. However, even with dewatering of active and inactive cells, the RWP cannot sustain the maximum return rate to the process plant as shown in Figure 18-8. The figure does not include water pumped from the environmental control dam into the RWP, which is discussed below in the surface water management section. When the RWP cannot meet the process plant requirements, water will be drawn from either pit dewatering or from the SCD 1.

Figure 18-7: TSF Cell 1 Average Water Balance



Source: Knight Piesold, 2023

Figure 18-8: Return Water Pond Average Water Balance Inputs and Output



Source: Knight Piesold, 2023

18.8 Waste Storage Facilities

Over 440 million bank cubic meters (Mbm^3) of waste overburden is identified in the mining schedule, of which 97.4 Mbm^3 comes from the South pit and the remainder from the North pit. According to geochemistry testwork previously completed, the waste overburden from the pits is either classified as non-acid-generating (NAG) or potentially acid-generating (PAG). The waste allocation of the NAG and PAG volumes from the South pit is based on a ratio of 67% NAG and 33% PAG; the testing of the North pit suggests a ratio of 87% NAG and 13% PAG. Mining of the North pit is not an issue with regard to waste storage, as material can either be placed into the South pit or North pit. The South pit, however, requires above-ground storage during Years 0, and 1 before pit infill can commence with small amounts in year 1 and fully in Year 2. According to the mine plan, NAG and PAG will be mined during this period, totaling 14.473 Mbm^3, of which an estimated 0.883 Mbm^3 is classified as PAG material.

Two waste overburden dumps, identified as waste dump no. 1 (WD-1) and waste dump no. 2 (WD-2), are planned outside of the pits, as shown in Figure 18-1. WD-1 will be used to store the PAG material generated at the South pit up to the end of Year 1. Prior to placement of PAG waste, the foundation of WD-1 requires clearing and grubbing, topsoil removal, contouring to promote runoff towards diversion channel 03 (DC03) which drains into the environmental control dam located directly west of WD-1. The area where PAG waste will be placed requires a 500 mm thick basal liner. For WD-1, an area of approximately 16.5 ha has been identified as sufficient to safely place and permanently store the estimated volume of PAG material. A suitable material has been identified during previous investigations that exhibits a very low permeability in the range of $1x10^{-9}$ to $1x10^{-10}$ m/s which is ideal for this application. Once the basal liner is established, truck traffic should be limited. Therefore, the initial placement of waste overburden shall be dumped and pushed over the basal liner.

PAG waste will be encapsulated by a minimum thickness of 6 m of NAG waste overburden (parallel to slope). The estimated maximum elevation of the PAG zone is 28 mamsl. This will be developed with a maximum overall slope similar to the NAG waste outer slope at 1V:5H. Based on stability requirements, WD-1 is limited to a 1V:5H overall side slope and a maximum elevation of 50 mamsl resulting in an average height of 37 m. An internal collection channel has been incorporated into the PAG foundation preparation to separate seepage from the PAG and NAG zones. This collection channel will drain into drainage channel no. 3 before discharging into the environmental control dam located west of the WD-1 PAG area.

WD-2 will only be used to permanently store NAG waste; therefore, the foundation excludes the basal liner requirement. The overall slope of WD-2 will not exceed 1V:5H to meet long-term stability requirements, while the elevation has been limited to 45 mamsl, resulting in an average height of 32 m.

To meet stability requirements, the overall slope of WD-1 and WD-2 will not exceed 1V:5H. Intermediate bench heights of 5 m at slopes of 1V:3.5H have been specified. A minimum setback of 8 m for each bench has also been included. The resulting storage volume for each waste dump using the described configuration is included in Table 18-8.

Table 18-8: Waste Dump Storage Capacities

Facility	Area (m^2)	Final Elevation (mRL)	Total Placed Volume (Mm^3)	NAG Placed Volume (Mm^3)	PAG Placed Volume (Mm^3)
WD-1	594,000	50	10.80	9.64	1.16
WD-2	553,500	45	8.50	8.50	0
Total	1,196,000		19.30	18.14	1.16

A bulking factor of 1.27 has been used for calculating the excavated and placed waste overburden material.

18.9 Power and Electrical

18.9.1 Power Supply

Power supply to the plant site and the Ponta Chugue Mineral Terminal facilities will be from an onsite hybrid power plant with solar and diesel power. These hybrid plants are commonly used in the region for isolated mine sites and generally can be managed through a build-own-operate-maintain (BOOM) model offered by third party power generation contractors. The current assumption is that 36% of power generation will be supplied by solar power over the life-of-mine.

The configuration of the onsite hybrid power plant at Farim is as follows:

- four 1.2 MW prime-rated 11 kV generators (3 duty, 1 standby)
- 11 kV switchroom
- solar cells and related electrical components (specifications TBD during detailed design).

The configuration for the onsite hybrid power plant at the Mineral Terminal is as follows:

- three 0.5 MW prime-rated 0.4 kV generators (2 duty, 1 standby)
- direct feed to the low-voltage switchroom
- Solar cells and related electrical components (specifications TBD during detailed design).

18.9.2 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 centers (MCCs) via four separate 11 kV / 433 V distribution transformers. The low-voltage switchrooms in the process plant area will house one low-voltage MCC each, while the plant services and reagents switchroom will hold two. These will be fed from an 11 kV feeder via an underground power cable. Power supply to the meter supply will be fed from an 11 kV feeder via an 11 kV underground powerline. Outdoor control panels and distribution boards have been allowed to provide plant lighting, small power distribution, and uninterruptible power supply (UPS) distribution.

18.9.3 Installed Load and Maximum Demand

The installed load and maximum demand are shown in Table 18-9 and Table 18-10 for the Farim plant and Ponta Chugue Mineral Terminal facilities, respectively.

Table 18-9: Farim Plant Power Demand

Plant Installed Load	Plant Maximum Demand	Plant Average Continuous Load
7,844 kW	4,911 kW	3,877 kW

Table 18-10: Ponta Chugue Port Power Demand

Port Installed Load	Port Maximum Demand	Port Average Continuous Load
2,265 kW	1,840 kW	1,282 kW

18.9.4 Electrical Buildings

The main high-voltage switchboard and all other plant electrical buildings will be prefabricated buildings, as follows:

- three low-voltage switchrooms for the beneficiation plant
- one low-voltage switchroom at the Mineral Terminal
- plant and Mineral Terminal control rooms.

These electrical buildings will have air conditioners and will be sealed to prevent dust ingress.

18.9.5 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.

18.9.6 11 kV Switchboards

The 11 kV switchboards will have a fully withdrawable design complete with protection, metering, and earthing facilities. The design fault level and circuit breaker ratings adopted are as follows:

- 11 kV switchboard busbar 1,250 A, 40 kA at 3 seconds
- 11 kV circuit breakers 630 A.

Protection will be provided by microprocessor-based protection relays.

18.9.7 Electronic Variable Speed Drives and Soft Starters

Low-voltage variable speed drive (VSD) units and soft starter ratings range from 2.2 kW up to 315 kW. These are mounted on the floor or wall (depending on size) along the internal wall of the low-voltage substation.

18.9.8 400 V Motor Control Center

The low-voltage MCCs will be double-sided (back-to-back) and housed in the low-voltage switchroom. Construction of all MCCs will have Form 4 segregation, Type 2 coordination. The MCC starters will be demountable and the main incoming circuit breakers will have a withdrawable design complete with protection. All motor starters will be equipped with smart overload relays. The low-voltage MCC's will supply power to the low-voltage motors, variable speed drives, and distribution boards.

18.9.9 Earth Fault Protection

Earth leakage protection will be applied to circuits with general purpose outlets (GPOs) and for lighting circuits.

18.9.10 Fire Protection

The high-voltage switchroom, low-voltage switchroom, and the plant and Mineral Terminal control rooms will be provided with fire detection systems. Signals from the fire detection system will be wired to the respective fire indication panel in the switchrooms and all signals will be monitored by a master fire detection panel in the security / emergency services control room in the corresponding administration buildings. Each fire indication panel will be wired to a local siren with a beacon to warn staff in the event of fire detection. The same fire and smoke activation alarm signals detected by the fire detection system will be monitored in the plant and Mineral Terminal control rooms.

18.9.11 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.

18.9.12 Cables

Direct buried cables will be provided with armoring. Cables up to 16 mm^2 will be insulated with polyvinyl chloride (PVC) and bigger cables will be insulated with cross-linked polyethylene (XLPE). VSD cables will be multiple core three-phase and three-earth cables symmetrically laid out within an overall shielded cable. Cables within the plant and Mineral Terminal areas will be installed above ground on cable ladders and will follow the mechanical pipe racks wherever possible.

18.9.13 Lighting

All lighting around the beneficiation plant and Mineral Terminal is designed in a fit-for-purpose manner to suit the operational requirements for each area.

18.9.14 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 Mineral Terminal building structures. Plant and Mineral Terminal 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.

18.10 Fuel

The diesel price used for the project is $0.86/L. This is based on the JP Morgan ICE Gasoil (diesel) swap curve through 2025, as of September 2022 with a decrease in diesel prices continuing until 2026. This aligns with the timeline for the start of the project and the phosphate rock pricing used in the financial analysis. Diesel fuel will be shipped to Ponta Chugue. From the Port of Ponte Chugue, diesel fuel will be transported via trucks to diesel fuel storage tanks at Farim. The estimated diesel fuel usage will be approximately 3 million liters per month (ML/month). The diesel fuel will be required primarily for the mining fleet, rotary dryer, and power generation plants.

The diesel storage tanks will be above ground, designed per the American Petroleum Institute standard (API 650), and inside a secondary containment berm. Both facilities will be equipped with fuel dispensing systems located on site.

18.11 Water Supply and Management

18.11.1 Process Water Management

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 operating, one on standby) will deliver process water to users distributed throughout the plant. A process water tank 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.

18.11.2 Surface Water Management

Surface water management in the project area and the supply of water for processing are critical aspects of the design. Five categories of water have been identified on site as follows:

- Undisturbed water (U)– runoff from undisturbed catchments
- Clean water (CW) – water from pit dewatering bores
- Contact clean water (CC) – runoff from mining disturbed catchment areas with some sediment pick-up; sources include inert waste and pit dewatering sumps
- Contact dirty water (CD) – runoff from mining disturbed catchment areas with potential for contamination; sources include mineralized waste
- Process water (P) – water that has passed through the process or come into contact with process water; sources include TSF decant water and process plant runoff.

Surface water management designs for the open pits, waste storage facilities, and tailings storage facility have been prepared to reflect the phasing of development of each component, adopting appropriate water management strategies for each of the identified water classes. The surface water management infrastructure includes surface water diversion channels, sediment and environmental control dams, an RWP, various flood protection bunds, and sumps within the open pits.

18.11.2.1 River Diversion and Flood Protection

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 runoff prior to discharge by means of sediment control dams (SCDs). 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 program should be implemented to further reduce the sediment load in runoff.

A combination of PAG and NAG waste will be stored within WD-1, while only NAG waste will be stored in WD-2. The PAG waste to be stored in the northern region of WD-1 will be placed within a dedicated cell with a basal liner during the initial years of the project's development. The PAG waste will subsequently be encapsulated using NAG waste once all PAG waste to be stored on the surface has been generated. Runoff water from the PAG area of WD-1 (Class CD water) will be intercepted using surface water diversion channels and directed to the HDPE-lined environmental control dam. This water will be pumped to the process plant for re-use or to the treatment plant for polishing before being released to the River Cacheu.

Runoff from the NAG area of WD-1 (Class CC) will report to SCD1, while runoff from the inert WD-2 (Class CC) will report to SCD2. Runoff from inactive or rehabilitated areas will be diverted around the sediment and environmental control dams to maximize the efficiency of those structures.

Other sources of water that report to the SCDs include clean water generated from pit dewatering boreholes (Class CW) and rainfall runoff collected within sumps in the internal pit area. Such runoff is classified as contact clean water (Class CC) and pumped to SCD1, prior to discharge. Runoff from the surrounding area is intercepted by perimeter diversion structures.

Sediment within the two sediment control dams (SCD1 and SCD2) will be treated with a flocculant to enable settling of suspended solids. Decant water from the dams (Class CW) will be utilized as make-up water for the plant or released to the River Cacheu.

The SCDs are designed to not spill to the environment at a reliability of 98%, while the environmental control dam is designed not to spill at a reliability of 99%.

18.11.2.2 Sediment Control

The surface water diversion dams and channels and flood protection bunds are designed to (1) divert runoff from the waste overburden dumps to the respective sediment / environmental control dams (Class CC and CD) and (2) divert runoff from undisturbed / rehabilitated catchments (Class U and CW) away from site infrastructure and to the nearest natural water course.

The Rio de Cavaras Marinhos flows through the site in a southeasterly direction. This river encroaches on the proposed footprints of the western limb of the north open pit, backwater dam no. 1, the environmental control dam, and SCD1. The river will initially be diverted via a diversion channel (backwater dam no. 1 spillway) constructed between the tailings storage facility and the environmental control dam / SCD1. Backwater dam no. 1, which separates the environmental control dam from the north open pit, will form the headworks of the initial diversion.

The North pit will be mined between Years 7 and 25, with mining starting on the southern side of the pit and progressing in a north-northeasterly direction. The pit will be progressively backfilled as mining advances. The western limb of the pit, the portion of the pit upon which Rio de Cavaras Marinhos encroaches, will be mined between Years 18 and 25. The second phase of the diversion will entail the construction of the diversion dam and river Diversion channel that diverts the river to the River Cacheu via the backfilled eastern portion of the North pit.

A second, smaller freshwater diversion channel included in the design diverts freshwater reporting to the eastern limb of the North pit. The flow is routed via backwater dam no. 2 and its spillway channel, which is routed between SCD2 and the process plant.

Two significant flood protection bunds are included in the design, one for the South pit and one for the North pit. The footprint of the South pit and process plant lies partially within the tidal flood plain of the River Cacheu, 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 River Cacheu. This bund will be constructed in stages with a temporary bund 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 5.5 m above mean sea level 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 and a wearing course along the crest.

The North pit will require the construction of flood protection bunds during the initial stages of pit development. Several smaller diversion channels and bunds will also be constructed to divert runoff from the upstream catchment past the active mining area.

18.11.3 Site Water Management Model

18.11.3.1 Model Description

A site water management model was developed to understand the flow of water to and from the following locations:

- open pit
- process plant
- tailings storage facility
- waste dumps
- sediment control dams
- environmental control dam.

Figure 18-9 presents a conceptual block model of the site water management system. The model was used to simulate expected water flows under average climatic conditions throughout the life of the project and determine the impact of extreme rainfall events at critical times during the operation. The implications for the individual structures are discussed in the following subsections.

18.11.3.2 Open Pit Dewatering

Pit inflows during the mining operation are expected to be of the following order of magnitude:

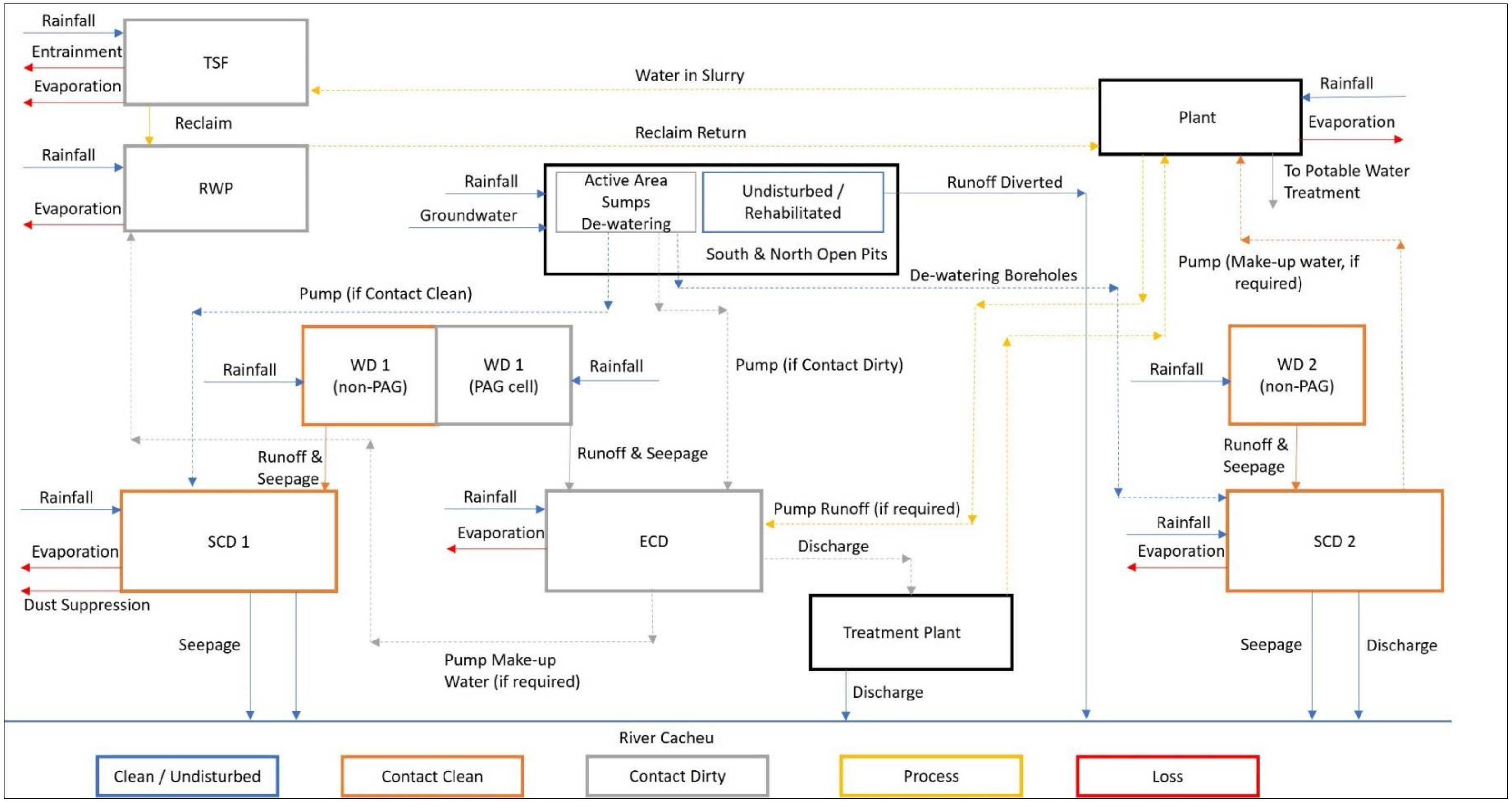
- The South pit average daily pit inflow will be approximately 13,000 m^3/d, ranging from 9,800 m^3/d to 16,700 m^3/d.
- The North pit average inflow will be 6,500 m^3/d, peaking at 8,900 m^3/d and 5,100 m^3/d at the end of mining (Year 26).

Water generated from the pit dewatering borehole is considered to be “clean” (it is better quality than the river water) and will be pumped directly to River Cacheu. Water pumped out of the pit sumps and well points will initially be pumped to an SCD to settle suspended sediment and conduct water quality monitoring prior to release into River Cacheu.

18.11.3.3 Tailings Storage Facility

The TSF maintains a neutral water balance, as long as water is reclaimed for processing. Reclaim rates during average climatic conditions vary between 2,840 m^3/d during the rainy season and 0 m^3/d during the dry season.

Figure 18-9: Farim Water Management Model Block Diagram



Source: Knight Piesold, 2023

18.11.4 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, while the one in Ponta Chugue will have a 17 m^3 capacity. Two potable water pumps (one operating and one on 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 brine will be pumped to a local area sump and periodically back into the process circuit.

18.12 Hazard Considerations

The TSF, RWP, sediment and environmental control dams, backwater dams, diversion dams, and flood protection bunds are all classified as dams with a safety risk due to flooding and river proximity.

These have been designed based on the following dam classifications, per the CDA Technical Bulletin on the Application of Dam Safety Guidelines to Mining Dams (2019):

- TSF .... high
- RWP .... significant
- SCD1 .... low
- SCD2 .... low
- Environmental control dam .... significant
- Backwater dam no. 1 .... high
- Backwater dam no. 2 .... high
- Diversion dam .... high
- Permanent flood protection bund: south .... high
- Permanent flood protection bund: north .... high

These dam classifications have been adopted considering the population at risk and potential incremental losses that may occur if any of these structures should fail.

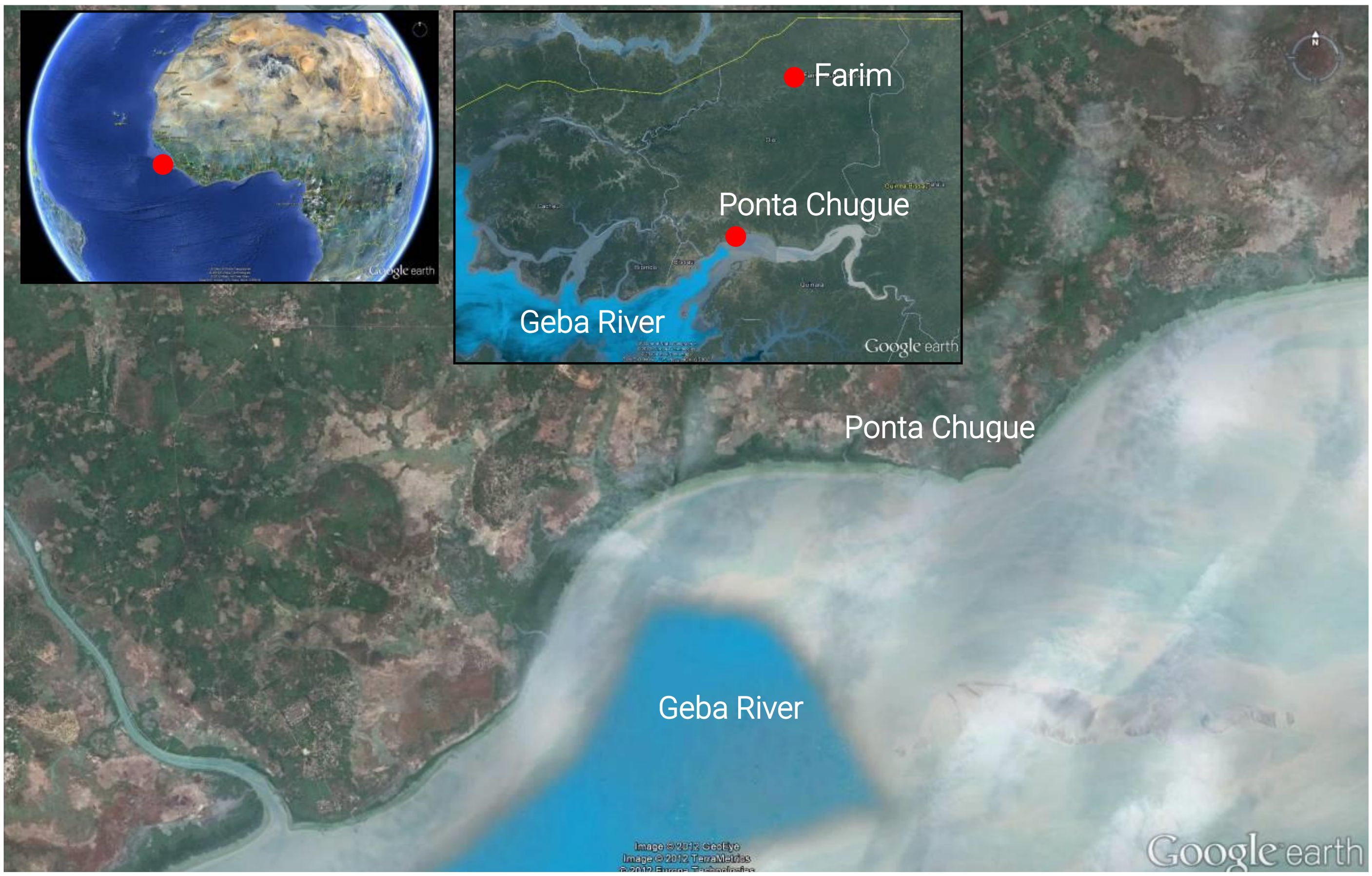
18.13 Marine Design at Ponta Chugue

The proposed marine terminal is located at Ponta Chugue on the Geba River estuary (Figure 18-10). The phosphate rock will be exported by bulk carriers that navigate the River Geba for 180 km to the project site for direct loading at a fixed wharf. Discussions around the land acquisition of the Mineral Terminal were put on hold during the pandemic and are meant to resume in the next phase of the project. Detailed preliminary designs of marine elements of the project were created to develop the project capital and operating cost estimates and generally consist of the following facilities (Figure 18-11):

- Shiploading infrastructure, including an access trestle composed of marine foundations, gallery support structures, a gallery-supported conveyance system and shiploader foundation structure(s), a radial telescoping shiploader, berthing/mooring dolphins, and various access gangways and catwalks
- docking infrastructure to house two tugboats and a pilot boat that are required to support berthing/de-berthing, pilotage, and warping operations

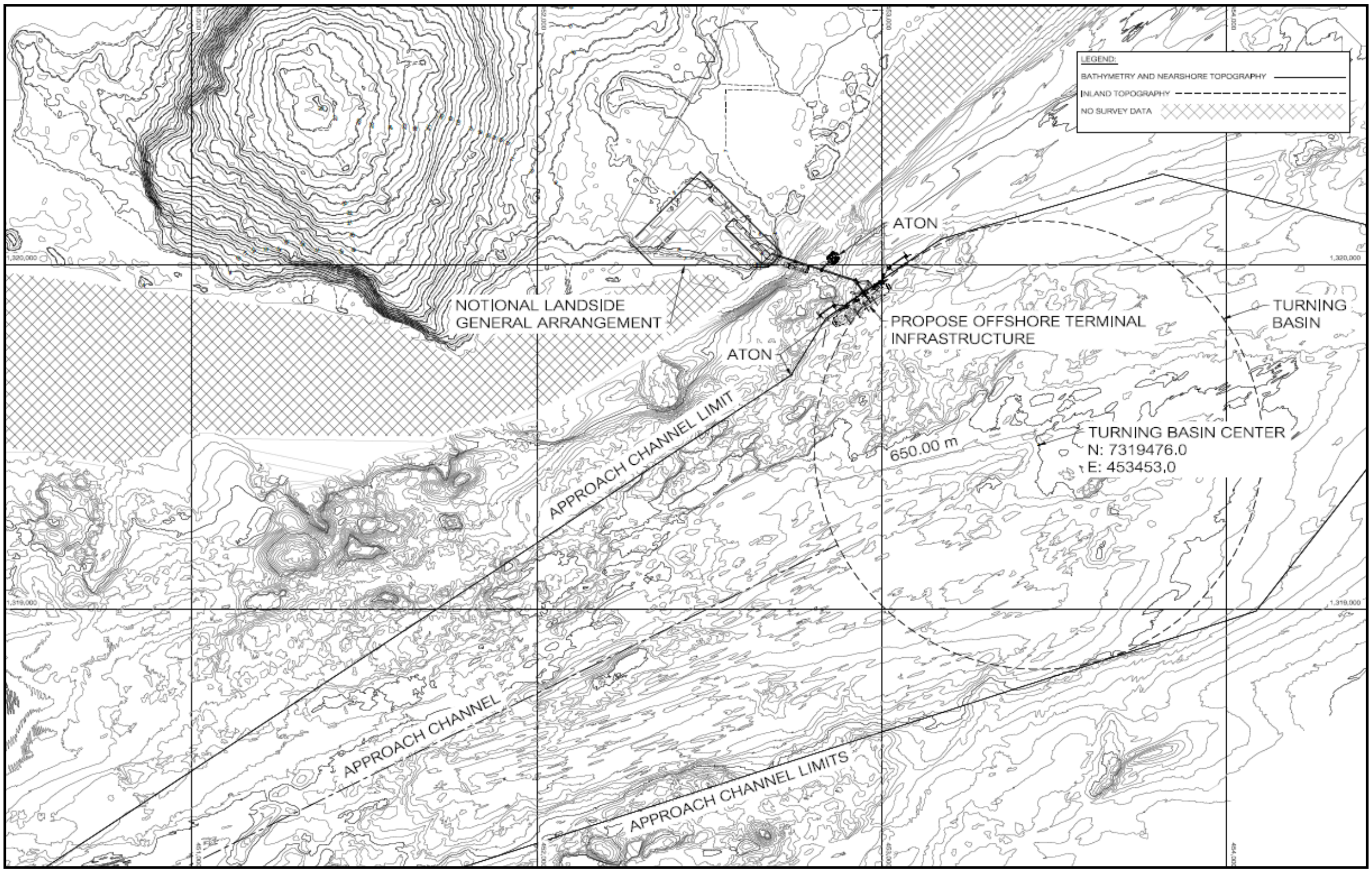
- maintenance barge infrastructure, including a material offloading facility and berthing infrastructure; the barge will be used to transport a crane from the landside to the terminal for maintenance purposes
- various aids for navigation
- fuel unloading infrastructure that allows fuel ships to be unloaded at the marine wharf and conveyed to a storage facility
- associated mechanical, electrical, plumbing, and lighting infrastructure, including a tug fueling system, ship fuel offload system, and water supply.

Figure 18-10: Marine Terminal Location



Source: Baird, 2019

Figure 18-11: Proposed Marine Terminal Facilities



Source: Baird, 2019

18.13.1 Operating Philosophy

Key aspects of the operating philosophy for the marine terminal are listed below:

- The marine terminal will be used to export phosphate product and import fuels to support the mine, processing equipment, and supply vessels. The use of the terminal for fuel offloading introduces a number of specific requirements related to marine spill mitigation and firefighting.
- The estimated annual phosphate export (single product type) is estimated at 1.32 Mt/a.
- The annual diesel fuel oil (grades 1, 2 or 4) import is estimated at 30,000 m^3.
- The design bulk carrier sizes range from Handysize (30,000 DWT) to Supramax (62,000 DWT). It is possible that Panamax vessels (65,000 to 80,000 DWT) might be utilized. The fuel tankers are expected to be of similar size to those calling at Bissau with a range of 4,000 to 13,000 DWT. Itafos will provide two tugboats and a pilot boat. Additional details are provided in Section 18.13.2.

- A radial telescoping shiploader system will be provided (refer to Section 18.13.4.11 for details). The shiploader has limited reach and, as a consequence, the vessel will need to be warped along the berth to align the holds of the vessel with the shiploader. Warping is conducted using tugboats and the vessel's engine. This type of movement is not recommended when current speeds exceed 2.5 knots.

- The phosphate product must be kept dry and will not be loaded when it is raining.

- The vessels will navigate a 180 km long channel in the Geba River from the Atlantic Ocean to the Ponta Chugue terminal (refer to Section 18.13.3 for details). There are depth constraints at two locations along the channel: (1) at Ponta Bernafel, approximately 28 km downriver from the terminal; and (2) offshore at the Bijagos Breaker area, approximately 120 to 170 km from the terminal. As a result, the deeper draft vessels arriving at the terminal will not be able to load to capacity and will have to navigate these shallow areas at or near high tide (i.e., "tidal assistance" is assumed).

- The arriving vessels will need to be turned in the river by tugs to achieve a starboard arrival at the berth (i.e., bow pointing downriver). Based on the results of real-time navigational simulations (refer to Section 18.13.3.4 for details), it is recommended that the vessels not turn unless the current speeds are less than 2.5 knots.

- Pilots will be employed to guide the arrival and sailing of the vessels. It is anticipated that a pilot will board the inbound vessel in the vicinity of Ponta de Caio and will deboard the outbound vessel beyond Bijagos Breaker (refer to Section 18.13.2.3.2 for details).

- The tugboats will be moored at the terminal alongside a floating pontoon system.

- A minimum capital cost approach to terminal maintenance has been selected that does not include vehicular access to the terminal facilities. A barge will be purchased and permanently stored at Ponta Chugue for maintaining marine infrastructure. Equipment and materials will be loaded onto the barge at a maintenance barge landing that will be constructed.

18.13.2 Vessel Requirements

This section defines design dimension associated with the range of vessels anticipated to use the terminal, including bulk carriers, fuel tankers, and services vessels (tugs, pilot boat, and maintenance barge). The capital and operating cost estimates were developed assuming the following operating philosophy:

- Bulk carriers and fuel tankers will be chartered.
- Service vessels (tugs, pilot boat, and maintenance barge) will be purchased and operated by Itafos.

18.13.2.1 Bulk Carriers

As part of the marine studies (Baird, 2019b), the dimensions of bulk carriers in the World Fleet and Automatic Identification System (AIS) data of bulk carrier traffic around West Africa were analyzed. The selected design vessels for the terminal range from GB Handysize to Supramax vessels, as shown in Table 18-11.

Table 18-11: Vessel Summary Based on the Seaweb 2015 World Fleet Register

Vessel Type	Vessel Size	DWT		LOA		Beam		Draft		Number of Vessels	% Gear
		Low	High	P10	P90	P10	P90	P10	P90		
Handy	Small Handysize	25.0	29.9	166.0	176.6	23.8	27.2	9.5	10.2	695	97%
	GB Handysize	30.0	34.9	175.5	181.0	27.0	30.0	9.6	10.4	776	99%
	Large Handysize	35.0	40.9	177.9	190.3	27.8	30.1	10.0	10.9	590	96%
Handy-max	Handymax	41.0	49.9	184.9	199.9	30.4	32.3	10.7	11.8	771	88%
	Supramax	50.0	61.9	189.9	199.9	32.3	32.3	12.0	13.0	2,133	98%
	Ultramax	62.0	64.9	199.9	225.0	32.3	33.0	12.3	13.3	173	91%
Panamax	Panamax	65.0	79.9	224.9	225.0	32.2	32.3	13.3	14.3	1,362	13%
	Kamsarmax	80.0	84.9	228.9	229.0	32.2	32.3	14.4	14.6	655	12%
	Post Panamax	85.0	109.9	229.0	244.5	36.8	43.0	13.1	14.9	448	08%
Capes	Minicapes	110.0	159.9	249.9	274.0	43.0	45.0	14.5	17.5	182	09%
	Capesize	160.0	210.0	288.9	299.9	45.0	50.0	17.7	18.3	1,219	07%
									Total	9,004	

Note: Orange highlighting indicates terminal target vessel range.

It is common practice in the design of bulk terminals to select representative examples of the smallest and largest vessels (in the expected range of vessels using the terminal) as the design vessels. Emphasis in the selection is placed on vessel length since this determines fender layout and mooring line angles. The dimensions of the two bulk carrier design vessels selected for the project are provided in Table 18-12.

Table 18-12: Dimensions of the Design GB Handysize Bulk Carriers

Class	Unit of Measure	GB Handysize	Supramax
Deadweight Tonnage	t	32,000	61,400
Length Over All	m	175.5	200.0
Beam	m	29.4	32.2
Draft Maximum	m	10.4	13.0
Sailing Draft for Ponta Chugue	m	10.4	11.2 to 11.6

The lengths of the design vessels are equal to the 10% and 90% probability values for the Handysize and Supramax vessels, respectively. “Draft maximum” is the listed maximum draft of the selected vessel. “Draft GB” is the laden draft assumed for the jetty design. The 11.6 m draft from laden sailing of the Supramax vessel from Ponta Chugue is less than the draft maximum of the Supramax vessels due to underkeel clearance (UKC) constraints associated with the departure channel. The maximum draft for Supramax vessels sailing laden from Ponta Chugue will vary depending on offshore wave conditions. Without offshore wave constraints, 11.6 m sailing draft could be achieved for most sailing windows; however, when larger northwesterly swells propagate to the coastal waters of Guinea-Bissau, the maximum sailing draft may decline. During January and February when swells from the northern hemisphere’s winter regularly occur, sailing drafts of 11.6 m will only be available on 20% of sailing windows. During this period, the most probable sailing draft to clear the outer shoals will be 11.3 m. The vessels, especially the Supramax classes, may be loaded to varying drafts, depending on the tide and the wave conditions. Table 18-13 provides the deadweight tonnage for drafts between 10 and 12 m for representative vessels in each class. The deadweight tonnage includes both phosphate rock and bunkering liquids. An approximation for product tonnage is generally 95% of DWT.

Table 18-13: Approximated Deadweight Tonnage (tonnes) at Different Drafts

Draft (m)	Handysize*	Supramax
10.0	31,300	43,200
10.5	33,600	46,200
11.0	-	49,300
11.5	-	52,300
12.0	-	55,300

Note: Handysize based on a 35,000 DWT vessel with maximum draft of 10.8 m.

18.13.2.2 Fuel Tankers

In addition to bulk handling, fuel delivery will be performed at a fuel offloading platform adjacent to the shiploader. The design vessels for these operations are anticipated to include a fleet of tankers similar to those currently calling at the Port of Bissau. Satellite-derived AIS data from 2016 indicates that the larger vessels calling at Bissau ranged in size from approximately 4,000 DWT (95 m LOA) to 13,000 DWT (129 m LOA).

Details for two selected vessels are shown in Table 18-14 (including manifold locations for two of the largest vessels in the 14-vessel dataset). The cargo-handling rates shown in Table 18-14 are expected to vary significantly between individual vessels and will be investigated further. In the interim, a variation of at least ±25% has been considered.

Table 18-14: Detailed Particulars for Two Selected Tanker Vessels

Vessel Name	Lisse (ex Lady Astrid)		Northsea Beta (ex Pyxis Beta)	
Built	2009		2010	
Dimensions				
LOA (m)	128.6		110	
LBP (m)	120.4		104.7	
Beam (m)	20.4		18.6	
Depth (m)	11.5		10	
Bow to Manifold CL (m)	52.1		53.5	
Stern to Manifold CL (m)	67.6		56.5	
Cargo Handling				
Pump Capacity			3 x 550 m³/h @ 0.7 MPa/70 m	
Simultaneous Pumping			2 pumps simultaneously	
Maximum Loading Rate (m³/h)	Unknown		1,000 per manifold connection	
Maximum Loading Rate (m³/h)	Unknown		2,000 through all manifolds	
Loadline Information				
Loading Condition	Summer Laden	Normal Ballast	Summer Laden	Normal Ballast
DWT (t)	13,062	6,478	8,615	4,270
Displacement (t)	17,462	10,878	11,751	7,406
Draft (m)	8.688	5.7	7.8	5.19
Freeboard to Main Deck (m)	2.812	5.8	2.2	4.81

18.13.2.3 Support Vessels

The adopted operating philosophy assumes Itafos owns and operates tugboats, pilot boats, and a maintenance barge; the following subsections provide the recommended specifications for these support vessels.

18.13.2.3.1 Tugboats

The navigation simulation studies included real-time, full-mission simulations with Tug Master-operated tug models (Baird, 2019b). The desktop and full-mission simulations identified that 40 tonnes of effective assistance from each of the two tugs is required for berthing and departures at Ponta Chugue. Due to the strong crosscurrents, only azimuth stern drive (ASD)-type tugs are recommended, as these tugs are better able to maintain towing position on the shoulder/quarter, while providing effective push/pull forces to assist with maneuvering the vessel. Simulations were completed with only one tug assisting, and while arrivals and departures may still be possible to/from the Mineral Terminal, further operational restrictions (i.e., tide or vessel size limits) would need to be implemented. Due to the strong operational currents, there is benefit in having a tug with low hull resistance and/or displacement to improve the performance of the tug in strong currents. It is recommended that the tug have a small, single centerline skeg and a forward-towing winch with heavy bow fendering to operate in the push-pull mode.

The Ponta Chugue Port is located in a remote location with limited access to parts and technician labor. The tugs will be required to have a comprehensive inventory of critical spare parts on board and crew that can perform servicing, maintenance, and repairs. The specifications for the tugboats are provided in Table 18-15.

Table 18-15: Approximate Tugboat Specifications

Bollard Pull (static)	50 tonnes (ahead)
Engine Power	3,000 kW
Tug Type	azimuth stern drive (ASD)
Length	20 to 30 m
Draft	3 to 4 m
Operational Cross-Current	2 m/s (4.0 knots)
Bollard Pull in Operational Current	40 tonnes (push/pull)
Astern Operating Speed (Line Fast)	7 knots

18.13.2.3.2 Pilot Boat

A pilot boat will be required to transport the pilot from Ponta Chugue to the pilot pickup point, where the pilot will board the bulk carrier. 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. It has been assumed that the dedicated Farim pilot will board at a similar location.

Metocean design criteria for the pilot boat are summarized as follows:

- significant wave heights up to Hm0 = 2.8 m at Bijagos breaker, Hm0 = 0.7m at Ponta Chugue (Tp~4.5 s)
- current speeds up to 5 knots at Ponta Chugue
- operating wind speed of approximately 20 m/s; extreme conditions (100-year ARI) of 50 m/s.

In general, pilot boat design should consider guidance in the Workboat Code (MCA, 2014). Preliminary vessel construction requirements are as follows:

- minimum service life of 15 years with limited capability for maintenance at the project site
- less than 24 m load line length
- proposed operating area UK MCA Area Category 2 (up to 60 miles from a safe haven)
- aluminum hull construction
- good seakeeping and crew comfort to mitigate pilot fatigue
- fuel, fresh water, and sanitary tank capacities to satisfy operating profile and outfit requirements.

18.13.2.3.3 Maintenance Barge

A maintenance barge (crane barge) is required for general maintenance at the terminal and at the fixed aids to navigation (ATONs). The barge will have the following general characteristics:

- 50 m length x 15 m breadth (approximate depth 3.1, draft 1.8 m)
- covered area and crew house
- deck rated for a minimum 110-tonne crane
- mooring/anchoring systems and/or spudwells sufficient for the depths at the terminal and adjacent to the fixed ATON (total water depth approximately 22 m at high tide).

18.13.3 Vessel Navigation

Vessels accessing the Mineral Terminal at Ponta Chugue from the Atlantic Ocean will navigate a 180 km long route through the Geba River estuary, as illustrated in Figure 18-13.

A series of marine studies were undertaken to assess and define the preferred navigation route and the required navigation techniques. The marine studies (Baird, 2019b) included a hydrographic survey of the estuary, a UKC study, and both desktop and full-mission navigation simulations.

The capital and operating cost estimates were developed assuming that capital and maintenance dredging to construct a navigable channel and berth is not required for the proposed operation of the marine terminal.

18.13.3.1 Channel and Berth Layout

A navigation route has been defined based on the results of the 2017 bathymetrical survey (EGS, 2017). The route has a length of almost 100 nm (180 km) from the Port at Ponta Chugue to deep water offshore in the Atlantic Ocean. The route can be divided into 20 (almost) straight segments between subsequent waypoints. A centerline and outline of the fairway was created to assist the pilot to navigate the vessel along the natural navigation route from the Mineral Terminal to deep water offshore. The width of the displayed fairway is 250 m over the entire route even though the width of navigable waters is often much wider. This is well above the required width recommended by PIANC (The World Association for Waterborne Transport Infrastructure).

There are two key maneuvering constraints for departures from the Port. The first is the section near Ponta Bernafel as indicated in Figure 18-13 at around 15 nm (28 km) after the start of the channel at Ponta Chugue. This section consists of a number of hard outcrops in the riverbed. The fairway has been defined around these outcrops to provide sufficient underkeel clearance. The recommended procedure is to leave the berth in flood currents after low tide and transit the

Bernafel section 2.5 to 3 hours later in flood currents close to high tide. In this way, the maneuvers near Ponta Bernafel are made in a counter current, which is beneficial for maneuvering.

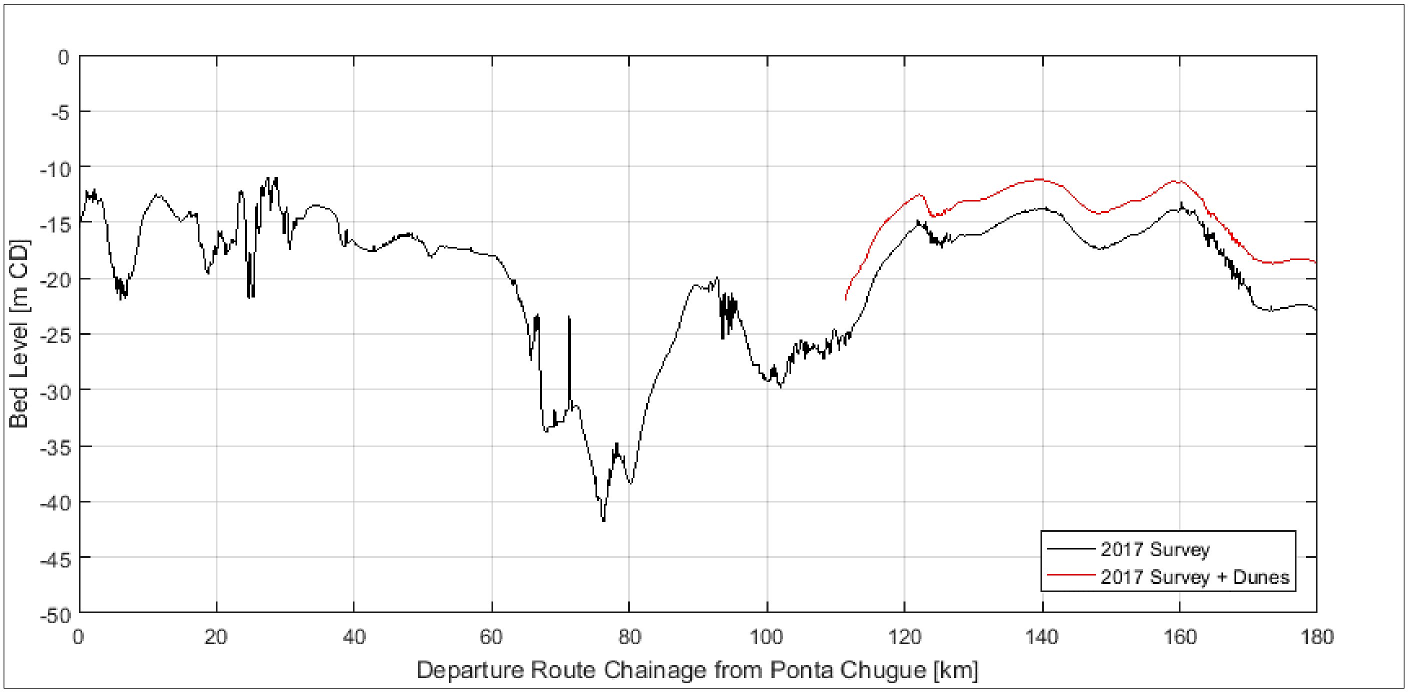
Figure 18-12: Main Marine Terminal Navigation Route

Source: Baird. 2019

The second maneuvering constraint is offshore in the Bijagos breaker region (see Figure 18-13) where an allowance is made at the offshore bars for the formation of dunes that can develop and migrate along the channel in this area due to the effects of tidal currents. The data available on bedforms in this area is limited; however, the marine studies completed have identified that the largest bedforms measured in the offshore region in the 2017 survey conform with empirical and analytical methods that are available to predict the amplitude and wavelength of these features.

The channel profile is given in Figure 18-13 relative to chart datum (CD) at each section from the results of the 2017 bathymetric survey.

Figure 18-13: Channel Profile from Ponta Chugue



Source: Baird, 2019

Note: Bed level along the navigation route marking the shallowest points in each 100 m section and allowance for formation of dunes in the offshore area.

Upon arrival, the vessels follow a more northerly route in the offshore area to the existing pilot station near Ponta de Caio. The vessels subsequently follow the same fairway as the departure route from Ponta Arlete to Ponta Chugue. The available fairway widens close to the Mineral Terminal. A turning circle with a radius of 650 m is available across the river in front of the berth with a natural water depth of at least 12.5 m. This wide, natural turning circle allows turning with a wide, curved track entering the circle from the southwest and ending close to and aligned with the berth. This enables safe turning of the vessel mostly relying on the vessel's engine and rudder in strong tidal currents.

18.13.3.2 Vessel Warping

The vessels will be warped alongside the berth to align the shiploader with the vessel's cargo holds. Vessel warping was simulated during the navigation simulations. The recommended procedure for warping is as follows:

- Have two tugs push at the shoulder and quarter positions to secure the vessel against the fenders
- Remove tension in all mooring lines at the winches on the vessel.
- Use the vessel's main engine to move the vessel forward or aft with a speed of up to 0.5 knots while the tugs continue to push to keep the vessel in contact with the fenders.
- Stop the vessel's engine to reduce speed close to the new warping position and, if the effect of fender friction is not sufficient, reverse the thrust to stop the vessel movement.
- Tension all mooring lines at the vessel's winches.
- Release the tugs.

Safe warping of the vessel is possible in flood currents with a speed of up to 4 knots and ebb currents with a speed of up 3 knots; however, for normal operations, particularly at the commencement of Mineral Terminal operations, it is recommended that warping maneuvers only be conducted if the current velocity is less than 2.5 knots.

18.13.3.3 Underkeel Clearance

The UKC study (Baird, 2019b) assessed the suitability of the navigation channel for several vessel classes, including: Handysize (30,000 to 35,000 DWT), Handymax (45,000 to 50,000 DWT), Supramax (55,000 to 60,000 DWT), and Panamax (65,000 to 80,000 DWT). The results of the UKC study confirmed the feasibility of the proposed navigation route; however, as previously stated, two critical sections were noted along the navigation route, as indicated in Figure 18-17 and described below:

- near the mouth of the estuary at “Bijagos Breaker”, due to large seabed “sand wave” formations
- near Ponta Bernafel due to the presence of shoals.

The UKC study indicated that laden vessels departing Ponta Chugue will require tidal assistance to maintain a safe UKC. Handysize vessels may depart fully laden with only marginal tidal restrictions—specifically, near low tide at the two locations noted above. Laden Handymax, Supramax, and Panamax vessels will always be tidally restricted; the latter two must only be partially laden to a draft of approximately 11 to 12 m.

During the months of December, January, and February, northwesterly swells from the northern hemisphere winter will be largest in the “Bijagos Breaker” region. Sailing drafts from Supramax and Panamax vessels will be most limited during this period.

18.13.3.4 Results of Navigation Simulations

The navigation simulations established the transit tidal windows and techniques to be employed during navigation to and from the Mineral Terminal. In addition, the simulations determined that the wharf must be oriented to align with the predominant ebb and flood current directions to assist de/berthing and warping operations. The simulations also assessed the towage requirements and confirmed the ATONs for the Mineral Terminal. The full-mission simulations were performed using Handysize, Supramax, and Panamax vessels, with the key results summarized below.

