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Bayovar 12 Phosphate Project

NI 43-101 Updated Pre-Feasibility Study

Department of Piura, Peru

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Phosphate Rock
for Better Farming

DATE AND SIGNATURES PAGE

This report is effective as of 11 May 2016. See Appendix A, Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

Cautionary Note to United States Investors Concerning Estimates of Reserves and Resources

Reserve and resource estimates included in this report have been prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and the Canadian Institute of Mining, Metallurgy, and Petroleum Definition Standards on Mineral Resources and Mineral Reserves. NI 43-101 is a rule developed by the Canadian Securities Administrators that establishes standards for public disclosure by a Canadian company of scientific and technical information concerning mineral projects. Canadian standards, including NI 43-101, differ significantly from the requirements of the United States Securities and Exchange Commission ("SEC"), and reserve and resource information contained in this report may not be comparable to similar information disclosed by U.S. companies. In particular, the term "resource" does not equate to the term "reserves". Under U.S. standards, mineralization may not be classified as a "reserve" unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the reserve determination is made.

The SEC's disclosure standards normally do not permit the inclusion of information concerning "measured mineral resources", "indicated mineral resources" or "inferred mineral resources" or other descriptions of the amount of mineralization in mineral deposits that do not constitute "reserves" by U.S. standards in documents filed with the SEC. You are cautioned not to assume that resources will ever be converted into reserves. You should also understand that "inferred mineral resources" have a great amount of uncertainty as to their existence and great uncertainty as to their economic and legal feasibility. You should also not assume that all or any part of an "inferred mineral resource" will ever be upgraded to a higher category. Under Canadian rules, estimated "inferred mineral resources" may not form the basis of feasibility or pre-feasibility studies except in rare cases. You are cautioned not to assume that all or any part of an "inferred mineral resource" exists or is economically or legally mineable. Disclosure of "contained ounces" in a resource is permitted disclosure under Canadian regulations; however, the SEC normally only permits issuers to report mineralization that does not constitute "reserves" by SEC standards as in-place tonnage and grade without reference to unit measures. The requirements of NI 43-101 for identification of "reserves" are also not the same as those of the SEC, and reserves reported in compliance with NI 43-101 may not qualify as "reserves" under SEC standards. Accordingly, information concerning mineral deposits set forth in this report may not be comparable with information made public by companies that report in accordance with U.S. standards.

BAYOVAR 12 PHOSPHATE PROJECT
FORM 43-101F1 TECHNICAL REPORT

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1 SUMMARY

The Bayovar 12 project, located within the Sechura basin in northern Peru, hosts a major deposit of what is considered to be the most reactive sedimentary phosphate rock in the world (Reactive Phosphate Rock or "RPR"). The project's relatively simple mine plan and flow sheet will produce a natural phosphate fertilizer that is suitable for use as direct application phosphate rock (DAPR) that can be applied directly to many soils without the need for conventional acid pre-treatment of the rock or the addition of other manufactured chemical compounds. Sechura RPR, due to its high reactivity, can outperform more expensive and non-organic single and triple-superphosphate fertilizers when used in tropical soil and climatic conditions.

The Project has been investigated here as an open pit mine and beneficiation plant to produce 24% P_2O_5 and 28% P_2O_5 DAPR products for local and international markets that will be suitable for organic certification. The results of this updated prefeasibility study indicate that phosphate resources contained on the Bayovar 12 concession are both technically and commercially feasible to produce and ship these DAPR products in both the near and long-term development of the concession.

The original PFS dated December 11, 2015 was based on producing two direct application phosphate rock (DAPR) products, one at 24% P_2O_5 and the other at 28% P_2O_5 in parallel process lines that were each capable of producing 500,000 tonnes of DAPR per year (mtpa). The first process line started at a reduced capacity of 300,000 mtpa in Year 1 ramping up to full production in Year 4. The second process line commenced at full production (500,000 mtpa) in Year 3.

The results of the original PFS, while not unacceptable by comparative standards, were deemed by FCV to be inadequate to obtain attractive project financing so it undertook a new look at The Project to improve the project economics. The mineral resources stayed the same but a new mine plan was developed to shorten waste rock hauls while making the 13 phosphate beds more accessible. The two process lines were combined into a single process line having a capacity of 1 million mtpa. The same two products, 24% P_2O_5 DAPR and 28% P_2O_5 DAPR, will be batch-produced from the single process line. The new mine plan and project design were estimated for capital and operating costs, and an updated financial model was developed. This report is the PFS Update to the original Bayovar 12 Phosphate Project PFS report.

1.1 PROPERTY DESCRIPTION AND OWNERSHIP

The Bayovar 12 concession is located in the Province of Sechura, Department of Piura in northwestern Peru. The property is located approximately 950 km north of the Peruvian capital, Lima, 65 km south of the town of Sechura and 90 km southwest of Piura. The concession is approximately 40 km east of the fishing village of Puerto Rico, which is situated on the southern margin of Sechura Bay on the Pacific coast of Peru.

The Bayovar 12 concession is located approximately 15 km northeast of Vale's operating Miski Mayo Phosphate Mine and directly east of the Fosfatos del Pacifico (FOSPAC) phosphate reserve area.

The concession is connected by sealed road to tidewater and the JPQ marine port facilities 40 km to the west, which is owned by Focus's Peruvian partners, Trabajos Maritimos S.A. and Inca Terminals and Mining Inc who are marine transport and service providers. The marine port facility is used by JPQ principally for the export of gypsum currently mined from the Bayovar 12 Concession and for phosphate rock produced locally from adjacent concessions. The JPQ port terminal was previously used to export phosphate rock extracted from the Bayovar Mine, prior to its acquisition by Vale.

1.2 MINERAL TENURE

Juan Paolo Quay S.A.C (JPQ) is the title holder of the Bayovar 12 mining concession. On March 26, 2015, Focus (via Peruvian subsidiary Agrifos) acquired an outright 70% interest in the issued share capital of JPQ, by paying $4 million cash to the owners of JPQ. The remaining 30% interest in JPQ is owned by Trabajos Maritimos S.A. and Inca Terminals and Mining Inc., subsidiaries of Grupo Romero (Peru) and Mamut Andino C.A. (Ecuador), respectively.

The Bayovar 12 Concession comprises 12,575 hectares and was acquired by JPQ in 2007 under a contract with state company Activos Mineros S.A.C. for the exploitation of gypsum rock by open pit methods from the claim.

1.3 GEOLOGY AND MINERALIZATION

The Bayovar-Sechura Phosphate Deposit occurs in the Sechura Basin, a shallow north trending basin situated in northwestern Peru. The basin is filled by a thick sequence of interlayered marine sediments including phosphorite, diatomite, sandstone, shale and volcanic tuff, ranging in age from Eocene (56.0 to 33.90 Ma) at the base to Pliocence (5.33 to 2.58 Ma) in the upper basin.

The phosphate bearing units occur in the upper 135 to 215 m of the Miocene (23.22 to 5.33 Ma) strata in the basin, within the Zapallal Formation. The phosphorite beds are comprised primarily of massively bedded phosphate pellets with lesser grains and fragments of diatoms, volcanic glass; evaporate salts; quartz; feldspar; sponge fragments; gypsum, mica flakes and organic matter. The phosphate is marine in origin and is generally in the form of the mineral francolite, a fluorhydroxycarbonate apatite.

The apatite is generally in the form of individual pellets although agglomerations of pellets, oolites, laminae, nodules and fragments of teeth, bones or shells are also present. The pellet grain size ranges from 0.4 to 2.0 mm in diameter, with larger pellets occurring in the phosphorite beds while finer grained pellets occur in the diatomite.

The diatomite ranges in color from white to brown to olive green. The diatomite generally has high porosity, often on the order of 90%; as a result of this and its resistance to compaction, the specific gravity is very low, typically the wet density is around 1.5.

The Zapallal Formation stratigraphy dips gently to the east within the Bayovar 12 Concession. No faulting or folding was identified within the concession.

Focus has intercepted 16 distinct and correlatable phosphorite beds (identified as PH01 through PH16) across the concession. Focus and Golder have interpreted the upper 13 phosphorite beds (PH01 to PH13) as Diana ore zone beds that have been modeled as mineral resources. The phosphorite beds range in thickness from 0.31 meters to 1.76 meters. The individual phosphorite beds exhibit relatively uniform thickness and P_2O_5 grade profiles across the concession; however, there is a pronounced zonation of P_2O_5 grades in both the phosphorite and diatomite beds that effectively divides the Diana ore zone into an Upper Diana ore zone and Lower Diana ore zone.

1.4 EXPLORATION STATUS

Detailed exploration drilling activities on the Bayovar 12 Concession to date have been limited to the Phase 1 (2014) and Phase 2 (2015) Focus exploration programs.

The Phase 1 exploration program resulted in the completion of 20 HQ (63.5 mm core diameter) vertical core holes totaling 2,027 m while the Phase 2 exploration program added an additional 42 HQ vertical core holes totaling 3,944 m for an overall project total of 62 drill holes and 5,971 m. The drill hole total depths for both programs ranged from 81 to 131 m (mean of 96 m); total depth variation was due to the location relative to the Tablazo.

The Phase 1 drilling was conducted on a nominal 800 by 800 m spaced grid covering approximately 27.36 km^2 (2,736 Ha) of the total 125.75 km^2 (12,575 Ha) of the Bayovar 12 Concession. The Phase 1 drilling program concentrated on the western portion of the Bayovar 12 Concession.

The Phase 2 drilling expanded the nominal 800 by 800 m spaced drilling grid towards the east of the Phase 1 drilling, while also including some closer 400 by 400 m spaced drilling to allow for evaluation of shorter range thickness and grade variability. As of the effective date of this technical report, a significant portion (approximately 65%) of the concession remained undrilled.

1.5 DEVELOPMENT AND OPERATIONS

As of the effective date of this technical report there has been no phosphate development work undertaken on the Bayovar 12 Concession.

Mining activity on the Bayovar 12 Concession property is limited to small scale surface mining of quaternary age gypsum that occurs at surface on the low ground immediately east of the Tablazo. The gypsum mining operation is carried out by JPQ, using a dozer to push the gypsum into piles that are then loaded on to a small road haul truck using an excavator. The gypsum is then transported by truck to the JPQ port facility on Sechura bay where it is stockpiled prior to loading onto ships.

1.6 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

1.6.1 Mineral Resource

Mineral Resources are subdivided into classes of Measured, Indicated, and Inferred, with the level of confidence reducing with each class respectively. Mineral Resources are always reported as in situ tonnage and are not adjusted for mining losses or mining recovery.

Geological modelling and subsequent mineral resource estimation was performed by Golder in accordance with Golder internal modelling and resource estimation guidelines and in accordance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (May 2003 edition).

The geological data compilation, interpretation, geological modelling and Mineral Resource estimation methods and procedures are described in the following sections.

1.6.2 Geological Database and Interpretation

All available Phase 1 and Phase 2 drill hole data and observations provided by Focus were compiled and loaded into a geological database. Golder independently reviewed all phosphorite bed picks and correlations. The phosphorite-diatomite contact roof and floor picks performed by the Focus drill site geologists were reviewed by Golder using the drill hole descriptive geological logs, core photographs and the down hole analytical results.

Once the drill hole geological intervals were reconciled with the downhole analytical results, Golder performed a review of the overburden, phosphorite and diatomite bed correlation interpretations that were provided by Focus. Correlation fences were created in both the east-west and north south directions across the Bayovar 12 Concession. All 62 of the Focus drill holes were included in the correlation fences.

1.6.3 Stratigraphic and Structural Model

Stratigraphic and structural data from the verified geological database was imported to the StratModel™ application of the Ventyx MineScape geological modelling and mine planning software in preparation for the construction of a gridded stratigraphic and structural model.

The stratigraphic grid model comprises gridded structure surfaces for each modeled overburden, phosphate, interburden and underburden unit. The structure grids created represent the individual unit roof, floor, vertical thickness (roof minus floor) and true thickness. The stratigraphic model is based on a series of gridded elevation and thickness horizons, with grid cell geometry of 50 by 50 m (east-west by north-south).

There were no faults included in the Bayovar 12 model schema as none were identified in the drill hole geological data nor were any faults identified in a review of the regional geological mapping coverage for the area surrounding the Bayovar 12 Concession.

1.6.4 Density/Specific Gravity

To facilitate the conversion of modelled volumes to tonnes Golder calculated dry basis and wet basis relative density values for all modelled phosphorite beds and waste units using relative density and moisture analyses data collected during the Phase 2 exploration drilling program. In a similar manner, default dry basis and wet basis relative density values were calculated for the diatomite interburden units and the overburden and underburden units.

1.6.5 Grade Model

Using the verified modelling database and the finalized stratigraphic and structural model, a phosphate grade gridded model was developed using the StratModel application of MineScape. The grade model was developed using the same 20 by 20 m spaced grid that was used for the stratigraphic and structural grid model.

The grade grid model comprises gridded surfaces for each modeled grade parameter for each individual phosphate and diatomite bed; the grade model grid surfaces are spatially associated with the corresponding stratigraphic model grid surfaces. The grade parameters included in the model were: P_2O_5; Al_2O_3; CaO; Fe_2O_3; MgO; and SiO_2.

1.7 MINERAL PROCESSING AND METALLURGICAL TESTING

No metallurgical testing has been performed prior to 2014 on the phosphorite layers underlying the Bayovar 12 Concession. The phosphorite layers are extensive and similar to those being mined by Vale (Bayovar Concession 2) and to those being developed by Fosfatos del Pacifico (Bayovar Concession 9). The unconsolidated phosphorite layers from Concessions 2 and 9 do not require grinding. The bench-scale test program for Bayovar 12 phosphorite layers examined the proven unit operations for recovering concentrate from the ore at Concessions 2 and 9.

The bench-scale testing in 2014/15 was performed by Jacobs Engineering (Jacobs) to develop a process for recovering a phosphate rock product from the Bayovar 12 phosphorite layers. The work by Jacobs is summarized in the reports "Beneficiation Testing Focus Ventures Ltd.," "Scrubbing, Settling, and Filtration Testing" and "Focus Bench Scale Flow Sheet Validation".

1.7.1 Metallurgical Samples

The metallurgical testing was performed on two sets of samples.

Focus Ventures geology team in Peru took the first set of samples from air-dried drill cores. Jacobs' preparation and characterization of these samples are described in Sections 13.1.1 and 13.1.2. Testing these samples was directed at recovering a phosphate rock concentrate containing +29% P_2O_5 that was suitable for conversion to phosphatic fertilizer by acidulation and granulation.

The second sample set comprised two composite samples – one for phosphorite layers 2 through 6 and the other for phosphorite layers 11 to 13, both of which had been prepared by ALS, an analytical laboratory in Vancouver, British Columbia. ALS had previously dried and crushed the cores to obtain a representative sample for chemical analysis. The two representative composites were prepared from PQ diameter drill core drilled specifically for metallurgical test work. Testing these samples was directed at recovering a concentrate containing +24% P_2O_5 that was suitable for use as direct application phosphate rock (DAPR). Jacobs' chemical analysis of these two composites is discussed in Section 13.1.4.

1.7.2 Bench-Scale Tests of Individual Layer Samples

1.7.2.1 Drum Scrubbing and Desliming

Drum scrubbing is a proven unit operation for disaggregating unconsolidated phosphorite. The retention time and slurry % solids for drum scrubbing were investigated using the larger composite samples (PH02, PH06, and PH13). The scrubbing parameters selected for subsequent tests were 37% solids slurry and 3 minutes retention time for all 13 composite samples.

When the confirmation tests were performed on each composite sample, the samples were moistened to about 30% moisture prior to scrubbing at 37% solids for 2 minutes.

1.7.2.2 Attrition Scrubbing and Desliming

The purpose of attrition scrubbing is to disaggregate the remaining diatomite so that it can be removed by desliming. One stage and two-stage attrition were tested. The lab data indicated that the % P_2O_5 of the deslimed attrition cell discharge was not significantly different for one or two stage attrition; however, two stage attrition reduced the recovery of P_2O_5 by as much as 6%.

Drum scrubbing used moist samples diluted to 37% solids with 3 minutes retention time. Attrition scrubbing used the +53 µm from the drum discharge diluted to 55% solids with 12 minutes retention time. The deslimed attrition cell discharge, including the >600 µm, contained 26.1% P_2O_5 on average, which is below the normal minimum for phosphate rock sold for use in phosphoric acid plants, but is suitable for use as DAPR.

The final washed products were sieved at 600 µm and 150 µm to determine which fraction was causing the grade dilution. Except for PH01, the grade dilution was consistently caused by the 150/53 µm fraction, which averaged 22.7% of the composite weight and 23.8% P_2O_5. Combining the +600 µm and the 600/150 µm fractions gave an average yield of 19.9% weight with 29.5% P_2O_5, which meets the normal minimum for commercial phosphate rock used to manufacture phosphoric acid, but the yield is low.

To achieve the normal minimum grade for commercial phosphate rock with an improved yield it is necessary to upgrade the 150/53 µm fraction by flotation. Flotation is not a required component in the current Bayovar 12 flowsheet for the production of DAPR, however results of laboratory test work by Jacobs on the 150/53 µm fraction can be found in Jacobs, 2015A "Beneficiation Testing Focus Ventures Ltd."

The apatite mineral at Bayovar 12 has a relatively low P_2O_5 content due to:

- CO_3 substitution for PO_4 in the crystal lattice
- The apatite grains are not completely liberated
- Quartz, feldspar, and calcium sulfate are not completely removed by beneficiation

The upper layer (PH01) produces lower % P_2O_5 concentrate with higher cadmium content and therefore will be monitored closely during exploitation. The chemical compositions of composites PH02 to PH13 indicate that the phosphate rock can readily be converted to phosphoric acid (% P_2O_5 > 28.5 and Calcium Oxide Ratio (COR) < 1.65) and high analysis fertilizers (MER < 0.100).

Bayovar phosphate concentrates typically contain +5% CO_2. The substitution of CO_3 for PO_4 in the crystal lattice causes the Bayovar phosphate concentrates to be highly reactive and well suited for use as direct application phosphate rock (DAPR).

The laboratory data indicate the simple washing flowsheet with desliming at 53 μm can produce a concentrate averaging more than 24% P_2O_5 except for layers PH01 and PH02. If PH01 is excluded, the yield and P_2O_5 grade average 44.6% and 24.8% respectively. The average grade of +24% P_2O_5 is acceptable for DAPR. To bring the washed products from PH01 and PH02 up to a grade of 24% P_2O_5 it will be necessary to coarsen the tertiary slimes cut point. Similarly, to obtain washed products with a composite grade of 28% P_2O_5 from layers PH03 to PH13 it will be necessary to coarsen the tertiary slimes cut point.

1.7.3 Metallurgy Conclusions

Testing of individual phosphorite layers established a robust process for production of a 29% P_2O_5 concentrate suitable for the manufacture of phosphoric acid. The same testwork demonstrated that the phosphorite layers were also suitable for the production of two qualities of DAPR. Focus selected the DAPR flowsheet to take advantage of the ease of beneficiation and unique reactivity of the Bayovar 12 phosphorite ores, and suitability as a natural fertilizer on the acid/slightly acidic soils typical throughout Latin America. Preliminary analysis indicated that the estimated capital and operating costs of a mine and plant for producing export quality phosphate concentrate suitable for manufacturing phosphoric acid were greater than for a similar operation producing DAPR.

The data from the same test work were used to establish a simpler flowsheet for producing 24% P_2O_5 DAPR (Mode A) and 28% P_2O_5 DAPR (Mode B).

The flowsheet for producing two qualities of DAPR is illustrated as a block flow diagram in Figure 1-1. Mode A produces 24% P_2O_5 DAPR by (tertiary desliming) at 53 μm and Mode B produces higher grade DAPR by coarsening the tertiary desliming cut point to reject lower grade near size material. The hydrosizer (tertiary) cut point may be adjusted by increasing the teeter bed density set point and increasing the flow of teeter water.

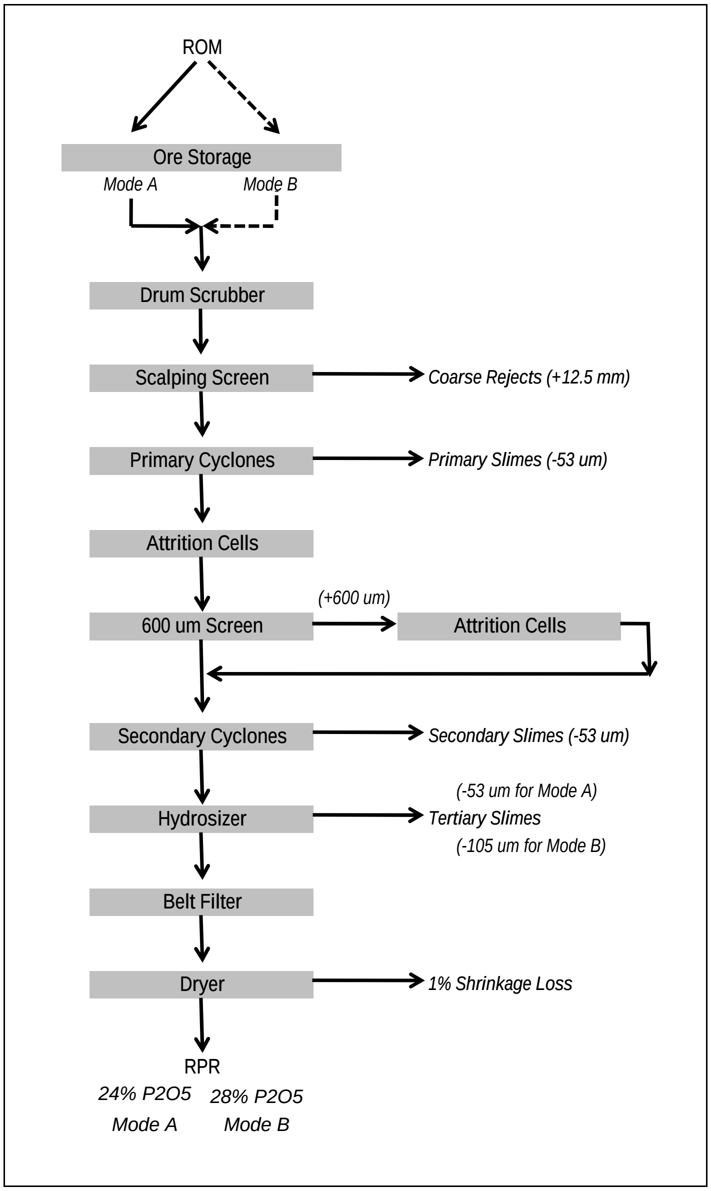


Figure 1-1: Block Flow Diagram for Two Grades of DAPR

1.8 MINERAL RESOURCE ESTIMATION AND CLASSIFICATION

Phosphate Mineral Resources were estimated for each individual phosphorite bed from PH01 through PH16.

Resource volumes and grade were estimated for each phosphorite and diatomite bed using the corresponding unit roof and floor grids from the structural grid model. The volumes for each phosphorite and diatomite bed were then converted to tonnes using the phosphorite bed specific relative density values.

Mineral resources have been classified into Measured, Indicated and Inferred Mineral Resource using area of influence polygons around points of observation. Classification was performed individually for each phosphorite bed using drill hole intercepts on the floor of the unit for the location of the point of observation. The area of influence polygons were generated on the floor surface for each phosphorite bed rather than on the horizontal plane to allow for the dip of stratigraphy.

Golder performed a statistical and geostatistical analysis of the phosphorite bed thickness and P_2O_5 grade data. Given the relatively limited dataset (62 drill holes) and the general uniformity of thickness and grade across the concession area for most of the phosphorite beds, the preliminary review of the thickness and P_2O_5 variograms suggested a broad range for the variograms.

The resultant areas of influence classification parameters used by Golder for the Bayovar 12 Concession Mineral Resource estimate are as follows:

- Measured Mineral Resources – 400 m spacing between points of observation
- Indicated Mineral Resources – 800 m spacing between points of observation
- Inferred Mineral Resources – 1,600 m spacing between points of observation

1.8.1 Statement of Mineral Resources

A summary of the classified Mineral Resources for phosphorite beds PH01 through PH16 from the Focus Bayovar 12 Concession is presented in Table 1-1. Mineral Resources reported are inclusive of tonnes converted to Mineral Reserves in subsequent sections of this report.

Table 1-1: Summary of Mineral Resources, Beds PH01 to PH16

Category	Tonnes (Mt; wet)	Tonnes (Mt; dry)	P_2O_5 Grade (wt. %)
Measured	23.4	17.7	13.16
Indicated	277.1	209.5	13.04
Inferred	135.0	102.2	13.11

Note: Mt = million tonnes
No minimum thickness, grade cut-off or other mining parameters applied
Phosphorite bed specific wet and dry relative densities used for tonnage calculations

1.9 MINERAL RESERVE ESTIMATES

1.9.1 Estimated Phosphorite Reserves

The Bayovar 12 phosphate deposit has been delineated over an area of approximately 34 km^2. The deposit consists of sixteen mineralised units. This Mineral Reserve Estimate only concerns the upper 13 of the 16 total modelled units (capas). The lower three beds, PH14, PH15 and PH16, were excluded from the mineral reserve estimate due to limited thickness and low grades.

1.9.2 Mining Model Development

1.9.2.1 Criteria for Determination of ROM Phosphorite

Run of Mine (ROM) mining surfaces were created to account for anticipated 7.5-cm roof and 7.5-cm floor dilution gain where the phosphorite capa was greater than the minimum mineable thickness of 30 cm. These assumed dilution and mining loss factors are based on extracting the phosphorite with surface miners to recover the entire thickness of the capas (plus dilution) to calculate ROM tonnages. ROM quality surfaces were also developed to account for the dilution gains using the quality data from the geological resource model.

1.9.2.2 Beneficiation Plant Yield and Project Quality Model

Beneficiation plant yields were estimated using a set of capa-specific predictive equations that are driven by the ROM (feed) P_2O_5 quality. Yield recommendations developed by Metallurgy QP Glenn Gruber and equations were based on the results of the laboratory testing performed on the metallurgical, geological holes drilled during 2014 and 2015.

1.9.2.3 Development of the 3D Block Model for Pit Optimisation

After developing the ROM and product surfaces, the mineral resource model was blocked into 3D cells 20 m by 20 m by 1.6 m in the X, Y, and Z, respectively, for the purposes of pit optimization, totaling 21.7 M blocks were created. The 20x20x1.6m blocks were sub-blocked to smaller size blocks in some areas to suspect the capas and interburden boundaries. The smallest subblock size is 5x5x0.2m.

1.9.2.4 Mineral Reserve Estimation Methodology

The assessment of surface-mineable phosphorite reserves within the Project area was based on a 20-year mine plan open-pit design which accounts for the effects of highwall laybacks on the estimated 20-year mineable reserve.

The 3D block resource model formed the initial basis of mining volume estimates for the extent of the project area. Using the modifying mining factors and plant performance yield and quality predictions, the blocks were populated with an economic value for the purpose of conducting Lerchs Grossmann (LG) pit optimization. Based on the requirement of the 20-year, 1.0 M tonnes phosphate concentrate (product tonnes) per year mine plan, the final pit configuration was designed on the $90/tonne of phosphate concentrate pit optimizations.

IMC started with the Golder pit design and expanded the design as needed to develop the updated 20 year Mineral Reserve. Pit boundary assessments were completed by IMC over a range of unit revenue values for phosphate concentrate (i.e., saleable product), with unit costs for mining, processing, general and administration, production transportation, etc. to assure the robustness of the modified pit design.

The design criteria for the final pit configuration are shown in Table 1-2 below.

Table 1-2: Summary of Mine Design Parameters

Description	Value
Pit Wall inter-ramp slope angle	20 degrees above -30 elevation; 26 degrees below
Bench Height	5m, double benched to 10m
Bench face design angle	30 degrees above -30 elevation; 35 degrees below
Bench dig face angle	~65 degrees
Pit Haul roads	25m wide, 8% maximum grade
Minimum mineable thickness	30 cm
Mining roof dilution gain	7.5 cm
Mining floor dilution gain	7.5 cm
Mining Recovery	100%
Pit Buffer from Bayovar Road	180 m
Target Average Product Grade	Product A (24+% P_2O_5) and Product B (28+%)

[1]For a complete discussion of the geotechnical units in the Bayovar 12 Project, please refer to Golder's geotechnical report "Focus Ventures Bayovar 12 Pre-Feasibility Report – Open Pit Design Recommendations"

1.9.3 Mineral Reserve Estimation Statement

The Phosphate Reserves expressed as mined phosphorite tonnes and phosphate concentrate (product) are shown in Table 1-3 and Table 1-4. The Mineral Reserve Estimate is based on a minimum capa thickness of 30 cm.

Table 1-3: Proven and Probable Reserves Expressed as ROM Mined Phosphorite

Capa	Proven Reserves	Probable Reserves	Total Reserves	P_2O_5 (%)
	Million Tonnes (dry)			
PH01	1.28	4.00	5.28	12.33
PH02	2.42	7.57	10.00	11.15
PH03	1.25	4.02	5.26	17.52
PH04	0.73	1.03	1.76	13.59
PH05	0.00	0.26	0.26	10.01
PH06	1.65	4.76	6.41	13.63
PH07	0.58	2.88	3.46	10.27
PH08	0.70	1.49	2.19	10.69
PH09	1.15	3.29	4.43	12.39
PH10	0.08	1.43	1.51	10.46
PH11	0.82	2.87	3.69	12.85
PH12	1.22	3.80	5.03	13.93
PH13	2.47	7.03	9.50	13.73
Total	14.35	44.42	58.77	12.93

Notes: ROM tonnes within the updated Pre-Feasibility Study pit design

Table 1-4: Proven and Probable Reserves Expressed as Phosphate Concentrate Product

Capa		Proven Reserves	Probable Reserves	Total Reserves	P_2O_5 (%)
		Million Tonnes (dry)			
PH01[1]		0.56	1.69	2.25	23.85
PH02[1]		1.03	3.23	4.26	22.50
PH03[2]		0.57	1.90	2.47	29.06
PH04[2]		0.28	0.38	0.66	29.66
PH05[1]		0.00	0.08	0.08	25.70
PH06[2]		0.44	1.40	1.84	28.68
PH07[1]		0.17	0.87	1.04	25.49
PH08[1]		0.19	0.41	0.61	26.05
PH09[1]		0.42	1.25	1.67	25.92
PH10[1]		0.02	0.44	0.46	26.01
PH11[2]		0.23	0.80	1.02	28.20
PH12[2]		0.37	1.16	1.53	28.05
PH13[2]		0.69	2.11	2.80	27.81
Total		4.97	15.72	20.70	26.24

Notes (1) Product A (24% P_20_5) (2) Product B (28% P_20_5) within the updated Pre-Feasibility Study pit design

For the Bayovar 12 Phosphate Deposit 20-year LOM plan, the total estimated Proven and Probable ROM Reserves are 58.7Mt (dry basis) with an average ROM P_2O_5 grade 12.93%. Total phosphate concentrate (product) tonnes after beneficiation are estimated to be 20.70 Mt (dry basis) with an average product P_2O_5 grade of 26.24%. The overall ROM strip ratio (SR) is estimated to be 7.19 dry tonnes of waste per dry tonne of ROM phosphorite. The overall product SR is estimated to be 20.4 Mt (dry) of waste per tonne of phosphate concentrate, requiring the removal of approximately 422.5 Mt of waste over the life of the mine.

1.9.4 Potential for Future Reserve Expansion

As stated in Section 15.2.4, the Mineral Reserves Estimate is based solely on the 20-year mine plan open-pit design with highwall laybacks and a target production rate of 1.0 M tonnes (dry) of phosphate concentrate per year. Although Mineral Resources exist outside the 20 year mine plan pit, the mine schedule and Mineral Reserves were limited to the 20 year pit shell.

1.10 MINING METHOD

The Project Site is contained within a low-lying, open, generally flat area, with elevations varying from 0 m to 30 m above mean sea level (amsl). The Bayovar 12 phosphorite deposit dips between 1°and 2° toward the north-northeast with sixteen primary capas, thirteen of which will be mined. Front end loaders (FEL) are the primary method of waste stripping with surface miners to extract the phosphorite and a portion of the interburden. All waste will be hauled to a waste storage facility (WSF), while phosphorite will be hauled to the plant using trucks.

The phosphorite extraction will be executed by a multiple bench open-pit haul-back mine. Initial mining requires ex-pit storage of waste rock, both overburden and interburden. Once a sufficient volume has been excavated, the waste rock is back-hauled into the mined-out area.

For the 1.0 Mtpy mine production plan, overburden will be stripped by Focus personnel. Overburden excavation will advance ahead of the phosphorite extraction in 5-m height production benches. Interburden will be stripped and removed with a front-end loader, or when the interburden is too thin, a dozer or surface miner.

To minimize mining dilution while maximizing recovery and production capabilities, the phosphorite will be mined with GPS controlled surface miners.

The mine plan maximizes the number of available production faces for the various capas throughout the plan life in order to accommodate blended plant feed requirements. Production equipment is included in the plan to source ore from multiple capas and maintain the blend and feed tonnage to the plant. The ROM stockpile has been designed as "fingers" to enable separation of individual or similar capas to provide feed for both mineral concentrates, which require slightly different ore characteristics.

The data from the test work were used to develop a flowsheet for producing 24% and 28% DAPR phosphate concentrate. Both mineral concentrates include scrubbing, attrition, and desliming to reach the required P_2O_5 concentrate grade. Capas were assigned to a plant operation mode using the majority process required to reach an acceptable concentrate P_2O_5 quality of 24% and or 28%. Mode A operation will produce a 24% P_2O_5 DAPR product through tertiary desliming at 53µm and Mode B operation will produce 28% DAPR) by coarsening the tertiary desliming at 105µm to reject lower grade near size material.

Table 1-5 below shows the process scheme to which each phosphate capa has been assigned.

Table 1-5: Beneficiation Process by Phosphate Capa

Phosphorite Capa	Mode A 24% P_2O_5	Mode B 28% P_2O_5
PH01	Yes	No
PH02	Yes	No
PH03	No	Yes
PH04	No	Yes
PH05	Yes	No
PH06	No	Yes
PH07	Yes	No
PH08	Yes	No
PH09	Yes	No
PH10	Yes	No
PH11	No	Yes
PH12	No	Yes
PH13	No	Yes

Optimization was conducted on Measured and Indicated Resources only; Inferred Resources were treated as waste. To prevent the optimized pits from encroaching on the Bayovar road, a 180-m offset buffer zone were established.

Figure 1-2 shows the pit phases, external waste dumps, and input backfilling where phosphorite mining is completed at the end of Year 5. The 20 year pit limit is shown as a bold dashed line. Figure 1-3 shows the pit progress at the end of Year 20.

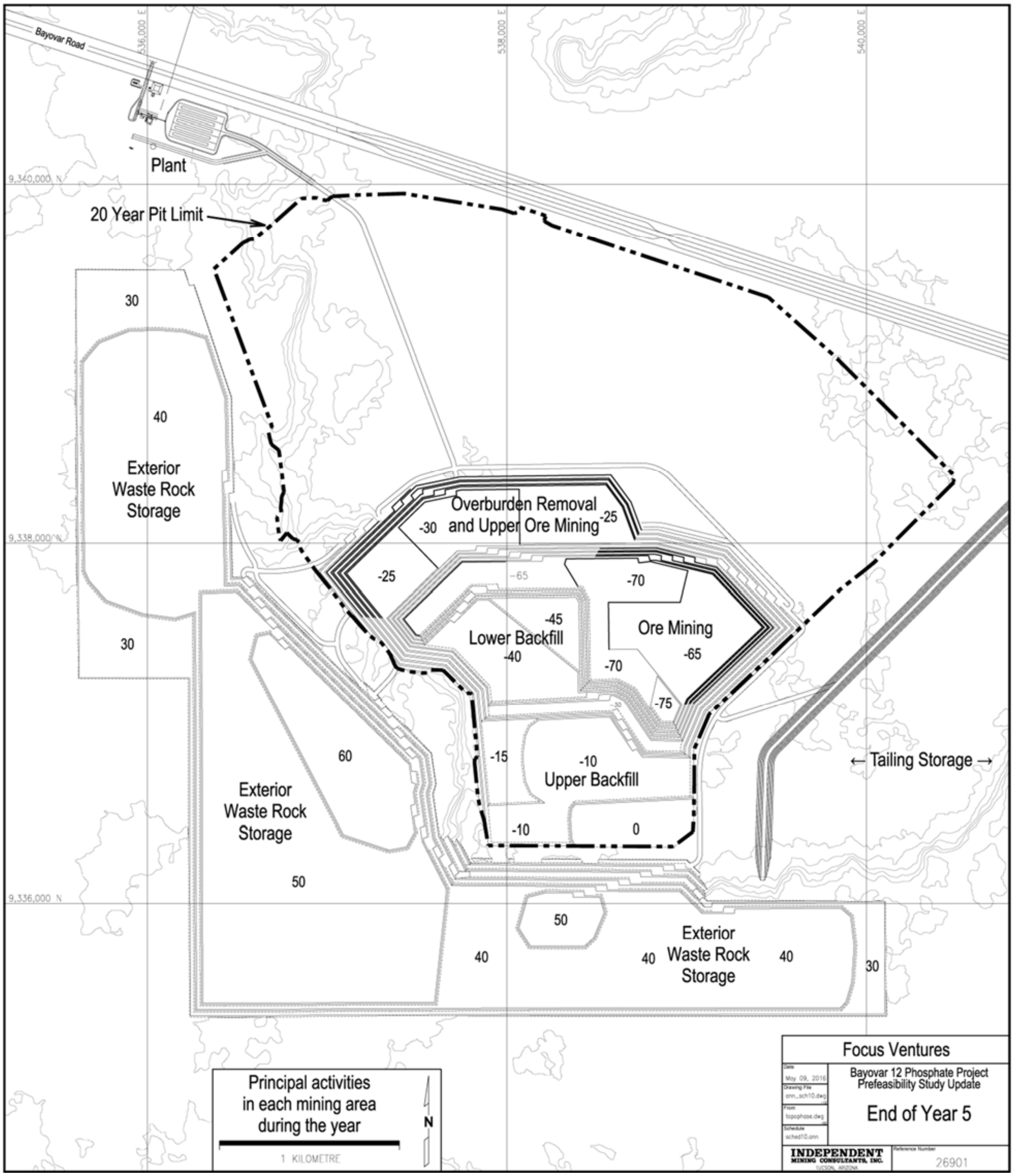


Figure 1-2: Pit phases, external waste dumps, and pit limits at the End of Year 5

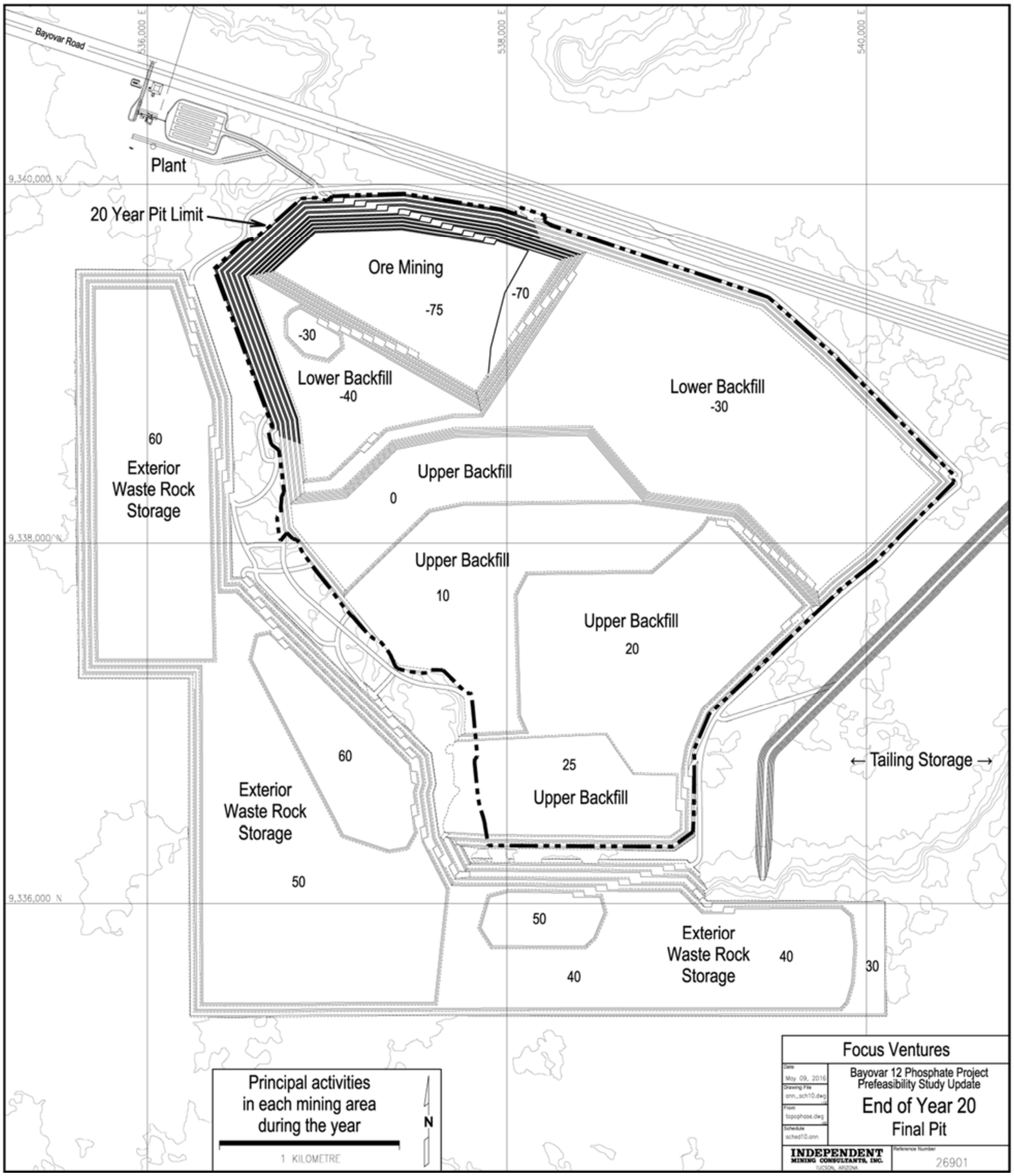


Figure 1-3: Pit phases, external waste dumps, and internal backfilling at End of Year 20

Mine waste will be used to construct the TSF containment embankment. The Phase 1 TSF embankment (constructed during pre-production) will require approximately, 2.6 Mm^3 of material placed in engineered lifts to 95% compaction. The Phase 2 embankment will require 3.3 Mm^3 of engineered material and construction will need to be completed during years 4 through 6.

The external waste dumps have been designed to store up to 134 M bcm of waste rock with the volume required calculated using a 20% swell factor. Backfilled waste within the open pit will accommodate 371 M bcm, also using a 20% swell factor for volume calculations.

Maximum in-pit backfill (IB) volumes were determined for each year by offsetting the pit toe 150 m and building lifts in 10 m increments until the facility crest intersected original topography.

Mine haul road will require upgrading with crushed aggregate to support the weight of the haul trucks given the low bearing capacity of the road surfaces.

Primary mine fleet requirements will include a maximum of two 1000 tph surface miners, four 31 cubic metre FEL, four track dozers, thirty seven 90-tonne haul trucks with 110 cubic metre coal beds, graders, compactors, water trucks, and other ancillary mining equipment.

1.11 RECOVERY METHODS

The phosphate beneficiation plant processes mined phosphate ore to produce phosphate concentrate. The ore is mined from thirteen phosphate-rich capas. A single plant will beneficiate the Bayovar 12 ore. Mode A will produce on average 500,000 dry mtpy of DAPR concentrate annually with a target grade of 24% P_2O_5. Mode B will produce 500,000 dry mtpy of DAPR with a target grade of 28% P_2O_5. The beneficiation process consists of desliming, dewatering to 15% moisture and drying to 4% moisture using unit operations including drum washing, size classification, attrition scrubbing, hydraulic classification, filtering, and fluid bed drying. The process uses seawater throughout: the product is not rinsed with freshwater to remove salts.

Each process plant consists of the following unit operations: as shown in Figure 1-4Figure 1-4. Mode A and Mode B are essentially identical except for the cut point for fines in the tertiary classifying hydrosizer. The cut point will be set at 53 µm to make 24% P_2O_5 DAPR (Mode A) while the cut point will be set at 105 µm to make 28% P_2O_5 DAPR (Mode B).