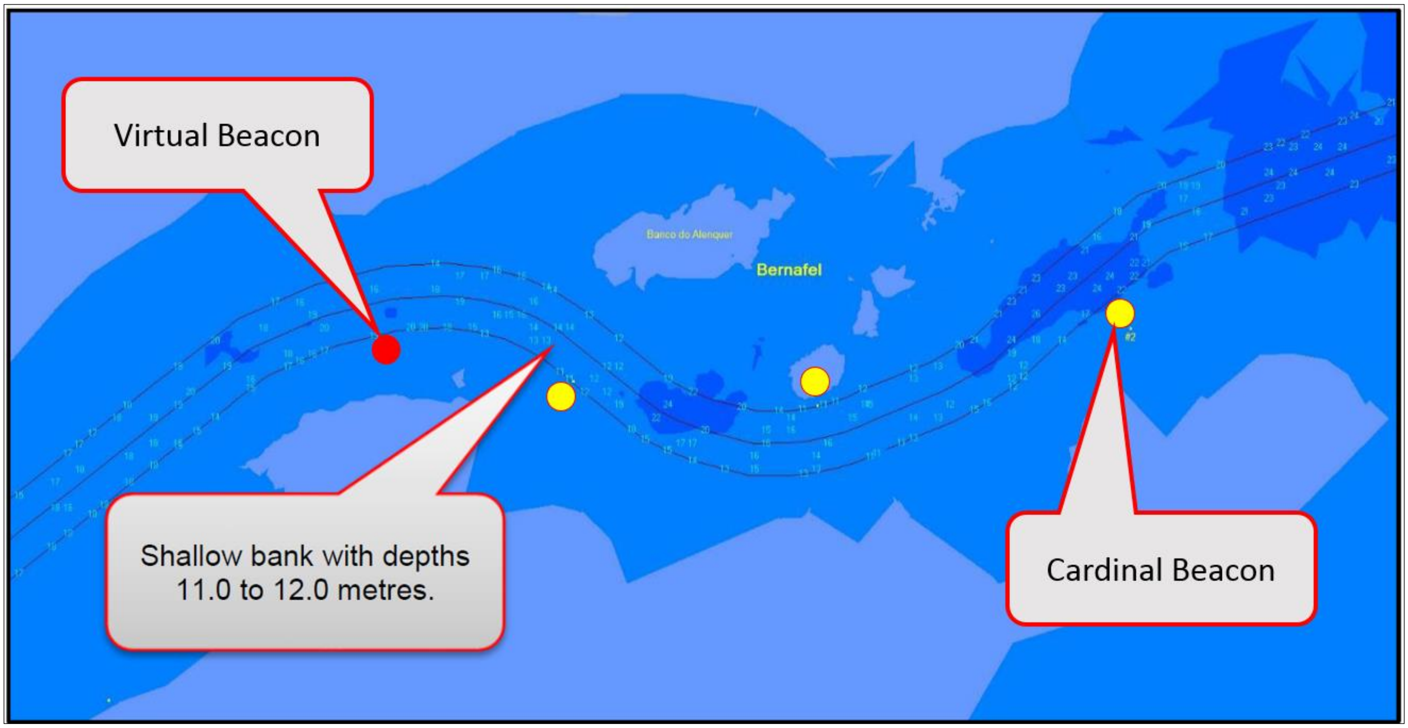
18.13.3.4.1 For Transits of the Geba River Estuary

The preferred outbound transit window is the 90 minutes leading up to high-water slack tide at Bernafel. This ensures that the vessel is stemming a current with a velocity of 1.5 knots or less; it also affords an additional degree of safety by providing the maximum tidal lift/UKC while transiting the shoals. The opportunity may exist in the future (once familiarity has been gained) to increase the outbound tidal window (incrementally) to a maximum tidal velocity of 3.0 knots on the flood tide and 2.0 knots on the ebb tide.

The three beacons (cardinal markers) proposed at Bernafel (Figure 18-14) provide sufficient visual cues when coupled with radar and portable pilot unit (PPU) information. It is recommended that the cardinal markers be fitted with AIS transponders to facilitate correlating their position on the radar/PPU.

Consideration should be given to providing a virtual beacon at the apex of the final bend through Bernafel (i.e., to provide guidance around the curve, but not limit the ability of a vessel to transit through the waters, if required).

Figure 18-14: Bernafel Section of Channel



Source: Baird, 2019

18.13.3.4.2 For Maneuvering Operations at the Berth

To meet the tidal window transit requirements at Bernafel, departures must be conducted during the flood tidal cycle, with preference to the latter portion of the flood.

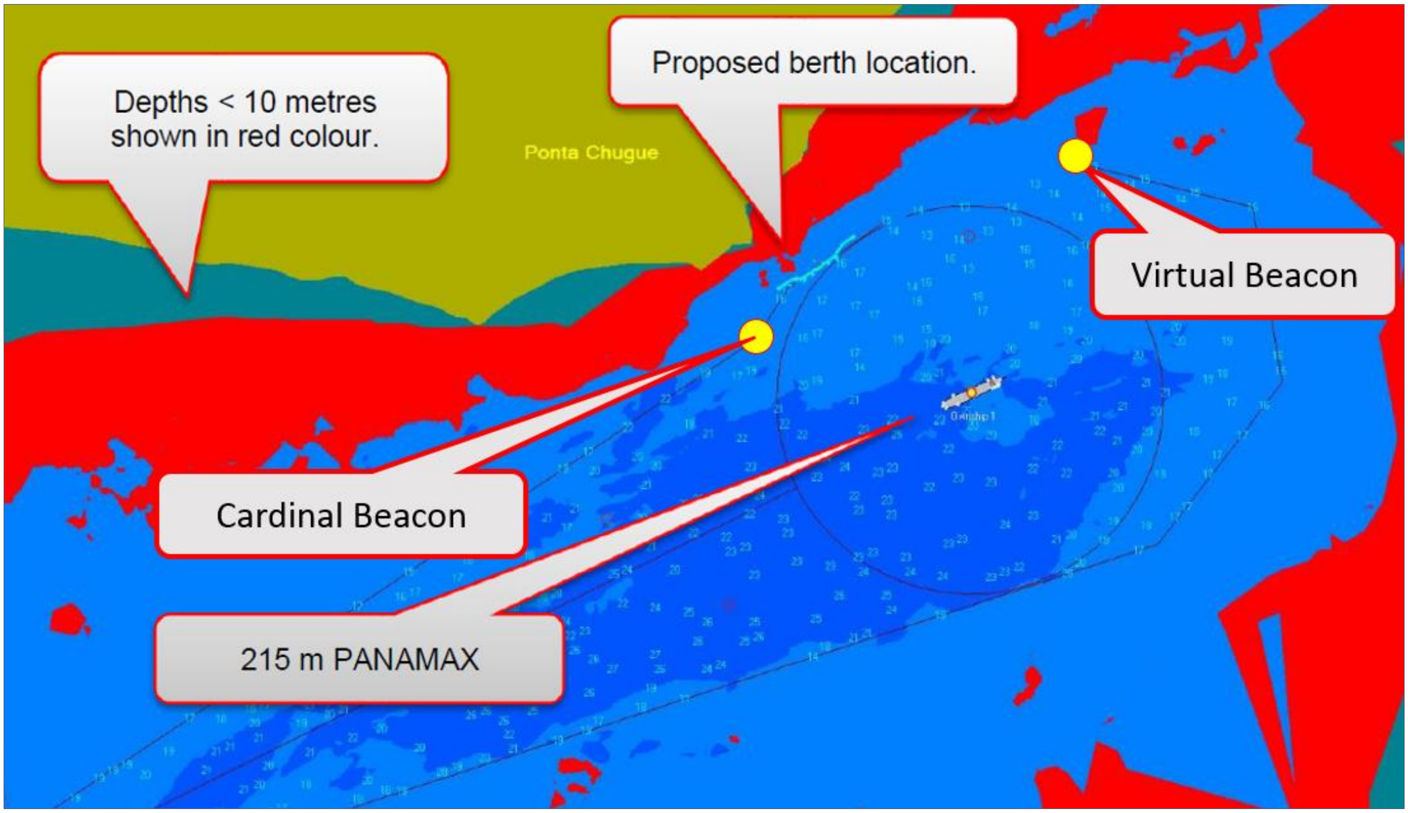
Port side departures during the flood tidal cycle are considerably more complicated than starboard side departures. Therefore, it is recommended that Mineral Terminal side arrivals (and the associated Mineral Terminal side departures) be avoided. The preferred arrival condition is a starboard side arrival when the current velocity is less than 2.5 knots.

Unless there is an operational necessity, it is recommended that ebb tide arrivals be avoided; however, if required, they should not be conducted if the current velocity exceeds 2.5 knots.

All maneuvers should be conducted with the assistance of two ASD tugs, with one tethered near the bow and the other near the stern. The tugs must be capable of working perpendicular to the side of the vessel in a current of 4.0 knots. This capability is required both when the vessel's ground speed is very low while at the berth (warping and arrival/departure) and when the vessel is being maneuvered in the basin at water speeds up to 4 knots.

The beacon marking the shoal southwest of the berth is very useful, particularly as a visual guide during Mineral Terminal side to berth departures. Consideration should be given to incorporating a virtual beacon to mark the shallow areas (i.e., 5 m depth) to the northeast of the Mineral Terminal limit, as illustrated in Figure 18-15.

Figure 18-15: Aids to Navigation in the Terminal Area



Source: Baird, 2019

18.13.3.5 Declared Berth Depth

The marine terminal will be provided with a minimum navigation depth (declared depth) at the berth of 12.6 m CD. This will provide 1 m UKC for a laden ship at berth at 0 m LAT tide. The berth depth will be -13.9 m CD to account for sedimentation and seabed survey tolerances/uncertainties, as summarized in Table 18-16.

Table 18-16: Berth Depth Calculation

Description	Chart Datum (m LAT)
Design Vessel Draft (Static)	11.6
Minimum 1 m UKC at Berth	1.0
Low Tide Offset	0.0
Declared Depth	-12.6
Sedimentation (Allowance)	1.0
Survey Tolerance	0.3
Dredging Tolerance	0.0
Berth Depth	-13.9

18.13.3.6 Aids to Navigation (ATONs)

The following ATONs will be installed/developed as part of the project:

- one pile-supported beacon marking the shoal directly to the southeast of the proposed jetty
- three pile-supported beacons marking the critical shallow areas in the Bernafel section
- electronic navigation aids, including the following:
  - use of electronic navigation charts (ENC) and electronic chart display and information system (ECDIS) onboard vessels
  - use of portable pilot units (PPUs) to assist the pilots in planning and monitoring their progress during navigation to and from the berth
  - incorporation of two virtual beacons, as described in Section 18.12.3.4.

Table 18-17 presents the position and properties of the physical ATONs.

Table 18-17: Position and properties of the marks and lights for the four beacons along the navigation route

#	Route Section	Position		Mark	Light		
		Latitude	Longitude		Elevation	Range	Intensity
1	Ponta Chugue	11.93749° N	15.43407° W	South Cardinal	8.8 m CD	2 nm	5 cd
2	Ponta Bernafel	11.77168° N	15.63757° W	North Cardinal	8.3 m CD	2 nm	5 cd
3	Ponta Bernafel	11.76898° N	15.64932° W	South Cardinal	8.3 m CD	2 nm	5 cd
4	Ponta Bernafel	11.76982° N	15.65845° W	East Cardinal	8.3 m CD	2 nm	5 cd

18.13.3.7 Real-Time Metocean Measurements

Real-time metocean data will be required for Mineral Terminal operations and navigation. As a minimum, the following will be required:

- weather station at Ponta Chugue measuring temperature, wind and rainfall
- real-time tide gauges referenced to LAT datum at Ponta Chugue (marine terminal) and Bernafel (located on an ATON).

For Mineral Terminal operations and planning for vessel warping, there may be a requirement that a real-time current measurement instrument be installed at Ponta Chugue to measure currents near the berth.

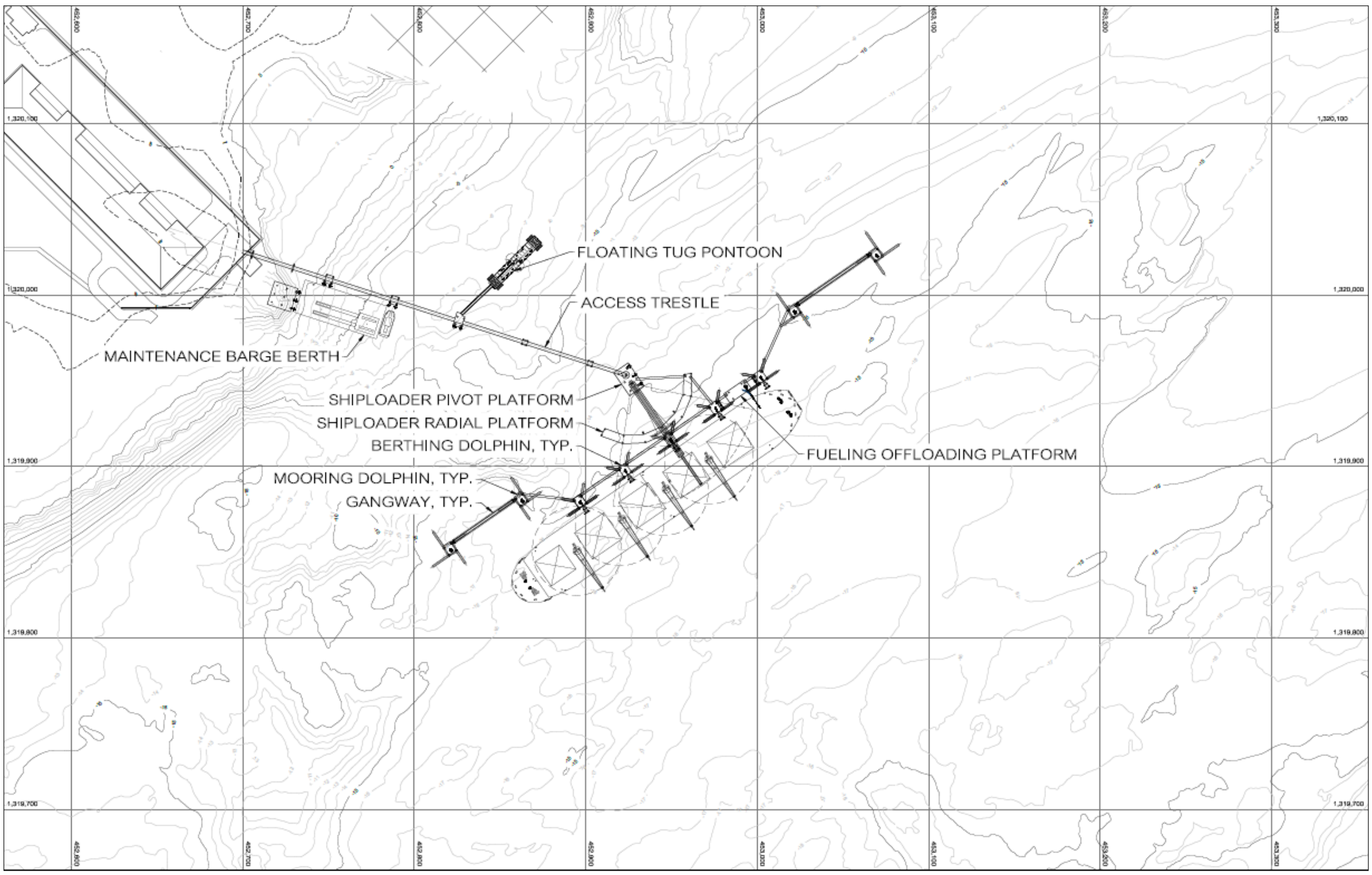
18.13.4 Marine Terminal Infrastructure

The overall layout of the proposed marine terminal to be constructed and operated by Itafos is presented in Figure 18-16. Bulk carriers will berth starboard side to the jetty. The vessel is warped along the jetty to align the cargo holds with the shiploader. Warping positions with the shiploader aligned with the second hold, second-to-last hold, and holds in between will be considered to determine the positions of the berthing and mooring dolphins. The proposed marine terminal consists of a number of major components, as discussed in the following subsections. Detailed marine terminal design criteria are provided in the basis of design document (Baird, 2019c).

18.13.4.1 Structural Design Concepts

The functional requirements and engineering design criteria outlined in this document assume the use of steel piles and structural members to facilitate pre-fabrication of major elements off site, with simple erection on site. This methodology has been selected for its advantages with respect to quality control and ease of construction considering the strong currents that exist on site.

Figure 18-16: Marine Terminal Layout



Source: Baird, 2019

18.13.4.2 Access Trestle Foundation

An access trestle, with the following primary characteristics, will support the conveyor galleries:

- ± 240 m long trestle with maximum 25 m spans between piled foundations and at least 9 marine foundations
- 9 piled foundations, 7 foundations with tubular jacket structures
- infrastructure for one davit crane to support maintenance activities
- mid-length stair access from conveyor to tug berth gangway
- pedestrian access to the wharf along walkway to either side of the conveyor (no vehicular access)
- berthing and mooring infrastructure on at least four of the foundations for the maintenance barge
- safety infrastructure.

18.13.4.3 Shiploader Foundation

A pivot platform and a radial foundation will be provided for a radial telescoping shiploader. The pivot platform foundation will generally consist of the following:

- piled foundation with tubular jacket
- pivot mount
- conveyor transfer tower to support conveyor, chute, and access stairway
- stair access from conveyor to pivot platform deck
- one ladder / grab handle at the rear of the dolphin extending to the LAT elevation
- area lighting
- handrail around the perimeter.

The radial foundation will generally consist of the following:

- piled foundations with tubular jackets
- access catwalk to pivot platform on one side of the radial beam
- one ladder / grab handle at the rear of the dolphin extending to the LAT elevation
- area lighting
- handrail around the perimeter.

18.13.4.4 Berthing Dolphins

The marine terminal will be provided with berthing infrastructure to accommodate the design vessels as described in Section 18.13.2. The berthing dolphins will generally consist of the following:

- five dolphins spaced 30 to 35 m apart, asymmetrically around the shiploader
- piled foundations with tubular jackets
- each berthing dolphin will include:
    - one double quick release hook with a safe working load of 65 tonnes per breasting dolphin

- one high performance cell fender with fender panel and ancillary furnishings
- one safety screen (mooring line snap-back protection)
- one ladder / grab handle at the rear of the dolphin extending to the LAT elevation

- bull rail around 50% of the perimeter and handrail around the remaining 50% of the perimeter, excluding the openings needed for catwalk landings.

18.13.4.5 Mooring Dolphins

The marine terminal will be provided with mooring infrastructure to accommodate the design vessels described in Section 18.13.2. The mooring dolphins will generally consist of the following:

- four dolphins, two on each side of and asymmetrically around the shiploader
- piled foundations with tubular jackets
- one quadruple quick release hook with a safe working load of 65 tonnes per mooring dolphin
- one safety screen
- one ladder / grab handle at the rear of the dolphin extending to the LAT elevation
- bull rail around 50% of the perimeter and handrail around the remaining 50% of the perimeter.

18.13.4.6 Fueling Offloading Platform

The marine terminal will be provided with a fuel offloading platform and fuel offloading system. The fuel offloading platform foundation will generally consist of the following:

- piled foundation with tubular jacket
- mount for 15 m fixed boom crane
- crane access and control platform
- one ladder / grab handle at the rear of the dolphin extending to the LAT elevation
- handrail around the perimeter
- fuel receiving equipment
- fuel pumphouse
- landside fuel storage.

18.13.4.7 Gangways and Decks

The deck elevation for all fixed marine terminal infrastructure is estimated at +8.5 m CD, which elevates the superstructure above the design wave crest during a severe storm at mean high water spring tide. This effectively eliminates wave uplift loads on the deck and significantly reduces lateral loading on the structure.

18.13.4.8 Maintenance Barge Mooring Piles

Additional marine infrastructure, if necessary, will be provided to facilitate maintenance of the marine terminal using a barge. The additional infrastructure will consist of two mono-piles for barge mooring and berthing, if necessary.

The design intent is to locate the maintenance barge berth near the trestle such that the trestle may serve as additional mooring (and fender) points for the maintenance barge. This would eliminate the need for additional discrete piles for mooring points.

18.13.4.9 Floating Tug Pontoon

A mooring location for the support tugs and pilot boat will be provided. All service vessels will be fueled and powered from the tug pontoon. The tug pontoon will generally consist of the following:

- steel pontoon
- piled mooring anchors
- 12 bollards
- six vertical rub strakes
- 30 m long gangway
- tug fueling equipment.

18.13.4.10 Maintenance Barge Berth

Infrastructure will be provided that allows for loading/offloading maintenance equipment and supplies to/from a maintenance barge. The maintenance barge berth will generally consist of the following:

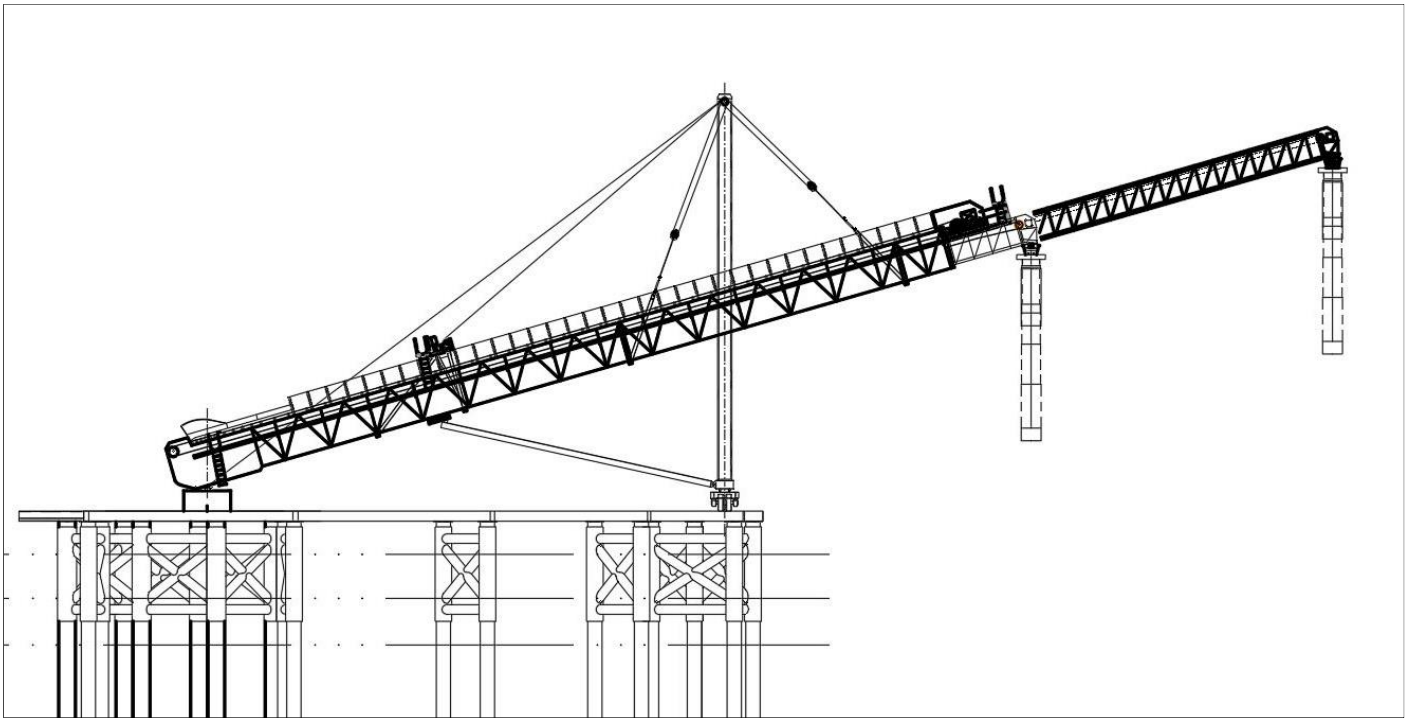
- vertical mooring and berthing piles, or integrated fenders and mooring points at trestle foundations near the maintenance barge berth
- open piled wharf
- landside bollards.

18.13.4.11 Shiploader

A radial telescoping shiploader system (refer to example in Figure 18-17) will be provided that generally consists of the following:

- Shiploading conveyor with telescoping boom
- capability of loading Handysize (35,000 DWT) and Supramax (60,000 DWT) vessels
- nominal conveyance rate of 1,200 t/h
- minimum 65 m telescoping boom conveyor length (customized model) and 21 m discharge height
- conveyor undercarriage with luffing and slewing capability
- fixed tail kingpin with rail or wheeled slewing axle
- design life of 25 years for primary structural frame and non-wear components.

Figure 18-17: Radial Telescoping Shiploader



Source: Baird, 2019

18.13.4.12 Conveyor Galleries

A transfer conveyor system will be provided that generally consists of the following:

- bulk material continuous belt conveyor with truss frame and intermediate support structures
- modularized construction with standard frame sizes and arrangements
- nominal conveyance rate of 630 t/h with maximum rate of 1,200 t/h
- approximate 25 m span length
- 15 m minimum discharge height at shiploader
- dual catwalk to provide pedestrian access to the wharf, tug berth, and access for conveyor maintenance
- conveyor modules provided with covers, integrated utilities for the terminal, etc.

18.13.4.13 Landside Support Facilities

High-level spatial requirements and equipment specifications will be provided for the following landside facilities:

- port office
- tug maintenance shed
- communications tower.

18.13.4.14 Utilities

Utilities will be provided from landside to the Mineral Terminal. Utilities will generally include the following:

- power distribution lines
- control cabling
- fuel lines to/from fuel unloading platform and barge fueling station
- firewater distribution piping at fuel unloading station.

18.13.4.14.1 Wharf Utilities

Utilities provided to the wharf will generally consist of the following:

- power
- compressed air
- water lines
- lighting near all working surfaces and walkways
- fire protection system, including water pump, valves, fittings, and water cannons.

At this time, it is anticipated that no ship to shore services will be provided at the marine terminal.

18.13.4.14.2 Tug Pontoon Utilities

Utilities provided to the tug pontoon will generally consist of the following:

- power
- compressed air
- potable water
- tug and pilot boat fueling
- lighting near all working surfaces and walkways.

18.13.5 Maintenance Philosophy

The marine terminal design is a minimum capital cost solution regarding maintenance. A radial telescoping shiploader is included, but not included are vehicular access along the trestle to facilitate maintenance of the conveyor, associated utilities, berthing and mooring infrastructure, or the shiploader. The design relies on a maintenance barge that would be purchased and permanently stored at Ponta Chugue for the purposes of generally maintaining the marine infrastructure (not including major structural repairs).

Common maintenance services would include the repair of belts, idlers, motors, pulleys, chutes, discharge spouts, and take-up systems. This work would likely only require laborers with hand tools and would be completed from access walkways; however, various other items in excess of 100 pounds would likely require the use of supplemental lifting equipment, including the maintenance barge and fixed davits located at critical locations.

18.13.6 Operational Modeling

The Mineral Terminal at Ponta Chugue will be operationally complex, requiring multiple risk mitigation measures for various environmental factors and the selected philosophy of minimizing infrastructure and equipment to reduce capital costs.

Several of the recommended operational risk mitigation measures will negatively impact operational costs related to crewing and shipping. These include the following:

- Navigation to and from the terminal is relatively complex, characterized by the length of the voyage, the presence of dynamic morphological features in the Geba River, a lack of reliable hydrographic survey data, a history of vessel groundings in the Geba River estuary, and tidal restraints at various locations in the channel during vessel departures due to the size of vessels Itafos plans to utilize (larger than any vessel currently navigating the Geba River to the port of Bissau). As a result of these factors, marine pilots, employed by Itafos, will be used as a risk mitigation measure to guide vessels during arrival and departure from Mineral Terminal. Given the length of the Geba River, navigating to and from the terminal will be demanding from a resource perspective; on average, 5 hours will be required for arrival and 14 hours for departure.
- The use of a single shiploader requires the vessel to warp at the terminal. Warping requires tugs to come alongside the vessel and hold it on berth while the lines are slackened, and the vessel moves itself ahead or astern on the berth utilizing its main engine. This operation is high risk and very complex for a vessel's crew due to the need to operate mooring lines in a dynamic environment. Warping also increases the time needed to load vessels and greatly increases the chances of an accident (allision, collision, or vessel breakaway) when compared to a terminal that does not require warping. It is anticipated that 4 to 6 warping maneuvers may be required per vessel (depending on size), and each maneuver will take about one hour to complete. The simulation study completed in 2018 indicated that vessel warping should have an operational limiting current speed of 1.5 m/sec (at least at commencement of operations).
- Due to the above factors, tugs will be needed 24 hours per day while the vessel is at berth in case of vessel breakaway, and warping will be undertaken under the supervision of the pilot / loadmaster. The pilot / loadmaster will actively manage all aspects of the loading of the vessel 24 hours per day. A pilot is experienced in vessel handling utilizing tugs and the unique operational conditions associated with the site. In contrast, ship masters do not have experience in vessel berthing, unberthing, and warping operations with tugs. It is also important to note, as the vessel owner's representative, the ship master's primary focus relates purely to the interests of the owner. There is often a different focus in safety and operations between the terminal and the vessel.
- Given that warping will occur multiple times per vessel, and the significant rate at which tidal conditions change at Ponta Chugue, the vessel's crew are needed to handle the vessel's lines 24 hours per day while it is loading. Note, constant line handling increases the chances of serious injuries or death at the terminal due to line handling mishaps (broken fingers, strained muscles, etc.), line snapping incidents, and worker-overboard occurrences.
- A significant portion of the operating costs associated with above items is generally due to the crew costs needed for pilotage and tugs, and these crew costs can be correlated to vessel turnaround time.
- It is understood that shipping rates will be based upon a negotiated guaranteed loading rate (tonnes per hour) for the terminal, which over a relatively short period of time will be adjusted through the shipper's experience at the terminal to include the influence of weather. As such, the shipping rates will also be correlated to the vessel turnaround times.
- Vessel turnaround time at the berth will be influenced by:
  - warping, as each warping maneuver requires shiploader shutdown
  - current velocity restrictions on warping, which prevent warping when currents exceed 1.5 m/s
  - precipitation, which necessitates shiploader shutdown (note: warping and the associated current restrictions increase the occurrence of precipitation related shutdowns, as they increase the overall duration of loading)
  - tidal current restrictions on berthing and deberthing (flood tide with currents less than 1.5 m/s), which prevent vessels from leaving the terminal until a particular tidal stage is achieved

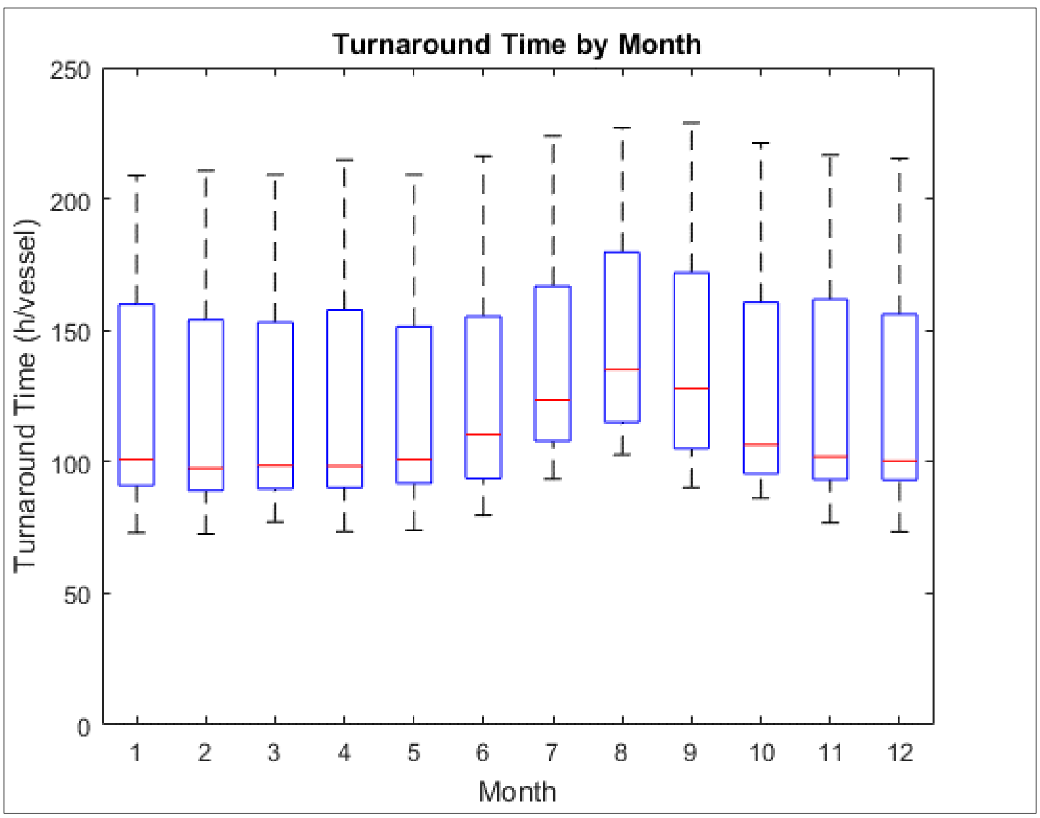
- the overall number of shiploaders and the nominal capacity of the shiploaders; faster shiploaders reduce turnaround time
- the capacity of the stockpile (e.g., smaller stockpiles are subject to increased occurrences of being emptied, which slows the shiploading rate; smaller stockpiles are also subject to increased occurrences of being full, which can impact overall throughput)
- vessel inter-arrival time at anchorage (there can be significant variability associated with each vessel's arrival time at anchorage)
- the above factors can contribute to congestion at the anchorage, where a vessel is waiting for a prior vessel to depart before sailing inbound to the berth.

The results of Baird's operational model (Baird, 2019a), which simulates operations at the terminal taking into account the results of the navigation simulations, are provided below to report the turnaround time for the base case. Three different cases have been examined to date to illustrate the impact of various items on turnaround time. For all cases, a fleet mix distribution of 20% Handysize, 20% Handymax, and 60% Supramax has been assumed.

18.13.6.1 Base Case

Figure 18-18 illustrates anticipated vessel turnaround time per month for the base case: a single shiploader at a rate of 750 t/h; a 60,000-tonne stockpile; and no minimum stockpile level.

Figure 18-18: Base Case Turnaround Times by Month



Source: Baird, 2019

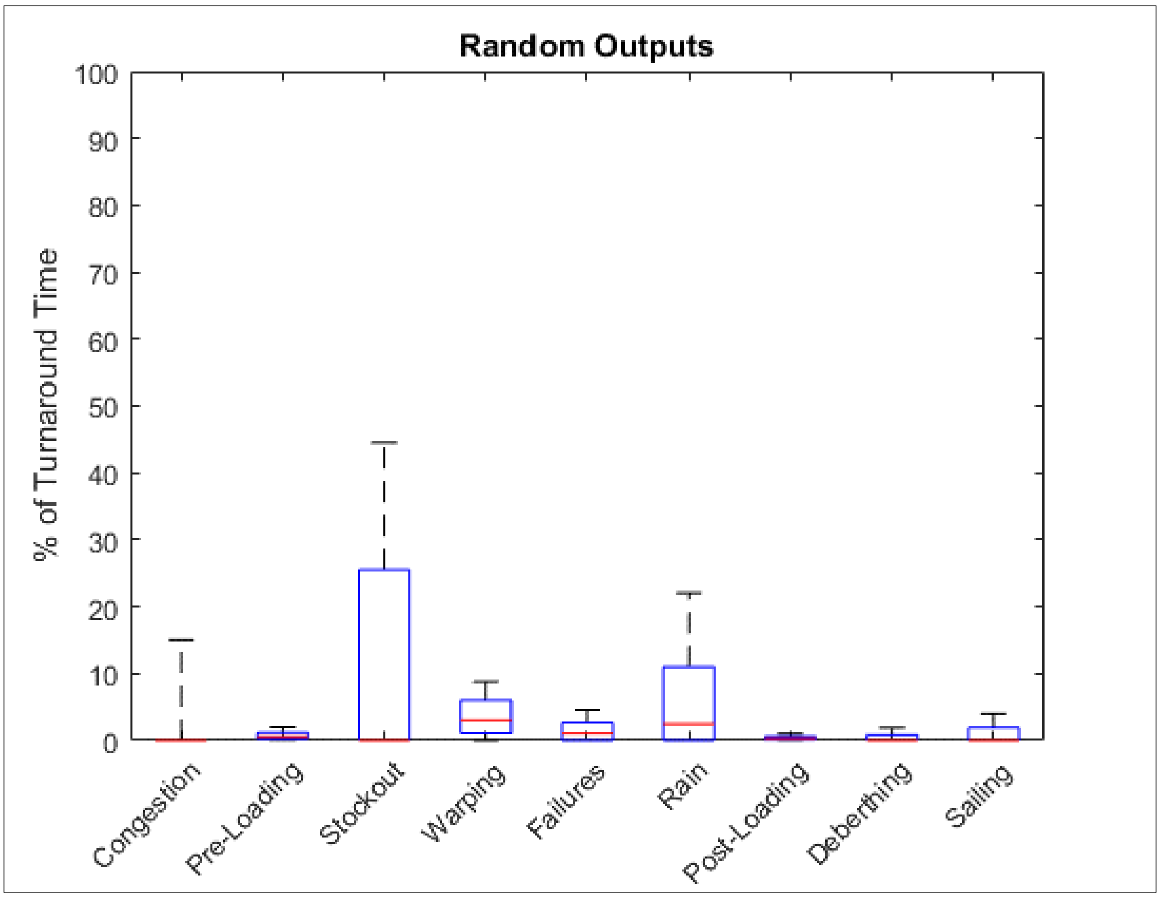
Note: The box represents the 25th and 75th percentiles of turnaround time and the whisker range is from 10% to 90% percentile. The red line is the median value.

Figure 18-19 represents the percentage of turnaround time associated with random variables in the model for the base case scenario, such as Mineral Terminal congestion, warping delay due to currents, deberthing delay due to currents, rain, and empty stockpiles. If the model were run without these random variables, all runs would produce the same result.

Figure 18-20 shows the percentage of turnaround time associated with the combined total of all random variables per month.

As can be seen from the figures, turnaround time is significantly influenced by the slow shiploading that results from empty stockpiles (largest of random variables). In addition, turnaround time is generally longer during the rainy season; median turnaround times generally range between 100 and 110 hours per vessel from November through June and 110 to 130 hours per vessel from July to October[2]. However, turnaround times can be significantly longer, as shown in the graphs.

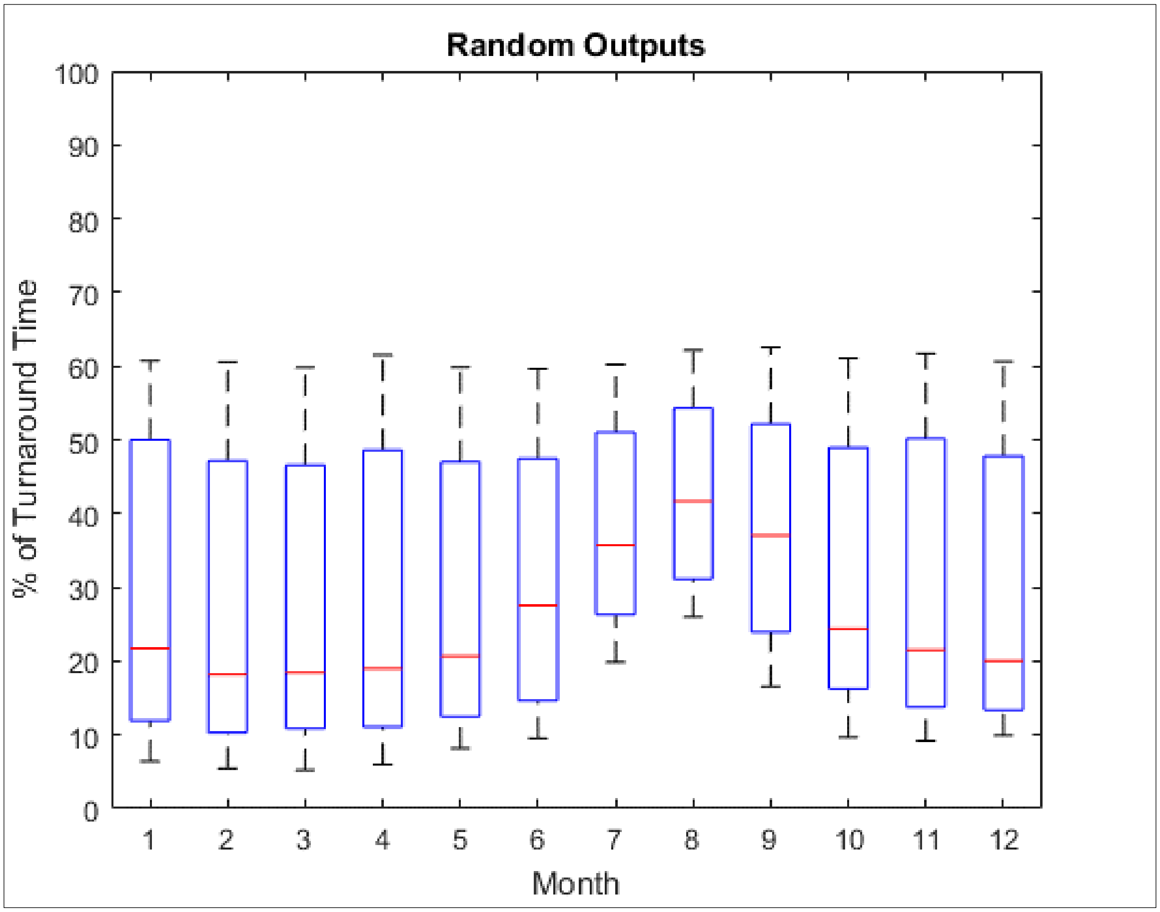
Figure 18-19: Base Case – Percentage of Turnaround Times from Individual Random Variables



Source: Baird, 2019

[2]The plots represent a distribution of vessel sizes visiting the terminal; 20% 30s, 20% 40s and 60% 60s.

Figure 18-20: Base Case – Percentage of Turnaround Times from Total Random Variables



Source: Baird, 2019

19 MARKET STUDIES AND CONTRACTS

19.1 Methodology

This analysis assesses the medium-term (2023-27) and long-term (2028-2040) phosphate outlook. The medium-term outlook is based on five-year demand and supply projections. The long-term outlook is based on demand forecasts and estimates of capacity and capital requirements needed to meet projected demand.

Demand forecasts are defined as the projected demand for downstream fertilizer and non-fertilizer products that are produced with phosphoric acid. Fertilizer products include the leading solid high-analysis products (DAP/MAP/NP/NPS/TSP) as well as other lower-analysis solid and leading liquid fertilizer products (NPK/PK/SPA/APP). The analysis excludes single superphosphate because it is not produced with phosphoric acid. Downstream non-fertilizer products include feed phosphate and purified acid (used for a variety of industrial and fertilizer applications). Some of the demand forecasts rely on recent CRU five-year and long-term projections. Potential medium-term and long-term demand accelerators are also highlighted in the analysis.

Medium term supply projections define effective phosphoric acid capacity as the lower of nameplate capacity or 105% of highest production achieved from 2010 to 2020. Phosphoric acid projects and expansions expected online by 2027 are included. Effective global phosphoric acid operating rates required to meet projected demand are then calculated.

Long-Term demand forecasts by nation and region are estimated using compound annual growth rates (CAGR) for two periods. These forecasts are calibrated or cross-checked with long-term fertilizer demand projections from IFA and CRU. The analysis does not speculate on what projects will get developed after 2027. Rather the analysis simply estimates the new capacity and capital in 2022 U.S. dollars that are required to meet projected demand in 2040.

Rock price forecasts are based on the historical relationship between the price of rock and the price of DAP FOB Jorf Lasfar. The price of DAP was forecast using a statistical model as a guide, and the model and statistical results are described in the price forecast section of the report.

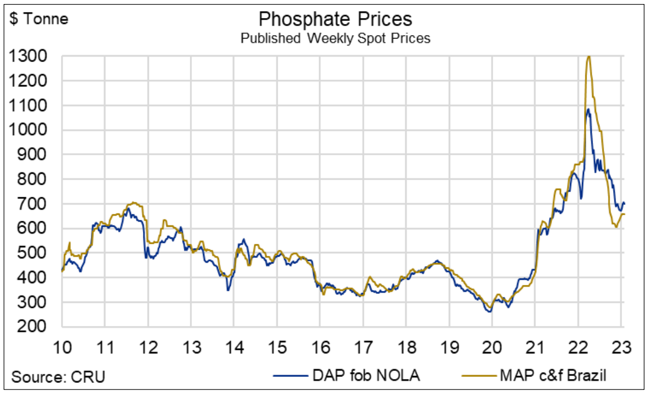
19.1.1 Current Phosphate Situation

19.1.2 Phosphate Prices and Market Developments

Phosphate prices surged to record highs in 2022 (see Figure 19-1). The price of MAP cfr Brazil peaked at $1,300/t and the price of DAP FOB New Orleans (NOLA) barge peaked at nearly $1,100/t in April. Prices collapsed from these extraordinary levels during the last half of 2022 but continue to trade at elevated levels. The Brazil MAP price plunged to $600/t before moving up to $655-$660/t in February 2023. The NOLA DAP price dropped to $670/t before increasing to $700/t in February 2023.

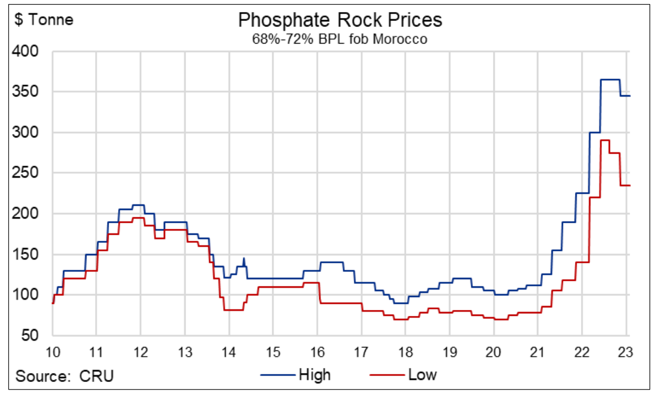
The price of 72% BPL (bone phosphate of lime) phosphate rock FOB Morocco also surged to a peak of $365/t in June 2022 (see Figure 19-2). That was about $100/t less than the 2008 peak, however. Contract prices stayed at this level until November when values dropped only $20/t to $345/t. OCP's strategy apparently is to maintain a price premium for its high-quality rock at the expense of losing market share. The price of 68% BPL rock FOB Morocco climbed to $290/t in June and then dropped $55/t in a couple of steps down to $235/t by November.

Figure 19-1: Phosphate Prices 2010-23



Source: CRU, 2022

Figure 19-2: Phosphate Rock Prices



Source: CRU, 2022

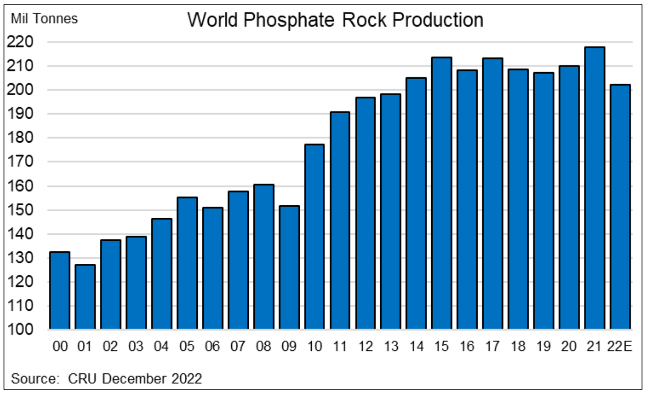
The rise and fall in phosphate prices were due to a combination of fundamental developments:

- Global demand surged in 2020 and 2021 in response to the climb in agricultural commodity prices. Phosphate demand outside China/East Asia surged 8.3% or 2.55 Mt P_2O_5 in 2020 and increased another 2.9% or 0.95 Mt in 2021, according to the latest statistics from the International Fertilizer Association (IFA).
- Chinese exports dropped in 2022. The National Development and Reform Commission (NDRC) announced in October 2021 that it would restrict phosphate exports to insure adequate supplies at affordable prices for domestic farmers. That caused importers dependent on China to seek alternative sources of supply and fueled the fly-up in prices during the first half of 2022.
- Raw materials costs, namely sulfur and ammonia values, skyrocketed in 2022.
- There were muted supply responses in several countries to higher phosphate prices due to aging industries already running at high rates as well as unplanned outages.
- The outbreak of the Russia-Ukraine war on February 24, 2022 added to supply uncertainties. Russian export supplies for the year turned out greater than initial expectations, but the fear of restricted availability contributed to the rapid increase in prices following the invasion.

19.1.3 Phosphate Rock Production and Apparent Demand

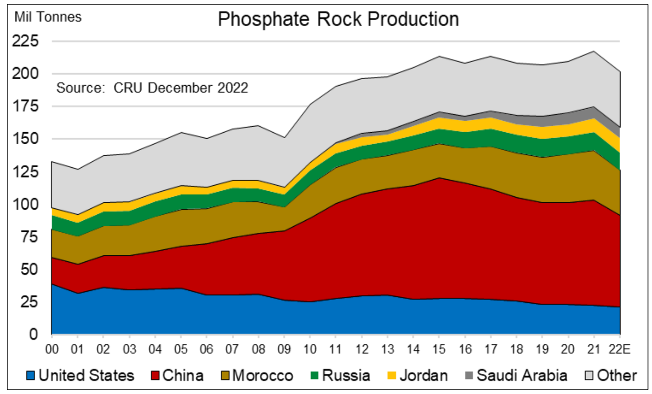
According to CRU estimates published in December 2022, global phosphate rock (concentrate) production and apparent demand (production plus imports minus exports) climbed to a record 217.7 Mt in 2021 and then dropped to just more than 202.0 Mt in 2022. China accounted for two-thirds of the estimated decline in 2022. Production and apparent demand have increased from about 130.0 Mt in 2000. However, output and use have plateaued at an average of about 210.0 Mt since 2015 (±3 Mt except for the 2022 estimate). Global phosphoric acid production has increased moderately since 2015, implying a drawdown of rock inventories worldwide.

Figure 19-3: World Phosphate Rock Production



Source: CRU, 2022

Figure 19-4: Rock Production by Leading Country



Source: CRU, 2022

Tables 19-1 and 19-2 list the top 10 phosphate rock producing and consuming countries during the last three years. China is the largest phosphate rock producing and consuming country, mining and consuming more than 78 Mt of phosphate rock per year and accounting for 37% of the global total for both during the last three years. Morocco ranks second with average annual output of 35.5 Mt and claiming 17% of the total during 2019-21. Morocco's rock use is less given the large amount of rock exports. The same holds true for Jordan, Egypt and Peru. India and the United States use more phosphate rock than produced. The top five producing countries accounted for more than three-quarters of global production and the top 10 claimed nearly 90% of the total during the last three years.

Table 19-1: Top 10 Rock Producing Countries (kt)

Rank	Country	2019-21 Average	Cumulative Production	Share	Cumulative Share	CAGR 2000-21
1	China	78,718	78,718	37%	37%	6.8%
2	Morocco	35,530	114,248	17%	54%	2.7%
3	United States	24,237	138,485	11%	65%	-2.5%
4	Russia	13,721	152,206	6%	72%	1.2%
5	Jordan	8,731	160,937	4%	76%	2.9%
6	Saudi Arabia	7,710	168,647	4%	80%	na
7	Egypt	5,523	174,170	3%	82%	8.9%
8	Brazil	4,935	179,104	2%	85%	0.4%
9	Tunisia	3,651	182,755	2%	86%	-3.6%
10	Peru	3,253	186,008	2%	88%	36.6%
	Other	25,536	211,544	12%	100%	0.5%
	Total	211,544		100%		2.4%

Source: CRU December 2022

Recent statistics mask a few important production trends. After increasing sharply from 2000 to 2015, annual Chinese production dropped 12.2 Mt from a peak of 92.8 Mt in 2015 to 80.6 Mt in 2021. CRU estimates that Chinese rock output dropped another 10.3 Mt to 70.3 Mt in 2022. After developing the largest phosphate industry in the world, it appears that the National Development and Reform Commission (NDRC) is implementing policies to restructure the domestic industry and prioritize domestic fertilizer and industrial needs over exports.

Table 19-2: Top 10 Rock Consuming Countries (kt)

Rank	Country	2019-21 Average	Cumulative Production	Share	Cumulative Share	CAGR 2000-21
1	China	78,420	78,420	37%	37%	7.7%
2	Morocco	26,832	105,251	13%	50%	4.5%
3	United States	25,582	130,834	12%	62%	-2.3%
4	Russia	12,298	143,131	6%	68%	2.9%
5	India	9,095	152,226	4%	72%	3.0%
6	Saudi Arabia	8,870	161,097	4%	76%	na
7	Brazil	6,825	167,922	3%	79%	0.9%
8	Jordan	4,482	172,404	2%	81%	3.3%
9	Tunisia	3,821	176,225	2%	83%	-2.7%
10	Egypt	2,569	178,794	1%	85%	7.3%
	Other	32,750	211,544	15%	100%	0.5%
	Total	211,544		100%		2.4%

Source: CRU December 2022

U.S. phosphate rock production continues to trend down. Output has dropped 2.5% per year since 2000. The decline is due to the depletion of mineral reserves over time. The United States ranked as the largest phosphate rock producer at the turn of the century, accounting for 30% or global output and topping the second largest producer (Morocco) by nearly a factor of two. The United States still ranks as the third largest producer today but accounts for just 11% of world production.

Annual output in Morocco has increased from about 22 Mt in 2000 to an average of 37 Mt during the last three years. The latest U.S. Geological Survey phosphate rock reserve estimates indicate that Morocco possesses nearly 70% of global rock reserves.

Table 19-3: Phosphate Rock Reserve Estimates (Mt)

Rank	Country	Reserves	Cumulative Reserves	Share	Cumulative Share
1	Morocco	50,000	50,000	69%	69%
2	Egypt	2,800	52,800	4%	73%
3	Algeria	2,200	55,000	3%	76%
4	China	1,900	56,900	3%	79%
5	Syria	1,800	58,700	3%	82%
6	Brazil	1,600	60,300	2%	84%
7	Saudi Arabia	1,400	61,700	2%	86%
8	South Africa	1,400	63,100	2%	88%
9	Australia	1,100	64,200	2%	89%
10	Finland	1,000	65,200	1%	91%
11	Jordan	1,000	66,200	1%	92%
12	United States	1,000	67,200	1%	93%
13	Russia	600	67,800	1%	94%
14	Kazakhstan	260	68,060	0%	95%
15	Peru	210	68,270	0%	95%
	Other	3,730	72,000	5%	100%
	Total	72,000			

Source: USGS Mineral Commodity Summaries, January 2023.

Several countries have emerged as significant phosphate rock producers during the last twenty years or so. Jordan has nearly doubled rock output this century. Egypt has increased rock output from slightly more than 1 Mt in 2000 to 6.5 Mt in

2022. Saudi Arabia and Peru produced no material quantities of rock in 2000, but output climbed to 8.4 Mt and 4.1 Mt, respectively, in 2022, according to CRU estimates.

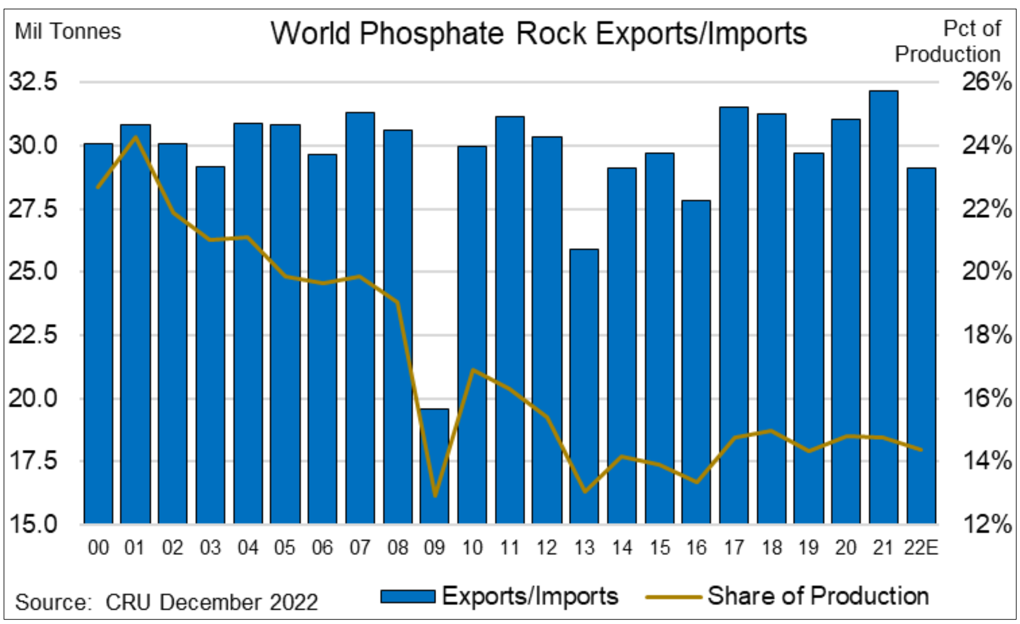
19.1.4 Phosphate Rock Trade

The production of finished phosphate products takes place in chemical complexes that range from fully integrated to non-integrated operations. Fully integrated operations mine phosphate rock, burn sulfur and may even manufacture ammonia to produce finished phosphate products. Non-integrated producers purchase phosphate rock or phosphoric acid as well as other inputs to fabricate final products.

India is a good example. The second largest phosphate consuming country possesses only small deposits of phosphate rock. Therefore, India relies on imports for nearly all of its phosphate needs. The country has diversified sources by stage of processing, importing raw materials (rock and sulfur), intermediate products (phosphoric acid and ammonia) as well as finished products (DAP/NP/NPS/NPKs).

There is significant trade in the raw material (phosphate rock) as well as the intermediate product (phosphoric acid). The chart shows that phosphate rock trade has stayed relatively stable at approximately 30 Mt per year. That makes sense in that non-integrated complexes typically operate and have consistent demand over time.

Figure 19-5: World Phosphate Rock Exports/Imports



Source: CRU, 2022

There have been some changes such as the closure of a few non-integrated operations (e.g., MissPhos Pascagoula) and switching from domestic to mostly imported rock by others (e.g., Mosaic Uncle Sam). Nevertheless, trade as a percentage of production has trended downward from more than 30% in 2001 to 14% in 2022.

Tables 19-4 and 19-5 show the top 10 phosphate rock exporting and importing countries. They highlight the dominance of Morocco and India. Morocco accounts for one-third of total exports, and the top five exporting countries account for nearly 80% of global exports.

India imports more than one-quarter of global phosphate rock trade. There are many more phosphate rock importing than phosphate rock exporting countries, but the top five importing countries claim almost one-half of world phosphate rock trade. The United States ranks third among importers with nearly all phosphate rock imports originating in Peru and purchased by Mosaic from their JV Miski Mayo operations for their Louisiana and Florida chemical plants.

Table 19-4: Top 10 Phosphate Rock Exporting Countries (kt)

Rank	Country	2019-21 Average	Cumulative Exports	Share	Cumulative Share
1	Morocco	10,108	10,108	33%	33%
2	Jordan	4,913	15,021	16%	49%
3	Peru	3,759	18,779	12%	61%
4	Egypt	3,375	22,154	11%	72%
5	Russia	2,314	24,468	7%	79%
6	Algeria	1,437	25,905	5%	84%
7	Togo	1,209	27,114	4%	88%
8	Syria	617	27,731	2%	90%
9	Kazakhstan	609	28,340	2%	92%
10	Senegal	486	28,826	2%	93%
	Other	2,141	30,967	7%	100%
	Total	30,967		100%	

Source: CRU December 2022

Table 19-5: Top 10 Phosphate Rock Importing Countries (kt)

Rank	Country	2019-21 Average	Cumulative Imports	Share	Cumulative Share
1	India	8,098	8,098	26%	26%
2	Indonesia	2,440	10,539	8%	34%
3	United States	2,360	12,899	8%	42%
4	Brazil	1,939	14,838	6%	48%
5	Mexico	1,743	16,582	6%	54%
6	Lithuania	1,407	17,989	5%	58%
7	Poland	1,078	19,067	3%	62%
8	Turkey	1,018	20,086	3%	65%
9	Belgium	804	20,890	3%	67%
10	Norway	718	21,607	2%	70%
	Other	9,360	30,967	30%	100%
	Total	30,967		100%	

Source: CRU December 2022

19.2 Medium- and Long-Term Demand

In the medium term (2023-27), new phosphate rock capacity and higher operating rates are required to meet projected demand during the next five years. Phosphoric acid demand, calculated from demand forecasts for downstream products, is projected to increase 6.36 Mt from 47.51 Mt in 2020 to 53.87 Mt in 2027.

Effective phosphoric acid capacity is expected to increase 4.46 Mt P_2O_5 and the global operating rate is projected to increase 500 to 600 basis points from 86% in 2020 to 91%-92% to meet projected demand during the next five years.

Assuming a 92% recovery rate, an additional 21.6 Mt of K-10 phosphate rock will be required to produce an additional 6.36 Mt P_2O_5 of phosphoric acid.

In the long term (2028-2040), demand is projected to increase 1.4% per year or 10.98 Mt P_2O_5 from 53.87 Mt in 2027 to 64.84 Mt in 2040. Fertilizer demand is forecast to increase 1.3% per year or 8.11 Mt P_2O_5. Non-fertilizer demand is forecast to increase at a faster pace of 2.4% per year or 2.86 Mt. Non-fertilizer demand is expected to grow at a faster pace largely due to rapid growth in purified phosphoric acid demand for the production of technical mono ammonium phosphate (tMAP). How battery technology evolves is still uncertain, but the growth of lithium iron phosphate (LFP) battery demand in electric vehicles may also potentially increase demand.

These estimates do not include the capacity or capital needed to replace phosphate rock mines that exhaust reserves during the forecast period. For example, several other U.S. mines are expected to exhaust reserves during the 2028-40 forecast period. In some cases, no viable reserves exist. In other cases, the permitting of viable reserves is uncertain.

Non-fertilizer demand is expected to grow at a faster pace largely due to rapid growth in purified phosphoric acid demand to produce technical MAP (tMAP). tMAP is used mostly in water soluble fertilizers today, but tMAP demand for the production of lithium iron phosphate (LFP) batteries for electric vehicles (EV) and storage is projected to grow exponentially during this period. How battery technology evolves is still uncertain, but the growth of LFP battery demand is a positive if not necessarily a game-changing demand development.

In North America, the rapid growth of renewable diesel (RD) and sustainable aviation fuel (SAF) production is expected to accelerate oilseed and phosphate demand growth during the next five years. Battery technologies are not well suited for long distance trucking or air transportation due to charging requirements and weight. As a result, trucking companies and airlines are turning to biofuels to reduce carbon footprints and meet new state regulations and corporate ESG objectives.

19.3 Rock Price Forecasts

19.3.1 Methodology

Phosphate rock price forecasts are based on the historical relationship between the price of phosphate rock the price of DAP FOB Jorf Lasfar in Morocco. Regression modeling demonstrates the price of phosphate rock and the price of DAP are correlated over time.

The regression analysis, shown in Table 19-17, used quarterly data from Q1 2010 to Q3 2022 (51 observations). The model hypothesized that the price of rock this quarter was a function of the price of DAP during the previous quarter. The R^2 was 0.92 indicating that variations in the price of DAP explained 92% of the variations in the price of rock during this period. The standard error was 17 resulting in a 68% prediction interval of approximately ± $17 per tonne of rock. The DAP coefficient was .336 and was statistically different from zero (t-statistic of 23.47) indicating that if the price of DAP increased (decreased) $10 per tonne then the price of rock typically increased (decreased) $3.36 per tonne in the subsequent quarter.

Phosphate rock price forecasts were then calculated using the Morocco Phosphate Rock Price (MRRC) and CRU International Ltd (CRU) Diammonium Phosphate price (DAP) forecasts using this statistical relationship (Figure 19-6).

Another statistical model was used by MRRC to provide guidance in making long term DAP price forecasts. This model utilized quarterly statistics from 2014 Q1 to 2022 Q3 (35 observations). The regression model hypothesized that the price of DAP FOB Jorf Lasfar vessel was a function of the prices of sulfur and ammonia FOB Mideast, the price of the front month futures corn price, and the phosphoric acid operating rate of the Moroccan industry (Figure 19-7).

The results of this model also are sound statistically. The R^2 value of 0.97 indicates that variations in the explanatory variables noted above explained 97% of the variation in the DAP price during this period. The standard error was 36 and implies a 68% prediction interval of about ± $36 tonne DAP.

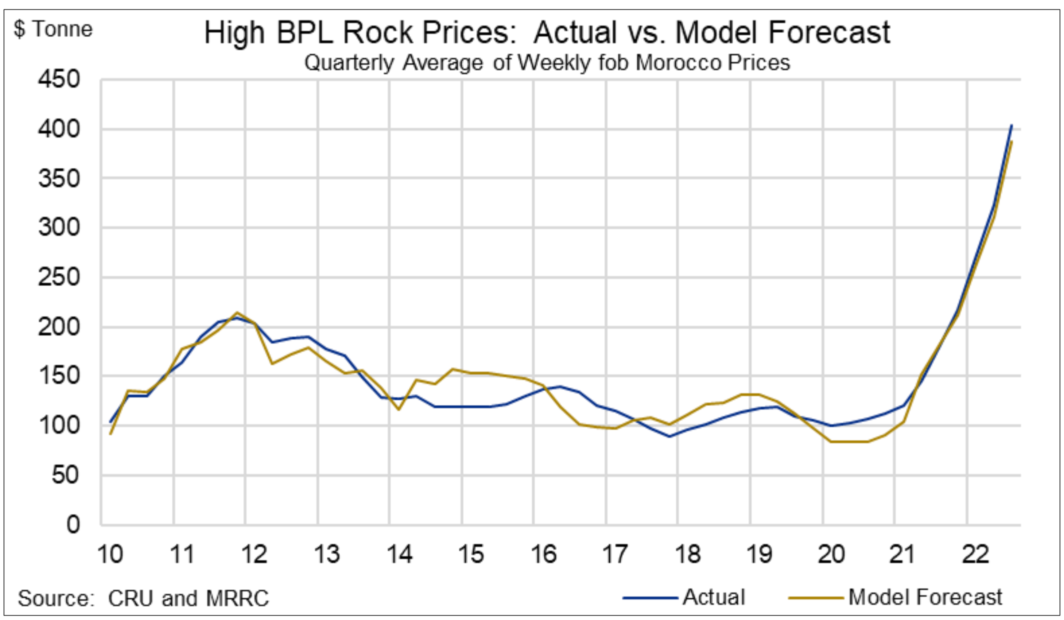
All explanatory variables in the DAP model have the correct expected signs, and all coefficients except the one for the Moroccan operating rate are statistically different from zero at the 0.95 level of confidence (i.e., t-statistics > 2.0).

Table 19-6: Rock-DAP Price Regression Results

Regression Statistics									
Multiple R	0.96								
R Square	0.92								
Adjusted R Square	0.92								
Standard Error	17.0								
Observations	51								
ANOVA									
	df	SS	MS	F	Significance F				
Regression	1	159,243	159,243	550.7	2.64E-28				
Residual	49	14,168	289						
Total	50	173,411							
	Coefficients	Standard Error	t Stat	P-value	Lower 95%	Upper 95%	Lower 95.0%	Upper 95.0%	
Intercept	-16.956	7.437	-2.28	0.03	-31.901	-2.011	-31.901	-2.011	
DAP Price	0.336	0.014	23.47	0.00	0.307	0.365	0.307	0.365	

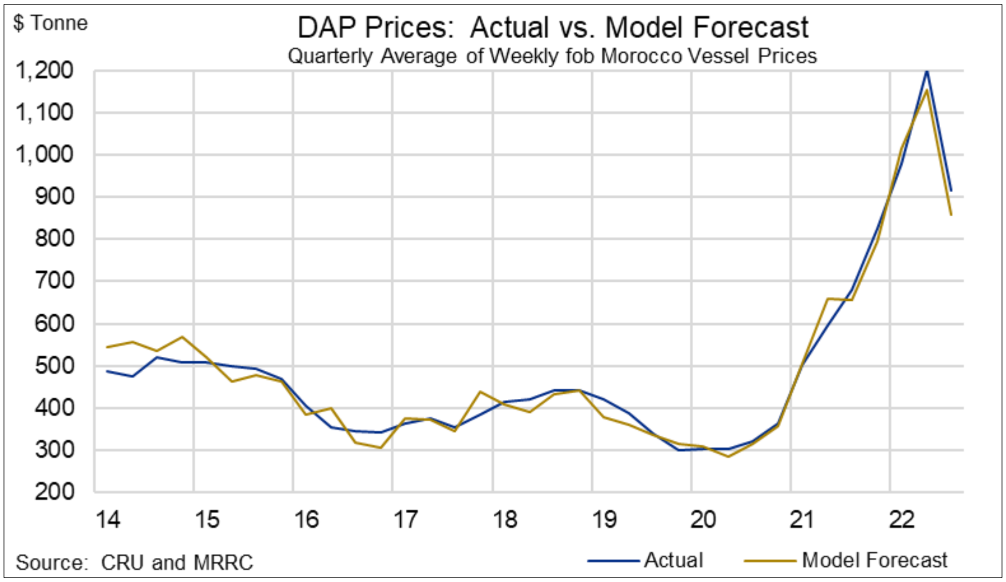
Based on the relationship between these variables during this period, the model indicates that if the front month futures corn price increases (decreases) 100 cents per bushel, then the price of DAP FOB Morocco will increase (decrease) about $34 per tonne. If the price of sulfur FOB Mideast decreases (increases) $50 tonne, then the price of DAP FOB Jorf Lasfar typically will decrease (increase) $42 per tonne. If the price of ammonia FOB Mideast decreases (increases) $50 tonne, then the price of DAP FOB Jorf Lasfar typically will decrease (increase) $24 per tonne. Comparisons between modeled compared to actual pricing are shown in Figures 19-6 and 19-7.

Figure 19-6: Rock–Prices – Actual vs. Model Forecasts



Source: CRU and MRRC, 2022

Figure 19-7: DAP Prices: Actual vs. Model Forecasts

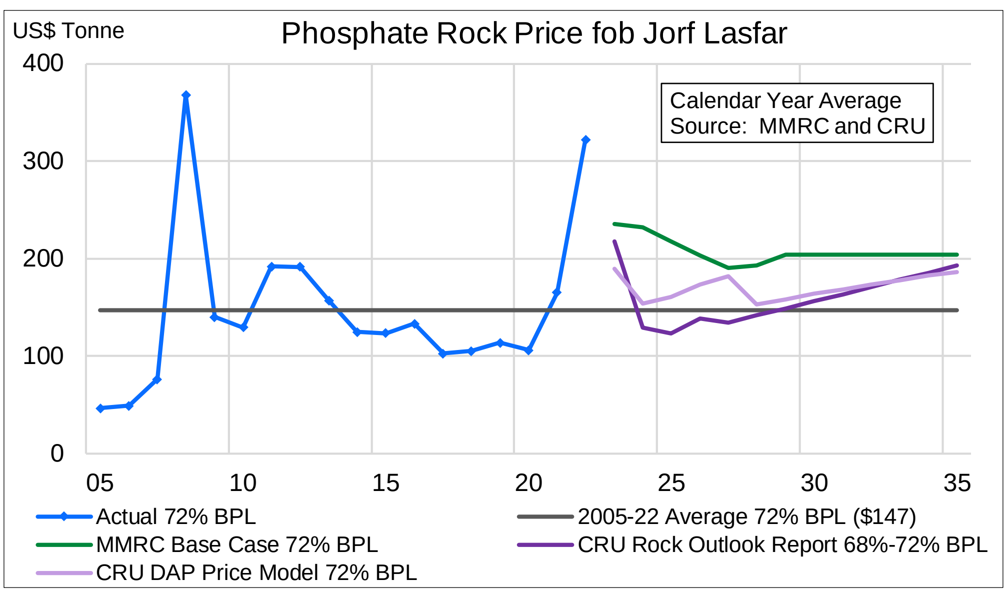


Source: CRU and MRRC, 2022

19.3.2 Phosphate Rock Price Forecasts 72% BPL Rock FOB Jorf Lasfar Morocco

MRRC phosphate rock price forecasts are derived from its long-term DAP price forecasts and the statistical relationship between rock and DAP prices. The average DAP price forecast from 2023 to 2035 is $668/t FOB Jorf, and that translates into an average rock price of $207/t for this period (see Figure 19-8 and Table 19-7).

Figure 19-8: Phosphate Rock Price Forecasts



Source: CRU and MMRC, 2022

Table 19-7: Phosphate Rock Price Forecasts

$ Tonne fob Jorf Lasfar	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	Average
MMRC Base Case DAP	753	741	698	655	618	624	656	656	656	656	656	656	656	668
MRRC Base Case Rock (72% BPL Rock)	236	232	218	203	190	193	203	203	203	203	203	203	203	207
CRU DAP	614	509	527	565	591	505	521	537	552	567	580	593	605	559
CRU Rock from DAP Model (72% BPL)	189	154	160	173	182	153	158	163	169	173	178	182	186	171
CRU Rock from Outlook Report (68%-72% BPL)	218	129	123	138	134	141	149	156	163	171	178	185	193	160

Source: MRRC and CRU

The average CRU DAP price forecast for this period is \$559/t FOB Jorf, and that corresponds to an average rock price of \$171/t. That is \$11/t greater than the CRU rock price forecasts for this period from its Phosphate Rock Market Outlook reports published in June and September 2022.

19.4 Contracts

The company has not entered into any contracts.

19.5 Comments on Market Studies and Contracts

The QP has reviewed the information provided by Itafos and confirms it can be used in the financial model. It is acceptable to use the DAP price for Jord LasFar in Morocco as the phosphate markets are driven by contracts and there isn't a publicly disclosed spot price. Phosphate from the Farim deposit would be considered representative as this best reflects market pricing in the absence of contracts for the output from the project.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Considerations

20.1.1 Baseline and Supporting Studies

Environmental studies were conducted on the project between 2011 and 2015 in support of an Environmental and Social Impact Assessment (ESIA) of the project. The studies took into consideration the mine area, product transport route, and Mineral Terminal (KP, 2015a).

Additional baseline studies were conducted from 2016 to 2019 in the areas of meteorology, air quality, noise, groundwater resources, and groundwater and surface water quality to establish an additional and contemporary pre-development baseline record that can be used for comparison in future monitoring programs. A summary of the scope of the baseline studies for each subject area is provided in Table 20-1. A discussion regarding the project's ESIAs and related approvals is provided in Section 20.5.

Environmental management plans were advanced for air quality, noise, and water in 2015 and 2016 (KP, 2015b, 2016a, 2016b, and 2016c), and Knight Piésold (KP) provided training to Itafos environmental technicians for these programs and assisted in their oversight and data management, including uploading the monitoring records into KP's web-based data management tool. Itafos' environmental technicians have implemented these monitoring programs since 2016, with some interruptions. In 2021, KP provided a status report to Itafos on these environmental monitoring programs, including a summary of the data collected to date, a data quality review, and recommendations for monitoring in future years in the absence of a construction decision on the project (KP, 2021). It was recommended that the meteorological station be serviced with new sensors, that groundwater level data continue to be collected from level loggers, and that hydrocensus work be completed. It was recommended that air quality, noise, and surface and groundwater quality programs be halted until a construction decision is made, as sufficient pre-development baseline data has been collected.

Additionally, updated social baselines, an updated land and asset survey for resettlement planning, and updated terrestrial and aquatic biodiversity studies are recommended when the project is advanced, as conditions may have changed since baseline data was collected in 2015.

Table 20-1: Summary of Environmental and Social Baseline Studies

Subject Area	Summary Description of Scope of Studies	Reference
Meteorology	A meteorology station operated nearly continuously at Farim between 2011-2016. An analysis of meteorology data was conducted in support of the 2015 feasibility study.	KP, 2015c
Air quality	Baseline measurements of particulate matter (PM), sulfur dioxide (SO_2), nitrogen oxides (NO_x) and dustfall collected at the mine in 2012 and from 2016-2018 to at representative locations at the mine site, along the transportation route and near the Mineral Terminal site.	Golder, 2013a; KP, 2015d and 2021
Noise	Noise measurements collected at receptor locations near the mine and transport route (2011-2013), and at the mine and Mineral Terminal (2015-2019).	Golder, 2014; KP, 2015e
Geochemistry	Geochemical testing of tailings and waste overburden, including static, kinetic, and radiological testing.	KP, 2015f, 2015g, 2017, 2018a, 2018b
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 Mineral Terminal site (metals and soil fertility parameters).	Golder, 2014; KP, 2015h
Surface water	Surface water sampling conducted over multiple wet and dry seasons at the mine site (2011-2013, 2015-2018)	Golder, 2014; KP, 2015i
Groundwater	Comprehensive groundwater investigations completed, and one dry season and wet season sampling campaign completed (2013). Additional wells installed at the mine site and pump tests conducted (2015-2018). Supplemental groundwater quality sampling conducted at select wells in the mine and Mineral Terminal areas (2015-2019).	Golder, 2012; KP, 2015i, 2015j, 2018c, 2018d, 2020, 2021
Aquatic ecology	Aquatic studies conducted in the River Cacheu and tributaries near the mine site in 2013, and in the River Cacheu and River Geba port site in 2015. River morphology studies conducted in the River Cacheu (2013).	Golder, 2013b and 2014; Aquatic Ecosystem Services, 2015
Terrestrial ecology	Terrestrial ecology studies conducted in the mine site area (2011-2015) and the Mineral Terminal site (2015).	Golder, 214; Hudson Ecology, 2015
Socioeconomics	Initial baseline studies including literature review, focus groups, household questionnaires (2011-2012). Household survey in support of a resettlement policy framework (2015); land and asset survey (2018). Traffic surveys in 2013 and 2015.	Tropica, 2011 and 2012; Golder, 2014; Eco Progresso, 2015a and 2015b; KP, 2015k

20.1.2 Physical and Biological Setting

20.1.2.1 Meteorological and Atmospheric Conditions

The climatic and seasonal variations are very distinct in Guinea-Bissau and follow the general West African climate conditions. It is hot and humid year-round with little fluctuation in average temperature. At the Farim climate station, mean monthly temperatures ranged from 21.9°C in January to 29.1°C in April for the period December 2011 – March 2015; maximum and minimum temperatures recorded over this same period were 42.8°C and 8.1°C, respectively (KP, 2015c). Monthly average relative humidity ranged from 49% in February to 92% in August between December 2011 and March 2015 (KP, 2015c).

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 predominantly 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 the north part of the country. Most 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, or 18 km/h (Golder, 2014). Extreme daily (24-hour) design precipitation depths at the Farim site, based on regional climate stations, were estimated to range from 94 mm for a two-year return period to 699 mm for a 250-year return period, while maximum annual precipitation was estimated as 2,152 mm (KP, 2015c). The maximum recorded average wind speed over a 10-minute period from the Farim weather station between December 2011 and March 2015 ranged from 3.5 m/s in November to 8.3 m/s in September (KP, 2015c).

Air quality data collected around the mine and Mineral Terminal sites indicates that the air quality is representative of a natural environment with low concentrations of anthropogenic gases. Particulate matter (measured at the mine site only) is elevated due to natural sources and wind erosion created by the Harmattan winds from the Sahara in the direction of the study region, particularly during the dry season of November to April (KP, 2015d). Other air quality parameters measured around the mine site during the baseline program (based on eight monthly datasets) were (Golder, 2014; KP, 2015d):

- Maximum nitrogen oxide (NOx) levels ranged from 8.43 µg/m^3 at Saliquenhe to11.68 µg/m^3 at Monsoa.
- Maximum nitrogen dioxide (NO_2) levels ranged from 4.58 µg/m^3 at the proposed plant area to 8.16 µg/m^3 at the Farim station.
- Maximum sulfur dioxide (SO_2) levels ranged from 1.44 µg/m^3 at the proposed plant area to 3.13 µg/m^3 at the Farim station.
- Maximum ozone (O_3) levels ranged from 83.77 µg/m^3 at Cansenhe to 120.60 µg/m^3 at Saliquenhe/Box Cut.
- The maximum NO_2 levels were much lower than the WHO annual mean guideline of 40 µg/m^3 and the maximum SO_2 levels were much lower than the WHO 24-hour mean value of 20 µg/m^3. The maximum O_3 level at Saliquenhe/Box Cut is higher than the WHO 8-hour mean guideline of 100 µg/m^3 but is less than the interim target of 160 µg/m^3 above which important health effects are noted (WHO, 2005).

The daytime and nighttime 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 LAeq 1 hour, respectively (Golder, 2014; KP, 2015e). Baseline noise surveys indicate that measured daytime noise levels are typically higher than the lowest measured nighttime noise levels. Daytime noise levels are most influenced by human activities. Noise levels increase around dusk due to the calling of crickets and toads, and steadily decline as the night passes.

20.1.2.2 Topography and Soils

Topographical variation in the mine area is relatively small, ranging from approximately 8 mamsl in the southwest to 55 mamsl in the north (Golder, 2014). The mine area is characterized as flat to gently undulating, dominated by the flat, low-lying valley through which the River Cacheu meanders in a generally westerly direction (Golder, 2014).

A soil survey evaluated the soil types and their respective agricultural potential within the mine footprint. The soils were grouped into five classifications according to the South African Taxonomical Soil Classification System: Avalon, Clovelly, and Hutton (all arable land), Katspruit (wetland), and Westleigh (forest). Most of the soil units defined at the mine were identified as having a high agricultural potential and arable land capabilities.

Land use types in the Mineral Terminal area consist of rice paddy, savannah, and coastal beaches.

20.1.2.3 Groundwater

The conceptual hydrostratigraphy of the regional area is as follows:

- An overburden layer comprising sands, clays and gravels, extends from the land surface to the absolute elevation of -30 to -40 mamsl. This unit can be considered an unconfined aquifer and is shown to be in limited hydraulic connection with the River 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 is better characterized as a calcareous clayey friable sandstone, justifying the low hydraulic conductivities of this layer.

Water supply drilling undertaken at the proposed worker camp located southeast of the mine site and at the Buredanfa Village site has identified a water-bearing sandstone unit beneath the limestone unit (KP, 2018).

Groundwater provides base flow to surface water bodies, including the River Cacheu and its tributaries (Golder, 2014).

The quality of groundwater collected by Golder (2014) and KP (2015e) in the mine site area is reflective of the undeveloped environment. Most of the samples collected met the WHO (2017) drinking water guidelines. 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 tributaries near the River Cacheu 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. This water will require treatment (iron and manganese removal and disinfection) before domestic consumption.

Hand-dug wells excavated into the shallow aquifer are numerous throughout the project area, as groundwater is the principal source of water. Shallow wells typically range in depth from 7 m to 27 m below ground level. Water levels respond to seasonal rainfall, dropping in the dry season and rising during the wet season. The intensive cropping of vegetables in irrigated horticultural gardens in local villages takes place during the dry season (January to May) and relies on wells that have been dug for this purpose. The wells are reported to dry toward the end of the dry season (Nomad Consulting and Africa-Wide Consulting, 2017).

20.1.2.4 Surface Water

20.1.2.4.1 Mine Area and the River Cacheu and Tributaries

The mine area is characterized by an undulating topography comprising lowland riparian areas of nearly zero elevation surrounded to the northwest by hilly areas with elevations of up to about 50 m.

The major surface water body in the area is the River Cacheu, a permanent water body about 300 m wide that lies adjacent to the mine site. This river is fed by the following tributaries (shown from left to right in Figure 20-1):

- Rio de Caur

- Rio de Cavaras Marinhos
- Rio de Bunja
- Rio de Banim.

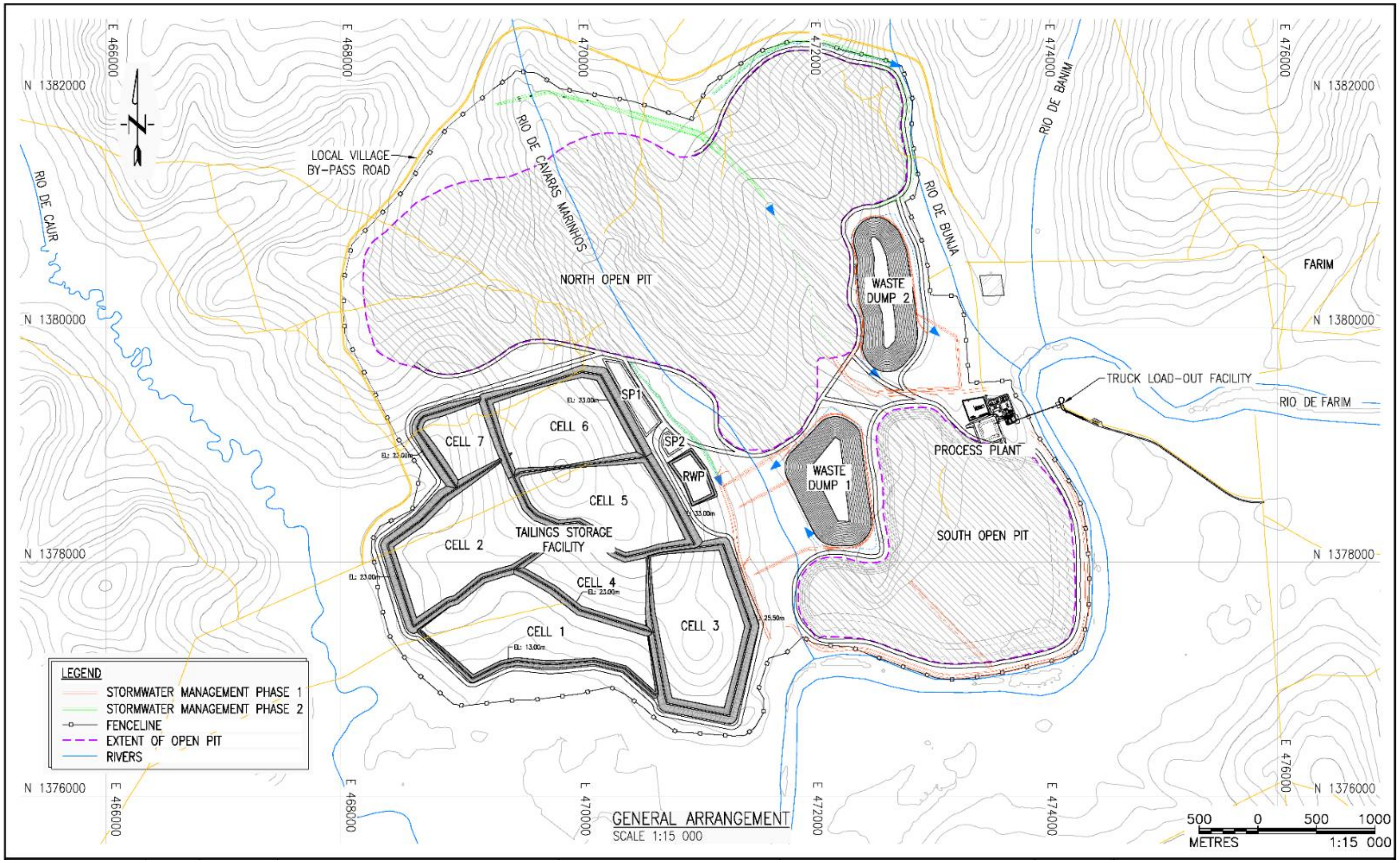
The River Cacheu is tidally influenced and is representative of an estuarine environment. The water level in the River Cacheu is affected by short- and longer-term tides with a daily range of about 2 m. The river tides also affect the water levels in the lower reaches of its tributaries. The upper reaches of the tributaries can be ephemeral during the dry season.

The lowland riparian areas are affected by tidal water level fluctuations, mainly over the mangrove coverage, as well as by seasonal inundations during the rainy season. The lowland area, which covers almost half of the South pit, can be completely inundated by water during the rainy season.

River Cacheu hydrodynamics was summarized as follows (from Golder 2014b):

- The River Cacheu in the reach from Farim to Cacheu is strongly influenced by the tidal regime (semi-diurnal); tides extend upstream of Farim even during the wet season, and influences some of the tributaries. Hydraulic analysis results show that the river currents are dominated by tidal conditions rather than runoff from surrounding upland areas.
- The variation in tide becomes dampened further upstream of Sao Vicente Bridge due to the channel geometry (width, depth, curvature) and also due to inflows from upland runoff. From available data measured from the July to August 2013 field program it was found that the maximum tide levels at Binta (immediately downstream of the mine site) are about 80 to 85% of the maximum tide levels at Cacheu.
- The tide range at Port Cacheu is estimated to be 2.8 m. The highest tide (spring tide) is estimated to be 2.9 m and the lowest tide (neap tide) is 0.1 m. At Binta, these tides are estimated to be 2.3 m (spring tide) and 0.1 m (neap tide).
- Maximum average velocities in the river range from 1.1 to 1.5 m/s according to simulated hydrodynamic conditions on the River Cacheu at Farim for both wet and dry seasons (Baird, 2012).

Figure 20-1: Water Features Around the Mine



Source: Knight Piesold, 2023

The geomorphology of the River Cacheu was summarized as follows (from Golder, 2014b):

- The River Cacheu estuary is a flooded river system resulting from a postglacial sea level rise (over the last 19,000 years BP), drowning the original river channel. Some examples of the drowned river channel are deep areas (up to 25 m) observed in the river channel between Farim and Cacheu.
- In both the wet and dry season, the riverbed sediments range from silt and clay at Farim to sand sized particles at the estuary near Bolor. This is consistent with literature and the understanding of the geomorphological development of the estuarine system.
- The riverbanks are composed of fine-grained sediments and are well-vegetated with mangroves and other vegetation. The fine-grained sediments are exposed at low tides and have the potential to be eroded through the action of waves and currents.
- The river morphology is currently dominated by the transport and deposition of fine-grained (predominantly silt, clay) sediments.
- Historical imagery indicates that no major channel changes (e.g., avulsions, meander cutoffs) occurred from 1979 to 2012 between Farim and Cacheu.

The River Cacheu is characterized by brackish waters with high levels of conductivity, TDS and chloride. The water quality standard for total phosphorus is based on baseline concentrations and is elevated relative to published total phosphorus guidelines from other jurisdictions.

Nutrient concentrations were generally below method detection limits (MDLs) in the dry season with the exception of phosphate and ammonia (as N), which ranged from 0.025 to 0.052 and 0.05 to 0.12 mg/L, respectively. Concentrations of ammonia (as N) were above the Australian Tropical Estuaries Standard of 0.015 mg/L and concentrations of phosphate were above the standard of 0.02 mg/L.

Several metals had concentrations below MDLs in the dry season samples including cadmium, mercury and silver. There were no exceedances of guideline limits in the dry or wet seasons.

The River Cacheu water quality displays considerable variations between dry and wet seasons. Chloride levels range from 1,278 to 5,320 mg/L in the dry season and from 1.7 to 1,970 mg/L in the wet season. The mean pH is slightly lower in the wet season compared to the dry season (7.0 vs. 7.5). Mean concentrations of TDS as well as sulfate are lower in the wet season relative to the dry season.

20.1.2.4.2 Mineral Terminal Area and the River Geba

The Mineral Terminal site area is characterized by relatively flat topography that gently slopes to the south towards the River Geba. The wetlands areas are located along the floodplains, most of which have been converted to rice paddies, with only a band of mangrove remaining along the shorelines.

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 Ponte Chugue (Baird, 2015). 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 Mineral Terminal site consisted of fine mud with a depth of 5 to 8 m.

Water quality in the River Geba is characterized by brackish waters with high levels of TDS and chloride. Water quality data was compared to World Bank General Environmental Guidelines (1998), South African Water Quality Guidelines (1995), and WHO (2011) water quality standards. Elevated levels of aluminum, boron, manganese, chromium, iron, arsenic and molybdenum were noted. Only chromium, nitrate and phosphorus exceeded the South African marine water quality standards.

20.1.2.5 Aquatic Ecology

20.1.2.5.1 Regional Overview

Guinea-Bissau is situated in Tropical West Africa on the edges of the Guinea Current Large Marine Ecosystem (LME) and the Canary Current LME (Belhabib and Pauly, 2015). The coastline is relatively short, yet the continental shelf is large and shallow (Mendy and Lobban, 2013) which contributes to turbulent coastal waters. Tidal currents are strong, and the tidal range can reach up to 6 m. High rainfall and freshwater input during the rainy season creates highly turbid coastal waters. Estuaries are lined with mangroves which provide important habitat for juvenile fish and crustacea. Despite having good fishery resources, the fishery sectors remain largely undeveloped, and income from fisheries is largely derived from license fees received from foreign vessels.

The tropical estuaries of West Africa have a rich ichthyofaunal diversity with over 200 species recorded from open and blind estuaries and coastal lakes (Blaber, 2000). Species composition is influenced by the freshwater inputs during the wet season, with marine species using estuarine systems temporarily for feeding, spawning and shelter (Baran, 2000). Estuaries in West Africa, and in particular the mangrove habitats, play an important role in the life histories of many species, especially the juvenile phases of many important fishery species. Clupeids typically dominate the fish fauna of West African estuaries numerically. The dominant ichthyofaunal families occurring in West African estuaries include the *Ariidae* (seacatfish), *Bagridae* (bagrid catfish), *Carangidae* (jacks), *Cichlidae* (cichlids), *Clupeidae* (sardines and shads), *Elopidae* (Elops), *Gerreidae* (mojarras), *Haemulidae* (grunts), *Polynemidae* (threadfins), *Sciaenidae* (drums), and *Sphyraenidae* (barracudas) (Blaber, 2000).

The Gambia Estuary can be considered a suitable reference point for West Africa estuaries as it is one of the last aquatic ecosystems of the area that has not been affected by strong environmental or anthropogenic changes (Simier et al., 2006). It also has a near zero drainage gradient over 500 km, and therefore brackish waters and tidal floodplains with mangrove swamps occur over the last 180 km (Daget, 1960, in Simier et al., 2006) which is similar to that of the Cacheu and Geba estuaries in Guinea-Bissau.

20.1.2.5.2 Cacheu Estuary

The Cacheu estuary is in a natural state (Golder 2014e). The habitat along the length of the Cacheu estuary has few modifications or existing impacts and minimal clearing of mangroves which extend beyond Farim. Sediments near the mine site are dominated by fine silts and clays (up to 90%) along the fringing banks, with gravel found in deeper channel areas. No seagrass beds or corals were identified anywhere along the length of Cacheu estuary (Golder 2014e). The estuary has a well-defined channel 100 to 250 m in width and a low gradient with a 5 m drop over 160 km which contributes to the strong tidal influence stretching beyond Farim. Water temperature ranges from 27°C to 30°C along the entire length of the estuary during both dry and wet seasons with no thermal stratification. Salinity at Farim ranged from 2.4 parts per trillion (ppt) in the dry season to 7.2 ppt during the wet season. Dissolved oxygen near the mine site was greater than 4.3 mg/L during wet and dry seasons.

Both marine and freshwater phytoplankton and zooplankton were present in the water column around the mine site region of the Cacheu estuary. Surveys captured 26 species from between the mouth of the River and Farim, a larger geographic area than the current study area. Ten species of fish have been captured during baseline surveys, and additional three species were identified through interviews with artisanal fisherman.

Eighteen infaunal taxa were distinguished. Benthic invertebrate density was generally low but increased downstream. Fish tissue sample analysis indicated no contamination from anthropogenic sources, confirming the near natural state and health of the system (Golder 2014e). Methyl mercury was not detected in any of the fish tissues, indicating that mercury methylation is not likely to occur in this aquatic system.

Sediment quality is acceptable for aquatic life when compared to international aquatic ecological guidelines (SC&A, 2015b). None of the sediment quality samples collected in 2015 exceed the Australian and New Zealand Guidelines for Fresh and

Marine Water Quality (ANZECC, 2000) or the sediment quality guideline values by Jackson (2000). If the River Cacheu sediments were to be disturbed by the project, none of the heavy metals contained in the fine sediments would pose a risk to aquatic organisms or humans. The water is highly turbid near Farim but based on the diversity and assemblages of the aquatic fauna, was not limiting to the overall abundance of fish and plankton.

River Cacheu water quality met various international standards (South Africa, Australian, Canadian, World Bank and WHO) applicable to the protection of aquatic ecosystems (SC&A, 2015b). One exception to this was total suspended solids (TSS), which ranged from 64 to 77 mg/L, in exceedance of the World Bank guideline value of 55mg/L. Other guidelines for TSS optionally allow for a 10% variance from baseline as a guideline when generic numerical guidelines are not appropriate, such as when TSS levels are either very low naturally or are very high such as in the River Cacheu and River Geba systems.

20.1.2.5.3 River Geba

There is a paucity of information available on the ecology of the Geba Estuary. The estuary is extremely large, with a width of 10 to 12 km near Bissau. The tidal range is in the region of 5 m, and tidal currents can reach speeds of up to 3 knots (Agardy, 1997). Sediments are soft to very soft muds (van der Veer, 1995). No known protected areas exist on the Geba Estuary.

The estuarine waters at Ponta Chugue were highly turbid with a strong tidal current (>3 knots) and large tidal range. Benthic sediments were very fine muds with rocky substrata limited to the immediate area around Ponta Chugue. The remaining samples contained weathered rock and shell fragments (> 80% gravels).

Three species of fish were captured, two of which (*Arius latiscutatis* and *Pentanemus quinquarius*) were not captured in the Cacheu Estuary. Numerous Penaeids were also captured. Interviews with artisanal fishermen confirmed the difficult fishing conditions and indicated few species of estuarine fish are caught in the area. Despite low catches, based on the size and characteristics of the Geba Estuary, the ichthyofaunal diversity should in theory be high, with a greater presence of marine dependent species than the upper reaches of the Cacheu Estuary at Farim.

Grab samples of the substrate were collected at eight sites of varying depth around the Mineral Terminal site. Few of the samples contained any infaunal species, most probably due to the very fine and anoxic mud sediments present across most of the study area.

Water quality and sediment samples were also taken at five sites during the 2015 surveys adjacent to the proposed wharf. Although limited by the prevailing wind and wave conditions, coupled with large volume of water that is exchanged during each tidal cycle within the 7 km wide portion of the system, the samples did provide some indication of the conditions within the estuary.

Water temperature ranged from 28°C to 28.9°C and salinity ranged from 32 to 38 ppt in the dry season, due to the large volume of seawater that is exchanged with the twice daily tides.

No spatial variation could be determined between the five sites sampled for water quality, as they showed little variation in the results obtained, indicating the high level of mixing within the water column. TSS values in the River Geba exceeded the WHO guideline value and were higher compared to TSS levels in the River Cacheu (SC&A, 2015b).

Similarly, the River Geba also showed within some of the samples elevated levels of aluminum, arsenic, boron, chromium, iron, manganese, molybdenum, and uranium. This will require additional investigation but is possibly related to the natural geology of the source catchments of these systems. Most these metals exceeded the WHO Drinking water standards, while only chromium, iron, and nutrients (nitrate and phosphorus) exceeded the South African marine water quality standards (SC&A, 2015b).

River Geba sediment, based on five of eight grab samples collected showed some similarity to sediments in the River Cacheu. None of the main elements recorded exceeded the applicable protection of aquatic life guidelines.

Only three species of fish were captured in the Geba Estuary at Ponta Chugue, two of which (*Arius latiscutatis* and *Pentanemus quinquarius*) were not captured in the Cacheu survey. Numerous Penaeids were also captured in the beam trawl. The fish species included *Arius latiscutatis, Pseudotolithus elongatus* and *Pentanemus quinquarius*. The low catch rates were due to the limited time available for sampling and due to the difficult sampling conditions typical of the lower Geba Estuary (strong tidal currents; large tidal range; rough wind chop). Interviews with artisanal fishermen based at Chugue confirmed the difficult fishing conditions and indicated few species of estuarine fish (*Arius spp., Pomadasys spp., Liza / Mugil spp., E. fimbriata, I.africana, P. quinquarius, Pseudotolithus spp*.) are caught in gillnets in the area. Despite low catches, based on the size and characteristics of the Geba Estuary, the ichthyofaunal diversity should in theory be high, with a greater presence of marine dependent species than the upper reaches of the Cacheu Estuary at the mine site.

Eight sites of varying depth were sampled around the Mineral Terminal site for benthic infauna. Few of the grab samples from the Geba Estuary port site had infaunal species present. This is most probably due to the very fine and anoxic mud sediments present across most of the study area, and the presence of rock and large gravel at sites off Ponta Chugue which are not suitable habitat for infaunal species.

20.1.2.5.4 Fishing within the Aquatic Study Areas

No information is available on the fisheries in the Cacheu or Geba estuaries. However, based on an understanding of the regional fisheries, it is unlikely that any industrial or recreational fishing occurs near the project sites in the Cacheu and Geba estuaries. Artisanal fishing effort from the local subsector, however, is likely to be high, with a high reliance on subsistence fishing.

The coastal population of Guinea-Bissau does not have strong, long-standing fishing tradition (Baran & Tous 1999 in Campredon and Cuq 2001) and fishing has been an off-season activity for local farmers (Tvedten 1990, Chavance 2004 in Belhabib and Pauly 2015). Estuarine and riverine fishing effort is largely unknown, but it is estimated that 10,000 to 12,000 mostly foreign fishermen harvest coastal resources in the estuaries and along the coast, with the Bijagós Archipelago on the continental shelf and the River Cacheu being particularly important areas (Megapesca, 2010, Mendy and Lobban 2013). A 12 nautical mile (NM) zone adjacent to the coast is set aside for artisanal fishing (Mendy and Lobban 2013).

The artisanal sector has two distinguishable sub-sectors, the first being the local domestic fishery, and the second the migrant, mainly Senegalese (with growing Guinean participation) fishery. Domestic artisanal fishing is limited to coastal waters within the 12 NM zone, while the foreign migrant fishers travel more widely, beyond the 12 NM industrial exclusion zone. This migrant sector of the artisanal fishery accounts for 70-80% of the artisanal harvest (Megapesca 2010). This sector contributes significantly to the food security of the coastal communities; however, little quantitative data is currently available.

20.1.2.5.5 Fishing Activities on River Cacheu near the Mine Site

Numerous artisanal fishermen were observed during the field survey on the Cacheu estuary. The most common forms of fishing involved gillnetting and longlining.

Gillnets are attached to floating platforms strung between mangroves across the main estuary channel. Fishing occurs predominantly at night with fishermen returning with catch in the early morning. Gillnets are constructed of multifilament meshing of varying mesh sizes, but mesh sizes are generally large. Net lengths are in the region of 90 m. During the field survey 10 permanent floating structures used by fishermen were observed over a 12 km section downstream of Farim.

Longlines consist of a length of rope attached to either an anchor which is dropped in mid-channel and marked by a buoy, or the end is tied off to the mangroves and set out into the river channel. Each line has 150-200 hooks attached to a weighted bottom set line. Over one short stretch of river (<2 km) of the estuary more than 20 longlines were counted.

The main artisanal species (from interviews) include *Chrysichthys spp., Sphyraena spp. Pseudotolithus spp.* and *Pomadasys spp*. Fishermen fish all year round, however, the peak period is from May to June and from August to December when catches are highest. There is a local fishing association, the Farim Association, with approximately 66 members.

20.1.2.5.6 Fishing Activities on River Geba near the Mineral Terminal Site

Only one fisherman was observed actively fishing while conducting the field survey on the Geba estuary, however, several young fishermen were encountered at the landing beach at Chugue. Interviews with the local fishing representative indicated that fishing in this region of the Geba Estuary is difficult due to the strong currents and large tidal range. Longline fishing is therefore the main type of fishing which occurs in the area. Longlines are approximately 200 m in length with up to 300 hooks.

Each fisherman has between 3 and 10 longlines. Due to the strong currents and deep channel at Ponta Chugue, the main fishing area is located on the opposite bank of the Geba Estuary. Fishing occurs between August and April when the currents are generally weaker and there is a lower abundance of large sharks which are avoided as they damage the fishing gear. Fishermen fish every day over this period setting longlines overnight, or over a full tidal cycle.