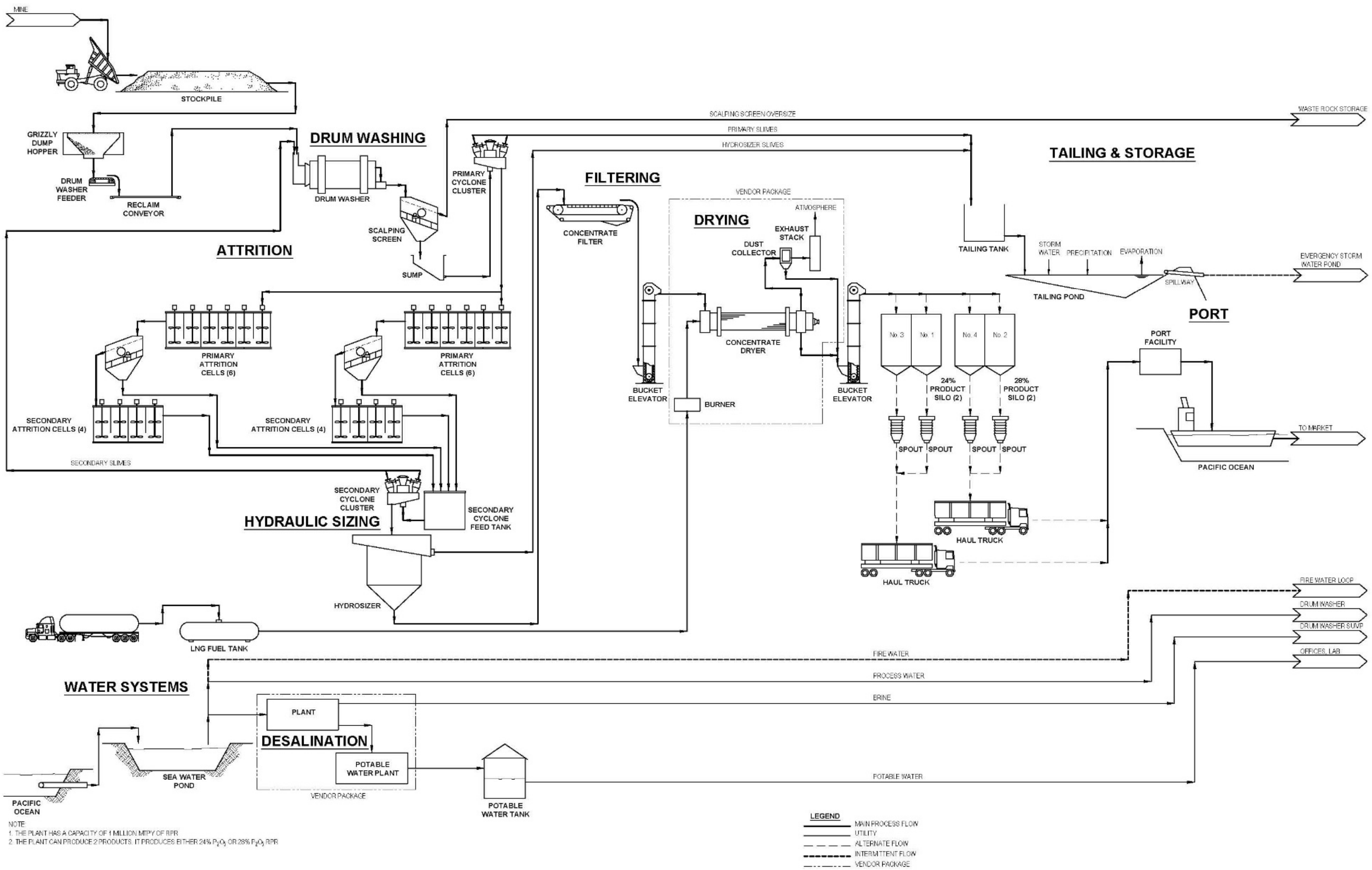


Figure 1-4: General Flowsheet for Bayovar 12 Process Plant

The test work showed that the phosphate recovery for Mode A is 81.5% P_2O_5 and for Mode B it is 66.4% P_2O_5 as shown in Table 1-6 below.

Table 1-6: Phosphate Recoveries by Capa for 24% P_2O_5 and 28%P_2O_5 by Fraction

Operating Mode A

	Resource			+28 mesh		28/270 mesh		+270 Concentrate		-270 mesh		% P_2O_5
Capa	Metric Tons	% P2O5	% WSS	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	Recovery
PH01	5,278,423	12.33	6.51	0.88	21.92	42.14	23.90	43.02	23.86	50.47	4.09	83.2
PH02	9,999,977	11.15	7.71	3.33	27.95	39.57	22.07	42.90	22.52	49.39	3.02	86.6
PH05	824,845	12.13	6.93	3.11	28.63	35.68	25.44	38.79	25.70	54.28	3.98	82.2
PH07	3,261,593	10.28	5.84	1.04	27.83	28.89	25.75	29.93	25.83	64.23	3.98	75.2
PH08	2,185,906	10.69	5.43	0.98	30.25	26.95	25.99	27.93	26.14	66.64	5.08	68.3
PH09	4,433,704	12.39	4.48	0.88	31.37	37.09	25.81	37.97	25.94	57.55	4.41	79.5
PH10	1,509,066	10.46	4.68	1.03	31.89	29.60	26.00	30.63	26.20	64.69	3.77	76.7
Combined	27,493,514	11.43	6.37	1.87	27.91	36.73	23.93	38.60	24.12	55.03	3.85	81.5

Operating Mode B

	Resource			+28 mesh		28/150 mesh		+150 Concentrate		-150 mesh		% P_2O_5
Capa	Metric Tons	% P2O5	% WSS	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	Recovery
PH03	5,264,531	17.52	5.23	6.59	31.12	40.64	30.33	47.23	30.44	47.54	6.62	82.0
PH04	1,757,167	13.59	5.35	4.04	30.58	33.24	29.48	37.27	29.60	57.38	4.47	81.1
PH06	6,044,550	13.56	4.26	1.29	29.45	27.54	28.88	28.82	28.90	66.92	7.82	61.4
PH11	3,685,348	12.85	4.13	2.20	30.93	25.71	27.95	27.90	28.18	67.97	7.34	61.2
PH12	5,027,668	13.93	2.98	2.80	31.78	27.71	27.86	30.50	28.22	66.52	8.00	61.8
PH13	9,499,165	13.73	2.93	1.05	30.92	28.68	27.66	29.73	27.77	67.34	8.13	60.1
Combined	31,278,429	14.26	3.86	2.61	30.98	30.22	28.65	32.83	28.83	63.31	7.56	66.4

Mode A will process 1.31 million mtpa of ore and produce 500,000 dry mtpy of concentrate with a grade of 24% P_2O_5. The capas designated for Mode A contain 11.4 percent P_2O_5, 6.4 percent water soluble salts (WSS) and 30 percent moisture. Testwork showed that the overall recovery of P_2O_5 is 81.5 percent. Losses are due to P_2O_5 reporting to slimes during washing, attrition scrubbing and classification with a 1 percent loss to dust and shrinkage. The plant operating availability is 85 percent based on 7,446 operating hours per year.

Mode B will process 1.50 million mtpa of ore and produce 500,000 dry mtpy of concentrate with a grade of 28% P_2O_5. The capas designated for Mode B contain 14.3 percent P_2O_5, 3.9 percent water soluble salts (WSS) and 30 percent moisture. Testwork shows that the overall recovery of P_2O_5 is 66.4 percent. Losses are due to P_2O_5 reporting to slimes during attrition scrubbing and classification with a 1 percent loss to dust and shrinkage. The plant operating availability is 85 percent based on 7446 operating hours per year.

Figure 1-5 is a layout of the Bayovar 12 plant showing the arrangement of equipment showing from ore feed to concentrate loadout. The figure shows the power line coming to the main plant substation adjacent to Seawater Pond.

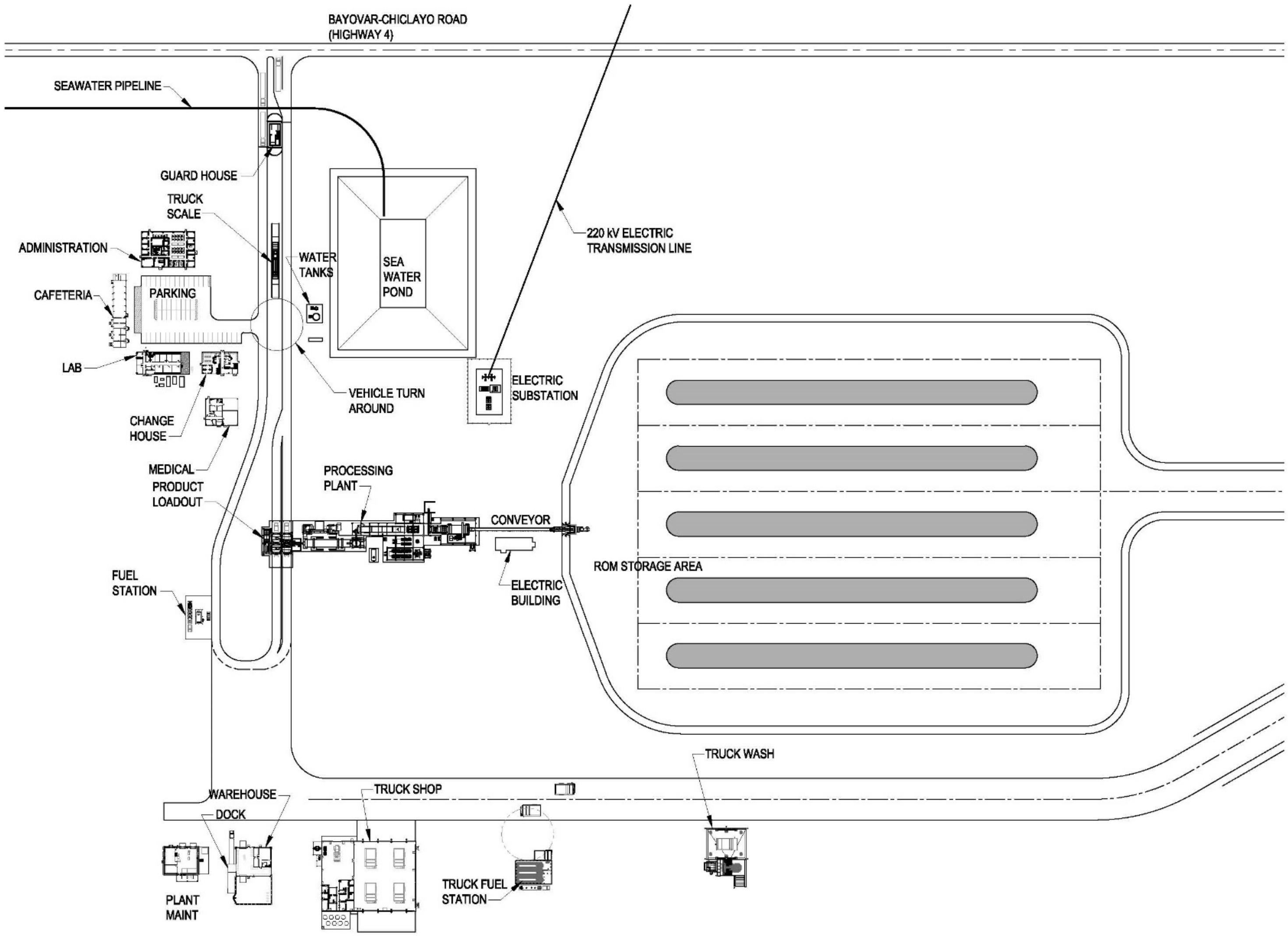


Figure 1-5: Bayovar 12 Process Plant and Facilities

1.12 INFRASTRUCTURE

The infrastructure for the Bayovar 12 Project includes site access and concentrate haulage roads, a port facility for shipping concentrate overseas, power supply and a new power transmission line, a seawater pipeline for process water supply, ancillary building facilities, a reverse osmosis water treatment plant to deliver desalinated and potable water for human consumption, fire protection and sanitary septic facilities, site communications, and the Tailings Storage Facility (TSF).

1.12.1 Site Access & Concentrate Haulage

Access to the Bayovar 12 plant is from the Bayovar-Chiclayo Highway, which is located approximately 17 km west of the Pan American Highway.

The proposed plan for the Bayovar 12 mine is to haul concentrate to the Port of Bayovar along the Bayovar-Chiclayo Highway over a distance of approximately 43 km to the port facility owned by JPQ shareholders using contractor-owned, 35-tonne end-dumping trailers.

1.12.2 Concentrate Shipping

The project is planning to use the port facilities owned by its partner, JPQ, located in the Port of Bayovar. The facility is a medium depth port that handles small Handysize ships that have a capacity of 20,000 long tons deadweight (DWT) – 28,000 DWT and Handysize ships, with a capacity of 28,000 – 40,000 DWT. The port has been actively shipping gypsum for JPQ from open stockpiles.

1.12.3 Power Transmission and Main Substation

Utility power is available from a 220 kV power transmission line which runs parallel to the route of the Pan American Highway. To supply power to the Bayovar 12 plant, a 220 kV 16-kilometer power transmission line from the La Niña substation to the Bayovar 12 main substation will need to be constructed. It would be stepped down at the main plant substation to the distribution voltage of 13.8 kV.

1.12.4 Seawater Supply System

The Bayovar 12 process plant operates using seawater that will be pumped at a maximum flow rate of 1,978 m^3/h. The seawater intake and vertical turbine pump platforms will be mounted on the existing JPQ ship loading conveyor structure.

The delivery points for the seawater is the plant Seawater Pondfrom which process water will be pumped to the various unit processes. A stream from the Seawater Pond will be taken to a reverse osmosis (RO) desalination plant, whose brine product will be discharged to the seawater ponds. The current design calls for a 36" HDPE seawater pipeline running 45 km from the seawater intake pumps to the seawater ponds. The seawater pipeline will run in an open trench parallel to the Bayovar-Chiclayo highway.

1.12.5 Water Desalination System

A packaged 4.5 m3/h (20 gpm) reverse osmosis plant has been included to supply desalinated potable water for human use and consumption for each. These plants can each produce 108 m3/d. The sizing of the RO plant is based on a consumption of 100 gallons per day per person for approximately 300 people on site per day at full production. Each RO plant is a single pass, 8-membrane system based on a fresh water (permeate) recovery of 50% and a salt rejection of 99.5%.

1.12.6 Tailing Disposal

The Bayovar 12 TSF will consist of approximately 9.8 Mm2 of available storage area as shown in Figure 1-6. The TSF is designed so that tailings can be impounded to a maximum height of 6.0 m, with spillway discharge and 1.0 meter freeboard.

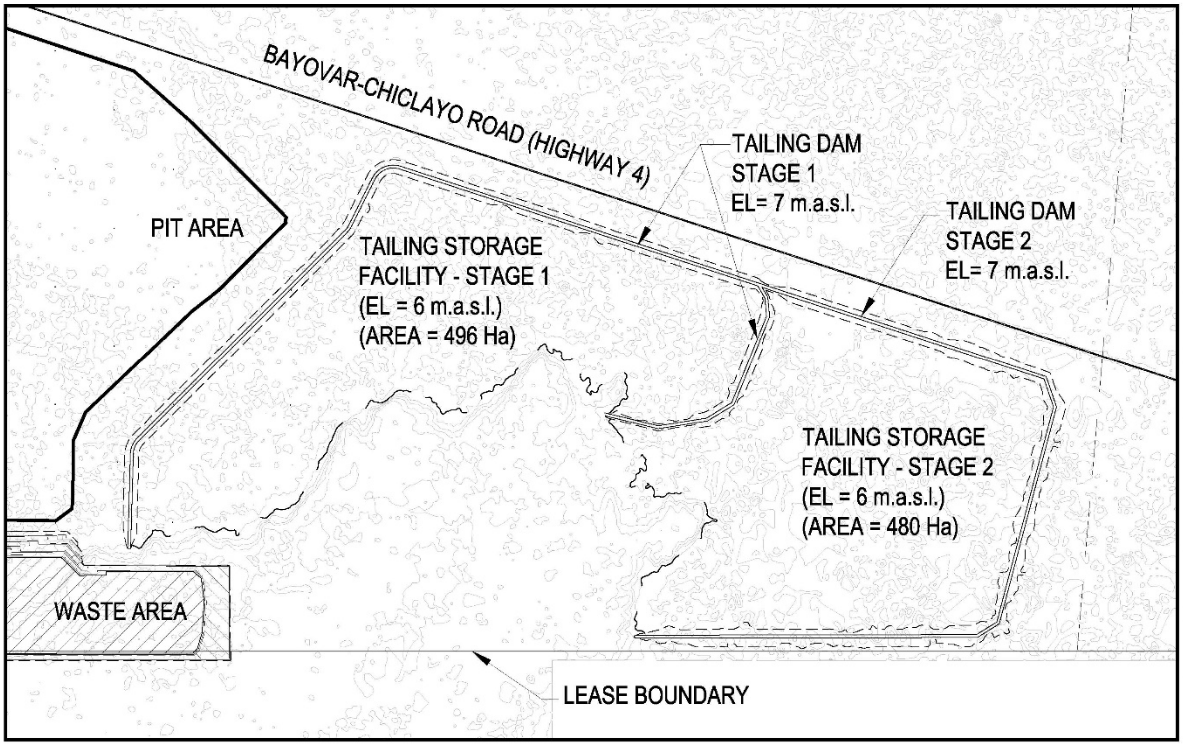


Figure 1-6: Tailing Storage Facility

1.12.7 Surface Water Management

Surface water diversions will be necessary for occasional rains and during El Niño years. Most surface water will report to the open pit where it will be channeled to collection sumps from where it will be pumped to the seawater or tailings ponds.

Diversion ditches and associated culvert systems, as well as ponds, sumps and pipelines, will be designed to address the majority of surface water flow at the project site. Water diversions will also be needed along the length of the south side of the Bayovar-Chiclayo highway to protect the berms of the TSF from erosion to prevent breaching of the slimes in the impoundment.

1.13 MARKET STUDIES AND CONTRACTS

The Bayovar 12 project, located within the Sechura basin in northern Peru, hosts a world-class deposit of what is arguably the most reactive sedimentary phosphate rock fertilizer in the world (Reactive Phosphate Rock or "DAPR"). The project will produce a highly sought-after, organic DAPR fertilizer. This product will service the rapidly-growing oil palm market in the Americas, especially the organic palm oil plantations of Colombia and Ecuador, including the local Peruvian market where phosphate nutrient needs are currently met entirely by imports.

1.13.1 Prices Paid FOB Bayovar for DAPR

Two mines close to Focus' Bayovar project have been producing and selling DAPR for several years. Fosyeiki S.A.C. and Corporación Agrosechura Peru S.A.C. produce and export ~22-24% P_2O_5 rock by truck and ship to Ecuador, Nicaragua, Bolivia and Colombia. Fosyeiki's product is claimed to range from 22-24% P_2O_5 and its sales volumes appear to be limited by its production capacity. FOB prices by ship to Central America have ranged from US$163 per tonne to US$195 per tonne. FOB prices achieved by Agrosechura by truck to Bolivia and Ecuador have ranged up to US$220 per tonne.

1.13.2 Price Assumption for 24% P_2O_5 DAPR Product

Retail prices of DAPR to organic farmers in Peru have remained fairly constant at about US$320 per tonne for the past few years. In Malaysia, Bayovar rock is currently being marketed by Union Harvest for US$160-170 per tonne. This Prefeasibility Study assumes a wholesale price of US$145 per tonne for Focus' 24% P_2O_5 product.

1.13.3 Price Assumption for 28% P_2O_5 DAPR Product

Several studies conducted by governmental organizations over the past decade have shown that on plantations located in areas with acidic soils and high rain fall, Sechura DAPR competes agronomically with Super Single Phosphate (SSP), achieving similar or higher crop yields. Consequently DAPR and SSP tend to compete directly and prices are similar, as seen in countries such as Brazil.

Focus will therefore be looking to displace SSP sales in South and Central America with a cheaper but more effective product. This Prefeasibility Study assumes a wholesale price of US$185 per tonne for Focus' higher grade product, which represents approximately a 25% discount to SSP prices in Latin America.

1.14 CAPITAL AND OPERATING COSTS

1.14.1 Mining Capital Costs

The estimated capital expenditure or capital costs (CAPEX) for the Bayovar 12 Project consists of two components:

(1) The initial CAPEX to design, permit, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, and utilities. The initial CAPEX also includes indirect costs for engineering, construction management, and Owner's costs.

(2) The sustaining CAPEX for facilities expansions, mining equipment additions, replacements and re-builds, expected replacements of process equipment and ongoing environmental mitigation activities;

The capital cost estimates reported in this section address the construction of a phosphate beneficiation plant capable of producing one million tonnes of DAPR concentrate (dry basis) from two process lines at full production.

Capital requirements associated ore production include purchasing mobile mine equipment for stripping overburden, interburden material and to mine phosphorite beds. Preproduction Year -1 mining is capitalized, as are the construction of berms and lifts for the TSF.

All mining will be performed by Focus using company-owned equipment and company employees. The mine plan for the 1.0 Mtpy beneficiated rock concentrate production was estimated based on 20 years of production.

Capital expenditures incorporate all mining equipment costs, tailings facility and mine haul road development. The mining cost model includes costs for ongoing reclamation during the 20 years of operation, but did not take into account the expenses after closure (e.g., final reclamation and re-vegetation, building and infrastructure demolition, and haul road re-grade). Table 1-7 lists the major mine capital costs encountered of the Life-of-Mine.

Table 1-7: Mine Equipment Capital Expenditures

	Initial Capital Cost	Sustaining Capital Cost	Total Capital Cost
	($1000)	($1000)	($1000)
WIRTGEN 2500SM Continuous Miner (2.5 M)	$ 2,932	$ 2,932	$ 5,864
CAT 994 Front End Loader (31 CuM)	$ 17,116	$ 25,674	$ 42,790
CAT 777 Coal Haul Truck (90 tn)	$ 48,042	$ 21,902	$ 69,944
CAT D9/D10 Track Dozers	$ 2,285	$ 11,425	$ 13,710
CAT 834 Wheel Dozer (450 HP)	$ 1,028	$ 1,028	$ 2,056
CAT 16M Motor Graders (297 HP)	$ 1,696	$ 5,088	$ 6,784
CAT 770-W Water Truck (30,000 Ltr)	$ 698	$ 1,396	$ 2,094
CAT 336 Aux Loader (1 CuM)	$ 307	$ 307	$ 614
CAT 777 Aux Truck (90 tn)	$ -	$ -	$ -
CAT CS-56 Compactor (147 HP)	$ 226	$ 904	$ 1,130
Subtotal Major Mining Equipment	$ 74,330	$ 70,656	$ 144,986
Subtotal Major Mining Equipment	$ 6,729	$ 4,746	$ 11,475
Shop Tools (3% of Major Equipment)	$ 2,230	$ 2,842	$ 5,072
Initial Spare Parts (5% of Major Equipment @ New Purchase Price)	$ 3,717	$ 4,737	$ 8,454
Contingency (5% of All Mine Equipment)	$ 4,053	$ 4,975	$ 9,028
TOTAL EQUIPMENT/FACILITIES CAPITAL	$ 91,058	$ 87,956	$ 179,014
Initial Capital 20%	$ 18,212	$ 160,802	$ 179,014
Pre-stripping Capital Costs	$ 41,175	$ -	$ 41,175
Total Mine Capital Cost	$ 59,387	$ 160,802*	$ 220,189

*The total of sustaining capital does not include interest payments which are expensed.

1.14.2 Plant Capital Costs

The Bayovar 12 beneficiation plant has been designed to produce 1 million tonnes per annum of 24% Mode A & 28% Mode B DAPR, for an initial plant capital cost is estimated to be $95.7 million which includes 20% contingency but does not include mining, pre-stripping, Owner's cost or IGV (Table 1-8).

The sustaining capital cost for the Bayovar 12 plant is based on an unspecified $1 million per year for 20 years totaling $20 million. This sustaining capital cost is in addition to annual maintenance, services, and supplies that are components of the operating cost estimate. The unspecified sustaining capital costs would cover replacements of the seawater pumps and slurry pumps, replacement of the vacuum pump for the belt filter, relining of the concentrate dryer, and similar unscheduled equipment replacements.

Initial CAPEX includes an estimate of contingency based on the accuracy and level of detail of the cost estimate. The purpose of the contingency provision is to make allowance for uncertain cost elements which are predicted to occur, but are not included in the cost estimate. These cost elements include uncertainties concerning completeness and accuracy of material takeoffs, accuracy of labor and material rates, accuracy of labor productivity expectations, and accuracy of equipment pricing.

Table 1-8: Initial Capital Cost Summary for Process Plant

Area	Plant Cost ($000s)
General Site Costs	2,548
ROM Dump Pocket and Feed Conveyor	2,044
Drum Washing and Desliming	4,394
Attrition Scrubbing	5,159
Concentrate Filtration	2054
Concentrate Drying and Loadout	6,978
Tailings Line	2,182
Seawater Supply	17,102
Desalination RO Plant & Firewater Supply	966
Power Transmission Line and Main Substation	5,158
Ancillaries	12,220
Direct Cost	$60,815
Contractor Indirects	3,039
EPCM Services	9,209
Commissioning and Vendor Reps	666
Capital & Commissioning Spare Parts & Initial Fills	1,467
Freight, Duties	4,604
Indirect Cost	$19,024
Contingency (Process Plant) at 20%	15,728
Total	$95,567

1.14.3 Tailing Storage Facility Capex

The TSF will be built in 2 stages. Each stage includes an estimate for incremental hauling of waste beyond the centroid of the waste, a cost for placing, spreading, and compacting the embankment fill. In Year -1, there is also a cost associated with constructing a berm to protect the open pit from inflows from tailings. These capital costs are summarized below in Table 1-9.

Table 1-9: TSF Capex by Stage

Item	Year	Cost ($000s)
Stage 1	-1	$10,255
Stage 2	5 & 6	$12,760
Total TSF Cost		$23,015

1.14.4 Owner's Costs

Owner's costs were estimated for several categories based on approximately 12 months of field construction for the mine and plant site. Owner's costs include Owners staff build-up and field expenses, staff hiring and training, Owner's commissioning team, construction insurance, environmental, legal and community development costs. The total estimated costs for Owner's costs is $2,508,000.

1.14.5 Capital Cost Summary

Table 1-10 summarizes the initial and sustaining capital costs for the Bayovar 12 Project over 20 years of production.

Table 1-10: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Total CAPEX ($000s)
Capex	Mine	59,387	160,802	220,189
	Processing Plant	95,567	20,000	115,567
	TSF	10,255	12,760	23,015
Owner's Costs		2,508	2,508	0
Total CAPEX with Contingency		$167,716	$193,562	$361,279

1.15 OPERATING COSTS

1.15.1 Mine Operating Costs

Operating costs of phosphate ore production including overburden and interburden stripping costs $39.72/tonne of concentrate see Table 1-10.

Overburden stripping, interburden stripping and phosphorite mining will be performed by Focus using company-owned equipment and company employees The mine cost model assumes the pre-production in Year -1 and that all pre-production mining is capitalized. After start-up, all overburden waste stripping will be expensed as an operating cost.

The mine plan for the 1.0 million mtpa phosphate concentrate production was estimated based on 20 years of production. The estimates encompassed all costs associated with all mining, phosphate rock and overburden handling, phosphorite stockpile processing, and other mine support services required for the delivery of phosphorite ore to the beneficiation plant. The operations support includes estimates for road grading, scraping, dust suppression, haul road maintenance, and other miscellaneous support activities. The supervision and administration function encompasses the cost of salaried supervisory and administrative personnel stationed at the mine and pickup truck fleet operations and maintenance.

The mine operating cost build up includes: wages for salaried and hourly labor, detailed equipment costs per hour of operation, hours estimates based on mining rate and haulage profiles, estimates for equipment maintenance consumables and labor hours, infill drilling costs, dewatering costs, and road construction costs.

Table 1-11: Summary of Mine Operating Costs

	Totals By Dry Bank Cubic Meters (x1000)				Totals By Dry Bank Tonnes					Mine Operating Cost - Total Dollars ($USx1000)							Totals By Dry Tonne		Totals By Bank Cubic Meter		Totals By Dry Product Tonne
Mining Year	Ore Dry (k bcm)	Interburden Dry (k bcm)	Overburden Dry (k bcm)	TOTAL Dry (k bcm)	Ore Dry (k tonne)	Interburden Dry (k tonne)	Overburden Dry (k tonne)	TOTAL Dry (k tonne)	Product Dry (tonne)	Loading ($x1000)	Hauling ($x1000)	Auxiliary ($x1000)	General Mine ($x1000)	General Maint. ($x1000)	Mine G&A ($x1000)	TOTAL ($x1000)	Cost Per ORE ($/kt)	Cost Per MINED ($/kt)	Cost Per ORE ($/bcm)	Cost Per MINED ($/bcm)	Dry PRODUCT ($/dry prod tne)
-1	298	5,632	21,642	27,572	364	4,280	19,694	24,338	137,189	8,566	24,893	2,446	1,714	1,318	2,239	41,175	113.158	1.692	138.052	1.493	300.135
1	1,332	13,605	13,648	28,586	1,625	10,340	12,420	24,385	623,152	8,912	25,685	3,073	1,744	1,358	2,317	43,089	26.509	1.767	32.342	1.507	63.886
2	2,529	13,057	11,915	27,501	3,085	9,924	10,843	23,851	1,022,848	8,983	26,867	3,714	1,712	1,334	2,360	44,970	14.578	1.885	17.785	1.635	42.933
3	2,649	16,124	8,914	27,687	3,231	12,255	8,112	23,598	1,123,985	8,970	23,430	4,357	1,717	1,328	2,310	42,113	13.033	1.785	15.901	1.521	37.310
4	2,410	16,075	8,871	27,356	2,940	12,217	8,072	23,230	958,362	8,806	20,329	4,366	1,707	1,305	2,255	38,769	13.185	1.669	16.086	1.417	40.254
5	2,411	12,621	12,058	27,090	2,941	9,592	10,973	23,506	1,007,201	8,841	23,289	4,360	1,699	1,310	2,308	41,806	14.214	1.779	17.341	1.543	41.508
6	2,163	10,657	14,291	27,111	2,639	8,099	13,005	23,743	951,043	8,835	26,072	4,357	1,700	1,323	2,360	44,647	16.918	1.880	20.639	1.647	45.740
7	2,447	13,876	11,892	28,214	2,985	10,546	10,822	24,352	1,020,179	9,139	23,081	4,360	1,733	1,343	2,305	41,961	14.058	1.723	17.151	1.487	41.131
8	2,640	16,507	8,451	27,598	3,221	12,545	7,690	23,457	1,214,810	8,932	21,631	4,364	1,715	1,319	2,280	40,241	12.493	1.716	15.242	1.458	32.998
9	2,297	13,622	10,663	26,582	2,802	10,353	9,703	22,858	849,309	8,627	21,127	4,364	1,684	1,285	2,269	39,356	14.046	1.722	17.136	1.481	46.339
10	2,237	11,583	12,422	26,241	2,729	8,803	11,304	22,835	1,029,868	8,576	23,246	4,361	1,674	1,284	2,308	41,449	15.190	1.815	18.532	1.580	40.192
11	2,502	15,931	8,802	27,235	3,052	12,108	8,010	23,170	1,097,207	8,800	23,468	4,358	1,704	1,314	2,310	41,954	13.746	1.811	16.770	1.540	38.237
12	2,507	10,647	14,728	27,882	3,058	8,092	13,402	24,553	1,112,908	9,166	25,430	4,356	1,723	1,343	2,349	44,367	14.506	1.807	17.698	1.591	39.776
13	2,487	15,278	9,987	27,752	3,035	11,611	9,088	23,734	1,007,461	8,973	24,908	4,367	1,719	1,342	2,345	43,655	14.386	1.839	17.551	1.573	43.332
14	2,333	12,822	11,599	26,754	2,847	9,744	10,555	23,146	1,067,876	8,716	27,080	4,365	1,689	1,321	2,383	45,554	16.002	1.968	19.522	1.703	42.392
15	2,643	17,571	7,602	27,816	3,224	13,354	6,918	23,496	1,090,389	8,959	23,096	4,361	1,721	1,332	2,305	41,774	12.958	1.778	15.808	1.502	38.311
16	2,479	9,835	17,073	29,387	3,024	7,475	15,537	26,035	1,188,762	9,624	28,326	4,356	1,768	1,404	2,405	47,882	15.834	1.839	19.318	1.629	38.632
17	2,495	14,621	11,447	28,563	3,043	11,112	10,417	24,572	975,929	9,226	26,980	4,353	1,743	1,369	2,376	46,047	15.130	1.874	18.459	1.612	47.183
18	1,981	11,606	10,488	24,075	2,417	8,820	9,544	20,782	968,935	7,866	18,854	4,366	1,609	1,197	2,225	36,117	14.943	1.738	18.231	1.500	36.743
19	2,606	18,319	26	20,951	3,179	13,922	24	17,125	1,137,438	6,819	16,868	4,365	1,515	1,091	1,716	32,374	10.184	1.890	12.425	1.545	28.404
20	2,730	14,758	0	17,488	3,331	11,216	-	14,547	1,110,796	5,868	10,322	4,375	1,352	915	975	23,808	7.148	1.637	8.720	1.361	21.390
TOTAL	48,174	284,748	226,519	559,441	58,772	216,408	206,133	481,313	20,695,647	181,204	484,982	87,742	35,342	27,135	46,701	863,108	14.686	1.793	17.917	1.543	39.72
									PERCENT	21.0%	56.2%	10.2%	4.1%	3.1%	5.4%	100.0%					

1.15.2 Plant Operating Costs

1.15.2.1 Process Plant Operating & Maintenance Cost

The process plant operating costs are summarized by area and then by cost element of labor, electric power, reagents, maintenance parts and supplies and services. Below in Table 1-12 is a summary of the cost by area for a typical operating year at full production. Note that the typical year plant operating cost is not the same as the LOM average which is $8.01/tonne of product.

Table 1-12: Process Operating Cost Summary

LOM Operating Summary		
Product – 24% DAPR	518,300 t/yr	
Product – 28% DAPR	516,500 t/yr	
LOM Average Process Plant Production		
Process Area	Annual Cost	Unit Cost per Tonne
ROM Stockpile	$314,883	$0.30
Washing & Scrubbing	$1,796,027	$1.74
RO Plant	$139,060	$0.13
Concentrate, Tailings	$4,983,381	$4.82
Ancillary	$1,053,984	$1.02

1.15.3 Life of Mine Operating Costs

Life-of-Mine (LOM) operating costs have been developed for mining, processing and general & administrative (G&A) costs. Operating costs include labor, equipment operation, power, fuel, reagent, and consumable consumption, maintenance and repairs, and outside services. Operating cost build-ups are described in the sections below. Table 1-13 summarizes the LOM operating costs that were derived from the financial model.

Table 1-13: Life-of-Mine Operating Costs based on 20,696,000 tonnes of DAPR concentrate

Item	Unit Cost ($USD per Product Tonne)	LOM Cost ($000s)
Mining	$39.72	$821,932
Process Plant	$8.01	$165,747
G&A	$2.38	$49,350
Transportation	$10.09	$208,819
Total Cost	$60.20	$1,245,848

1.15.4 Economic Analysis

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine (LOM) based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of phosphate ore. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

1.15.5 Plant Production Statistics

The process plant has two operational modes capable of producing 1,370 tonnes per day each of concentrate at 85% availability. The product is categorized into two grade 24% and 28%. Presented below in Table 1-14 is the life of mine production for both products.

Table 1-14: Life-of-Mine Production

	Thousand tonnes
Phosphate concentrate - 24% DAPR	10,366
Phosphate concentrate - 28% DAPR	10,329

1.15.6 Marketing Terms

The phosphate production is assumed to be shipped to end consumer and the terms are negotiable at the time of the agreement. The financial analysis presented here does not consider any deductions or penalties are being accessed. The product is priced to be shipped FOB, Port of Bayovar, Peru.

1.15.7 Capital Expenditures

1.15.7.1 Initial Capital

The financial indicators have been determined with equity financing for the initial capital. Any acquisition cost or expenditures prior to start of the full project period have been treated as “sunk” cost and have not been included in the analysis.

The total initial capital carried in the financial model for new construction and pre-production mine development is expended over a 2-year period. The initial capital includes Owner’s costs and contingency. The cash flow will be expended in the years before production.

The initial capital is presented in Table 1-15.

Table 1-15: Initial Capital

	$ in millions
Mining (includes preproduction)	$59.4
Process Plant	$95.6
Owner's Cost	$2.5
TSF	$10.3
Total	$167.7

1.15.8 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital (Table 1-16). Included in the sustaining is capital for the replacement of the mining fleet, unspecified plant upgrades, and an increase in the TSF capacity. The total LOM sustaining capital is estimated to be $193.5 million. This capital will be expended during an 20-year period.

Table 1-16: Sustaining Capital

	$ in millions
Mining	$160.8
Process Plant	$20.0
TSF	$12.8
Total	$193.6

1.15.9 Revenue

Annual revenue is determined by applying phosphate prices to the annual product by grade for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. Prices used in the evaluation are as follows:

- Phosphate Ore - 24% DAPR $145.00/tonne
- Phosphate Ore – 28% RPF $185.00/tonne

1.15.10 Total Production Cost

The life of mine Production Cost over the life of the mine is estimated to be $69.83 per tonne of product being sold, excluding the cost of the capitalized pre-stripping. The Production Cost includes mine operations, process plant operations, general administrative cost, corporate overhead, shipping charges, royalties, and closure/reclamation and salvage value. Table 1-18 shows the estimated production cost by area per metric ton of product sold.

Table 1-17: Total Production Cost

Operating Cost	US$/tonne product sold	LOM Cost ($millions)
Mining	$39.72	$821.9
Process Plant	$8.01	$165.7
General & Administration	$2.38	$49.4
Transportation	$10.09	$208.8
Total	$60.20	$1,245.8
Royalty	$9.07	$187.8
Interest	$0.98	$20.3
Reclamation/Closure	$0.06	$1.3
Salvage Value	($0.48)	($10.0)
Total Production Cost	$69.83	$1,445.2

1.15.11 Royalty and Export Duties

The royalty basis is 3.5% of gross revenues to the Peruvian government plus 2% to Radius Gold. A royalty is estimated at $187.8 million for the life of the mine.

1.15.12 Reclamation and Closure

Much of reclamation is going to be concurrent with mining from the backfilling of the open pit with waste. An allowance for reclamation and closure was included in the cash flow of $1.26 million for the life of the mine to cover monitoring. Reclamation will be limited to removal of structures that will be offset by sale of equipment and structures from the plant.

1.15.13 Salvage Value

At end of the mine life an estimated salvage value was shown of $10.0 million has been included mainly from salvaging the large diameter HDPE pipe used for the seawater supply line, and the tailings line.

1.15.14 Taxation

1.15.14.1 Depreciation

Ten-year straight line method for depreciation has be used for both initial and sustaining capital.

1.15.14.2 Income Tax

A corporate income tax rate of 26% was included in the economic model. This is applied to net profits of the company. Income taxes paid are estimated to be $418.1 million.

1.15.14.2 Value Added Tax

Value added tax (IGV) is levied on the supply of goods and services subject to the tax. The financial model applies an 18% rate and IGV tax is also reimbursed and it is assumed that the IGV paid and IGV recovered are the same; it is shown in the working capital section.

1.15.14.3 Project Financing

The project was based on a combination of equity and debt financing.

1.15.14.4 Net Income after Tax

Net Income after Tax amounts to $1,189 million.

1.15.14.5 NPV and IRR

The NPV calculation includes Years 1 through 20 and adds the pre-production capital in Years -2 and -1. The economic analysis indicates that the project has an after-tax Internal Rate of Return (IRR) of 26.3% with a payback period of 3.9 years and a Net Present Value at 7.5% of $457.7 million.

The sensitivity analysis shown in Table 1-18 compares the project discounted cash flow, IRR, and payback period against the base case when the commodity prices, initial capital and operating cost are varied. The project is most sensitive to variation to the commodity prices; while the initial capital and operating costs are similar.

Table 1-18: Price Sensitivities after Taxes

Commodity Price Sensitivity after Taxes (costs in $000's)			
	NPV @7.5	IRR	Payback
20%	$693,384	34.7%	2.9
10%	$575,562	30.6%	3.3
Base Case	$457,741	26.3%	3.9
-10%	$339,919	21.9%	5.0
-20%	$222,098	17.2%	6.2
Operating Cost Sensitivity after Taxes			
	NPV @7.5	IRR	Payback
20%	$362,258	22.5%	4.8
10%	$409,999	24.4%	4.3
Base Case	$457,741	26.3%	3.9
-10%	$505,482	28.2%	3.6
-20%	$553,223	30.1%	3.3
Initial Capital Sensitivity after Taxes			
	NPV @7.5	IRR	Payback
20%	$432,749	23.5%	4.4
10%	$445,245	24.8%	4.2
Base Case	$457,741	26.3%	3.9
-10%	$470,237	28.0%	3.7
-20%	$482,733	30.0%	3.4

2 INTRODUCTION

2.1 PURPOSE

In 2015, Focus Ventures Ltd (FCV) commissioned M3 Engineering and Technology Corporation (M3) to prepare a prefeasibility study (PFS) for the Bayovar 12 Phosphate Project (the Project) in Piura Department, Peru. The Project Contains a large, open pittable, soft rock phosphate resource that is areally extensive across Bayovar Concession #12 in NW Peru.

The original PFS compiled the results of mineral resource estimates, metallurgical studies, mine engineering, process/plant engineering, engineering of infrastructure, environmental and social studies, development of capital and operating costs, and an indepth financial model. The results were reported in a January 5, 2016 press release and the PFS was published in February 2016. The technical report conformed to Canadian National Instrument 43-101 (NI 43-101) and summarized the outcomes of an engineering study completed by several authors to Prefeasibility Study (PFS) standards.

The original PFS was based on producing two direct application phosphate rock (DAPR) products, one at 24% P_2O_5 and the other at 28% P_2O_5 in parallel process lines that were each capable of producing 500,000 tonnes of DAPR per year (mtpa). The first process line started at a reduced capacity of 300,000 mtpa in Year 1 ramping up to full production in Year 4. The second process line commenced at full production (500,000 mtpa) in Year 3.

The results of the original PFS, while not unacceptable by comparative standards, were deemed by FCV to be inadequate to obtain attractive project financing so it undertook a new change in scope focusing on the mine and plant in order to improve the project economics. The mineral resources remained unchanged but a new mine plan was developed to shorten waste rock hauls while making the 13 phosphate beds more accessible. The two process lines were combined into a single process line having a capacity of 1 million mtpa. The same two products, 24% P_2O_5 DAPR and 28% P_2O_5 DAPR, will be batch-produced from the single process line. The new mine plan and project design were estimated for capital and operating costs, and an updated financial model was developed. This report is the PFS Update to the original Bayovar 12 Phosphate Project PFS report (Effective Date December 18, 2015).

The Project is located in the Sechura desert, a marine depression near the coast of NW Peru. The mine consists of selective open pit mining of phosphate beds in a sequence of diatomite beds of Miocene age. The PFS Update investigates various aspects of the Project including the land status, mineral resource development, mining methodology, metallurgy and process methodology, geotechnical and hydrological characterization, plant engineering, environmental engineering, capital and operating cost estimation and financial analysis to determine whether the Bayovar 12 Project is technically, commercially, socially, and environmentally feasible.

FCV began work on the Bayovar 12 concession, one of several mineral concessions in the Sechura desert in 2013.

The mineral resource estimate was completed by Golder Associates (Golder), a global consulting company with a participating office in Calgary, Alberta. The mineral resources are based on 62 drill holes a have been prepared in accordance with NI 43-101 (Golder, 2015).

To-date, the following have been completed:

- Drilling of 62 exploration holes for a total of 5,971 meters of exploration drilling;
- Developed mineral resource estimates for Measured, Indicated, and Inferred, according to CIM definitions and compliant with NI 43-101 reporting requirements,

- Conducted various metallurgical testwork to determine yields, recoveries, and concentrate grades of the phosphate beds using standard beneficiation processes, settling rates, filtration rates, and other testwork;

- Developed process flowsheets, mass balances and process design criteria to make 24% P_2O_5 DAPR and 28% P_2O_5 DAPR,

- Designed a site layout that minimizes sterilizing mineral resources, optimizes capital cost expenditures for one process line produce two reactive phosphate rock (DAPR) products at a combined rate of 1 mtpa.