Approximately 30 to 40 fishermen use the Ponta Chugue landing beach where roughly 10 canoes are based during the fishing season. Seacatfish (*Arius spp*.) is the main catch on the longlines, while more species including *Pomadasys spp., Mugilidae, E. fimbriata, I. africana, P. quinquarius, Pseudotolithus spp* may be caught by gillnet fishermen. A gillnet fisherman was observed during the field survey, however, the habitat available for setting gillnets is limited as nets need to be carefully set in deeper channels in between shoals close to the shoreline to ensure that they are not washed away or pushed down by the strong tidal currents. Crab fishing using baskets is also undertaken. Larger ferry canoes also utilize the landing beach to load/unload supplies and transport people.

20.1.2.5.7 Marine Mammals and Reptiles

The Government of Guinea-Bissau has protected the country's biodiversity by setting aside and managing approximately 536,972 hectares of its territory in six coastal and marine protected areas. Protected areas near the project include the Rio Cacheu Mangrove Natural Park, the Varela National Park, and the Pelundo Faunal Reserve.

These protected areas were largely defined to firstly protect the natural coastal habitats and forests, but more importantly provide protection for several important species of marine mammals and reptiles that occur in high numbers along the Guinea-Bissau coastline. None of these protected areas are located near the mine or Mineral Terminal sites, due to the lack of suitable habitat for the target species.

The West African subregion supports a diverse marine mammal fauna. Six baleen whale species and 22 toothed whale and dolphin species most likely occur in the region. Three of these whale species are endangered (blue and fin whales), two are vulnerable (i.e., humpback and sperm whales) and several others are in lower-risk categories. Coastal areas and offshore of West Africa are possible breeding and nursery areas for the humpback whale, which migrates along the coast of Southern Africa to mate, calve, and nurse their young during the austral winter.

Observations of these species within the project area was assessed during the Golder baseline studies with particular reference to potential impact of shipping collisions with these species, water quality and food resources impacts, and the disturbance of any turtle nesting sites within the project area. This is due to the lack of available habitat such as sandy beaches for turtle nesting sites, as well as seagrass beds for food (turtles and West African manatee). However, these maybe be encountered by shipping traffic from the proposed Mineral Terminal area on route to the open ocean.

20.1.2.6 Terrestrial Ecology

20.1.2.6.1 Flora

During the ecological studies conducted between 2011 and 2015, a total of 341 plant species were recorded (Hudson Ecology, 2015). Floral species diversity in the area is moderate to high, but not as high as many regions of West Africa, such as the Upper Guinea Forest zone. A large proportion of the species recorded are indigenous with few exotic species occurring in the area although, in areas of higher anthropogenic disturbances, exotic species are more prevalent. As is the case in tropical forests all over the world, the nutrient capacity of soil in the forests of Guinea-Bissau is poor; the reason for this is that most of the nutrients are stored in the plants themselves. In any forest, dead organic matter falls to the ground, providing valuable nutrients for new growth. In cooler or drier climates, the nutrients build up in the soil. However, in tropical rain forest, with its abundance and variety of life, those nutrients are reabsorbed almost as fast as they are deposited. This nutrient cycle is further exacerbated by the fact that the sandy soil in Guinea-Bissau is easily leached of nutrients during the wet season.

Due to the nutrient poor soil of the forest areas, little agriculture is practiced on the soils underlying the forest areas: the agriculture that is practiced is mainly subsistence agriculture (including maize (*Zea mays*), sorghum (*Sorghum bicolor*), bananas (*Musa* spp.) and cassava (*Manihot esculenta*)), which is conducted on a slash and burn basis. Agroforestry is conducted in Guinea-Bissau with large tracts of natural forest being clear-felled in order to plant Cashew trees (*Anacardium occidentale*). Generally, agriculture is practiced in the freshwater wetland areas, with rice (*Oryza glaberrima*) being the crop produced on large scale. Rice production has, in fact resulted in the transformation of almost all the freshwater wetlands in Guinea-Bissau.

Based on physiognomy, moisture regime, rockiness, slope, and soil properties, the following six main vegetation communities were recorded:

- Rhizophora – Avicennia Mangrove community
- Natural Forest vegetation community
- Secondary Forest community
- Elias – Cyperus Floodplain community
- Oryza Paddy vegetation community
- Anadelphia afzeliana seasonally wet grassland community.

The natural forest vegetation community is under threat due mainly to slash and burn agricultural practices for the cultivation of food crops or cashew nuts. Although only one International Union for Conservation of Nature (IUCN) Red Data species was recorded in this vegetation community, the likelihood of occurrence of Red Data species in this community is high. A total of 209 flora species were recorded in this vegetation community.

The secondary forest vegetation community comprises large sections of natural forests that have been cleared to grow cashew nuts and other crops. The cashew plantations vary from areas which are dominated by cashew trees (cashew monoculture), to areas of mixed cashews and secondary forest. A total of 145 flora species were recorded in this vegetation community.

Land use in the Mineral Terminal area 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 (Hudson Ecology, 2015).

20.1.2.6.2 Flora Species of Conservation Importance

Two IUCN Red Data listed species were recorded in the natural forest vegetation community: Raphia Palm (*Raphia palma-pinus*), listed as Data Deficient (DD); and the Musase tree (*Albizia ferruginea*), listed as Vulnerable (VU). *Albizia ferruginea* was recorded in the natural forest vegetation community which is now encompassed by the restoration area; however according to the vegetation remapping much of the forest, extant from the 2015 surveys, has now been cleared. This species is often first to be chopped down in forest fragments as it is much sought after for furniture making.

20.1.2.6.3 Fauna

Non-chordate diversity within the study area was relatively high with 124 arthropod species being recorded during the study. Most species recorded were common species with some specialized species being recorded in the mangrove communities. Most of the species recorded are not restricted in terms of range and habitat preferences. Common species included Red Winged Dropwing and locust.

The herpetofauna of the region can be classified as having moderate diversity, of the 69 reptile species known to occur in Guinea-Bissau, only 11 species were recorded. This may be due to the proximity of the project area to the town of Farim and other settlements in the area. Common species occurring in the area include Ornate Monitor and Tree Agama.

The region can be classified as having low amphibian diversity; of the 34 amphibian species recorded in Guinea-Bissau, only five species were recorded during project surveys, none of which are IUCN listed. None of the five recorded species appear to be utilized by the local community for food, although some species are said to have superstitious importance or medicinal uses. The only species recorded in the area that will form part of the restoration area was *Amietophrynus regularis* (Common African Toad).

Avifaunal diversity in the study area was very high with a large number of upper trophic level species occurring in the area (Table 20-2). The Hooded Vulture is currently listed as Critically Endangered by the IUCN (BirdLife International, 2017). Seventy-five species were recorded, including the Palmnut Vulture, Longcrested eagle, Hooded vulture and Gymnogene. In general, species diversity was moderate to high throughout the study area with the rice paddies and natural forests showing the highest levels of species diversity.

Mammal species diversity was very low in the study area, probably due to severe subsistence hunting (Hudson Ecology, 2015). Hunters were regularly seen or heard during the surveys, often with animals ranging from snakes to monkeys. This not only reduces the number of animals and species in the area, but also causes the remaining animals to be very shy of humans, which in turn makes accurate survey of species occurring in the area very difficult. Fifteen of the 192 mammal species known to occur in the study area were recorded during project surveys. The species recorded in the study area include Striped Ground Squirrel, Musk shrew, Lesser Spot-nosed Guenon, and Red colobus, all of which are common species, with the exception of Red colobus, listed as Endangered by IUCN (2018), which was recorded after being killed by a hunter and transported south into the study area from the forests in the north.

A total of 28 Red Data fauna species may occur in the RSA, according to the IUCN Red Data list and these species are listed in Table 20-2. Some of the animals listed are believed to be locally extinct, but as this is not confirmed, these species have been included in the table. Considering habitat suitability, the probability of the species occurring in the project area is provided in Table 20-2.

Table 20-2: Probability of Occurrence of Red Data Faunal Species in the Project Area

Scientific Name	Common Name	IUCN Status	Probability of Occurrence
Balearica pavonina	Black Crowned-Crane	VU	High
Ceratogymna elata	Yellow-casqued Hornbill	NT	High
Circaetus beaudouini	Beaudouin's Snake-Eagle	VU	High
Circus macrourus	Pallid Harrier	NT	Moderate
Gallinago media	Great Snipe	NT	Low
Gyps africanus	White-backed Vulture	NT	Low
Gyps rueppellii	Rueppell's Griffon	NT	Low
Limosa	Black-tailed Godwit	NT	Moderate
Necrosyrtes monachus	Hooded Vulture	EN	Recorded
Neophron percnopterus	Egyptian Vulture	EN	Low
Neotis denhami	Stanley Bustard	NT	Low
Numenius arquata	Eurasian Curlew	NT	Moderate

Scientific Name	Common Name	IUCN Status	Probability of Occurrence
Phoenicopterus minor	Lesser Flamingo	NT	Moderate
Psittacus erithacus	Gray Parrot	NT	Moderate
Trigonoceps occipitalis	White-headed Vulture	VU	Low
Lycaon pictus	African Wild Dog	EN	Locally Extinct
Cercocebus atys	Sooty Mangabey	VU	Low
Colobus polykomos	King Colobus	VU	Low
Papio	Guinea Baboon	NT	Low
Procolubus badius	Red Colobus	EN	Recorded
Loxodonta africana	African Bush Elephant	VU	Locally Extinct
Panthera leo	Lion	VU	Locally Extinct
Panthera pardus	Leopard	NT	Locally Extinct
Hippopotamus amphibius	Common Hippopotamus	VU	Locally Extinct
Pan troglodytes	Common Chimpanzee	EN	Locally Extinct
Eidolon helvum	Straw-Colored Fruit Bat	NT	High

Notes: EN – Endangered; VU – Vulnerable; DD – Data deficient; NT – near threatened.

20.1.3 Socioeconomic and Cultural Setting

20.1.3.1 National Socioeconomic Setting of Guinea-Bissau

The national socioeconomic 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. Guinea-Bissau is ranked 177 out of 187 countries according to the 2018 United Nations Development Program (UNDP) Human Development Index and has one of the lowest per capita gross domestic products in the world (United Nations Development Program, 2018).

Guinea-Bissau's Human Development Index value has increased slightly year over year since 2012. Only 14% of the population speak the official language (Portuguese). 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). Indigenous ethnic groups such as the Papels were not identified in the vicinity of the project.

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, in which 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 is characterized 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, most villages have fewer than 500 inhabitants.

20.1.3.2 Local Socioeconomic Setting

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 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 for 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.

The following points summarize the local population within the local study area:

- Ethnicity – The mine area includes eight ethnic groups: Mandinga (66% of the population), the Mansonka (17%), Fula (7.6%), and Balante (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%).
- 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 animism is practiced by 4% of the population. These latter religions are mainly practiced by the Balante.
- Housing – Households are located in clusters as rural villages rather than widely distributed. Households may comprise 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, neighboring 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.
- 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 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 has the opportunity to express their opinion. Heads of villages are under the authority of the administrator of the district to whom 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 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.

- Housing – The majority of the population of Guinea-Bissau live in rural villages or small towns. The villages were found to be dispersed around the landscape study area, most likely because of agricultural practices and land tenure. These settlement patterns can be described as being nucleated around key services such as water points or forming a linear alignment along main roads.

At the village level, a common pattern is for multi-generations of the same family to reside in a common compound or homestead and share the agricultural land which they cultivate collaboratively. Residential areas are dispersed in space. A "homestead" is defined as the physical property owned under customary law, and includes all physical assets located on the property. A "household" refers to the family unit that occupies a homestead. A "house" refers to the primary building in which people sleep.

20.1.3.3 Demographics

The national population of the country in July 2017 was estimated at 1,792,338 (Central Intelligence Agency [CIA], 2016). The ethnic breakdown of the population as of 2008 is as follows (CIA, 2016):

- Fula 28.5%
- Balanta 22.5%
- Mandinga 14.7%
- Papel 9.1%
- Manjaco 8.3%
- Beafada 3.5%
- Mancanha 3.1%
- Bijago 2.1%
- Felupe 1.7%
- Mansoanca 1.4%
- BalantaMane 1%
- Other 1.8%
- None 2.2%

The dominant language spoken by 90.4% of the population is Crioulo, a Portuguese-based Creole language. The official language in Guinea-Bissau is Portuguese; however, only 27.1% of the population speaks the language. Other languages spoken, based on a 2008 census, include French (5.1%), English (2.9%), and other languages (2.4%) (CIA, 2016).

The religions practiced, based on a 2008 estimate, include Islam (45.1%), Christianity (22.1%), Animism (14.9%), none (2%), and unspecified (15.9%) (CIA, 2018). The Constitution of Guinea-Bissau encompasses freedom of religion.

The Republic of Guinea-Bissau is divided into three provinces: Leste (East), Norte (North) and Sui (South). All project components are located within the North Province, and also entirely within the Oio Region, which is one of eight regions in the country. The population of the Oio Region is estimated at 215,259 inhabitants; the total population within the Farim sector is 48,264, of which 8,661 reside in the Town of Farim (Eco Progresso, 2015).

The resettlement-affected villages and surrounding villages are represented by six ethnic groups (Table 20-3). In Tambato Mandinka and Canico 100% of the households are Muslim, while in Saliquenhe 97% of the households are Muslim and 3% are Catholic Christians. The majority of the households in Ponta Capsec/Cabisseki are Animist (57%), with the remainder

Christian (29% Catholic and 14% Protestant). The majority of the households in Saliquenhe Porto/Ponta Zeca are Christian (58% Catholic and 33% Protestant) with a few Animist (8%).

The villages of Sandjal, Tambandinto, Temanto, Sintchan Tierno and Sintchan Maudi were not included in the 2015 household survey.

Table 20-3: Village Demographics in Project Area

Village	Ethnic Origin / Tribal Association(s)	No. of Households	Average No. of People per Household	Total Population
Villages to be Resettled at Buredanfa				
Saliquenhe	Mandinga (97%); Mancanha (3%)	76	18.9	1,476
Ponta Capsec/Cabisseki	Balanta (100%)	8	16	128
Saliquenhe Porto	Manjaca, Mancanha	2	17	34
Ponta Zeca	Balanta	12	20.2	242
Tambato Mandinka	Mandinga	22	18.1	400
Canico	Mandinga	55	17.1	940
Villages within Restoration Area				
Sintchan Tierno	Fula	2		
Adjacent Villages to Buredanfa and Restoration Area				
Canico Tumana	Mandinga	29	11.3	329
Urqui (Seidi)	Fula	21	11.4	239
Other Villages Outside Buredanfa and Restoration Area				
Ufude	Mandinga, Fula (1); Balanta (1)	15	17.9	268

20.1.3.4 Household Size and Composition

Preliminary land use mapping within the mine area socioeconomic study in 2015 identified 233 individual building structures, with most households consisting of more than one building, for an assumed 175 households. Houses and secondary buildings are mainly built out of clay; the majority have tin roofs, while a small number have traditional thatched roofs. The floors of these structures are primarily made from earth, although concrete is utilized in some households. Within surveyed households, the primary assets generally associated with a typical household include a multifunctional building (the main house), a separate kitchen, a fenced garden, and one or more areas to hold livestock.

The homesteads are generally shared by several nuclear families related by kinship and a "chief" who is the father or grandfather common to them all. Families also share the land for agriculture. The head of each household is typically male, with only 0.4% of the surveyed households containing female headed households. Patriarchal customs dominate with property and title most often held by male lineage. Sons and daughters typically reside at the father's homestead until such a time that they move away or establish their own household on the same homestead property. The presence of multiple nuclear families within each household indicates that there is limited movement of extended families or relations between households.

20.1.3.5 Economic Conditions

The following summarizes the economic conditions:

- 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 an estimated 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, which is the most important crop, is cultivated by more than 51% of 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 and 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 saltwater intrusion from the River Cacheu. 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 Mineral Terminal 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 riverbed directly adjacent to the Chugue shoreline, the fishermen 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 fibers, 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 leaves, and wooden poles).
- Landscape – Four main landscape types were identified in the mine 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, no nationally or internationally recognized geographical features or landmarks are in the mine study area. There are many very old trees, including giant Baobab trees within the study area, 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.

20.1.3.6 Income and Expenditure

Livelihoods in the study area are agriculture-based, with emphasis on cashew production as a cash crop. Listed below are the primary livelihood activities:

- agriculture (for household consumption and cash trade)
- tree production (mainly but not exclusively cashew trees)
- livestock (almost 93% of surveyed households own cattle, sheep, goats and/or pigs)
- natural resource use (includes a variety of harvesting and production activities such as fishing; salt production; collection of wood, medicinal plants and other resources)
- wage employment (few people in the study area rely on wage employment).

The primary livelihoods adopted by households in the resettlement-affected villages are the following:

- agricultural cropping
- fruit tree cultivation
- livestock rearing
- salt production
- fishing
- natural resource harvesting.

Households in the villages affected by resettlement were surveyed in 2016. The households claimed to have, on average, five to seven agricultural fields, of which four were typically actively farmed, with the remainder in fallow (Nomad Consulting and Africa-Wide Consulting, 2017). The primary crops for surveyed households were maize, sorghum/millet, and rice. Approximately 58% of the crops produced were solely for household nutrition; 4% of crops are reportedly sold for cash income, while the remainder is used for household consumption or traded. Produce that is sold for cash income is largely traded in the same village (70% of surveyed households), with a limited (1% of surveyed households) intervillage trade. The trade of produce is primarily done via setting up stalls in front of homesteads, as there are no markets in the resettlement-affected villages (Nomad Consulting and Africa-Wide Consulting, 2017). Trade of crop produce outside of the villages primarily occurs at markets in Farim (16% of households), with a smaller proportion of households travelling as far as Senegal (6.5%) and Bissau (3%). Travel to other villages for trade is by use of local trucks, personal bicycles, or by foot.

Tree crops play a proportionally larger role in terms of sustaining livelihoods in this region in comparison with other villages in West Africa. Production of cashew fruit/nuts, and to a lesser extent mango fruit, is a major source of cash income. Trade in cashew nuts, mango, and other tree products is primarily focused within the village (51%) or at markets in Farim (31%). Intra-village trade is through local traders who visit the village to buy their produce. Travel time and costs restrict trade further afield, though 6% of trade is reported to occur in Bissau.

Livestock rearing is a key livelihood and primary protein source for households. Chickens and goats are kept by 90% and 80% of households, respectively, likely due to their smaller size and relatively easy upkeep. Pig rearing is limited to the smaller, primarily Christian, villages of Saliquenhe Porto/Ponto Zeca, Canico Tumana, and Tambato Mandinga, since the larger villages of Saliquenhe and Canico are predominately Muslim. Livestock, (notably chicken, goats and sheep) are used primarily for household food and nutrition, but trade occurs in times of surplus or if additional cash income is required. The trade in livestock largely occurs within the village (56% of market trade) or at markets in Farim (34% of market trade). Only

26% of households claim to trade in animal products (e.g., cow milk, eggs), suggesting that livestock is maintained to primarily sustain its household.

Salt farming and the trade in salt is nearly exclusively undertaken by females and is range restricted to the floodplains of the river. The salt is transported by females to the homestead.

The trade of salt at markets in Farim accounts for 43% of all trade, while the sale of salt in other locations in the Oio Region accounts for 20%. Salt is also traded within the village (20% of trade); however, this is largely limited to local traders visiting the village to buy salt stocks.

A total of 56% of the surveyed households undertake fishing as a source of livelihood. Fishing is restricted to the salt-water River Cacheu and is largely restricted to shore fishing using handlines or nets, while only 6% of the surveyed households reported having access to a boat. There are a limited number of species that are targeted and commonly caught including Bentana, Esquilão and Tainha. Fish catch is primarily a source of cash income and only secondarily a source of household food. Fish products are traded intravillage via small stalls established near the homestead or at markets in Farim.

Villagers informally trade in surplus vegetables, fruit and animal products; however, these are not defined as businesses. Forms of trade are limited to taverns and small shops.

Natural resources harvesting is undertaken by all surveyed households. The degree of harvesting varies by the type of natural resource, with most surveyed households harvesting firewood, wild fruit and vegetables, and nearly half also sourcing wood for charcoal production and medicinal plants. Households primarily trade in natural resources using a dual strategy: first to secure food, with any surplus traded for cash income. However, in some cases where there is a surplus of crops or household livestock harvested resources can be used to supplement household income.

Only 5% of males and 1% of females reported wage labor as a primary livelihood.

The main source of income earned by the villages can be attributed to collecting wood used for charcoal making (Nomad Consulting and Africa-Wide Consulting, 2017). Approximately one-third of surveyed households reported using their homestead as a place of business. Business activities reported include the following:

- sale of salt, animals, milk, smoked fish, medicine, and other necessities
- tavern
- trading
- two households were reported to be rentals.

The majority of income is spent on personal and food items (35.6%), clothes (13.3%), agricultural expenses (8.9%), telecommunications (8.8%) and household goods such as furniture (8.4%). Expenditure is thus concentrated on the household and sustaining farming practices.

20.1.3.7 Community Infrastructure

There is no distributed electricity within the surveyed villages, and few generators. The main source of fuel for cooking is wood (98% of households), with the remainder using paraffin, although one household reported using electricity. Candles, paraffin lamps, and battery torches (flashlights) are primarily used for household lighting; a small number of affluent families utilize solar energy.

Within the RIZ, water utilized by the villages is primarily sourced from a number of wells or boreholes. These wells are primarily located in communal areas and within the residences of individual homesteads. Approximately 25% of surveyed households rely on deep wells; 30% on hand pumps, and 45% on shallow hand-dug wells. A small proportion of the surveyed households reported water shortages at all times of the year, while the majority of the surveyed population experienced water shortages at the end of the dry season (April and May). Local rivers and streams are not used as a water source: River Cacheu, though flowing year-round, is brackish, and smaller tributaries are ephemeral.

There is no formalized sanitation infrastructure in the study area: the majority of surveyed households (77%) utilize pit latrines (mostly located within the boundary of each household), with the remainder using the bush. Of households with pit latrines only approximately 9.4% of households use a ventilated improved pit latrine.

There is no formal waste collection and disposal system within the villages surrounding the mine: the majority of households litter (42%) or burn their trash (41%), while a minority bury their waste or report disposing of it within a community waste disposal facility.

There is a limited range of community or public facilities located in the five resettlement affected villages. Cabisseki / Ponta Capsec, Saliquenhe Porto/ Ponta Zeca and Tambato Mandinka have negligible community facilities – limited to one grass hut school at Tambato Mandinka. The larger village of Saliquenhe supports one Mosque, clinic and school; while Canico Tumana has one closed school and one Mosque (Nomad Consulting, 2017).

The most common forms of communication in the study area are mobile phones and radio, with nearly every household owning one or more cell phones, and 80% owning a radio. Twelve percent of households reported owning a television, and 1.6% of households reported owning a computer.

The modes of transportation most used in the surveyed villages are bicycle (75% of households); motorcycle (11% of households); and motorized car (1.6% of households). Donkeys are also likely used in part for transportation.

20.1.3.8 Land Use

The Land Law (Law No. 5/98) states that land is the property of the State and the heritage of the population; however, land is administered following traditional law by customary authorities. In the traditional system common in rural areas, the "ax right" allows a person to either clear land not previously occupied before they register it with the village chief; or clear the land upon allocation by the chief. The landowner may also sell part of it to a third party and declare the "transaction" with the village chief.

The replacement land identified for resettlement is at Buredanfa and is situated adjacent to the North pit mining area, at an elevation ranging from approximately 7 mamsl in the east to 50 mamsl in the west and covering approximately 2,575 ha. The area will allow for a wide range of cropping, as well as livestock raising and natural resource purposes to be undertaken consistent with resettlement needs. The cadastral office will be issuing title of the land to GB Minerals, who will in turn offer certificates of occupancy to all households being resettled. Any remaining land will be donated back to the communities. Many of the resettlement-affected villages have already established cashew/mango/oil palm trees and cropping areas across an extensive area of land from the north open pit into Buredanfa. Prior to resettlement a full land-use and land capability audit of the replacement land will be conducted to map existing farmland. Existing farmland will be protected and excluded as replacement farmland. Approximately 775 ha area in the most north-westerly portion of the Buredanfa Host Site area is forested, and the forest is already used to a certain extent by local people to secure natural resources.

Additional changes in land use associated with the mine will be creation of the Cansenha bypass road to replace the existing road that connects Cansenha to Farim via Saliquenhe (Farim West Access Road) to preserve residents' accessibility to Farim and allow for safe passage around the mine (KP, 2015a). The Cansenha bypass road will abut the southeast end of the Restoration Area and connect to the existing village road immediately east of Canico Tumana. The Farim West Access Road currently provides access between Farim and Cacheu for a number of villages located north and south of this route, including Cansenha, Tambato, Saliquenhe, Ponta Zeca and Bani. Traffic surveys conducted in 2013 found that traffic flow in this area is moderate, with predominantly pedestrian traffic and bicycle traffic.

A large proportion of the land that will be acquired is secondary forest (a mosaic of natural vegetation and transformed land): secondary forest is important to local communities in the provision of natural resources and additional land for the establishment of agricultural plots or cashew orchards (Nomad Consulting, 2017). The current remaining major land-uses in the restoration area include farmland, grazing-land, clear felled areas, rice farms and cashew monoculture – totaling approximately 1,102 ha, or 37% of the total land acquisition area (Nomad Consulting, 2017).

20.1.4 Health and Safety

20.1.4.1 Health and Nutrition

At the national level the maternal mortality rate is 549 female deaths per 100,000 live births (2015 estimate), placing it 18th highest out of 184 countries included in the World Factbook (CIA, 2016). The high maternal mortality rate is attributed to the prevalence of early childbearing, a lack of birth spacing, the high percentage of births outside of health care facilities, and a shortage of medicines and supplies. The infant mortality rate (the number of deaths of infants under one-year-old) is 85.7 deaths per 1,000 live births (2017 estimate, male and female combined), placing it 4th highest out of 225 countries included in the World Factbook (CIA, 2016). The infant mortality rate is often used as an indicator of the level of health in a country. Guinea-Bissau is ranked 35th out of 138 countries in terms of the number of underweight children under five years old, with a reported 17% (2014 estimate). Health care services are poor: the physician density was 0.08 per 1,000 people and the hospital bed density was 1 bed per 1,000 people in 2009 (CIA, 2016).

The availability of health-related services is poor at the regional level, and only one doctor and one midwife were reported for the RIZ (KP, 2015a). The majority of respondents (89%) reported seeking medical assistance at the Medical Care Center in Farim in response to a medical issue; respondents who indicated that they did not make use of existing medical facilities reported using a home remedy (5%), purchasing medicine at a pharmacy (2.6%), not doing anything (1.6%), seeing a medicine man (1.5%), and taking the person to a priest or church/mosque to pray (<1%) (KP, 2015a).

Lack of accessibility was cited as the primary reason for not making use of existing medical facilities, due to facilities being too far away, no staff being available, or lack of transport. An additional 22% of respondents indicated the cost or lack of economic means as the primary reason for not seeking medical treatment, while 6.1% indicated that medical treatment was not necessary and about 1% indicated that traditional treatment was typically sought.

The most prominent ailments affecting the villages are malaria and diarrhea. Other diseases and illnesses included tuberculosis, high blood pressure, pneumonia, heart problems, typhoid, measles, asthma, diabetes, hepatitis, meningitis, and cholera (KP, 2015a).

20.1.4.2 Social Conflict

Guinea-Bissau has a history of political instability, a civil war, and several coups (the latest attempt was in 2012), which have resulted in a weak economy, high unemployment, rampant corruption, widespread poverty, and thriving drug and child trafficking (CIA, 2016).

20.1.4.3 Archaeology and Cultural Heritage

Cultural heritage baseline surveys of the mine area were undertaken in 2012 by the Senegalese Laboratoire d'Archéologie IFAN-UCAD (Golder, 2014), and of the mine and Mineral Terminal areas with a primary focus on archaeological resources and a secondary focus on sacred sites, cemeteries and mosques (ERM, 2015). Additionally, Eco Progresso also conducted a supplemental assessment in 2015, focusing on in-depth village interviews to identify living heritage resources, including sacred sites, cemeteries and mosques (Eco Progresso, 2015a). The results of these three surveys constitute the cultural heritage baseline for the project.

ERM visited each of the 25 cultural sites listed in the 2014 Golder baseline report, only 14 of which could be relocated. Sites identified in the Golder survey that could not be relocated may have been isolated ceramic fragments that were collected during the survey, which would remove all surface evidence. It is also possible that some of the sites were misidentified or recorded as areas where the local population indicated they had seen ceramics in the past. Information from the Golder baseline report (2014a) does not help to clarify this issue, as only coordinates were provided without site descriptions.

In addition to relocating 14 previously identified sites, ERM recorded five new sites in the mine area. An additional five new sites were identified by ERM in the Mineral Terminal area, which was not visited during the Tropica survey. Eco Progresso identified 46 separate living heritage sites. However, coordinates were not recorded for most sites. As a result, only 15 living heritage sites identified by Eco Progresso are included in this analysis. The location of these 15 sites is not precisely known. Future efforts should target the remaining living heritage sites that could not be relocated.

Combining the confirmed previously recorded sites and the newly recorded sites, the baseline survey identified 39 cultural sites: 15 archaeological sites; seven cemeteries; 11 sacred sites; and six mosques.

Fifteen cultural heritage sites were recorded within the maximum area of disturbance (MAD) and are likely to be directly affected by activities associated with the project (Table 20-4). Five are classified as religious sites, four as sacred sites, and six as archaeological sites. The sensitivity of each site has been identified based on the cultural heritage sensitivity criteria established by ERM and presented in the mine ESIA (KP, 2015a).

Table 20-4: Cultural Sites Identified within the Maximum Area of Disturbance

Mine Component	Site No.	Classification	Type	Sensitivity
North Pit	MSQ-1*	Religious	Mosque	High
	MSQ-2*	Religious	Mosque	High
	SS-1	Sacred Site	Sacred Forest	High
	SS-9*	Sacred Site	Sacred Forest	Low
	CM-4	Religious	Cemetery	High
	CM-3	Religious	Cemetery	Low
	AR-3	Archaeological	Low-density ceramic scatter	Low
South Pit	CM-7*	Religious	Cemetery	Low
	AR-5, AR-7, AR-8	Archaeological	Low-density ceramics scatters	Low
	AR-6	Archaeological	High-density iron smelting site	Medium
Dam Water	SS-2	Sacred Site	Sacred Forest	Low
WD-2	SS-11*	Sacred Site	Sacred Forest	Low
Processing Facility	AR-4	Archaeological	High-density ceramic scatter	Medium

20.1.5 Tailings Geochemistry

A summary of the tailings geochemical characterization programs is provided in Table 20-5.

In 2015, an initial tailings geochemical characterization program was presented. The tailings materials were sent to KP Perth in Australia and arrived dry. As such, the tailings solids were mixed with local tap water to create a proxy for the supernatant water quality (KP, 2015a). Given that the tailings were dry upon delivery, these tailings are considered to be less representative of the expected tailings for the project. Therefore, the geochemical characterization of the tailings and supernatant, based on the samples derived from the two pilot plant programs in 2017, are summarized below:

As outlined in Table 20-5, these tailings solid samples underwent a suite of static and kinetic testing to assess the overall geochemical characterization of the materials, as well as their reaction rates over time. Test results suggest the tailings will be non-potentially acid generating (non-PAG) (KP, 2018a). As such, there is no perceived risk from acidification of the tailings and there are no specific controls required.

The tailings solids contain elements with high levels of element enrichment, with antimony, bismuth, phosphorous, selenium and uranium. However, both short-term leach testing and humidity cell testing demonstrated that only cadmium and nickel were prone to leaching when screened against the River Cacheu target receiving water quality standards (KP, 2017; KP, 2018b). The tailings supernatant also demonstrated elevated cadmium and nickel concentrations, which suggest that these parameters may be a result of the ore processing and not from metal leaching of the tailings solids (KP, 2017).

Table 20-5: Summary of Testing on the Tailings Material (Including Supernatant) to Date

Year	Static testing	Kinetic testing	Radiological testing	Objective	Reference
2015	1 tailings solid sample 1 modified supernatant sample (tailings solids mixed with tap water from Perth, Australia)	N/A	1 sample of course product 1 sample of fine product	Initial geochemical characterization program on the tailings materials available in 2015.	KP Perth, 2015a
2017	3 samples of coarse rejects, 3 samples of combined product, 5 samples of cyclone tails from pilot plant 1 3 samples of coarse tails and 3 samples of fine tails from pilot plant 2	1 combined tailings sample from pilot plant 1	2 samples of coarse rejects, 3 samples of combined product, 5 samples of combined tailings from pilot plant 1 3 samples of supernatant from pilot plant 1	Subsequent geochemical characterization program inclusive of industry standard static testing for both acid base accounting and short-term leach testing to assess the ARD/ML and radionuclide potential from the tailings solids and liquids products from pilot plant 1 and pilot plant 2.	KP, 2017
2018	12 samples from the pilot plant 1 6 samples from the pilot plant 2	1 combined tailings sample from pilot plant 2	N/A	Kinetic testing of a combined tailings sample from the second pilot plant was recommended to verify the non-PAG classification of the tailings from the first pilot plant. The second pilot plant produced a larger sample of tailings material. Additional kinetic testing was recommended to confirm the geochemical characterization of the tailings material.	KP, 2018

The 2017 radionuclide testing on the tailings materials indicated that the both lead-210 and radium-226 concentrations from all samples were above the Health Canada release limits (Health Canada, 2011); however, this was not demonstrated within the tailings supernatant. Given that no exceedances were noted within the tailings liquid, it is likely that the near-neutral pH of the tailings leachate/supernatant does not allow for the isotopes to be mobilized in water. A decrease in pH could result in these isotopes becoming mobile, however this testing was recommended in 2016 and not assessed. With respect to public exposure, based on the available data for Farim, exposure to the tailings and ore could theoretically result in doses that exceed the Health Canada (2011) dose constraint of 0.3 mSv/y and the incremental dose of 1 mSv/y that are used for the protection of the public (NECA, 2015). The public would have to be exposed for a full year, however, which is unlikely as the operating mine will not be accessible to the public. At closure, when local residents may seek to inhabit the area again, long-term exposure can be managed by constructing a suitably designed cover on the tailings storage facility that adequately shields exposure.

Limiting exposure to radium-226 can also be achieved by constructing open walled structures in proximity to the TSF. radium-226 can build up within walled structures over time, which could result in a higher dose to anyone entering those buildings. Open walled structures allow for the passage of air, which alleviates the risk of radium-226 buildup.

20.1.6 Ore and Waste Overburden Geochemistry

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 20-6.

Table 20-6: 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 (FPP)	7 (single intercept)
	Upper Dolomitic Limestone	Clayey limestone	>2[1] (single intercept)
	Decarbonized 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]

Note: 1. Base of unit not encountered.

An initial geochemical assessment comprising testing of 20 composite samples of overburden was undertaken in 2012 as part of a previous phase of the project. Based on the 2012 data, 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, sulfate, arsenic, cadmium, chromium, iron, manganese, nickel, lead and zinc exceed the reference surface water quality guidelines, with cadmium and nickel exceeding the WHO drinking water guidelines.

Since this initial program, additional geochemical characterization and radionuclide testing programs were completed between 2014 and 2017. A summary of these programs is provided in Table 20-7.

Geochemical testing was completed on samples of ore and product, tailings and waste overburden as outlined below.

- Phosphate Ore and Product Geochemistry – 156 ore samples were tested for mineralogy and metals in 2014 (Golder, 2014b). In 2015, one phosphate composite ore sample and one associated phosphate rock product sample from pilot plant metallurgical testing were analyzed for metals (KP, 2015e, 2015f). An additional 14 samples of ore were collected by KP (7 from the South pit and 7 from the North pit) during a 2016 sampling program (KP, 2017a). These 14 samples underwent a full suite of geochemical characterization testing to assess the overall acid generating and metal leaching potential of the phosphate ore. Based on these data the following conclusions were made:
- Acid Generating Potential of Ore – Approximately 11 of the 14 samples (~78%) have elevated sulfide sulfur concentrations, however only 1 of these 11 samples (from the North pit) is classified as PAG, given that the ore does host a low to moderate concentration of carbonate, which buffers the sulfide acidification. It should be noted that the ore will undergo processing, and the overall acid rock drainage (ARD) potential of the material is not a project concern, unless short-term surface storage of ore is required.
- Metals in Ore – While the ore is enriched in Ag, Sb, Bi, P, Se, U, and Y, some of which may be of an environmental concern, (i.e., only Cd, Ni, Se, Zn was prone to leaching under acidic conditions, based on short-term leach testing). Similar analytes are noted below from the tailings geochemistry testwork (discussed below).
- Phosphate Rock Product – The metals concentrations within the phosphate rock product are similar to the ore, based on a single metals scan of product and the corresponding feed ore.

The ore and phosphate rock have higher metals concentrations overall, but both are expected to have lower environmental exposure. It is expected that most of the ore will be processed to produce phosphate rock, with waste deposited as tailings within the tailings storage facility (TSF).

- Tailings Geochemistry – Geochemical test results of the tailings are inclusive of samples derived from two pilot plant programs (both occurring in 2016) as part of the feasibility study review and update (KP, 2017a). It should be noted that an initial tailings sample was assessed in 2015 (KP, 2015g); however, the two pilot plant programs that occurred in 2016 were based on a larger number of samples (13 and 60 drillholes) representing a larger proportion of the south deposit, so therefore the 2015 data was not considered. These 2016 samples underwent a suite of static and kinetic testing to assess the overall geochemical characterization of the materials, as well as their reaction rates over time. Test results suggest the tailings are non-acid generating (NAG) (KP, 2018a). As such, there is no perceived risk from acidification of the tailings and there are no specific controls required.

Table 20-7: Summary of Geochemical Characterization Testing on Waste Overburden and Ore

Year	Static testing	Kinetic Testing	Radiological Testing	Objective	Reference
2012	20 composite overburden samples 156 ore samples	N/A	N/A	Initial geochemical characterization program on the waste overburden. Data did not include metal leaching testing but focused on whole rock compositions to make ARD/ML assessments.	Golder, 2013; 2014a; Golder 2014b
2015	38 overburden samples from five drillholes 1 phosphate composite ore sample 1 phosphate rock product sample	N/A	N/A	Subsequent geochemical characterization program inclusive of industry standard static testing for both acid base accounting and short-term leach testing to assess the ARD/ML of the waste overburden.	KP Perth, 2015b
2015	N/A	N/A	12 surface overburden samples from the project footprint and Mineral Terminal site 6 ore samples 2 product samples generated by pilot plant testing	To collect baseline radionuclide data for the project during the 2015 EIS	NECA, 2015
2016	71 overburden samples from 14 drillholes (39 samples from the North pit and 32 samples from the South pit)	4 humidity cells inclusive of two PAG cells from the North pit and two PAG cells from the South pit.	3 samples of waste overburden, 2 samples from metallurgic drillholes	A targeted geochemical characterization program to address a gap in previous sampling which did not isolate samples by material type and composite samples consisting of multiple materials were tested. KP was interested in testing these materials separately to determine if the PAG overburden was associated with overburden material type or by spatial distribution.	KP, 2017
2017	19 samples from South pit – 5 Lôdo Clay samples and 14 from North Lobe 2 drillholes	N/A	N/A	Targeted sampling program to satisfy the recommendations from the 2016 sampling program. ➢ Determine the extents of the Lôdo Clay unit within the South pit, as this unit is considered PAG and will require proper storage ➢ Delineate the area of PAG material surrounding drillhole DH16-GC-09 within the South pit to better estimate the volume of overburden that will require PAG storage	KP, 2018b

- Based on inductively coupled plasma mass spectrometry (ICP-MS), the tailings solids contain elevated levels of element enrichment for Sb, Bi, P, Se and U. However, both short-term leach testing and humidity cell testing demonstrated that only Cd and Ni were prone to leaching when screened against the River Cacheu Target Receiving Water Quality Standards (KP, 2017a and 2018a). The tailings supernatant also demonstrates elevated Cd and Ni concentrations, which suggest that these parameters may be released by the ore processing and not specifically from metal leaching of the tailings solids (KP, 2017a).
- Waste Overburden Geochemistry – A total of 126 samples of the waste overburden have been collected and undergone geochemical characterization testing since 2015 (KP, 2015f; 2017a; 2018b). Of these 126 samples, 59 were from the North pit and 67 were from the South pit. It should be noted that in 2017, 19 samples from the South pit were collected to specifically target likely PAG waste overburden (KP, 2018a), as such, these data are not included in the following statistics.
    - North Pit – 9 of 59 samples (~15%) collected from the North pit are considered PAG or Uncertain. The PAG samples are primarily within proximity to the ore body at depth. The sulfide content is the main driver for ARD within the waste overburden, as there is little to no buffering capacity (readily reactive carbon minerals) to react with the acid producing minerals. The onset to ARD for known PAG materials is immediate, and the leachate remains steadily acidic over time. Both short-term leach testing and kinetic testing demonstrated Metal Leaching (ML), with exceedances of Al, As, Cd, Cr, Cu, Fe and Zn. Though, it should be noted that these metal exceedances are more prevalent with acidic leachate. If the waste overburden is stored such that the PAG material is segregated and encapsulated with non-acid-generating (NPAG) waste overburden, such that there is limited ingress of either air or water to react with the waste, the metal leaching potential should be negligible.
    - South Pit – 12 of 48 samples (25%) collected from the South pit in 2015 and 2016 are PAG or uncertain. The PAG samples are associated with the following three conditions:
        - proximity to the ore body at depth (similar to what was noted in the North pit)
        - surficial samples within the Lôdo Clay (or lignitic clay), which was later confirmed to be a PAG unit during the additional 2017 testing
        - DH16-GC-09 within the north lobe of the South pit had an anomalous number of PAG samples, associated with high sulfide content. This area was targeted in 2017 and it was concluded that this part of the South pit has a higher sulfide content, regardless of depth, and should be classified as PAG waste.

As mentioned previously regarding the North pit samples, the sulfide content is the main driver for ARD within the waste overburden, as there is little to no buffering capacity (readily reactive carbon minerals) to react with the acid producing minerals (sulfides). The onset to ARD for known PAG wastes from the South pit is also immediate and the leachate remains steadily acidic over time. Metal leaching (ML) from the short-term leach testing and kinetic testing demonstrates that Al, Cd, Co, Cu, Fe, Ni and Zn are prone to leaching. As previously stated for the North pit samples, these metal exceedances are more prevalent with acidic leachate. If the waste overburden is stored such that the PAG material is segregated and encapsulated with NPAG waste such that there is limited ingress of either air or water to react with, the metal leaching potential is negligible.

20.1.7 Radiological Characteristics of Ore, Tailings and Waste Overburden

A preliminary radiological assessment of the ore, surface soils, and tailings was completed by Northern Environmental Consulting and Analysis in 2015 (NECA, 2015). Additional testing was completed on the ore (product), tailings solids, and supernatant in 2016 (KP, 2017a). The preliminary assessment calculated exposure dose to humans based on uranium concentrations in the subject media. Results from the preliminary assessment indicated that the background radiation from uranium in the soil of the Farim area is variable, but the external doses to people in the area from the uranium are consistent with the global average.

The 2017 dataset indicated that the both lead-210 and radium-226 concentrations from all samples were above the Health Canada release limits (Health Canada, 2011). Both lead-210 and radium-226 are daughter products that develop from the decay of uranium. Elevated concentrations of lead-210 and radium-226 were not detected within the tailings supernatant. Given that no exceedances were noted within the tailings liquid (supernatant), it is likely that the near neutral pH of the tailings leachate/supernatant does not allow for the isotopes to mobilize in water. A decrease in pH may result in these isotopes becoming mobile; however, this testing, recommended in 2016, has not yet been carried out.

With respect to public exposure, based on the available data, exposure to the tailings and ore could theoretically result in doses that exceed the Health Canada (2011) dose constraint for uranium of 0.3 mSv/y and the incremental dose of 1 mSv/y that are used for the protection of the public (NECA, 2015). However, the public would have to be exposed for a full year, which is unlikely as the operating mine site will not be accessible to the public. At closure, when local residents may seek to inhabit the area again, long-term exposure will be managed by constructing a cover on the tailings storage facility that should reduce exposure.

Limiting exposure to radium-226 can also be achieved by constructing open walled structures in proximity to the TSF. Radium-226 can build up within walled structures over time, which could result in a higher dose to anyone entering those buildings. Open walled structures allow for the passage of air, which alleviates the risk of radium-226 buildup. No buildings will remain on the covered TSF after closure.

Doses decrease significantly with lower hours of workers, and as such, the preliminary external dose estimate for workers is considerably less than the Health Canada (2011) dose constraint for occupationally exposed workers.

A conservative approach has been taken to account for the uncertainty in the preliminary assessment. The TSF and its closure cover have taken the potential radiological risks into account.

20.1.8 Water Management

The River Cacheu is the primary receiving water for discharges from the mine site. The river experiences tidal flows resulting in a changing mixture of fresh and brackish water. As such, despite the distance of the mine site to the sea, it is appropriate in the evaluation of surface water impacts to apply water quality standards applicable to estuarine and inner coastal water quality.

Standards for estuarine water quality tend to vary significantly as they are dependent on both geology and geography. In addition, many water quality thresholds derived from available standards are already exceeded by the baseline water quality in the River Cacheu. Hence a “no worsening” approach is adopted for parameters that exceed the target water quality standards.

There are no published mine effluent discharge standards or estuarine or inner coastal receiving water quality guidelines in Guinea-Bissau. IFC’s mine effluent guidelines (IFC, 2007b) have been adopted as the targeted end-of–pipe discharge standards (Table 20-8).

Target receiving water quality standards for the project were developed considering the estuarine receiving water quality guidelines from the following jurisdictions:

- South African Water Quality Guidelines for Coastal Marine Waters (Department of Water Affairs and Forestry, 1996)
- Australian Tropical Estuaries Standards (Australian and New Zealand Environment and Conservation Council – ANZECC), 2000)
- Trinidad and Tobago Coastal Nearshore Standards (Trinidad and Tobago Water Pollution Management Program, 2005)
- Canadian Environmental Quality Guidelines – Guidelines for the Protection of Marine Aquatic Life (Canadian Council of Ministers of the Environment, 2015).

Table 20-8 presents target water quality standards for the River Cacheu which were established considering the most stringent of the receiving water quality guidelines listed above. For parameters in which the 75th percentile of dry season baseline water quality exceeded the most stringent published guideline value, the 75th percentile of the dry baseline value was adopted under the principle that the project should not worsen water quality within the river.

Applying the approach of "no worsening" of water quality and considering the apparent differences in wet and dry season water quality in the River Cacheu, it is appropriate to establish separate water quality objectives for the wet season (and enhance the baseline dataset for the dry season) once sufficient baseline data has been generated. The adopted dry season water quality standards are presented in Table 20-8.

Table 20-8: Mine Effluent Discharge Guidelines and Target Receiving Water Quality Standards for the Dry Season

Parameters	Units	IFC Mine Effluent Guidelines	River Cacheu Target Receiving Water Quality Standards
Physical Parameters			
pH Value	pH	6 to 9	6.5 to 8.5
Dissolved Oxygen	%		80 to 120
General Parameters and Nutrients			
Ammonia	mg/L		0.02
Ammonia as N	mg/L		0.074
Chloride	mg/L		4280
Nitrate as N	mg/L		0.42
Nitrite as N	mg/L		0.0014
Total suspended solids	mg/L	50	22 to 112
Total Nitrogen as N	mg/L		0.25
Total Phosphorus as P	mg/L		0.043
Turbidity	mg/L		20 to 46
Total Metals			
Arsenic	µg/L	100	12
Cadmium	µg/L	50	0.12
Chromium	µg/L		8
Copper	µg/L	300	5
Iron	µg/L	2,000	3,500
Lead	µg/L	200	12
Mercury	µg/L	2	0.016
Nickel	µg/L	500	25
Silver	µg/L		5
Zinc	µg/L	500	25

20.1.9 Environmental Monitoring

An Environmental and Social Management Plan (ESMP) was prepared as part of the 2015 ESIA. The ESMP summarizes the Company`s commitments to address and mitigate risks and impacts identified as part of the ESIA. Mitigation strategies include avoidance, minimization, and compensation/offset. The ESMP consists of the following:

- Level 1 – Management System – This includes 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 ongoing 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 was part of the ESIA but will be a live, stand-alone document to 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 have been developed to date:

- Air Quality Management Plan
- Noise Management Plan
- Erosion Control and Sedimentation Plan
- Water Management Plan and related Water Monitoring Manual
- 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 Action Plan.

These plans will need to be updated prior to construction. The Resettlement Action Plan (RAP) requires updating as described in Section 20.2. 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.

Additionally, an Influx Management Plan will be developed and implemented in consultation with the local communities, to minimize in-migration to the area, and to prepare to respond to the influx that does occur.

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 sites) and Grievance Mechanism are two Level 3 SOPs that will be developed as part of the ESMP.

20.2 Expected Material Environmental and Social Impacts and Risks

Based on the work to date, there have been no environmental or social issues that are expected to prevent Itafos from developing the project.

The project is expected to result in adverse environmental, socioeconomic 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:

- 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, which could become mobilized into the environment under acidic leaching conditions. A portion of the waste overburden is potentially acid generating (PAG). Additionally, a 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; KP, 2017). Containment of these materials will be necessary to prevent seepage of adverse quality effluent to groundwater and to prevent discharge to surface waters. The PAG overburden will be placed on a basal liner in WD-1 and become encapsulated by NPAG materials to limit the ingress of oxygen and water during operations. At closure the waste dump will be covered, such that oxygen ingress to the PAG material is significantly reduced and atmospheric water will be shed from the waste dump to limit water ingress resulting in seepage. The tailings storage facility will require sufficient cover at closure to shield future land users from any radioactivity. Additionally, no PAG material will be placed in the TSF to reduce the likelihood for acidic leachate, which may cause the radioactive isotopes from the tailings solids to mobilize into solution.
- Pit Dewatering Affecting Household and Community Wells – Dewatering of the open pits will create a drawdown cone that will likely affect some existing community wells. The company will need to establish plans to provide water of at least equivalent quantity and quality to affected groundwater users in the area. A hydrocensus monitoring program has been started and should be completed to be able to verify any claims of effects to household and community wells.
- Ecological Impacts – The project will result in the loss of mangroves, salt marsh and freshwater areas, as well as secondary forest. Lost mangrove habitat may in turn contribute to riverbank instability and erosion coupled with a loss of crocodile habitat. Establishment of the Buredanfa Resettlement Village and associated livelihood restoration area will result in the loss of indigenous forest. A decrease in forest habitats represents a loss of habitat for primates. Updated ecological studies should be completed before significant construction activities begin to confirm current inventories and status of species of conservation concern, and to implement an updated Biodiversity Management Plan, potentially with biodiversity offsets.
- Community Health, Safety and Security – The project will interrupt the current flow of mostly pedestrian and bicycle traffic between the regional service center of Farim to villages to the west and north of the mine. In addition, the presence of the mine 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 Mineral Terminal 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. The effects can be far-reaching in terms of social unrest and conflict, overloading of available public services and infrastructure, and causing increased pressures on ecological resources. Significant influx has occurred in the nearby Boké region of Guinea, as large numbers of people moved into the area in response to the development of the bauxite industry in that region. A Community Health, Safety and Security Management Plan identifies these issues and proposes preliminary mitigation measures that can be discussed with the appropriate authorities. Influx should be addressed in the CHSS management plan or a separate plan.
- 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). Nonetheless, a monitoring program is recommended to verify that external doses to workers are within acceptable limits. A monitoring program is described in a preliminary Occupational Health and Safety Plan.