- Developed waste rock strategies to minimize haul distances, while opening up areas for in pit disposal of waste rock at the earliest possible time.

- Conducted geotechnical and hydrogeological investigations including drilling to determine the conditions for open pit design and TSF construction;

- Developed an open pit mine design and Life-of-Mine extraction schedule to optimize the current resources that extends for 21 years,

- Developed a phased Tailings Storage Facility (TSF) to minimize initial capital costs that covers the current Life of Mine;

- Developed base line environmental conditions, in accordance with the preparation of an EIA;

- Investigated the social license requirements for operating a mine and plant facility on the Bayovar 12 concession;

- Designed a seawater intake and 45-km water supply pipeline system to the Bayovar 12 mine site from the Port of Bayovar.

- Organized the site plan with respect to plant ancillary buildings to accommodate the new design;

- Incorporated a power transmission line from the La Niña substation to the plant site;

- Developed capital and operating costs based on capital equipment and material supply prices, labor rates, fuel prices, current consumable costs; and other standard industry metrics;

- Prepared a financial model based on all of the above.

This PFS Update incorporates the results of an updated resource estimation performed in an NI 43-101 October 2015 by Golder Associates.

2.2 SOURCES OF INFORMATION

This Report is the product of technical contributions from a number of consultants; together with FCV personnel. Listed below are the primary “Qualified Persons” (as defined in the National Instrument 43-101) that compiled different sections of the report. Table 2-1 describes the primary contributors by section.

- Conrad Huss, P.E., M3 Engineering & Technology will be the principal Qualified Person (“QP”) and author of the study.

- Jerry DeWolfe, P.Geo. Golder Associates (Golder) – Geological Modeling and Mineral Resource Estimation

- Herb Welhener, SME RM, Independent Mining Consultants – Mine Engineering and Mineral Reserve Determination;
- Glenn Gruber, Phosphate Beneficiation (BF) – Metallurgical Testwork
- Tom Drielick, P.E., M3 – Process Engineering
- Ted Minnes, P.E., Golder – Environmental and Social Studies

Table 2-1: List of Contributing Authors

Section	Section Name	Main Contributor	Qualified Person
1	Summary	M3, PB, IMC & Golder	Conrad Huss – M3, Jerry DeWolfe – Golder Herb Welhener – IMC Glenn Gruber – PB
2	Introduction	M3	Conrad Huss – M3
3	Reliance on Other Experts	M3	Conrad Huss – M3
4	Property Description and Location	FCV	Conrad Huss – M3
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	FCV	Conrad Huss – M3
6	History	FCV	Conrad Huss – M3
7	Geological Setting and Mineralization	FCV	Jerry DeWolfe – Golder
8	Deposit Types	FCV	Jerry DeWolfe – Golder
9	Exploration	FCV	Jerry DeWolfe – Golder
10	Drilling	Golder	Jerry DeWolfe – Golder
11	Sample Preparation, Analyses and Security	PB & Golder	Jerry DeWolfe – Golder
12	Data Verification	Golder	Jerry DeWolfe – Golder
13	Mineral Processing and Metallurgical Testing	PB	Glenn Gruber –PB
14	Mineral Resource Estimates	Golder	Jerry DeWolfe – Golder
15	Mineral Reserve Estimates	IMC	Herb Welhener – IMC
16	Mining Methods	IMC	Herb Welhener - IMC
17	Recovery Methods	M3	Tom Drielick – M3
18	Project Infrastructure	M3 & IMC	Conrad Huss – M3 & Herb Welhener – IMC
19	Market Studies and Contracts	FCV	Conrad Huss – M3
20	Environmental Studies, Permitting and Social or Community Impact	Golder	Ted Minnes – Golder
21	Capital and Operating Costs	M3 & IMC	Conrad Huss – M3 Herb Welhener – IMC
22	Economic Analysis	M3	Conrad Huss – M3
23	Adjacent Properties	FCV	Conrad Huss – M3
24	Other Relevant Data and Information	M3	Conrad Huss – M3
25	Interpretation and Conclusions	All	Conrad Huss – M3
26	Recommendations	All	Conrad Huss – M3
27	References	M3	Conrad Huss – M3

- Abbreviations: ALL – All QP Contributors; FCV – Focus Ventures Ltd; Golder – Golder Associates (both North America & Lima offices); M3 – M3 Engineering & Technology Corporation, IMC – Independent Mining Consultants; PB – Phosphate Benefication Ltd
- Note: Where multiple authors are cited, refer to author certificate (Appendix A) for specific responsibilities.

This Report has been compiled for FCV by M3, Golder, IMC, and Phosphate Beneficiation, collectively the Authors. The Report is based on information and data supplied to the Authors by FCV and other parties. The Authors have relied upon the data and information supplied by the various qualified persons listed above as being accurate and complete.

The Authors have relied on information provided by FCV and on information provided from previous studies. Where possible, the Authors have confirmed the information provided by comparison against other data sources, similar projects in Peru and South America or by field verification.

Where checks and confirmations were not possible, the Authors have assumed that all information supplied in the previous technical report is complete and reliable within normally accepted limits of error. During the normal course of the review, the Authors have not discovered any reason to doubt that assumption.

This Report conforms to the standards of a Prefeasibility Study and a NI 43-101 Technical Report. This Report, based on the work completed to date, is intended to summarize the work performed to date on the Project. This study evaluates the economics of the process plant to produce DAPR concentrate having a P_2O_5 grades of 24% & 28%, operating at a combined 1 million mtpa capacity for 20 years. The study sets forth conclusions and recommendations, based on the Authors' experience and professional opinion, which result from their analysis of work and data collected.

In accordance with the feasibility nature of the Report, M3 and the other contributors have used estimates and approximations based on experience and expertise. Where such estimates and approximations have been used, it is so noted and the assumptions made in making such estimates and approximations are so noted.

This Report should be construed in light of the methodology, procedures and techniques used for its preparation, and should be read in original context - all readers should refer to referenced documents for clarification of the original context.

2.3 SITE VISIT & PERSONAL INSPECTIONS

The following site visits were made by the groups and individuals listed below.

Conrad Huss, M3 Engineering and Technology (M3), Principal Author of the Bayovar 12 PFS Update has not visited the site. Stephen Simpson of M3 visited the Project site from October 1 thru 4, 2014. The primary focus of the site visit was to observe first-hand the project site, the Port of Bayovar and gain an understanding of potential issues related to the general infrastructure, power supply, process plant location options and availability of services required for plant operation.

Jerry DeWolfe, Associate and Senior Geological Consultant for Golder, visited the site from July 2 through July 5, 2014. The purpose of the visit was to familiarize the QP with the general geology of the area and detailed geology of the Bayovar 12 Phosphate Project property, review the project exploration history, verify drill hole locations in the field, perform a laboratory site visit, review available information and to discuss procedures and methods applied during the recent and historical exploration programs.

Herb Welhener, Vice President, Independent Mining Consultants, who is responsible for mineral reserves, mine engineering/planning, mine capital/operating cost development, and the tailings design, visited the Bayovar 12 site in February 2016.

Ted Minnes, Golder Associates, who is responsible for environmental, permitting, social licence and community relations studies has not visited the site. However, Martha Ly of Golder Associates, a specialist in environmental studies, permitting, social and community relations visited the site in October of 2015.

The units of production in this report are metric unless otherwise noted. Production is in tonnes (t). All monetary amounts are in 1st Quarter 2016 US dollars along with other variables such as the price of DAPR concentrate and phosphate fertilizer prices, unless otherwise noted.

Table 2-2: List of Acronyms

Abbreviation	Definition
AFPC	Association of Fertilizer and Phosphate Chemists
CIM	Canadian Institute of Mining
CRM	Certified Reference Material
DAP	Di-ammonium Phosphate
DAPR	Direct Application Phosphate Rock
DEM	Digital Elevation Model
DSM	Digital Surface Model
DWT	Deadweight Tonnage
EPCM	Engineering Procurement and Construction Management
ESIA	Environmental and Social Impact Assessment
FCV	Focus Ventures Ltd.
FOB	Free on Board
FOSPAC	Fosfatos del Pacifico
GA	General Arrangement
GMO	Genetically Modified Crops
ICP	Inductively Coupled Plasma
IFA	International Fertilizer Association
IFDC	International Fertilizer Development Center
IGV	Impuesto General a las Ventas (Peruvian value added tax)
INACC	Instituto Nacional de Concesiones y Catastro Minera
INDECOPI	Peruvian Government National Accreditation Service
INGEMMET	Instituto Geológico, Minero y Metalúrgico
IRR	Internal Rate of Return
JPQ	Juan Pablo Quay
LNG	Liquefied Natural Gas
M3	M3 Engineering & Technology Corporation
MAP	Mono-ammonium Phosphate
MGA	Merchant Grade Phosphoric Acid
MINAGRI	Peruvian Agriculture Ministry
NAC	Neutral Ammonium Citrate
NPV	Net Present Value
NSR	Net Smelter Return
PFS	Prefeasibility Study
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy.
QP	Qualified Person
ROM	Run of Mine
RPR	Reactive Phosphate Rock
RQD	Rock Quality Designation

Abbreviation	Definition
SSP	Single Super Phosphate
SUNARP	Superintendencia Nacional de Registros Públicos
TCR	Total Core Recovery
TSF	Tailings Storage Facility
TSP	Triple Super Phosphate
WSS	Water Soluble Salts

Table 2-3: Units of Measure

Unit Abbreviation	Definition
amsl	above mean sea level
cm	centimeter
d	day
dmt	dry metric tonne
ft	foot
g	gram
g/t	gram per tonne (metric)
gm/t	gram per tonne (metric); alternate spelling
h	hour
ha	hectare
hp	horsepower
HQ	core size
kg	kilogram
kg/t	kilogram per tonne (metric)
km	kilometer
km^2	square kilometers
kph	kilometers per hour
kW	kilowatt
kWh	kilowatt hours
kWh/t	kilowatt hours per tonne (metric)
m	meter
m/s	meter per second
m^2	square meter
m^3	cubic meter
mA	Milliampere
masl	Meters above sea level
min	minutes
mm	millimeter
mtpa	tonnes (metric) per annum
mtpd	tonnes (metric) per day

Unit Abbreviation	Definition
mtph	tonnes (metric) per hour
P	Chemical symbol for phosphorus
t	tonne (metric)
tpy	Tonnes per year
wmt	wet metric tonne
μm	micrometer

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs) for this report have relied upon certain reports, opinions and statements of legal and technical experts who are not necessarily considered “Qualified Persons”, as defined by NI 43-101. Reports received from other experts have been reviewed for factual errors by the relevant QPs and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this Report. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

3.1 MINING CONCESSIONS

Legal review concerning the status of mineral concessions covering the Bayovar 12 Project was conducted by Estudio Echecopar, a Peruvian law firm affiliated with the global firm, Baker & Mackenzie. The status report (February 9, 2015) with respect to the mining concessions on the Bayovar 12 Project property states that the claims are in good standing and that Agrifos, the Peruvian subsidiary of Focus Ventures, holds title in good standing. A description of the mineral tenure can be found in Section 4.2 of this report.

3.2 WATER SUPPLY

The updated water balance and water model prepared for this study will be used to prepare a technical document required to obtain the authorization and permit from the Water National Authority (ANA), designated approving authority. This is the regular procedure in Peru.

3.3 ENVIRONMENTAL AND PERMITTING

The environmental, socioeconomic and permitting studies presented in Section 20 of this Technical Report were prepared as part of the current Project by Martha Ly of Golder. Mrs. Ly is a senior environmental scientist with extensive experience in performing environmental, socioeconomic and permitting studies for similar mining projects in Peru and internationally.

The design and operating parameters incorporated in the 2015 Bayovar 12 Prefeasibility Study are expected to meet the requirements of an ESIA. Furthermore, as the environmental impact of the proposed Bayovar 12 operation has been engineered to minimize the impact on existing infrastructure including the highway, power lines and existing port facilities.

FCV’s plans for the Project are to focus on preparing an ESIA in 2016.

3.4 MARKETING STUDIES

Marketing studies were prepared for FCV by Integer Research Limited (2015), and Dr. Ricardo Melgar.

The research by Integer Research resulted in a report, Direct Application Phosphate Rock Market Study for Focus Ventures – Final Report.

Dr. Ricardo Melgar issued in 2015 the report, Use of Phosphoric Rock for Direct Application in southern South America Argentina – Chile - Uruguay – Paraguay.

3.5 TRANSPORTATION STUDIES

Land transportation rates and terms were provided by Mamut Peru, a local concentrate trucking company that is active in the Sechura-Bayovar, Peru area.

Port handling rates were provided FCV's partner and port operator, Juan Paulo Quay (JPQ), an operator of port terminals in Peru and Ecuador.

M3 also investigated concentrate transportation and port costs in the cities of Paita, Peru, and Callao, Peru, using RANSA, a logistics and freight forwarding company based in Callao, Peru.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Bayovar 12 Concession is located in the Sechura Province, Piura Region of northwestern Peru (Figure 4-1). The property is located approximately 950 km north of the Peruvian capital, Lima, 65 km south of the town of Sechura and 90 km southwest of Piura. The concession is approximately 40 km east of the fishing village of Puerto Rico, situated on the southern margin of Sechura Bay on the Pacific coast of Peru.

The Bayovar 12 Concession is located approximately 15 km northeast of Vale's operating Miski Mayo Phosphate Mine Figure 4-2. The Bayovar 12 Concession is directly east of the Fosfatos del Pacifico (FOSPAC) phosphate reserve area and directly south of the GrowMax Agri Corp. phosphate exploration properties.

The concession is connected by sealed road to tidewater and the JPQ marine port facilities 40 km to the west. The marine port facility is used by JPQ principally for the export of gypsum currently mined from the Bayovar 12 Concession and for phosphate rock produced locally from adjacent concessions. The JPQ port terminal was previously used to export phosphate rock extracted from the Bayovar Mine, prior to its acquisition by Vale. The JPQ marine port facilities are owned by Focus's Peruvian partners, Trabajos Maritimos S.A. and Inca Terminals and Mining Inc, who are marine transport and service providers. The marine port facility is used principally for the export of gypsum currently mined from the Bayovar 12 Concession and for phosphate rock produced locally from adjacent concessions. The JPQ port terminal was previously used to export phosphate rock extracted from the Bayovar Mine, prior to its acquisition by Vale.

The Pan-American Highway crosses the claim at its eastern end and power transmission lines for Vale's Bayovar Mine transect the Property at its northern end.

The following sections contain information relating to mineral titles, legal agreements as well as permitting and regulatory matters in Peru. The Golder Qualified Person is not qualified to verify these matters and has relied upon information provided by Focus including lease agreements and legal opinions concerning Focus' mineral and surface rights prepared by Asociado a Baker & McKenzie International, a Peruvian law firm, for the benefit of Focus.

4.2 MINERAL TENURE

Juan Paolo Quay S.A.C (JPQ) is the title holder of the Bayovar 12 mining concession (Cueva, 2015). On March 26, 2015, Focus (via Peruvian subsidiary Agrifos) acquired an outright 70% interest in the issued share capital of JPQ, by paying $4 million cash to the owners of JPQ. The remaining 30% interest in JPQ is owned by Trabajos Maritimos S.A. and Inca Terminals and Mining Inc., subsidiaries of Grupo Romero (Peru) and Mamut Andino C.A. (Ecuador), respectively.

The Bayovar 12 Concession comprises 12,575 hectares and was acquired by JPQ in 2007 under a contract with state company Activos Mineros S.A.C. for the exploitation of gypsum rock by open pit methods from the claim. The boundary node coordinates for the Bayovar 12 Concession are presented in Figure 4-3.

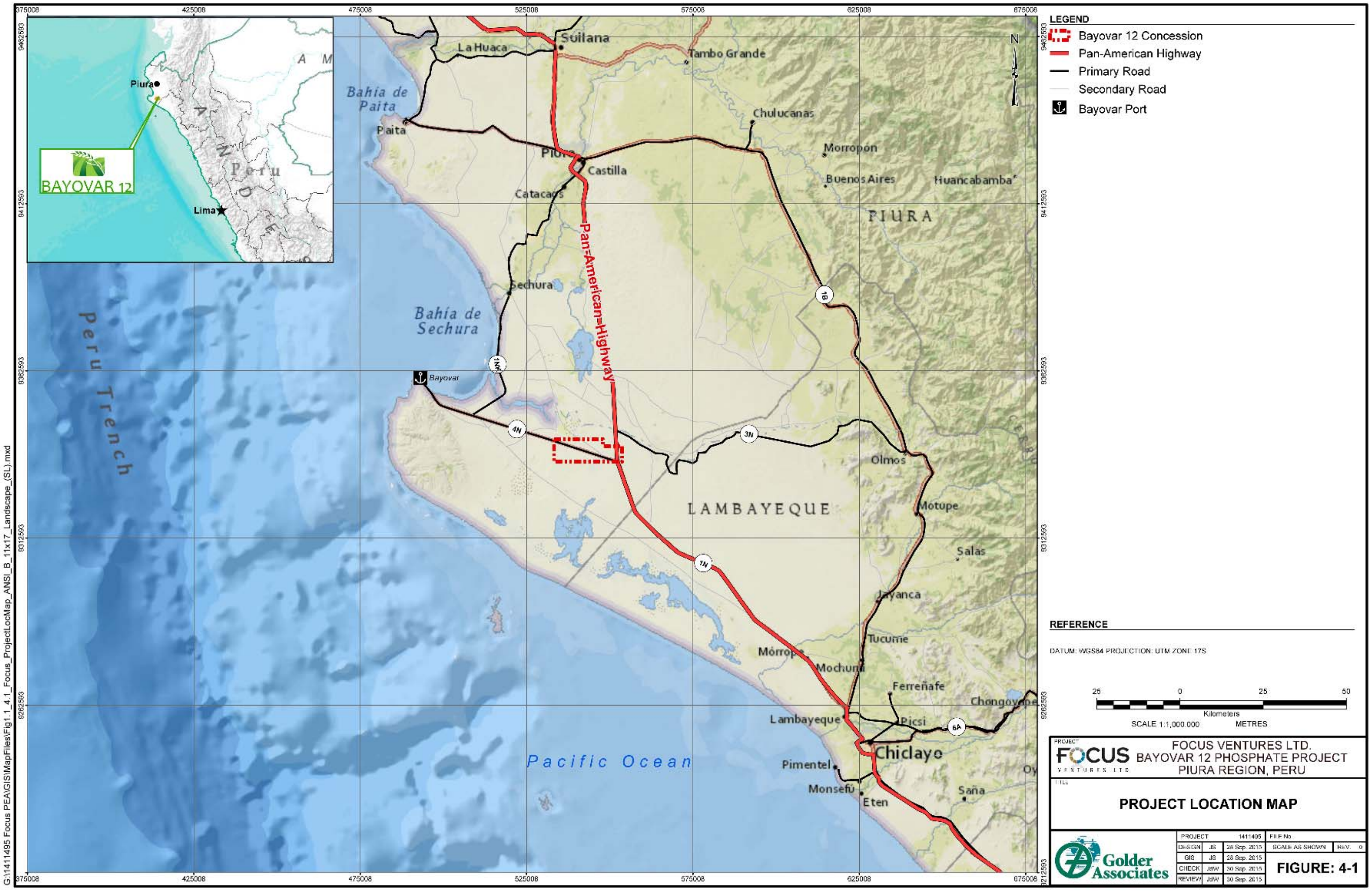


Figure 4-1: Project Location Map

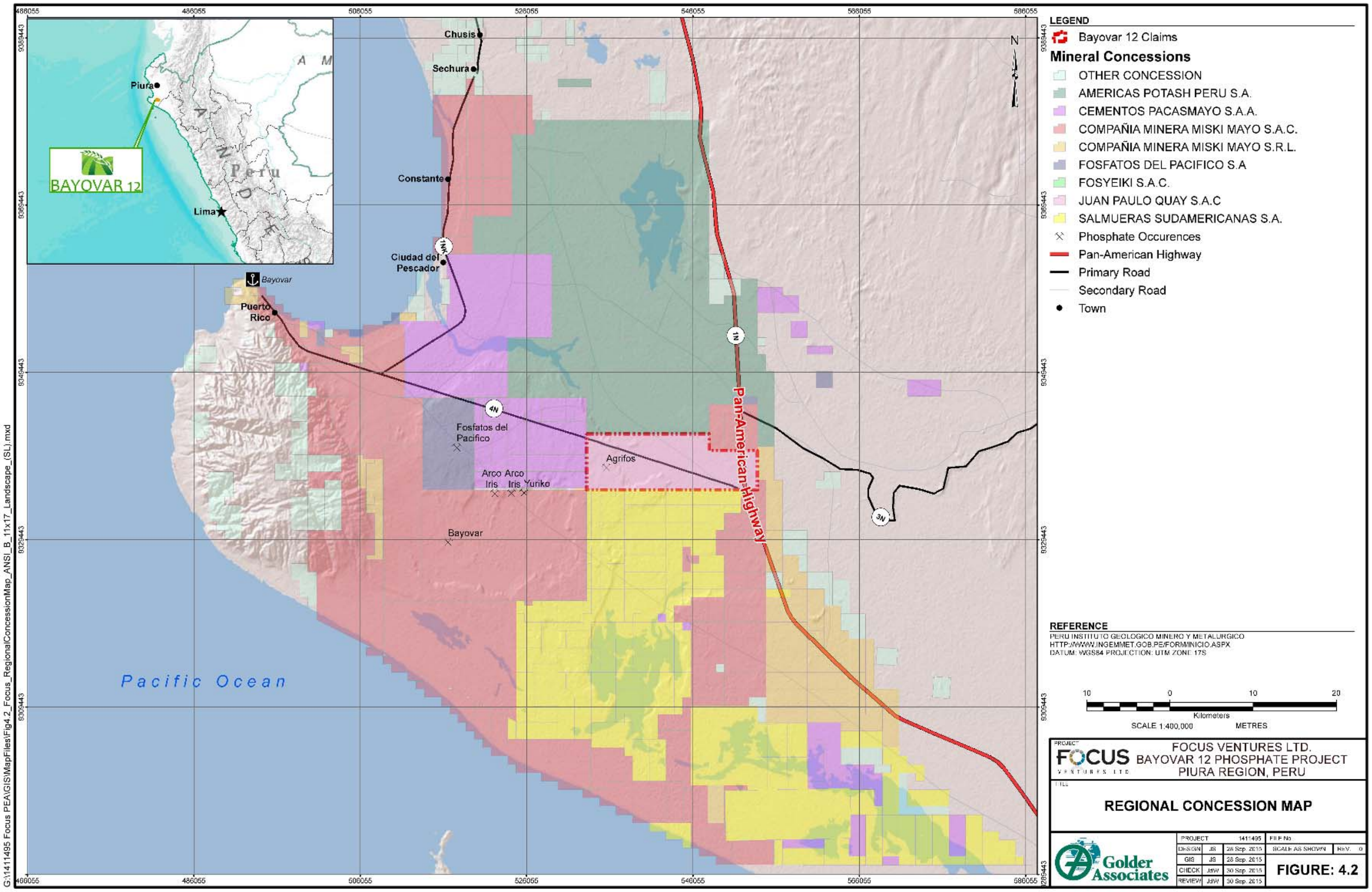


Figure 4-2: Regional Concession Map

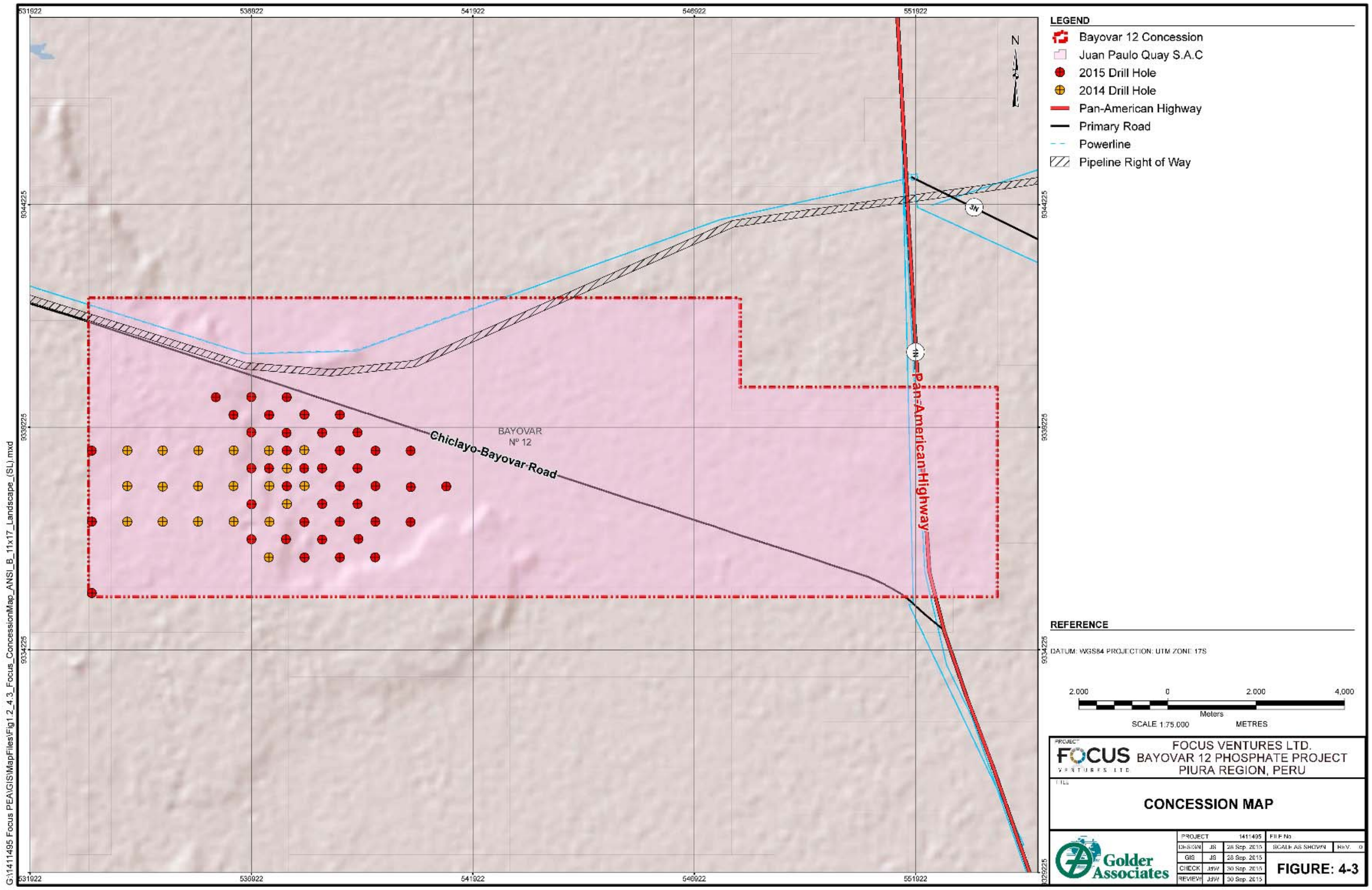


Figure 4-3: Concession Map

Table 4-1: Concession Boundary Coordinates

Lease Node	Easting (m)	Northing (m)
1	533,243	9,342,130
2	547,942	9,342,130
3	547,942	9,340,130
4	553,742	9,340,130
5	553,742	9,335,430
6	533,243	9,335,430

Note: UTM Zone 17S Projection, WGS84 Datum

In Peru, concessions are map-registered using a grid system at the Instituto Geologico, Minero y Metalurgico ("INGEMMET") and the Superintendencia Nacional de Registros Publicos ("SUNARP"). Concessions can be granted for either metallic or non-metallic minerals and allow both exploration and exploitation. Mining concessions are granted for an indefinite period; however, in order to maintain concessions in good standing, titleholders must pay a Mining Good Standing license fee equal to US$3.00 per hectare per year.

4.3 SURFACE RIGHTS

Under Peruvian law, holding a mining concession does not grant title for surface rights. JPQ was granted surface rights access for 99 years and a 30 year land use easement (renewable) under agreements signed with the community of San Martin de Sechura (Fundacion Comunal San Martin de Sechura).

4.4 AGREEMENTS AND ENCUMBRANCES

On January 14, 2014, Focus (via Agrifos), signed a formal option agreement for the acquisition of shares in JPQ, the titleholder of the Bayovar 12 mining concession, where Focus (via Agrifos) had the option to acquire a 70% interest in the issued share capital of JPQ by fulfilling the following;

- Completing a positive PFS within 48 months from February 26 2014, being the date (the "Permit Date") that the Company received an exploration permit from the Peruvian authorities;
- Spending at least US$1,000,000 on exploration and drilling of the property within 12 months of the Permit Date (completed as of the effective date of this technical report); and
- Paying to JPQ a minimum of US$4,000,000 and a maximum of US$7,000,000 as follows:
  - i) US$50,000 on signing of the Letter of Intent (paid as of the effective date of this technical report);
  - ii) US$200,000 on signing of the formal option agreement (paid);
  - iii) US$750,000 no later than six months after the Permit Date (paid);
  - iv) US$3,000,000 no later than the earlier of the option exercise or 12 months of the Permit Date;

On March 26, 2015, Focus (via Agrifos) acquired an outright 70% interest in the issued share capital of JPQ, by paying $4 million cash to the Vendors, thereby cancelling its previously granted option agreement to earn such interest. Focus committed to spending a minimum of US$14 million in development of the Project, without dilution to the Vendors' remaining 30% interest. Focus has agreed to complete a PFS by December 31, 2015. Failure to complete the PFS requires the payment of a US$500,000 penalty, plus additional $500,000 penalty payments for each additional year that the study is not completed, up to a maximum of US$2,000,000 in penalty payments.

Port and loading services for the future export of phosphate rock will be provided by the Vendors at commercial rates at the Puerto Bayovar Maritime Terminal located 40 km west of the Bayovar 12 Project. Focus will retain a right of first refusal for the purchase of the Vendors' 30% interest in JPQ. In order to fund the purchase and for further advancement of the Bayovar 12 Project, Focus executed a US$5.0 million secured loan facility with Sprott Resource Lending Partnership, of which US$3.5 million was outstanding at the time of writing.

In April 2015, Focus completed the sale to Radius Gold Inc. of a royalty equal to 2% of Focus' 70% interest in future phosphate production from the Bayovar 12 Project for the sum of US$1.0 million. Under the terms of the sale agreement, Focus has the right for 12 months to buy back one-half of the royalty for US$1.0 million. If Radius decides to sell any of its royalty interest in the future, Focus will retain a first right of refusal.

4.5 MINING ROYALTIES AND TAXES

In order to maintain concessions in good standing, titleholders must pay a Mining Good Standing license fee equal to US$3.00 per hectare per year.

Under Peruvian mining laws, Concession holders must reach an annual production of at least US$100.00 per hectare in gross sales within six (6) years from January 1st of the year following the date the title was granted. If there is no production on the concession within that period, the titleholder must pay a penalty of US$6.00 per hectare or US$1.00 for small scale miners and US$0.50 for artisan miners, during the 7th through 11th years following the granting of the concession. From the 12th year onwards, the penalty is equal to US$20.00 per hectare under the general regime, US$5.00 for small scale miners and US$3.00 for artisan miners. The titleholder is exempt from the penalty if exploration expenditures incurred during the previous year was 10 times the amount of the applicable penalty. Failure to pay the license fees or the penalty for two consecutive years will result in the forfeiture of the concession.

The concession holder also has a royalty agreement with the local community of San Martin de Sechura (Fundacion Comunal San Martin de Sechura) and Activos Mineros S.A.C. The concession holder maintains title by sustaining an annual production of 80,000 tonnes of gypsum rock and by paying each a royalty of $0.60 per tonne mined. This royalty is also applicable for any other non-metallic minerals extracted via a simple conversion formula.

4.6 ENVIRONMENTAL LIABILITIES

Golder is not aware of any environmental liabilities on the Bayovar 12 Project property.

4.7 PERMITTING

Mining reconnaissance or prospecting does not require an environmental assessment. The Environmental Regulations for Mining Exploration Projects requires the submittal of an Environmental Declaration for projects that include a maximum of 20 drill holes and less than 10 ha of disturbed areas or tunnels up to 50 m long; for projects exceeding 20 drill holes, 10 ha of disturbed area or tunneling in excess of 50 m, a Semi-detailed Environmental Impact Assessment is required to be submitted prior to any exploration activities. Focus received its initial environmental permit for the Phase 1 drill program on the 22 January, 2014, and on the 15th January 2015 for the Phase 2 drill program.

As per Environmental Mining Regulations, the mining concession holder must submit an Environmental Impact Assessment once the exploration stage of the project is complete and prior to the commencement of mining activities.

4.8 OTHER SIGNIFICANT FACTORS AND RISKS THAT MAY AFFECT ACCESS, TITLE, OR THE RIGHT OR ABILITY TO PERFORM WORK ON THE PROPERTY

The Bayovar 12 Concession is not associated with any Natural Protected Area, nor is the concession within any urban or urban expansion zone, archeological site or agricultural area. Golder is not aware of any other significant factors and risks that may affect access, title or the right or ability to perform work on the Bayovar 12 Concession property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PHYSIOGRAPHY

The Focus Bayovar 12 Project area is situated within the Sechura Desert in northwestern Peru. The Sechura Desert occurs as a continuous narrow strip along much of the Pacific coast of Peru, stretching inland from 20 to 100 km, covering an area of approximately 22,000 km^2. The desert slopes gently westward from the foothills of the Andes mountain range to the Pacific Coast. In the Bayovar area the desert is partially bound on the western side by the Illescas Mountains that form the Illescas Peninsula on the southern margin of Sechura Bay (Figure 5-1).

The generally featureless, low-relief character of the Sechura Desert (Figure 5-1) is marked in the Bayovar area by several distinct physiographic features, namely the Virilla Estuary, the Tablazo and the Sechura Depression (Figure 5-1). The physiographic features present are a result of combined local uplift and subsidence as well as erosional activity.

Figure 5-1: Typical Landscape on the Bayovar 12 Concession

The Virilla Estuary is a network of shallow channels that connect Sechura Bay on the Pacific Coast with Ramon Lake, a large inland lake situated north of the project area. While the region is classified as a desert and there is limited year-round surface water present, the area is subject to tsunami and flooding associated with weather and seismic events occurring along the nearby pacific coast. The Bayovar 12 project area was impacted by the floodway associated with the 1998 El Niño event which saw widespread flooding in the low ground surrounding the Virrilla Estuary to the north.

The Tablazo (Figure 5-2) is a prominent regional scale flat-topped table land that runs north-south through the Bayovar area, separating a central plateau of higher ground from lower ground to the north, east and west. The steep ridge line marking the edge of the Tablazo ranges in height from 15 to 75 m amsl; elevation on top of the Tablazo ranges from 15 to 75 m amsl compared to 0 to 10 m amsl on the lower plain to the east and 30 to 60 m amsl on the low ground to the west that separates the Tablazo from the Illescas Mountains. The Bayovar 12 property straddles the eastern ridge of the Tablazo, with the western third of the property situated on top of the Tablazo and the eastern two thirds falling on the low ground east of the Tablazo ridge line.

Figure 5-2: Tablazo Ridge on the Bayovar 12 Concession, Looking West

The Sechura Depression is a steep sided, flat bottomed, topographic depression that transects the Tablazo, breaking it into northern and southern regions. The floor of the Sechura Depression is approximately 35 m below mean sea level. The northeastern limit of the Sechura depression is located adjacent to the southwestern limit of the Bayovar 12 Concession boundary.

Much of the Bayovar 12 project surface area is marked by a thin layer of hard packed sand, thin gravel and localized gypsum, all of which are quaternary in age. Large crescent shaped barchan sand dunes (Figure 5-3) that slowly migrate across the property in a north-easterly direction are present across most of the project area. Vegetation in the Bayovar12 Project area is sparse, consisting for the most part of drought tolerant low bushes (Figure 5-3 and Figure 5-4) sparse grass and salt tolerant plants in lower elevation areas.

Figure 5-3: Barchan Sand Dunes on the Bayovar 12 Concession

Figure 5-4: Typical Vegetation on the Bayovar 12 Concession

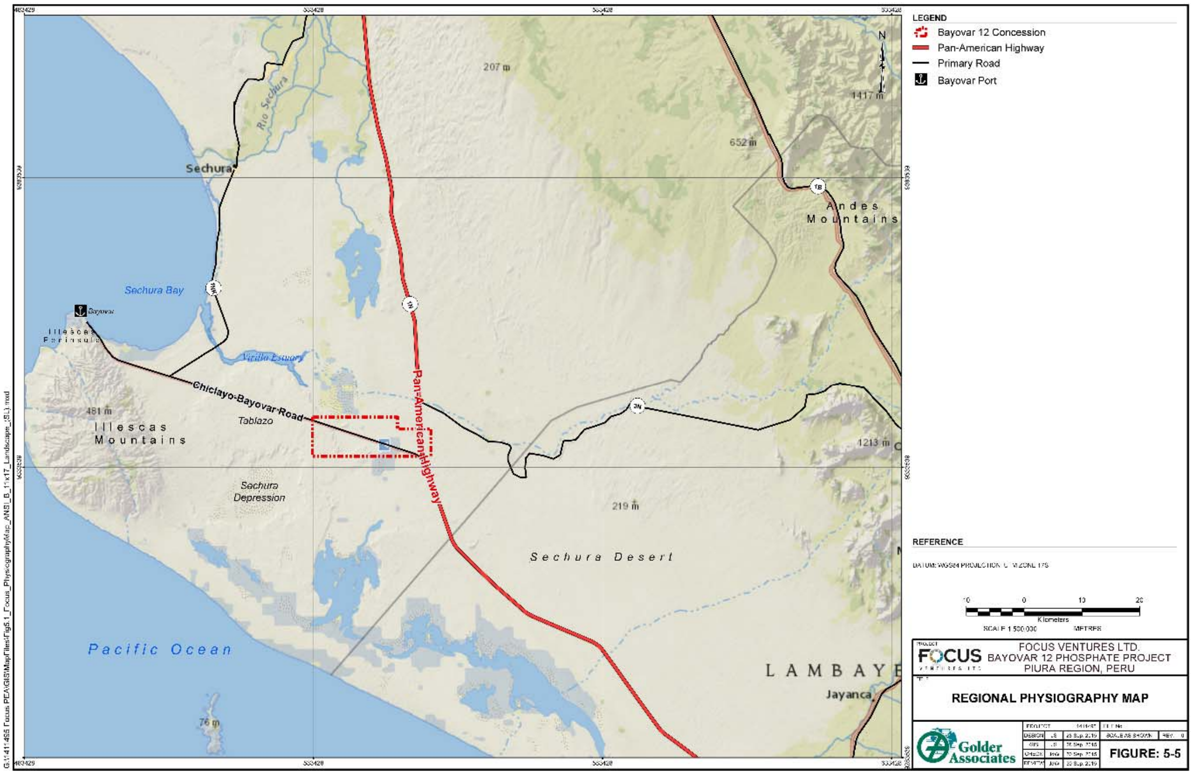


Figure 5-5: Regional Physiography

5.2 CLIMATE

The Bayovar 12 Project area is situated within the mild desert climate of the Sechura Desert. The Sechura Desert climate is heavily influenced by the confluence of the Humboldt (cold water) and Equatorial (warm water) ocean currents that circulate in contrary directions; this typically results in zones of high temperature and low precipitation.

The proximity to the Pacific Ocean results in relatively moderate temperatures year round; the mean monthly temperature in the summer months (December to April) is approximately 25°C and in the winter months (May to October) is approximately 18°C. The annual precipitation is approximately 50 millimetres (mm) of rain but can increase to in excess of 150 mm in El Niño years. Wind in the Bayovar 12 Project area is predominantly from the southeast, with average wind speed values of 4.1 metres per second (m/s; weak breeze); peak gusts are generally around 7 m/s (moderate breeze).

A summary of the historical climate data for the project area, sourced from the Sechura, Peru weather station data as compiled on the www.weatherbase.com website (accessed September 2015), is presented in Figure 5-2. The historical climate data was collected over a 30 to 112 year period.

Figure 5-6: Sechura-Bayovar Area Historical Climate Data

5.3 ACCESSIBILITY

The Bayovar 12 project area is accessible year round via a series of multi-lane sealed roads and highways (Figure 5-3). The Pan-American Highway crosses the eastern end of the property and the Chiclayo-Bayovar road (Figure 5-5) transects the property. A network of un-maintained drill roads and access roads for minor surface gypsum mining operations provide four wheel drive vehicle access to the remainder of the property.

Figure 5-7: Chiclayo-Bayovar Road on the Bayovar 12 Concession with the Tablazo in the Background

Travel time from Piura to the Bayovar 12 Concession is approximately 1.5 hours by car via the Pan-American Highway. Piura is serviced by a modern domestic airport with commercial daily service to Lima and other airports in the region. Air travel flying time from Piura to Lima is approximately 1.5 hours.

The concession is also located 40 km inland by paved road from the JPQ marine port facility (Figure 5-6) near the fishing village of Puerto Rico, located in Sechura Bay on the pacific coast. Water depth adjacent to the jetty at the JPQ port facility is approximately 8 m, currently allowing for loading of 24,0000 Deadweight tonnage (DWT) capacity vessels.

Figure 5-8: JPQ Marine Port Facility on Sechura Bay

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

The Bayovar 12 project area is approximately 40 km by paved multi-lane road from the JPQ port facility located near the fishing village of Puerto Rico, located on Sechura Bay on the pacific coast. The JPQ port facility is situated adjacent to the Vale port facility where phosphate from Vale's Miski Mayo operation (Bayovar Mine) is loaded for shipping.

Power transmission lines for the Vale Miksi Mayo Bayovar Mine also transect the northwest corner of the Focus Bayovar 12 property (Figure 4-3). An easement for power transmission lines to the FOSPAC property also transects the northwest corner of the Bayovar 12 property.

Mining operation at the Bayovar 12 property could be conducted year-round and would not be affected by the climate.

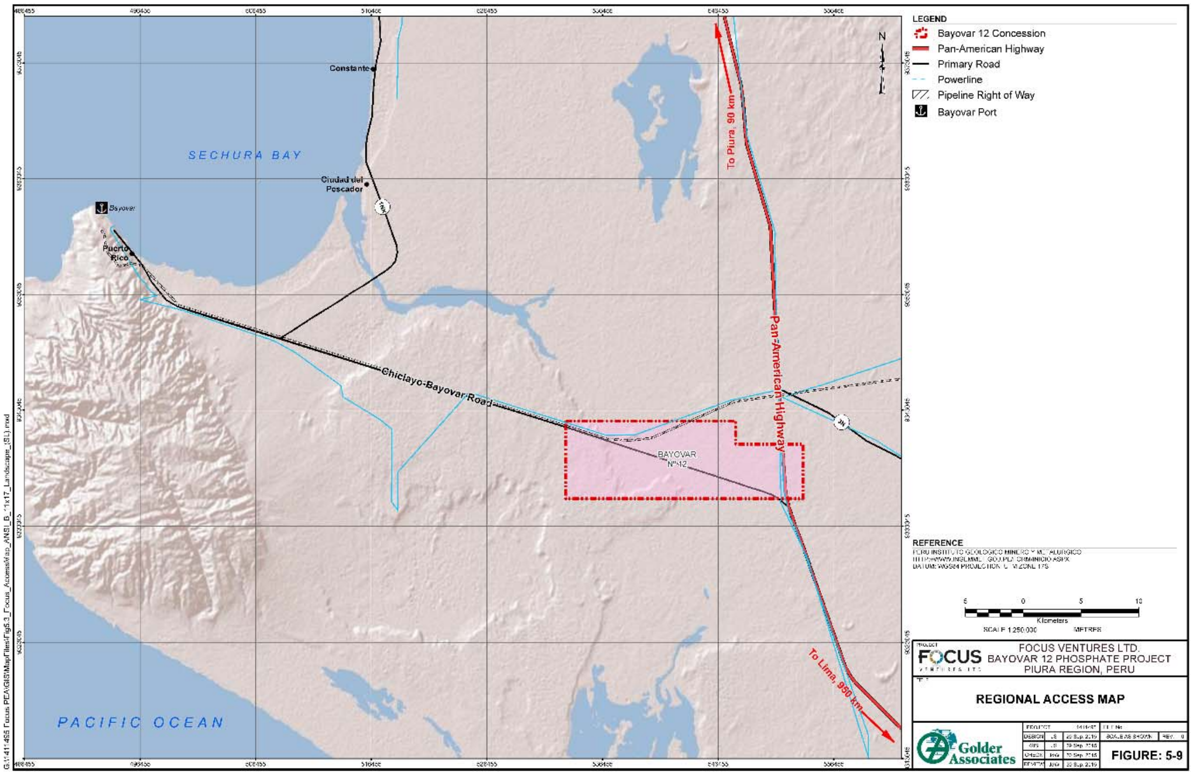


Figure 5-9: Regional Access

6 HISTORY

6.1 OWNERSHIP HISTORY

On January 14, 2014, Focus' Peruvian subsidiary, Agrifos, signed a formal option agreement for the acquisition of shares in JPQ, the titleholder of the Bayovar 12 mining concession. On March 26, 2015, Focus (via Agrifos) acquired an outright 70% interest in the issued share capital of JPQ, by paying $4 million cash to JPQ, thereby cancelling its previously granted option agreement (see Section 4, Property Description and Location, for details pertaining to the agreement).