- 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.
- Involuntary Resettlement – The project will require the acquisition of approximately 3,000 ha of land resulting in the physical and/or economic displacement of an estimated 175 households in villages in the mine area. A Resettlement Policy Framework was presented in the 2015 ESIA (Nomad Socio Economic Management and Consultancy (Pty) Ltd. [Nomad Consulting], 2015). This framework document and the ESIA proposed the relocation of the households in these villages to a new village to be established to the north of the proposed mine. Other mine project components such as the truck loadout facility, highway bypass around the town of Mansoa, and the Mineral Terminal facility and associated access road will be positioned on lands held by others. Compensation is planned as part of securing land tenure for these areas, though no household resettlement is required. Since completion of the ESIA in 2015, Itafos has undertaken the following studies supporting its resettlement planning efforts:
  - Candidate host sites were identified, and the preferred site was selected at Buredanfa (Nomad Consulting, 2016).
  - A livelihoods baseline and restoration strategy was prepared (Nomad Consulting and Africa-Wide Consulting, 2017).
  - A Resettlement Action Plan was prepared (Nomad Consulting, 2017).
  - Itafos has identified a restoration area for the Buredanfa Resettlement Project and a proposed alignment for a village bypass road around the northern boundary of the mine. Preliminary village designs have been discussed with the affected villages.

  Because of the time that has passed since this work was completed, the communities that require resettlement may have grown, and it will be necessary to conduct another land and asset survey and update the Resettlement Action Plan. It is possible that social conditions may have meaningfully changed since 2015, such as influx into the proposed land restoration area.
- Terrestrial Ecology – The restoration area is comprised of mainly natural forest and secondary forest vegetation communities, and it is anticipated that much of this area could be lost over time due to vegetation clearing and conversion of the land to agricultural land or community lands. The magnitude of impacts on the natural forest and secondary forest vegetation communities are moderate are high and low, respectively. The significance of vegetation clearing effects on the natural forest community is rated "major" in the absence of any mitigation, decreasing to "moderate" with protection and offsetting mitigation measures, with any offsetting accomplished through incorporation of the Buredanfa Restoration Area into any offsetting proposed under the project's Biodiversity Management Plan. The significance of effects on the secondary forest community was rated as "low".
- Cultural Heritage – Development of the project will result in direct and unavoidable physical impacts on the following cultural heritage resources:
  - three cemeteries (one of high and two of low sensitivity)
  - two mosques (both of high sensitivity)
  - three sacred sites (one of high and two of low sensitivity)
  - six archaeological sites (two of medium and four of low sensitivity).

  Mitigation of these resources will involve the development of a grave relocation plan, a sacred site relocation plan, and a mosque relocation plan. The re-interments should be conducted in consultation with local communities, with the involvement and agreement of the local community and relatives of the deceased. Sacred forests and mosques should also be relocated in consultation with affected communities. Relocation of sacred forests or sites usually refers to moving spirits as well as any ritual huts or offering jars from one tree or grove to another through the appropriate ritual ceremony and conditions. The archaeological sites will require data recovery/rescue excavations prior to construction. A Cultural Heritage Management Plan was developed as part of the ESIA (KP, 2015m). The potential risk of mitigating these cultural heritage features is the potential for conflict with local communities while negotiating the mitigation plans, and any potential delays to construction that such issues may cause.

20.3 Closure and Reclamation Planning

20.3.1 Closure and Reclamation Plans

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 20.4). 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 specifications and practices established by international standards and regulations.

A preliminary MRCP and closure cost estimate has been prepared as part of the feasibility study (KP, 2022). The MRCP adopts the IFC closure objectives, as follows:

- future public health and safety are not compromised
- after-use of the site is beneficial and sustainable to the affected communities in the long term
- adverse socioeconomic impacts are minimized, and socioeconomic benefits are maximized.

The MRCP contemplates the progressive rehabilitation of several facilities at the mine including TSF cells, the overburden waste dumps, and the North and South open pits. The South pit and most of the North pit will be backfilled with waste overburden as mining progresses. 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 that will be connected to the Rio de Cavaras Marinhos (Figure 20-2).

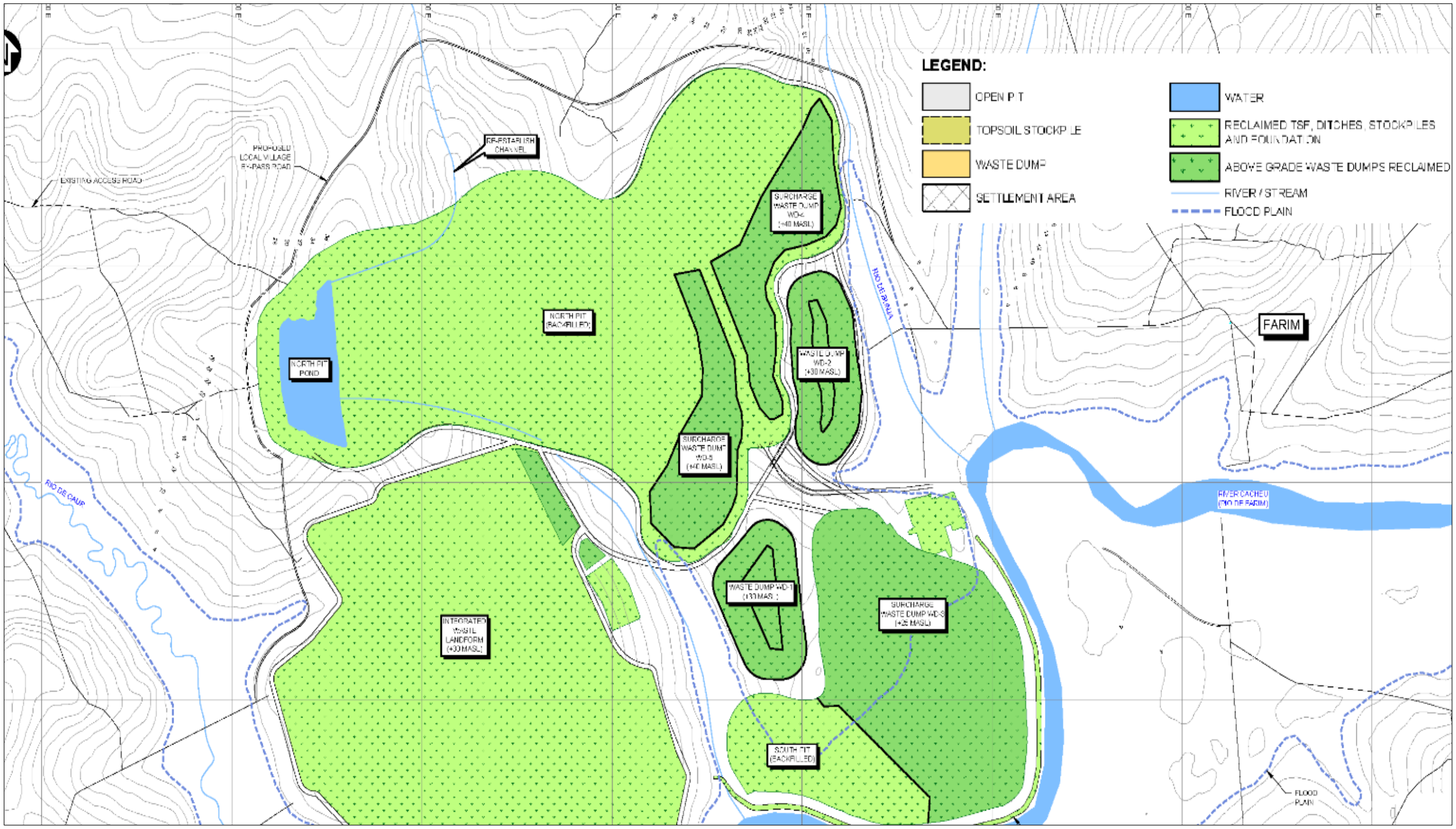
The TSF will be progressively covered with 1.5 m of NAG waste overburden. About 8 bMm^3 of waste overburden will be required (of an estimated 430 bMm^3). The first six cells will be covered during mine operations by diverting NAG waste overburden. Cell 7 will be closed out during the two-year active closure phase.

The 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 Mineral Terminal site, buildings, machinery and equipment will be decommissioned and removed from the site. Remediation will be undertaken, as required, so that the Mineral Terminal 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 years 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 socioeconomic impacts of closure as part of the company's Community Development Plan.

Figure 20-2: Post-Closure Mine Layout



Source: Knight Piesold, 2022

20.3.2 Water Management Post-closure

Post-closure water management at the mine site will consist of the following:

- Surface and storm water will continue to be diverted away from the mining wastes and managed using 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-3 in Figure 20-2), 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.
- Most of the North pit will be backfilled during operations, and two surcharge waste dumps (WD-4 and WD-5 in Figure 20-2) 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-4 and WD-5. The option of not backfilling and allowing a small pit lake or pond to form at the most westerly portion of the North pit and connecting the pit lake to a re-established Rio de Caravas Marinhos 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 runoff. The Environmental Control Dams will have been decommissioned and the local drainage patterns (Rio de Caur, Rio de Caravas 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.

20.4 Permitting Considerations

20.4.1 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, several laws related to environmental protection and management have been passed. The legislation most relevant to the project is summarized in Table 20-10.

The Secretary of State of the Environment is the public institution responsible for defining, coordinating, and implementing public environmental policies and actions towards sustainable development, environmental protection and international environmental commitments.

Table 20-9: Applicable National Legislation

Subject	Laws and Decrees	Notes
Basic Legislation on Environment	Law nº 1/2011, 2nd of March	It defines the basic concepts and specifies the norms, and the basic principles related to policies and activities of protection, preservation and conservation of the environment of the Republic of Guinea-Bissau. It also promotes the improvement of the quality of life through correct management of the National environment and a rational use of natural resources, in order to optimize and to guarantee the sustainability and continuity of the use of such resources. In addition, it creates the Environmental Fund.
Environmental Assessment Law	Law nº 10/2010, 24th of September	Defines the fundamental principles and methodologies of national environmental assessment process for projects, plans and programs. Projects subject to EA need a positive environmental certificate before issuing any License and works start. Project categorization on A, B and C regime. This law leaves open for future diplomas to rule and detail subjects such as public participation procedures, environmental audits, revenue distribution from taxes and fines, likewise accreditation of companies to prepare ESIA reports and studies.
Public Participation	Decree nº 5/2017, 28th of June	Defines the different procedures of public participation under the process of Environmental (and Social) Assessment law.
Environmental Fund	Decree nº 6/2017, 28th of June	A fund created to promote the protection of national natural resources and the environment, dedicated to promoting activities of sustainable natural resource management, environmental education, restoration of degraded habitats, support environmental inspection and the environmental assessment process, among others.
Environmental and Social Impact Study	Decree nº 7/2017, 28th of June	Definition of different stages of EA process such as previous examination and project categorization, the terms of reference of the ESIA study, duties of Competent Environmental Assessment Authority (AAAC), sanctions, fines, among others.
Environmental License	Decree nº 8/2017, 28th of June	Regulates the procedures of environmental licensing of projects, different stages of the process and different entity duties, sanctions, fines, among others.
Environmental Audit	Decree nº 9/2017, 28th of June	Defines the procedures for environmental audits to projects, plans, programs and policies; the role of different entities, sanctions, fines, among others.
Environmental Inspection	Decree nº 10/2017, 28th of June	Defines the procedures for environmental inspection, sanctions, fines, among others.
Protected Areas Law	Decree-law nº 5-A/2011, 1st of March	Defines the protection of fauna, flora and ecosystems inside protected areas, including the procedures to take into account such as environmental assessment of projects and activities inside these areas.
Forestry Law	Decree-law nº 5/2011, 22nd of February	Regulates forestry activities in the country; it stipulates concessions or other forestry activities that require an environmental license.
Water Code	Decree-law nº 5-A/1992, 17th of September	Framework for water resources management in Guinea-Bissau. Stipulates the requirement of an environmental impact study on waters when a project may affect water quality.
Law of Mines and Minerals	Law nº 3/2014, de 29 de Abril	Regulates the extraction activities of minerals and mines. It stipulates that in order to be awarded with the mining title/permit an environmental impact assessment must be prepared to prevent, reduce, control and compensate for the project's environmental and social impacts.
Law of Petroleum	Law nº 4/2014, 15th of April	Defines the regime of oil/hydrocarbons search and exploitation. It stipulates that in order to be awarded with the search or exploitation title/permit an environmental impact assessment must be prepared to prevent, reduce, control and compensate for the project's environmental and social impacts.
Law of Land	Law nº 5/1998, 28th of April	Defines the regime of access to land in Guinea-Bissau. The land belongs to the State, it is State property, only through concession people and private sector may have access to land.
Labor Law (health and safety procedures)	Decree nº 2/2012, 3rd of January	Stipulates the requirement at work of health and safety plans to ensure adequate working conditions and basic medical service, as well as minimum wage, among others.
Gender based violence Laws	Law nº 14/2011, 6th of July Law nº 6/2014, 4th of February	First law establishes de combat and repression to the practice of female genital mutilation; the second law relates to domestic violence (all different forms) and, among other aspects, considers it a public crime.
Gender Parity Quota Law	Law in 2018, 12th of September	Stipulates a minimum quota for women in decision-making positions and at elections positions of 36%.
Planning and Land Use	Decree nº 17/95, 30th of October	Approves the Urban Plan of the City of Bissau, its zoning and regulation for the next 20 years. Now outdated.

The Secretary of State of the Environment is the head of five other important institutions that work together to accomplish these objectives:

- Directorate General of the Environment (DGA)
- Directorate General of Sustainable Development (DGDD)
- Competent Environmental Assessment Authority – Autoridade da Avaliação Ambiental Competente (AAAC)
- Institute for Biodiversity and Protected Areas (IBAP)
- General Inspection of the Environment.

The Competent Environmental Assessment Authority (AAAC), the Institute for Biodiversity and Protected Areas (IBAP), and the General Inspection of the Environment play important roles on project licensing, on the environmental and social impact assessment process, on the management of country's protected areas for nature conservation purposes and on the surveillance/control of environmental compliance of all actors of all sectors and activities.

The AAAC, previously named Célula de Avalição de Impacte Ambiental (CAIA) is the lead authority responsible for coordinating review of ESIAs. This department is responsible for ensuring, through collaboration with other relevant government departments, that all development projects are analyzed 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 makes a recommendation to the Ministry of Natural Resources and Energy regarding the implementation of the project based on AAAC's review of the ESIA. AAAC will then issue an environmental license 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 license 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 license.

Institute for Biodiversity and Protected Areas (IBAP) manages all protected areas in Guinea-Bissau; it plays a very important role on the environment and social impact assessment process of projects that affect protected areas (marine and terrestrial). Nowadays, 26% of the territory of Guinea-Bissau is classified under legal protection for nature conservation.

The Directorate General of the Environment (DGA) and Directorate General of Sustainable Development (DGDD) are responsible for designing and implementation of the country's environmental policies and international environmental commitments.

20.4.2 Environmental Permits

An ESIA for the project was published in September 2015 and was shared with the Government of Guinea-Bissau and prospective lenders (KP, 2015a). An ESIA was subsequently prepared for the Buredanfa Resettlement Village (KP, 2019). The project and the Buredanfa Resettlement Village were approved by the Government of Guinea-Bissau, according to a Declaração de Conformidade Ambiental (Declaration of Environmental Compliance) issued to Itafos on September 14, 2018 (Secretaria de Estado do Ambiente, 2018).

While the approval (declaration) expired in 2019, the Autoridade da Avaliação Ambiental Competente (Competent Environmental Assessment Authority) notified Itafos in March 2020 that the Authority had almost completely suspended its operations, and that the process of renewal of the environmental license will resume when the pandemic is over (Autoridade da Avaliação Ambiental Competente, 2020). To date, no further notification from the Autoridade da Avaliação Ambiental Competente has been received indicating the resumption of operations and the envisaged renewal of the environmental license.

20.4.3 Mining Permits

The project is covered by a Mining Agreement and Mining Lease 004/2009, covering 30,625 hectares (ha), granted by the Government of Guinea-Bissau on May 28, 2009 to GB Minerals AG (GBMAG). Details regarding the Mining Agreement and Mining Lease are provided in Section 4.2.

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 willful intent is demonstrated.
- The licensee shall not be held liable for environmental damages that may result from Mineral Terminal 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 behavior.
- 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.

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).

20.5 Social Considerations

Public consultation was carried out between 2011 to 2015 leading up to the issuance of the project's ESIA, and these activities are described in Volume I of the ESIA (KP, 2015a). Key issues and concerns summarized in that document include a distrust of the project because of claims of promises regarding resettlement and compensation that have never materialized, a distrust of the government in relation to the project, concern over needing to be resettled, and opportunities for employment that the project represented.

Itafos maintained an active dialogue with the local communities around the mine until the COVID-19 pandemic required these activities to cease. However, active dialogue was re-established in late 2022. Engagement following the ESIA submission focused on presenting the outcomes of the assessment, and this transitioned to a greater focus on resettlement in 2016 through 2018 as Itafos developed its Resettlement Action Plan in consultation with the communities. Review of Itafos' most recent consultation records on resettlement from 2018 indicated concerns about the following key issues:

- resettlement
- bypass road around the mine.

Other than consultations on resettlement, there have been no negotiations or agreements discussed with local communities or authorities to date.

Potential social impacts that need to be carefully monitored as the project advances to construction include the following:

- Potential impacts to community groundwater users due to mine dewatering – This is identified as a material environmental and social impact or risk in Section 20.2. It will be important to complete the previously started hydrocensus so that Itafos has an accurate account of where wells are located, and their tendency to run dry or other issues in the absence of the project, so that future claims of impacts can be investigated for potential project impacts. Similarly, continued measurement of water levels in the areas around the community will be important to establish a baseline of sufficient duration as to be helpful in sorting out effects from the project.

- Cansenha Bypass Road – Because the mine will cut off access to the Town of Farim by villages west of the mine, it will be necessary to establish a new road around the mine. This was identified as a concern in 2018 community meetings, where community members expressed concern that the proposed bypass road doubled the distance to Farim. Many women carry goods on their head, and the new distance will be difficult for them. Consideration should be given to further mitigation measures, such as operating a basic bus service.
- Involuntary Resettlement – The International Finance Corporation's (IFC's) Guidance Note 5 on Land Acquisition and Involuntary Resettlement states, *"Unless properly managed, involuntary resettlement may result in long-term hardship and impoverishment for the Affected Communities and persons, as well as environmental damage and adverse socioeconomic impacts in areas to which they have been displaced."* (IFC, 2012). It is important to recognize the risks that resettlement poses and to dedicate adequate resources to its implementation, to minimize unintended impacts.
- Relocation of cemeteries, sacred forest, and a mosque – Plans for relocation of these important cultural heritage features will need to be developed in consultation with the community. This could take time and could become on the critical path for construction if meaningful delays are experienced. Negotiation of these plans is also a potential source of conflict between the company and the communities.

21 CAPITAL AND OPERATING COSTS

The capital cost estimate summarized in Table 21-1 provides a summary of the total project capital cost estimate, with costs grouped into major scope areas by initial and sustaining capital costs. The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International). The costs are expressed in Q4 2022 US Dollars.

Table 21-1: Summary of the Farim Project Capital Costs

Description	Initial Capital (US$M)	Sustaining & Closure Capital (US$M)	Total Capital (US$M)
Mining	32.243	265.348	297.591
Process Plant and Infrastructure	68.934	-	68.934
Ponte Chugue Infrastructure (Mineral Terminal & Drying)	99.728	12.050	111.778
Tailings Storage Facility & Water Management	14.049	57.722	71.771
South Pit Dewatering	4.420	12.737	17.157
North Pit Dewatering	-	20.995	20.995
Resettlement and Livelihood Restitution	11.985	5.635	17.620
EPCM	27.452	-	27.452
Indirects	6.057	-	6.057
Owners' Cost	11.637	-	11.637
Contingency	31.765	-	31.765
Progressive Closure and Rehabilitation (TSF)	-	58.817	58.817
Progressive Closure and Rehabilitation (Pits & WDs)	-	21.169	21.169
Total Site Closure	-	33.997	33.997
Salvage Value – Mine	-	-12.893	-12.893
Salvage Value – Port	-	-8.433	-8.433
Total	308.270	467.142	775.413

Operating costs include the ongoing cost of operations related to mining, processing, tailings disposal, shiploading, and general administration activities. The operating cost estimate is presented separately for the south and North pit operations due to the varying quantities of concentrate produced in each phase.

The operating cost estimate was developed by Ausenco, WSP, and Baird for processing, mining, and shiploading costs respectively. Labor rates and general administration costs were provided by Itafos. A summary of the operating costs is presented in Table 21-2.

Table 21-2: Summary of Farim Project Operating Costs (with fuel itemized separately)

Description	Life-of-Mine Operating Cost			South Pit			North Pit		
	US$M	US$/t Feed	US$/t Conc.	US$M/y	US$/t Feed	US$/t Conc.	US$M/y	US$/t Feed	US$/t Conc.
Mining	661.4	15.1	20.1	31.3	17.9	23.1	24.6	14.0	18.9
Process	343.0	7.8	10.4	13.9	7.9	10.3	13.6	7.8	10.5
Shiploading	111.3	2.5	3.4	4.5	2.5	3.3	4.5	2.5	3.4
Tailings, Environment, Water	15.7	0.4	0.5	0.6	0.4	0.5	0.6	0.4	0.5
G&A	186.8	4.3	5.7	7.5	4.3	5.5	7.5	4.3	5.7
Fuel	952.3	21.8	28.9	35.4	20.2	26.1	39.1	22.4	30.1
Total	**2,270.5**	**51.9**	**69.0**	**93.2**	**53.2**	**68.7**	**89.9**	**51.4**	**69.1**

Note: Fuel is itemized separately and is not included in mining, processing, shiploading or G&A costs.

21.1 Capital Costs

21.1.1 Mining Capital Costs

Capital costs represent the investment in physical assets required to facilitate overburden removal, matrix production, processing and delivery of the finished rock product to the Mineral Terminal. 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 QP 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 operational life. Sustaining capital includes equipment replacement and rebuilds, equipment capital additions, haul road development, and minor miscellaneous capital requirements.

The capital expenditures developed for this Section 21.1.1 refer to those items directly related to mining and include mining equipment, road development, and water management within the developing pit. The QP estimated the capital requirements associated with the 1.75 Mt/a (product tonnage) mine plan. The overall mining cost estimate was predicated upon the assumption that a mining contractor will be used for Years 0 through 5 and will perform all direct mining functions, excluding waste dump base development and ground water pit dewatering. The development of the contractor mining cost estimate is described further in Section 21.1.1.1. Beginning in Year 6 of the mining plan, costs are estimated based on the assumption that the mine will be owner operated. The mining cost estimate reference point is the plant ROM hopper.

For this report, 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.

21.1.1.1 Contractor Mining Capital Cost

In 2019, the QP requested quotes for contract mining of the deposit from Year 0 through Year 5. The quote, provided by PW Mining International Ltd., (PW) headquartered in Accra, Ghana, provided the quote based on using similar-sized mining equipment and truck shovel methods through Year 5 of the mining plan. For this technical report, the QP escalated the contractor bid to 2022 US dollars. The escalation was based on the producer price index (PPI) inflation between 2019 and 2022, which resulted in an escalation rate of 1.11 for labor and 1.12 for equipment. The escalated costs provided included a fixed cost for mobilization and set up of $3.76 million in Year 0 and another $1.98 million in Year 5 for demobilization. Additionally, a capital cost of $2.97 million has been included for site establishment in Year 0. The Operating Costs for the contractor mining period are discussed in Section 21.2.3.1. The overall contractor costs for Years 0 through 5 are summarized in Table 21-3.

Table 21-3: Summary of Contractor Cost Model

Total 2022 Costs	Units	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mobilization	$k	3,762	0	0	0	0	0	3,762
Demobilization	$k	0	0	0	0	0	1,978	1,978
Site Establishment	$k	2,965	0	0	0	0	0	2,965
Road Construction Cost	$k	1,764	718	881	1,301	1,133	1,468	7,265
Subtotal - Contractor Capital Expenditures	$k	8,491	718	881	1,301	1,133	3,446	15,970
Management Fee	$k	9,052	8,995	8,976	8,889	8,919	8,868	53,699
Mining Works Load Haul	$k	12,479	30,002	37,051	34,482	35,462	33,234	182,710
Mining Works Ore Handling	$k	0	1007	987	989	984	992	4,959
Subtotal - Contractor Operating Expenditures	$k	21,531	40,004	47,014	44,359	45,365	43,094	241,368
Contractor Total	$k	30,023	40,723	47,894	45,660	46,498	46,540	257,338

When the bid was submitted in 2019, the QP identified critical items missing from the PW bid that were necessary for the total project cost estimate. These costs were primarily related to the haul road and included road construction and armoring during the first five years with armoring material sourced from local quarries. The cost model incorporated the escalated contractor costs and modifications discussed above for Years 0 through 5.

21.1.1.2 Owner-Operated Mining Capital Costs

Unit capital costs for primary production equipment for Year 6 and beyond were generally based on dealer/manufacturer budgetary price quotes and Golder file data. The QP obtained capital cost estimates from dealers or suppliers during the second quarter (Q2) of 2015 and escalated these quotes to 2022 US dollars. 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 2015 manufacturer quotes, primarily from local manufacturers in Africa, and escalated to 2022 US dollars.

Primary equipment requirements for the 1.75 Mt/a case for this study are listed in Table 21-4. The detailed equipment requirements were previously shown in Section 16.8. The table shows the initial units required for the preproduction 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.

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 in Table 21-5.

Table 21-4: Summary of Equipment Requirements

Description	Initial Units Required	Additional/ Replacement Units	Total Units Purchased	Rebuilds
Primary Mining Equipment				
Wheel Loader - 12.2 m^3 Class	6	17	23	22
Backhoe - 5.0 m^3 Class	3	0	3	0
End Dump Truck – 97 t Capacity	16	44	60	41
End Dump Truck – 36 t Capacity	7	5	12	8
Major Support Equipment				
Backhoe - 2.1 m^3 Class	1	0	1	0
Dozer – 405 hp Class	5	2	7	11
Compactor – 147 hp Class	6	9	15	14
Motor Grader – 297 hp Class	3	6	9	6
Backhoe - 1.0 m^3 Class	1	1	2	1
Water Truck - 34,000 L Tank Capacity	2	2	4	4

Table 21-5: 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
Backhoe	1.0 m^3 bucket	60,000	30,000
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

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, the QP used major equipment rebuilds to extend the effective lives of the backhoes, wheel loaders, haul trucks, water trucks, graders, compactors and dozers.

The QP quantified equipment replacement in terms of cumulative machine operating hours. Actual operating hours are a function of operating conditions, intensity of equipment use, and basic machine design. The QP based replacement and rebuild intervals operations on experience and available manufacturer/dealer guidelines. The QP's equipment replacement occurred once the cumulative operating hours for an individual equipment unit surpassed or approached the estimated machine life.

Table 21-6 summarizes estimates the equipment capital requirements for the project mine plan. New unit purchase requirements total $65.0 million with primary mining equipment accounting for 93% of the initial requirements and Support equipment and spare parts account for the remaining 7% of new unit purchase requirements.

Other major capital expenditures including early road construction in Years 0-5 ($4.7 M in Years 0-1, $10.3 M in Years 2-5) and the in-pit mine dewatering system estimated at $4.8 M. Dewatering of groundwater in advance of mining and surface water management costs were covered by others. An additional $158.6 million in equipment replacement and rebuild costs is anticipated over the life of mine. The overall capital cost schedule is provided in Table 21-7 and Figure 21-1. The estimated capital did not include the costs related to maintenance facility construction and other support infrastructure, such as office buildings and fuel islands. These costs are covered in Sections 21.1.2 and 21.1.3.

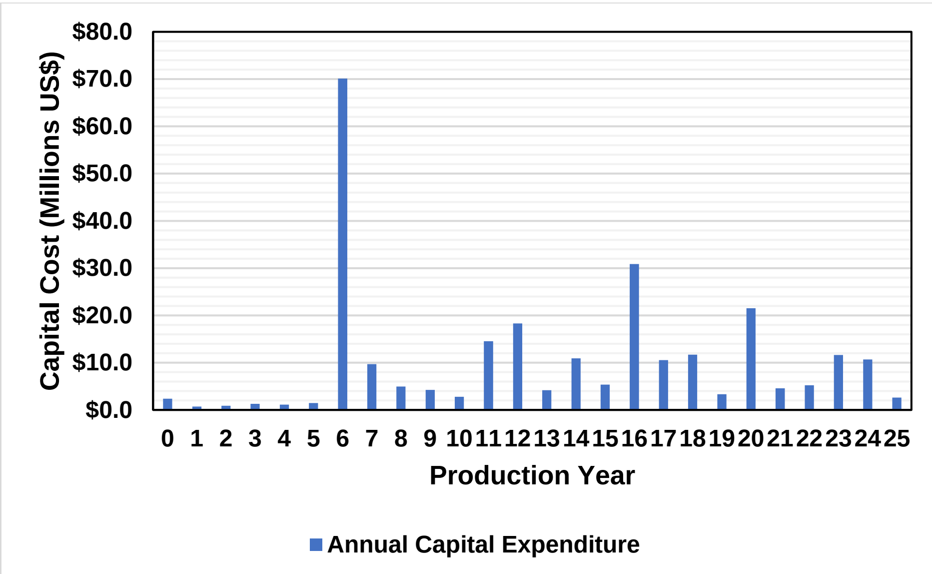
Table 21-6: Summary of Mining Equipment Capital Expenditures

Description	New Unit Purchase ($k)	Replacement & Rebuild Costs ($k)	Total Equipment Capital Cost ($k)
Primary Mining Equipment ($k)	59,433	153,214	212,648
Wheel Loaders	15,364	55,427	70,791
Excavators / Backhoes	3,108	155	3,263
Dozers	4,728	3,221	7,949
Haul Trucks	30,269	83,223	113,492
Motor Grader	2,898	6,636	9,534
Water Trucks	1,881	2,385	4,267
Compactors	1,184	2,167	3,352
Support Equipment	4,438	6,458	10,895
Total Estimated Equipment Capital Expenditure	63,871	159,672	223,543

Table 21-7: Summary of Yearly Estimated Capital Expenditures for Mining

Machine / Item	Capital New / Rebuild	Replacement Life (hours)	Unit Cost (US$k)	Depreciation 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	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Stripping & Loading Equipment																														
Caterpillar 992K - Wheel Loader	New / Replace	36,000	2,606	7	0	0	0	0	0	0	15,634	0	2,606	0	0	2,606	15,634	0	2,606	0	0	0	2,606	0	15,634	0	2,606	0	0	0
Caterpillar 992K - Wheel Loader	Rebuild	18,000	357	3	0	0	0	0	0	0	0	0	0	2,145	0	357	0	0	357	0	2,145	0	357	0	0	0	357	0	2,145	0
Caterpillar 374DL - Backhoe	New / Replace	60,000	1,036	10	0	0	0	0	0	0	3,108	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Caterpillar 374DL - Backhoe	Rebuild	30,000	124	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Caterpillar 336DL - Backhoe	New / Replace	48,000	464	7	0	0	0	0	0	0	464	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Caterpillar 336DL - Backhoe	Rebuild	24,000	55	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Caterpillar D9R - Dozer	New / Replace	48,000	946	10	0	0	0	0	0	0	4,728	0	0	0	946	0	0	0	0	0	0	0	0	0	946	0	0	0	0	0
Caterpillar D9R - Dozer	Rebuild	24,000	91	5	0	0	0	0	0	0	0	0	0	0	0	454	0	0	0	91	0	0	0	0	0	454	0	0	0	0
Spare Parts Inventory (@ 5%)	Other	n/a	n/a	5	0	0	0	0	0	0	1,197	0	130	0	47	130	782	0	130	0	0	0	130	0	829	0	130	0	0	0
Haul Trucks																														
Caterpillar 777G - End Dump Truck	New / Replace	60,000	1,559	10	0	0	0	0	0	0	24,949	7,797	0	0	0	6,237	0	1,559	4,678	0	24,949	6,237	6,237	1,559	1,559	1,559	0	6,237	0	0
Caterpillar 777G - End Dump Truck	Rebuild	30,000	126	5	0	0	0	0	0	0	0	0	0	0	0	2,647	0	0	0	504	0	0	0	126	378	0	0	1,513	0	0
Caterpillar 770 - End Dump Truck	New / Replace	60,000	760	10	0	0	0	0	0	0	5,320	0	0	760	0	0	0	760	0	0	0	0	0	0	0	760	0	760	760	0
Caterpillar 770 - End Dump Truck	Rebuild	30,000	64	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	446	0	0	64	0	0
Spare Parts Inventory (@ 5%)	Other	n/a	n/a		0	0	0	0	0	0	1,513	390	0	38	0	312	0	116	234	0	1,247	312	312	78	78	116	0	350	38	0
Mobile Equipment																														
Caterpillar 16M - Motor Grader	New / Replace	42,000	966	7	0	0	0	0	0	0	2,898	0	0	0	0	0	0	0	0	2,898	0	0	0	0	0	0	0	0	2,898	0
Caterpillar 16M - Motor Grader	Rebuild	28,000	17	3	0	0	0	0	0	0	0	0	0	0	0	0	202	0	0	0	0	0	0	0	0	202	0	0	0	0
Caterpillar 770 - Water Truck	New / Replace	LOM	941	10	0	0	0	0	0	0	1,881	0	0	0	0	0	0	0	0	0	0	1,881	0	0	0	0	0	0	0	0
Caterpillar 770 - Water Truck	Rebuild	24,000	79	5	0	0	0	0	0	0	0	0	0	0	0	158	0	0	0	0	0	0	0	0	0	0	158	0	0	0
Caterpillar CS-56 - Compactor	New / Replace	42,000	197	10	0	0	0	0	0	0	1,184	0	197	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1,382	197
Caterpillar CS-56 - Compactor	Rebuild	28,000	17	5	0	0	0	0	0	0	0	0	0	0	0	0	104	0	17	0	0	0	0	0	0	104	0	17	0	0
Spare Parts Inventory (@ 5%)	Other	n/a	n/a		0	0	0	0	0	0	298	0	10	0	0	0	0	0	0	145	0	94	0	0	0	0	0	0	214	10
Service & Support Equipment																														
Caterpillar 428F - Backhoe Loader	New / Replace	60,000	130	10	0	0	0	0	0	0	130	0	0	0	0	0	0	0	0	0	0	0	130	0	0	0	0	0	0	0
Caterpillar 428F - Backhoe Loader	Rebuild	30,000	12	5	0	0	0	0	0	0	0	0	0	0	0	0	12	0	0	0	0	0	0	0	0	0	0	0	0	0
Fuel / Lube Truck	New / Replace	40,000	425	10	0	0	0	0	0	0	1,275	0	425	0	0	0	0	0	1,275	0	425	0	0	0	0	0	0	0	1,275	425
Fuel / Lube Truck	Rebuild	20,000	64	5	0	0	0	0	0	0	0	0	0	0	191	0	64	0	0	0	0	0	191	0	0	64	0	0	0	0
Mechanic's Truck	New / Replace	30,000	87	10	0	0	0	0	0	0	173	0	0	0	0	0	0	173	0	0	0	0	0	0	173	0	0	0	0	0
Mechanic's Truck	Rebuild	30,000	13	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pickup Truck	New / Replace	30,000	52	7	569	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	569	0	0
Pickup Truck	Rebuild	30,000	8	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Liebherr LTM 1095 - Mobile Crane	New / Replace	LOM	1,390	10	0	0	0	0	0	0	1,390	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Liebherr LTM 1095 - Mobile Crane	Rebuild	35,000	278	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	278	0	0	0
10-tonne Forklift	New / Replace	LOM	73	10	0	0	0	0	0	0	73	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
10-tonne Forklift	Rebuild	35,000	13	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Welding Machine	New / Replace	20,000	11	10	0	0	0	0	0	0	11	0	0	0	0	11	0	0	0	11	0	0	0	11	11	0	0	0	11	0
Welding Machine	Rebuild	20,000	1	5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Screening Plant	New / Replace	10,000	27	20	0	0	0	0	0	0	27	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Screening Plant	Rebuild	10,000	3	n/a	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Light Plant	New / Replace	40,000	10	7	0	0	0	0	0	0	103	10	10	0	0	10	0	0	103	21	0	0	0	10	0	0	103	10	21	0
Light Plant	Rebuild	20,000	1	5	0	0	0	0	0	0	0	0	0	0	12	2	0	0	1	0	0	0	12	1	1	0	0	1	0	0
Spare Parts Inventory (@ 5%)	Other	n/a	n/a		28	0	0	0	0	0	159	1	22	0	0	1	0	9	69	2	21	0	6	1	9	0	5	29	65	21
Infrastructure & Misc.																														
Dewatering System	New / Replace	LOM	n/a	25	0	0	0	0	0	0	2,407	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Haul Road Construction	New / Replace	LOM	n/a		1,764	718	881	1,301	1,133	1,468	1,180	1,516	1,559	1,312	1,573	1,604	1,493	1,552	1,444	1,691	2,095	2,010	1,700	1,524	1,451	1,330	1,602	2,067	1,861	1,963

Figure 21-1: Total Estimated Capital Expenditures



Source; WSP Golder, 2023

The costs associated with initial haul road construction account for $4.7 M of the total estimated initial capital in Years 0-1. Based on haul road design criteria, as shown in Section 16.7.1, the roads were designed for use of 97 t overburden and 36 t matrix trucks. As seen in Table 21-8, the QP calculated the unit cost of road construction to be $209 and $66 per meter of road built for overburden and matrix trucks, respectively. These costs incurred for the aggregates supply of about $20.23/m^3, the use of 2.1 m^3 bucket backhoe and 36 t truck fleet, a mobile rock screening plant, grader and compactor. Labor cost to operate the required equipment was also included in the estimate.

Table 21-8: Unit Cost of Road Construction

Road Purpose		Overburden Haulage	Matrix Haulage
Truck Payload (tonnes)		97	36
Driving Surface (Road) Width (m)		24.5	14
Subgrade / Sub-base / Base Course	Thickness (m)	1.2	0.45
	Estimated Unit Cost per meter of road built	$109.56	$23.62
Surface Course	Thickness (m)	0.2	0.15
	Estimated Unit Cost per meter of road built	$99.12	$42.48
Total	Thickness (m)	1.4	0.6
	Estimated Unit Cost per meter of road built	$208.68	$66.11

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.

Due to the remoteness of the mine site, it is assumed that the project will have to be self-supporting with regard to necessary facilities and services. Most infrastructure capital expenditures, such as the processing plant, offices, and maintenance facilities, were the responsibility of Itafos or other QPs.

21.1.2 Process and Infrastructure Capital Costs

A summary of the process plant and site infrastructure capital cost estimate is presented in Table 21-9.

Table 21-9: Capital Cost Estimate Summary – Process Plant & Site Infrastructure

WBS	Description (Processing)	Cost (US$M)
001	Construction Contractor – Indirects	12.82
010	Site Construction Indirects General	6.35
020	Site Construction Facilities	0.23
030	Site Construction Facilities Other	0.12
040	Construction Operations	1.90
050	Construction Accommodation	3.72
060	Vendor Representatives	0.18
101	Treatment Plant – General	1.52
120	Feed Preparation	2.73
130	Scrubber Feed	0.77
140	Scrubbing/Desliming/Tailings	12.23
150	Fine Concentrate Thickening	1.02
190	Concentrate Filtration & Storage	17.79
210	Reagents	0.88
230	Water Services	3.33
240	Plant Services	0.55
250	Plant Air Services	0.38
270	Electrical Services	2.18
300	Infrastructure - General	0.23
Total		68.93

21.1.2.1 Estimating Sources

The process plant and associated infrastructure pre-production capital cost is US$68.93 million and includes provision for treatment plant (process plant), feed preparation, scrubber feed, scrubber desliming, concentrate thickening, concentrate filtration and storage, reagents, water services, plant services, plant air services, electrical services, infrastructure, power supply, tailings storage facility, plant site building and truck loadout.

All major processing equipment for all phases were sized based upon the process design criteria. Once the mechanical equipment list was outlined, the mechanical scopes of work were derived and sent to the market for budgetary pricing by international equipment suppliers (see Table 21-10 and Table 21-11). Once the price quotations were reviewed and integrated, in total 77% of the value of the mechanical equipment was sourced from budgetary quotations, with the remainder of the process equipment pricing escalated from 2019 pricing provided from budgetary quotes from previous study phase completed by Lycopodium. The escalation factor was selected based on escalated pricing observed when comparing 2019 and 2022 budgetary pricing. For Mechanical Equipment observed pricing increase was close to 48%.

Table 21-10: Mechanical Equipment Supply Price Basis

Source	Number of Packages	Supply Cost (US$)
Budget Quotes	10	19,313,000
Escalated from 2019 Pricing	21	5,754,000

*Note: costs exclude freight

Table 21-11: Mechanical Equipment Supply Price Basis

Package No.	Equipment	Pricing Source
5067	Pressure Filters	Budgetary Quote
5184	Belt Conveyors	Budgetary Quote
5043	Belt Feeders	Budgetary Quote
5006	High-Rate Thickeners	Budgetary Quote
5011	Cyclones	Budgetary Quote
5123	Drum Scrubber	Budgetary Quote
5193	Fuel Storage and Distribution	Budgetary Quote
5026	Flocculant Mixing Systems	Budgetary Quote
5187	Attrition Scrubbers	Budgetary Quote
5165	Hydroseparators	Budgetary Quote
5007	Slurry Pumps	Escalated from 2019 Pricing
5140	Off-Gas Scrubbing Systems	Escalated from 2019 Pricing
5085	Fire Water Pumps	Escalated from 2019 Pricing
5023	Water Treatment Systems	Escalated from 2019 Pricing
5014	Centrifugal Solution Pumps	Escalated from 2019 Pricing
5005	Vibrating Screens	Escalated from 2019 Pricing
5155	Truck Weighbridges	Escalated from 2019 Pricing
5033	Compressed Air Systems	Escalated from 2019 Pricing
5195	Truck Wash Systems	Escalated from 2019 Pricing
5025	Sewage Treatment Plants	Escalated from 2019 Pricing
5185	Analyzers	Escalated from 2019 Pricing
5008	Slurry Agitators	Escalated from 2019 Pricing
5038	Slurry Samplers	Escalated from 2019 Pricing
5189	Floating Pumps	Escalated from 2019 Pricing
5012	Vertical Cantilever Pumps	Escalated from 2019 Pricing
5186	Samplers	Escalated from 2019 Pricing
5029	Cranes and Hoists	Escalated from 2019 Pricing
5051	Fire Hydrants	Escalated from 2019 Pricing
5060	Safety Showers and Eyewash	Escalated from 2019 Pricing
5013	Positive Displacement Pumps	Escalated from 2019 Pricing
5015	Submersible Pumps	Escalated from 2019 Pricing
5199	Fire Hose Reels	Escalated from 2019 Pricing

Similar to the above, major electrical equipment was sized based on the mechanical equipment list. Once the electrical equipment was outlined, scopes of work were derived and sent to the market for budgetary pricing by international suppliers, as outlined in Table 21-12 and Table 21-13. Once the budgetary quotations were reviewed and integrated total of 51% of the value of the electrical and instrumentation equipment was sourced from budgetary quotations, with the reminder escalated from 2019 pricing provided from budgetary quotes from previous study phase completed by Lycopodium. The escalation factor was selected based on escalated pricing observed when comparing 2019 and 2022 budgetary pricing.

Table 21-12: Electrical Equipment Supply Price Basis

Source	Number of Packages	Supply Cost (US$)*
Budget Quotes	2	$4,210,000
Escalated from 2019 Pricing	6	$3,971,000

*Note: costs exclude freight.

Table 21-13: Electrical and Instrumentation Equipment Supply Price Basis

Package No.	Equipment	Pricing Source
6000	Low Voltage Motors	Escalated from 2019 Pricing
6500	Electrical Bulks	Escalated from 2019 Pricing
6600	Instrumentation Bulks	Escalated from 2019 Pricing
6700	Transmitters	Escalated from 2019 Pricing
6800	Process Control System	Escalated from 2019 Pricing
6850	Station Valve Panel	Escalated from 2019 Pricing
7002	Integrated E-Rooms	Budgetary Quote
7005	Transformers	Budgetary Quote

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering design were completed to a feasibility study level of definition, allowing for the bulk material quantities (earthworks, concrete, structural steel, electrical and instrumentation bulks) to be derived for the major commodities, as outlined in Table 21-14.

Table 21-14: Material Commodity Codes

Commodity Code	Commodity Description
A	Architectural
B	Earthworks
C	Concrete
D	Mining
E	Electrical Equipment and Bulks
F	Platework and Mechanical Bulks
G	Site Development
H	Spare
I	Instrumentation
J	Spare
K	Marine
L	Spare
M	Mechanical
N	Plant and Miscellaneous Equipment
O	Mobile Equipment
P	Pipework
Q	Electrical Bulks
R	Rail
S	Structural Steelwork
T	Spare
U	Field indirects
V	Third Party Packages / Other
W	EPCM, EPC & EP
Y	Owner's Costs
Z	Taxes

After the derivation of all the bulk material quantities, for the process plant and infrastructure areas, major construction contracts were formed and tendered to contractors for budgetary pricing bid for the process plant and infrastructure, as per Table 21-15.

Table 21-15: Contracts Price Basis

Package No.	Construction Package	Pricing Source
1100	Earthworks and Site Establishment	Budgetary Quote
1200	Concrete	Budgetary Quote
1500	SMP	Budgetary Quote
1700	Electrical and Instrumentation Install	Escalated from 2019 Pricing

21.1.3 Site Infrastructure

21.1.3.1 Basis of Estimate

21.1.3.1.1 TSF, RWP and WD

Capital and sustaining capital cost construction rates for the TSF, RWP and WD capital cost are based on a construction quote provided to Golder / WSP by PW Mining in 2020. The site infrastructure basis of estimate for the TSF, RWP and WDs are based on the following:

- The PW 2020 rates have been adjusted for 2022 with rates increasing between approximately 1% to over 50% depending on the activity.
- The 2022 construction rates adopted were not verified with local or international contractors.
- The 2022 rates do not include a cost escalation as insufficient information was made available to establish local construction trends.
- A 10% contingency has been applied to the TSF, WD, RWP and SWM infrastructure capital costs excluding preliminary and general costs.
- Preliminary and general costs which include contractor mobilization, demobilization, contractual requirements, and other fixed charged items is 25% of total construction costs.
- The bulk of the TSF embankment construction material will be sourced from pit waste overburden. The construction rate adopted assumes approved waste overburden meeting construction specifications will be delivered to the embankment footprint under construction with load, haul and delivery costs accounted for within the mining costs.

21.1.3.1.2 Surface Water Management & Pit Dewatering

Water infrastructure includes surface water management infrastructure as well as the South pit and North pit dewatering systems. The surface water management will be constructed in two phases to align with mining operations of the south and North pits. Surface water management infrastructure for Phase 1 includes the following systems:

- Sediment Control Dam 1 (SCD1) (Year 0)
    - SCD 1 and Spillway
    - Backwater Dam 1 and Spillway
    - Catchment Diversion Channel 03 (DC03)

  - Catchment Diversion Channel 04 (DC04)
- Environmental Control Dam (Year 0)
- Sediment Control Dam 2 (SCD2) (Year 1)
  - SCD2 and Spillway
  - Backwater Dam 2 and Spillway
  - Catchment Diversion Channel 03 (DC01)
  - Catchment Diversion Channel 04 (DC02)
- South Pit Flood Protection Bund (constructed in segments between Year 0 to Year 6)
- Process Plant Flood Protection Bund (Year 0)
- South Pit Dewatering System (Developed in phases according to mining schedule (Year 0 to 7)

Surface water management infrastructure for Phase 2 which is planned for Year 7 onwards includes the following systems:

- North Pit Diversion: Catchment Diversion Channel 05 (DC05)
- SCD 1 Catchment Diversion Channel 06 (DC06)
- North Pit Flood Protection Bund
- North Pit Diversion Dam
- North Pit River Diversion
- North Pit Dewatering System (Developed in phases according to mining schedule (Year 7 to 25)

The site infrastructure basis of estimate for the water infrastructure are based on the following:

- The PW 2020 rates have been adjusted for 2022 with rates increasing between approximately 1% to over 50% depending on the activity.
- The 2022 construction rates adopted were not verified with local or international contractors.
- The 2022 rates do not include a cost escalation as insufficient information was made available to establish local construction trends.
- A 10% contingency has been applied to the overall capital costs excluding preliminary and general costs for surface water management.
- A 15% contingency has been applied to the overall capital costs excluding preliminary and general costs for pit dewatering.
- Preliminary and general costs which include contractor mobilization, demobilization, contractual requirements and other fixed charged items is 25% of total construction costs.

21.1.3.2 Site Infrastructure Capital Costs

21.1.3.2.1 TSF, RWP and WD

Two surface waste dump areas have been identified as part of the mine layout. Waste dump 1 (WD-1) located adjacent to the South pit will be developed at the beginning of Year 0 to permanently store NAG and PAG waste overburden generated in Year 0 and Year 1. Waste dump 2 (WD-2) will be developed at the beginning of Year 1 to permanently store NAG waste overburden.

The estimated costs to develop the WD-1 foundation in preparation for receiving waste overburden including applicable P&Gs is shown in Table 21-16.

Table 21-16: Waste Dump Capital Cost Estimate

Area	Construction Costs (US$M)	P&G Costs (US$M)	Contingency (10%) (US$M)	Total Costs (US$M)	Construction Year
Waste Dump 1	1.329	0.332	0.133	1.794	0
Total	1.329	0.332	0.133	1.794	

The return water pond (RWP) will be constructed in two phases with the first phase constructed in Year 0 and the second phase in Year 1. The estimated capital costs for the RWP Phase 1 are summarized in Table 21-17.

Table 21-17: Return Water Pond Capital Cost Estimate

Area	Construction Costs (US$M)	P&G Costs (US$M)	Contingency (10%) (US$M)	Total Costs (US$M)	Construction Year
RWP Phase 1	1.119	0.280	0.112	1.511	0
Total	1.119	0.280	0.112	1.511	

Following topsoil removal, earthworks will proceed for the individual TSF embankment construction with materials supplied from the mining fleet delivering waste overburden directly onto the cell embankment footprint. The estimate assumes the waste overburden will be constructed in maximum 300 mm thick layers with the rate including, spreading, conditioning, compaction and testing. A 500 mm thick blanket drain included in the embankment design is comprised of sand and gravel appropriately sized to act as a filter and prevent piping or loss of fines. A local source on site has not been identified, therefore the rate selected assumes supply, haul and place in one lift will be sourced from an offsite source. This project update did not identify a specific offsite source and is an overall risk to the project.

The individual TSF cells will be constructed over the life of mine to spread construction costs. Cell 1 will be constructed in Year 0 while the final cell, cell 7 will be constructed in Year 22 based on the current 2022 mining plan. Each TSF cell embankment will include a 1.5 mm HDPE geomembrane liner to minimize seepage through the embankment but also to provide an erosion protection barrier in lieu of an expensive rip rap rock option. A perimeter road along the downstream toe of the TSF embankments will be developed in Year 0 and Year 1. The perimeter road has been designed to accommodate delivery of mine waste overburden from the mining fleet, however, current costs assume a rock subbase is not required.

The estimated costs for the TSF cell 1 construction in Year 0 are summarized in Table 21-18.