The Bayovar 12 Concession was acquired by JPQ in 2007 under a contract with state company Activos Mineros S.A.C. for the exploitation of gypsum rock and other non-metallic minerals by open pit methods from the concession.

Prior to the acquisition of the Bayovar 12 Concession by JPQ from Activos Mineros S.A.C in 2007, stretching back to the 1950's there have been numerous government and commercial entities that have owned portions of the overall Bayovar-Sechura Phosphate deposit as summarized in Nardi and Gruber (2008) and Apaza (2012). Historical entities with interests in the Bayovar area include Minerales Industriales del Peru (MIDEDSA), ESSO-Homestake, Minera Bayovar S.A., Kaiser Aluminum and PROBAYOVAR.

Golder has not been able to establish historical land tenure boundaries for the various past owners in the region. As a result, it is possible that a portion or all of the current Bayovar 12 Concession area may have been included in the land holdings of some of these historical operators in the Bayovar-Sechura Phosphate deposit.

6.2 EXPLORATION HISTORY

The phosphate deposits of the Bayovar area were discovered in 1955 during regional oil and gas exploration. Phosphorite was discovered in the immediate project area in an abandoned road cut in 1958.

As mentioned in the previous section, there were a number of different historical entities with interests in portions of the Bayovar-Sechura Phosphate Deposit since its discovery in the 1950's; however, Golder has not been able to establish what, if any, historical exploration work may have been conducted within the boundary of the area that now comprises the Bayovar 12 Concession area. There are no records of any historical exploration activity specific to the property prior to JPQ ownership in 2007.

JPQ performed limited reconnaissance exploration work on the Bayovar 12 Concession in 2012; however, Golder and Focus could not verify the methodology and results from the 2012 JPQ work to a level where they could be relied upon for use in the geological modelling process and resultant resource estimates. As a result the 2012 JPQ work was not used for modelling and resource estimation as reported in this technical report.

The only significant detailed exploration programs on the Bayovar 12 Concession are the Phase 1 (2014) and Phase 2 (2015) exploration programs implemented by Focus; a detailed discussion of the Phase 1 and Phase 2 Focus exploration programs is presented in Section 10 of this Technical Report.

The drill holes from the Focus Phase 1 and Phase 2 exploration programs are presented in Figure 6-1.

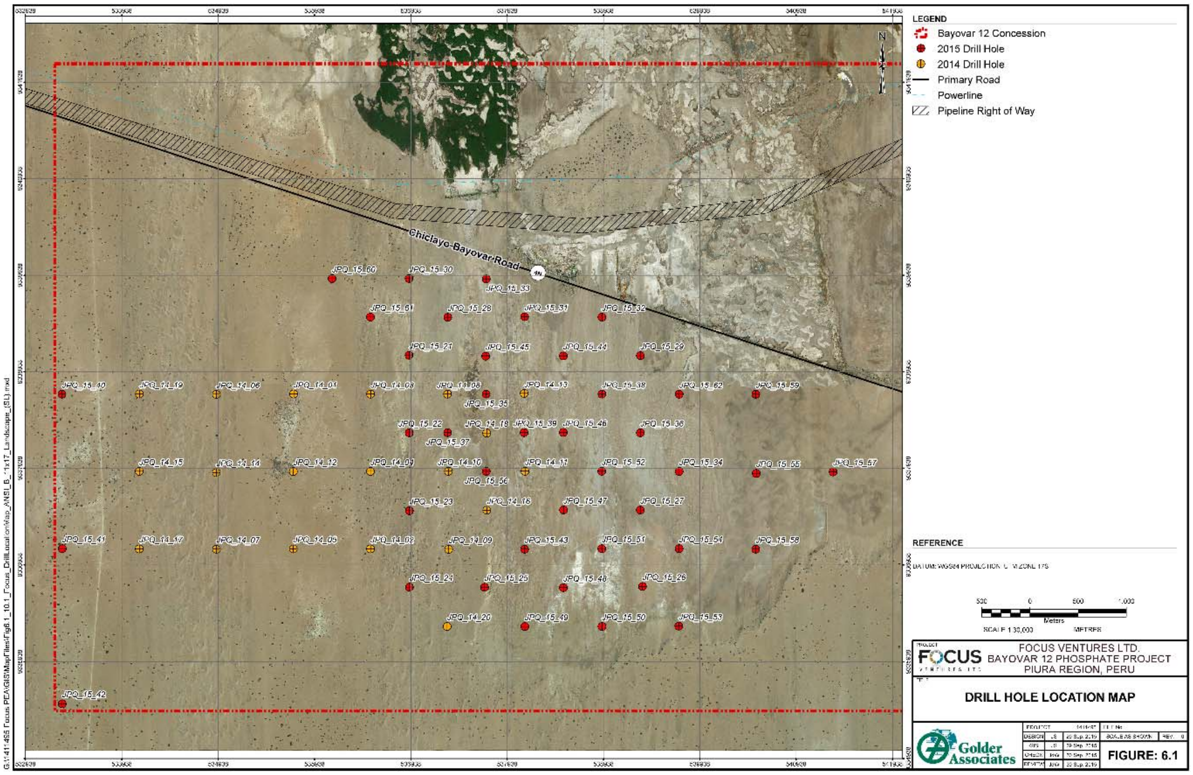


Figure 6-1: Drill Hole Location Map

6.3 DEVELOPMENT HISTORY

As of the effective date of this technical report there has been no phosphate development work undertaken on the Bayovar 12 Concession by current or previous owners or operators.

6.4 HISTORICAL MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The Company published an inaugural Resource estimate on October 23, 2014 and an updated Resource estimate on October 5 2015.

Table 6-1: Summary of Mineral Resources, October 5, 2015

Beds PH01 to PH16

Category	Tonnes (Mt; wet)	Tonnes (Mt; dry)	P_2O_5 Grade (wt.%)
Measured	23.4	17.7	13.16
Indicated	277.1	209.5	13.04
Measured & Indicated	300.5	227.2	13.05
Inferred	135.0	102.2	13.11

Note: Mt = million tonnes
No minimum thickness, grade cut-off or other mining parameters applied
Phosphorite bed specific wet and dry relative densities used for tonnage calculations

As of the effective date of this technical report there are no historical phosphate mineral reserve estimates for the Bayovar 12 Concession property.

6.5 PRODUCTION HISTORY

As of the effective date of this technical report there has been no commercial phosphate mining production from the Bayovar 12 Concession property.

Mining activity on the Bayovar 12 Concession property is limited to small scale surface mining of quaternary age gypsum that occurs at surface on the low ground immediately east of the Tablazo. The gypsum mining operation is carried out by JPQ, using a dozer to push the gypsum into piles that are then loaded on to a small road haul truck using and excavator. The gypsum is then transported by truck to the JPQ port facility on Sechura bay where it is stockpiled prior to loading onto ships.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

7.1.1 Regional Stratigraphy

The following section summarizes the regional geology as presented by McClellan (1989), Cheney et al., (1979) Bech (2009) and references therein.

The Bayovar-Sechura Phosphate Deposit occurs in the Sechura Basin (Figure 7-1), a shallow north trending basin situated in northwestern Peru. The Sechura Basin is bordered by the Illescas Mountains to the west and the foothills of the Andes Mountains to the east. The basin is filled by a thick sequence of interlayered marine sediments including phosphorite, diatomite, sandstone, shale and volcanic tuff, ranging in age from Eocene (56.0 to 33.90 Ma) at the base to Pliocence (5.33 to 2.58 Ma) in the upper basin.

The Sechura basin formed as a result of subsidence along the paleo-continental shelf. Cycles of uplift and subsidence modified the basin during its long infilling history, with basement faults partially controlling basin geometry during deposition. The stratigraphy is subhorizontal, dipping gently at 2° to 3° across the basin.

The phosphate bearing units occur in the upper 135 to 215 m of the Miocene (23.22 to 5.33 Ma) strata in the basin, within the Zapallal Formation. The Zapallal Formation comprises a cyclical series of interlayered marine basin fill diatomites and phosphorites with lesser sandstone and tuff. A detailed discussion of the Zapallal Formation stratigraphy and structure is presented in the section below.

The Zapallal Formation is underlain by older Miocene, Oligocene and Eocene age marine basin fill sedimentary units which unconformably overlay metamorphic and igneous basement rocks of Paleozoic (541 to 252 Ma) and Precambrian (greater than 541 Ma) in age. As a result of late Eocene basement uplift and basin subsidence the Paleozoic basement is exposed in the Illescas Mountains along the western margin of the basin.

The Zapallal Formation is unconformably overlain by Pliocene age interbedded coquina, sandstone and shale. The stratigraphy is capped by a thin cover of unconsolidated Quaternary (2.58 Ma to present) age alluvial and aeolian sand with localized occurrences of gypsum.

7.1.2 Zapallal Formation Detailed Stratigraphy

The marine basin fill Zapallal Formation is subdivided into four members, which are in turn subdivided into distinct units or zones (Figure 7-2). The formation and its subdivision presented from oldest to youngest are as follows:

- Lower Diatomite and Phosphorite Member
  - Tuffaceous Diatomite – Thickness is in excess of 50 m, predominantly foraminifera-rich diatomite with numerous thin beds of tuff and three isolated phosphorite beds; P_2O_5 grades in the phosphorite beds range from 10 to 18 wt% and from 1 to 3 wt% in the diatomite. The upper contact is gradational with the overlying Diana ore zone (note: please refer to Section 2.5 of this technical report concerning the usage of the term “ore zone”).
  - Diana ore zone – Mean thickness of 39 m (range of 36 to 41 m). The zone is subdivided into seven regionally correlatable phosphorite beds alternating with diatomite beds. The Diana ore zone contains the highest P_2O_5 grades (ranging from 10 to 25 wt% in the phosphorite beds and 3 to 7 wt% in the diatomite beds) and thickest phosphorite bearing zones in the Bayovar-Sechura Phosphate deposit. The upper contact is gradational with the overlying Grey Tuff.

  - Grey Tuff – Thickness ranges from 0 to 21 m. Comprises generally massive beds of soft grey diatomaceous tuff. Preservation of the unit varies across the area due to variable and locally significant erosion by the overlying Clam Bore Sandstone. There are no identified or correlatable phosphorite beds present within the unit and no significant presence of phosphate pellets, resulting in negligible P_2O_5 grades. The upper contact with the overlying Clam Bore Sandstone is marked by a hiatus and angular unconformity; units below the unconformity dip to the east while units above dip to the southeast.

- Clam Bore Sandstone Member – Thickness ranges from 0 to 23 m; the unit is thinnest in the southern portion of the basin and thickest in the north and western portions of the basin. Comprises fine to medium grained quartz sand with abundant trace fossils across most of the basin; in areas where the unit is thicker the upper portion of the unit comprises a limestone coquina (oyster bank). The upper contact is gradational with the overlying Zero ore zone.

- Upper Diatomite and Phosphorite Member

  - Zero (or Cero) ore zone – Mean thickness of 6 m (range of 3 to 11 m). Comprises a single thick phosphorite bed (the Zero Bed) and is overlain by diatomite; P_2O_5 grade in the Zero Bed can be up to 18 wt.%, with a mean P_2O_5 grade of 9 wt.% over the entire zone (phosphorite and overlying diatomite). The upper contact is gradational with the overlying Inca Diatomite.

  - Inca Diatomite – Mean thickness of 10 m (range of 5 to 17 m). Comprises massive beds of diatomite with minor pellets but no distinct or correlatable phosphorite beds; P_2O_5 grade ranges from 1.0 to 2.8 wt% throughout the unit. The upper contact is gradational with the overlying Minerva ore zone.

  - Minerva ore zone – Mean thickness of 26 m (range of 23 to 35 m). Comprises two to three phosphorite beds interlayered with diatomite; overall P_2O_5 grade in the phosphorite beds is lower than those in the other zones with grades generally in the 5 to 6 wt.% range although in areas where the unit thins there is often an associated increase in P_2O_5 grade, with values in the 8 to 10 wt.% range. Grades in the diatomite beds range from 2 to 3 wt% P_2O_5. The upper contact is gradational with the overlying Quechua Diatomite.

  - Quechua Diatomite – Mean thickness of 17 m (range of 12 to 19 m). Comprises massive beds of diatomite with localized occurrences of one or two thin phosphorite beds. Grade across the unit (diatomite and localized phosphorite inclusive) is typically 2 wt% P_2O_5. The upper contact is gradational with the overlying Barren Diatomite Member.

- Barren Diatomite Member – Thickness range of 0 to 31 m. Comprises massive diatomite and mica flakes are commonly present. There are no identified or correlatable phosphorite beds present within the unit and no significant presence of phosphate pellets, resulting in negligible P_2O_5 grades. The upper contact is marked by a hiatus and angular unconformity separating the Miocene Zapallal Formation from overlying Pliocene sandstone and shale; units below the unconformity dip to the southeast while units above are horizontal.

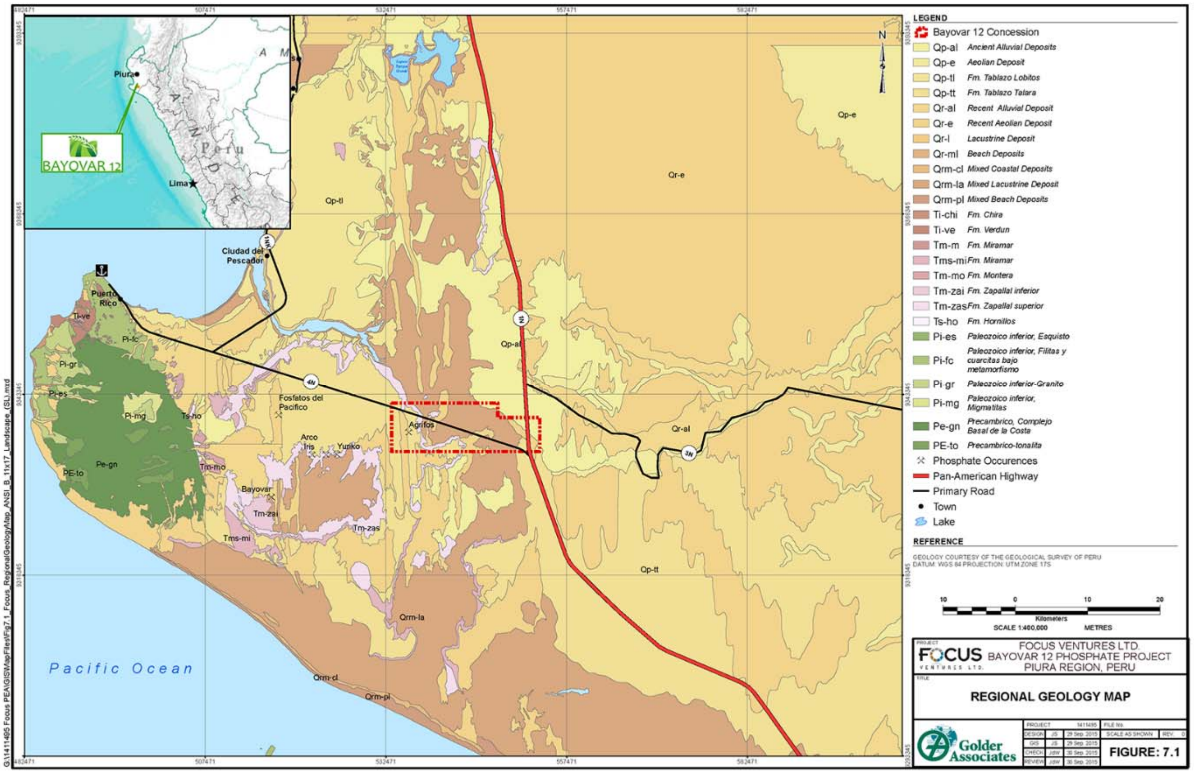


Figure 7-1: Regional Geology Map

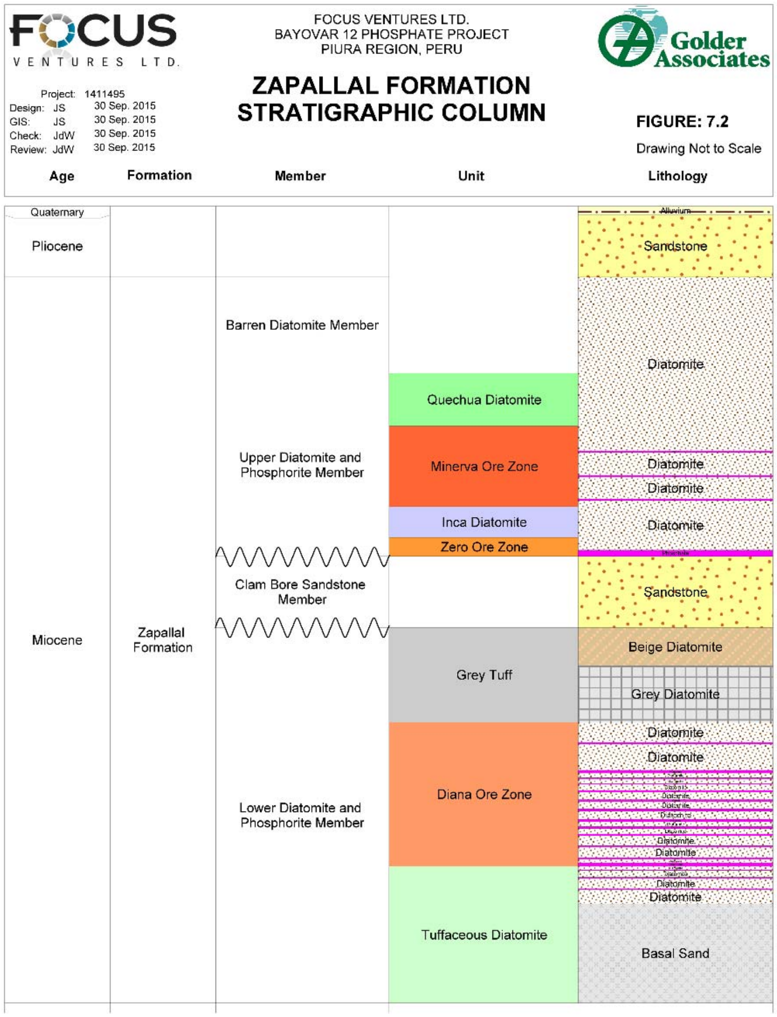


Figure 7-2: Zapallal Formation Stratigraphic Column

7.1.3 Regional Structure

As identified in the detailed discussion of the Zapallal Formation, there are two regional scale angular unconformities identified as impacting the stratigraphy in the basin. The uppermost unconformity occurs at the top of the Zapallal Formation (Figure 7-2), marking the contact between barren diatomite of Miocene age and coquina, sandstone and shale of the overlying Pliocene sedimentary units. The second unconformity occurs within the Zapallal Formation, marking the contact between the Lower Diatomite and Phosphorite Member and the Clam Bore Member.

There are isolated occurrences of regional scale faulting and folding in the basin, including a regional scale fault that transects the FOSPAC Bayovar 9 concession to the west, but for the most part the stratigraphy has seen minimal post-depositional tectonic modification.

7.2 PHOSPHORITE AND DIATOMITE COMPOSITION

The phosphorite beds (Figure 7-1) are comprised primarily of massively bedded phosphate pellets with lesser grains and fragments of diatoms, volcanic glass; sodium, potassium and magnesium salts; quartz; feldspar; sponge fragments; gypsum, mica flakes and organic matter. The phosphate is marine in nature and is generally in the form of fluorhydroxycarbonate apatite.

The apatite is generally in the form of individual pellets although agglomerations of pellets, oolites, laminae, nodules and fragments of teeth, bones or shells are also present. The pellets are generally subrounded but elongated and irregular shaped pellets also present. The pellet grain size ranges from 0.4 to 2.0 mm in diameter, with larger pellets occurring in the phosphorite beds while finer grained pellets occur in the diatomite.

The pellets range in color from white to brown to black. Although the apatite pellets and other grains and fragments are generally well sorted within the beds, most of the pore space is filled in with fine fragments of diatoms and silt. The specific gravity of individual pellets is typically around 2.9.

Figure 7-3: Typical Phosphorite Bed Showing Layering of Phosphorite (dark) and Diatomite (light)

7.2.1 Diatomite

The diatomite beds are comprised primarily of massively bedded aggregates of microscopic diatom fragments with variable finer grained apatite pellets and lesser grains and fragments of volcanic glass, shell and bone fragments and sponge fragments. The diatoms are typically composed of opaline silica.

The diatomite ranges in color from white to brown to olive green. The diatomite generally has high porosity, often on the order of 90%; as a result of this and its resistance to compaction, the specific gravity is very low, typically around 1.5.

7.3 PROPERTY GEOLOGY AND MINERALIZATION

The Zapallal Formation stratigraphy dips gently to the east within the Bayovar 12 Concession (Figure 7-3). No faulting or folding was identified within the concession. Interpretation of the Phase 1 and Phase 2 Focus exploration drilling in the western portion of the Bayovar 12 Concession indicated that the following stratigraphic units (from top downwards) were intercepted on the property:

- Quaternary sand, gravel and localized gypsum (distinct upper and lower gypsum horizons)
- Zapallal Formation
  - Clambore Sandstone Member
  - Lower Diatomite and Phosphorite Member
    - Gray Tuff
    - Diana ore zone
    - Tuffaceous Diatomite

Summary statistics for the various overburden beds are presented in Table 7-1. Representative cross sections across the Focus drilling area are presented in Figure 7-4 and Figure 7-5.

Table 7-1: Overburden Unit Thickness Summary Statistics

Overburden Unit	Intercept Count	Mean Thickness (m)	Minimum Thickness (m)	Maximum Thickness (m)
Quaternary	62	4.24	0.15	12.70
Upper Gypsum	27	0.62	0.05	2.10
Lower Gypsum	12	0.28	0.05	0.52
Clambore Sandstone	20	6.72	1.90	15.80
Grey Tuff - Beige Diatomite	59	7.56	0.50	23.90
Grey Tuff - Grey Diatomite	62	19.64	12.68	33.19

As a result of Quaternary erosional surfaces, the uppermost members of the Zapallal Formation, namely the Upper Diatomite and Phosphorite Member and the Barren Diatomite Member, were absent within the Phase 1 and Phase 2 drilling area. This included the absence of the Zero ore zone phosphorite bed (lower most phosphorite bed in the Upper Member) that is present several kilometres to the west on the FOSPAC Bayovar 9 Concession. As the stratigraphy is dipping gently to the east, it is possible that the upper portion of the Zapallal Formation may be encountered when planned exploration drilling on the property advances to the east.

Focus has intercepted 16 distinct and correlatable phosphorite beds (identified as PH01 through PH16) across the concession. Focus and Golder have interpreted the upper 13 phosphorite beds (PH01 to PH13) as Diana ore zone Beds with the lower three beds (PH14 to PH16) interpreted as phosphorite beds occurring in the underlying Tuffaceous Diatomite unit. The literature on the Bayovar-Sechura Phosphate Deposit generally identifies seven regionally correlatable phosphorite beds in the Diana ore zone; however, Focus and Golder interpret the six additional phosphorite beds encountered in the Diana ore zone as locally continuous and correlatable beds that aren't necessarily present or correlatable across the entire basin. Summary thickness and P_2O_5 grade statistics for the phosphorite are presented in Table 7-2, Phosphorite Bed Thickness and P_2O_5 Grade Summary Statistics. Summary thickness and P_2O_5 grade statistics for the diatomite beds are presented in Table 7-3, Diatomite Bed Thickness and P_2O_5 Grade Summary Statistics.

The individual phosphorite beds exhibit relatively uniform thickness and P_2O_5 grade profiles across the concession; however, there is a pronounced zonation of P_2O_5 grades in both the phosphorite and diatomite beds that effectively divides the Diana ore zone into an Upper Diana ore zone and Lower Diana ore zone.

The Upper Diana ore zone phosphorite beds (PH01 through PH05) exhibit high mean grades and are separated by low grade diatomite beds with mean grades all below 2 wt.% P_2O_5.

Below PH05 there is a marked change in the nature of the diatomite beds with mean grades all in excess of 2 wt.% P_2O_5. The sole exception is diatomite bed IB11, located between PH10 and PH11, which consistently exhibits mean P_2O_5 grades below 2 wt.% across the concession. Because of the signature low grade profile, the IB11 diatomite was used by both Focus and Golder as a marker bed during the correlation process.

The Zapallal Formation units are interpreted to continue in all directions beyond the limits of the Phase 1 and Phase 2 drilling programs. Due to concession boundary limits planned future exploration drilling will concentrate on expanding the resource to the east of the Tablazo ridge, where the phosphorite beds are closer to surface due to lower surface elevations.

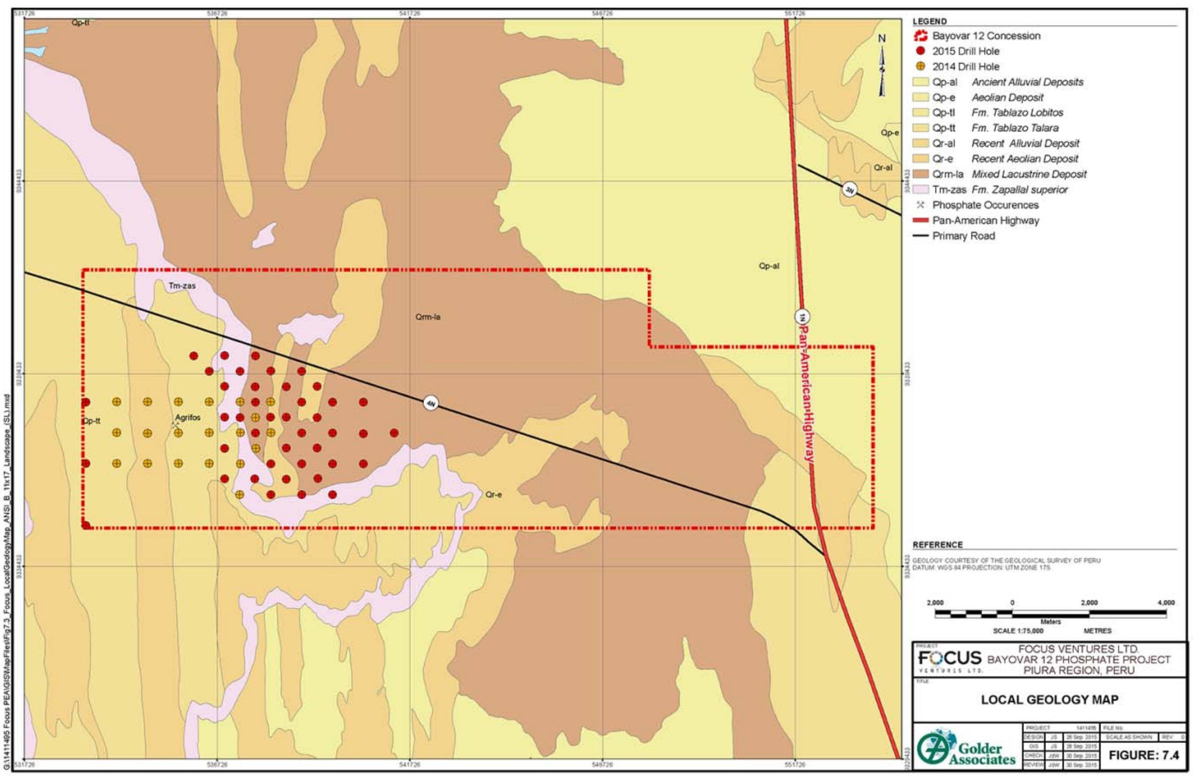


Figure 7-4: Local Geology Map

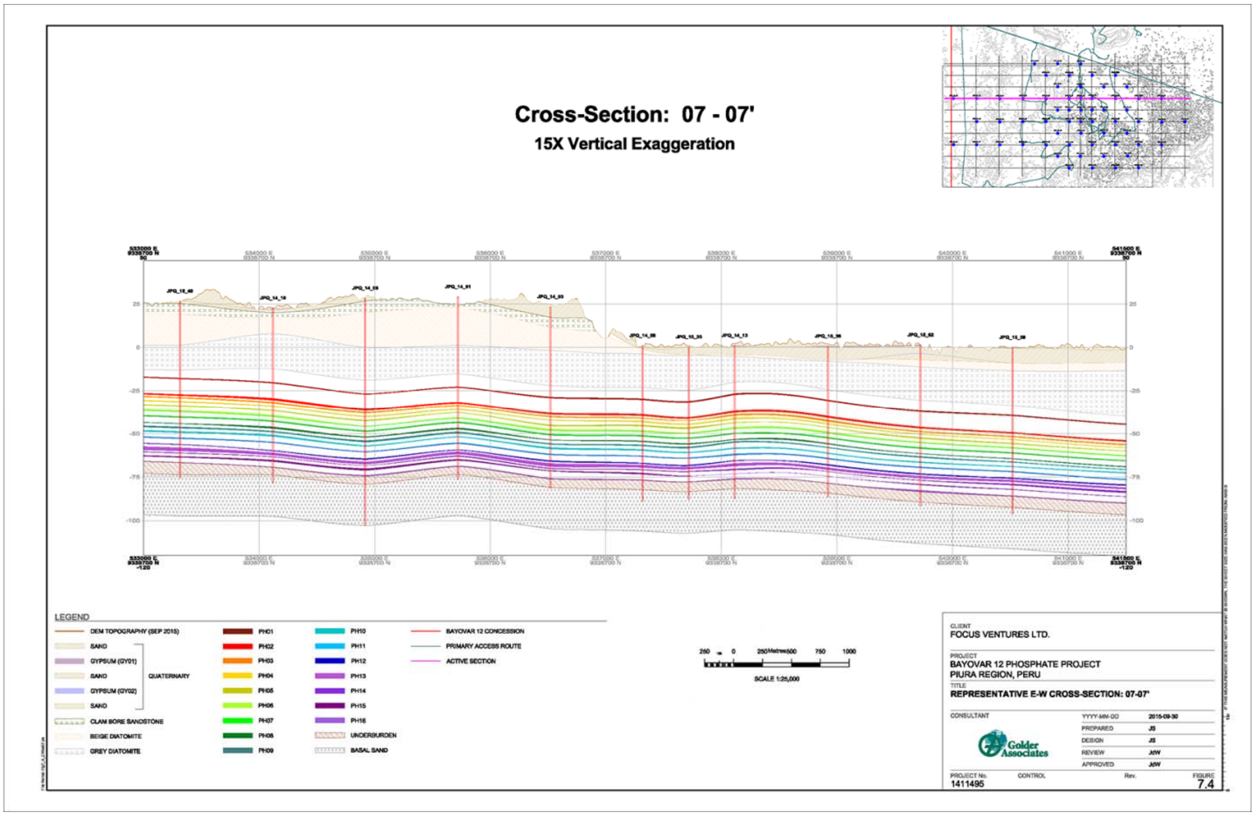


Figure 7-5: Representative East-West Cross Section

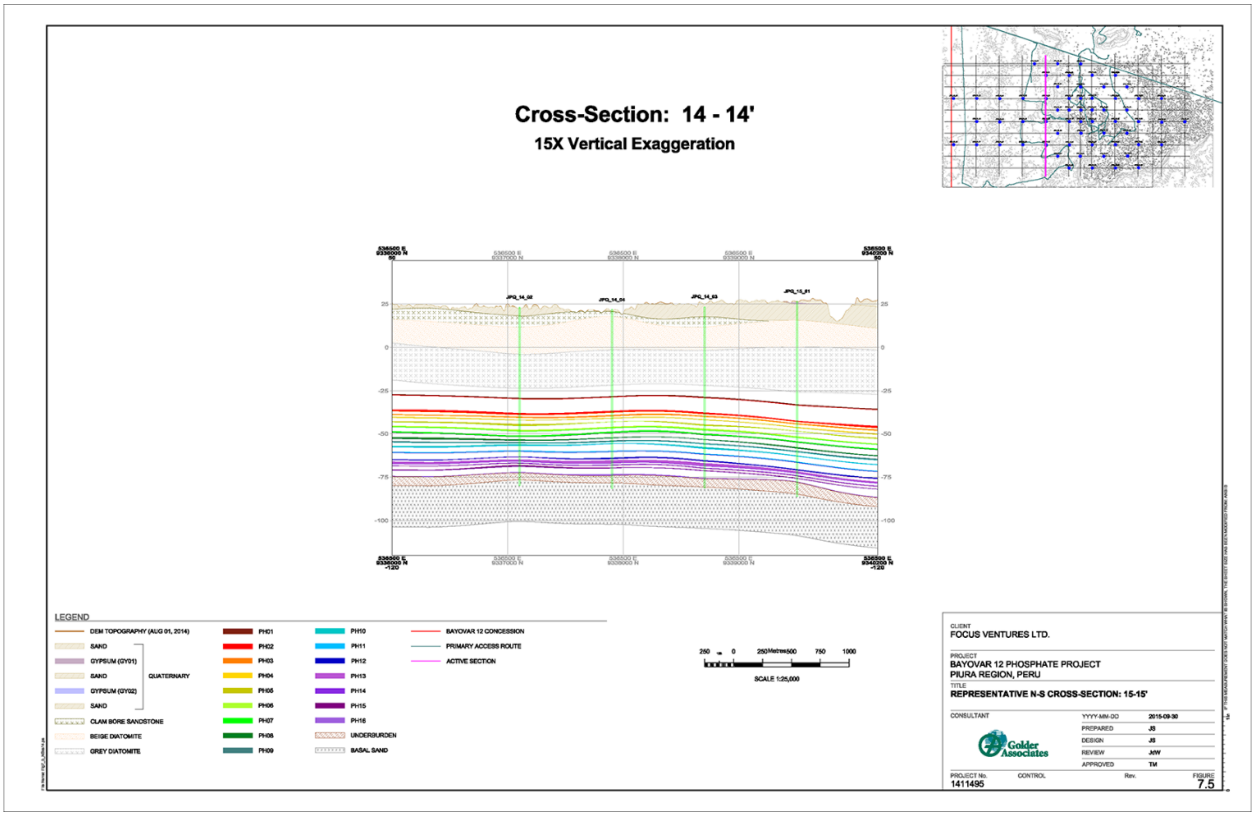


Figure 7-6: Representative North-South Cross Section

Table 7-2: Phosphorite Bed Thickness and P_2O_5 Grade Summary Statistics

Phosphorite Bed	Intercept Count	Mean Thickness (m)	Minimum Thickness (m)	Maximum Thickness (m)	Mean P_2O_5 (wt. %)	Minimum P_2O_5 (wt. %)	Maximum P_2O_5 (wt. %)
PH01	62	0.50	0.22	0.72	12.60	6.14	17.12
PH02	62	0.89	0.47	1.26	11.77	7.13	15.87
PH03	62	0.48	0.10	0.94	19.59	10.89	24.32
PH04	60	0.31	0.10	1.04	16.08	6.78	23.72
PH05	61	0.45	0.10	1.06	9.49	5.08	16.07
PH06	62	0.58	0.25	0.92	12.99	6.35	19.12
PH07	62	0.60	0.24	1.58	10.20	5.56	14.94
PH08	62	0.48	0.10	1.45	10.85	7.00	20.83
PH09	62	0.50	0.10	0.87	11.72	7.08	18.31
PH10	62	0.47	0.10	1.03	10.70	5.88	16.71
PH11	62	0.40	0.16	0.89	13.38	7.04	19.56
PH12	62	0.51	0.10	0.94	14.82	6.41	23.64
PH13	62	1.06	0.43	1.76	13.47	7.94	19.23
PH14	57	0.27	0.10	1.00	9.15	4.81	17.37
PH15	55	0.40	0.10	1.04	8.55	4.05	15.85
PH16	61	0.36	0.10	0.97	7.44	2.91	11.47
All Beds	976	0.52	0.10	1.76	12.25	2.91	24.32

Note: Mean P_2O_5 grades are thickness weighted.

Table 7-3: Diatomite Bed Thickness and P_2O_5 Grade Summary Statistics

Diatomite Bed	Intercept Count	Mean Thickness (m)	Minimum Thickness (m)	Maximum Thickness (m)	Mean P_2O_5 (wt. %)	Minimum P_2O_5 (wt. %)	Maximum P_2O_5 (wt. %)
IB02	62	8.76	7.85	9.93	1.85	1.12	3.54
IB03	61	1.45	0.77	1.99	1.77	1.00	3.72
IB04	62	1.40	0.54	2.83	1.62	1.00	3.31
IB05	62	1.96	1.39	2.76	2.07	1.00	4.71
IB06	62	2.32	1.57	3.20	2.70	1.00	5.44
IB07	62	2.43	1.80	3.38	2.96	1.30	5.19
IB08	62	2.87	1.66	3.54	3.56	1.91	5.15
IB09	62	1.67	0.90	2.60	4.05	2.39	5.59
IB10	62	2.15	0.63	3.64	4.66	1.45	6.50
IB11	62	2.96	2.26	4.41	1.51	1.00	3.18
IB12	62	3.48	2.31	4.74	3.28	1.83	5.71
IB13	62	1.11	0.41	1.90	3.62	2.00	5.64
IB14	61	1.25	0.46	1.79	3.31	1.56	6.24
IB15	62	2.05	1.10	2.97	2.42	1.00	3.70
IB16	62	3.35	1.23	4.55	2.16	1.00	5.29
All Beds	928	2.62	0.41	9.93	2.60	1.00	6.50

Note: Mean P_2O_5 grades are thickness weighted.

8 DEPOSIT TYPES

The following section summarizes the deposit type and genetic model as presented by Simandl et al. (2012), Mosier (in Cox and Singer, 1992), Garrison (1992), Follmi (1996) Froelich et al. (1988), and Cheney et al. (1979).

The Bayovar-Sechura phosphate deposit is a sedimentary phosphate deposit, also commonly referred to as upwelling phosphate deposits, stratiform phosphate deposits or phosphorite deposits. Sedimentary phosphate deposits are stratiform bodies that commonly comprise alternating mineralized and barren zones; the individual zones can range from sub-metre thickness up to tens of metres thick, with the overall thickness of mineralized and barren sequence commonly forming in excess of several hundred metres. The deposits typically cover significant areal extents, often extending for tens or hundreds of kilometres in their maximum lateral dimensions.

Sedimentary phosphate deposits are biochemical in origin. The formation of sedimentary phosphate deposits occurs throughout geological time, spanning as far back as the Proterozoic (2,500 to 542 Ma) to the present day. On the basis of stratigraphy the Bayovar-Sechura Phosphate deposit formed during the middle Miocene with most of the phosphate deposition occurring between 8.5 to 7 Ma (Garrison, 1992). Modern sedimentary phosphate deposits are currently forming off the Pacific coast of Peru under similar depositional conditions and controls that were in place during the Miocene deposition and formation of the Bayovar Phosphate deposit (Froelich et al., 1988).

Both paleo and modern sedimentary phosphate deposits typically develop in marine sedimentary basins that occur along passive continental margins. Conditions favourable for the depositional and biochemical process are found in areas of warm paleoclimate (or current climate for modern day equivalents), typically occurring between the 40th parallels at the time of deposition. The water depth at time of deposition and biogenic activity can range from 40 m to in excess of 300 m.

Sedimentary phosphate deposits form in marine sedimentary basins where upwelling, nutrient-rich, cold waters interact with the warm surface seawater layer, creating favourable conditions for intense algal bloom. Algal blooms develop as algae multiply at rapid rates in nutrient rich, shallow marine environments (allowing for significant sunlight input to aid in the photosynthetic process). During algal blooms the algae biomass increases significantly as algae multiply. The combination of high biomass and rapid multiplication of the algae often leads to toxic and/or anoxic conditions that prove fatal to both algae and other marine organisms.

Cycles of algal bloom and death lead to significant seafloor accumulation of organic phosphate released by the algae along with accumulation of skeletons, scales, fecal pellets and other organic debris from algae and other marine life forms in areas of upwelling activity. The decomposition of the phosphate bearing organic debris by bacteria along with the dissolution of fish bones and scales result in the precipitation of phosphate minerals in an anoxic environment within the unconsolidated seafloor sediment near the sediment-water interface. This process is known as phosphogenesis.

Multiple cycles of marine regression and transgression and cycles of upwelling activity result in changes in the depositional environment and associated biochemical processes occurring within the host marine sedimentary basin. These changes commonly result in the cyclical nature of the deposits, where the deposit comprises a series of alternating phosphorite and barren (or non-phosphorite) horizons of varying thickness.

The primary phosphate minerals present in most sedimentary phosphate deposits are microcrystalline Fluorapatite ($Ca_5(PO_4)_3F$) or Francolite (carbonate-rich Fluorapatite; $Ca_5(PO_4)_{2.5}(CO_3)_{0.5}F$). The phosphate minerals in sedimentary phosphate deposits commonly occur as pellets, oolites, laminae, nodules and fragments of teeth, bones or shells.

The host rocks and the barren horizons within the sedimentary phosphate deposits are most commonly diatomite although mudstone, marl, limestone, volcanic ash and sandstone are also known to occur depending on changes in the depositional environment and sediment input within the basin.

9 EXPLORATION

9.1 SUMMARY OF NON-DRILLING EXPLORATION ACTIVITY

Detailed exploration activities on the Bayovar 12 Concession to date have been limited to exploration drilling during the Phase 1 (2014) and Phase 2 (2015) Focus exploration programs.

There have been no surface geochemical sampling programs conducted on the Bayovar 12 Concession and there have been no surface or airborne geophysical surveys conducted on the Bayovar 12 Concession.

The only non-drilling exploration activity on the Bayovar 12 Concession was the development of a digital topography model in August 2014.

9.2 DIGITAL SURFACE (TOPOGRAPHY) MODEL

Focus engaged Pacific Geomatics Ltd. of Vancouver, Canada, in August 2014 to prepare a Digital Surface Model (DSM) for the Bayovar 12 Concession. The DSM covered the entire extent of the 2014 Focus exploration area as well as most of the remainder of the Bayovar 12 Concession. In August 2015 the coverage was extended by an additional 79km^2 to cover the entire Bayovar 12 Concession and surrounding areas (Figure 9-1).

The DSM was prepared using 1.5 m SPOT6 Tristereo satellite imagery. Gross errors were fixed in the DSM in stereo. The easting and northing data was adjusted by Pacific Geomatics Ltd. to fit with available 0.50 m data for the area. Although DSM models differ from Digital Elevation Models (DEM) in that DSM's do not process out features like vegetation and building, there are no such features present on the Bayovar 12 Concession.