Table 21-18: TSF Cell 1 Capital Cost Estimate

Area	Construction Costs (US$M)	P&G Costs (US$M)	Contingency (10%) (US$M)	Total Costs (US$M)	Construction Year
Cell 1	3.662	0.916	0.366	4.944	0
Total	3.662	0.916	0.366	4.944	

21.1.3.2.2 Surface Water Management & Pit Dewatering

The estimated costs to develop the South pit initial phase of dewatering including the 15% contingency is shown in Table 21-19.

Table 21-19: South Pit Dewatering Capital Cost Estimate

Area	Construction Costs (US$M)	Contingency (15%) (US$M)	Total Costs (US$M)
Well Development	2.057	0.308	2.365
Mechanical Systems	1.992	0.299	2.291
Indirects	0.286	0	0.286
Operating	0.302		0.302
Total	4.638	0.607	5.245

Installation and commissioning of the borehole pumps must be completed as early as possible in the Year 0 or possibly earlier to enable dewatering to commence ahead of South pit pre-strip. Operating costs have been included in the capital costs for Year 0 as dewatering will need to commence ahead of the mine pre-strip.

A summary of the surface water management infrastructure capital costs is given in Table 21-20.

Table 21-20: Surface Water Management Infrastructure Capital Cost Estimate

Area	Construction Costs	P&G Costs	Contingency (10%)	Total Costs	Construction Year
Sediment Control Dam 1	1.558	0.390	0.156	2.103	0
Environmental Control Dam	1.290	0.323	0.129	1.742	0
Flood Protection Bunds	0.797	0.199	0.080	1.077	0
Total	3.646	0.911	0.365	4.922	

21.1.3.3 Site Infrastructure Sustaining Capital Costs

21.1.3.3.1 TSF, RWP and WD

The estimated costs to develop the WD-2 foundation in preparation for receiving NAG waste overburden including applicable P&Gs is shown in Table 21-21.

Table 21-21: Waste Dump Sustaining Capital Cost Estimate

Area	Construction Costs (US$M)	P&G Costs (US$M)	Contingency (10%) (US$M)	Total Costs (US$M)	Construction Year
Waste Dump 2	0.623	0.156	0.062	0.841	1
Total	0.623	0.156	0.062	0.841	

The estimated capital costs for the RWP Phase 2 construction are summarized in Table 21-22.

Table 21-22: Return Water Pond Sustaining Capital Cost Estimate

Area	Construction Costs (US$M)	P&G Costs (US$M)	Contingency (10%) (US$M)	Total Costs (US$M)	Construction Year
RWP Phase 2	0.819	0.205	0.082	1.106	1
Total	0.819	0.205	0.082	1.106	

The estimated costs for the TSF construction over the LOM between years 1 and 22 are summarized in Table 21-23.

Table 21-23: TSF Cell 2 to 7 Sustaining Capital Cost Estimate

Area	Construction Costs (US$M)	P&G Costs (US$M)	Contingency (10%) (US$M)	Total Costs (US$M)	Construction Year
Cell 2	6.552	1.638	0.655	8.845	2
Cell 3, 4, 5, 6 & 7	24.720	6.180	2.472	33.372	7 - 22
Total	31.272	7.818	3.127	42.217	

21.1.3.3.2 Surface Water Management & Pit Dewatering

The estimated costs to develop the South pit dewatering for Years 1 through 7 including the 15% contingency is shown in Table 21-24.

Table 21-24: South Pit Dewatering Sustaining Capital Cost Estimate Years 1 to 7

Area	Construction Costs (US$M)	Contingency (15%) (US$M)	Total Costs (US$M)
Well Development	4.368	0.655	5.023
Mechanical Systems	2.163	0.324	2.487
Indirects	0.737	0	0.737
Total	7.268	0.979	8.247

The estimated costs to develop the North pit dewatering including the 15% contingency is shown in Table 21-25.

Table 21-25: North Pit Dewatering Sustaining Capital Cost Estimate

Area	Construction Costs (US$M)	Contingency (15%) (US$M)	Total Costs (US$M)
Well Development	6.426	0.964	7.390
Mechanical Systems	4.155	0.623	4.778
Indirects	1.023		1.023
Total	11.604	1.587	13.191

Installation and commissioning of the borehole pumps should be completed as early as possible in Year 7 to enable dewatering to commence.

A summary of the surface water management infrastructure sustaining costs is given in Table 21-26.

Table 21-26: Surface Water Management Infrastructure Sustaining Capital Cost Estimate

Area	Construction Costs	P&G Costs	Contingency (10%)	Total Costs	Construction Year
Sediment Control Dam 2	1.070	0.268	0.107	1,445	1
Flood Protection Bunds	3.659	0.915	0.366	4.940	1 - 8
North Pit Diversion	0.683	0.171	0.068	0.923	8
Diversion Dam	0.144	0.036	0.014	0.194	20
River Diversion	2.618	0.654	0.262	3.534	20
Total	8.174	2.044	0.817	11.036	

21.1.4 Marine Terminal Capital Costs

The marine terminal capital cost estimate is summarized in Table 21-27 and grouped into the following main categories:

- Employer procured items (EPI) capital costs – major equipment procurements would be procured directly by ITAFOS.
- Marine works contractor (MWC) procurement capital costs – major infrastructure procurements which will be the responsibility of and coordinated by the MWC and that are associated with a long lead time.
- MWC Construction capital costs – mobilization/demobilization, the supply of labor and equipment for the installation of the marine works, and the procurement of materials with shorter lead times.
- Cacheu River Crossing capital costs – mobilization/demobilization from Ponte Chugue, the supply of labor and equipment for the installation of the conveyor crossing foundations, and the procurement of materials.
- EPCM Fee costs – Engineering, procurement, and construction management fee.

Table 21-27: Marine Terminal Capital Costs Summary

Description	Cost ($M)
Total EPI	17.1
Total MWC Procurement	19.8
Total MWC Construction	31.6
Total Cacheu River Crossing	1.2
Total EPCM Fee	7.1
Total Marine Terminal Capital Cost	76.7

21.1.4.1 Employer Procured Items

The capital costs for the EPI were determined primarily by budgetary quotes from shiploader suppliers and new build vessel prices. The cost breakdown is shown in Table 21-28.

Table 21-28: Employer Procured Items Capital Costs

Description	Cost ($M)
Shiploader	3.4
Tugboats (x2)	9.7
Pilot Boat	0.9
Maintenance Barge	3.0
eNavigation System	0.1
Total EPI Capital Cost	17.1

21.1.4.2 MWC Procurement

The capital costs for the MWC Procurement were determined from budgetary quotes provided by two regional fabricators and based off drawings and detailed material take-offs provided by Baird. The cost breakdown is shown in Table 21-29.

Table 21-29: MWC Procurement Capital Costs

Description	Cost ($M)
Piles	3.8
Jackets	4.5
Topsides	1.9
Trestle Towers	0.3
Floating Tug Berth	1.9
Walkways	1.6
Quick Release Hooks	0.2
Pile and Fabricated Steel Freight	4.1
MWC Markup on Procurement	1.4
Total MWC Procurement Capital Cost	19.8

21.1.4.3 MWC Construction

The capital costs for the MWC Construction were determined from a budgetary quote provided by a marine contractor familiar with the construction of similar marine structures in the Port of Bissau and based off drawings and detailed material take-offs provided by Baird. The cost breakdown is shown in Table 21-30.

Table 21-30: MWC Construction Capital Costs

Description	Cost ($M)
Indirect Costs	5.0
General Conditions	3.6
Mobilization / Demobilization	7.5
Shiploader Foundations – Pivot and Radial	1.3
Fuel Platform	0.2
(5) Berthing Dolphins	2.3
(4) Mooring Dolphins	1.8
(9) Trestle Foundations	1.4
Walkways	0.2
Maintenance Barge Berth	1.4
Floating Tug Berth	0.4
Aids to Navigation	1.5
Material Conveyance System	2.3
Fuel System	1.4
Electrical System	0.8
Fire Suppression and Detection System	0.4
Potable Water Supply System	0.1
Compressed Air System	0.1
Total MWC Construction Capital Cost	31.6

21.1.4.4 Cacheu River Crossing

The capital costs for the Cacheu River Crossing were escalated from the 2019 tender price provided by a marine contractor and based off conceptual drawings and detailed material take-offs provided by Baird. The cost breakdown is shown in Table 21-31.

Table 21-31: Cacheu River Crossing Capital Costs

Description	Cost ($M)
Indirect Costs & General Conditions	0.5
Piles & Jackets Procurement	0.4
Topsides & Towers Procurement	0.1
Installation	0.2
Total Cacheu River Crossing Capital Cost	1.2

21.2 Operating Costs

Operating costs include the ongoing cost of operations related to mining, processing, tailings disposal, shiploading, and general administration activities. The operating cost estimate is presented separately for the south and North pit operations due to the varying quantities of concentrate produced in each phase.

The operating cost estimate was developed by Ausenco, WSP, and Baird for processing, mining, and shiploading costs respectively. Labor rates and general administration costs were provided by Itafos. A summary of the operating costs is presented in Table 21-32.

Table 21-32: Operating Cost Summary (with fuel itemized separately)

Description	Life-of-Mine Operating Cost			South Pit			North Pit		
	US$M	US$/t Feed	US$/t Conc.	US$M/ year	US$/t Feed	US$/t Conc.	US$M/ year	US$/t Feed	US$/t Conc.
Mining	661.4	15.1	20.1	31.3	17.9	23.1	24.6	14.0	18.9
Process	343.0	7.8	10.4	13.9	7.9	10.3	13.6	7.8	10.5
Shiploading	111.3	2.5	3.4	4.5	2.5	3.3	4.5	2.5	3.4
Tailings, Environment, Water	15.7	0.4	0.5	0.6	0.4	0.5	0.6	0.4	0.5
G&A	186.8	4.3	5.7	7.5	4.3	5.5	7.5	4.3	5.7
Fuel	952.3	21.8	28.9	35.4	20.2	26.1	39.1	22.4	30.1
Total	2270.5	51.9	69.0	93.2	53.2	68.7	89.9	51.4	69.1

Note: Fuel is itemized separately and is not included in mining, processing, shiploading or G&A costs.

21.2.1 Overall Basis of Estimate

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q3 2022 pricing without allowances for inflation.
- Costs are expressed in United States dollars (USD or US$).
- For material sourced in Canadian dollars, an exchange rate of 1.29 dollar per US dollar was assumed.
- For material sourced in euros, an exchange rate of 0.98 euros per US dollar was assumed.
- For material sourced in West African CFA Francs, an exchange rate of 640 francs per US dollar was assumed.
- Labor is assumed to come from neighboring municipalities and from expatriate employees.
- Processing unit operations were benchmarked against similar or comparable processing plants.
- Equipment, consumables, and maintenance materials will be purchased as new.
- Reagent consumption rates have been estimated based on tested metallurgical characteristics.
- Mobile equipment costs provision for fuel and maintenance.
- Dryer and power plant fuel consumption rates are derived from vendor quotations.
- Diesel price of US$0.86/L.

21.2.2 Mine Operating Costs

As stated previously, direct mine operating costs include the required labor, supply, and materials costs based on the mine plan schedule. Labor costs include wages for hourly production, maintenance and support employees, and salaries for mine administration and supervisory staff. Labor 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. The estimates of the quantity of labor and materials necessary to fulfill the requirements of the mine plan became the basis of all cost estimates.

21.2.2.1 Contractor Mining Operating Costs

PW provided variable unit rates for overburden removal based upon varying haul distances between the mining area and dumps. These unit rates range from $1.95 per bcm up to $3.46 per bcm. Matrix mining and handling averaged $0.57 per ROM tonne. Total cost of delivered matrix for the contractor option averaged $25.93/ROM tonne (dry basis) for Years 1 through 5. A summary of the contractor costs is provided in Table 21-33. Contract operations will be for production of 1.75 Mt/a (dry basis) of feed to the processing facility as well as appropriate handling of all overburden and other mining ancillary functions required to meet the design ore production.

Table 21-33: Summary of Contractor Cost Model

Total 2022 Costs	Units	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mobilization	$k	3,762	0	0	0	0	0	3,762
Demobilization	$k	0	0	0	0	0	1,978	1,978
Site Establishment	$k	2,965	0	0	0	0	0	2,965
Road Construction Cost	$k	1,764	718	881	1,301	1,133	1,468	7,265
Subtotal - Contractor Capital Expenditures	$k	8,491	718	881	1,301	1,133	3,446	15,970
Management Fee	$k	9,052	8,995	8,976	8,889	8,919	8,868	53,699
Mining Works Load Haul	$k	12,479	30,002	37,051	34,482	35,462	33,234	182,710
Mining Works Ore Handling	$k	0	1007	987	989	984	992	4,959
Subtotal - Contractor Operating Expenditures	$k	21,531	40,004	47,014	44,359	45,365	43,094	241,368
Contractor Total	$k	30,023	40,723	47,894	45,660	46,498	46,540	257,338

21.2.2.2 Owner-Operated Mining Operating Costs

The QP estimated costs for the remainder of the 25-year mine plan to produce 1.75 Mt/a (dry basis) of feed for the process facility. The estimates encompassed all costs associated with all mining, matrix and overburden handling, ROM stockpile ore handling, and other mine support services, up to the processing plant for Years 6 through 25 based on an owner-operated model starting in Year 6. The estimate does not include product transport costs to the Mineral Terminal. 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 were not included in this cost model. The QP included ongoing reclamation costs during the mine life (including dozer work for backfill pit re-grading and re-vegetation during mining; however, final mine closure and infrastructure demolition were not included in the mining cost model and were covered by others.

The cost estimates for this mine plan were developed using the 2015 Feasibility Study zero-based cost estimate. To account for changes in diesel, materials, supply, labor, equipment, and other miscellaneous costs, an escalation factor of 1.2037 (20.37%) developed from the most recent gross domestic product implicit price deflator (GDP-IPD) published by the US Energy Information Administration (EIA) was applied to escalate the costs to approximate 2022 US dollars without obtaining updated budgetary quotes.

Using the total zero-based direct operating cost estimates from the 2015 Feasibility Study, Golder developed unit costs by mining function and escalated these developed unit costs to 2022 US dollars to estimate direct operating costs. The QP 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
- mine supervision and administration.

The waste stripping and haulage functional cost category and FPA mining and haulage functional cost category were each delineated into three different subcategories describing the activities that aggregate these functions, including loading, haulage, and support. Haulage costs were further delineated to account for the varying haul distances and average haul cycle times from each point of origin (South pit and North pit) to each destination.

The overburden stripping cost center represents overburden excavation and removal of overburden by the excavator/loader fleets and truck haulage of excavated overburden material to designated waste storage 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 offsite location).

For reporting purposes, pit dewatering was treated as single cost center, separated from the operation support function. 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 management 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 overburden stockpiles, in-pit backfill grading, revegetation and re-vegetation monitoring. Mine maintenance functions include in-pit equipment fueling 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.

Costs for pit dewatering and FPA stockpiling were developed on a cost per ROM FPA tonne basis, whereas costs for reclamation, mine maintenance, and operations support were developed based on a cost per bank cubic material of material moved (waste and FPA).

21.2.2.2.1 Mine Operating Costs – Labor

The QP estimated operating labor requirements using a zero-based approach with annual staffing levels determined by the level of equipment or facility usage dictated by the mine plan. The QP allotted maintenance labor, support labor, 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 labor and mine supervisory and administrative personnel were assigned as deemed necessary to adequately support production.

For this report, the QP 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 during the dayshift only, one 12-hour shift per day, seven days per week, due to higher mining dilution consideration during a nightshift. 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.

Labor cost comprises wages for hourly employees; salaries for supervisory and administrative personnel were provided by Itafos in 2015 and escalated to 2022 US dollars. 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, worker'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 develop the workforce sourced locally.

The QP used the information to estimate operating labor costs for six pay-grade categories using hourly operating labor rates. Table 21-34 shows the hourly wage rates for the six paygrades and the respective job descriptions. Hourly rates are the base rates and annual rates include the additional burden items previously noted. Total hourly costs formed the yearly equivalent of the base rate charges.

Table 21-34: Summary of Hourly Wage Rates

Pay Grade	Job Classification	Wage Rate	
1	Senior Operator	$2.27 per hour	$9,867 annually
2	Skilled Trades	$2.27 per hour	$9,867 annually
3	Operators	$1.87 per hour	$8,256 annually
4	Junior Operator	$1.87 per hour	$8,256 annually
5	Drivers	$2.07 per hour	$8,436 annually
6	Laborers (Semi-skilled)	$2.07 per hour	$8,436 annually

Itafos supplied the QP with in-country labor rates based on salary surveys. Expatriate salaries were estimated using base salaries deemed competitive within the region. Table 21-35 lists a summary of base salaries for mine administration and supervisory staff.

Table 21-35: Summary of Salaried Labor Positions

Position	Base Salary (US$ per year)	Other Benefits (US$ per year)	Total Compensation (US$ per year)
Mining Manager (Expat)	173,324	89,837	263,161
Maintenance Manager (Expat)	173,324	89,837	263,161
Mining Engineer/Geologist (Expat)	151,979	81,943	233,922
Mining Superintendent (Expat)	130,633	74,048	204,681
Maintenance Superintendent (Expat)	130,633	74,048	204,681
Department Head	35,088	23,100	58,188
Deputy	18,246	12,152	30,398
Shift Supervisor	13,334	8,959	22,293
Officers	11,930	8,047	19,977
Senior Qualified Staff	9,825	6,678	16,503
Junior Qualified Staff	8,421	6,205	14,626
Graduate Entry Level Qualified Staff	7,836	5,825	13,661
Technician/Secretary	7,252	5,445	12,697

21.2.2.2.2 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 in dollars per liter ($/L) and lubricants (in $/L or dollars per kilogram ($/kg)) were based on vendor budgetary pricing data. Table 21-36 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 21-36: Summary of Unit Consumable Costs

Lubricant	Grade	Equipment Usage	Unit Cost
Engine Oil	SAE 5W-40	Excavators / Loaders	$7.66
	SAE 10W-30	Mobile Equipment	$7.66
	SAE 15W-40	Haul Trucks	$7.66
Transmission Fluid	SAE 0W-30	Excavators / Loaders	$6.33
	SAE 30	Mobile Equipment	$6.33
	SAE 5W-30	Haul Trucks	$6.33
Final Drive & Differential Fluids	SAE 50	Excavators	$6.56
	SAE 60	Haul trucks	$6.56
	SAE 75W-90	Small Excavators	$6.80
	SAE 80W-90	Mobile equipment	$6.80
Hydraulic Fluid	SAE 0W-20	Excavators	$6.17
	SAE 5W-30	Mobile Equipment	$6.17
	SAE 10W	Haul Trucks	$6.17
Grease	Multi-purpose	Mobile Equipment	$9.61
	3% Moly	Hydraulic / Excavators	$14.22
	5% Moly	Shovels / Loaders	$14.22
	Synthetic	Excavators	$14.22
Fuel Diesel	n/a	All Equipment	$0.86/L

21.2.2.2.3 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. 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 the QP's experience. 0 lists the unit costs applied in the cost model for consumable items.

The QP 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 equipment production rates. Support equipment was assigned as deemed necessary to facilitate an effective mining operation.

Table 21-37 also lists the base price of each equipment obtained from major equipment suppliers or manufacturers. The QP secured the quotes for support equipment, such as welding machines, 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, commissions, and other applicable fees.

Table 21-37: Summary of Equipment Base Price and Hourly Operating Costs

Equipment Type	Manufacturer & Model	Size Class		Total Capital Cost	Fuel	Lube	Tire	Filter	U.C.	R&R	Total
				($)	($)	($)	($)	($)	($)	($)	($)
Backhoe	Caterpillar 374F - Backhoe	5	m^3 bucket	993,697	35.69	7.13	n/a	1.38	16.56	56.31	**117.07**
Backhoe	Caterpillar 336D - Backhoe	2.1	m^3 bucket	437,297	33.54	4.30	n/a	1.32	4.46	18.04	**61.66**
Wheel Loader	Caterpillar 992K - Wheel Loader	12.2	m^3 bucket	2,383,030	84.62	13.12	17.60	2.17	n/a	77.45	**194.96**
Dozer	Caterpillar D9R - Dozer	405	hp	907,755	47.21	4.54	n/a	1.20	18.91	28.37	**100.23**
End Dump Truck	Caterpillar 777D - End Dump Truck	97	tonnes	1,260,680	64.50	10.82	30.14	1.08	n/a	37.82	**144.36**
End Dump Truck	Caterpillar 770 - End Dump Truck	36	tonnes	636,656	31.48	10.82	8.61	0.54	n/a	14.86	**66.31**
Motor Grader	Caterpillar 16M - Motor Grader	297	hp	899,883	28.29	3.12	3.51	0.60	n/a	25.71	**61.23**
Compactor	Caterpillar CS-56 - Compactor	147	hp	173,282	14.62	6.11	0.00	0.72	n/a	4.33	**25.78**
Scraper	Caterpillar 637G - Scraper	26	m^3 bucket	1,474,293	81.01	6.11	0.00	1.62	n/a	33.96	**122.70**
Water Truck	Caterpillar 770 - Water Truck	34,000	liter	790,339	35.09	10.82	8.61	0.48	n/a	22.56	**77.56**
Fuel/Lube Truck	Fuel/Lube Truck	400	hp	424,901	23.13	3.94	0.00	1.32	n/a	31.87	**60.26**
Mechanic's Truck	Mechanic's Truck	150	hp	86,665	7.83	1.33	0.00	0.54	n/a	3.76	**13.46**
Pickup Truck	Pickup Truck	128	hp	51,759	4.90	0.64	0.00	0.60	n/a	0.82	**6.96**
Crew Bus	Crew Bus	94	hp	71,018	4.90	3.94	0.00	0.66	n/a	0.94	**10.44**
95-tonne Crane	Liebherr LTM 1095 - Mobile Crane	95	tonne	1,390,258	59.86	6.55	0.00	1.38	n/a	24.83	**92.62**
10-tonne Forklift	10-tonne Forklift	10	tonne	73,361	16.94	4.07	0.00	0.66	n/a	1.31	**22.98**
Welding Machine	Welding Machine	24	hp	9,639	1.29	0.11	n/a	n/a	n/a	0.43	**1.83**
Light Plant	Light Plant	2,300	m2	7,782	2.32	0.19	n/a	0.12	n/a	0.31	**2.94**
Screening Plant	Screening Plant	266	hp	22,864	22.49	0.19	n/a	0.00	n/a	1.83	**24.51**

21.2.2.2.4 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 preproduction 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 Itafos are listed in Table 21-38. As the mining consultant for the project, the QP calculated and reported all cash costs directly associated with mining, including preproduction Year 0. As part of the integration into the total project cost estimate, Ausenco, as the lead consultant and under the direction of Itafos, elected to categorize preproduction costs as capital expenses.

The operating costs comprise of material and supply estimates and labor cost. The total direct mine operating cost, including preproduction and fuel is $1,048 million, or $31.86/t product. Annual costs range from $21.5 million in preproduction Year 0 to $48.9 million in Year 2. When the estimated preproduction costs of $21.5 million and mining fuel cost of $365.3 million are excluded from the direct operating costs, the total becomes $661.4 million, or $20.1/t product.

The primary direct mine operating cost drivers in Table 21-38 are overburden stripping and FPA (matrix) mining. These costs account for 77% 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 $256,000 in Year 8 to $738,000 in Year 16.

Table 21-39 details annual labor requirements and indicates that productivity is approximately 3,861 product tonnes per employee over the life of mine.

As previously noted, expenses related to haul road construction are not included in direct operating expenses. These were categorized as capital costs and are explained in Section 21.1.1.2. However, the costs for road maintenance are included in the operations support component and include the costs to operate a grader, utility backhoe and water truck.

Table 21-38: Summary of Direct Mine Operating Costs

Production Statistics	Units	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	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
ROM Waste Stripping	kbcm	5,812	8,661	10,081	14,892	12,969	16,800	13,510	17,348	17,847	15,017	18,005	18,361	17,083	17,768	16,525	19,356	23,981	23,006	19,457	17,438	16,609	15,228	18,335	23,654	21,300	22,470
ROM (Plant Feed) Tonnes, Dry Basis	kt	0	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	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,766
Total Prime Material (Waste + Ore) Moved	kbcm	5,812	10,050	11,456	16,275	14,350	18,181	14,893	24,448	12,942	16,267	19,255	19,611	18,333	19,018	17,775	20,606	25,231	24,256	20,707	18,688	17,859	16,478	19,585	24,904	22,550	23,641
Total Product Tonnes, Dry Basis	kt	0	1,356	1,356	1,356	1,356	1,356	1,356	1,345	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,312
Total Tailings Tonnes, Dry Basis	kt	0	280	280	280	280	280	280	299	380	380	380	380	380	380	380	380	380	380	380	380	380	380	380	380	380	383
Prime ROM Strip Ratio, Dry Basis	bcm / ROM tonne	n/a	4.95	5.76	8.51	7.41	9.60	7.72	9.91	10.20	8.58	10.29	10.49	9.76	10.15	9.44	11.06	13.70	13.15	11.12	9.96	9.49	8.70	10.48	13.52	12.17	12.72
Effective Product Strip Ratio, Dry Basis	bcm / product tonne	n/a	6.39	7.43	10.98	9.56	12.39	9.96	12.89	13.73	11.55	13.85	14.12	13.14	13.66	12.71	14.89	18.44	17.69	14.96	13.41	12.77	11.71	14.10	18.19	16.38	17.13
Effective Waste Haulage Volumes	kbcm	5,812	8,661	10,081	14,892	12,969	16,800	13,510	17,348	17,847	15,017	18,005	18,361	17,083	17,768	16,524	19,356	23,981	23,006	19,457	17,438	16,609	15,228	18,335	23,654	21,300	22,470
In-Pit Overburden Backfilling (IOB)	kbcm	0	3,021	7,246	12,099	9,904	15,034	11,979	14,522	12,138	12,034	12,433	11,017	9,426	7,085	11,296	10,672	21,291	13,738	18,009	16,892	15,070	15,148	16,705	13,683	18,913	22,470
Surcharge Overburden Stockpiling (SOS)	kbcm	0	0	0	525	2,144	1,691	1,317	0	5,709	2,982	5,572	7,344	7,658	10,188	5,228	7,050	2,690	9,268	0	546	1,538	80	939	9,971	1,606	0
Construction Material Haulage	kbcm	742	175	1,919	224	100	75	214	2,826	0	0	0	0	0	494	0	1,634	0	0	1,448	0	0	0	691	0	0	0
Ex-Pit	kbcm	5,070	5,466	916	2,045	822	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	781	0
Direct Operating Costs																											
Contractor Mining Costs																											
Contractor Mining Cost (FPA + OB)	US$k	12,479	30,002	37,051	34,482	35,462	33,234																				
Operations Support	US$k	9,052	8,995	8,976	8,889	8,919	8,868																				
FPA Rehandle	US$k	0	1,007	987	989	984	992																				
Subtotal - Contractor Mining Costs	US$k	21,531	40,004	47,014	44,359	45,365	43,094	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner Mining Costs	US$k																										
Waste Stripping	US$k							19,449	29,444	28,413	24,116	29,223	31,866	30,350	33,792	24,506	30,504	34,964	34,412	29,546	25,241	24,219	22,059	27,434	35,438	31,460	21,657
FPA Mining	US$k							2,410	2,119	2,238	2,602	2,454	2,576	2,641	2,703	2,601	2,850	2,809	2,635	2,701	2,850	2,935	3,059	3,172	3,438	3,485	3,583
Pit Dewatering	US$k							413	456	256	362	440	507	546	586	698	692	738	619	590	676	681	689	627	440	523	632
Ongoing Reclamation	US$k							1,033	1,146	1,196	905	1,005	1,219	1,116	1,226	1,103	999	1,212	1,348	941	1,287	803	756	1,112	1,329	1,178	1,791
Mine Maintenance	US$k							2,020	2,426	2,494	2,200	2,512	2,782	2,411	2,512	2,353	2,890	3,383	3,263	2,895	2,422	2,356	2,211	2,531	3,313	3,073	3,166
Operations Support	US$k							1,388	1,403	1,377	1,381	1,407	1,390	1,404	1,428	1,408	1,395	1,478	1,393	1,404	1,390	1,374	1,378	1,371	1,351	1,358	1,336
FPA Stockpiling	US$k							992	992	992	992	992	992	992	992	992	991	991	992	991	992	991	992	992	992	992	1,001
Mine Supervision & Administration	US$k	n/a	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927	1,927
Subtotal - Owner Mining Costs	US$k	0	1,927	1,927	1,927	1,927	1,927	29,632	39,913	38,894	34,485	39,959	43,260	41,388	45,166	35,589	42,249	47,503	46,588	40,996	36,784	35,286	33,071	39,165	48,228	43,996	35,092
Total Direct Mine Operating Costs	US$k	21,531	41,932	48,941	46,286	47,292	45,022	29,632	39,913	38,894	34,485	39,959	43,260	41,388	45,166	35,589	42,249	47,503	46,588	40,996	36,784	35,286	33,071	39,165	48,228	43,996	35,092

Table 21-39: Summary of Labor Requirement

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	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Production Statistics																										
Total Product Tonnage (kt, dry basis)	0	1,356	1,356	1,356	1,356	1,356	1,356	1,345	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,312
Total Stripping Volume (kbcm)	5,812	8,661	10,081	14,892	12,969	16,800	13,510	17,348	17,847	15,017	18,005	18,361	17,083	17,768	16,525	19,356	23,981	23,006	19,457	17,438	16,609	15,228	18,335	23,654	21,300	22,470
Stripping Ratio (bcm / ROM tonne)	n/a	4.95	5.76	8.51	7.41	9.60	7.72	9.91	10.20	8.58	10.29	10.49	9.76	10.15	9.44	11.06	13.70	13.15	11.12	9.96	9.49	8.70	10.48	13.52	12.17	12.72
Productivity (Product Tonne / Total Employees)	0	6,183	5,610	4,464	4,974	4,213	4,804	3,899	3,774	4,150	3,676	3,298	3,650	3,424	3,924	3,235	2,887	2,979	3,297	3,824	3,892	4,094	3,632	2,919	3,121	3,428
Operations Labor																										
Shovel / Backhoe / Excavator Operators	4	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
Loader Operators	10	16	18	25	23	28	23	29	30	25	30	30	28	29	28	32	38	37	32	29	28	26	30	38	35	36
Haul & Water Truck Operators	39	61	71	85	73	87	76	109	104	95	109	122	119	133	93	115	123	121	109	96	95	90	107	132	120	73
Compactor Operators	9	13	16	23	20	26	21	27	28	23	28	28	26	28	26	30	37	36	30	27	26	24	28	37	33	35
Dozer Operators	8	11	12	18	16	21	17	21	22	18	22	23	21	22	20	24	29	28	24	21	20	19	22	29	26	28
Grader & Utility Equipment Operators	6	8	8	10	10	11	10	11	11	10	11	11	11	11	11	12	13	13	12	11	11	10	11	13	13	13
Pumper / Laborer	0	0	2	5	6	7	6	5	3	4	6	7	8	9	12	11	13	10	9	12	12	13	11	5	7	10
Subtotal - Operations Labor	76	118	136	175	156	188	161	211	206	185	215	230	222	241	198	233	263	254	225	206	200	189	219	263	242	203
Maintenance Labor																										
Fuel Truck Driver / Serviceman	10	14	16	24	20	26	22	26	28	24	28	28	26	28	26	30	36	34	30	26	26	24	28	34	32	34
Electricians	5	5	5	5	5	5	5	5	5	5	5	10	5	5	5	10	10	10	10	5	5	5	5	10	10	10
Crane / Forklift Operators	4	8	9	12	10	13	11	13	13	12	13	13	13	13	12	14	17	16	14	13	12	12	13	16	15	16
Mechanics / Welders	8	8	8	8	8	8	8	8	8	8	8	16	8	8	8	16	16	16	16	8	8	8	8	16	16	16
Tire Servicemen, Maintenance Helpers, Trainees	26	36	38	50	43	52	45	52	55	50	55	67	52	55	52	69	78	76	69	52	52	50	55	76	72	74
Subtotal - Maintenance Labor	53	71	76	98	87	104	91	104	109	98	109	134	104	109	104	139	157	152	139	104	104	98	109	153	145	150
Total Hourly Labor	129	189	212	274	243	292	252	315	315	283	324	364	326	350	301	372	420	406	364	310	304	288	328	415	387	353
Supervision & Administration																										
Mine Supervision & Administration	30	30	30	30	30	30	30	30	30	30	30	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	30	30	30	30	30	30	30	30	30	30	30
Total Workforce	159	219	242	304	273	322	282	345	345	313	354	394	356	380	331	402	450	436	394	340	334	318	358	445	417	383

21.2.2.3 Indirect Mining Costs

Indirect mine operating costs are costs incurred by the mining operation that are 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; laboratory 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, the QP did not include overhead expenses in the cost model, as it was assumed these expenses would be provided by Itafos or Ausenco.

21.2.3 Process Operating Costs

The average annual process operating costs (including general and administrative or G&A costs) vary between the processing of the South pit and North pit ores. Table 21-40 provides a summary of the operating costs of the two ore types, which will be processed at different times in the mine life.

- South Pit: Years 1 – 7
- North Pit: Years 8 – 25

Table 21-40: Overall Operating Costs for Process Activities

Description	South Pit			North Pit		
	US$M	US$/t Feed	US$/t Conc	US$M	US$/t Feed	US$/t Conc
Plant Maintenance	1.07	0.61	0.79	1.07	0.61	0.82
Labor (O&M)	4.43	2.53	3.27	4.43	2.53	3.41
Consumables and Reagents (excl. Dryer Fuel)	1.30	0.74	0.96	1.24	0.71	0.95
Dryer fuel	15.73	8.99	11.60	15.08	8.62	11.60
Total Consumables and Reagents	17.03	9.73	12.56	16.32	9.33	12.55
Mobile Equipment (excl. Fuel)	3.18	1.82	2.34	3.06	1.75	2.36
Mobile Equipment Fuel	2.52	1.44	1.86	2.43	1.39	1.87
Total Mobile Equipment	5.70	3.26	4.21	5.50	3.14	4.23
Power (excl. Fuel)	3.92	2.24	2.89	3.84	2.19	2.95
Power Generation Fuel	5.07	2.90	3.74	5.07	2.90	3.90
Total Power	8.99	5.14	6.63	8.91	5.09	6.85
Total Operating Cost	**37.24**	**21.28**	**27.46**	**36.23**	**20.70**	**27.87**

Source: Ausenco, 2022

In addition, the process operating costs are split between the plant site and the Mineral Terminal site. The battery limit for the process operating costs are as follows:

- receipt of ore to the ROM bin
- discharge of concentrate to the shiploading conveyor

- discharge of tailings from the overland tailings pipeline to the tailings distribution system
- discharge of coarse rejects to the coarse rejects stockpile.

21.2.3.1 Maintenance Parts and Supplies

The beneficiation and Mineral Terminal site annual maintenance costs were derived from the total installed mechanical equipment cost for each phase based on the mechanical equipment list using a factor of 4%. The factor was derived based on internal benchmarks for maintenance costs of operating plants. The annual maintenance costs are summarized in Table 21-41.

Table 21-41: Annual Equipment Maintenance Costs

WBS	Area	Equipment Cost (US$)	Total Maintenance Cost (US$)	Unit Cost (US$/t)
120	Feed Preparation	629,581	25,183	0.01
130	Scrubber Feed	431,681	17,267	0.01
140	Scrubbing/ Desliming/ Tailings	5,841,931	233,677	0.13
150	Fine Concentrate Thickening	528,636	21,145	0.01
190	Concentrate Filtration & Storage	9,076,406	363,056	0.21
210	Reagents	109,203	4,368	0.00
230	Water Services	795,091	31,804	0.02
240	Plant Services	71,633	2,865	0.00
250	Plant Air Services	100,939	4,038	0.00
330	Power Supply	4,230	169	0.00
340	Tailings Storage Facility	100,166	4,007	0.00
350	Buildings - Plant Site	8,450	338	0.00
440	Mining Facilities	232,608	9,304	0.01
730	Mineral Terminal Water Services	452,857	18,114	0.01
740	Concentrate Drying/ Storage/ Loadout	7,838,698	313,548	0.18
750	Mineral Terminal Utilities & Services	327,362	13,094	0.01
760	Mineral Terminal Fuels	234,314	9,373	0.01
770	Mineral Terminal Electrical Services	2,684	107	0.00
780	Buildings - Mineral Terminal	12,240	490	0.00
	Total	26,798,709	1,071,948	0.61

21.2.3.2 Labor

A summary of process labor requirements is presented in Table 21-42.

Table 21-42: Labor Requirements for the Process Plant and Mineral Terminal Site

Labor / Contractor Summary	Local / Expat	Quantity	Total Cost (US$/a)
Process Upper Management			
Process Manager	Expat	1	210,000
Secretary	Local	1	9,581
Subtotal		**2**	**219,581**
Beneficiation Plant Operations			
Plant Superintendent	Expat	2	370,909
General Foreman	Local	2	226,991
Process Trainer	Local	2	226,991
Shift Supervisors	Local	4	64,035
Plant Sampler	Local	4	28,424
FEL Operators	Local	12	74,664
Plant Operators	Local	16	99,552
Power Plant Operators	Local	8	49,776
Truck Drivers	Local	94	625,726
Subtotal		**144**	**1,767,067**
Mineral Terminal Operations			
General Foreman	Expat	2	363,185
Shift Supervisors	Local	4	64,035
Plant Sampler	Local	4	28,424
FEL Operators	Local	20	124,440
Dryer Operators	Local	8	49,776
Plant Operators (Product Receiving)	Local	4	24,888
Plant Operators (Stockpile Management)	Local	4	24,888
Power Plant Operators	Local	8	49,776
Subtotal		**54**	**729,411**
Metallurgy			
Senior Metallurgist	Local	1	115,909
Lab Technicians	Local	8	53,253
Subtotal		**9**	**169,162**
Mill Maintenance			
Maintenance Supervisor	Expat	2	370,909
Maintenance Planner	Local	1	115,131
Maintenance Trainer	Local	2	230,262
Mechanical Supervisor	Local	4	306,667
Electrical Supervisor	Local	4	306,667
Mechanical Fitters	Local	8	49,776
Workshop Fitters	Local	4	24,888
Trades Assistants	Local	16	106,507
Electricians	Local	4	24,888
Clerk	Local	2	13,313
Subtotal		**47**	**1,549,007**
Total Process Labor		**256**	**4,434,228**

Staffing estimates were provided by Itafos. The labor costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay laboratory, mobile equipment operators, and haul truck drivers. Salaries and wages are based on escalated rates from the previous study phase, expected local industrial rates, and benchmarks of similar projects. The estimated annual labor cost for the beneficiation plant, concentrate hauling, and Mineral Terminal site operations is US$4.434 million per annum.

21.2.3.3 Consumables and Reagents

Various reagents, consumables, and fuels are required for the operation of the beneficiation plant and Mineral Terminal site. The annual costs for each category are presented in Table 21-43. The derivation of each category is described as follows:

- Drum Scrubber Liners: it was assumed the liner will be replaced once per year.
- Flocculant: based on testwork conducted on fine concentrate samples, the flocculant consumption rate was determined to be 35 g/t of thickener feed.
- Filter Cloths, Plates, and Pressing Diaphragms: each component was assigned a lifespan based on benchmark projects and vendor recommendations.
- Antiscalant: a nominal consumption rate of 5 g/L of process water was used.
- Dryer Diesel: consumption rates were derived based on first principles calculations and confirmed by vendor estimates of diesel consumption. The nominal annual throughput of the dryer was used to calculate the annual fuel consumption.

It should be noted that the lower mass yield associated with the North pit ore has a corresponding impact on the consumption of reagents and consumables for impacted unit operations such as the concentrate dryer.

Table 21-43: Summary of Reagent and Consumables

Area	Description	South Pit			North Pit		
		Annual Cost, US$	Unit Cost, US$/t Feed	% of Total	Annual Cost, US$	Unit Cost, US$/t Feed	% of Total
Drum Scrubbing	Liners	100,000	0.06	0.6	100,000	0.06	0.6
Flocculant	Flocculant	171,284	0.10	1.0	157,432	0.09	1.0
Concentrate Filtration	Filter Cloths, Plates, & Diaphragms	999,249	0.57	5.9	957,990	0.55	5.9
Water Services	Antiscalant	32,546	0.02	0.2	25,889	0.01	0.2
Mineral Terminal Concentrate Drying	Diesel	15,730,183	8.99	92.3	15,080,679	8.62	92.4
Total		17,033,263	9.73	100.0	16,321,990	9.33	100.0
Total Ex. Diesel		1,303,080	0.74	7.7	1,241,311	0.71	7.6

21.2.3.4 Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment, and include fuel, maintenance, consumables, and annual registration and insurance fees. A summary by area of the mobile equipment requirements is presented in Table 21-44.

Table 21-44: Mobile Equipment Summary

Description	Quantity	Annual Cost, US$	Percentage
Beneficiation Plant	21	783,888	14
Concentrate Trucking	37	4,714,674	84
Mineral Terminal	7	106,101	2
Total		5,604,662	100

Concentrate trucking refers to the transport of filtered concentrate from the truck loadout area to the Mineral Terminal site. Concentrate trucking operates 16 h/d, and each truck makes four round trips per day. The operation involves the following mobile equipment:

- Front End Loader
  - Model: CAT 980L
  - Quantity: 2 Duty
- Haul Truck
  - Model: Sinotruck HOWO Truck with 30 CBM Trailer
  - Quantity: 31 Duty, 4 Standby

21.2.3.5 Power

The power costs of the project facilities were calculated from the nominal power requirements summarized in the electrical load list. The applicable power cost is calculated based on information provided by the power plant supplier and is variable depending on how much diesel generation is required to supplement the solar generation costs. Using a diesel price of US$0.86/L, the power cost is US$0.229/kWh for the beneficiation plant and US$0.257/kWh for the Mineral Terminal site. A summary of the power consumption and costs is provided in Table 21-45.

Table 21-45: Power Consumption Summary

WBS	Area	Installed (kW)	South Pit		North Pit	
			Consumption (MWh/a)	Cost (US$/a)	Consumption (MWh/a)	Cost (US$/a)
Plant Site						
120	Feed Preparation	37	127	29,028	127	29,028
130	Scrubber Feed	56	222	50,797	222	50,797
140	Scrubbing/ Desliming/ Tailings	2,547	9,707	2,221,728	9,707	2,221,728
150	Fine Concentrate Thickening	100	284	65,089	284	65,089
190	Concentrate Filtration & Storage	2,717	10,519	2,407,587	10,085	2,308,176
210	Reagents	10	38	8,711	36	8,352
230	Water Services	1,061	3,135	717,450	3135	717,450
250	Plant Air Services	125	332	76,022	332	76,022
270	Electrical Services	25	111	25,341	111	25,341
340	Tailings Storage Facility	110	326	74,570	326	74,570
350	Buildings - Plant Site	370	1801	412,136	1,801	412,136
N/A	TSF Dewatering	500	2900	*	1,892	*
440	Mining Facilities	187	676	154,820	676	154,820
	Subtotal	7,844	30,178	6,243,278	28,734	6,143,509
Mineral Terminal Site						
730	Mineral Terminal Water Services	107	204	52,490	204	52,490
740	Concentrate Drying/ Storage/ Loadout	1,623	8,329	2,143,428	7,985	2,054,925
750	Mineral Terminal Utilities & Services	129	437	112,379	437	112,379
760	Mineral Terminal Fuels	131	380	97,847	380	97,847
770	Mineral Terminal Electrical Services	15	66	17,096	66	17,096
780	Buildings - Mineral Terminal	260	1,265	325,642	1,265	325,642
	Subtotal	2,265	10,681	2,748,881	10,337	2,660,379
Total		10,109	40,859	8,992,160	39,071	8,803,888

Note: *Carried in TSF operating costs.

21.2.4 Shiploading Operating Costs

The marine terminal operating cost estimate is summarized in Table 21-46 grouped into the following four main categories:

- Equipment operating costs – fuel, electricity, operating supplies, and repair costs for support vessels (tugboats, pilot boat, and maintenance barge) and the shiploader. Equipment operating costs are variable costs.
- Labor operating costs – salaries and overhead costs for the operation of support vessels, shiploader, and fuel import terminal. Labor operating costs are fixed costs.
- Miscellaneous operating costs – miscellaneous consumables, services, and training. Miscellaneous operating costs are a blend of fixed and variable costs.
- Sustaining capital operating costs – maintenance and repair costs for the marine terminal infrastructure. Major rehabilitation costs are not included and are not expected to be required for the 25-year design life of the marine terminal. Sustaining capital operating costs are fixed costs.

Table 21-46: Mineral Terminal Operating Costs

Description	Rate per Tonne (US$/t Concentrate)									
	Y1-6	Y7	Y8-9	Y10	Y11-14	Y15	Y16-19	Y20	Y21-24	Y25
Equipment Operating Costs (incl. Fuel)	1.35	1.35	1.41	1.41	1.41	1.41	1.41	1.41	1.41	1.41
Labor Operating Costs	2.28	2.28	2.38	2.38	2.38	2.38	2.38	2.38	2.38	2.38
Miscellaneous Operating Costs	0.16	0.16	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.17
Total	3.79	3.79	3.94	3.94	3.94	3.94	3.94	3.94	3.94	3.94

21.2.4.1 Equipment

Equipment operating costs were estimated by Baird leveraging the results of the marine studies (i.e., navigation simulations and operational modeling) for fuel consumption and historical rates for operating supplies and repairs. The cost breakdown is shown in Table 21-47.

Table 21-47: Mineral Terminal Equipment Operating Costs

Equipment Description	Annual Operating Hours	Operating Supplies (US$/h)	Minor Repairs (US$/h)	Major Repairs (US$/h)	Power Cost (US$/h)	Fuel Cost (US$/h)	Hourly Operating Costs (US$)	Annual Operating Costs (US$k)
Tugboats	2268	39.17	103.16	206.33		245.87	579.95	1,315
Pilot Boat	750	24.66	22.00	44.00		154.80	236.28	177
Maintenance Barge	457	58.18	26.27	210.19		21.62	314.99	144
Shiploader	2640	28.88	12.60	22.08	9.36		72.92	193
Total								1,829

21.2.4.2 Labor

Labor operating costs were estimated by Baird assuming employees are paid a yearly salary while only loading three vessels a month. The cost breakdown is shown in Table 21-48.

Table 21-48: Mineral Terminal Labor Operating Costs

Position	Employees	Annual Salary (US$)	Total Annual Salary (US$M)
Pilots	4	259,392	1,038
Tug Master/Captain	6	155,635	934
Tug Engineer	4	113,702	455
Tug Mate	4	61,286	245
Tug Able Seaman	4	33,869	135
Marine Office Attendant	3	10,752	32
Drivers	6	7,392	44
Semi-Skilled Labor	12	7,392	89
Other Labor Costs			
Pilot Roundtrip Flights			120
Total			3,092

21.2.4.3 Miscellaneous Costs

Miscellaneous operating costs were estimated by Baird using historical rates. The cost breakdown is shown in Table 21-49.

Table 21-49: Mineral Terminal Miscellaneous Operating Costs

Description	Annual Amount (US$k)
Consumables	60
Miscellaneous Materials and Services	129
ISPS/ISO Accreditation & Training	25
Total	215

21.2.4.4 Sustaining Capital

Sustaining capital costs were estimated by Baird using historical rates. The rail-mounted radial telescoping shiploader is a low capital cost option for shiploading and it is anticipated that the shiploader will need to be replaced once during the design life of the marine terminal. The cost breakdown is shown in Table 21-50.

Table 21-50: Mineral Terminal Marine Terminal Sustaining Capital Costs

Description	Annual Amount (US$M)									
	Y1-4	Y5	Y6-9	Y10	Y11-14	Y15	Y16-19	Y20	Y21-24	Y25
Equipment Replacement	0	0	0	0	0	3.0	0	0	0	0
Marine Terminal Maintenance & Repair	0.1	0.4	0.1	2.8	0.1	0.4	0.1	2.8	0.1	0.4
Total	0.1	0.4	0.1	2.8	0.1	3.4	0.1	2.8	0.1	0.4

21.2.5 Site Infrastructure Operating Costs

21.2.5.1 TSF, RWP and WD

21.2.5.1.1 Basis of Estimate

The operating cost basis of estimate for the TSF, RWP and WDs include the following:

- TSF and RWP operating costs include monitoring with an allowance for bi-annual audits and a dam safety review every five years.
- The TSF and RWP operating costs have not included an allowance for dedicated site staff to be allocated to these facilities. It is assumed process plant operations will complete routine (daily) inspections of the TSF and RWP facilities.
- TSF and RWP operating costs include an allowance for annual routine maintenance set at 0.05% of the total TSF construction costs excluding P&Gs and contingency cost allowances.
- Pumping power costs are based on an electricity unit rate of $0.281 per kWh.
- TSF pump usage and annual power consumption is based on the annual LOM water balance with dewatering from active and inactive TSF cells. Annual power consumption from the TSF ranges from 63,000 kWh to 84,752 with an average annual consumption of 78,226 kWh (excluding Year 26 closure).
- RWP pump usage and annual power consumption is based on the annual LOM water balance with dewatering from active and inactive TSF cells feeding into the RWP. Annual power consumption from the RWP averages 78,921 kWh.

21.2.5.1.2 Operating Cost Summary

The average annual and total LOM operating costs for the TSF and RWP for Years 1 to 25 are summarized in Table 21-51. The monitoring and maintenance costs are combined for the TSF and RWP.

Table 21-51: TSF Cell 1 to 7 & RWP Operating Cost Estimate

Description	TSF (US$)		RWP (US$)		Combined Total (US$)	
	Annual Average	LOM Total	Annual Average	LOM Total	Annual Average	LOM Total
Monitoring	75,000	1,875,000			75,000	1,875,000
Power	21,982	549,540	22,177	554,420	44,159	1,103,960
Maintenance	17,200	430,000			17,200	430,000
Subtotal	114,182	2,854,540	22,177	554,420	136,359	3,408,960
+15% Contingency	17,127	428,181	3,327	83,163	20,454	511,344
Total	**131,309**	**3,282,721**	**25,504**	**637,583**	**156,813**	**3,920,304**

21.2.5.2 Surface Water and Pit Dewatering Operating Costs

21.2.5.2.1 Basis of Estimate

The operating cost basis of estimate for the surface water and pit dewatering includes the following:

- Pumping power costs are based on electricity unit rate of $0.229/kWh
- South pit dewatering pump usage and annual power consumption is based on low-, medium- and high-flow wells.

- Annual power consumption for the low flow wells ranges from 234,000 kWh (Year 7) to 653,000 kWh (Year 2) with an average annual consumption of 499,000 kWh.
- Annual power consumption for the medium flow wells ranges from 928,000 kWh (Year 1) to 1,152,000 kWh (Year 2) with an average annual consumption of 1,006,000 kWh.
- Annual power consumption for the high flow wells is estimated at 1,296,000 kWh.

21.2.5.2.2 Operating Cost Summary

The annual and total operating costs for the low flow, medium flow and high flow wells for the South pit dewatering are summarized in Table 21-52. Maintenance and replacement costs are built into the 15% contingency identified above in the operating costs.

The average annual and total operating costs for the low flow, medium flow and high flow wells for the North pit dewatering are summarized in Table 21-53. Maintenance and replacement costs are built into the 15% contingency identified above in the operating costs. A summary of the surface water management infrastructure operating costs is given in Table 21-54.