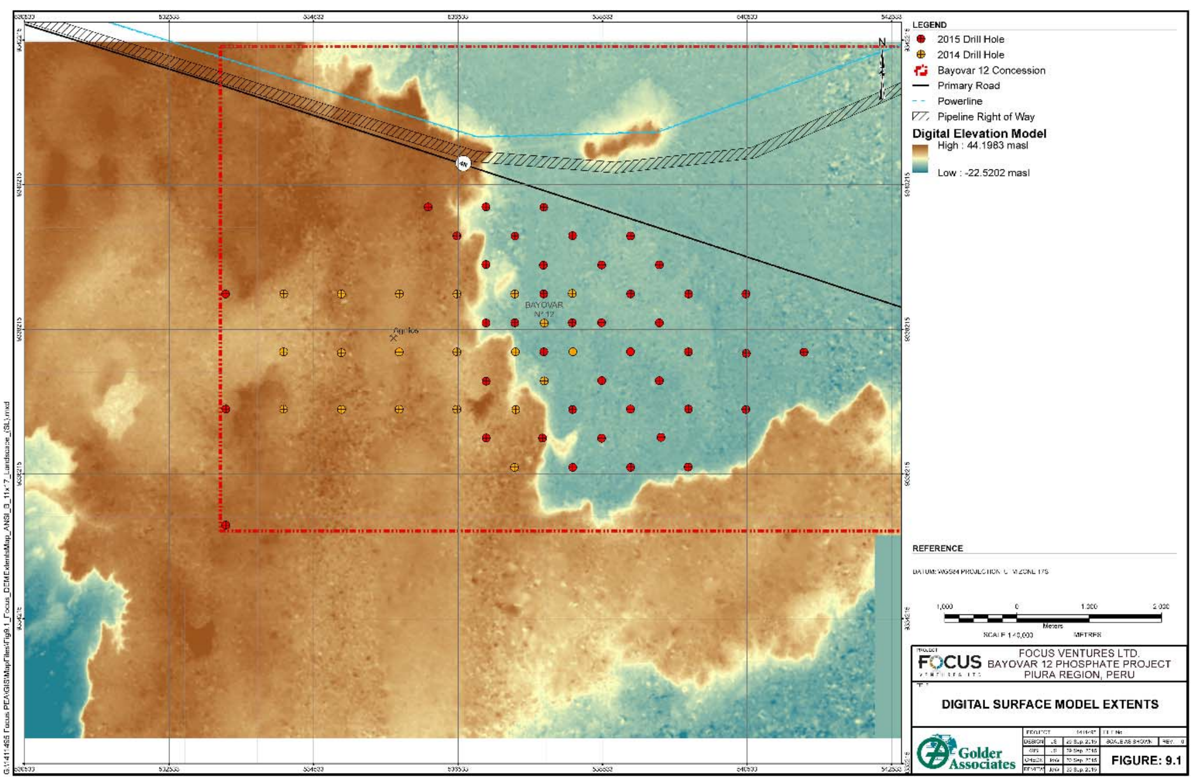


Figure 9-1: Digital Surface Model Extents

10 DRILLING

10.1 DRILLING SUMMARY

Detailed exploration drilling activities on the Bayovar 12 Concession to date have been limited to the Phase 1 (2014) and Phase 2 (2015) Focus exploration programs.

The Phase 1 exploration program resulted in the completion of 20 HQ (63.5 mm core diameter) vertical core holes totaling 2,027 m while the Phase 2 exploration program added an additional 42 HQ vertical core holes totalling 3,944 m for an overall project total of 62 drill holes and 5,971 m. The drill hole total depths for both programs ranged from 81 to 131 m (mean of 96 m); total depth variation was due to the location relative to the Tablazo. All of the Phase 1 and Phase 2 drill holes were completed to their planned total depths; no drill holes were lost or abandoned due to technical or ground issues. A summary of the Phase 1 and Phase 2 Focus drill holes is presented in Figure 10-1.

The Phase 1 drilling was conducted on a nominal 800 by 800 m spaced grid covering approximately 27.36 km^2 (2,736 Ha) of the total 125.75 km^2 (12,575 Ha) of the Bayovar 12 Concession. The Phase 1 drilling program concentrated on the western portion of the Bayovar 12 Concession. The Phase 2 drilling expanded the nominal 800 by 800 m spaced drilling grid towards the east of the Phase 1 drilling, while also including some closer 400 by 400 m spaced drilling to allow for evaluation of shorter range thickness and grade variability. As of the effective date of this technical report, a significant portion of the concession remained undrilled.

Table 10-1: Phase 1 and Phase 2 Focus Drill Hole Summary

Hole Name	Easting (m)	Northing (m)	Elevation (m amsl)	Total Depth (m)	Overburden (m)	Diatomite (m)	Phosphorite (m)	Underburden (m)
JPQ_14_01	535,722	9,338,704	29.2	105.40	47.35	45.04	8.56	4.45
JPQ_14_02	536,519	9,337,103	23.1	103.00	50.02	38.70	10.18	4.10
JPQ_14_03	536,520	9,338,703	23.4	104.60	45.75	44.48	9.62	4.75
JPQ_14_04	536,518	9,337,903	21.7	103.10	46.55	43.29	9.14	4.12
JPQ_14_05	535,717	9,337,103	28.4	112.50	53.74	45.29	9.39	4.08
JPQ_14_06	534,919	9,338,701	28.4	131.30	71.15	46.71	8.87	4.57
JPQ_14_07	534,920	9,337,101	28.3	118.60	58.59	46.43	9.61	3.97
JPQ_14_08	537,318	9,338,702	1.1	90.00	30.81	44.54	9.39	5.26
JPQ_14_09	537,331	9,337,099	24.3	107.70	50.36	42.68	10.00	4.66
JPQ_14_10	537,327	9,337,905	7.4	91.40	34.33	42.97	9.54	4.56
JPQ_14_11	538,122	9,337,906	1.6	82.00	23.17	43.72	9.63	5.48
JPQ_14_12	535,718	9,337,902	25.1	104.90	47.09	43.82	10.09	3.90
JPQ_14_13	538,114	9,338,710	1.1	88.50	27.02	46.57	9.70	5.21
JPQ_14_14	534,919	9,337,891	26.6	104.40	44.93	47.25	8.60	3.62
JPQ_14_15	534,119	9,337,904	21.4	97.50	38.00	46.82	8.84	3.84
JPQ_14_16	537,727	9,337,502	3.5	81.10	23.91	44.45	7.92	4.82
JPQ_14_17	534,118	9,337,103	27.0	104.00	46.47	46.23	7.83	3.47
JPQ_14_18	537,725	9,338,301	0.6	85.50	26.36	45.92	7.69	5.53
JPQ_14_19	534,120	9,338,704	22.7	101.00	39.51	47.43	9.75	4.31
JPQ_14_20	537,314	9,336,302	29.4	110.80	52.31	42.74	10.27	5.48
JPQ_15_21	536,920	9,339,109	5.9	92.40	31.15	47.98	6.95	6.25
JPQ_15_22	536,921	9,338,303	4.5	91.40	30.13	37.72	6.91	5.58
JPQ_15_23	536,922	9,337,498	23.7	105.50	48.52	42.08	7.06	4.94
JPQ_15_24	536,924	9,336,708	24.0	105.00	47.96	45.43	6.21	5.40
JPQ_15_25	537,703	9,336,707	28.3	107.60	51.59	42.33	7.23	4.77
JPQ_15_26	539,341	9,336,716	0.3	91.80	33.10	44.09	6.95	5.45
JPQ_15_27	539,319	9,337,504	-0.7	90.60	30.81	44.02	7.47	5.40
JPQ_15_28	537,320	9,339,502	0.6	87.20	21.84	48.80	8.64	5.82
JPQ_15_29	539,319	9,339,105	0.5	90.50	26.90	46.99	7.44	5.66
JPQ_15_30	536,920	9,339,904	5.9	98.20	32.16	49.57	8.88	4.95
JPQ_15_31	538,119	9,339,508	1.4	86.70	21.94	47.26	9.80	5.47
JPQ_15_32	538,921	9,339,504	1.1	92.10	23.27	47.55	8.85	5.83
JPQ_15_33	537,720	9,339,900	1.2	91.30	28.60	47.63	6.99	6.84
JPQ_15_34	539,723	9,337,904	1.0	95.20	34.08	44.04	8.49	5.73
JPQ_15_35	537,719	9,338,703	0.3	88.20	25.00	45.48	7.73	5.43
JPQ_15_36	539,319	9,338,303	1.1	90.60	30.00	45.99	7.04	6.17
JPQ_15_37	537,319	9,338,304	0.2	84.40	22.38	43.56	9.01	5.01
JPQ_15_38	538,922	9,338,704	0.8	87.00	25.16	46.33	7.83	5.53
JPQ_15_39	538,113	9,338,305	1.1	85.20	22.62	47.22	6.97	5.40
JPQ_15_40	533,316	9,338,702	26.7	102.00	41.55	44.81	8.85	6.79
JPQ_15_41	533,319	9,337,106	27.0	104.50	43.42	49.45	8.56	3.07
JPQ_15_42	533,319	9,335,504	31.8	106.70	45.80	49.53	7.07	4.30

Hole Name	Easting (m)	Northing (m)	Elevation (m amsl)	Total Depth (m)	Overburden (m)	Diatomite (m)	Phosphorite (m)	Underburden (m)
JPQ_15_43	538,118	9,337,100	0.1	84.45	27.33	43.49	8.49	5.14
JPQ_15_44	538,521	9,339,103	1.1	87.80	25.23	46.70	8.84	7.03
JPQ_15_45	537,714	9,339,101	1.3	88.10	28.55	46.67	6.92	5.96
JPQ_15_46	538,520	9,338,306	1.1	86.40	23.86	48.29	7.54	6.71
JPQ_15_47	538,522	9,337,506	0.8	82.50	21.60	47.08	5.68	6.42
JPQ_15_48	538,520	9,336,703	1.0	86.10	28.33	43.26	7.95	6.56
JPQ_15_49	538,121	9,336,301	1.8	85.00	30.00	42.08	7.74	5.18
JPQ_15_50	538,921	9,336,303	0.4	88.90	32.99	43.48	7.01	5.42
JPQ_15_51	538,920	9,337,106	0.8	87.50	31.25	43.43	7.45	5.37
JPQ_15_52	538,919	9,337,906	1.0	90.00	30.98	45.38	7.93	5.71
JPQ_15_53	539,718	9,336,306	1.5	97.00	41.46	41.24	7.96	6.34
JPQ_15_54	539,722	9,337,106	0.1	95.40	40.17	42.54	7.15	5.54
JPQ_15_55	540,522	9,337,883	0.7	100.50	39.40	43.71	7.22	6.51
JPQ_15_56	537,721	9,337,904	0.1	82.20	19.65	45.79	7.61	5.11
JPQ_15_57	541,321	9,337,898	0.4	105.60	44.28	42.01	7.94	6.49
JPQ_15_58	540,517	9,337,100	-0.5	100.40	42.47	43.29	6.56	6.45
JPQ_15_59	540,518	9,338,702	0.1	96.20	33.55	45.56	7.00	6.29
JPQ_15_60	536,119	9,339,903	29.7	120.20	54.30	49.07	7.17	9.66
JPQ_15_61	536,517	9,339,505	26.5	112.60	52.20	44.70	7.52	6.88
JPQ_15_62	539,723	9,338,702	1.5	93.00	31.86	44.42	8.63	5.82

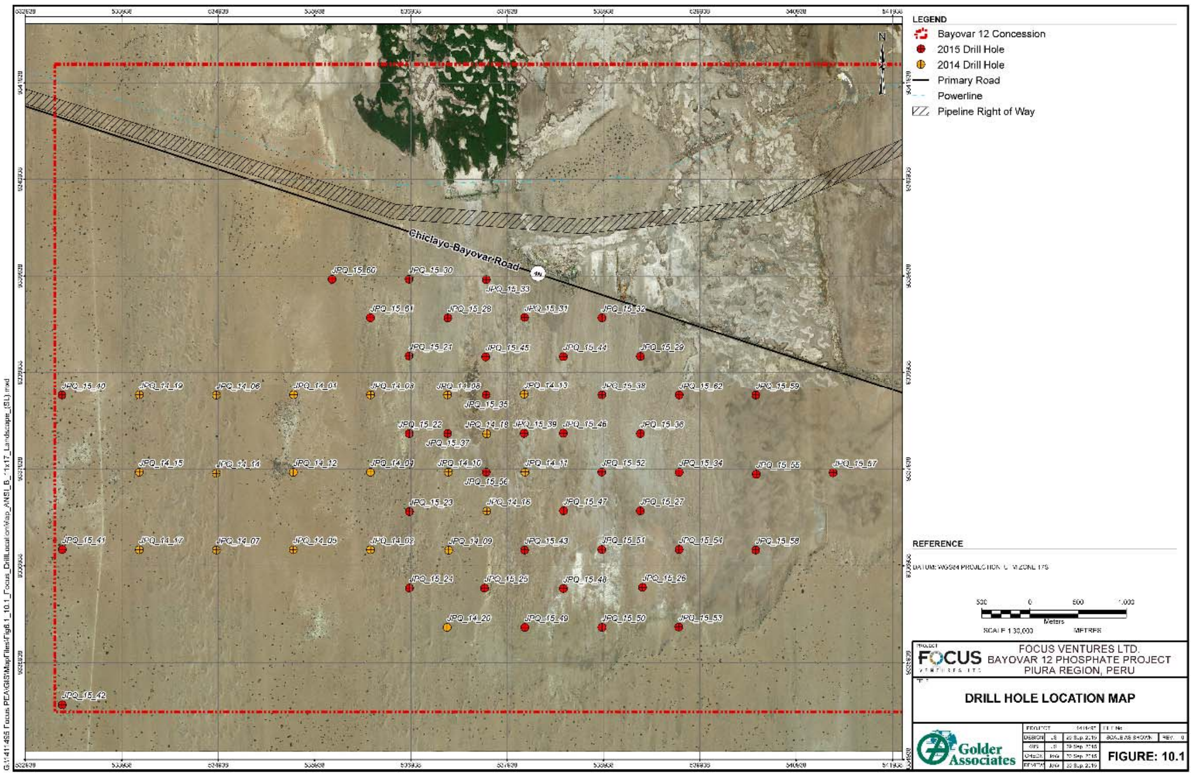


Figure 10-1: Drill Hole Location Map

10.2 DRILLING RESULTS

All 62 of the Phase 1 and Phase 2 drill holes intercepted the full sequence of target phosphorite beds. The entire sequence of 16 phosphorite beds spanned a total mean thickness of 47.4 m (range of 43.3 to 50.6 m) including interburden diatomite beds. Depths from surface to the roof of the upper most phosphorite bed (PH01) ranged from 62.6 m below surface in the Tablazo area to 25.5 m below surface in the low area to the east of the Tablazo (overall mean of 39.8 m below surface). The floor of the lower most phosphorite bed (PH16) ranged from 104.4 m below surface in the Tablazo area to 73.1 m below surface in the low area to the east (overall mean of 86.8 m below surface).

Drill hole thickness and grade statistics for the individual phosphorite beds are presented in Table 10-2. Drill hole thickness and grade statistics for the individual diatomite beds are presented in Table 10-3, Diatomite Bed Thickness and P_2O_5 Grade Summary Statistics Geological sections and isopleth maps are presented in Section 14 of this technical report.

Table 10-2: Phosphorite Bed Thickness and P_2O_5 Grade Summary Statistics

Phosphorite Bed	Intercept Count	Mean Thickness (m)	Minimum Thickness (m)	Maximum Thickness (m)	Mean P_2O_5 (wt.%)	Minimum P_2O_5 (wt.%)	Maximum P_2O_5 (wt.%)
PH01	62	0.50	0.22	0.72	12.60	6.14	17.12
PH02	62	0.89	0.47	1.26	11.77	7.13	15.87
PH03	62	0.48	0.10	0.94	19.59	10.89	24.32
PH04	60	0.31	0.10	1.04	16.08	6.78	23.72
PH05	61	0.45	0.10	1.06	9.49	5.08	16.07
PH06	62	0.58	0.25	0.92	12.99	6.35	19.12
PH07	62	0.60	0.24	1.58	10.20	5.56	14.94
PH08	62	0.48	0.10	1.45	10.85	7.00	20.83
PH09	62	0.50	0.10	0.87	11.72	7.08	18.31
PH10	62	0.47	0.10	1.03	10.70	5.88	16.71
PH11	62	0.40	0.16	0.89	13.38	7.04	19.56
PH12	62	0.51	0.10	0.94	14.82	6.41	23.64
PH13	62	1.06	0.43	1.76	13.47	7.94	19.23
PH14	57	0.27	0.10	1.00	9.15	4.81	17.37
PH15	55	0.40	0.10	1.04	8.55	4.05	15.85
PH16	61	0.36	0.10	0.97	7.44	2.91	11.47
All Beds	976	0.52	0.10	1.76	12.25	2.91	24.32

Note: Mean grades are thickness weighted.

Table 10-3: Diatomite Bed Thickness and P_2O_5 Grade Summary Statistics

Diatomite Bed	Intercept Count	Mean Thickness (m)	Minimum Thickness (m)	Maximum Thickness (m)	Mean P_2O_5 (wt.%)	Minimum P_2O_5 (wt.%)	Maximum P_2O_5 (wt.%)
IB02	62	8.76	7.85	9.93	1.85	1.12	3.54
IB03	61	1.45	0.77	1.99	1.77	1.00	3.72
IB04	62	1.40	0.54	2.83	1.62	1.00	3.31
IB05	62	1.96	1.39	2.76	2.07	1.00	4.71
IB06	62	2.32	1.57	3.20	2.70	1.00	5.44
IB07	62	2.43	1.80	3.38	2.96	1.30	5.19
IB08	62	2.87	1.66	3.54	3.56	1.91	5.15
IB09	62	1.67	0.90	2.60	4.05	2.39	5.59
IB10	62	2.15	0.63	3.64	4.66	1.45	6.50
IB11	62	2.96	2.26	4.41	1.51	1.00	3.18
IB12	62	3.48	2.31	4.74	3.28	1.83	5.71
IB13	62	1.11	0.41	1.90	3.62	2.00	5.64
IB14	61	1.25	0.46	1.79	3.31	1.56	6.24
IB15	62	2.05	1.10	2.97	2.42	1.00	3.70
IB16	62	3.35	1.23	4.55	2.16	1.00	5.29
All Beds	928	2.62	0.41	9.93	2.60	1.00	6.50

Note: Mean grades are thickness weighted.

Core recovery for all units was very good, with mean core recovery of 99% (range of 42% to 100%) during the Phase 1 program and mean core recovery of 99% (range of 19% to 100%) during the Phase 2 program. There were 125 occurrences with core recovery less than 90% and 19 occurrences where core recovery was less than 50%. The core recovery within the phosphorite beds mirrored the overall recovery values with a mean of 99% (range of 42% to 100%) but with 34 occurrences of recovery less than 90% and two occurrences where core recovery was less than 50%.

Overall the Rock Quality Designation (RQD) for all units was fair to excellent with a mean RQD of 76% (range of 0% to 100%). The RQD within the phosphorite beds showed a slight improvement over the RQD for all units, with a mean of 85% (range of 0% to 100%), considered good to excellent.

A total of 6,980 half core (hand split) samples were collected through the entire diatomite (4,494 samples) and phosphorite (2,845 samples) sequence in all 62 drill holes. Sample interval lengths ranged from 0.08 to 0.97 m (mean of 0.25 m) in the phosphorite and 0.01 to 1.27 m (mean of 0.58 m) in the diatomite. A detailed discussion of the analysis methods and the analytical results from the sampling program are presented in Section 11 of this technical report.

10.3 DRILLING PROCEDURES AND METHODOLOGY

The following sections detail the exploration drilling program procedures and methodology employed by Focus during the Phase 1 and Phase 2 Focus exploration programs.

10.3.1 Drilling Methodology

All 62 drill holes in the Phase 1 and Phase 2 Focus Exploration programs were drilled by RAM Peru S.A.C using two skid-mounted Boart Longyear (one LY-44 and one LM-75 model) wireline drill rigs (Figure 10-2) with a maximum depth capacity of approximately 530 m for HQ core drilling. All 62 drill holes were drilled vertical, recovering HQ size (63.5 mm core diameter) core. Downhole directional surveys were not performed on the drill holes; given the short total length of the holes, the orientation of stratigraphy or fabrics in the rocks (oriented normal to the drill hole) and the broad overall drill spacing (400 to 800 m centres) lateral deviation of the drill holes was deemed negligible.

Drilling was conducted on a single 12 hour shift each day. Typical drilling rate was 1 to 1.5 days per drill hole. Drilling of the 20 Phase 1 drill holes commenced on March 1, 2014 and was completed on April 5, 2014 while drilling of the 42 Phase 2 drill holes commenced on April 14, 2015 and was completed on May 20, 2015.

Drill site supervision, core logging and sampling duties were performed by Focus senior geologists and technical personnel.

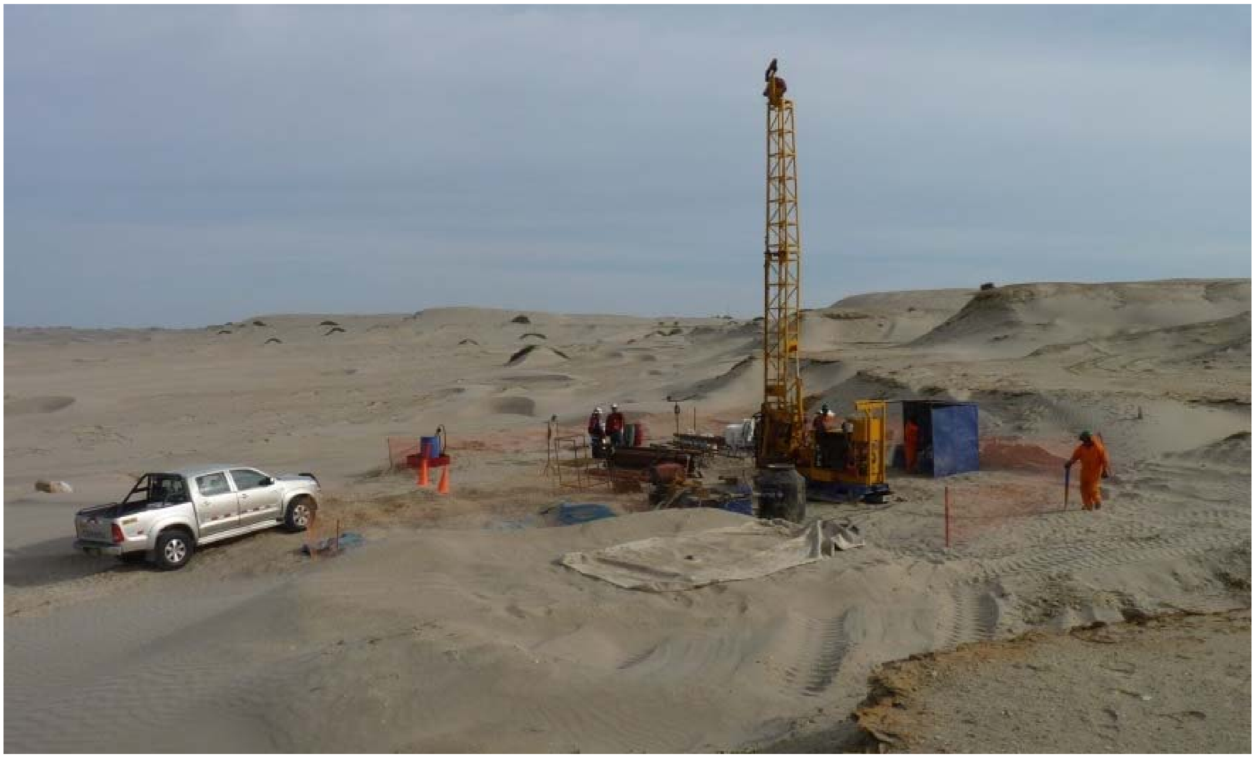


Figure 10-2: Drilling on the Bayovar 12 Concession

10.3.2 Drill Hole Location Methodology

All drill hole platforms were located by Focus senior geologists using a handheld Garmin GPS. The planned drill holes were located on a nominal 800 by 800 m spaced grid with the exception of a small area where drilling was performed on 400 by 400 m centres to allow for evaluation of short range variability. In total the Phase 1 and Phase 2 drilling covered approximately 27.36 km^2 (2,736 Ha).

After completion, drill holes are sealed with a cement monument (Figure 10-3) and marked clearly using PVC pipe or wooden stakes to withstand wind and sand dune cover. The drill hole name, total depth and completion date were inscribed in the cement monument prior to the cement setting.

On completion of the drill program all cement monuments were surveyed by a professional land surveyor using Total Station GPS to record collar surveys to an accuracy of +/-0.1m in X, Y and Z dimensions. The surveying was performed by Peruanas de Inversiones R & L S.A.C., of Chiclayo Peru.

Figure 10-3: Cement monument marking 2014 Bayovar 12 Drill Hole

10.3.3 Core Handling and Visual Logging Methodology

Exploration Data Collection and Documentation

All measurements, observations, sample intervals and other associated information collected during the core logging and sampling process were recorded by the Focus geologists directly into an Excel drill hole logging datasheet for each individual drill hole. All digital data entry was performed by the geologist at the time of logging rather than being transcribed at a later date. All information pertaining to an individual drill hole was recorded on specialized tabs within the single excel file for that drill hole, allowing for single source for records for each individual drill hole. The drill hole logging datasheet included tabs for:

- Collar location and completion details;
- Drill hole orientation;
- Downhole lithology observations;
- Phosphate mineralization observations;
- RQD and Total Core Recovery measurements and calculations;
- Geotechnical observations and measurements;
- Sampling intervals and analytical QA/QC insertion records; and,

- Imported analytical results

Core Handling

Core was boxed at the drill site and transported to the secure core logging facility (Figure 10-4) by Focus or RAM drilling personnel on a daily basis. The core boxes were laid out sequentially and visually inspected to ensure all boxes were accounted for and that core boxes and depth markers were clearly labeled and in the correct downhole order.

For the phase 1 drill program the Focus core logging facility was located inside the secure (gated and armed guard) JPQ port facility. The core logging area was purpose built and included areas for logging, core splitting and sampling. Core was stored on covered steel core racks while awaiting logging and sampling. For the Phase 2 drilling program up to present, logging and storage facilities were re-located to a secured property in Piura (Figure 10-5).

Figure 10-4: Phase 1 Focus Core Logging Facility

Figure 10-5: Core Storage Racks at the Focus Phase 2 Core Logging Facility

Core Photography

Before core splitting, logging and sampling commenced each complete core box was photographed using a tripod mounted digital camera and photos were archived for reference purposes (Figure 10-6). The core photography set up was standardized so that all core box photographs were consistent in terms of quality, scale and resolution. Drill hole names and depth marker blocks were oriented so as to be clearly read in each photograph. In addition to the core box photographs, detailed close-up photographs were also taken for any stratigraphic, structural, mineralization or other features of interest. All photos were labeled using a systematic numbering system that clearly indicates drill hole name and depth interval in the photograph name. Photographs were reviewed by the geologist to ensure they met the required standards prior to splitting the core.

Geotechnical Logging

Once photographed the core was reoriented and fitted together as appropriate prior to measuring the length of core recovered for every drill interval and calculating the Total Core Recovery; the recovered length and Total Core Recovery were recorded in the drill logging sheet. The RQD was also calculated for each drill run by counting the number of whole core pieces that were equal to or greater than 10 cm in length. The RQD results were recorded in the drill logging datasheet. Observations on the spacing, frequency and infill material on joints and fractures were also recorded in the drill logging datasheet.

Figure 10-6: Example Core Box Photograph

Core Splitting

Once the Total Core Recovery, RQD and other geotechnical observations were recorded on the whole core, a longitudinal cut line was marked on the core by the logging geologist. The logging geologist also marked the core to indicate which side would be sampled and which side would be retained for reference so that the same side of the core was submitted from consecutive sample intervals to avoid any sample selection bias.

Once marked up the core was moved to the sample splitting stations in the Focus core facility. The core for the entire drill hole was split longitudinally by hand by Focus technical personnel. A core technician placed individual core segments in a form to hold the core in place on the splitting bench. A 3 to 5 mm deep groove was then cut along the longitudinal cut line on the core segment using a hand saw (Figure 10-7). A cleaver was then placed in the cut and gently struck with a hammer to split the core in half (Figure 10-8). This procedure was used on most of the core except in instances where the material was very soft, in which case it was split by a knife or sampling spoons. This core splitting method allowed for the best results in terms of maintaining core segment integrity (over a mechanical vise style sample splitter or water cooled saw) as well as providing a natural fracture surface that allowed for easy evaluation of fine textural features and estimation of apatite pellets that was not possible on the polished cut surfaces created by a rock saw.

Figure 10-7: Cutting Longitudinal Line on Core Prior to Splitting

Figure 10-8: Splitting Core with Cleaver

Descriptive Logging

Once split, both halves of the core segments were carefully returned to the core box and then the boxes were returned to the core logging benches and laid out in down hole order to allow for detailed logging of the lithology, structure and mineralization.

Lithology descriptions included down hole depth intervals, color, grain size, porosity, facies type, interpreted unit name and a detailed comment or description of each interval. A new interval record was created in the descriptive log any time a change in the color, grain size, facies or geological unit occurred. The logging geologist marked observed and interpreted geological unit and facies interval boundaries on the core during the visual logging process (Figure 10-9). All lithology interval descriptions were then recorded directly into the drill logging datasheet by the logging geologist (Figure 10-10).

Figure 10-9: Geologist Logging Core

Figure 10-10: Geologist Entering Data and Observations

Grade Estimation

Once the lithology logging process was completed and the geological unit and facies boundaries were determined and marked on the core by the logging geologist, the diatomite and phosphorite intervals identified in the visual logging process were inspected by the logging geologist using a magnified hand lens. Using the hand lens and a set of percentage composition estimation charts the logging geologist identified grade zones based on estimating the apatite pellet contents in each interval (Figure 10-11). Individual diatomite and phosphorite intervals were subdivided into multiple grade intervals whenever there were 5% changes (positive or negative) in the estimated pellet content.

The grade estimation process for all 62 Phase 1 and Phase 2 drill holes was performed by two senior geologists to minimize variability and/or bias in the grade estimation process. The two senior geologists regularly cross checked each other's estimates to ensure they were consistently estimating grade intervals.

Once the grade intervals were established and marked on the drill core, the logging geologist recorded the percentage, type (pellets, ooids, teeth, bone fragments etc.), grain size, shape, color and a detailed description for each interval directly into the drill logging datasheet.

Once the grade estimation process was completed the core was ready for the sampling process. A detailed discussion of the sampling, analyses and analytical QA/QC process is presented in Section 11 of this technical report.

Figure 10-11: Geologist Estimating Pellet Content in Phosphorite and Diatomite Bed

10.4 DRILLING FACTORS IMPACTING ACCURACY AND RELIABILITY OF RESULTS

It is Golder's opinion that the Phase 1 and Phase 2 exploration programs were carried out by Focus personnel according to appropriate professional methodologies and procedures, including those presented in the CIM Exploration Best Practice Guidelines (August 2000 edition). The methodology and procedures were well defined and documented prior to commencing with the drilling and sampling programs. All components of the programs were conducted according to the methodology and procedures and were well documented by Focus technical personnel. All Phase 1 and Phase 2 exploration work for the drilling programs appears to have been performed by experienced and qualified personnel, including Focus personnel as well as third party contractors.

The overall drill core recovery was very good (mean of 99%), and was very good within the phosphorite beds (mean of 99%). The RQD was fair to excellent overall (mean of 76%) and was good to excellent within the phosphorite beds (mean of 85%). All drill holes were completed to their planned total depths and all drill holes intercepted the complete sequence of phosphorite and diatomite beds as predicted prior to drilling.

Golder is not aware of any factors or concerns regarding the accuracy and reliability of the results from the Phase 1 and Phase 2 Focus exploration programs.

10.5 INTERPRETATION OF DRILLING RESULTS

The drilling results from the Phase 1 and Phase 2 exploration programs were reviewed and interpreted independently by the Focus senior geologists and by the Golder Qualified Person. Drill hole lithology and grade data was used to confirm the roof and floor picks for each of the phosphorite intervals as well as the various overburden, interburden and underburden units. Drill hole fences were used to confirm the hole to hole correlation of the phosphorite beds and overburden, interburden and underburden units. A detailed discussion of the interpretation and geological modelling process is presented in Section 14 of this technical report.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE SUMMARY

Sampling and phosphate analyses activities on the Bayovar 12 Concession to date have been limited to sampling of exploration drill core during the Focus Phase 1 and Phase 2 exploration programs.

As part of the Phase 1 and Phase 2 exploration programs a total of 6,980 half core (hand split) samples were collected through the entire diatomite (4,494 samples) and phosphorite (2,845 samples) sequence in all 62 drill holes. Sample interval lengths ranged from 0.08 to 0.97 m (mean of 0.25 m) in the phosphorite and 0.10 to 1.27 m (mean of 0.58 m) in the diatomite.

All Phase 1 and Phase 2 samples were submitted to Certimin laboratory in Lima, Peru for primary analyses. Pulp duplicates were submitted to the SGS Laboratory in Lima for secondary check assay analyses.

The standard analytical package performed on all diatomite and phosphorite samples was as follows:

- P_2O_5 (gravimetric analysis);
- major oxides (ICP-OES analysis); and,
- SiO_2 (gravimetric analysis).

In addition to the analyses indicated above, samples were collected and submitted to Certimin for moisture and relative density analysis by water displacement method during Phase 1 and Phase 2 drilling programs. Due to sample handling and processing issues with the Phase 1 density samples it was determined by Focus and Golder that the Phase 1 relative density samples had experienced moisture loss due to air drying and the samples and analytical results were deemed unreliable for determining moisture content and relative density. During Phase 2 exploration work Focus personnel applied special sample selection and handling procedures to ensure collection of reliable relative density samples for the individual phosphorite and interburden beds as well as for the general overburden and underburden units.

The following sections detail the sample selection, collection, transport, preparation and analyses procedures and methodology employed by Focus during the Phase 1 and Phase 2 exploration programs.

11.2 SAMPLING METHODOLOGY AND PROCEDURES

11.2.1 Sample Interval Identification

Sample intervals were marked on the drill core and recorded in the drill logging datasheet by the Focus logging geologist. Sample interval lengths ranged from 0.08 to 0.97 m (mean of 0.25 m) in the phosphorite and 0.10 to 1.27 m (mean of 0.58 m) in the diatomite.

Focus sampled the entire continuous sequence of diatomite and phosphorite in each drill hole so that there were no gaps in the downhole sampling record. This was done to provide detailed interburden dilution grade data and to allow for the potential evaluation of bulk mining of closely spaced during later modelling, mine planning and processing activities. The continuous sampling also provided flexibility in adjusting phosphorite and diatomite boundaries should the analytical results support such adjustment in areas where the contacts were gradational or grade variation was difficult to assess.

11.2.2 Sample Collection and Packaging

Once each sample interval was recorded in the drill logging datasheet the logging geologist selected each sample in sequence (Figure 11-1) and placed each one inside a plastic sample bag pre-labeled with the sample number. A sample tag was also placed inside the sample bag before the bag was sealed with a cable tie (Figure 11-2). When the core was moist the phosphorite samples were wrapped in brown paper (Figure 11-3) before being placed in the sample bags to prevent the phosphorite sample from sticking to the sample bag. The sample number and the sample interval from and to depths were recorded in the directly into the drill logging datasheet by the logging geologist.

The sealed sample bags were then placed in 70 litre plastic sample barrels (Figure 11-4). Each sample barrel held 43 packaged samples; once full the barrels were sealed with a metal clamp and were held in the core facility until a shipment batch (approximately 8 barrels) were ready for transport to the primary analytical laboratory facility.

The remaining un-sampled core was carefully reorganized in the core box and the lids were returned to the boxes before they were racked on the metal core storage racks at the logging facility.

Figure 11-1: Geologist Sampling Core

Figure 11-2: Bagged Samples

Figure 11-3: Wrapping Phosphorite Sample in Brown Paper to Prevent Sticking in Sample Bag

Figure 11-4: Samples Packed for Shipping

11.2.3 Insertion of Field Quality Assurance/Quality Control Standards

Field QA/QC samples were inserted into the sampling stream by the logging geologist during the sampling process. The QA/QC field standards used by Focus included:

- ¼ Core Duplicates – ¼ core duplicate assay sample (using half of the split core that was retained for reference purposes.
- Coarse Blanks – coarse, locally sourced diatomite from the barren Upper Diatomite Series (above the Diana ore zone). Five samples were processed and analysed for P_2O_5 (gravimetric analysis) at Certimin and 5 samples were analysed at SGS (both laboratories located in Lima, Peru); all of the coarse blank characterization samples returned P_2O5 values of less than 5 wt.%.
- Pulp Blanks – Three commercially prepared pulp blanks were purchased from Canadian Resource Laboratories Ltd. of Canada. The pulp blanks (P5B, CDN-BL-4 and CDN-BL-10) were sourced from igneous rocks and were not ideally suited to phosphate and will be replaced with a more appropriate matrix matched commercial pulp blank in the future.
- Certified Reference Materials – Four commercially prepared certified reference material standards (CRM's) were purchased from Geostats Pty Ltd of Australia. The CRM's were matrix matched from sedimentary phosphate in Tunisia and Australia. The following standards were used by Focus:
  - GPO 14 - 24.52 wt.% P_2O_5
  - GPO 16 - 17.76 wt.% P_2O_5

- GPO 17 – 13.55 wt.% P_2O_5
- GPO 18 – 15.09 wt.% P_2O_5

The field standards were inserted randomly into the sample number sequence by the logging geologist. The field standards were placed in a plastic sample bag and secured with a cable tie in the same fashion as the regular analytical samples. Generally, one CRM and one blank (either coarse or pulp blank) was inserted every 20 samples but were not inserted at regular intervals or at the same location in each drill hole. The logging geologist generally tried to insert the mineralized CRM standards within zones of similar estimated grade of mineralization in the sample sequence before, within or immediately after the mineralized sample. Blanks were inserted at the end of mineralized runs to measure carry-over etc.

Each sample submission batch was nominally the same size (43 samples) and was designed to include as a minimum the following standards in each batch:

- 2 ¼ Core duplicates
- 2 CRM's (grade matched to the estimated mineralization)
- 2 blanks (either coarse or pulp)

A detailed discussion of the QA/QC analysis and results is presented later in this section.

11.3 SAMPLE PREPARATION AND ANALYTICAL METHODOLOGY AND PROCEDURES

11.3.1 Primary and Secondary Analytical Laboratories

All sample preparation and primary analyses for all samples from the Phase 1 and Phase 2 exploration programs was performed at the Certimin S.A (Certimin) laboratory in Lima Peru. The Certimin laboratory is an ISO 9001:2008 and ISO 14001:2004 certified and Peruvian Government National Accreditation Service (INDECOPI) accredited analytical laboratory with certificates in good standing (certificate renewal date : May 2019). Certimin has significant experience providing analytical services to the phosphate exploration and other exploration industries in Peru.

As part of the Phase 1 and Phase 2 analytical QA/QC programs, select pulp duplicates were submitted to the SGS del Peru S.A.C. (SGS) Laboratory in Callao (Lima) Peru for the purpose of performing check assay analyses. The SGS laboratory is an ISO 9001:2008 certified and Peruvian Government National Accreditation Service (INDECOPI) accredited analytical laboratory with certificates in good standing (certificate renewal date : December 2017). SGS has significant experience providing analytical services to the phosphate exploration and other exploration industries in Peru.

11.3.2 Sample Preparation

All Phase 1 and Phase 2 sample preparation work was performed at the Certimin laboratory. Focus personnel delivered the samples to the sample receiving area (Figure 11-5). Certimin immediately inspected the sample batch and sample submission sheets from Focus to ensure all samples were accounted for and the required analyses were clearly indicated.

Figure 11-5: Certimin Sample Reception and Check-In Area

The Focus samples were then checked into the laboratory computer database and a bar code label was printed and placed on each sample bag; a unique sample bar code was provided for the sample preparation phase and then a second unique sample bar code was provided for the analytical sample upon completion of the sample preparation process.

Once the samples were checked in to the database they were weighed on a balance (Figure 11-6) and the value was entered in to the database sample record. Temperature and humidity were recorded (once daily at 8 am) in the sample reception and weighing area to ensure that all samples are processed and weighed under constant conditions.

The samples were then placed on trays and dried in an oven (Figure 11-7) to remove any free moisture.

Following the drying process, the samples move to the primary crusher (Figure 11-8) where they were crushed to pass a 2 mm (#10 mesh) screen. Quartz was used to clean the crusher after every 10 samples.

The crushed material was then passed through a riffle splitter to separate the sample and reject material. Every 43 samples Certimin creates a lab duplicate sample; for batches of less than 43 samples they will prepare a minimum of one lab duplicate sample.

The reject from the riffle splitter was placed in a sample bag and then placed in rice bags and stored in the reject storage area (stored for up to three months).

Figure 11-6: Sample Weighing Station

Figure 11-7: Sample Drying Oven

Figure 11-8: Primary Crusher

Following crushing and splitting, the sample then moved to a disc grinder (Figure 11-9) where it was milled to pass a 106 micron (#140 mesh) screen. The disc grinder was cleaned with quartz every 5 samples.

The milled product was then weighed, entered into the computer system and placed in a sample envelope. A new sample bar code was assigned to the milled sample and the sample was placed in a box (Figure 11-10) with the other samples from the batch prior to being delivered to the analytical laboratory for analyses.

Prior to delivering the sample boxes to the analytical laboratory the laboratory internal standards and replicates were inserted into the sample batch. The laboratory internal standards were assigned sample number bar codes and were packaged in the same manner as the analysis samples so they couldn't be identified as standards by the analytical laboratory personnel.

Figure 11-9: Disc Grinder

Figure 11-10: Boxed Samples Ready for Analysis

The Laboratory internal standards and replicates were inserted into each sample batch according to the following schedule:

- 2 laboratory duplicates;
- 1 blank;
- 1 standard rock;
- 1 standard oxide;
- 1 pulp duplicate; and,
- 1 reject duplicate.

Following the completion of the analyses, the remaining analysis sample material was returned to the sample preparation area and the samples were stored in boxes (Figure 11-11) along with the sample reject material for three months.

Typical sample preparation processing time, from receipt of the samples to delivery to the analytical lab, was two days.

Figure 11-11: Sample Storage Area

11.3.3 Sample Analyses

The standard analytical package performed on all diatomite and phosphorite samples was as follows:

- P_2O_5 by gravimetric analysis (Figure 11-12);

- major oxides by ICP-OES analysis (Figure 11-13); and,
- SiO_2 by gravimetric analysis.

During the Phase 1 analytical program the standard analytical package was performed on all samples (both phosphorite and diatomite) while during the Phase 2 program P_2O_5 (gravimetric analysis) was first performed on all samples followed by major oxides (ICP-OES analysis) and SiO2 (gravimetric analysis).

In addition to the analyses indicated above, samples were collected and submitted to Certimin for moisture and relative density analysis by water displacement method during Phase 1 and Phase 2 drilling programs. Due to sample handling and processing issues with the Phase 1 density samples it was determined by Focus and Golder that the Phase 1 relative density samples had experienced moisture loss due to air drying and the samples and analytical results were deemed unreliable for determining moisture content and relative density. During Phase 2 exploration work Focus personnel applied special sample selection and handling procedures to ensure collection of reliable relative density samples for the individual phosphorite and interburden beds as well as for the general overburden and underburden units.

Balances for the gravimetric analyses undergo annual calibration/certification and are checked daily using a set of mass standards. The Certimin ICP-OES is set up to run a check analysis on a suite of internal standards and blanks every 30 samples.

Following completion of the required analyses the results are reviewed by the laboratory internal QA/QC manager to ensure all internal standard and replicate analysis results are within the accepted tolerance. Once approved the database reassigns the original client sample number to the record in the database and the analysis certificates and data spreadsheets are prepared and delivered to the client.

11.3.4 Analytical Results

Analyses were performed by Certimin on a total of 6,980 samples from the entire diatomite (4,494 samples) and phosphorite (2,845 samples) sequence in all 62 drill holes.

A summary of the analytical results for the individual phosphorite beds is presented in Table 11-1. A summary of the analytical results for the individual diatomite beds is presented in Table 11-2.

Figure 11-12: Gravimetric Analysis for P_2O_5

Figure 11-13: ICP-OES analysis for major oxides

Table 11-1: Summary of Phosphorite Bed Analytical Results

Phosphorite Bed	Intercept Count	P_2O_5 (wt.%)			Al_2O_3 (wt.%)			CaO (wt.%)			Fe_2O_3 (wt.%)		
		Mean	Min	Max	Mean	Min	Max	Mean	Min	Max	Mean	Min	Max
PH01	62	12.60	6.14	17.12	3.11	3.42	6.12	13.97	10.26	25.89	1.97	2.39	3.77
PH02	62	11.77	7.13	15.87	3.95	2.60	7.70	19.54	12.40	25.98	2.03	1.62	2.84
PH03	62	19.59	10.89	24.32	2.63	1.78	4.66	29.35	18.08	38.39	1.21	0.84	1.78
PH04	60	16.08	6.78	23.72	2.83	1.84	5.54	25.93	11.77	37.33	1.47	0.83	2.19
PH05	61	9.49	5.08	16.07	3.61	1.92	6.75	16.35	8.68	25.40	1.84	1.51	2.58
PH06	62	12.99	6.35	19.12	3.57	2.44	5.74	20.94	10.14	30.40	1.85	1.34	2.44
PH07	62	10.20	5.56	14.94	2.40	2.42	4.51	12.32	10.16	23.09	1.49	1.71	2.61
PH08	62	10.85	7.00	20.83	1.98	1.89	3.77	12.87	12.93	31.01	1.24	1.22	2.26
PH09	62	11.72	7.08	18.31	2.26	1.93	4.17	14.99	13.15	28.30	1.44	1.30	2.56
PH10	62	10.70	5.88	16.71	2.39	2.23	4.86	11.34	10.56	25.58	1.50	1.71	2.68
PH11	62	13.38	7.04	19.56	3.93	2.53	5.92	21.68	12.03	29.70	2.03	1.47	2.38
PH12	62	14.82	6.41	23.64	4.09	2.14	6.76	24.20	9.83	37.14	2.09	1.14	2.99
PH13	62	13.47	7.94	19.23	4.71	3.07	8.21	22.67	15.50	30.23	2.44	1.78	3.40
PH14	57	9.15	4.81	17.37	1.62	1.70	3.72	10.13	7.81	26.84	1.08	1.34	2.46
PH15	55	8.55	4.05	15.85	2.15	2.32	5.15	12.82	11.81	27.64	1.42	1.67	2.68
PH16	61	7.44	2.91	11.47	2.00	1.65	4.32	14.12	11.35	27.48	1.27	1.17	2.59
All Beds	976	12.25	2.91	24.32	3.22	1.65	8.21	18.30	7.81	38.39	1.76	0.83	3.77

Note: *Mean grades are thickness weighted*
Min = minimum value
Max = Maximum value

Table 11-2: Summary of Diatomite Bed Analytical Results

Diatomite Bed	Intercept Count	P_2O_5 (wt.%)			Al_2O_3 (wt.%)			CaO (wt.%)			Fe_2O_3 (wt.%)		
		Mean	Min	Max	Mean	Min	Max	Mean	Min	Max	Mean	Min	Max
IB02	62	1.85	1.12	3.54	2.25	2.09	5.02	2.44	2.19	6.15	1.99	2.60	3.83
IB03	61	1.77	1.00	3.72	1.74	1.64	4.16	2.57	1.86	7.44	1.50	1.88	2.68
IB04	62	1.62	1.00	3.31	1.34	1.33	3.27	3.99	4.43	8.99	1.15	1.38	2.11
IB05	62	2.07	1.00	4.71	1.92	1.70	5.27	3.34	2.96	8.48	1.61	1.95	2.96
IB06	62	2.70	1.00	5.44	2.35	1.45	5.35	5.45	4.26	10.65	1.70	1.75	3.08
IB07	62	2.96	1.30	5.19	1.63	1.58	3.62	4.07	3.28	9.10	1.29	1.68	2.29
IB08	62	3.56	1.91	5.15	1.57	1.52	4.73	4.25	4.08	8.56	1.22	1.54	2.83
IB09	62	4.05	2.39	5.59	2.18	2.39	4.20	6.04	5.00	12.94	1.64	2.23	2.93
IB10	62	4.66	1.45	6.50	2.09	2.24	4.49	6.76	7.00	13.49	1.49	2.01	3.16
IB11	62	1.51	1.00	3.18	1.73	1.85	3.53	1.81	1.88	5.75	1.53	2.15	2.64
IB12	62	3.28	1.83	5.71	2.32	2.59	4.70	4.91	4.98	11.54	1.75	2.33	3.02
IB13	62	3.62	2.00	5.64	2.25	1.83	4.54	7.70	5.22	18.66	1.67	1.82	3.28
IB14	61	3.31	1.56	6.24	1.86	1.74	5.67	3.77	3.90	14.78	1.42	1.92	3.37
IB15	62	2.42	1.00	3.70	1.88	1.72	4.19	4.76	5.07	10.98	1.51	2.06	2.60
IB16	62	2.16	1.00	5.29	2.25	2.20	5.32	3.55	3.12	8.41	1.88	2.55	3.38
All Beds	928	2.60	1.00	6.50	2.03	1.33	5.67	3.93	1.86	18.66	1.65	1.38	3.83

Note: *Mean grades are thickness weighted*
Min = minimum value
Max = Maximum value

Table 11-3: Summary of Diatomite Bed Analytical Results (Continued)

Diatomite Bed	Intercept Count	K_2O (wt.%)			MgO (wt.%)			Na_2O (wt.%)			SiO_2 (wt.%)		
		Mean	Min	Max	Mean	Min	Max	Mean	Min	Max	Mean	Min	Max
IB02	62	0.36	0.39	0.76	1.07	1.30	2.59	1.53	0.99	7.08	35.85	48.97	60.77
IB03	61	0.29	0.33	0.61	0.97	0.99	2.29	1.58	1.18	7.62	39.22	49.18	66.34
IB04	62	0.23	0.26	0.54	1.73	1.94	3.65	1.43	1.17	7.44	36.90	48.22	64.87
IB05	62	0.30	0.31	0.69	1.12	1.04	2.60	1.48	1.17	7.16	39.34	51.17	64.26
IB06	62	0.31	0.31	0.68	2.26	1.09	5.04	1.08	1.02	4.69	32.01	41.95	63.42
IB07	62	0.25	0.29	0.52	1.12	1.05	2.97	1.21	1.18	5.91	37.16	50.91	62.60
IB08	62	0.24	0.26	0.65	1.00	0.98	2.67	1.17	1.27	5.49	37.65	51.75	65.23
IB09	62	0.30	0.32	0.61	1.88	1.62	4.13	1.02	1.12	3.80	31.80	40.39	58.00
IB10	62	0.29	0.33	0.66	1.88	1.89	4.80	1.02	1.19	3.48	31.06	43.42	55.60
IB11	62	0.26	0.31	0.52	0.64	0.79	1.73	1.14	1.35	3.95	42.17	60.33	68.06
IB12	62	0.31	0.37	0.63	1.53	1.59	3.34	0.93	1.16	2.68	33.80	46.86	58.17
IB13	62	0.31	0.28	0.67	2.86	1.19	8.79	0.91	0.71	2.68	31.95	29.38	55.23
IB14	61	0.27	0.28	0.79	0.96	0.89	5.27	0.94	1.17	2.34	36.53	40.69	66.37
IB15	62	0.27	0.29	0.57	1.77	1.80	4.18	0.91	1.08	2.11	35.32	48.74	61.56
IB16	62	0.33	0.36	0.75	1.38	1.56	3.21	0.88	1.14	1.93	36.77	52.00	60.48
All Beds	928	0.30	0.26	0.79	1.36	0.79	8.79	1.20	0.71	7.62	35.99	29.38	68.06

Note: Mean grades are thickness weighted
Min = minimum value
Max = Maximum value

11.4 SAMPLE SECURITY

All drill core from the Phase 1 and Phase 2 drilling programs was transported back to the secure core logging and storage facility on a daily basis. The core logging and storage facility is located inside a locked compound with an armed security guard at the JPQ port facility.

All core logging and core sampling was performed at the secure Focus core logging facility. The core facility was a purpose built structure adjacent to the Focus field office and consisted of plywood walls with plastic windows and a sheet metal roof. Core logging benches were positioned along the walls of the logging area, with core splitting benches located in a separate area. A work bench for storing sampling supplies, field standards and blanks, laptops and other core logging and sampling materials was located down the center of the core logging facility

The Sample selection and packaging was performed by Focus geologists and core technicians under the supervision of the Focus project manager.

The drill core was placed in core boxes immediately upon removal from the core barrel. Once a core box was complete a box top was prepared with the drill hole number, box number and depth from and to intervals and the top was then placed on the core box.

All core was transported to the secure core storage facility on a daily basis; core was never left unattended at the drill site. The drill core was stored in covered core boxes and racked on metal core racks while awaiting logging. The core racks are located outside but have a sheet metal roof covering them to protect them from direct sunlight and the elements.

11.5 QUALITY ASSURANCE AND QUALITY CONTROL METHODOLOGY AND PROCEDURES

11.5.1 Focus Field Quality Assurance and Quality Control

Focus implemented a comprehensive analytical QA/QC program during the Phase 1 and Phase 2 drilling programs that included the insertion of blind CRM standards, duplicates and blanks to evaluate analytical precision, accuracy and potential contamination during the sample preparation and analytical process. The field QA/QC samples were inserted by Focus geologists during the core logging process; for details on the field QA/QC sample insertion process please refer to Section 11.2.3 of this technical report.

In addition to the field inserted QA/QC samples, Focus selected 64 pulps after sample preparation at Certimin and submitted them to SGS for check assay purposes during Phase 1. No check assay information was provided for Phase 2.

The QA/QC sample summary and insertion rates for the Phase 1 and Phase 2 exploration programs are presented in Table 11-4.

Table 11-4: Focus Quality Assurance and Quality Control Samples

Field QA/QC Sample Type	Pulp/Coarse	Laboratory	QA/QC Sample Count			QA/QC Insertion Rate (total sample count = 5,771)
			Phase 1	Phase 2	Total	
Control Reference Material	GPO-14 - Pulp	Certimin	37	45	82	1%
	GPO-16 - Pulp	Certimin	40	38	78	1%
	GPO-17 - Pulp	Certimin	42	30	72	1%
	GPO-18 - Pulp	Certimin	11	51	62	1%
CRM Sub-Total			130	164	294	5%
Duplicates	Coarse	Certimin	127	164	291	5%
	Pulps	SGS	64	N/A	64	1%
Duplicates Sub-Total			191	164	355	6%
Blanks	Coarse	Certimin	124	164	288	5%
	Pulps	Certimin	128	164	292	5%
Blanks Sub-Total			252	328	580	10%
Total Field QA/QC Samples			573	656	1229	21%

Certified Reference Material Standards.

Focus used four commercially prepared phosphate CRM standards to monitor laboratory analytical accuracy. The CRM standards were purchased from Geostats Pty Ltd. of Australia. The CRM's were matrix matched from a sedimentary phosphate in Tunisia. The following standards were used by Focus:

- GPO 14 - 24.52 wt.% P_2O_5 (certified standard deviation as per Geostats Pty Ltd. Certificate is 0.288)
- GPO 16 - 17.76 wt.% P_2O_5 (certified standard deviation as per Geostats Pty Ltd. Certificate is 0.147)
- GPO 17 – 13.55 wt.% P_2O_5 (certified standard deviation as per Geostats Pty Ltd. Certificate is 0.117)
- GPO 18 – 15.09 wt.% P_2O_5 (certified standard deviation as per Geostats Pty Ltd. Certificate is 0.117)

A total of 294 blind CRM standards were submitted to Certimin for analysis for the two phases. Focus and Golder prepared and evaluated QA/QC control charts for each of the CRM standards (Figure 11-14). The CRM control charts show that all CRM standard results fell within the upper and lower warning limits for P_2O_5 grade. All four CRMs show an upward drift in the last two lots of samples submitted on May 17, 2015 and May 25, 2015. While the results are within the acceptable tolerance, the accuracy of the analysis should be monitored to avoid any positive bias.

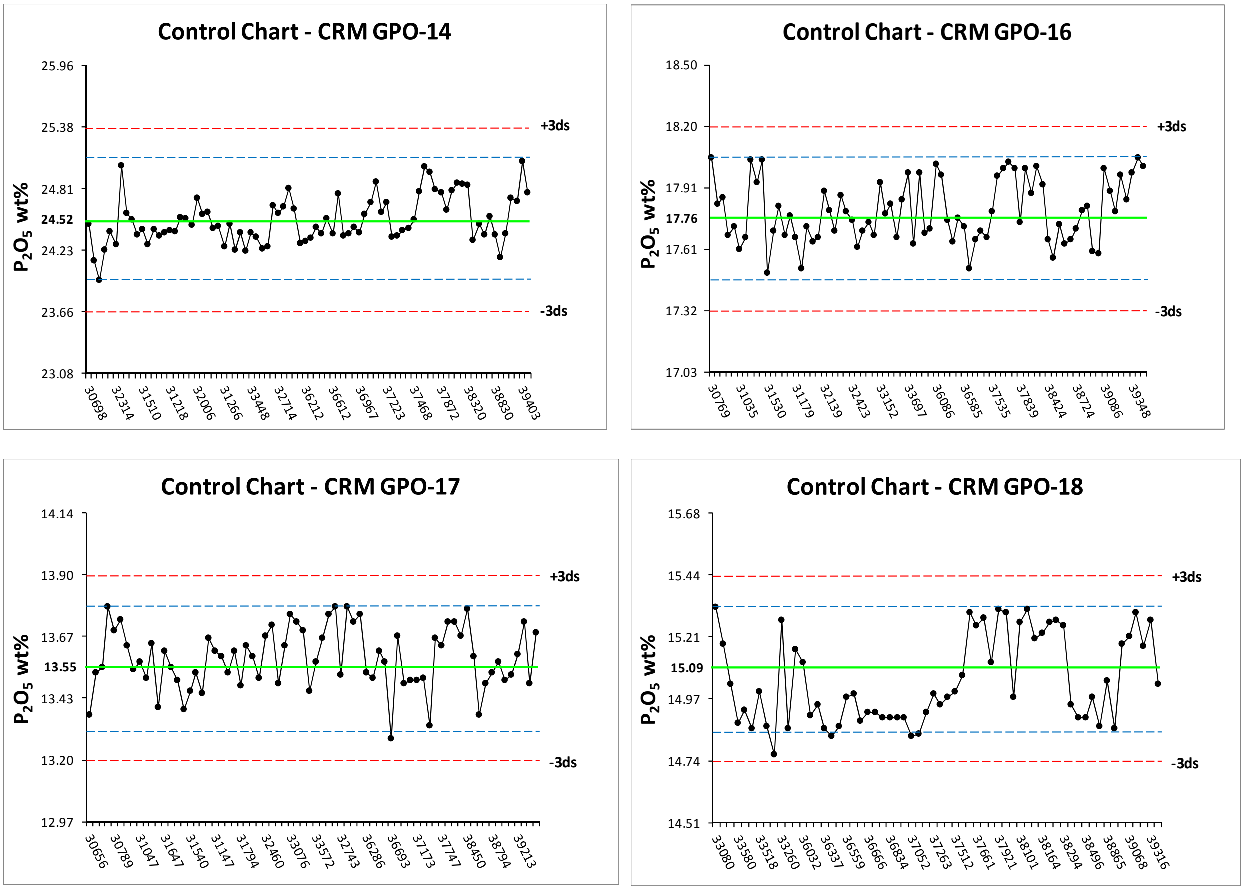


Figure 11-14: Control Charts – P_2O_5 Certified Reference Material Standards

Coarse and Pulp Duplicates

Focus submitted blind coarse duplicates comprising ¼ core samples (half of the core split retained for reference purposes) to the primary laboratory and pulp duplicates for check assays at a secondary laboratory to evaluate for analytical precision.

A total of 294 blind coarse duplicates were submitted to Certimin for analysis during Phase 1 and 2. Focus and Golder prepared and evaluated QA/QC control charts comparing the original and duplicate analyses performed at Certimin (Figure 11-15, Control Charts – P2O5 Coarse and Pulp Duplicates). The P_2O_5 control charts identified a two occurrences where a duplicate analysis was significantly different to the primary analysis (6.34 wt.% versus 13.07 wt.% and 1 wt% versus 2.02 wt%).

A total of 64 pulp duplicates were submitted to SGS for check-assay analysis in Phase 1. No information on check-assay analysis from Phase 2were provided. Golder prepared and evaluated QA/QC control charts comparing the original Certimin and duplicate SGS analyses (Figure 11-15, Control Charts – P2O5 Coarse and Pulp Duplicates) for Phase 1. The P_2O_5 control charts identified a small cluster of pulp duplicates that fall outside of the control limits; this cluster occurs at the low grade end of the results and is a result of the difference between the P_2O_5 detection limits for Certimin and SGS. All remaining duplicate analyses returned results matching the original analysis.

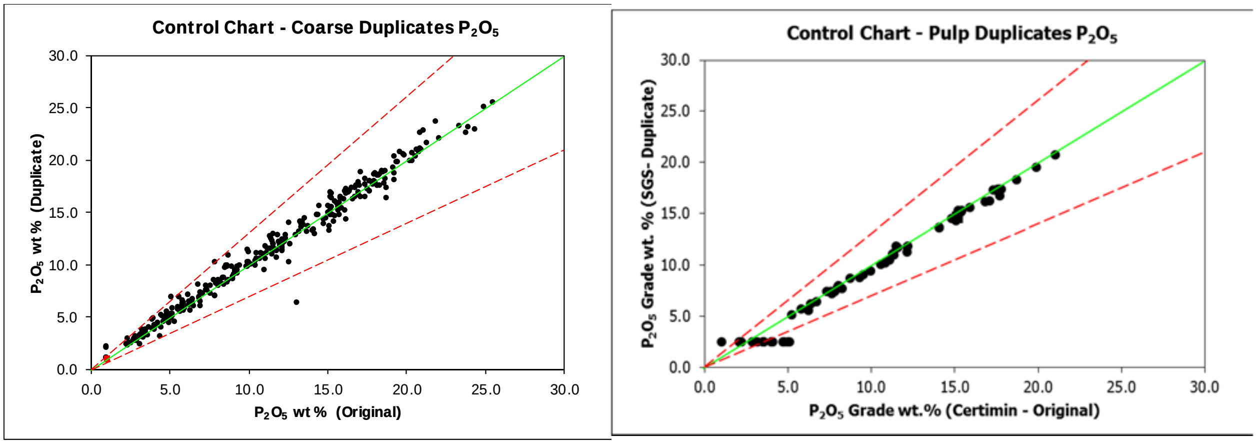


Figure 11-15: Control Charts – P_2O_5 Coarse and Pulp Duplicates

Coarse and Pulp Blanks

Focus used two commercially prepared pulp blank standards and one internally prepared (by Focus) coarse blank standard to monitor potential laboratory sample preparation and analytical contamination. The pulp blank standards were purchased from Canadian Resource Laboratories Ltd. of Canada. The pulp blank standards (P5B, CDN-BL-4 and CDN-BL-10) were sourced from igneous rocks and were not ideally suited to phosphate and will be replaced with a more appropriate matrix matched commercial pulp blank in the future. The coarse blank standard was prepared using locally sourced diatomite from the barren Upper Diatomite Series (above the Diana ore zone). Five samples were processed and analysed for P_2O_5 (gravimetric analysis) at Certimin and five samples were analysed at SGS (both laboratories located in Lima, Peru); all of the coarse blank characterization samples returned P_2O_5 values of less than 5 wt.%.

A total of 252 blind blank standards were submitted to Certimin for analysis. Focus and Golder prepared and evaluated QA/QC control charts for the pulp and coarse blank standards (Figure 11-16). The majority of the coarse blank standards plotted within the control limit of 5.0 wt. % P_2O_5. There were three episodes where the coarse blank standards exceeded the warning limit of 3.0 wt% P_2O_5 from the samples submitted on April 28, May 17 and May 25 2015. This could have been resulted from the coarse blank sample containing lenses with higher phosphate content, as pulp Material used for making the coarse blank standard should be reviewed. All of the pulp blanks plotted at or below 1.0 wt.% P_2O_5 and well within the control limit of 3.0 wt.% P_2O_5 for the coarse blanks.

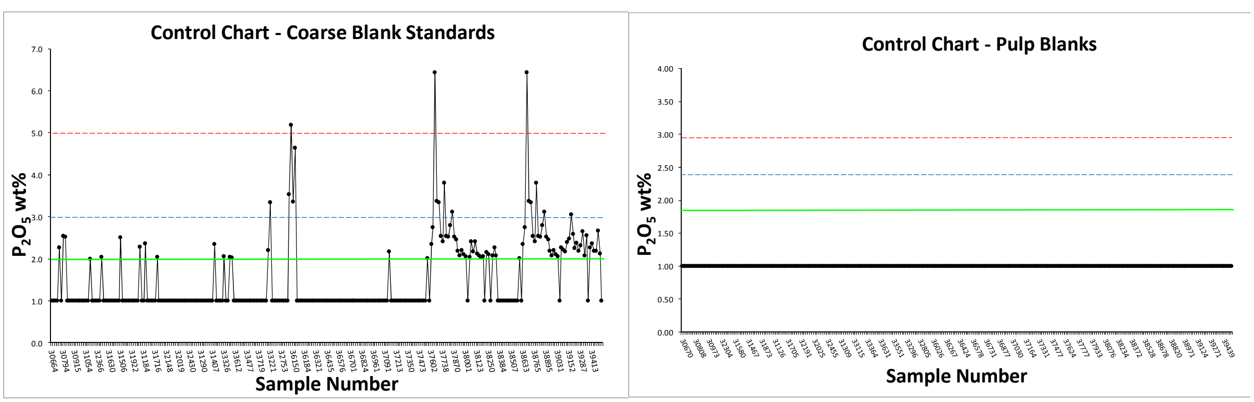


Figure 11-16: Control Charts – Coarse and Pulp Blank Standards

11.5.2 Certimin Internal Laboratory Analytical Quality Assurance and Quality Control

In addition to the field based QA/QC program implemented by Focus, Certimin also performed their own internal analytical QA/QC program that included the insertion of blind CRM standards, duplicates and blanks to evaluate analytical precision, accuracy and potential contamination during the sample preparation and analytical process. The laboratory standards were a combination of blind standards inserted by laboratory sample preparation and QA/QC personnel prior to analysis, as well as routine testing of reference standards during the analytical process (i.e. during ICP-OES analysis).

11.5.3 Qualified Person Comment on Analytical Quality Assurance and Quality Control Program

It is Golder's opinion that the Focus QA/QC protocol and the laboratory internal QA/QC protocol applied during the Phase 1 and Phase 2 exploration program were appropriate, followed and well documented during the analytical process. It is Golder's opinion that analytical samples showing no significant bias and that the quality of Certimin analyses results can be considered reliable for use in estimating Mineral Resources.

11.6 PRIMARY LABORATORY AUDIT

As part of the 2014 Qualified Person site visit the Golder Qualified Person performed a laboratory audit visit to the During the laboratory audit visit the Golder Qualified Person reviewed the sample chain of custody, sample receiving, sample preparation, analytical process and reporting of results procedures with the senior Certimin laboratory personnel, including the Certimin manager for internal QA/QC; the Focus database and QA/QC manager was also present on the site visit as well as a senior technical manager from the Golder Lima office.

The Certimin sample receiving, sample preparation, analytical and sample storage areas were visited and standard laboratory procedures were reviewed with the technical personnel responsible for each area or analysis. Golder also reviewed the internal laboratory QA/QC documentation present at each stage of the process; this documentation included daily records of temperature and humidity in sample receiving and sample preparation areas, scale and analytical instrument daily standard calibration records, annual inspection/certification seals and/or certificates for scales and analytical instruments and the results of monthly and annual round robin testing results for the analytical instruments.

11.7 QUALIFIED PERSON STATEMENT ON SAMPLING, ANALYSIS AND QUALITY CONTROL

It is Golder's opinion that appropriate chain of custody and internationally recognized sample selection, sample preparation, analysis and QA/QC procedures were followed during the sample preparation and analytical process for the Phase 1 and Phase 2 exploration programs. It is Golder's opinion that the samples collected during the Phase 1 and Phase 2 exploration programs were of high quality and were representative of the phosphorite mineralization within the Focus Bayovar 12 Concession with no significant sample bias.

12 DATA VERIFICATION

12.1 DATA VERIFICATION PROCEDURES

12.1.1 Focus Data Verification

As Golder personnel were not involved directly during the implementation of the Phase 1 and Phase 2 exploration programs the primary quality control and data verification measures taken were in the form of a desktop review of the data and observations provided by Focus. As the procedures and methodology used in the Phase 1 and Phase 2 exploration programs were developed collaboratively by Focus and Golder personnel, Golder is satisfied that the data and observations from the exploration programs can be considered reliable for use in geological modelling and resource estimation. The key areas of the exploration program data and observation verification carried out by Golder are presented in the following sections.

Drill Hole Collar Location Verification

The Golder Qualified Person visited 12 of the 62 drill hole locations on the Bayovar 12 Concession property in order to verify and document the reported drill hole locations. The drill holes visited for collar location verification were selected at random by the Golder Qualified Person while in the field to ensure there was no bias in drill hole selection by the Focus personnel. The Golder Qualified Person did not conduct a site visit during the Phase 2 drilling program; however, Golder personnel were on site performing geotechnical logging and collection hydrogeological data and confirmed the presence of a number of the Phase 2 drill holes.

Drill hole collar monuments, indicating the drill hole name, completion date and depth, were photographed (Figure 12-1 and Figure 12-2) and drill hole collar coordinates for each of the 12 drill holes visited were recorded using a handheld non-differential GPS. The handheld GPS coordinates were compared to the surveyed collar coordinates and differences in easting and northing were calculated. The results of the collar coordinate comparison are presented in Table 12-1. The differences between the drill hole verification coordinates and the surveyed collar coordinates are within the error limits of the handheld GPS.

Logging and Sampling Procedure Verification

Golder did not actively participate in the implementation of the Phase 1 exploration drilling and sampling program; however, Golder did work collaboratively with Focus to develop the exploration drilling, logging, sampling and analytical program procedures and methodology that was implemented. The Golder Qualified Person was able to observe the implementation of the core splitting, logging and sampling procedures during the Qualified Person current personal inspection site visit (Figure 12-3 and Figure 12-4). Golder provided senior geological support during the implementation of the Phase 2 exploration drilling program; additionally, Golder personnel were directly involved in geotechnical logging and collection of hydrogeological data for two of the Phase 2 drill holes.

Figure 12-1: Example Drill Hole Monument for JPQ-14-05

Figure 12-2: Example Drill Hole Monument for JPQ-14-19

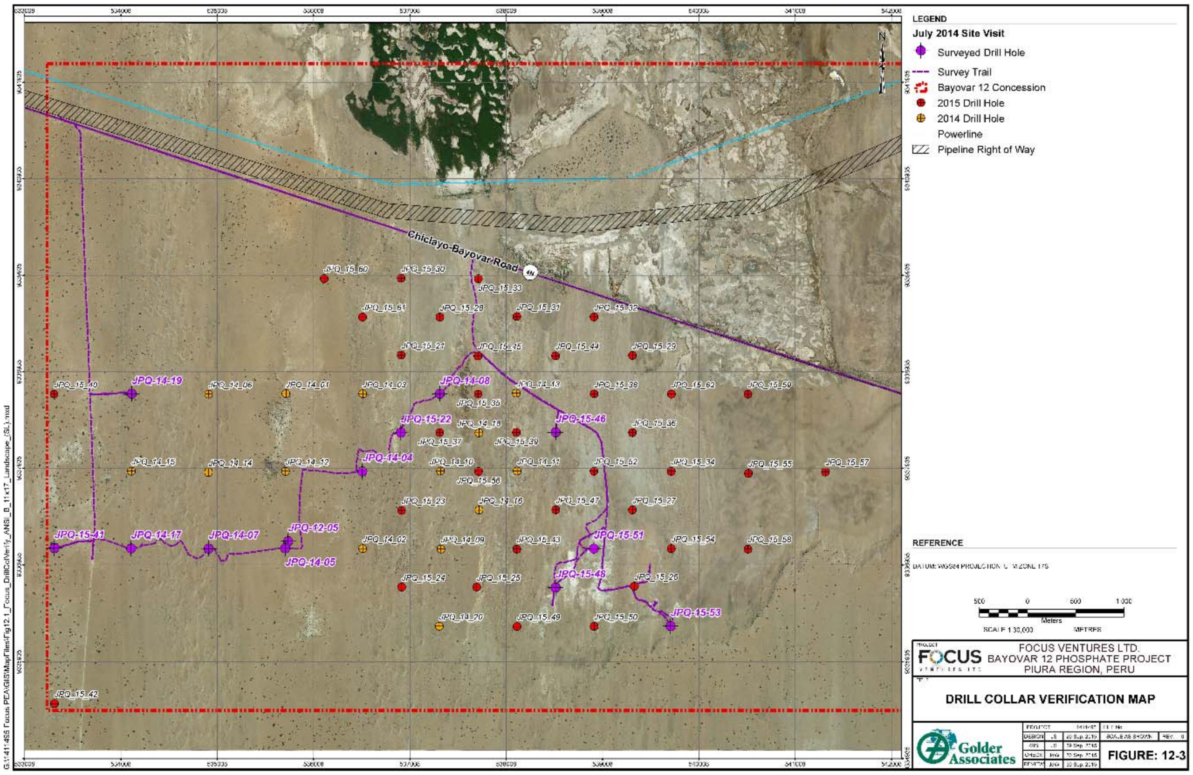


Figure 12-3: Drill Collar Verification Map

Table 12-1: Summary of Drill Hole Collar Coordinate Comparison

Drill Hole	Golder Site Visit GPS Coordinates (m)		Focus Surveyed Coordinates (m)		Difference (m)	
	Easting	Northing	Easting	Northing	Easting	Northing
JPQ-14-04	536,515.0	9,337,901.0	536,518.0	9,337,903.1	3.0	2.1
JPQ-14-05	535,714.0	9,337,106.0	535,717.4	9,337,103.5	3.4	-2.5
JPQ-14-07	534,919.0	9,337,103.0	534,919.6	9,337,101.0	0.6	-2.0
JPQ-14-08	537,319.0	9,338,703.0	537,318.1	9,338,702.3	-0.9	-0.7
JPQ-14-17	534,115.0	9,337,105.0	534,118.4	9,337,103.4	3.4	-1.6
JPQ-14-19	534,121.0	9,338,704.0	534,120.3	9,338,704.3	-0.7	0.3
JPQ-15-22	536,915.0	9,338,301.0	536,920.9	9,338,302.9	5.9	1.9
JPQ-15-41	533,318.0	9,337,108.0	533,319.1	9,337,105.9	1.1	-2.1
JPQ-15-46	538,522.0	9,338,304.0	538,520.3	9,338,306.2	-1.7	2.2
JPQ-15-48	538,521.0	9,336,702.0	538,520.0	9,336,702.7	-1.0	0.7
JPQ-15-51	538,918.0	9,337,104.0	538,919.6	9,337,106.1	1.6	2.1
JPQ-15-53	539,714.0	9,336,303.0	539,718.0	9,336,305.9	4.0	2.8

Figure 12-4: Core Splitting Being Performed During the Qualified Person Site Visit

Figure 12-5: Core Logging Being Performed During the Qualified Person Site Visit

Golder also performed a desk top review of the Phase 1 and Phase 2 descriptive logs, sample interval data, analytical data and core photos provided by Focus to verify that all data and observations were collected in a manner consistent with the prepared exploration drilling, logging, sampling and analytical program procedures and methodology.

Geological Data and Interpretation Verification

Geological data and interpretation verification performed by Golder was in the form of a desktop review of the descriptive logs, sample interval data, analytical data and core photos to ensure the geological database was free from typographic errors or omissions.

Golder prepared graphic logs for each of the 62 drill holes using all available data for each drill hole. Lithology intervals were reviewed and where minor errors or omissions were identified Golder performed these adjustments. Likewise, Golder reviewed phosphorite and diatomite bed correlations between drill holes and where minor errors or omissions were identified Golder performed these adjustments.

Analytical Data Verification

Analytical data verification performed by Golder includes cross referencing the spreadsheet analytical data against pdf copies of the Certimin laboratory certificates to ensure the analytical database was free from typographic errors or omissions.

Golder independently compiled and reviewed the Focus analytical QA/QC analyses results (see Section 11 for a detailed discussion on the analytical QA/QC review), including analytical blank, standard and duplicate analyses for test work performed by Certimin as well as reviewing check-assay analyses performed at SGS Peru, the secondary laboratory.

Basic statistics of the analytical data were reviewed by Golder on a unit by unit basis to evaluate the potential for phosphorite bed miscorrelations as well as to identify any potential outliers or errors. The modelled grade parameters were also reviewed by Golder on a unit by unit basis as a final check for potential miscorrelations, outliers or errors.

Golder did not independently collect samples to submit for analyses.

12.2 LIMITATIONS ON DATA VERIFICATION

As discussed previously, JPQ performed limited reconnaissance exploration work on the Bayovar 12 Concession in 2012; however, Golder and Focus could not verify the methodology and results from the 2012 JPQ work to a level where they could be relied upon for use in the geological modelling process and resultant resource estimates. As a result the 2012 JPQ work was not used for modelling and resource estimation as reported in this technical report.

Given the fact that the Focus Phase 1 and Phase 2 drilling programs were designed such that the Focus drill hole spacing pattern was complete on its own and did not rely on any previous work for points of observation, Golder does not see the exclusion of the 2012 JPQ work as an issue or limitation that impacts the reliability or representativeness of the current geological model and the resultant Mineral Resource estimate.

12.3 QUALIFIED PERSON STATEMENT ON DATA VERIFICATION

It is Golder's opinion that the exploration data and observations from the 62 drill holes completed during the Phase 1 and Phase 2 exploration drilling programs have been appropriately verified for the purpose of completing a geological model, estimating Mineral Resources and preparing an NI 43-101 compliant Mineral Resource estimate technical report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

No metallurgical testing has been performed prior to 2014 on the phosphorite layers underlying the Bayovar 12 Concession. The phosphorite layers are extensive and similar to those being mined by Vale (Bayovar Concession 2) and to those being developed by Fosfatos del Pacifico (Bayovar Concession 9). The unconsolidated phosphorite layers from Concessions 2 and 9 do not require grinding. The bench-scale test program for Bayovar 12 phosphorite layers examined the proven unit operations for recovering concentrate from the ore at Concessions 2 and 9.

The bench-scale testing in 2014/15 was performed by Jacobs Engineering (Jacobs) to develop a process for recovering a phosphate rock product from the Bayovar 12 phosphorite layers. The work by Jacobs is summarized in the reports "Beneficiation Testing Focus Ventures Ltd.," "Scrubbing, Settling, and Filtration Testing" and "Focus Bench Scale Flow Sheet Validation". Jacobs has operated a laboratory and pilot plant for evaluating phosphate ore samples since 1977. The Metallurgical laboratory does not have ISO certification, but the chemist is a member in good standing of the Association of Fertilizer and Phosphate Chemists (AFPC) and participates in the AFPC quality control program. The analytical laboratory normally uses check sample 22 and AFPC approved procedures to chemically analyze samples.

13.1 METALLURGICAL SAMPLES

The metallurgical testing was performed on two sets of samples.

Focus Ventures geology team in Peru took the first set of samples from air-dried drill cores. Jacobs' preparation and characterization of these samples are described in Sections 13.1.1 and 13.1.2. Testing these samples was directed at recovering a phosphate rock concentrate containing +29% P_2O_5 that was suitable for conversion to phosphatic fertilizer by acidulation and granulation.

The second sample set comprised two composite samples – one for phosphorite layers 2 through 6 and the other for phosphorite layers 11 to 13, both of which had been prepared by ALS, an analytical laboratory in Vancouver, British Columbia. ALS had previously dried and crushed the cores to obtain a representative sample for chemical analysis. The two representative composites were prepared from PQ diameter drill core drilled specifically for metallurgical test work. Testing these samples was directed at recovering a concentrate containing +24% P_2O_5 that was suitable for use as direct application phosphate rock (DAPR). Jacobs' chemical analysis of these two composites is discussed in Section 13.4.

13.1.1 Preparation of 13 Individual Layer Composites

For purposes of metallurgical testing, 13 composite samples were prepared, each representing a major phosphorite layer drilled in the Bayovar 12 concession. The Focus geological team selected the drill holes used for making the composite samples. Table 13-1 identifies the composite samples and the drill holes from which the cores for the 13 phosphorite layers were obtained. The samples are arranged in the table below from the top layer (PH01) to the bottom (PH13).

Table 13-1: Composite Samples and Source Drill Holes

Composite	JPQ-14 Drill Hole Cores Used								
Sample	06PQ	07PQ	08PQ	10PQ	11PQ	16PQ	17PQ	19PQ	43PQ
PH01		yes		yes	yes	yes			yes
PH02		yes	yes						yes
PH03		yes	yes						yes
PH04	yes	yes		yes	yes	yes	yes	yes	
PH05		yes	yes						yes
PH06		yes	yes						yes
PH07		yes	yes						yes
PH08		yes	yes						yes
PH09		yes	yes						yes
PH10		yes	yes						yes
PH11		yes	yes						yes
PH12		yes	yes						yes
PH13		yes	yes						yes

The above samples and the sample inventory sheets were received in good condition. The core samples for all composites except PH01 and PH04 were dried at nominally 100 ºC, weighed, stage crushed in a roll crusher to pass 3 mm, then blended and riffle split to obtain a head sample and 1000 gram aliquots for subsequent testing. PH01 and PH04 were weighed as received, stage crushed in a roll crusher to pass 3 mm, then blended and riffle split to obtain a head sample and 1000 gram aliquots for subsequent testing. PH01 and PH04 were not dried.

13.1.2 Sample Characterization

A portion of the head sample for each composite was pulverized and chemically analyzed using AFPC approved procedures.

The water-soluble salt content of each pulverized sample was measured by weight loss after water washing and rinsing on a filter with a glass frit. The mineral density of each composite was determined by water displacement using a pycnometer. The bulk density of dry crushed material for each composite was determined with a graduate cylinder, by recording the sample weight and then the volume before (loose bulk density) and after consolidation (tamped bulk density).

The data obtained for each composite are presented in Table 13-2. Composite PH01 contained more Fe2O3 and Al2O3 than the other composites and also had the highest mineral density (specific gravity) and bulk density.

QEMSCAN analysis performed at the Colorado School of Mines determined that the major minerals in the washed material (+53 µm) were predominantly apatite, with lesser quantities of gangue minerals such as quartz, feldspars, and gypsum. The washed material was examined as three size fractions, +600 µm, 600/150 µm, and 150/53 µm. The QEMSCAN report is presented in Jacobs, 2015, “Beneficiation Testing Focus Ventures Ltd.” For the +600 µm fraction, the percentages of liberated apatite particles were low for PH01 (8%), PH02 (60%), and PH04 (40%), and ranged from 68% to 93% for the other composites.

For the 600/150 µm fraction, the percentages of liberated apatite particles were below 40%, except for PH01, PH04, PH10, and PH11.

For the 150/53 µm fraction, the percentage of liberated apatite particles was greater than 90%, except for PH01 and PH04, in which only 13% and 42% of the phosphate particles were liberated. This size fraction contained more quartz and feldspar than the coarser fractions.

Concentrates obtained from the other Bayovar phosphorite deposits are typically low grade for export phosphate rock. For Bayovar 12 many of the apatite particles in the +600 µm and 600/150 µm fractions are not liberated, partly explaining why the average P_2O_5 content of the washed concentrate seldom exceeds 30%. Also, the quartz, feldspars, and gypsum in the 150/53 µm fraction are diluents that lower the P_2O_5 content of the washed concentrates.

The -53 µm fraction of the phosphorite from Bayovar 12 is comprised mostly of diatomite, with some carbonate minerals and clays.

13.1.3 Sample Representation

The Focus geological team selected the drill holes so that the 13 composite samples would have P_2O_5 contents similar to those of the 13 phosphorite layers (NI 43-101 Mineral Resource Technical Report on the Bayovar 12 Phosphate Project). The P_2O_5 contents are compared onTable 13-3. As shown on the table, the P_2O_5 contents of the composite samples are similar to the weighted average P_2O_5 contents of the corresponding phosphorite beds. The mean difference of the 13 data pairs was 0.73% P_2O_5.

The P_2O_5 contents of the composite samples are considered not representative of the P_2O_5 contents of the indicated resource, if the mean difference of 0.73% P_2O_5 is significantly greater than 0.00% P_2O_5. The 90% confidence limits of the mean difference can be determined from the following relationship.

mean ± t x std. deviation/(count)$^{0.5}$, where

mean = 0.73 from Table 13.1.3

t = 1.796 from student t value for α = 5% at 12 degrees freedom

std. deviation = 1.76 from Table 13.1.3

count = 13 from Table 13.1.3, (13)$^{0.5}$ = 3.606

lower 95% limit = 0.73 – 1.796 x 1.76/3.606 = 0.73 – 0.88 = <u>-0.15</u>

upper 95% limit = 0.73 + 1.796 x 1.76/3.606 = 0.73 + 0.88 = <u>1.61</u>

The 90% confidence limits (-0.15 ≤ 0.73 ≤ 1.61) include 0 and therefore the P_2O_5 contents of the composite samples used for metallurgical tests did not differ significantly from the P_2O_5 contents of the indicated resources at the 5% level.

Table 13-2: Chemical Analysis and Densities of Composite Samples

Phosphorite	Chemical Analyses – AFPC Approved Procedures							Dry Density (t/m³)		
Bed	% P_2O_5	% Insol	% Fe_2O_3	% MgO	% Al_2O_3	% CaO	% Salt[a]	Mineral	Bulk[b]	Bulk[c]
PH01	13.78	32.87	2.40	1.76	3.24	22.43	6.51	2.75	0.96	1.02
PH02	10.59	42.90	1.70	1.83	1.99	18.14	7.71	2.62	0.69	0.73
PH03	19.48	24.25	1.00	1.14	1.58	30.85	5.23	2.43	0.71	0.76
PH04	18.29	25.90	1.09	1.48	1.67	29.85	5.35	2.71	0.85	0.94
PH05	10.00	42.83	1.54	1.95	1.74	17.36	6.93	2.52	0.54	0.58
PH06	13.71	35.82	1.59	1.65	1.94	22.67	4.26	2.59	0.59	0.64
PH07	7.94	48.14	1.83	2.09	1.94	14.52	5.84	2.37	0.53	0.58
PH08	5.53	56.97	1.62	1.66	1.53	10.27	5.43	2.34	0.41	0.44
PH09	10.35	43.86	1.95	1.50	2.42	17.32	4.48	2.52	0.59	0.64
PH10	8.35	46.28	1.76	2.09	1.93	15.82	4.68	2.47	0.52	0.58
PH11	13.91	38.01	1.81	1.19	2.34	22.39	4.12	2.57	0.63	0.68
PH12	12.51	40.11	1.95	1.64	2.34	21.04	2.98	2.57	0.64	0.69
PH13	11.07	38.66	1.91	2.53	2.31	20.63	2.93	2.55	0.64	0.75

a – water soluble salt
b – loose bulk density
c – tamped bulked density

Table 13-3: Composite Sample % P_2O_5 vs. Indicated Resource % P_2O_5

Phosphorite	% P_2O_5		
Bed	A	B	=A-B
PH01	13.80	13.78	0.02
PH02	11.61	10.59	1.02
PH03	19.56	19.48	0.08
PH04	15.67	18.29	(2.62)
PH05	8.56	10.00	(1.44)
PH06	13.50	13.71	(0.21)
PH07	9.93	7.94	1.99
PH08	9.93	5.53	4.40
PH09	11.20	10.35	0.85
PH10	10.05	8.35	1.70
PH11	13.87	13.91	(0.04)
PH12	14.21	12.51	1.70
PH13	13.13	11.07	2.06
Average	12.69	11.96	0.73
Std. Deviation	2.95	3.94	1.76
Count			13

A – weighted average % P_2O_5 from Indicated Resource
B – composite sample % P_2O_5

13.1.4 ALS Combined Layer Composite Samples

The samples, labeled as 1 of 2 and 2 of 2, were received in good condition. Sample 1 of 1 comprised PH02, PH03, PH04, PH05, and PH06 in proportion with the indicated resource tonnage. At the time the sample was prepared, it was envisioned that PH 01 would not be processed because it made lower grade concentrate than the other phosphorite layers. Sample 2 of 2 comprised PH11, PH12, and PH13 in proportion with the indicated resource tonnage. Phosphorite layers PH07, PH08, PH09, and PH10 were not considered in this particular round of test work.

The chemical analyses of the two samples are reported in Table 13-4. These samples were tested to re-examine drum scrubbing and the establish unit areas for tailings thickening and concentrate filtration.