Table 21-52: South Pit Dewatering Operating Cost Estimate (Year 1 to 7) in US Dollars

Well Flow	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Low	123,595	149,344	125,655	136,984	104,026	106,086	53,558	799,247
Medium	212,400	263,669	227,048	227,048	227,048	227,048	227,048	1,611,312
High	296,628	296,628	296,628	296,628	296,628	296,628	296,628	2,076,395
Subtotal	632,623	709,641	649,331	660,661	627,702	629,762	577,234	4,486,954
+15% Contingency	94,893	106,446	97,400	99,099	94,155	94,464	86,585	673,043
Total	727,516	816,087	746,731	759,760	721,857	724,226	663,819	5,159,997

Table 21-53: North Pit Dewatering Operating Cost Estimate (Year 7 to 25) in US Dollars

Well Flow	Annual Average	Total
Low	38,596	733,330
Medium	161,131	3,061,492
High	210,762	4,004,476
Total	410,489	7,799,298
+15% Contingency	61,573	1,169,895
Total	472,062	8,969,193

Table 21-54: Surface Water Management Infrastructure Operating Cost Estimate in US Dollars

Description	Average Annual	LOM Total
Sediment Control Dam 1	49,175	1,229,380
Sediment Control Dam 2	34,804	835,299
Environmental Control Dam	17,164	429,101
Flood Protection Bund	33,800	845,000
Power (water pumps)	29,995	749,875
Subtotal	164,938	4,088,655
+15% Contingency	24,741	613,298
Total	189,679	4,701,953

21.2.6 General and Administrative Operating Costs

General and administrative (G&A) costs are expenses not directly related to the production of the phosphate concentrate, and include expenses not included in mining, process, and transportation costs. These costs were developed based on inputs from Itafos and from a G&A labor estimate. A summary of G&A costs is presented in Table 21-55.

Table 21-55: General and Administrative Cost Summary

Category	Cost (US$/year)
G&A Expenses	4,049,434
G&A Labor	3,108,051
Corporate Costs	313,000
Total G&A	7,470,486

The G&A costs are divided into the following areas:

- G&A Expenses
  - G&A maintenance, including mobile equipment for G&A use
  - town office, including rental and other costs
  - site office, including telecommunications and stationery
  - insurance
  - fees, including tenement maintenance
  - consultants, including metallurgical testing
  - personnel, including first aid and medical, recreation, safety supplies, travel, recruiting, and training
  - contracts, including cleaning and sanitation services, catering and worker transport
  - general, including the site laboratory and other general services.
- Corporate Costs
  - banking charges
  - auditing costs
  - travel and accommodation
- G&A Labor

G&A labor costs were developed from a first principles estimate, and the required personnel are summarized in Table 21-56.

Table 21-56: G&A Labor Summary

Labor / Contractor Summary	Local / Expat	No. Per Shift	No. Shifts	Quantity
Management				
Site Manager	Local	1	1	1
Manager Corporate Affairs	Local	1	1	1
Secretary	Local	1	1	1
Itafos Office				
Resident Manager (GB Minerals)	Expat	1	1	1
Motor Pool Driver (GB Minerals)	Local	1	1	1
Environment, Health and Safety				
SHE Manager	Expat	1	1	1
OH&S Officer	Local	1	1	1
Environmental Monitoring Officer	Local	1	1	1
Environmental Rehabilitation Officer	Local	1	1	1
Environmental Technicians	Local	1	1	1
Doctor	Local	1	1	1
Nurses	Local	1	1	1
First Aid Officer	Local	1	1	1
Human Resources				
HR Manager	Expat	1	1	1
Senior HR Officer	Local	1	1	1
HR Officer	Local	1	1	1
Training Superintendent	Expat	1	1	1
Mine Security				
Manager Security	Local	1	1	1
Security Supervisors	Local	4	1	4
Security Staff	Local	6	4	24
Mineral Terminal Security				
Manager Security	Local	1	1	1
Security Supervisors	Local	1	1	1
Security Staff	Local	3	4	12
Finance & Administration				
Administration Manager	Expat	1	1	1
Senior Accountant	Local	1	1	1
Accountant	Local	1	1	1
Accounts Clerk	Local	2	1	2
Payroll Clerk	Local	2	1	2
Purchasing Officer	Local	1	1	1
Warehouse Manager	Local	1	1	1
Warehouse Officer	Local	1	1	1
Warehouse Labor	Local	4	1	4

21.3 Closure Cost Estimate

Most closure activities will be completed progressively during mining. The estimated cost to implement the identified progressive reclamation activities is $80.6 million, and final closure costs are estimated at an additional $34.6 million. Progressive and final closure costs are broken down by project component in Table 21-57.

Table 21-57: Breakdown of Progressive and Final Closure Costs by Component

Area/Component	Progressive Reclamation Cost (US$)	Final Closure Cost (US$)
Mine Site		
Tailings Storage Facility	47,053,496	4,151,109
Waste Dumps	12,211,363	-
Process Plant	-	566,350
Truck Loadout Facility	-	507,321
Electrical Infrastructure	25,000	223,250
Buildings and Other Structures	25,000	978,716
Open Pits	3,346,377	11,981,264
Haul and Access Roads	-	150,799
Equipment and Machinery	-	300,000
Landfills and Waste Management	-	283,600
Water Management	560,206	3,622,758
Subtotal Mine Site	63,221,441	22,765,165
Mineral Terminal Site		
Buildings and Other Structures	25,000	822,790
Equipment & Machinery	-	50,000
Electrical Infrastructure	-	57,500
Water Management	-	4,600
Waste Management	-	15,500
Subtotal Mineral Terminal Site	25,000	950,390
Post-Closure, Monitoring and Maintenance	742,464	3,287,156
Subtotal	63,988,906	27,002,712
Project Management (5%)	N/A	1,350,136
Engineering Supervision (5%)	3,199,445	1,350,136
General and Administrative Costs (5%)	N/A	1,350,136
Recommended 20% Contingency	12,797,781	5,400,542
Total	**79,986,132**	**36,453,661**

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.

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this chapter represent forward-looking information as defined under Canadian securities law.

The results of the economic analysis are subject to several known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

Forward-looking information includes the following:

- mineral resource and mineral reserve estimates
- assumed commodity price and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- sustaining costs and proposed operating costs
- interpretations and assumptions regarding contract mining terms
- assumptions as to closure costs and closure requirements
- assumptions about environmental, permitting, and social risks.

Additional risks to the forward-looking information include the following:

- changes to costs of production from what is assumed
- changes in the estimated timing and quantity of production
- unrecognized environmental and social risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade, or recovery rates
- geotechnical or hydrogeological considerations during mining being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment, or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- ability to maintain the social license to operate
- accidents, labor disputes, and other risks of the mining industry
- changes to interest rates
- changes to tax rates and incentive programs
- changes in government regulation of mining operations and
- potential delays in the issuance of permits and any conditions imposed with the permits that are granted.

22.2 Methodologies Used

The economic analysis is based on the mineral reserves as defined in Section 15, the mining methods and production schedule as expressed in Section 16, the recovery and processing methods as described in Section 17, and the capital and operating costs as outlined in Section 21.

The project has been evaluated using a discounted cash flow (DCF) analysis based on a 10% discount rate. This is the typical rate; however, Table 22-2 also presents DCF at 5%, 8%, 10% and 15%. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including preproduction costs, operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections.

Cash flows are taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in rock prices, discount rate, fuel prices, operating costs, and capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Section 21 in Q4 2022 United States dollars. The economic analysis has been run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

The financial model is based on a grade-adjusted P_2O_5 average price (see Section 19 of this report). The forecasts are meant to reflect the average rock price expectation over the life of the project.

The economic analysis was performed using the following assumptions:

- detailed design, construction, commissioning, and ramp-up duration as per Section 24.3
- mine life of 25 years
- exchange rate of 0.98 (USD:Euro)
- cost estimates in constant Q4 2022 US dollars without any inflation or escalation adjustment
- results are based on 100% ownership
- capital costs funded with 100% equity (no financing costs assumed)
- all cash flows discounted to start of construction using mid-period discounting convention
- rock is assumed to be sold in the same year it is produced
- project revenue is derived from the sale of the rock into the international marketplace.

22.3.1 Taxes

The project has been evaluated on a post-tax basis to provide an approximate value of the potential economics. The tax model was compiled based on ongoing discussions with the Guinea-Bissau government. The calculations are based on the tax regime as of the date of the feasibility study. At the effective date of the cashflow analysis, the project was assumed to be subject to the following tax regime:

- Guinea-Bissau corporate income tax of 22%
- additional 7% surcharge tax on the income allocated for tax collection
- three-year tax holiday from commencement of production.

At the rock price assumption, total tax payments are estimated to be US$703.5 million over the life of mine.

22.3.2 Royalty

A 2.0% net revenue royalty (revenue generated multiplied by the applicable royalty percentages) has been assumed for the project, resulting in approximately US$135.9 million in royalty payments over the life of mine.

22.3.3 Closure Costs

Closure costs include all the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude (this includes a 100-year post-closure monitoring period). The closure costs total US$27.3 million. Additional to this is a sum of US$71.5 million for the progressive closure of TSF cells, and the south and north dewatering pits.

22.4 Economic Analysis

The economic analysis was performed assuming a 10% discount rate. The pre-tax $NPV_{10\%}$ is US$730 million; the IRR is 37.8%; and payback period is 4.2 years. On a post-tax basis, the $NPV_{10\%}$ is US$572 million; the IRR is 34.9%; and the payback period is 4.2 years. A summary of project economics is listed in Tables 22-1, 22-2, 22-3 and shown graphically in Figures 22-1 and 22-2. The analysis was done on an annual cashflow basis as shown in Table 22-4.

Table 22-1: Financial Data

Description	Life-of-Mine (US$M)
Revenue	6,497.2
Total Preproduction Capital	308.3
Total All-in LOM Operating Costs (see below)	2,332.1
Total Sustaining Capital (including Progressive Closure and Final Closure Costs – See Below)	467.1
Operating Margin Ratio (Operating Revenue / Operating Cost)	2.8
Royalties	129.9
Income Taxes	714.8
Pre-Tax Cumulative Cash Flow	3,259.8
After-Tax Cumulative Cash Flow	2,545.0
Detail of Expenditures	
Total Operating Costs	2,270.5
Total Other Costs (Corporate Overhead)	61.7
Total All-in LOM Operating Costs	2,332.1
Sustaining Capital Cost	374.5
Sustaining Capital Cost – Progressive Closure	80.0
Closure Capital Cost	12.7
Total Sustaining Capital (including Progressive Closure and Final Closure Costs)	467.1

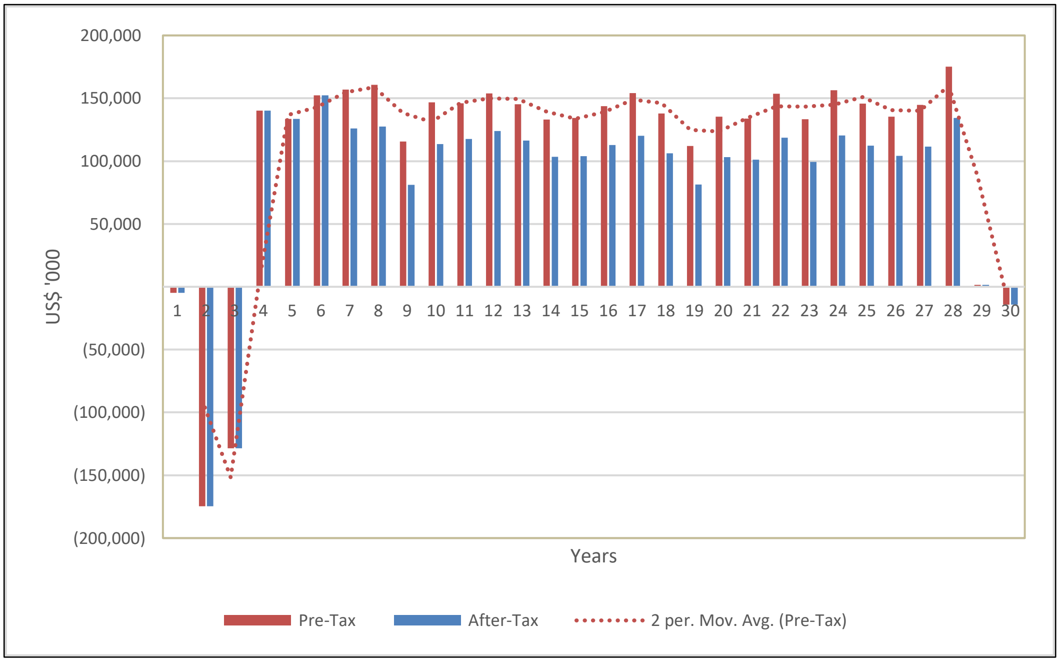
Table 22-2: Financial Statistics

Description	Unit	After-Tax	Pre-tax
Cumulative Net Cash Flow			
Undiscounted (Base Year 2024)	US$k	2,544,960	3,259,753
Net Present Value			
Discounted at 5%	US$k	1,148,827	1,464,435
Discounted at 8%	US$k	749,268	954,843
Discounted at 10%	US$k	572,028	729,998
Discounted at 15%	US$k	300,575	387,968
Internal Rate of Return	%	34.9	37.8
Payback Period	Years	4.2	4.2

Table 22-3: Income Tax Holiday Impact on After-Tax Financial Statistics

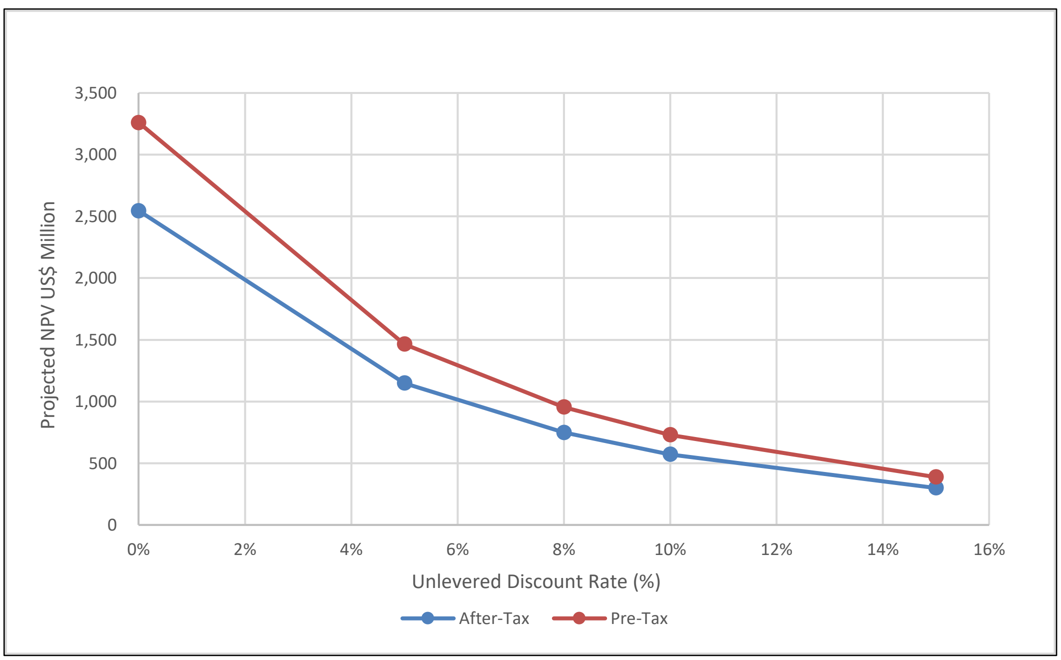
Description	Unit	With 3 Years Tax Holiday	Without Tax Holiday
Net Present Value Discounted at 10%	US$k	572,028	526,660
Internal Rate of Return	%	34.9	31.5
Income Tax Payable	US$k	714,793	788,150
Payback Period	Years	4.2	4.2

Figure 22-1: Pre-Tax and After-Tax Cashflow



Source: Kristal Font, 2023

Figure 22-2: Projected NPV at Various Unlevered Discount Rates



Source: Kristal Font, 2023

Table 22-4: Projected Cash Flow on an Annualized Basis

Project Year					0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29
Production Year					-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
			Life-of-Mine Total					South	South	South	South	South	South	South	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North		
Ore																																		
Overburden	kbcm		441,513				5,812	8,661	10,081	14,892	12,969	16,800	13,510	17,348	17,847	15,017	18,005	18,361	17,083	17,768	16,525	19,356	23,981	23,006	19,457	17,438	16,609	15,228	18,335	23,654	21,300	22,470		
Ore	kt		43,750					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	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750		
Recovered Ore	kt		32,898					1,356	1,356	1,356	1,356	1,356	1,356	1,356	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300		
Strip Ratio (BCM:Tons)								4.95	5.76	8.51	7.41	9.60	7.72	9.91	10.20	8.58	10.29	10.49	9.76	10.15	9.44	11.06	13.70	13.15	11.12	9.96	9.49	8.70	10.48	13.52	12.17			
Revenue																																		
MRRC Base Case Rock (72% BPL Rock)	US$k	100%																																
33.5% P2O5 Phos rock	US$k	100%	6,497,243					275,319	257,688	261,756	275,319	275,319	275,319	275,319	251,493	251,493	251,493	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448	256,448		
Total Revenue	**US$k**		**6,497,243**					**275,319**	**257,688**	**261,756**	**275,319**	**275,319**	**275,319**	**275,319**	**251,493**	**251,493**	**251,493**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**	**256,448**		
				Average US$/t																														
Royalty & Freight Cost																																		
Ocean freight (FOB)	US$k																																	
Royalties	US$k		(129,945)					(5,506)	(5,154)	(5,235)	(5,506)	(5,506)	(5,506)	(5,506)	(5,030)	(5,030)	(5,030)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)	(5,129)		
Total Royalty & Freight	**US$k**		**(129,945)**					**(5,506)**	**(5,154)**	**(5,235)**	**(5,506)**	**(5,506)**	**(5,506)**	**(5,506)**	**(5,030)**	**(5,030)**	**(5,030)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**	**(5,129)**		
Net Revenue	**US$k**		**6,367,298**					**269,812**	**252,534**	**256,521**	**269,812**	**269,812**	**269,812**	**269,812**	**246,463**	**246,463**	**246,463**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**	**251,319**		
Operating costs																																		
Mining																																		
Mining opex (excl fuel costs)	US$k	100%	661,368	20.1				33,817	39,712	34,239	36,872	32,069	18,326	24,147	23,623	20,974	24,148	26,235	25,038	27,261	21,795	25,632	28,712	28,221	24,869	22,575	21,537	20,261	23,827	29,049	26,598	21,830		
	US$k	100%																																
Non-Mining	US$k																																	
G&A	US$k	100%	186,762	5.7				7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470	7,470		
Labor (excl. Mining & Shiploading)	US$k	100%	110,856	3.4				4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434	4,434		
Operating Consumables (excluding Fuel)	US$k	100%	31,465	1.0				1,303	1,303	1,303	1,303	1,303	1,303	1,303	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241	1,241		
Power (excluding fuel)	US$k	100%	96,467	2.9				3,919	3,919	3,919	3,919	3,919	3,919	3,919	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835	3,835		
Plant Maintenance	US$k	100%	26,799	0.8				1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072	1,072		
Mobile Equipment	US$k	100%	77,410	2.4				3,180	3,180	3,180	3,180	3,180	3,180	3,180	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064		
Tailings & Water Management	US$k	100%	7,760	0.2				256	296	303	303	360	303	296	296	303	360	303	303	296	296	360	303	303	303	299	360	303	303	296	303	360		
Environmental & Resettlement	US$k	100%	7,987	0.2				430	289	311	289	374	294	292	458	292	342	307	294	336	289	340	289	307	313	292	342	292	289	311	294	322		
Shiploading Costs																																		
Labor	US$k	100%	77,305	2.3				3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092	3,092		
Equipment (excl vessel fuel)	US$k	100%	28,635	0.9				1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145	1,145		
Maintenance, Consumables & Others	US$k	100%	5,364	0.2				215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215	215		
Fuel	US$k																																	
Mine fuel consumption	`kL	100%	424,822	kL				9,435	10,731	14,008	12,116	15,062	13,146	18,332	17,756	15,710	18,385	19,796	19,011	20,819	16,039	19,321	21,851	21,356	18,753	16,522	15,988	14,895	17,835	22,302	20,230	15,422		
Power Fuel Consumption	`kL		147,476					5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899	5,899		
Mobile Equipment fuel Consumption	`kL		71,479					2,936	2,936	2,936	2,936	2,936	2,936	2,936	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829	2,829		
Vessel Fuel Consumption	`kL		19,874					795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795	795		
Others - Concentrate Drying	`kL		443,678					18,291	18,291	18,291	18,291	18,291	18,291	18,291	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536	17,536		
Total Fuel Consumption	**`kL**		**1,107,329**					**37,356**	**38,652**	**41,929**	**40,037**	**42,982**	**41,067**	**46,253**	**44,815**	**42,769**	**45,444**	**46,855**	**46,070**	**47,878**	**43,098**	**46,380**	**48,910**	**48,415**	**45,812**	**43,581**	**43,047**	**41,954**	**44,894**	**49,361**	**47,289**	**42,481**		
Total Fuel Cost	US$k	100%	952,303	28.9				32,126	33,241	36,059	34,432	36,965	35,317	39,778	38,541	36,782	39,082	40,295	39,620	41,175	37,064	39,887	42,062	41,637	39,398	37,480	37,020	36,081	38,609	42,450	40,668	36,533		
	US$k																																	
Total Operating Costs	**US$k**	**100%**	**2,270,479**	69.02				**92,460**	**99,368**	**96,742**	**97,726**	**95,598**	**80,071**	**90,343**	**88,487**	**83,920**	**89,501**	**92,710**	**90,825**	**94,637**	**85,013**	**91,789**	**96,935**	**96,038**	**90,451**	**86,215**	**84,828**	**82,506**	**88,596**	**97,675**	**93,433**	**84,615**		
Other Costs																																		
Corporate Overhead			58,938	1.8				2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358	2,358		
Pre-production Cost(ramp up- Salaries)			2,721	0.1				2,721																										
Principal Repayment																																		
Financing Costs																																		
Total Other costs			**61,659**	1.87				**5,079**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**	**2,358**		
Capital expenditure																																		
Construction capital	US$k		**308,270**		4,871	174,773	128,627																											
SUSEX - Shiploading	US$k		**12,050**					120	120	120	120	360	120	120	120	120	2,760	120	120	120	120	3,410	120	120	120	120	2,760	120	120	120	120	360		
SUSEX -Resettlement and Other	US$k		**5,635**					2,606	606	606	606	606	606																					
SUSEX -Mining	US$k		**265,348**					718	881	1,301	1,133	3,446	70,105	9,713	4,960	4,255	2,769	14,531	18,291	4,170	11,054	5,363	30,883	10,534	11,682	3,311	21,516	4,590	5,239	11,617	10,670	2,617		
SUSEX -Water Management	US$k		**10,482**					1,602	286	239	581	225	195	593	1,567	100	100	100	100	100	100	100	100	100	100	100	3,593	100	100	100	100	100		
SUSEX -TSF	US$k		**47,240**					3,269	8,382	107	537	102	67	11,301	1,130	25	25	25	25	3,151	26	7,722	26	25	6,519	25	1,551	25	3,100	25	25	25		
SUSEX -South Pit Dewatering (Includes Pump Power Costs)	US$k		**12,735**					2,011	3,517	1,756	2,127	1,526	1,112	687																				
SUSEX -North Pit Dewatering (Includes Pump Power Costs)	US$k		**20,991**											5,094	883	883	883	883	883	883	883	883	883	883	883	883	883	883	883	883	883	883		
Working capital	US$k							21,738	1,815	(677)	178	(532)	(3,882)	2,568	(345)	(1,142)	1,395	778	(471)	953	(2,406)	1,694	1,286	(224)	(1,397)	(1,059)	(347)	(581)	1,523	2,270	(1,060)	(22,076)		
Closure & salvage																																		
SUSEX -Progressive Closure and Rehabilitation (TSF Only)	US$k		**58,817**								7,424	5,431	1,298	156	156	156	156	6,685	4,738	1,026				6,158	4,469	3,766	818	4,906	3,560	771		7,142		
SUSEX -Progressive Closure and Rehabilitation (Pits & WD)			**21,169**						1,644	1,534	34	34	2,037	46	1,198	1,960	1,164	34	34	34	34	34	6,743	34	2,204	2,006	34	34	34	59	34	165		
Total site	US$k		**33,997**																														19,800	14,196
Salvage value - mine	US$k		**(12,893)**																														(12,893)	
Salvage value - Plant & Port	US$k		**(8,433)**																														(8,433)	

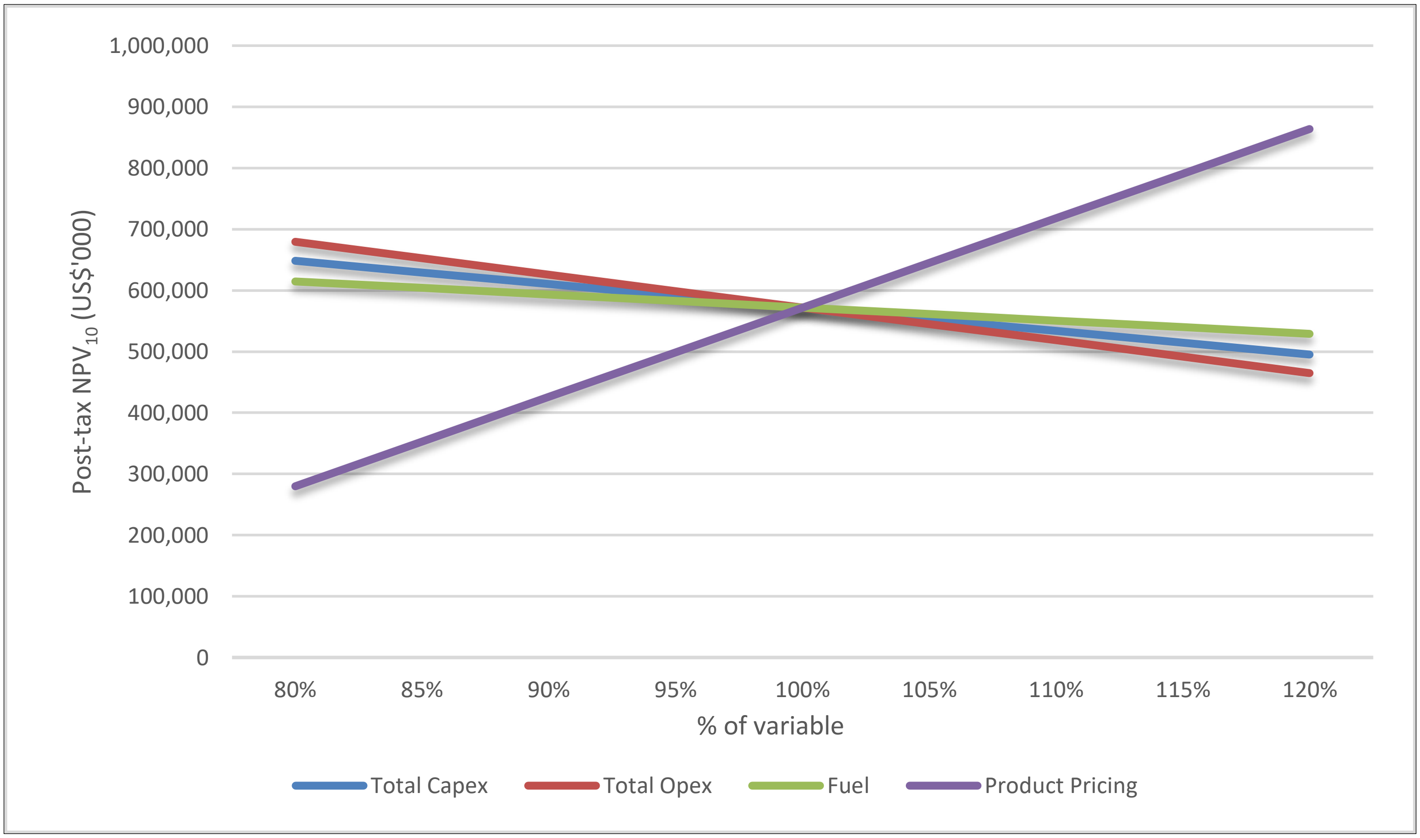
Project Year				0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29
Production Year				-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
			Life-of-Mine Total				South	South	South	South	South	South	South	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North	North		
Total Capital Cost	**US$k**	**100%**	**775,407**	**4,871**	**174,773**	**128,627**	**32,064**	**17,251**	**4,986**	**12,740**	**11,197**	**71,659**	**30,278**	**9,670**	**6,357**	**9,253**	**23,156**	**23,720**	**10,437**	**9,812**	**19,206**	**40,041**	**17,630**	**24,579**	**9,153**	**30,809**	**10,078**	**14,559**	**15,845**	**10,772**	**(10,784)**	**(1,526)**	**14,196**
Earnings																																	
EBITDA (Earnings before taxes, depreciation & amortization)	**US$k**		**4,035,160**				**172,274**	**150,809**	**157,422**	**169,729**	**171,857**	**187,384**	**177,112**	**155,618**	**160,186**	**154,605**	**156,252**	**158,137**	**154,325**	**163,949**	**157,173**	**152,027**	**152,924**	**158,511**	**162,747**	**164,134**	**166,456**	**160,366**	**151,287**	**155,529**	**164,347**		
Depreciation																																	
Non-Equipment Depreciable amount per year																																	
Opening Balance	US$k					166,502	189,970	296,749	266,045	239,301	199,346	229,191	152,235	162,830	154,631	128,412	113,921	91,749	62,928	56,249	48,009	59,384	35,220	54,874	33,885	53,674	22,552	34,612	26,016	16,594	12,732		
Current Year Depreciation	US$k		**436,142**			24,975	28,495	44,512	39,907	35,895	29,902	34,379	22,835	24,425	23,195	19,262	17,088	13,762	9,439	8,437	7,201	8,908	5,283	8,231	5,083	8,051	3,383	5,192	3,902	2,489	1,910		
Closing Balance	US$k					141,527	161,474	252,237	226,138	203,406	169,444	194,812	129,400	138,406	131,436	109,150	96,833	77,987	53,489	47,812	40,807	50,476	29,937	46,643	28,802	45,623	19,170	29,421	22,113	14,105	10,822		
Depreciation Equipment	US$k		**250,884**			1,982	2,042	2,042	2,042	2,042	2,042	9,536	10,356	10,813	11,199	11,260	12,686	15,069	12,519	13,614	13,540	11,572	11,605	12,227	9,981	12,570	11,471	11,929	12,165	12,636	11,943		
	US$k																																
Total Depreciation			**687,026**			**26,958**	**30,538**	**46,554**	**41,949**	**37,937**	**31,944**	**43,915**	**33,191**	**35,238**	**34,394**	**30,522**	**29,774**	**28,831**	**21,959**	**22,051**	**20,741**	**20,480**	**16,888**	**20,458**	**15,064**	**20,621**	**14,854**	**17,121**	**16,067**	**15,125**	**13,852**	**0**	**0**
Taxation																																	
Operating cash flow	US$k		**4,035,160**	0	0	0	150,536	148,994	158,098	169,551	172,389	191,266	174,544	155,963	161,328	153,210	155,475	158,609	153,371	166,355	155,479	150,741	153,148	159,907	163,806	164,480	167,037	158,843	149,018	156,590	186,423	0	
Depreciation	US$k		**(687,026)**	0	0	(26,958)	(30,538)	(46,554)	(41,949)	(37,937)	(31,944)	(43,915)	(33,191)	(35,238)	(34,394)	(30,522)	(29,774)	(28,831)	(21,959)	(22,051)	(20,741)	(20,480)	(16,888)	(20,458)	(15,064)	(20,621)	(14,854)	(17,121)	(16,067)	(15,125)	(13,852)	0	
Pre-tax net cash flow	US$k		**3,348,134**	0	0	(26,958)	119,998	102,439	116,150	131,613	140,445	147,351	141,353	120,725	126,934	122,688	125,701	129,777	131,413	144,303	134,738	130,261	136,261	139,449	148,742	143,859	152,183	141,722	132,951	141,465	172,571	0	
Carryover NOL available	US$k			0	0	0	26,958	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Carryover NOL used	US$k		**26,958**	0	0	0	26,958	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Carry forward NOL	US$k			0	0	26,958	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Net taxable income	US$k		**3,321,177**	0	0	(26,958)	93,041	102,439	116,150	131,613	140,445	147,351	141,353	120,725	126,934	122,688	125,701	129,777	131,413	144,303	134,738	130,261	136,261	139,449	148,742	143,859	152,183	141,722	132,951	141,465	172,571	0	
Income tax payable	**US$k**	**0%**	**(714,793)**	**0**	**0**	**0**	**0**	**0**	**0**	**(30,982)**	**(33,061)**	**(34,686)**	**(33,274)**	**(28,419)**	**(29,880)**	**(28,881)**	**(29,590)**	**(30,550)**	**(30,935)**	**(33,969)**	**(31,717)**	**(30,663)**	**(32,076)**	**(32,826)**	**(35,014)**	**(33,864)**	**(35,824)**	**(33,361)**	**(31,297)**	**(33,301)**	**(40,623)**	**0**	**0**
Net earnings	US$k		**3,320,367**				172,274	150,809	157,422	138,747	138,796	152,698	143,837	127,200	130,306	125,724	126,662	127,588	123,390	129,980	125,456	121,364	120,848	125,684	127,733	130,269	130,632	127,004	119,991	122,229	123,724		
Net project cash flow																																	
Pre-tax	US$k		**3,259,753**	(4,871)	(174,773)	(128,627)	140,210	133,557	152,436	156,989	160,660	115,725	146,833	145,948	153,829	145,352	133,096	134,418	143,888	154,137	137,967	111,986	135,294	133,931	153,594	133,325	156,378	145,806	135,443	144,758	175,132	1,526	(14,196)
After tax	US$k		**2,544,960**	(4,871)	(174,773)	(128,627)	140,210	133,557	152,436	126,007	127,599	81,039	113,559	117,529	123,948	116,472	103,507	103,868	112,953	120,168	106,250	81,323	103,218	101,105	118,580	99,460	120,554	112,445	104,146	111,457	134,508	1,526	(14,196)
Cumulative Cashflow																																	
Pre-tax	US$k			(4,871)	(179,644)	(308,270)	(168,060)	(34,503)	117,933	274,922	435,582	551,307	698,141	844,089	997,917	1,143,269	1,276,366	1,410,784	1,554,671	1,708,809	1,846,776	1,958,762	2,094,056	2,227,987	2,381,582	2,514,906	2,671,285	2,817,091	2,952,534	3,097,291	3,272,423	3,273,949	3,259,753
After-tax	US$k			(4,871)	(179,644)	(308,270)	(168,060)	(34,503)	117,933	243,941	371,540	452,578	566,137	683,667	807,615	924,087	1,027,593	1,131,461	1,244,414	1,364,583	1,470,833	1,552,155	1,655,373	1,756,478	1,875,059	1,974,519	2,095,073	2,207,519	2,311,664	2,423,121	2,557,630	2,559,156	2,544,960

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the post-tax NPV and IRR of the project using the following variables: revenue (P_2O_5 rock price), operating cost, total capital cost, and fuel. Post-tax sensitivity results are shown in Figure 22-3 and Tables 22-5 to 22-8.

The analysis revealed that the project is most sensitive to changes in P_2O_5 rock price, total operating cost, total capital cost, and fuel, in the order listed.

Figure 22-3: After-Tax NPV_{10} Sensitivity Graph (US$k)



Source: Kristal Font, 2023

Table 22-5: Operating Cost vs. Revenue NPV Sensitivities

Description		Operating Cost (Mining, Non-Mining, Shiploading & Fuel)								
After-Tax NPV @ 10% (US$k)	572,028	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue, Product Pricing	120%	172,798	245,793	318,788	391,782	464,777	537,772	610,767	683,761	756,756
	115%	199,611	272,606	345,600	418,595	491,590	564,585	637,579	710,574	783,569
	110%	226,424	299,418	372,413	445,408	518,402	591,397	664,392	737,387	810,381
	105%	253,236	326,231	399,226	472,220	545,215	618,210	691,205	764,199	837,194
	100%	280,049	353,044	426,038	499,033	**572,028**	645,022	718,017	791,012	864,007
	95%	306,861	379,856	452,851	525,846	598,840	671,835	744,830	817,825	890,819
	90%	333,674	406,669	479,664	552,658	625,653	698,648	771,642	844,637	917,632
	85%	360,487	433,481	506,476	579,471	652,466	725,460	798,455	871,450	944,445
	80%	387,299	460,294	533,289	606,284	679,278	752,273	825,268	898,262	971,257
	75%	414,112	487,107	560,101	633,096	706,091	779,086	852,080	925,075	998,070

Note: Sensitivities shown in rows, top to bottom.

Table 22-6: Total Capital Cost vs. Revenue NPV Sensitivities

Description		Capital Cost (including Owner's Cost, Sustaining & Closure Costs)								
After-Tax NPV @ 10% (US$k)	572,028	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue, Product Pricing	120%	203,455	276,458	349,461	422,464	495,467	568,470	641,473	714,476	787,479
	115%	222,603	295,604	368,605	441,606	514,607	587,608	660,609	733,610	806,611
	110%	241,752	314,751	387,750	460,748	533,747	606,746	679,745	752,744	825,743
	105%	260,900	333,897	406,894	479,891	552,887	625,884	698,881	771,878	844,875
	100%	280,049	353,044	426,038	499,033	**572,028**	645,022	718,017	791,012	864,007
	95%	299,197	372,190	445,183	518,175	591,168	664,161	737,153	810,146	883,139
	90%	318,346	391,336	464,327	537,318	610,308	683,299	756,289	829,280	902,271
	85%	337,494	410,483	483,471	556,460	629,448	702,437	775,426	848,414	921,403
	80%	356,643	429,629	502,616	575,602	648,589	721,575	794,562	867,548	940,535
	75%	375,791	448,776	521,760	594,745	667,729	740,713	813,698	886,682	959,667

Note: Sensitivities shown in rows, top to bottom.

Table 22-7: Fuel Cost vs. Revenue NPV Sensitivities

Description		Fuel								
After-Tax NPV @ 10% (US$k)	572,028	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue, Product Pricing	120%	237,424	310,419	383,414	456,409	529,403	602,398	675,393	748,388	821,382
	115%	248,081	321,075	394,070	467,065	540,059	613,054	686,049	759,044	832,038
	110%	258,737	331,731	404,726	477,721	550,716	623,710	696,705	769,700	842,694
	105%	269,393	342,387	415,382	488,377	561,372	634,366	707,361	780,356	853,351
	100%	280,049	353,044	426,038	499,033	572,028	645,022	718,017	791,012	864,007
	95%	290,705	363,700	436,694	509,689	582,684	655,679	728,673	801,668	874,663
	90%	301,361	374,356	447,350	520,345	593,340	666,335	739,329	812,324	885,319
	85%	312,017	385,012	458,007	531,001	603,996	676,991	749,985	822,980	895,975
	80%	322,673	395,668	468,663	541,657	614,652	687,647	760,642	833,636	906,631
	75%	333,329	406,324	479,319	552,313	625,308	698,303	771,298	844,292	917,287

Note: Sensitivities shown in rows, top to bottom.

Table 22-8: Operating Cost vs. Total Capital Cost NPV Sensitivities

Description		Operating Cost (Mining, Non-Mining, Shiploading & Fuel								
After-Tax NPV @ 10% (US$k)	572,028	80%	85%	90%	95%	100%	105%	110%	115%	120%
Total Capital Cost	120%	541,916	522,631	503,347	484,062	464,777	445,492	426,208	406,923	387,638
	115%	568,584	549,336	530,087	510,838	491,590	472,341	453,093	433,844	414,595
	110%	595,252	576,040	556,827	537,615	518,402	499,190	479,977	460,765	441,552
	105%	621,921	602,744	583,568	564,391	545,215	526,039	506,862	487,686	468,510
	100%	648,589	629,448	610,308	591,168	572,028	552,887	533,747	514,607	495,467
	95%	675,257	656,153	637,049	617,944	598,840	579,736	560,632	541,528	522,424
	90%	701,925	682,857	663,789	644,721	625,653	606,585	587,517	568,449	549,381
	85%	728,593	709,561	690,529	671,498	652,466	633,434	614,402	595,370	576,338
	80%	755,261	736,266	717,270	698,274	679,278	660,282	641,287	622,291	603,295
	75%	781,929	762,970	744,010	725,051	706,091	687,131	668,172	649,212	630,252

Note: Sensitivities shown in rows, top to bottom.

Table 22-9: Operating Cost vs. Revenue IRR Sensitivities

Description		Operating Cost (Mining, Non-Mining, Shiploading & Fuel)								
After-Tax IRR	34.9%	80%	85%	90%	95%	100%	105%	110%	115%	120%
Revenue, Product Pricing	120%	17.4%	20.9%	24.2%	27.3%	30.3%	33.3%	36.2%	39.0%	41.8%
	115%	18.8%	22.2%	25.4%	28.5%	31.5%	34.4%	37.3%	40.1%	42.8%
	110%	20.1%	23.4%	26.6%	29.6%	32.6%	35.5%	38.4%	41.2%	43.9%
	105%	21.3%	24.6%	27.7%	30.8%	33.7%	36.6%	39.5%	42.2%	45.0%
	100%	22.6%	25.8%	28.9%	31.9%	34.9%	37.7%	40.5%	43.3%	46.0%
	95%	23.8%	27.0%	30.1%	33.1%	36.0%	38.8%	41.6%	44.4%	47.1%
	90%	25.0%	28.2%	31.2%	34.2%	37.1%	39.9%	42.7%	45.4%	48.1%
	85%	26.3%	29.4%	32.4%	35.3%	38.2%	41.0%	43.8%	46.5%	49.1%
	80%	27.4%	30.5%	33.5%	36.4%	39.3%	42.1%	44.8%	47.5%	50.2%
	75%	28.6%	31.7%	34.7%	37.5%	40.4%	43.2%	45.9%	48.6%	51.2%

23 ADJACENT PROPERTIES

There are no material mineral properties adjacent to the project.

24 OTHER RELEVANT DATA AND INFORMATION

This section describes the proposed execution strategy, implementation plan and project schedule for the effective design, engineering, construction and commissioning of the Farim Phosphate Project and associated infrastructure.

24.1 Project Execution Strategy

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 will be engaged to provide EPCM services for the overall development of the project, including the process plant and the associated infrastructure. Specialist consultants will be contracted on an EPCM basis to address specific elements of the project outside the core competency of the engineering firm. These elements include mining, geotechnical, resettlement, environmental, marine construction of the Mineral Terminal, surface and sub-surface water management and the tailings storage facility. The specialist consultants will form an integrated project team under the overarching leadership of the engineering firm, who will be responsible for the overall project management and coordination between the various parties. Cost estimates assumed in the cash flow model are based on this approach.

24.1.1 Engineer and Specialist Consultants

The Itafos Farim Project has worked with experienced consulting companies since 2015. The project execution strategy, and underlying execution schedule, assumes the continued involvement of the core consulting companies to maintain continuity of Project knowledge. The core consulting companies are as follows:

- EPCM – Ausenco or Lycopodium Canada
- Marine Terminal – WF Baird and Associates
- Resource/Reserve statement, Mine design and Contract Mining RFP – WSP Golder
- Geotechnical, hydrogeology and in-pit dewatering, infrastructure, tailings design and surface water management – Knight Piésold
- Environmental Management, including Resettlement Action Plan – Knight Piésold (UK Office)
- Resettlement village design – Halyard
- Architecture of resettlement – ERM
- Hybrid power generation (build-own-operate-maintain model) – Aggreko.

24.2 Project Considerations

Guinea-Bissau has one of the least developed economies in the world. In the Doing Business 2020 report published by the World Bank, Guinea-Bissau ranked #174 from the 190 economies evaluated on ease of doing business, with 'getting electricity' ranked at #182. Due to the limited investments in infrastructure development and political instability, little progress has been made in expanding other sectors such as manufacturing and construction. The construction and civil engineering sectors are therefore mainly informal with only a small number of local companies present in the market, and with no professional umbrella organization. The large international construction companies have no local representation in Guinea-Bissau. This broader socio-economic context underpins the development of the project implementation plan.

Guinea-Bissau is part of the UEMOA (West African Economic and Monetary Union) which is west Africa's customs and currency union. The UEMOA works for economic integration and a common market among its members.

Part of these efforts involve drafting common Codes on key sectors of the economy, which are adopted into national law by members in an attempt to ensure a common investment framework. Amongst these sectors is natural resources; accordingly, Guinea-Bissau (and the other 7 UEMOA members) have adopted a common Mining Code as national law, providing legal stability across the region and including in Guinea-Bissau.

24.2.1 Construction Power

Although the Government of Guinea-Bissau is currently planning and constructing an electrical grid infrastructure system, the current Farim project design assumes that power supply to the plant site and the Ponta Chugue Mineral Terminal facilities will be from a dedicated onsite hybrid power plant with solar and diesel power. These hybrid plants are commonly used for isolated mine sites in the region.

As per Section 18.9.1, the Farim power plant comprises four 1.2 MW 11 kV generators, while the Marine Terminal plant comprises three 0.5MW 400V generators. Although the full system forms part of the long-lead items, staged delivery of the generators will be required, where one of each of the generators are delivered early and connected to construction-power minisubs from where construction power will be distributed. Power for the pre-mine pit dewatering system will be drawn from the temporary Farim power generator system.

For marine construction activities at the Marine Terminal, the floating marine plants will be self-sufficient and supply their own power via gensets.

24.2.2 Transport and Logistics Services

Equipment and materials will arrive at either the Port of Dakar (Senegal) or the port of Banjul (Gambia) and be transported to the construction areas. A quote was received from a transport and logistics services company for the handling of imports of materials, transport and freight of the key mechanical and electrical items. The Port of Bissau was not considered due to its occupancy during the cashew season, low draft, and major container carriers that have withdrawn service to the Port of Bissau.

Carriers have indicated their aversion for containers to cross the international border into Guinea-Bissau and it is anticipated in the estimate that container deposits will apply. The project will utilize dedicated ground agents to facilitate cargo delivery to Farim. Although not a requirement, a Customs Agreement will be drafted with the Government of Guinea-Bissau as part of the revised Mining Agreement. The purpose of the Customs Agreement is to facilitate the import of project-related materials, and to establish procedures for VAT and other tax exemptions.

For oversize cargoes, it will be necessary to mobilize fleet from outside Guinea-Bissau. Mobilization and demobilization costs were included in the estimate for this process. The availability of trucks compliant with Quality, Health, Safety & Environmental standards remains very limited, and the overall condition of the country fleet is poor. Further, supply of trucks is restricted during the cashew season (May to September), with most of the fleet deployed to cashew export traffic. The project transport campaign is planned to avoid this period and agreements will be set up with logistics companies from neighboring countries to truck containers to site.

24.2.3 Project Labor

Due to the informal construction and civil engineering sector in the country, skilled contractors from South Africa were approached and provided bids for their works. The bids included for most of the skilled labor to come from South Africa and live in temporary accommodations provided by the contractor. The respective supervisory and management teams will be accommodated in the existing project contractor’s camp.

Unskilled labor will be drawn from the local communities. Contractors will provide mandatory task training, while the project will provide mandatory safety training, as part of the induction process. The project schedule allows for extended mobilization and site establishment for the various contractors to accommodate the upskilling of local labor.

Although not a requirement, a Labor Agreement has been discussed with the Labor Ministry and will be implemented as best practice to establish harmonious labor relationships between workers and contractors. The purpose of the labor agreement is to establish labor rates, benefits and other guidelines with the unions and government so that all contractors compensate personnel appropriately and on a similar basis.

24.2.4 Consolidated Construction Contracts

Given the remoteness of the project, it is anticipated that contractors will have to be incentivized to provide competitive bids. The project therefore considers the consolidation of similar scopes of work into construction packages, including:

- The bulk earthworks contractor will also construct the North pit bypass road, fencing and the RAP village.
- The Year 0 to 5 mining contractor will also construct the flood protection bund, waste dump, tailings storage facility, and all surface water management infrastructure.
- The structural steel and mechanical erection contractor will also supply and erect all platework and piping (SMPP contract).

The additional advantage of consolidating construction contracts will be a reduction in the number of contractors on site, especially during the early phases of construction.

24.2.5 QA/QC Management

Due to the lack of skilled labor and fabrication facilities in Guinea-Bissau, equipment and materials have been priced to be imported from reputable fabricators outside of the country. Fabricators and equipment suppliers will have strict QA/QC plans with hold-points prior to the items being released to site. QA/QC management will be provided by the EPCM contractor, and EPCM pricing allows QA/QC inspectors to fly from North America for hold-point release. Furthermore, EPCM management will regularly visit the site to oversee construction quality management.

Supply contracts will be drafted so that the cost for re-inspection, as well as any QA/QC rectification, will be recovered from the supplier or fabricator. This will help maintain quality and reduce re-work due to poor materials and equipment arriving to site. Furthermore, consolidated SMPP contracts will be drafted such that the contractor only receives payment once all the material for a specific structure is on site. This places the onus for material handling squarely on the SMPP contractor.

24.2.6 Long-Lead Items

The aim during the design phase would be to prioritize the design and procurement of long-lead items, along with the contractor packages. This will give plenty of mobilization time and delivery time for the items and crews to arrive on site. The following major long-lead delivery items have been considered in the schedule:

- Radial shiploader – 11 months
- Diesel power plants– 12 months (excluding construction power). Note that generators will be staged during initial construction to mitigate impacts from lead-time delays.
- Rotary dryer – 10 months
- Vertical plant-and-frame filters – 11 months
- Belt conveyors and feeders – 10 months
- Drum scrubber – 10 months.

24.2.7 Access Control, Security Management and North Pit Bypass Road

To prevent unauthorized access to the mining and construction area, a security fence will be installed around the perimeter of the Mining Area, including both the South and North pits. Local villages to the west of the mine site will not have access to Farim once the mine site fence is installed and the existing village road that passes directly through the mine site is decommissioned. A North pit bypass road around the northern extent of the property must be constructed prior to the fence being closed in to replace the existing village access road.

In addition, to ensure a safe construction area and minimize losses to equipment of the plant and Marine Terminal sites, the plant, concentrate outloading and Marine Terminal sites will be enclosed with dedicated fences. Strict access control will be enforced to ensure that only trained authorized personnel can enter the construction areas. A security team will be contracted to monitor and survey the site.

24.2.8 Resettlement

The update of the Resettlement Action Plan (RAP) is linked to the overall project development schedule. Although the RAP work could begin early, it is linked to the investment decision to avoid community tensions and fatigue around the resettlement process, as well as avoiding the expiration of the RAP and asset data before the resettlement process can be implemented.

An integral part of updating the land and asset survey and the RAP is the restart of the stakeholder engagement process to get the affected communities back up to speed on the RAP process, while attempting to alleviate concerns related to project delays. The project will need to be sensitive to social response, and additional time has therefore been allowed for community engagement in the project schedule. Therefore, Phase 1 (Saliquenhe Porto & Ponto Zeca) and Phase 2 (Canico) of the RAP will have to be complete prior to pre-stripping commencing in the South pit and construction of project infrastructure.

24.3 Project Schedule

A high-level integrated project execution schedule has been prepared. The major project milestones are summarized in Table 24-1.

The overall schedule duration from the start of detailed engineering to the end of commissioning is 31 months. The ramp-up period will commence in Month 32, and the date of first commercial production is expected to be achieved within 6 months following commissioning.