Table 13-4: Sample Analyses

Sample	% P_2O_5	% Insol	% Fe_2O_3	% Al_2O_3	% MgO	% CaO	Cd ppm	% WSS
1	14.01	38.50	1.49	1.90	1.52	22.87	62	8.14
2	14.57	35.85	1.93	2.65	1.64	23.97	64	5.30

13.2 BENCH-SCALE TESTS OF INDIVIDUAL LAYER SAMPLES

13.2.1 Drum Scrubbing and Desliming

Drum scrubbing is a proven unit operation for disaggregating unconsolidated phosphorite. The retention time and slurry % solids for drum scrubbing were investigated using the larger composite samples (PH02, PH06, and PH13). Dry ore was placed in a 20 cm diameter drum and diluted to 37% solids, 45% solids, or 50% solids and rotated at 49% of critical speed for 2 minutes, 2.5 minutes, and 3 minutes respectively. The drum discharge was wet screened at 53 µm. The laboratory data indicated that the yield and % P_2O_5 of +53 µm material were similar for each condition; however, the wet screening technique may have confounded the data because soft lumps were pushed through the screen by hand. Based on visual observation, the fewest soft lumps remained after scrubbing at 37% solids. Consequently, the scrubbing parameters selected for subsequent tests were 37% solids slurry and 3 minutes retention time for all 13 composite samples.

PH01 and PH04 composites were not dried prior to drum scrubbing at 37% solids. For the other composites, dry sample was added to the scrubber and diluted to 37% solids. The general perception was that the moist samples were more completely disaggregated by drum scrubbing than the dried samples. When the confirmation tests were performed on each composite sample, the samples were moistened to about 30% moisture prior to scrubbing at 37% solids for 2 minutes. A comparison of the yields and % P_2O_5 of the +53 µm material is provided in Table 13-5. The yields expressed on Table 13-5 are based on dry material without salt.

Table 13-5: Dry Scrubbing vs. Moist Scrubbing

Phosphorite	% Weight of +53 um			% P_2O_5 of +53 um		
Bed	B1	B2	=B1- B2	B1	B2	=B1-B2
PH01	65.1	59.3	5.8	22.7	23.7	(1.0)
PH04	68.0	66.8	1.2	28.3	28.5	(0.2)
Average	66.6	63.1	3.5	25.5	26.1	(0.6)
	A	B	=A-B	A	B	=A-B
PH02	54.5	57.3	(2.8)	18.7	19.1	(0.4)
PH03	69.9	64.7	5.2	25.4	28.7	(3.3)
PH05	41.6	40.2	1.4	22.7	23.3	(0.6)
PH06	56.3	45.7	10.6	21.4	26.8	(5.4)
PH07	48.6	29.9	18.7	16.5	24.8	(8.3)
PH08	49.1	19.5	29.6	9.6	24.1	(14.5)
PH09	53.1	38.7	14.4	19.3	25.3	(6.0)
PH10	52.4	30.3	22.1	13.2	23.5	(10.3)
PH11	53.3	49.4	3.9	22.4	25.0	(2.6)
PH12	59.0	49.8	9.2	20.0	23.5	(3.5)
PH13	61.2	43.3	17.9	18.9	25.0	(6.1)
Average	54.5	42.6	11.8	18.9	24.5	(5.5)

Note: all tests performed with 37% solids slurry and 2 minutes retention time
B1 – damp sample fed to scrubber in first test
B2 – damp sample fed to scrubber in confirmation test
A – dry sample fed to scrubber in first test
B – damp sample fed to scrubber in confirmation test

The first two rows of Table 13-5 deal with PH01 and PH04, which were tested in a moist condition both times. The yield was lower by 3.5% weight and the grade was higher by 0.6 % P_2O_5 for the confirmation tests.

For the remaining 11 comparisons on Table 13-5, the "A" set of tests was performed by adding dry sample to the scrubber, and the "B" set of tests were performed by adding moist samples to the scrubber. On average, the yield from moist samples was 11.8 % weight lower than the yield from dry samples. The grade of +53 um material obtained from moistened samples averaged 24.5% P_2O_5 compared to 18.9% from dry samples. The data demonstrate that drying the samples before testing changes the test result and that moistening samples before they are tested can reverse the effect of drying.

The average P_2O_5 content of the +53 µm material recovered from moist samples after the drum scrubbing and desliming step was 24.5 % P_2O_5, which is a suitable grade for direct application phosphate rock (DAPR).

The scrubbing procedure using moist sample instead of dry sample is an important consideration for future tests. Preferably future metallurgical samples should not be dried prior to scrubbing. Drying appears to make the samples more refractory to scrubbing by hardening the diatomite aggregates. If the future samples have been dried, a 5-minute rehydration period prior to scrubbing should be allowed.

13.2.2 Attrition Scrubbing and Desliming

The purpose of attrition scrubbing is to disaggregate the remaining diatomite so that it can be removed by desliming. In a commercial plant the attrition feed % solids will be controlled by the primary cyclone underflow. The slurry viscosity is increased by the diatomite and the cyclone underflow is expected to be about 55% solids. Consequently the laboratory attrition tests were performed at 55% solids.

One stage and two-stage attrition were tested. The lab data indicated that the % P_2O_5 of the deslimed attrition cell discharge was not significantly different for one or two stage attrition; however, two stage attrition reduced the recovery of P_2O_5 by as much as 6%.

Attrition cell retention times of 8 minutes, 12 minutes, and 16 minutes were tested. The test data did not show significant changes in the yield or P_2O_5 content of the +53 µm material due to retention time. Nevertheless, 12 minutes retention was selected for confirmation tests.

The attrition scrubber discharge was wet screened and the size fractions were analyzed to investigate the desliming cut points of 53 µm, 74 µm, 105 µm, and 150 µm. The P_2O_5 content of the <600 µm coarse product with progressively finer cut points is presented in Table 13-6. Similarly, the weight yield of <600 µm coarse product with progressively finer cut points is presented in Table 13-7. Only the <600 µm fraction is dealt with here because the >600 µm must undergo a second stage of attrition.

From Table 13-6 it is apparent the 600/150 µm fraction has a higher and more consistent P_2O_5 content than the three fractions with finer cut points. The % P_2O_5 declines as the cut point becomes progressively finer, averaging 29.25% for the 600/150 µm fraction and declining to an average 25.58% for the 600/53 µm fraction.

From Table 13-7 it is apparent that the 600/150 µm fraction has lowest weight recovery (average = 34.86%) and the most variable weight recovery (standard deviation = ±16.77%) of the four size fractions. There is considerable variation in weight recovery from layer to layer, which means that performance predictions depending on particle size are specific to each layer.

Table 13-6: % P_2O_5 of <600 µm Material vs. Cut Point

	% P_2O_5			
Composite	600/150 um	600/105 um	600/74 um	600/53 um
PH01	26.39	25.17	24.22	23.98
PH02	29.12	26.05	22.48	21.53
PH03	30.08	28.73	27.45	27.19
PH04	30.15	29.55	28.73	28.55
PH05	29.78	27.34	25.63	25.18
PH06	29.71	28.64	27.91	27.45
PH07	28.91	27.41	26.26	25.55
PH08	29.04	27.34	25.76	25.05
PH09	29.29	27.52	26.22	25.52
PH10	29.32	27.53	26.16	25.42
PH11	30.21	27.97	26.58	25.64
PH12	29.48	27.67	26.24	25.26
PH13	28.74	27.70	26.75	26.24
Average	29.25	27.59	26.18	25.58
STDEV	0.982	1.110	1.570	1.705

Table 13-7: Weight Recovery of <600 μm Material vs. Cut Point

	% Weighted Recovery			
Composite	600/150 um	600/105 um	600/74 um	600/53 um
PH01	72.87	92.28	98.51	100.00
PH02	38.97	67.77	92.45	100.00
PH03	54.54	86.26	97.38	100.00
PH04	57.06	88.72	97.78	100.00
PH05	33.97	71.62	93.69	100.00
PH06	29.32	68.17	92.92	100.00
PH07	26.67	61.45	88.59	100.00
PH08	21.84	53.47	82.91	100.00
PH09	20.54	59.92	89.51	100.00
PH10	17.63	54.48	88.21	100.00
PH11	20.40	64.60	89.79	100.00
PH12	29.21	64.59	90.02	100.00
PH13	30.14	66.79	94.16	100.00
Average	34.86	69.24	91.99	100.00
STDEV	16.772	12.492	4.443	100.00

The yields of slimes (-53 μm) and product (+53 μm) after drum scrubbing, desliming, attrition scrubbing, and desliming of each composite sample are shown on Table 13-8, as are the P_2O_5 contents of the slimes and product. Drum scrubbing used moist samples diluted to 37% solids with 3 minutes retention time. Attrition scrubbing used the +53 μm from the drum discharge diluted to 55% solids with 12 minutes retention time. The deslimed attrition cell discharge, including the >600 μm, contained 26.1% P_2O_5 on average, which is below the normal minimum for phosphate rock sold for use in phosphoric acid plants, but is suitable for use as DAPR.

Table 13-8: Confirmation Test Balances – After Second Attrition

	% Weight (dry salt free basis)			Material % P_2O_5		
Composite	1st -53 um	2nd -53 um	2nd +53 um	1st -53 um	2nd -53 um	2nd +53 um
PH01	40.7	1.7	57.6	3.3	19.4	23.9
PH02	42.6	9.9	47.4	1.0	2.6	22.5
PH03	35.4	1.7	63.0	2.4	14.1	29.0
PH04	33.2	0.1	66.6	4.1	4.1	28.5
PH05	59.8	4.5	35.7	1.7	4.4	25.7
PH06	54.3	2.2	43.5	2.4	8.0	27.7
PH07	70.1	1.9	28.1	2.4	9.0	25.8
PH08	80.6	2.1	17.4	2.0	6.8	26.1
PH09	61.3	1.8	37.0	2.8	12.0	25.9
PH10	69.7	3.5	26.8	2.2	4.5	26.0
PH11	50.6	2.5	46.9	2.7	8.9	25.9
PH12	50.1	5.5	44.4	2.5	5.6	25.8
PH13	56.7	3.0	40.4	2.5	7.2	26.3
Average	54.2	3.1	42.7	2.5	8.2	26.1

The final washed products were sieved at 600 μm and 150 μm to determine which fraction was causing the grade dilution. Except for PH01, the grade dilution was consistently caused by the 150/53 μm fraction, which averaged 22.7% of the composite weight and 23.8% P_2O_5. Combining the +600 μm and the 600/150 μm fractions gave an average yield of 19.9% weight with 29.5% P_2O_5, which meets the normal minimum for commercial phosphate rock used to manufacture phosphoric acid, but the yield is low.

To achieve the normal minimum grade for commercial phosphate rock with an improved yield it is necessary to upgrade the 150/53 µm fraction by flotation. Flotation is not a required component in the current Bayovar 12 flowsheet for the production of DAPR, however results of laboratory test work by Jacobs on the 150/53 µm fraction can be found in Section 3.10 of Jacobs, 2015 "Beneficiation Testing Focus Ventures Ltd.,".

13.2.3 Observation of Apatite Liberation

When apatite particles are classified as liberated that means that less than 5% of the particle area was gangue, classified as a middling means 5% to 35% of the particle area was gangue, and classified as locked means that more than 35% of the particle area was gangue. Practically, the Bayovar 12 apatite particles were either liberated or middling, as only trace amounts of locked particles were observed.

The liberation of phosphate (apatite) in three size fractions is summarized in Table 13-9.

With the exception of PH01 and PH04, the 150/53 µm apatite particles were well liberated. Generally the 600/150 µm apatite particles were not well liberated. The +600 µm apatite particles were normally better liberated than the 600/150 µm apatite particles; however the chemical analyses did not support that observation (see Table 13-7). Very likely the 600/150 µm apatite particles had gangue smeared on their surfaces which gave them the appearance of middling and locked particles.

Table 13-9: % Liberated Apatite Particles

Composite	+600 µm	600/150 µm	150/53 µm
PH01	8%	72%	13%
PH02	60%	22%	91%
PH03	93%	37%	91%
PH04	40%	91%	42%
PH05	92%	25%	92%
PH06	84%	25%	95%
PH07	80%	19%	92%
PH08	90%	35%	95%
PH09	79%	37%	93%
PH10	84%	55%	92%
PH11	88%	61%	92%
PH12	79%	32%	92%
PH13	68%	21%	92%

13.3 BENCH-SCALE TESTS OF COMBINED LAYER COMPOSITE SAMPLES

13.3.1 Additional Drum Scrubbing and Desliming

As stated in Section 13.2, there was concern that the wet screening technique may have confounded the data because soft lumps were pushed through the screen by hand.

Composite samples 1 of 2 and 2 of 2 were drum scrubbed for 3 minutes at 37% solids and then wet screened at 600 µm and 53 µm to obtain +600 µm, 600/53 µm, and -53 µm fractions. The soft lumps remaining after scrubbing were not pushed through the screens by hand. The test results are summarized in Table 13-10 and Table 13-11.

Table 13-10: Sample 1 of 2 – Test Results (A)

(µm)	Analysis		Distribution		
	P_2O_5 %	Insol %	Weight %	P_2O_5 %	Insol %
>600	15.44	35.60	13.4	15.6	12.2
600/53	22.83	22.10	45.9	79.2	26.4
<53	1.67	57.70	40.7	5.2	61.2
Overall	13.22	38.41	100.0	100.0	100.0

Table 13-11: Sample 2 of 2 – Test Results (B)

(µm)	Analysis		Distribution		
	P_2O_5 %	Insol %	Weight %	P_2O_5 %	Insol %
>600	10.74	40.17	26.7	21.7	30.6
600/53	24.54	15.26	38.6	71.9	16.8
<53	2.40	53.22	34.7	6.4	52.6
Overall	13.18	35.06	100.0	100.0	100.0

The above data show that considerable acid insoluble material remained in the 600/53 µm and +600 µm fractions when the soft lumps are not forced through the screen by hand. Consequently the total +53 µm material after drum scrubbing and primary desliming would contain less than 24% P_2O_5. Focus Ventures intends to produce direct application phosphate rock (DAPR) containing a minimum of 24% P_2O_5. Consequently, further processing by attrition scrubbing and desliming would required to complete disaggregation of the diatomite and allow the production of DAPR containing more than 24% P_2O_5.

13.3.2 Vacuum Filter Leaf Tests

Filtration tests were performed on the 600/53 µm product from sample 1 of 2 and sample 2 of 2 to determine the filter cake moisture content and the filtration unit area for dewatering service and also for dewatering plus counter current washing service. Sieve analysis of each filter cake was performed and the results are summarized in Table 13-12.

The original particle size distribution of the filter feed was 100% passing 600 µm and 100% coarser than 53 µm. The presence of 3.5% to 4.6% weight of -53 µm in the filter cakes is attributed to the disaggregation of diatomite lumps during filtration and sieving.

Table 13-12: Sieve and Chemical Analysis of Filter Cakes

	Sample 1 of 2		Sample 2 of 2	
Size (µm)	% P_2O_5	% Insol	% P_2O_5	% Insol
600/212	23.43	10.49	27.32	6.12
212/150	27.24	5.20	27.92	5.62
150/106	24.76	14.20	27.19	10.22
106/53	15.77	41.12	19.76	20.97
-53	3.52	59.12	4.64	51.50
Composite	21.01	22.01	23.49	15.11

If the filter feed had been attritioned and deslimed the diatomite aggregates would have been removed and the P_2O_5 analysis of the feed would have increased. The diatomite remaining in the filter feed also adversely influenced the filtration, as shown in Table 13-13.

Table 13-13: Vacuum Filter Unit Area ($m^2/t/h$) Requirements

	Sample 1		Sample 2	
	D & R	D only	D & R	D only
$m^2/t/h$	12.5	4.0	11.1	3.7

D & R = dewater and 3 countercurrent rinses with seawater
D only = dewater only, no rinses

The unit area of 4.0 m2/t/h for dewatering service (D only) is excessive because the cake dry time exceeded the cake form time due to the remaining diatomite reducing the rate of water removal. The expected unit area for dewatering and rinsing of properly scrubbed and deslimed Bayovar concentrate would be in the range of 0.3 to 0.5 m2/t/h. The expected unit area for dewatering only of properly scrubbed and deslimed Bayovar concentrate would be less than 0.3 to 0.5 m2/t/h.

The moisture content of the above filter cakes ranged from 24% to 24.7%. This high moisture content is attributed to moisture held within the voids of the remaining diatomite aggregates. The expected moisture content of properly scrubbed and deslimed Bayovar concentrate filter cake would be 15 to 18% by weight.

The counter current rinsing did not reduce the cadmium content of the filter cakes, so the dimensions of the filters can be based on tonnage rate and the unit area for dewatering only.

13.3.3 NAC Analysis of Filter Cake

The percentage of available P_2O_5 for each filter cake was determined by neutral ammonium citrate analyses (NAC). The filter cakes from sample 1 of 2 and 2 of 2 contained 6.06% and 8.90% available P_2O_5, respectively. According to the International Fertilizer Association (IFA), phosphate rock with more than 5.4% available P_2O_5, as determined by NAC analysis, has a “High” classification for Direct Application.

13.3.4 Attrition Scrubbing of +600 µm Material

The +600 µm material recovered after drum scrubbing and desliming contained aggregates of diatomite because the soft diatomite lumps had not been pushed through the screen by hand. The +600 µm material from sample 1 of 2 and sample 2 of 2 was attritioned for 6, 9, 12, and 15 minutes and then deslimed at 53 µm. The high diatomite content significantly increased the slurry viscosity and consequently the % solids of the attrition feed had to be reduced from 55% to 40% by weight to. Figure 13-1 presents the P_2O_5 grade vs. recovery after the attrition cell discharge was deslimed at 53 µm. Generally, the recovery decreased and the grade increased as attrition time increased. The lower right most points on Figure 13-1 are for 6 minutes attrition, and the next 3 points up and to the left are for 9, 12, and 15 minutes attrition. For sample 1 of 2 there was no improvement in grade after 9 minutes. For sample 2 of 2 the grade continued to improve as attrition time increased. Twelve 12 minutes of attrition were required to attain a grade of 24% P_2O_5 for sample 2 of 2.

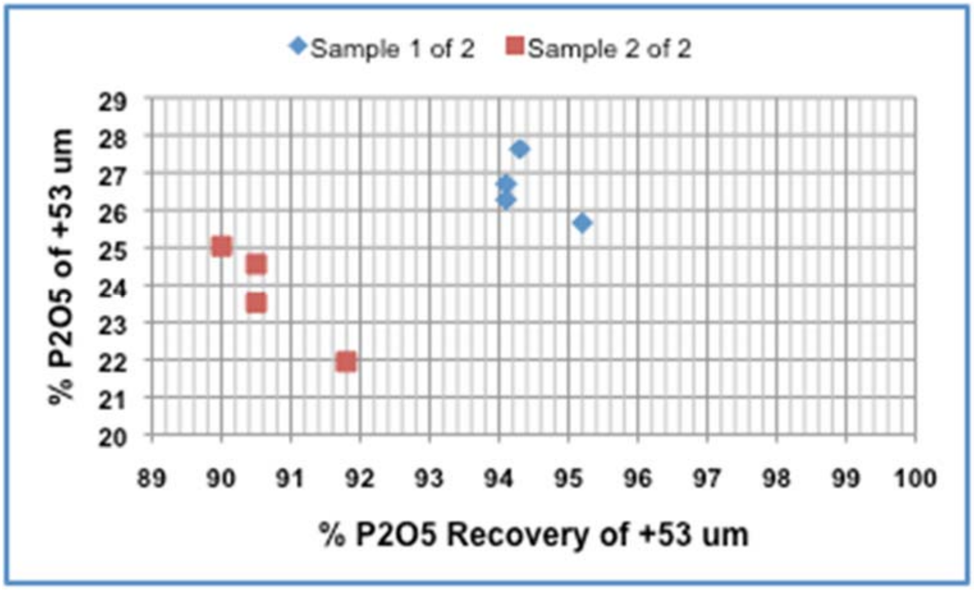


Figure 13-1: Grade vs. Recovery after Attrition of +600 µm Material

13.3.5 Static Cylinder Settling Tests

Several settling tests were performed on -53 µm material from samples 1 of 2 and 2 of 2 to select an appropriate flocculant and to determine the unit area for thickening the material to about 20% solids. Arrmaz flocculant #1 performed well for both samples and was used for the remainder of the settling tests. The test results are summarized in Table 13-14. The unit area requirement is reduced by adding flocculant or by increasing the feed slurry % solids. The unit area requirements for conventional thickeners were excessive.

Table 13-14: Thickener Unit Area (m_2/t/d) Requirements

Sample >	Sample 1 of 2			Sample 2 of 2		
% solids >	2.5%	5%	7.5%	2.5%	5%	7.5%
Floc (g/L)						
0	24.7	27.7	17.2	21.3	16.2	12.5
10	24.2	24.8	13.8	15.5	13.4	9.7
30	24.2	22.9	15.0	15.5	11.5	8.1
90	23.2	19.1	13.2	11.6	11.5	9.4
120	15.5	17.2	12.5	11.6	10.5	11.0

13.4 MATERIAL ISSUES AND DELETERIOUS ELEMENTS

As shown in Table 13-2, the phosphorite layers contain water-soluble salt, ranging from about 3% to 8% weight. The salt must be removed from the plant water system with the tailings to avoid saturation of the process water. Consequently, recycle water must be recovered upstream of the tailings pond. The salt bleed stream (tailings) will dictate the makeup water required by the plant.

The Apatite mineral at Bayovar 12 has a relatively low P_2O_5 content due to:

- CO_3 substitution for PO_4 in the crystal lattice
- The apatite grains are not completely liberated

- Quartz, feldspar, and calcium sulfate are not completely removed by beneficiation

Table 13-15 gives the chemical quality of combined concentrate obtained from each composite by drum scrubbing, desliming, attrition scrubbing, desliming, size classification and flotation of the 150/53 µm fraction.

The upper layer (PH01) produces lower % P_2O_5 concentrate with higher cadmium content and therefore will be monitored closely during exploitation. The above chemical compositions of composites PH02 to PH13 indicate that the phosphate rock can readily be converted to phosphoric acid (% P_2O_5 > 28.5 and COR < 1.65) and high analysis fertilizers (MER < 0.100).

Bayovar phosphate concentrates typically contain +5% CO_2 and +10% LOI (at 1000°C), which indicates that concentrates exceeding 30% P_2O_5 may be obtained by calcination to remove water of hydration as well as organic and inorganic carbon. The substitution of CO_3 for PO_4 in the crystal lattice causes the Bayovar phosphate concentrates to be highly reactive and well suited for use as direct application phosphate rock (DAPR).

Table 13-15: Concentrate Quality after Washing and Flotation

Composite	% P_2O_5	% Fe_2O_3	% MgO	% Al_2O_3	% CaO	MER[1]	COR[2]	Cd[3]
PH01	26.90	1.35	0.56	1.19	41.41	0.115	1.54	101
PH02	28.97	0.61	0.61	0.79	44.24	0.069	1.53	46
PH03	30.81	0.44	0.53	0.61	45.34	0.051	1.47	39
PH04	29.01	0.45	0.54	0.64	44.46	0.056	1.53	32
PH05	29.29	0.52	0.55	0.67	45.56	0.059	1.56	35
PH06	29.59	0.51	0.52	0.63	45.67	0.056	1.54	39
PH07	29.01	0.67	0.58	0.94	44.78	0.075	1.54	49
PH08	29.26	0.57	0.49	0.77	45.45	0.063	1.55	66
PH09	28.92	0.58	0.52	0.80	45.13	0.066	1.56	78
PH10	29.14	0.61	0.53	0.73	44.98	0.064	1.54	83
PH11	29.28	0.59	0.53	0.79	45.25	0.065	1.55	94
PH12	29.23	0.57	0.50	0.78	45.69	0.063	1.56	53
PH13	28.65	0.75	0.62	0.90	44.98	0.079	1.57	47

1: Minor Element Ratio = (%Fe_2O_3 + % MGO + %Al_2O_3)/%P_2O_5
2: Calcium Oxide Ratio = %CaO/%P_2O_5
3: Cadmium = parts per million

The Jacobs data for washed products after drum scrubbing, primary desliming, attrition scrubbing of 600/53 µm, attrition scrubbing of +600 µm, secondary desliming, and tertiary desliming are presented on Table 13-16. The "Deviation" given in the last row of the table is calculated by dividing the standard deviation by the mean.

The grade Deviations for the +600 µm and 600/53 µm are relatively small. In other words the +600 µm and 600/53 µm have similar grades for all phosphorite layers. Except for PH01, the +600 µm and 600/53 µm fractions consistently contain well in excess of 24% P_2O_5. The grade Deviation for the 150/53 µm fraction is 11.1%. This increased deviation results from the variable amount of near size gangue minerals in the different layers.

The % weight Deviations for the three size fractions are very large, indicating that the relative proportions of +600 µm, 600/53 µm, and 150/53 µm fractions differ considerably from layer to layer.

The laboratory data indicate the simple washing flowsheet with desliming at 53 µm can produce a concentrate averaging more than 24% P_2O_5 except for layers PH01 and PH02. If PH01 is excluded, the yield and P_2O_5 grade average 44.6% and 24.8% respectively. The average grade of +24% P_2O_5 is acceptable for DAPR. To bring the washed products from PH01 and PH02 up to a grade of 24% P_2O_5 it would be necessary to coarsen the tertiary slimes cut point. Similarly, to obtain washed products with a composite grade of 28% P_2O_5 from layers PH03 to PH13 it would be necessary to coarsen the tertiary slimes cut point.

Table 13-16: Laboratory Data for Washed Products

	% P_2O_5			% Weight of Ore		
	+600 um	600/150 um	150/53 um	+600 um	600/150 um	150/53 um
PH01	21.92	27.04	18.28	1.18	36.14	20.22
PH02	27.95	29.24	16.48	3.67	19.07	24.57
PH03	31.12	32.18	24.08	7.69	31.98	23.01
PH04	30.58	28.78	26.57	6.39	51.15	8.16
PH05	28.63	29.72	22.86	2.86	12.31	20.46
PH06	29.45	29.43	26.80	1.34	14.29	27.81
PH07	27.83	29.32	24.56	0.97	6.79	20.27
PH08	30.25	29.78	24.90	0.61	3.74	13.03
PH09	31.37	29.81	24.91	0.86	6.64	29.46
PH10	31.89	30.23	24.92	0.89	5.23	20.44
PH11	30.93	29.58	24.24	2.44	11.40	32.82
PH12	31.78	29.59	23.38	2.69	13.73	27.54
PH13	30.92	29.73	24.70	0.96	11.75	27.66
Mean	29.59	29.57	23.59	2.50	17.25	22.73
STDEV	1.43	0.84	2.63	2.30	13.48	7.05
Deviation	4.8%	2.8%	11.1%	92.0%	78.2%	31.0%

13.5 CONCLUSIONS

Testing of individual phosphorite layers established a robust process for production of a 29% P_2O_5 concentrate suitable for the manufacture of phosphoric acid. The same testwork demonstrated that the phosphorite layers were also suitable for the production of two qualities of DAPR. Focus selected the DAPR flowsheet to take advantage of the ease of beneficiation and unique reactivity of the Bayovar 12 phosphorite ores.

The flowsheet for producing two qualities of DAPR is illustrated as a block flow diagram in Figure 13-2. With reference to Figure 13-2, 24% P_2O_5 DAPR is produced when the tertiary desliming cut point is adjusted to 53 µm and 28% P_2O_5 DAPR is produced when the tertiary desliming cut point is adjusted to 105 µm. Coarsening the cut point from 53 µm to 105 µm rejects more lower grade near size material and thereby increases the concentrate grade. The hydrosizer (tertiary) cut point may be adjusted by increasing the teeter bed density set point or by increasing the flow of teeter water.

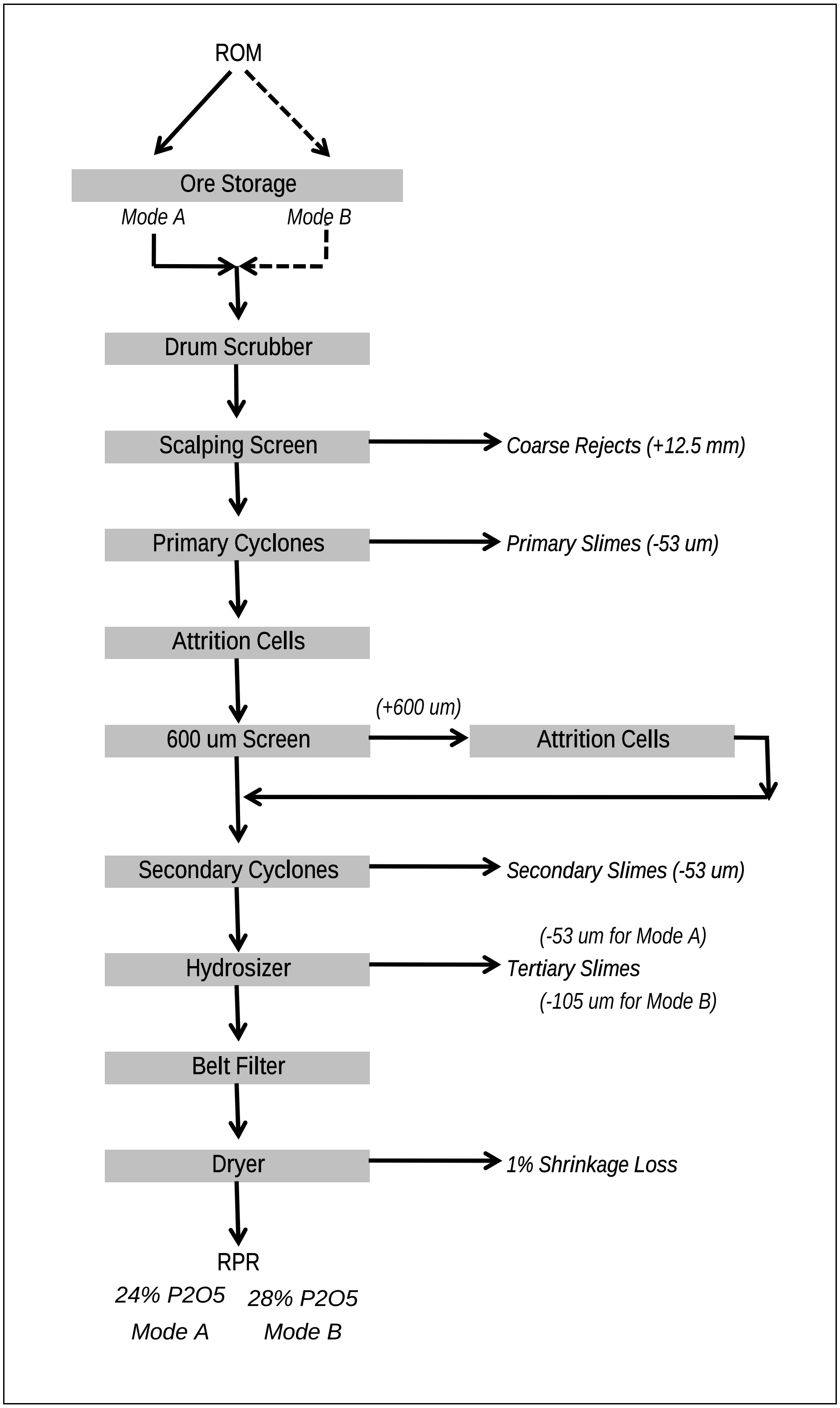


Figure 13-2: DAPR Flowsheet, Mode A (24%) and Mode B (28%)

13.6 RECOVERY ASSUMPTIONS

13.6.1 Recovery Assumptions

Statistical analysis of the lab data show that concentrate yield is predictable based on the P_2O_5 content of the plant feed; however, from Tables 13-7 and 13-16 it is evident that the particle size distribution (PSD) varies considerably from layer to layer.

The proposed beneficiation plant has two operating modes. Mode A is for producing nominally 24% P_2O_5 concentrate and Mode B is for producing nominally 28% P_2O_5 concentrate. Once the mine pit is fully opened, material from layers PH01, PH02, PH05, PH07, PH08, PH09 and PH10 will normally be treated with the plant in Mode A and material from layers PH03, PH04, PH06, PH11, PH12, and PH13 will be treated with the plant in Mode B. The process disaggregates the unconsolidated phosphorite material and then separates it according to size into a coarse fraction enriched in phosphate (concentrate) and a fine fraction that is phosphate poor (waste). Because recovery is dependent on both the PSD and the P_2O_5 content of the plant feed, the concentrate yield is predicted layer-by-layer and block-by-block within the geologic model. The concentrate yield increases if the P_2O_5 content of the plant feed increases and the yield decreases if the P_2O_5 content of the plant feed decreases.

Test data and projected run of mine (ROM) data for Mode A are presented on Table 13-17.

The weight recovery (yield) of concentrate from each phosphorite layer is projected based on the laboratory results, considering the % P_2O_5 and % soluble salt of the plant feed. The projected yield = (lab yield x ROM % P_2O_5 / lab feed % P_2O_5) x (1-%Salt/100).

Examples:

>600 µm concentrate for PH01

yield = Test % Wt >600 µm x ROM % P_2O_5 / Test feed % P_2O_5 x (1-ROM %salt/100)

yield = 1.18 x 12.33 / 15.42 x (1-6.51/100) = 0.88 % of ROM (dry basis with salt)

600/53 µm concentrate for PH02

yield = Test % Wt 600/53 µm x ROM % P_2O_5 / Test feed % P_2O_5 x (1-ROM %salt/100)

yield = 43.64 x 11.15 / 11.35 x (1-7.71/100) = 39.58 % of ROM (dry basis with salt)

Because the tertiary desliming cut point is 53 µm in Mode A, no recovery adjustment to the 600/53 µm fraction is required.

The combined yield is the sum of the >600 µm and 600/53 µm concentrate size fractions. The 27.493 million metric dry tons of ROM processed in Mode A is projected to contain 1.77 million tons salt, 10.614 million tons of concentrate, and by difference 15.110 million tons of -53um (-270 mesh) plant waste.

The % P_2O_5 of concentrate recovered from each phosphorite layer is projected to be the same as the % P_2O_5 of the concentrate recovered from the composite sample of that layer by laboratory testing. This projection holds if the ROM % P_2O_5 differs from the composite sample tested. The weight averaged concentrate grade from Mode A is projected to be 24.14% P_2O_5. The projected Mode A LOM feed tonnes in Table 13-17 are 1.3% higher than those forecast in mine production schedule Table 16-1. The projected Mode B LOM feed tonnes in Table 13-18 are 1.3% lower than those forecast in Table 16-1. The total LOM feed tonnes for combined products is balanced.

Table 13-17: Projected Grade and Yield of Resources Processed from Operating Mode A (24%)

Data from Laboratory Testing of Composite Samples [1]

	% P2O5				% Weight		
Layer	Feed	+600 um	600/53 um	Combined	+600 um	600/53 um	Combined
PH01	15.42	21.92	23.90	23.86	1.18	56.37	57.55
PH02	11.35	27.95	22.07	22.52	3.67	43.64	47.31
PH05	10.37	28.63	25.45	25.70	2.86	32.77	35.63
PH07	9.07	27.83	25.75	25.83	0.97	27.06	28.03
PH08	6.29	30.25	25.99	26.14	0.61	16.77	17.38
PH09	11.52	31.37	25.81	25.94	0.86	36.10	36.96
PH10	8.65	31.89	26.00	26.20	0.89	25.67	26.56

Data Projected for ROM [2]

	ROM		% P2O5				% Weight			600/53 um Factors[3]	
Layer	Mtons	Salt %	ROM	+600 um	600/53 um	Combined	+600 um	600/53 um	Combined	Grade	Recovery
PH01	5.278	6.51	12.33	21.92	23.90	23.86	0.88	42.15	43.03	-	100%
PH02	10.000	7.71	11.15	27.95	22.07	22.52	3.33	39.58	42.91	-	100%
PH05	0.825	6.93	12.13	28.63	25.45	25.70	3.11	35.68	38.80	-	100%
PH07	3.261	5.84	10.28	27.83	25.75	25.83	1.03	28.87	29.91	-	100%
PH08	2.186	5.43	10.69	30.25	25.99	26.14	0.98	26.96	27.94	-	100%
PH09	4.434	4.48	12.39	31.37	25.81	26.14	0.88	37.09	37.97	-	100%
PH10	1.509	4.68	10.46	31.89	26.00	25.94	1.03	29.59	30.61	-	100%
Total	27.493	6.43	11.54			24.14	1.87	36.73	38.61		
Mtons	27.493	1.769	3.172			2.562			10.614		

(1) The Lab data are on a dry-salt-free basis.
(2) The ROM data are on a dry with salt basis. The % P2O5 of the +600 um, 600/53 um fractions are projected to remain constant for each phosphorite layer. If the ROM % P2O5 exceeds the test feed % P2O5, the concentrate yield increase proportionately. Similarly if the ROM % P2O5 is less than the test feed % P2O5, the concentrate yield decreases proportionately.
(3) The desliming cut point is 53 um. From Tables 13.6 & 13.7, the grade increase = 0% P2O5 and the % weight recovery of 600/53 um =100%

Test data and projected resource data for Mode B are presented on Table 13-18.

The weight recovery (yield) of concentrate from each phosphorite layer is projected based on the laboratory results, considering the plant feed % P_2O_5 and % soluble salt.

The projected yield = lab yield x resource % P_2O_5 / lab feed % P_2O_5 x (1-%Salt/100). For Mode B, the concentrate size fractions are +600 µm and 600/105 µm. Consequently the laboratory data for the 600/53 µm concentrate fraction must be adjusted to 600/105 µm. The adjustments for each phosphorite layer are based on the data in Tables 13-6 (grade) and 13-7 (recovery)

Examples:

+600 µm concentrate for PH03

yield = Test % Wt >600 µm x ROM % P_2O_5 / Test feed % P_2O_5 x (1-ROM %salt/100)

yield = 7.69 x 17.52 / 19.38 x (1-5.23/100) = 6.59 % of ROM (dry basis with salt)

600/53 µm fraction for PH04

yield = Test % Wt 600/53 µm x ROM % P_2O_5 / Test feed % P_2O_5 x (1-ROM %salt/100)

yield = 59.31 x 13.59 / 20.37 x (1-5.35/100) = 37.45

Because the tertiary desliming cut point is 105 µm in Mode B, a recovery adjustment to the 600/53 µm fraction is required. The recovery adjustments are shown on Table 13-18.

600/105 µm concentrate for PH04

yield = 600/53 µm % of ROM x recovery adjustment

yield = 37.45 x 88.72% = 33.23 % of ROM (dry basis with salt)

The combined yield is the sum of the +600 µm and 600/105 µm concentrate size fractions. The 31.278 million metric dry tons of ROM processed by Mode B is projected to contain 1.207 million tons salt, 10.269 million tons of concentrate, and by difference 19.802 million tons of plant waste.

The % P_2O_5 of concentrate recovered from each phosphorite layer is projected to be the same as the % P_2O_5 of the concentrate recovered from the composite sample of that layer by laboratory testing. This projection holds if the resource % P_2O_5 differs from the composite sample tested. The grade adjustment to the 600/53 µm depends on the phosphorite layer and the desliming cut point. The weight averaged concentrate grade produced by Mode B is projected to be 28.83% P_2O_5.

The grade adjustments for a cut point of 105 µm are shown on Table 13-18.

Example:

600/53 µm fraction for PH04

fraction % P_2O_5 = lab grade = 28.48% P_2O_5

600/105 µm concentrate for PH04

Concentrate % P_2O_5 = lab grade + adjustment

Concentrate % P_2O_5 =28.48 +1.00 = 29.48 P_2O_5

Table 13-18: Projected Grade and Yield from Operating Mode B (28%)

Table 13-18: Projected Grade and Yield with Mode B

			Data from Laboratory Testing of Composite Samples (1)								
			% P2O5				% Weight				
Layer			Feed	+600 um	600/53 um	Combined	+600 um	600/53 um	Combined		
PH03			19.38	31.12	28.79	29.08	7.69	54.99	62.68		
PH04			20.37	30.58	28.48	28.68	6.39	59.31	65.70		
PH06			13.53	29.45	27.69	27.75	1.34	42.10	43.44		
PH11			13.69	30.93	25.62	25.89	2.44	44.22	46.66		
PH12			13.00	31.78	25.45	25.83	2.69	41.27	43.96		
PH13			12.23	30.92	26.20	26.31	0.96	39.41	40.37		
	Data Projected for ROM(2)										
	ROM		% P2O5				% Weight			600/53 um Factors(3)	
Layer	Mtons	Salt %	ROM	+600 um	600/105 um	Combined	+600 um	600/105 um	Combined	Grade	Recovery
PH03	5.264	5.23	17.52	31.12	30.33	30.44	6.59	40.64	47.23	1.54	86.26%
PH04	1.757	5.35	13.59	30.58	29.48	29.60	4.03	33.22	37.25	1.00	88.72%
PH06	6.045	4.26	13.56	29.45	28.88	28.91	1.29	27.54	28.82	1.19	68.17%
PH11	3.685	4.13	12.85	30.93	27.95	28.18	2.20	25.71	27.90	2.33	64.60%
PH12	5.028	2.98	13.93	31.78	27.86	28.22	2.80	27.71	30.51	2.41	64.59%
PH13	9.499	2.93	13.73	30.92	27.66	27.77	1.05	28.68	29.73	1.46	66.79%
Total	31.278	3.86	14.26	30.98	28.65	28.83	2.61	30.22	32.83		
Mtons	31.278	1.207	4.459			2.961			10.269		

(1) The Lab data are on a dry-salt-free basis.
(2) The ROM data are on a dry with salt basis. The % P2O5 of the +600 um, 600/53 um fractions are projected to remain constant for each phosphorite layer. If the ROM % P2O5 exceeds the test feed % P2O5, the concentrate yield increase proportionately. Similarly if the ROM % P2O5 is less than the test feed % P2O5, the concentrate yield decreases proportionately.
(3) The desliming cut point is 105 um. From Tables 13-6 & 13-7, the Grade increases and the % weight recoveries of 600/53 um are shown above.

13.7 RECOMMENDATIONS

The individual layer test work used composite samples prepared from the cores of nine drill holes within the global resource. Subsequent resource drilling and chemical analyses of 62 drillholes indicate that the average grade of the phosphorite layers has increased from 12.7% to 13.7% P_2O_5. It is therefore recommended that additional testing be performed with core samples of phosphorite layers from different locations within the proposed mine pit to determine if the recovery assumptions and predicted concentrate grades are applicable over the entire mine pit. The proposed number of drill holes to be included in this variability testing is five for the first 5-year mining block and two each for the 2nd, 3rd, and 4th 5-year mining blocks, for a total of 11 holes. If the proposed test work results demonstrate that the recovery assumptions and predicted concentrate grades are valid over the entire mine pit, the variability testing may be terminated and pilot plant testwork initiated. However, if the recovery assumptions and predicted concentrate grades are not applicable over the entire mine pit, more variability testing within the problematic mining blocks would be recommended prior to pilot plant testing.

Additional settling tests and if applicable, filtration tests are recommended to establish unit area requirements for high rate thickeners and filtration rates for concentrates prepared according to the flowsheet. This work could be performed in conjunction with the variability testing program. Future bench-scale metallurgical testing, if practical, should be performed with samples that have not been dried. If dried samples are available for use, they should be rehydrated to approximate the insitu moisture content prior to testing.

14 MINERAL RESOURCE ESTIMATES

14.1 DEFINITION OF MINERAL RESOURCES

For estimating the phosphate Mineral Resources for Bayovar 12 Project, Golder has applied the definitions of "Mineral Resource" as set forth in the CIM Definitions Standards adopted May 10, 2014 (CIMDS) by the Canadian Institute of Mining, Metallurgy and Petroleum Council.

Under CIMDS, a Mineral Resource is defined as:

"…a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

Mineral Resources are subdivided into classes of Measured, Indicated, and Inferred, with the level of confidence reducing with each class respectively. Mineral Resources are always reported as in situ tonnage and are not adjusted for mining losses or mining recovery.

14.2 MINERAL RESOURCE ESTIMATION METHODOLOGY

14.2.1 General

Geological modelling and subsequent mineral resource estimation was performed by the Golder Qualified Person in accordance with Golder internal modelling and resource estimation guidelines and in accordance with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (May 2003 edition).

The geological data compilation, interpretation, geological modelling and Mineral Resource estimation methods and procedures are described in the following sections.