Project schedules will be updated during the detailed engineering stage to mitigate risks associated with critical path activities. Specific areas that will be managed include the following:

- Detailed engineering associated with the pre-mining dewatering, the installation and equipping of the pre-mining dewatering wells, pumping the pre-mine wells for at least six months prior to the commencement of pre-stripping and completing the pre-stripping before the ramp-up period of the plant.
- Design, procurement, fabrication, delivery, and commissioning of construction power, which needs to be available for dewatering pumping to start.
- Updating the RAP and Biodiversity management plan, followed by detailed engineering associated with the RAP village, the mining bypass road, the fence, and the bulk earthworks for the plant. Phase 1 and Phase 2 of the relocation, as well as the construction of the bypass road and fence must be complete prior to the commencement of infrastructure construction by the mining contractor.

Detailed engineering, procurement, fabrication, delivery, installation and commissioning of long-lead items, including the concentrate dryer, horizontal scrubber and belt filters.

Table 24-1: Major Project Milestones

Major Milestone	Month
Start of Detailed Engineering for the Plant (Section 17), Concentrate Handling & Drying (Section 17) and Project Infrastructure (Sections 18.7, 18.8, 18.11)	1
Start Update to Biodiversity Management Plan and the Land and Asset Survey of the Relocation Action Plan (RAP) (see Section 20.2)	2
Start detailed engineering for Mining (Section 16)	3
Issue Procurement Packages for Long-Lead Sections, including Power Generation (Section 18.9), Concentrate Dryer (Section 17.4.1), Vertical Plate-and-Frame Filters (Section 17.3.3) and Horizontal Scrubber (Section 17.3.2)	4
Award Contract for Construction of Pre-Mining Pit Dewatering System (Section 16.4.3)	6
Award Combined Contract for PLANT BULK EARTHWORKS, NORTH PIT BYPASS ROAD (Section 18.2), Mine Fence (Section 18.2), Ponta Chugue Access Road (18.3.1) and Buredanfa Resettlement Village Construction (Section 20.2)	7
Issue Year 0 to Year 5 Contractor Mining RFQ (see Section 21.1.1.1) into the Market	7
Start Drilling of Pre-mine Dewatering Boreholes (Pre-mining Pit Dewatering System)	8
Start Detailed Engineering for Marine Terminal (Section 18.13)	8
Publish Updated Biodiversity Management Plan and Resettlement Action Plan (RAP)	9
Start Construction, including Plant Bulk Earthworks, North Pit Bypass Road, Fence around Mining Area, Ponta Chugue Access Road and Buredanfa Resettlement Village	10
Issue Procurement Packages for Marine Works Package (Section 18.13.4) and Shiploader Long-Lead Section (Section 18.13.4.11)	10
Award Year 0 to Year 5 Contractor Mining Contract	13
Commissioning Complete of Phase 1 (Construction Power) of Diesel-Hybrid Power Generation Systems	13
Complete Construction of Pre-mining Pit Dewatering System, incl. Piping and Overhead Line	13
Commence Pre-mine Pit Dewatering (Six Months Prior to Pre-stripping)	14
Complete Phase 1 Relocation (Saliquenhe Porto & Ponto Zeca), as per RAP	14
Start Process Plant, Outloading and Drying Concrete Works	15
Complete North Pit Bypass Road	16
Complete Mine Fence and Establish Access Control to Construction Site	16
Complete Phase 2 Relocation (Canico) as per RAP	16
Complete Year 0 to Year 5 Contractor Mining Contract Mobilization and Site Establishment	16
Mining Contractor to Start Construction of Infrastructure, including Flood Protection Bund, BD1, TSF1, WD1, ECD and the SCD1	17
Start Process Plant, Outloading and Drying SMPP Installation	17
Start Piling and Other Marine Works Construction	20
Start Mining Pre-strip	20
Start Process Plant, Outloading and Drying E&I INSTALLATION	21
Start Commissioning of Process Plant, Outloading and Drying	29
Start Commissioning of Mineral Terminal	31
Commissioning Complete	31
Start of Ramp-up Period	32

25 INTERPRETATION AND CONCLUSIONS

The QPs have provided the following interpretations and conclusions in their respective areas of expertise based on the review of data available for this study.

25.1 Hydrogeology

25.1.1 Interpretation and Conclusions

25.1.1.1 Hydrostratigraphic Units

The hydrogeological stratigraphy of the Farim area can be summarized from top to bottom as follows:

- Lodo clay near the River Cacheu ranging from a few meters to over 30 m near the river, and although it has not been tested, it is expected to have low permeability (10^{-9} to 10^{-8} m/s) pinching out with distance from the river. Where present, it acts as a low permeability barrier between the river and underlying sand aquifer.
- Overburden aquifer ranges in thickness from 28 to 48 m with an average of 35 m and is comprised of well-graded silt and sand units. The transmissivity ranges between 1.6 x 10^{-4} and 2.5 x 10^{-3} m^2/day and a storativity range of between 1 x 10^{-5} and 1 x 10^{-3} is reported. The sand units can be up to 25 m thick and are key dewatering targets with hydraulic conductivity in the range of 10^{-6} to 10^{-4} m/s based on most recent testing in 2017.
- An intermediate aquitard comprising the blue gray lignitic clay lies at the base of the overburden is present but is not laterally persistent.
- There is a phosphate ore aquifer comprising the FPO, FPA and FPB units which lie at the base of the overburden aquifer. They are described as poorly cemented sandy units ranging in thickness from 1 to 15 m and form part of the dewatering targets.
- At the bottom is the bedrock aquifer composed of calcareous to clayey sandy limestones that have not been drilled far into as they underlie the phosphate units. This layer has a poor rock mass rating; it does not show evidence of karstic weathering although the potential for karstic features in the bedrock remains under consideration. The bedrock aquifer generally has slightly higher transmissivity (ranging between 4 x 10^{-4} and 3.3 x 10^{-3} m^2/s) than the overburden aquifer (ranging between 3.1 x10^{-5} and 1.8 x 10^{-3} m^2/s), although more recent testing in 2016 of geotechnical drillholes indicates a hydraulic conductivity of 10^{-6} m/s.

25.1.1.2 Groundwater Levels

The groundwater elevations recorded since 2012 to 2017 in both the overburden and the underlying geology indicate that groundwater flows from the northwest (5 to 10 m below ground surface) where groundwater elevations are highest, towards the River Cacheu in the southeast (at 0 to 2 m below ground surface). Seasonal fluctuations range from 0.5 m in the North pit areas to 1.5 m, including tidal influence near the River Cacheu. Water levels near the South pit fluctuate 0.5 m semi-diurnally consistent with the tide.

The groundwater elevations recorded between August 2009 and February 2017 ranged between -1 and 4 1 meters above mean sea level (mamsl) in the lower bedrock aquifer and between -0.81 and 4.5 mamsl in the overburden aquifer. In the vicinity of the proposed TSF cells, the depth to groundwater is 2.8 to 13.6 m below ground surface based on the geotechnical drilling in 2017 and the water table in MW04 is 1 mamsl or 17 meters below ground level.

Seasonal monitoring of paired boreholes in the overburden and bedrock aquifers suggests vertical flow of groundwater changes seasonally, with downward gradients dominating during the wet season and upward gradients dominating in the dry season. More recent borehole measurements in 2018 and results from vibrating wire piezometers installed since 2016 indicate consistent downward vertical gradients in the South pit area during the wet season.

25.1.1.3 Groundwater Quality

Baseline water quality monitoring has been ongoing since 2012 and includes the boreholes installed in the South pit since 2016. The results indicate that the ion concentrations (measured as total dissolved solids (TDS) and electrical conductivity (EC)) generally increase at sample locations closer to the tidally influenced River Cacheu in both the overburden and bedrock aquifers. Water chemistry in the overburden aquifer near the River Cacheu changes from sodium-chloride (Na-Cl) type water during the dry season to Ca/Mg-HCO_3 during the wet season due to precipitation. The pH is generally neutral.

The field data collected indicates that the groundwater in the bedrock aquifer is slightly less acidic but more saline (higher electrical conductivity) than the groundwater in the shallow overburden aquifer due to the rainfall infiltrating the upper aquifer system above the denser saline water in the bedrock.

The groundwater quality is classified as "good" with only iron (total and dissolved) and manganese (dissolved) reporting above World Health Organization (WHO) guidelines. The groundwater has corrosive potential, especially for groundwater monitoring boreholes located near the South pit and the River Cacheu, due to higher chloride and sulfate ions.

25.1.2 Risks and Opportunities

The hydrogeological risks identified are as follows:

- The dewatering plan is based on stratigraphic interpretation from a limited database and the results of two pumping tests. The hydrogeological conditions are likely to be variable across the site, which could result in changes required to the dewatering design. Higher inflows and pumping costs could be anticipated if the Lodo Clay is discontinuous allowing for direct connection between the River Cacheu and the sandy overburden aquifer.
- The current location and design of the cells comprising the TSF and other infrastructure (overburden dumps) has not been included in the 2015 numerical groundwater model or taken into consideration in the South pit dewatering plan. The hydraulic head in the TSF will drive groundwater flow and result in additional ingress to the pits as well as potentially higher phreatic surface or pore pressures on the pit walls adjacent to the TSF. The assumption that the pit bottom is dry during mining may not be met, which could have an impact on the slope stability of the pit walls.
- Additional dewatering boreholes may be required to manage the artificial recharge from the TSF, overburden dumps and reduce ingress to the pits. Drilling and test pumping of groundwater boreholes is required in these areas and the data will be used as input to the numerical flow model to determine the additional number of dewatering boreholes and associated costs that may be required.
- The dewatering design (South pit) is feasible but there must be sufficient flexibility and adaptation to the plan once the Year 0 -Year1 boreholes have been drilled and the measured water level data compared to updated numerical model with refined geology.
- A hydraulic engineering assessment of the pipe reticulation has indicated that the surface pipes for the borehole dewatering system must be sized at larger diameter and additional discharge points are required to manage the pressures and flow velocities.
- Clogging, corrosion and encrustation of the boreholes is likely due to the corrosivity of the water that will be pumped, and maintenance and replacement costs must be included.

25.2 Mineral Processing and Metallurgical Testing

25.2.1 Interpretation and Conclusions

The following general conclusions were obtained:

- The process developed for the beneficiation of Farim phosphate ore is robust, continuous, and reliable, rendering reproducible metallurgical results.
- The combined concentrate assayed over 33.5% P_2O_5 with a mass yield of 77.5% and a P_2O_5 recovery of over 81%.
- The phosphate parameters were all within specifications with a CaO/P_2O_5 ratio of about 1.40; an MER of approximately 0.100; and an MER* of about 0.080.
- It was demonstrated that no flocculants were required for both the fine concentrate and tailings thickening. Since no reagents are used in the process it is likely that the concentrate could be certified as "organic".

The characterization studies, head chemical analysis, screen analyses, screen assays, and mineralogical QEMSCAN showed that the Farim composite prepared was representative of the South pit area of the deposit. This metallurgical work was done to develop the process flowsheet which eliminates flotation for the first seven years of mining of the Farim phosphate deposit. This beneficiation process was applied to the Farim North pit phosphate ore. The results indicated that the process was suitable for the North pit, but it requires further studies to determine the optimum operating conditions to obtain higher P_2O_5 grade and recovery. The metallurgical results and parameters obtained are outlined in Table 25-1.

Table 25-1: Attrition Scrubbing Metallurgical Results for Farim Ore

Parameter	South Pit	North Pit
Mass Yield	73.9%	74.3%
P_2O_5 Recovery	77.2%	76.8%
CaO/ P_2O_5 Ratio	1.5	1.4
MER	0.075	0.116
MER*	0.070	0.078
P_2O_5 Grade	33.8%	32.3%

The results of the pilot plant testwork confirmed KEMWorks' circuit design using horizontal and attrition scrubbing to remove the impurities from the ore and to achieve a concentrate product of 34% P_2O_5. The metallurgical results of the continuous pilot plant showed that they are reproducible and reliable. The results obtained in the best pilot plant test adjusted to the bench-scale operating conditions showed even better results than the bench-scale tests. The results were as follows:

- Mass Yield 76.2%
- P_2O_5 Recovery 80.3%
- CaO/ P_2O_5 Ratio 1.4
- MER 0.109
- MER* 0.078
- P_2O_5 Grade 33.7%

Thus, the continuous pilot plant tests showed that the most likely results in an industrial-scale plant that are possible to obtain were: A yield (mass recovery) of 77.5%, and P_2O_5 recoveries estimated between 81.4% and 84.3%, with the most likely P_2O_5 recovery being 81.8%. In the case of the P_2O_5 grade of the combined concentrate, the results were between 33.6% and 34.7%, with the most likely P_2O_5 grade being 33.6%. The most likely material balance and parameters were as follows:

- Mass Yield 77.5%
- P_2O_5 Recovery 81.8%
- CaO/ P_2O_5 Ratio 1.4
- MER 0.108
- MER* 0.078
- P_2O_5 Grade 33.6%

25.2.2 Risks and Opportunities

The beneficiation process developed for the South pit, which represents the first seven years of mining, was applied to the North pit. The results indicated that the process was suitable for the North pit, but it requires further studies to determine the optimum operating conditions to obtain higher P_2O_5 grade and recovery.

25.3 Mineral Resource and Mineral Reserves

25.3.1 Interpretation and Conclusions

25.3.1.1 Mineral Resources

The data provided through various exploration and sampling programs, combined with a detailed processing analysis, infrastructure and cost analysis, is sufficient to support the prefeasibility study and associated mineral reserves.

New drilling has been performed since the development of the current resource model. The QP has determined that the infill drilling data did not have a material impact on the current resource model; however, it is recommended that this new drilling on the property be used to update the next geological resource model.

25.3.1.2 Mineral Reserves

The mineral reserves outlined in the study are based on a targeted mine life of 25 years at a rate of 1.75 Mt/a (dry basis). Additional measured and inferred mineral resources have been delineated on the property, which have the potential to add substantial additional mineral reserves. For this study, equipment was assumed to be 12 m³ front-end loaders (FEL) matched with 97 t capacity haul trucks for the overburden, and 5 m³ bucket class backhoes matched with 36 t capacity trucks for the matrix mining. These equipment sizes were selected 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 m³ 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. Water inflow will be of paramount concern during all stages of mining. Precautions include dewatering in advance of mining, reduction of production rates during rainy season, in-pit sumps and pumping, as well as construction of both permanent and temporary flood bunds.

Key modifying factors used to convert mineral resources to mineral reserves include: 100 mm roof mining loss, 75 mm floor dilution gain, minimum mineable thickness of 1 m, expected mass recovery of 74.3% for the North pit and 77.5% for the South pit, and product grade of 34% P_2O_5.

The mining sequence was developed to achieve production targets and to defer mining of the area with potential acid-generating (PAG) material and the areas adjacent to the River Cacheu until sufficient neutralizing material could be stripped to mitigate the PAG overburden. The yearly strip ratio remains under 10 bcm / ROM tonne as mining progresses through the South pit and then increases in Year 7 as mining transitions to the higher strip ratio North pit.

The waste dumps at Farim include the short-haul ex-pit dumps WD-1 and WD-2 located between the North pit and South pit extents and the surcharge overburden storage (SOS) facilities located within the South pit and North pit extents The WD-1 is designed to handle up to 0.9 M LCM of PAG material in a specifically designated lined area. WD-2 ex-pit dumps can only handle NAG material. All other potentially leachable material must be dumped to IOB and covered with sufficient NPAG waste.

The total direct mine operating cost, including preproduction, is $1,048 million or $31.86 per tonne of product. Annual costs range from $21.5 million in preproduction Year 0 to over $48 million in Years 2 and 23. When the estimated preproduction costs of $21.5 million and mining fuel cost of $365.3 million are excluded from the direct operating costs, the total becomes $661.4 million or $20.1 per tonne of product.

25.3.2 Risks and Opportunities

Risks include modifying factors which were used to convert mineral resources to mineral reserves being significantly different from factors used in the estimate. Additional risks include the following:

- significantly different commodity market conditions and prices in the future
- significant increases in operating costs
- extraction conditions during rainy seasons proving more adverse than anticipated
- significant reduction in equipment and labor productivity from what has been estimated in the reserve analysis.

25.4 Recovery Methods

25.4.1 Interpretation and Conclusions

The beneficiation plant design is based on bench-scale and pilot plant testing for optimum recovery and minimum operating costs while meeting concentrate grade requirements. The flowsheet is based upon unit operations that are proven within the industry. The beneficiation plant utilizes physical separation processes to reject specific size fractions and produce a filtered bulk concentrate that achieves product specification targets. The beneficiation plant is designed to produce 1.36 Mt/a of concentrate from the South pit and 1.30 Mt/a from the North pit.

25.4.2 Risks and Opportunities

The following risks were identified for the project, with respect to recovery methods:

- The North pit ore has limited testwork relative to the South pit ore and has limited variability testing done to date.

The following opportunities were identified through the development of the process design:

- Further tailings thickening and rheology testwork should be conducted to optimize the thickener underflow density and confirm the performance of tailings pumping to the management facility.

- The completion of further concentrate dewatering testwork could present benefits in the design of concentrate handling equipment. It would allow the moisture content requirements to be optimized in combination with the required transportation moisture limits. Evaluation of alternative drying technologies could also reduce diesel consumption rates in relation to the selected concentrate filtration design.
- The location of the concentrate dryer should be evaluated to determine if cost savings could be realized by reducing the mass trucked to the Mineral Terminal, consolidating the fuel storage and distribution infrastructure, and simplifying the Mineral Terminal operations.
- Investigate alternative drying technologies to determine if diesel consumption can be decreased while achieving discharge moisture targets; the evaluation should be performed in conjunction with a filter press design trade-off to optimize the feed moisture to the dryer in relation to the capital and operating costs.

25.5 Site Infrastructure

25.5.1 Interpretation and Conclusions

25.5.1.1 TSF, Return Water Pond and Waste Dumps

KP established the scope and quantities for the seven-cell TSF, return water pond (RWP), and waste dump (WD) foundation treatment. KP developed 3D models to estimate earthworks and material costs such as piping and HDPE geomembrane liner quantities based on West African contractor rates provided in 2020. The 2020 rates were reviewed and adjusted to reflect 2022 assumed local conditions; however, the adjusted rates were not verified through new quotes. The TSF will be constructed in stages with cell 1 only being constructed in Year 0. Embankment fill for each TSF cell will be sourced from waste overburden delivered directly to the embankment footprint. Haul and overhaul costs are included within the mining budget. The RWP will be constructed in two phases: Phase 1 will be constructed in Year 0, while Phase 2 will be constructed in Year 1 to reduce initial capital costs.

Based on the work to date, there have been no geotechnical or material supply or handling issues to prevent Itafos from developing the project.

25.5.1.2 Surface Water Management infrastructure

KP established the scope and quantities for the surface water management infrastructure and pit dewatering. KP developed 3D models to estimate earthworks and material costs based on West African contractor rates provided in 2020 as well as material and equipment costs from recent project experience. The 2020 contractor rates were reviewed and adjusted to reflect 2022 assumed local conditions; however, the adjusted rates were not verified through new quotes or by contacting local suppliers.

25.5.2 Risks and Opportunities

25.5.2.1 TSF, Return Water Pond and Waste Dumps

The following risks were identified for the project, with respect to tailings storage, out of pit waste overburden NAG and PAG storage and TSF reclaim water management:

- Changes to the existing fenced boundary need to be reflected in an updated RAP (Section 26.7).
- Changes to tailings delivery design with the completion of rheology testwork on tailings.
- Changes to the tailings settled dry density could increase construction requirements in terms of additional land and embankment fill materials and make placement of closure cover more challenging.

- Changes in the tailings settling characteristics impacts water released from tailings resulting in excess water stored in the active cell.
- Changes in tailings geochemistry in terms of potential ARD, metal leaching and radionuclide leeching. Current testing indicates low potential for Acid Rock Drainage; however, metal leaching of cadmium and nickel are above target receiving water quality levels and there is radionuclide leaching potential from exposed tailings. An engineered containment and capping system is required with progressive closure to be adopted to reduce exposure once each cell reaches capacity.
- Construction timing for the individual TSF cells is impacted by the rainy season and construction must be brought forward.
- Incomplete soil and rock testwork related to runoff coefficients directly impacts water balance assumptions and site operates as increased deficit or increased surplus with excess water for treatment.
- Climate conditions specific to site differ from data source used to establish water balance and extreme rainfall events causing under design of water management infrastructure and additional costs or leading to significant changes to the water balance results and sizing of storm water management infrastructure.
- Characteristics of soil layers under the plant site, truck loadout facility, WD and TSF may result in foundation instability without a campaign that targets specific plant equipment and embankment placement.
- Changes to the TSF embankment stability with the completion of additional geotechnical investigations as foundation and pore pressure conditions are not well understood with the available data not targeting specific footprints.
- Cyclic and static liquefaction potential of the foundation soils should be assessed for undrained conditions on critical areas through the updated TSF, flood bund and pit overburden waste dump locations. The investigation and findings could impact the geotechnical design of various foundations.
- An offsite rock source has not been identified, costs for supply of crushed rock for roads, embankment drains and erosion protection could increase significantly.
- Unlined TSF basin cells may introduce high hydraulic gradients for seepage between these dams and the pit shells that could introduce instability.

The following opportunities were identified through the development of the tailings storage, out of pit waste overburden NAG and PAG storage and TSF reclaim water management:

- Tailings settling testwork suggests tailings settled dry density increases with an increase to the tailings slurry % solids by mass. An increase in settled dry density can provide additional storage capacity, or a reduced cell development and or crest elevation.
- Research on phosphate tailings may identify available options for achieving higher settled dry densities. A cost comparison between cell construction and options may allow for a reduced TSF footprint.
- Follow up geotechnical site investigation based on the latest TSF embankment footprint focusing on key soil conditions provides opportunities to reduce embankment construction volumes requirements.

25.5.2.2 Surface Water Management infrastructure

The following risks were identified for the project, with respect to surface water management infrastructure:

- Unlined sediment control dams may introduce high hydraulic gradients for seepage between these dams and the pit shells that could introduce instability.

- Changes from a water classification of Contact Clean to Contact Dirty for runoff water managed within active pit areas would require that such water be treated prior to release to the River Cacheu.
- Climate conditions specific to site differ from data source used to establish water balance and extreme rainfall events causing under design of water management infrastructure and additional costs or leading to significant changes to the water balance results and sizing of storm water management infrastructure.
- Incomplete soil and rock testwork related to runoff coefficients directly impacts water balance assumptions and site operates as increased deficit or increased surplus with excess water for treatment.

The following opportunities were identified through the development of the surface water management infrastructure:

- The Rio de Cavaras Marinhos may be diverted to a watercourse on the western side of the site, simplifying the surface water management approach and reducing both CAPEX and OPEX requirements. The alternate diversion alignment would need to be included as part of an updated ESIA.

25.6 Electrical Infrastructure

25.6.2 Interpretation and Conclusions

The current base case for electrical supply is through hybrid generator stations at the mine site and Mineral Terminal as described in Chapter 18. The government of Guinea-Bissau is constructing an electrical power grid within the country including transmission lines, sub-stations, and distribution networks. The government will purchase power from hydroelectric power stations located in the neighboring nations of Senegal and Guinea and sell it to users. Although this electrical power grid construction is well-advanced, the Farim project currently assumes that all power will be generated through hybrid power plants.

25.6.3 Risks and Opportunities

The opportunity exists to potentially utilize grid power at the Marine Terminal or mine site. Based on discussions with local government officials, it is unlikely that the electrical grid will have sufficient capacity to support electrical needs at the processing plant. However, the grid design has considered the Farim project and could be utilized at selected locations including the housing camp, offices, security, auxiliary lighting, truck-loading facilities, service wells, selected pumping, and miscellaneous power. During detailed design, this opportunity should be assessed based on the status of the government’s electrical network, costs, reliability, and available power capacity.

25.7 Marine

25.7.1 Interpretation and Conclusions

Given the marine studies and design completed to date, the proposed vessel loading facility at Ponta Chugue is feasible.

Metocean conditions for the project have been defined to a suitable level of detail based on high resolution modeling of water levels, currents, and waves.

The foundation design for marine structures was based on geotechnical information collected in 2017 generally along the alignment of the trestle and wharf. As such, the foundation design is based on suitable assumptions.

The channel design has been assessed against PIANC channel design guidelines and with desktop and real-time navigation simulations. The channel alignment, including through the Bernafel section, is suitable for the water depths, design depths and prevailing currents. The navigation fairway surrounding the Ponte Chugue Marine Terminal is suitable and provides a generous maneuvering area for inbound and departing vessels.

There are depth constraints at two locations along the channel: (1) at Ponta Bernafel, approximately 28 km downriver from the terminal; and (2) offshore at the Bijagos Breaker area, approximately 120 km to 170 km from the terminal. Deeper draft vessels arriving at the terminal will not be able to load to capacity and will have to navigate these shallow areas at or near high tide (i.e., "tidal assistance" is assumed). An operational UKC system will be required to plan departures from the Marine Terminal, and to monitor the available UKC and sailing window during a departure.

An assessment of estuarine morphology and its potential impact on the marine operations of the Ponte Chugue Marine Terminal have been completed using a combination of approaches including analysis of survey data, qualitative numerical modeling and desktop analyses including application of empirical and analytical models. The sediment transport processes in the Geba River estuary are complex and the estuary is subject to large sediment loads and re-working of existing sediment deposits. 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.

25.7.2 Risks and Opportunities

The following marine risks were identified:

- 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.
- The vessels will navigate a 180 km long channel in the Geba River from the Atlantic Ocean to the Ponta Chugue terminal. There are depth constraints at two locations along the channel: (1) at Ponta Bernafel, approximately 28 km downriver from the terminal; and (2) offshore at the Bijagos Breaker area, approximately 120 km to 170 km from the terminal. As a result, the deeper draft vessels arriving at the terminal will not be able to load to capacity and will have to navigate these shallow areas at or near high tide (i.e., "tidal assistance" is assumed).
- The design of the vessel loading facility and aids to navigation were developed with a goal of minimizing capital cost, which has impacts on the required sustaining capital for the facility and requires that stringent safety protocols be followed during vessel loading operations.
- The use of a single shiploader requires the vessel to warp at the terminal. Warping requires tugs to come alongside the vessel and hold it on berth while the lines are slackened, and the vessel moves itself ahead or astern on the berth utilizing its main engine. This operation is high risk and complex for a vessel's crew due to the need to operate mooring lines in a dynamic environment. Warping also increases the time needed to load vessels and greatly increases the chances of an accident (allision, collision, or vessel breakaway) when compared to a terminal that does not require warping.
- During the feasibility study, Itafos selected a minimum capital cost solution for maintenance that included a radial telescoping shiploader and did not include vehicular access along the trestle to facilitate maintenance of the conveyor, associated utilities, berthing and mooring infrastructure, or the shiploader. The scheme instead relied upon a maintenance barge that would be purchased and permanently stored at Ponta Chugue for the purposes of generally maintaining the marine infrastructure (not including major structural repairs).
- In the event of an extreme squall where sustained winds could exceed 20 m/s, it is likely that numerous mooring lines would part on Supramax and Handysize ships. It is recommended that operational procedures be developed for the terminal to address squalls that may include ship completing an emergency deberthing and getting clear of the marine terminal before anchoring to 'ride-out' the squall event which may persist between 10 and 60 minutes.

The following marine opportunities were identified:

- Transshipping Opportunity
  - The project base case of shipping concentrate through the Mineral Terminal at Ponte Chugue is feasible as currently designed. However, there is an opportunity to change the project design to transship concentrate using barges at either Ponte Chugue (PC) or at the River Cacheu (RC). At RC, barges would be loaded near the process plant therefore eliminating the need for a conveyor over RC. Transshipping conceptual design and costing work was done in 2015. Although determined to be feasible, transshipping was not used as the base case for this feasibility study. This design and cost estimate should be revisited prior to a construction decision to reassess its potential benefits. Key elements to consider regarding the costs and benefits of transshipping include:
- Ponte Chugue Transshipping
  - It is uncertain that transshipping at PC will lead to significantly cheaper capital costs, as there would still need to be a barge loading terminal which is similar to the OGV terminal.
  - The 2015 transshipping marine terminal capital cost was 15% lower than the 2015 direct load marine terminal. This is a comparison of the marine structures, excluding material handling, ATONs, and service vessels. Since 2015, the design ocean going vessel has increased in size thus the CAPEX savings should be expected to increase when compared to the current marine terminal design.
  - Including service vessels, the 2015 transshipping marine terminal capital cost was 13% higher than the 2015 direct load marine terminal. The additional cost of purchasing barges reduces any potential capital cost savings.
  - The initial assessment assumed 6 barges at 2,700 tons per barge for PC, as at the time there was a strong desire to use standard vessels from Europe (Europa IIa). Barge marshalling was relatively easily incorporated into the barge loading terminal at Ponte Chugue as it was conceptually designed for this.
- River Cacheu Transshipping
  - Due to the separation between the barge loading terminal at Binta and the transshipping area in the mouth of the RC, a standalone marshalling area was required for the RC option. Much like the PC option, the design considered 2,700-ton Europa II barges (estimated 20 to 25 barges).
  - Since the RC option would not need the river crossing conveyor, truck-loading facilities on the east side of the river and infrastructure at Ponte Chugue will lead to capital cost savings. However, the drying facility and concentrate storage area will have to be moved to the processing plant site, which is already congested. The larger construction area might necessitate the expansion of the flood protection berm to the north.
  - Transshipping eliminates the need for a fleet of trucks to transport dry concentrate from RC to the Mineral Terminal. However, this will be offset by a fleet of barges and tugs to run the operation. Using barges and loading at Binta (where water depth exists) would likely decrease operating costs due to economies of scale with barges (efficiencies relating to labor and fuel).
  - Transshipping at RC eliminates haul road bypass construction to complete the haul road to the Mineral Terminal resulting in lower capital cost. Also, eliminating concentrate haulage along a shared road would have social benefits for local communities and residence.
  - River Cacheu is used by fishermen on a regular basis and also provides a transportation route for local residence. Barge operations would impact this activity therefore social aspects must be well understood.
  - A typical rate for a transshipping is between 9,000 and 11,000 tons per day (assuming ships gear, calm weather and one barge on each side of the OGV). However, the waves are not calm offshore, so the conceptual design

estimated a daily rate between 3,500 to 5,000 tons per day. This was somewhat due to the selection of Europa IIa barges which do not handle waves well. A better barge would improve loading rate and reliability. Note. this is only an issue at RC.

- Items Common to PC and RC
  - Regarding demurrage, the current plan is loading via shiploader at 1,200 tonnes per hour. However, it is unlikely that rate would be achievable with the ship gear. By implication, it will take significantly longer to load a vessel therefore adding demerge charges. Note demurrage is the cost of time over anticipated time to load. While demurrage for a slow loading rate will likely be greater due to the greater probability of encountering downtime events like storms and maintenance outages, it should not be significantly longer if the shipping contract correctly incorporate expected ship turnaround time.
  - By not constructing the OGV terminal (both PC and RC), it eliminates the plan to import fuel. Diesel fuel supply will have to be addressed with this option and could include local sources in-country or importing from Senegal. Another possibility is to design fuel delivery through PC or RC but that would require a very specialty barge.

Reassessing transshipping options must consider all aspects listed above Furthermore; local leaders should be involved during the social impact assessment. Even if transshipping is more expensive, practical and social benefits may justify a design change.

25.8 Geotechnical

25.8.1 Interpretation & Conclusions

The mine site lies 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 over-consolidated deposits at greater distance from the rivers.

Foundation parameters have been selected based on site investigations along the boundary for the South pit flood bund, Tailings storage facility, waste dumps, process plant and truck load out facility. Selected sedimentary soils within the project limits have been identified to be suitable for low permeability applications and for general fill. An onsite source for drainage sand/gravel, road base and concrete aggregate is not available, and the project will rely on offsite sources to support the mine. The nearest known hard rock quarry is approximately 150 km from the site. For the next phase of the project, an agreement should be in place for the supply of various aggregates as required by the project to confirm local suppliers can meet demand and to confirm costs.

25.8.2 Risks and Opportunities

25.8.2.1 Risks

- Characteristics of soil layers under the plant site, truck loadout facility, WD and TSF may result in foundation instability without a campaign that targets specific plant equipment and embankment placement.
- Changes to the TSF embankment stability with the completion of additional geotechnical investigations as foundation and pore pressure conditions are not well understood with the available data not targeting specific footprints.
- Cyclic and static liquefaction potential of the foundation soils should be assessed for undrained conditions on critical areas through the updated TSF, flood bund and pit overburden waste dump locations. The investigation and findings could impact the geotechnical design of various foundations.

- An offsite rock source has not been approached for specific costs for the supply of crushed rock for roads, embankment drains, and erosion protection could increase significantly.
- Unlined sediment control dams may introduce high hydraulic gradients for seepage between these dams and the pit shells that could introduce instability.
- The geotechnical characterization of the subsurface data for the South pit was based on relatively large-spaced boreholes and CPT soundings mainly concentrated along the east and south walls of the South pit. Variability in the subsurface conditions could result in geotechnical conditions different than what was estimated for this study, which could lead to instability. As the slopes of clayey units with estimated weakest shear strengths are exposed, they may begin to deteriorate and may undergo strength loss due to exposure, opening and softening of fissures, or degradation of the organic materials.

25.8.2.2 Opportunities

- Follow-up geotechnical site investigation based on the latest TSF embankment footprint focusing on key soil conditions provides opportunities to reduce embankment construction volumes requirements.
- Steeper inter-ramp design slope angles in Years 4 through 7 at the south and west walls of the South pit may be feasible if geological conditions are more favorable than the assumptions documented in this report. Additional geotechnical data and lessons learned from mining in Years 1 and 2 can also help with his decision.

25.9 Environmental, Permitting and Social Considerations

25.9.1 Interpretation and Conclusions

ESIAs for the project and the Buredanfa Resettlement Village were approved by the Government of Guinea-Bissau, according to a Declaração de Conformidade Ambiental (Declaration of Environmental Compliance) issued to Itafos on September 14, 2018 (Secretaria de Estado do Ambiente, 2018). While the approval (declaration) expired in 2019, the Autoridade da Avaliação Ambiental Competente (Competent Environmental Assessment Authority) notified Itafos in March 2020 that the Authority had almost completely suspended its operations, and that the process of renewal of the environmental license will resume when the pandemic is over (Autoridade da Avaliação Ambiental Competente, 2020). To date, no further notification from the Autoridade da Avaliação Ambiental Competente has been received indicating the resumption of operations and the envisaged renewal of the environmental license.

The project is expected to result in adverse environmental, socioeconomic and cultural heritage impacts that can be reduced to acceptable levels, through the implementation of mitigation measures. The most material impacts identified to date include the following:

- potential effects to groundwater and surface water
- pit dewatering affecting household and community wells
- ecological impacts
- community health, safety and security
- radiological exposure to workers
- cultural heritage impacts
- employment and training opportunities to Guinea-Bissau nationals
- influx and associated social disruption and ecological impacts
- involuntary resettlement.

The above impacts are described in more detail in Section 20.2, Based on the work to date, there have been no environmental or social issues that are expected to prevent Itafos from developing the project.

Because biodiversity and social conditions may have changed since the 2015 ESIA and 2018 Resettlement Action Plan were completed, additional work in these areas is recommended ahead of construction, as described in Section 26.

25.9.2 Risks and Opportunities

The following risks were identified regarding the environmental, social and community aspects of the project:

- Conditions related to resettlement may have changed – It is possible that communities have grown and/or people have moved into the area anticipating a resettlement process, which could increase land acquisition and resettlement costs relative to the 2018 RAP and associated costs.
- Biodiversity conditions may have changed – The conservative status of wildlife may have changed, requiring additional measures to address or offset Red List species.
- Relocation of significant cultural heritage features takes longer and more money – There is a risk that development of the required cultural heritage plans (grave relocation plan, sacred site relocation plan, and mosque relocation plan) takes more time and money than expected due to difficulties establishing consensus with local communities.
- Political instability – Ongoing political instability and a lack of continuity of the highest positions in the government that has been experienced in the past could present difficulties in operating the mine. Since its independence in 1974, the country has faced many coups d'état along with many additional coup attempts, the highest number in the world; a very high rotation of prime-ministers, governments, ministries, deputies, directorate generals, etc., often in charge not even for one year (Landesz, 2019). This political instability does not provide the necessary conditions to rule the country in a way that policies are well design and implemented, natural resources are sustainably planned and supervised, basic public services are provided to people throughout the country. Corruption also undermines the availability of public resources for the implementation of public policies and providing basic public services to the Bissau-Guineans.
- Geochemical uncertainty – There is a risk that the quantity of PAG waste overburden requiring surface disposal is higher than estimated, and higher costs will be incurred to manage this waste.
- TSF closure – The low-density tailings do not consolidate appreciably in each of the closed cells, and placement of the 1.5 m closure cover proves to be more challenging and costly.

25.10 Economic Analysis

25.10.1 Interpretation and Conclusions

The feasibility study 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.

25.10.2 Risks and Opportunities

Costs have been estimated to a level of accuracy suitable for a feasibility study. Overall economic risks include financing, price escalation, inflation, commodity sales price variability, and general global economic conditions. Sensitivity analysis shows the impact of key economic drivers for various changes. The economic analysis has been undertaken only from the point of detailed design; costs prior to that time are excluded.

26 RECOMMENDATIONS

26.1 Overall

The financial analysis of this feasibility study demonstrates that the Farim Phosphate Project has robust economics. It is recommended to continue developing the project through engineering and de-risking, towards a construction decision.

Analysis of the results and findings from each area of investigation completed as part of this feasibility study suggests numerous recommendations for further investigations to mitigate risks and/or improve the base case designs. Costs associated with future recommendations are included within the detailed design initial capital costs or operating costs.

The following sections summarize the key recommendations arising from this FS. Each recommendation is not contingent to a subsequent one. Table 26-1 presents a summary of recommended work and associated costs for the next project phase. Other recommendations will be important but should be addressed during the detailed design phase.

Table 26-1: Recommended Work and Budget for Next Phase

Area	Estimated Cost (US$)
Mineral Resources and Mineral Reserves	55,000
Mineral Processing & Metallurgical Testing	150,000
Recovery Methods – Tailings Thickening Testing	10,000
Marine – Evaluation of Transshipping Option	50,000
Tailings Characterization and Settling Testwork	35,000
Updated South Pit Ground Investigation for Pit Dewatering (includes Drilling, CPT Program, Vibrating Wire Piezometers Supply and Install, and Supervision)	625,000
Ongoing community engagement including renewal of its Declaration of Environmental Compliance from the Competent Environmental Assessment Authority to ensure continuity of project approvals	N/A
Total	925,000

26.2 Mineral Resources and Mineral Reserves

Work programs related to resources and reserves for the next phase include the following:

- Confirm that the dry density values used in Section 14 and Section 15.6, are representative for future resource and reserve estimations. Additional density measurements should be taken to verify these values. The QP anticipates that this would cost approximately US$5,000.
- As noted in Section 16.7.4.6.1, a lack of samples in Area 4 of the pit (as designated by Figure 16-7) has prevented a thorough evaluation of the liquefaction susceptibility in this Area. Samples in Area 4 should be collected and screened prior to excavation to evaluate the soil's liquefaction susceptibility. The QP anticipates that this would cost approximately US$20,000.
- As stated in Section 16.7.5.2, an important component of the slope development will be to monitor the degree of pore pressure reduction that has been achieved in the bench face that is being excavated. This can be achieved by installing piezometers or pushed probes with pressure transducers into critical areas along the pit slopes.

Supplemental pumping wells or horizontal drains will be needed where isolated pressurized zones are encountered. Further studies should be done to advise the precise locations of these piezometers for optimized performance. The QP anticipates that this would cost approximately US$30,000.

Additional recommendations to de-risk the project are listed below. Costs for these recommendations are included within capital or operating costs.

- The results from the post-2015 drilling, although determined to not have a material impact on the current model, should be used to update the geologic resource model in advance of short-term mine planning for any preproduction or production activities.
- Update the contractor cost estimates through the development of a new contract miner bid and include mining functions that were not included in the 2019 bid (note: excluded mining functions were estimated and included in this technical report cost estimate).
- Update capital cost estimates through quotes from equipment vendors in the project area.
- In the North pit, geotechnical field testing should be completed to develop a detailed geotechnical design compatible with the work performed in the South pit prior to opening the North pit area adjacent to waste dump 2 (WD-2).

26.3 Mineral Processing and Metallurgical Testing

Mineral processing and metallurgical testing work programs for the next phase include:

- Conduct continuous phosphoric acid plant tests to assess likely performance in an industrial plant. Conduct bench-scale phosphoric acid concentration and clarification tests, and bench scale fertilizer testwork for both the South pit and North pit concentrates of the Farim phosphate deposit especially if flotation is required. The QP anticipates that this would cost approximately US$150,000. Results from this testwork will be used in product off-take negotiations and are independent of the investment decision therefore do not have to be complete prior to detailed design.

Additional recommendations to de-risk the project relating to metallurgical testwork and mineral processing are listed below. Costs associated with these recommendations are included in the capital or operating costs.

- Perform further variability testwork on the North pit ores to confirm process performance.
- Additional drilling in the North pit of the Farim deposit is recommended according to mine planning to prepare selected-representative samples for metallurgical studies.
- Characterization studies for the North pit phosphate ore on representative samples including physical, chemical, and mineralogical and QEMSCAN studies.
- Evaluation of different solids content for the optimization of horizontal scrubbing for the North pit phosphate ore since the retention time will depend on the solids content, with the horizontal scrubber design being already defined. Confirmation tests should be considered.
- Carry out attrition scrubbing studies for the North pit phosphate ore to reject more Al_2O_3 and Fe_2O_3 resulting in $C_{organic}$ and $S_{pyritic}$ (pyrite) being rejected since both $C_{organic}$ and $S_{pyritic}$ (pyrite) are tied to alumina and iron-bearing minerals. In addition, some SiO_2 and silicates may also be rejected. This may upgrade the coarse concentrate (1180 x 106 µm), and mainly the fine concentrate (106 x 20 µm).
- Flowability studies for the North pit phosphate ore and concentrates should be evaluated for bins, and transfer chutes design.
- Screen out the 420 x 212 µm size fraction of the North pit phosphate ore to be floated if required since it is about only 18% of the feed to the beneficiation plant, reducing the overall size of the flotation circuit.

- Flotation of the 420 x 212 µm size fraction of the North pit phosphate ore requires that the concentrate obtained should be joined to the 1180 x 420 µm and the 212 x 106 µm size fractions to reconstruct the coarse concentrate. Amine selection should be considered for coarse material.
- Flotation, if required, could be applied to the coarse concentrate (1180 x 106 µm size fraction) If the same flowsheet developed for the South pit is to be maintained for the North pit. This may result in upgrading the North pit coarse concentrate to 35% P_2O_5, the combined concentrate resulting in 34% P_2O_5 and lower $C_{organic}$.
- Optimization of the selected flotation process (if required) should be carried out, which includes the size fraction submitted to flotation, the flotation conditions (conditioning, flotation time, and pH), and amine flotation reagent (type and dosage).
- Flotation feed size separation equipment system, reagents yard, storage facilities, reagents feeding systems, and safety considerations should be included in the project layout and design if flotation is required.
- Rheology of the flotation tailings should be studied, dewatering tests must be carried out, and the flotation TSF needs to be designed if flotation is required.
- Update and evaluate the rheological characteristics of the products and tailings to determine the slurry and pumping characteristics, static and dynamic settling and filtration characteristics for the South pit and North pit of the Farim phosphate deposit.
- Evaluate the settling and filtration parameters in the absence and presence of coagulants and/or flocculants for the design of the thickeners and filtration devices for products and tailings using samples of both South pit and North pit.
- Implement a metallurgical testwork program to include:
  - vacuum belt filter dewatering and optimization for coarse concentrate
  - pressure filter dewatering and optimization for fine concentrate
  - bulk material handling flowability tests for product bin design
  - dry optimization tests.
- Perform variability bench scale tests for different areas of the South pit and the North pit of the deposit applying the beneficiation technology developed.
- Characterization studies for the North pit phosphate ore including physical, chemical, and mineralogical and QEMSCAN studies.
- Carry out additional pilot plant tests for each South pit and North pit phosphate ores to obtain enough information on the material balances, operating conditions, variability effects, products and their marketing; and to evaluate the use of column flotation cells for 106 x 20 µm size fraction when high iron-bearing minerals are present, and advanced coarse flotation cells for the 1180 x 106 µm, and 420 x 212 µm size fraction for the North pit.

26.4 Recovery Methods

The following work is recommended to advance to the next stage:

- Further evaluate tailings thickening and dewatering to maximize achievable underflow density and optimize thickener sizing. The QP anticipates that this would cost approximately US$10,000.

Further recommendations to de-risk the project or enhance project economics are listed below. Costs associated with these recommendations are included in capital or operating costs.

- Investigate alternative drying technologies to determine if diesel consumption can be decreased while achieving discharge moisture targets; the evaluation should be performed in conjunction with a filter press design trade-off to optimize the feed moisture to the dryer in relation to the capital and operating costs.
- Perform a trade-off study to evaluate the dryer location between the beneficiation plant versus the Mineral Terminal site.
- Evaluate alternative fuel sources for the power plant and rotary dryer to determine if lower unit costs could be realized.

26.5 Marine

The marine field data acquisition activities and engineering studies have largely concluded. To advance to the next stage, the following marine work program is recommended.

- Update the transshipping trade-off study to evaluate barge loading rather than hauling filtered concentrate to the planned Mineral Terminal at Ponte Chugue. This includes updating the costs from the previously performed work, re-evaluating barge, vessel requirements and throughput, updating the social impacts, and overall project benefits. This trade-off update is estimated to cost US$50,000.

Recommendations for future work to de-risk the project include the following:

- Reengage with marine contractors to complete the detailed design of the marine terminal.
- Develop a logistics plan to define the strategy and execution principles that will direct the project logistics team in undertaking the logistics program for the Mineral Terminal. Outline transport modes, routes and capabilities of transport in Guinea-Bissau. Outline hazardous material transportation, supply chain security, local policies, and insurance and claims.
- Develop a construction management plan that defines the roles, responsibilities, and objectives of the construction management team. The plan should also define construction scope of the Mineral Terminal facilities and outline subcontracting strategies, industrial relations, and local human resource management.
- Develop a commissioning and handover plan that describes the commissioning scope, responsibilities, processes, and sequence of testing the project.
- Develop a marine operational readiness plan that details the necessary training required for vessel operators, logistics channels for sourcing spare parts, international ship and Mineral Terminal facility security code (ISPS) requirements, safety procedures, equipment and personnel required to maintain the marine facility.
- Develop an asset management strategy plan to summarize the methodology, assumptions, and key results of maintenance strategy defined for the Mineral Terminal assets. The plan should also define asset management principles, the project maintenance strategy, and maintenance KPIs.

26.6 Site Infrastructure

The feasibility study has outlined many other infrastructure recommendations to de-risk the project or enhance project economics. These recommendations will be addressed during the detailed design stage or during operations and are described below by main area including tailings storage facility, dewatering, and geotechnical.

To advance to the next stage, the following tailing storage facility work program is recommended:

- Complete additional tailings characterization and settling testwork to improve TSF design including tailings settled dry density and tailings entrainment among other design parameters. The QP estimates this work will cost $35,000.

To improve the understanding of the dewatering needs and the potential hydrogeological impacts the following is recommended:

- Additional closer spaced drilling and testing of boreholes (including CPT survey) to determine the depth to bedrock, continuity of clay and sandstone lenses with installation of more vibrating wire piezometers (VWP) to monitor pressure heads in different units, particularly in the vicinity of the pit walls closest to planned infrastructure (TSF, overburden dumps). Updated South pit ground investigation for pit dewatering (includes Drilling, CPT Program, Vibrating Wire Piezometers Supply and Install, and Supervision). The QP estimates this work will cost $625,000 and must be completed prior to detailed design.

Additional recommendations to de-risk the project by main area are listed below.

26.6.1 TSF

The following are recommendations to support the TSF design:

- A review and update of the dam hazard classification for the TSF once a comprehensive and detailed quantitative dam break analysis has been completed that considers the potential failure modes, flood wave travel times, the flow arrival times, depths and velocities and the depth of material deposition. The IDF, earthquake design ground motion, required freeboard and general design guidelines will be determined based on the updated classification.
- Complete additional tailings characterization and settling testwork to improve TSF design including tailings settled dry density and tailings entrainment among other design parameters.
- Conduct site investigations and laboratory testing at TSF embankment locations to better define the foundation conditions; verify the groundwater levels directly under the proposed embankments; and estimate the extent of the clay. A geotechnical site investigation along the latest TSF embankment footprint may provide an opportunity to improve the embankment design through verifying stability leading to a reduction in embankment slopes.

26.6.2 Pit Dewatering

Additional dewatering recommendations to reduce risks based on the feasibility study include:

- Updated hydrocensus to identify all current water users that may be impacted due to dewatering in terms of water quantity and quality.
- Use the updated geotechnical data, hydrogeological data (VWP, drilling and testing results) and geological model to refine the hydrogeological domains and incorporate it into the numerical flow model.
- Include the TSF cell design in the numerical model to determine the recharge to the Overburden Aquifer and impacts of a higher phreatic surface on the pit slope stability and inflow volumes that would need to be managed. This may require changes to the cell design and sequencing to minimize the additional inflows.
- Additional dewatering boreholes should be modelled between the mine infrastructure and the pits, to optimize dewatering for wet and dry seasons and to determine the capital and operating costs with more accuracy.
- Salinity differences between the River Cacheu, creeks and aquifers should be taken into consideration in the numerical flow model as this is an additional hydraulic stress that will drive groundwater flow and the interaction between the brackish surface water bodies under current and dewatering conditions. This will also be useful in prediction of increasing salinity levels to be managed.
- A groundwater monitoring network has been established but must be expanded continuously and the data used in the updates of the numerical flow model.
- Update integrated surface water and groundwater management plan.

- Revise operating dewatering borehole operating strategy based on revised inflows from numerical flow model and including equipment recycling and maintenance requirements.
- Increase the pipe size and number of discharge points to the River Cacheu and the dewatering channel to manage flow volumes and pressures more effectively from the dewatering boreholes.

26.7 Environmental Studies and Permitting

The project has been de-risked from an environmental perspective to the extent that an environmental baseline has already been established. Nonetheless, ongoing community engagement should continue through the normal care and maintenance activities currently happening on site, including seeking the renewal of the Declaration of Environmental Compliance from the Competent Environmental Assessment Authority. Costs associated with this ongoing work are part of the Itafos project development budget.

For typical mining projects of this nature, the next phase would be used to develop an environmental baseline and a Resettlement Action Plan. However, as discussed in Section 24, the update of the Resettlement Action Plan (RAP) is linked to the overall project development schedule. Although the RAP work could begin early, it is linked to the investment decision to avoid community tensions and fatigue around the resettlement process, as well as avoiding the expiration of the RAP and asset data before the resettlement process can be implemented.

An integral part of updating the land and asset survey and the RAP is the restart of the stakeholder engagement process to get the affected communities back up to speed on the RAP process, while attempting to alleviate concerns related to project delays. KP has assumed that this initial environmental work program includes tasks that should be done ahead of project implementation (i.e., during detailed engineering).

The following tasks are recommended for completion ahead of construction.

- Complete an updated land and asset survey and update the RAP.
- Complete hydrocensus in mine area (mostly by Itafos staff by purchasing and installing level loggers).
- Complete biodiversity surveys and update the Biodiversity Management Plan.
- Refresh the pre-development baselines for air quality, noise, surface water and groundwater once a construction decision is made, for comparison to future monitoring.
- Develop an influx or project-induced migration management plan in consultation with local and regional authorities.
- Further develop human resources policies, training, and recruitment plans for environmental and social management.
- Itafos should seek a renewal of its Declaration of Environmental Compliance from the Competent Environmental Assessment Authority to ensure continuity of project approvals.
- Revise the existing management plans listed in Section 20.1.9 ahead of construction.

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