14.2.2 Geological Database

All available Phase 1 and Phase 2 drill hole data and observations provided by Focus were compiled and loaded into an MS Access geological database. Using the database Golder performed a series of in-house visual basic scripts, designed to review and identify common problems in geological base data, on the raw data to ensure that the base data were free of errors or omissions.

Golder identified a limited number of minor typographic errors and omissions that were reviewed with Focus personnel prior to being corrected by Golder.

Based on the differences identified between the collar elevation versus the DEM topography model elevation it was decided that the precision of the DEM data were more reliable than the collar elevation surveys, and as a result, all collar elevations were adjusted to the topography surface elevation. The structural model was reviewed for all drill holes to ensure this adjustment did not result in the creation of structural anomalies and none were identified.

14.2.3 Geological Interpretation

Once the geological base data was reviewed and deemed to be free of errors or omissions Golder independently reviewed all phosphorite bed picks and correlations.

Golder used Golden Software's Strater™ program to assist with the unit picks and correlation. The lithological data and observations, sample intervals and analytical results for each drill hole were imported into Strater and a series of downhole geological logs were created for each of the 62 Focus Phase 1 and Phase 2 drill holes. The phosphorite-diatomite contact roof and floor picks performed by the Focus drill site geologists were reviewed by Golder using the drill hole descriptive geological logs, core photographs and the down hole analytical results.

The review process identified a small number of occurrences where the phosphorite bed roof and floor contacts were not consistent with the analytical results; for example, a sample identified as occurring above the roof of the phosphorite bed in the geological log returned P_2O_5 grade results consistent with the phosphorite bed rather than the overlying diatomite bed. Minor mismatches between roof and floor contacts logged in the field and analytical results are not surprising given the gradational nature of the phosphorite and diatomite contacts and the difficulty in estimating subtle decreases or increases in the apatite pellet content in the drill core when establishing geological unit contacts during core logging. The instances were reviewed with Focus personnel prior to being adjusted by Golder to ensure the geological intervals were consistent with the analytical results.

Once the drill hole geological intervals were reconciled with the downhole analytical results, Golder performed a review of the overburden, phosphorite and diatomite bed correlation interpretations that were provided by Focus. Using the Strater drill hole geological logs, correlation fences were created in both the east-west and north south directions across the Bayovar 12 Concession. An example of the fence sections used for correlation purposes is presented in Figure 14-1. All 62 of the Focus drill holes were included in the correlation fences. Golder reviewed the drill hole to drill hole correlations for each overburden, phosphate and diatomite bed in the sequence.

As a result of this review process, a small number of phosphorite bed miscorrelations were identified by Golder; the miscorrelations commonly occurred in sequences where there were numerous thin, closely spaced phosphorite beds and units were incorrectly correlated between holes. There were also a few isolated instances where a unit was inadvertently mislabelled in the Focus drill hole record, resulting in a miscorrelation. The miscorrelations were reviewed with Focus personnel prior to being adjusted by Golder to ensure the geological intervals were properly correlated across the Focus drilling area.

Any revisions to the drill hole unit picks and or unit correlations were tabulated and updated in the geological database prior to commencing with the geological modelling process.

14.2.4 Topographic Modelling

Focus provided Golder with topography data for the project area in the form of a Digital Surface Model (DSM) in registered raster file format (.tif). The DSM was prepared by Pacific Geomatics Ltd. (in August 2014) using 1.5 m SPOT6 Tristereo satellite imagery. Gross errors were fixed in the DSM in stereo. The easting and northing data was adjusted by Pacific Geomatics Ltd. to fit with available 0.50 m data for the area. Although DSM models differ from Digital Elevation Models (DEM) in that DSM's do not process out features like vegetation and building, there are no such features present on the Bayovar 12 Concession.

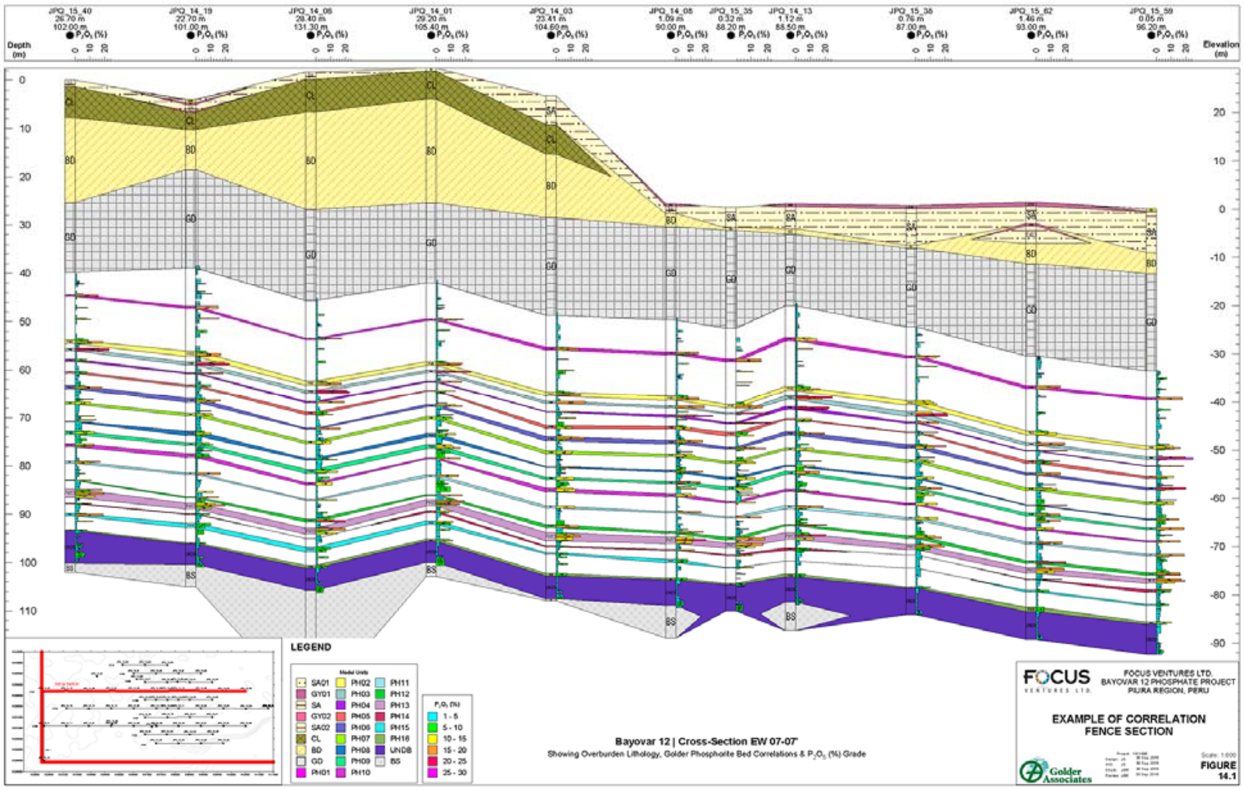


Figure 14-1: Example of Correlation Fence Section

The DSM covered the entire extent of the Focus Phase 1 and Phase 2 exploration program area as well as most of the remainder of the Bayovar 12 Concession. Golder processed the DSM provided to extract the data as ASCII format xyz point data.

The ASCII xyz format elevation data were then imported into MineScape StratModel and gridded using a 10 by 10 m grid covering the Bayovar 12 Concession. The gridded topography surface was then contoured on 2 m contour intervals and visually inspected to evaluate for potential problem areas.

The contoured topography surface was compared against publically available lower resolution SRTM and ASTER topography data for the area to ensure there were no significant differences in the DSM based topography model.

As a final check of the modelled topography surface, drill hole collar elevations were compared against the topography model elevations at the drill hole collar coordinates. For the most part the surveyed collar elevations were in good agreement with the topography elevations but there were a small number of isolated holes with differences in excess of 1 m (mean difference of 0.77 m, range of 0.07 to 4.11 m).

Based on the differences identified between the collar elevation versus the DSM topography model elevation it was decided that the precision of the DSM data were more reliable than the collar elevation surveys, and as a result, all collar elevations were adjusted to the topography surface elevation. Once the stratigraphic and structural model was generated (see below) the model surfaces were reviewed for all drill holes to ensure this adjustment did not result in the creation of structural anomalies and none were identified.

14.2.5 Stratigraphic and Structural Model

Stratigraphic and structural data from the verified geological database was imported to the StratModel™ application of the Ventyx MineScape geological modelling and mine planning software in preparation for the construction of a gridded stratigraphic and structural model.

The stratigraphic and structural grid modelling process in StratModel™ is controlled by a schema that defines the rules and procedures used in the construction of the stratigraphic and structural model. The schema includes parameters that indicate the type of interpolator, search radii, and extrapolation distances to be used in the modelling process. The schema also defines the stratigraphic sequence and the conformable and non-conformable relationships between adjacent stratigraphic units, as well as the relationships between stratigraphic units and structural features.

The stratigraphic grid model comprises gridded structure surfaces for each modeled overburden, phosphate, interburden and underburden unit. The modeled units in their stratigraphic order are presented in Figure 14-2. The structure grids created represent the individual unit roof, floor, vertical thickness (roof minus floor) and true thickness.

There were no faults included in the Bayovar 12 model schema as none were identified in the drill hole geological data nor were any faults identified in a review of the regional geological mapping coverage for the area surrounding the Bayovar 12 Concession.

A planar interpolator was used for thickness calculations while a Finite Element Method interpolator was used for surface calculations. The stratigraphic model is based on a series of gridded elevation and thickness horizons, with grid cell geometry of 50 by 50 m (east-west by north-south).

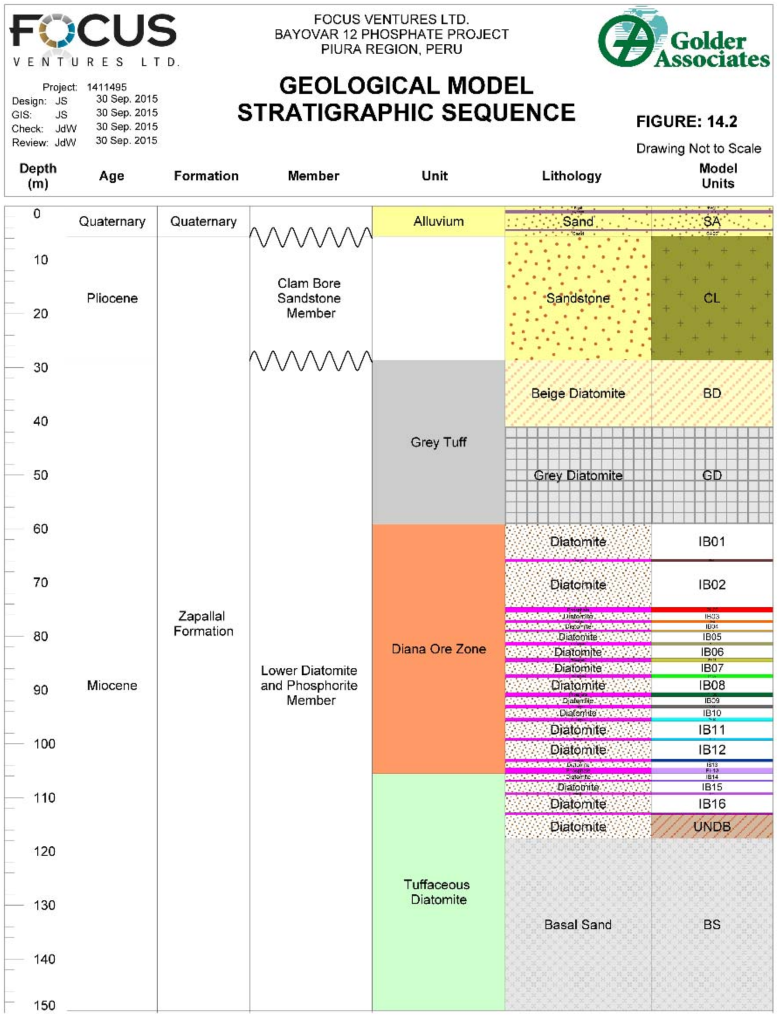


Figure 14-2: Geological Model Stratigraphic Sequence

Development of the MineScape stratigraphic and structural model is an iterative process involving gridding the data, checking the results by way of visual review of cross sections and structure isopleth maps, and then adding interpretive control points where needed before re-gridding. For the sake of brevity an example of a representative geological section from the Bayovar 12 model is presented in Figure 14-3, Representative Cross Section from the Geological Model and a representative structure isopleth map is presented in Figure 14-4, Representative Phosphorite Bed Thickness Isopleth Map; all additional geological sections and structure isopleth maps, including coverage for all 16 phosphate units, are available on the Focus website (www.focusventuresltd.com) in a downloadable supplemental graphics package (pdf format) for the Bayovar 12 Concession geological model.

14.2.6 Density/Specific Gravity

To facilitate the conversion of modelled volumes to tonnes Golder calculated dry basis and wet basis relative density values for all modelled phosphorite beds and waste units using relative density and moisture analyses data collected during the Phase 2 exploration drilling program. The replaces the global default relative density values used during the previous Mineral Resource estimate report.

In instances where there were three or more relative density samples for a specific phosphorite unit, a mean value was calculated using the dry density data and a wet density value was calculated using the dry density data and moisture data specific to that phosphorite unit. Where there were fewer than three relative density samples for a specific phosphorite unit a mean value was calculated using the dry density data and a wet density value was calculated using the dry density data and moisture data from the entire set of phosphorite relative density analyses (Table 14-1).

In a similar manner, default dry basis and wet basis relative density values were calculated for the diatomite interburden units and the overburden and underburden units (Table 14-2).

14.2.7 Grade Model

Using the verified modelling database and the finalized stratigraphic and structural model, a phosphate grade gridded model was developed using the StratModel application of MineScape. The grade model was developed using the same 20 by 20 m spaced grid that was used for the stratigraphic and structural grid model.

The grade grid model comprises gridded surfaces for each modeled grade parameter for each individual phosphate and diatomite bed; the grade model grid surfaces are spatially associated with the corresponding stratigraphic model grid surfaces. The grade parameters included in the model were: P_2O_5; Al_2O_3; CaO; Fe_2O_3; MgO; and SiO_2.

Ply basis grade samples were composited on a unit basis, creating a single composite sample interval for each phosphorite bed and each diatomite bed that is intersected in each drill hole. The grade composites were length and density weighted. The composited grade data were then gridded using a Finite Element Method interpolator.

Table 14-1: Phosphorite Unit Default Relative Density Values

Phosphorite Unit	Relative Density (g/cm³)	
	RD, Dry	RD, Wet
PH01	1.16	1.54
PH02	1.36	1.73
PH03	1.16	1.54
PH04	1.16	1.54
PH05	1.16	1.54
PH06	1.27	1.66
PH07	1.11	1.50
PH08	0.93	1.31
PH09	1.14	1.51
PH10	1.02	1.40
PH11	1.20	1.59
PH12	1.24	1.63
PH13	1.27	1.65
PH14	1.16	1.54
PH15	1.16	1.54
PH16	1.16	1.54

Table 14-2: Waste Unit Default Relative Density Values

Waste Unit	Relative Density (g/cm³)	
	RD, Dry	RD, Wet
Overburden	0.91	1.42
Interburden	0.76	0.98
Underburden	0.76	0.98

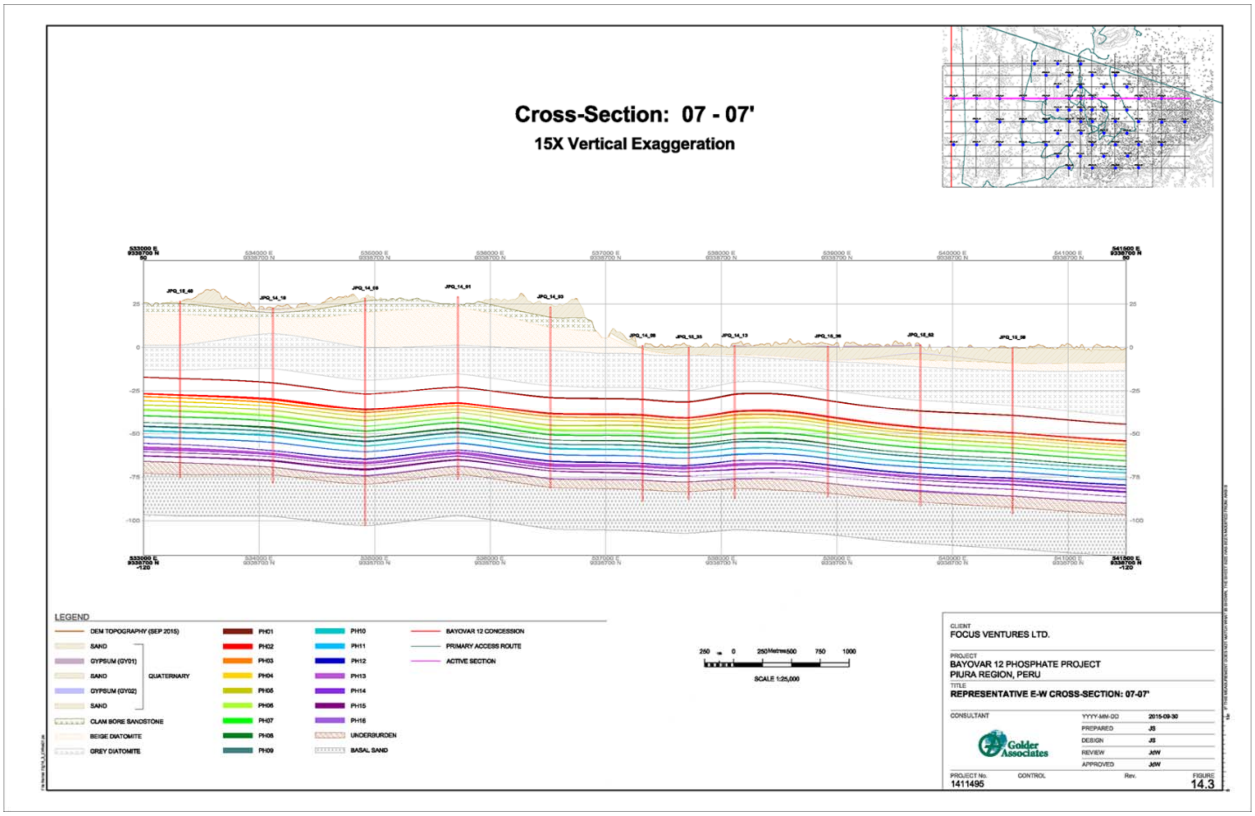


Figure 14-3: Representative Cross Section from the Geological Model

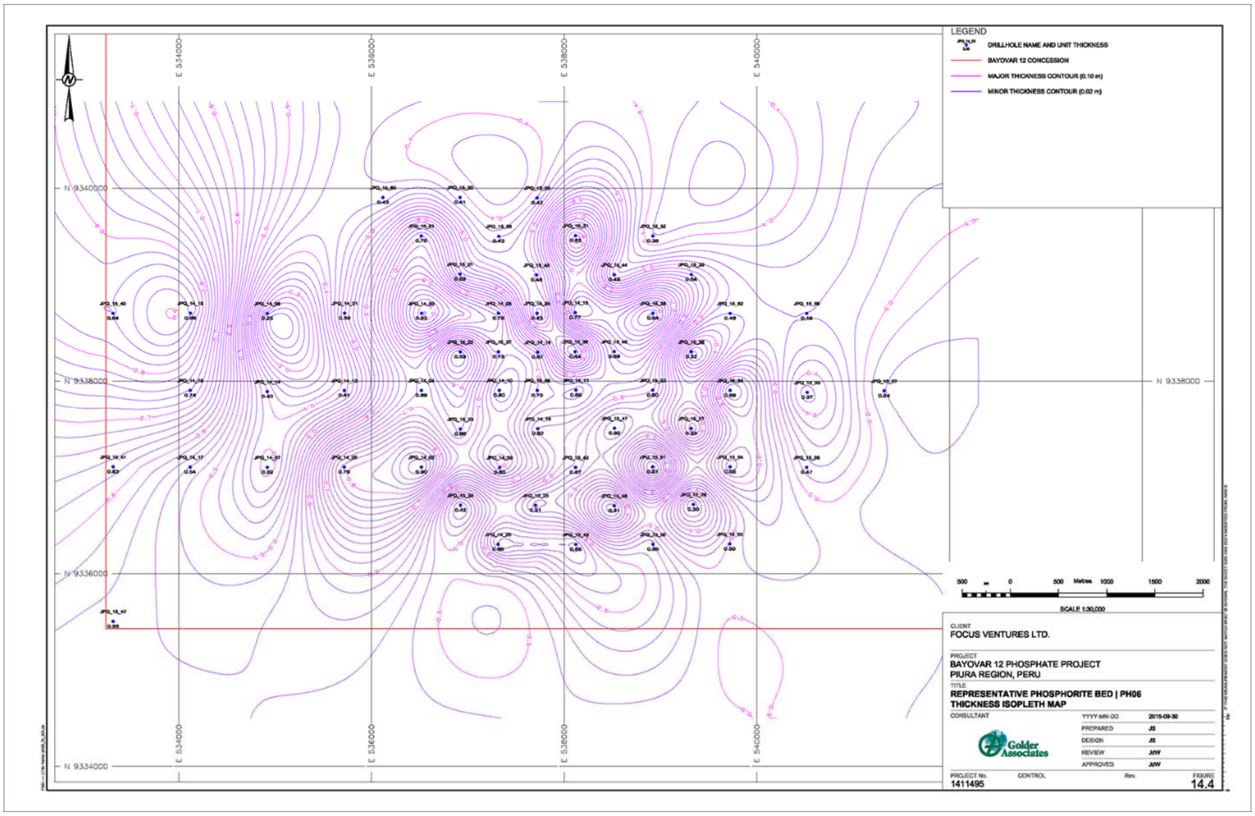


Figure 14-4: Representative Phosphorite Bed Thickness Isopleth Map

Grade isopleth maps were prepared for each gridded grade parameter for each individual phosphate unit. The grade contours were compared against postings from the drill hole composites to ensure the model was representative of the base drill hole analytical data. For the sake of brevity a representative grade isopleth map from the Bayovar 12 model is presented in Figure 14-5; all additional grade isopleth maps, including coverage for all 16 phosphate units, are available on the Focus website (www.focusventuresltd.com) in a downloadable supplemental graphics package (pdf format) for the Bayovar 12 Concession geological model.

Summary statistics for the composite grade parameters were also reviewed for each individual phosphate unit. Any potential outliers or issues identified in the visual inspection and statistical review were followed up by Golder to ensure the model was free from erroneous data, compositing errors or interpolation errors.

14.3 MINERAL RESOURCE ESTIMATION AND CLASSIFICATION

Using the finalized stratigraphic and structural model and the grade model, Golder estimated phosphate Mineral Resources for the Bayovar 12 Concession using the StratModel application of MineScape. Phosphate Mineral Resources were estimated for each individual phosphorite bed from PH01 through PH13. The lower three beds, PH14, PH15 and PH16, were excluded from the resource estimate due to limited thickness and low grades.

As per NI 43-101 guidelines and CIMDS definitions the Mineral Resources were reported as in situ tonnage and were not adjusted for mining losses or mining recovery. No minimum mining thickness or grade cut-off parameters were applied.

Resource volumes and grade were estimated for each phosphorite and diatomite bed using the corresponding unit roof and floor grids from the structural grid model. The volumes for each phosphorite and diatomite bed were then converted to tonnes using the phosphorite bed specific relative density values.

Golder performed classification of the Mineral Resources in the StratModel application of MineScape according to the CIMDS definitions as referenced in NI 43-101. Mineral resources have been classified into Measured, Indicated and Inferred Mineral Resource using area of influence polygons around points of observation. A point of observation is defined as a complete intercept of the bed (both roof and floor intercepted) with core recovery within the bed exceeding 90%.

Classification was performed individually for each phosphorite bed using drill hole intercepts on the floor of the unit for the location of the point of observation. The area of influence polygons were generated on the floor surface for each phosphorite bed rather than on the horizontal plane to allow for the dip of stratigraphy.

To aid in establishing Measured, Indicated and Inferred Mineral Resource area of influence polygons for use in resource classification, Golder performed a statistical and geostatistical analysis of the phosphorite bed thickness and P_2O_5 grade data. The review included evaluation of basic descriptive statistics as well as a preliminary review of variograms for phosphorite thickness and P_2O_5 grade for all 16 phosphorite beds. Given the relatively limited dataset (62 drill holes) and the general uniformity of thickness and grade across the concession area for most of the phosphorite beds, the preliminary review of the thickness and P_2O_5 variograms suggested a broad range for the variograms.

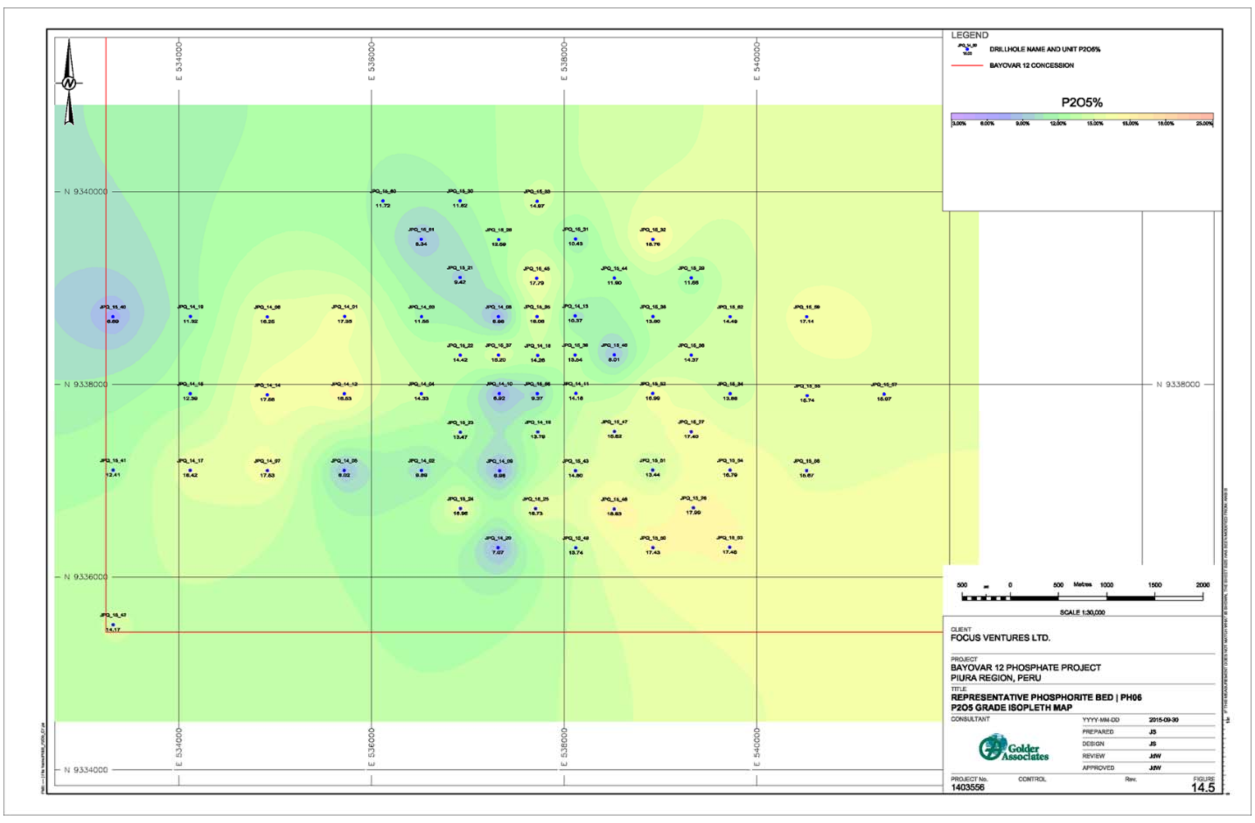


Figure 14-5: Representative Phosphorite Bed P_2O_5 Grade Isopleth Map

Golder recommends that additional close spaced drilling is required to improve the evaluation of close range variance and hopefully improve the variogram modelling. Golder recommends that the geostatistics be evaluated further once additional drilling and analytical data are available to determine if the results support a less conservative area of influence based classification for the Bayovar 12 Concession Mineral Resources.

The resultant areas of influence classification parameters used by Golder for the Bayovar 12 Concession Mineral Resource estimate are as follows:

- Measured Mineral Resources – 400 m spacing between points of observation
- Indicated Mineral Resources – 800 m spacing between points of observation
- Inferred Mineral Resources – 1,600 m spacing between points of observation

For the sake of brevity a representative Mineral Resource Classification map from the Bayovar 12 Mineral Resource estimate is presented in Figure 14-6, Representative Mineral Resource Classification Map; all additional Mineral Resource Classification maps, including coverage for all 13 phosphorite beds, are available on the Focus website (www.focusventuresltd.com) in a downloadable supplemental graphics package (pdf format) for the Bayovar 12 Concession Mineral Resource estimate.

14.4 STATEMENT OF MINERAL RESOURCES

A summary of the classified Mineral Resources for phosphorite beds PH01 through PH16 from the Focus Bayovar 12 Concession is presented in Table 14-3. Mineral Resources reported are inclusive of tonnes converted to Mineral Reserves in subsequent sections of this report.

Table 14-3 Summary of Mineral Resources, Beds PH01 to PH16

Category	Tonnes (Mt; wet)	Tonnes (Mt; dry)	P_2O_5 Grade (wt.%)
Measured	23.4	17.7	13.16
Indicated	277.1	209.5	13.04
Inferred	135.0	102.2	13.11

Note: Mt = million tonnes
Mineral Resources that are not Mineral Reserves have not demonstrated economic viability
No minimum thickness, grade cut-off or other mining parameters applied
Phosphorite bed specific wet and dry relative densities used for tonnage calculations

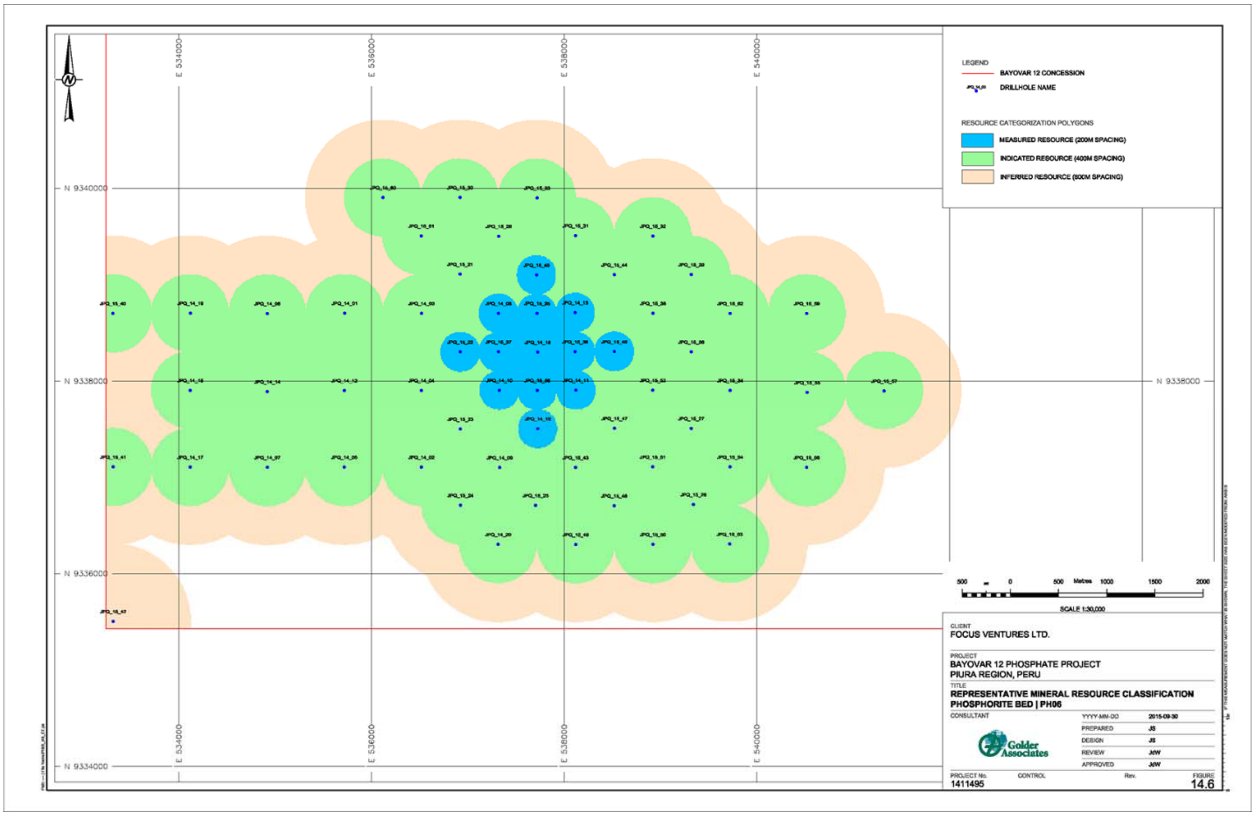


Figure 14-6: Representative Mineral Resource Classification Map

Estimated Mineral Resources on an individual phosphorite bed basis are presented in Figure 14-6, Table 14-5 and Table 14-6.

Table 14-4: Summary of Measured Mineral Resources, Beds PH01 to PH16

Phosphorite Bed	Vertical Thickness (m)	Volume (x 1000 m³)	Tonnes (Mt; wet)	Tonnes (Mt; dry)	P_2O_5 (wt.%)
PH01	0.53	964	1.5	1.1	14.5
PH02	0.92	1,666	2.9	2.3	12.0
PH03	0.53	973	1.5	1.1	19.8
PH04	0.38	692	1.1	0.8	16.3
PH05	0.53	957	1.5	1.1	9.4
PH06	0.64	1,169	1.9	1.5	14.0
PH07	0.68	1,241	1.9	1.4	10.5
PH08	0.43	782	1.0	0.7	12.7
PH09	0.49	889	1.3	1.0	13.4
PH10	0.41	742	1.0	0.8	10.6
PH11	0.37	679	1.1	0.8	15.4
PH12	0.49	888	1.4	1.1	15.6
PH13	1.03	1,882	3.1	2.4	13.8
PH14	0.27	489	0.8	0.6	10.4
PH15	0.32	377	0.6	0.4	9.2
PH16	0.28	517	0.8	0.6	8.1
All 16 Beds	0.62	14,906	23.4	17.7	13.2

Note: Mt = million tonnes
Mineral Resources that are not Mineral Reserves have not demonstrated economic viability
No minimum thickness, grade cut-off or other mining parameters applied
Phosphorite bed specific wet and dry relative densities used for tonnage calculations

Table 14-5: Summary of Indicated Mineral Resources, Beds PH01 to PH16

Phosphorite Bed	Vertical Thickness (m)	Volume (x 1000 m^3)	Tonnes (Mt; wet)	Tonnes (Mt; dry)	P_2O_5 (wt.%)
PH01	0.50	10,754	16.6	12.5	14.1
PH02	0.88	18,910	32.7	25.7	11.8
PH03	0.48	10,197	15.7	11.8	20.5
PH04	0.30	6,154	9.5	7.1	16.3
PH05	0.42	8,886	13.7	10.3	9.9
PH06	0.57	12,121	20.1	15.4	15.2
PH07	0.60	12,685	19.0	14.1	10.7
PH08	0.49	10,719	14.0	10.0	11.5
PH09	0.50	10,901	16.5	12.4	13.0
PH10	0.48	10,393	14.6	10.6	11.2
PH11	0.41	8,821	14.0	10.6	14.9
PH12	0.51	11,089	18.1	13.8	15.1
PH13	1.07	23,123	38.2	29.4	14.2
PH14	0.27	5,435	8.4	6.3	9.6
PH15	0.40	8,772	13.5	10.2	9.2
PH16	0.38	8,193	12.6	9.5	8.0
All 16 Beds	0.60	177,153	277.1	209.5	13.0

Note: Mt = million tonnes
Mineral Resources that are not Mineral Reserves have not demonstrated economic viability
No minimum thickness, grade cut-off or other mining parameters applied
Phosphorite bed specific wet and dry relative densities used for tonnage calculations

Table 14-6: Summary of Inferred Mineral Resources, Beds PH01 to PH16

Phosphorite Bed	Vertical Thickness (m)	Volume (x 1000 m^3)	Tonnes (Mt; wet)	Tonnes (Mt; dry)	P_2O_5 (wt.%)
PH01	0.50	5,593	8.6	6.5	14.2
PH02	0.87	9,164	15.9	12.5	11.8
PH03	0.47	5,030	7.7	5.8	20.5
PH04	0.31	3,460	5.3	4.0	16.5
PH05	0.41	4,292	6.6	5.0	9.9
PH06	0.56	5,796	9.6	7.4	15.3
PH07	0.57	5,552	8.3	6.2	10.8
PH08	0.45	3,790	5.0	3.5	11.6
PH09	0.50	5,399	8.2	6.1	13.0
PH10	0.46	4,571	6.4	4.7	11.4
PH11	0.40	4,183	6.7	5.0	14.9
PH12	0.50	5,270	8.6	6.5	15.1
PH13	1.07	11,820	19.5	15.0	14.2
PH14	0.28	3,502	5.4	4.1	9.5
PH15	0.42	5,297	8.2	6.1	9.3
PH16	0.36	3,297	5.1	3.8	8.1
All 16 Beds	0.60	86,016	135.0	102.2	13.1

Note: Mt = million tonnes
Mineral Resources that are not Mineral Reserves have not demonstrated economic viability
No minimum thickness, grade cut-off or other mining parameters applied
Phosphorite bed specific wet and dry relative densities used for tonnage calculations

As a result of the area of influence classification parameters applied and the 800 m nominal spacing of drill holes across most of the drill coverage, the bulk of the classified mineral Resources fall within the Indicated and Inferred mineral resource categories. A small area of Measured Mineral Resources was classified in the area of the 400 m spaced infill drilling. Additional infill drilling to 400 m spacing between drill holes will be required for the estimation of additional Measured Mineral Resources under the current Mineral Resource classification parameters.

15 MINERAL RESERVE ESTIMATES

15.1 GENERAL

The Mineral Reserve update developed by IMC uses the mineral resource model developed by Golder as described in Section 14 and expanded the open pit to develop the 20 Year Mineral Reserve encompassing the increased production of P_20_5 products during Years 1 through 3. The Mineral Reserve increases by 7.5% from 54.7 to 58.8 million dry tonnes. The updated open pit design criteria are the same as developed by Golder for the previous design work presented in the Pre-Feasibility Study dated December 18, 2015. The design criteria are included in this section for completeness.

In accordance with NI 43-101 for estimating Resources and Reserves of the Focus Bayovar 12 Phosphate Project, IMC has applied the definitions of "Mineral Resource" and "Mineral Reserve," as set forth in the updated CIM Definition Standards (CIMDS), effective in May 10, 2014, by the Canadian Institute of Mining, Metallurgy, and Petroleum Council.

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 reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study."

A Mineral Reserve is subdivided into two classes, Proven and Probable, with the level of confidence reducing with each class respectively. The CIMDS provides for a direct relationship between Indicated Mineral Resources and Probable Mineral Reserves, and between Measured Mineral Resources and Proven Mineral Reserves. Inferred Mineral Resources cannot be combined or reported with other categories.

Except as stated herein, IMC 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.

15.2 ESTIMATED PHOSPHORITE RESERVES

15.2.1 Introduction

The Bayovar 12 phosphate deposit has been delineated over an area of approximately 34 km^2. The deposit consists of sixteen mineralised units. This Mineral Reserve Estimate only concerns the upper 13 of the 16 total modeled units. No additional mineralization outside the deposit modeled was considered in the Mineral Resource and Reserve Estimates.

The updated Mineral Reserve statement is effective May 11, 2016.

15.2.2 Mining Model Development

15.2.2.1 Criteria for Determination of ROM Phosphorite

ROM mining surfaces were created by Golder in Ventyx, Minescape software Version 4.119 to account for anticipated 7.5-cm roof and 7.5-cm floor dilution gain where the phosphorite capa was greater than the minimum mineable thickness of 30 cm. These estimated dilution and mining loss factors are based on extracting the phosphorite with surface miners to recover the entire thickness of the capas to calculate ROM tonnages. ROM quality surfaces were also developed to account for the dilution gains using the quality data from the geological resource model.

15.2.2.2 Beneficiation Plant Yield and Product Quality Model

Once the ROM model had been developed and checked, a product model was developed in Vulcan to account for yields due to plant beneficiation. Golder developed the Vulcan product model using the recommendations published by Mr. Glenn Gruber (Gruber) in November 2015. Beneficiation plant yields are estimated using a set of capa-specific predictive equations that are driven by the ROM (feed) P_2O_5 quality. Gruber based his updated recommendations and equations on the results of bench-scale metallurgical testing performed on the metallurgical, geological holes drilled during 2014 and 2015. The overall effects of mining losses, dilution gains, and beneficiation on production are discussed in further detail in Section 16.5.

IMC has relied exclusively upon the process plant yields developed and recommended by Gruber and has not independently verified the results of this analysis.

15.2.2.3 Development of the 3D Block Model for Pit Optimisation

After developing the ROM and product surfaces, the Golder Vulcan model was blocked into 3D cells 40 m by 40 m by 1.6 m in the X, Y, and Z, respectively, for the purposes of pit optimisation. Using the same limits as the Vulcan resource model, approximately 21.7 M blocks were created. In areas requiring better definition of the phosphorite capas and interburden, sub-blocks were created with a minimum size of 5x5x0.2m. The relevant ROM and product quality attributes for each block were populated using Vulcan's block calculation file (bcf). The 3D block model was checked against the Minescape estimates of ROM and product tonnages and qualities to ensure that data integrity was maintained and that volumes, tonnages, or assay data were the same prior to 3D blocking. After review, the 3D block model was loaded into Maptek's Vulcan mine planning and mine design software for pit optimisation.

The 3D block model was transferred to IMC for use in the updated open pit design and mine production schedule. IMC has reviewed the procedures used to develop the 3D block model and its attributes and accepts it for use in the development of the updated mineral reserve estimate.

15.2.2.4 Mineral Reserve Estimation Methodology

The assessment of surface-mineable phosphorite reserves within the project area was based on a 20-year mine plan open-pit design which incorporates highwall laybacks on the estimated 20-year mineable reserve. The pit design resulted from a targeting process beginning with pit optimisation and proceeding through final pit design. The development of the 20-year mine plan pit is discussed in Section 16.

IMC started with the Golder pit design and expanded the design as needed to develop the updated 20 Year mineral reserve. The pit boundary assessments were completed over a range of unit revenue values for phosphate concentrate on a dry basis (i.e. saleable product), with unit costs for mining, processing, general and administration, production transportation etc. Golder's unit costs shown in Table 15-1 below, were based on their experience and studies on similar deposits.

Table 15-1: Summary of the Pro Forma Unit Costs used in the Golder Pit Optimisation Analysis

Description	Value (US$)
Overburden stripping and haulage	$1.89 / BCM
Phosphorite mining and haulage	$2.12 / ROM tonne
Mine supervision and admin	$3.00 / Product tonne
Beneficiation – Plant Line No. 1 & 2	$20.14 / Product tonne
Trucking and Port Costs	$10.09 /Product tonne

Notes:
Costs developed from similar open-pit mining operations in Chile, Argentina, and Peru.
Cost based on Golder's experience

The 3D block model loaded into Vulcan formed the initial basis of volume estimates for the extent of the project area. Using the modifying mining factors and plant performance yield and quality predictions, the blocks were populated with an economic value for the purpose of conducting LG pit optimisation.

Based on the requirement of the 20-year, 1.0 M tonnes phosphate concentrate (product tonnes) per year mine plan, the Golder final pit configuration was designed on the $90/tonne of phosphate concentrate pit optimisations. The resulting pit shell limits for these incremental pits were used for phasing design, refinement of the pit shell and developing the mine plan and schedule.

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

Table 15-2: Summary of Mine Design Parameters

Description	Value
Pit Wall angle	26 degrees
Bench Height	10m
Bench face (batter angle)	~ 65 degrees
Minimum mineable thickness	30 cm
Mining roof dilution gain	7.5 cm
Mining floor dilution gain	7.5 cm
Mining Recovery	100%
Pit Buffer from Bayovar Road	180 m
Target Average Product Grade	24+% P_2O_5 and 28+% P_2O_5

[1]For a complete discussion of the geotechnical units in the Bayovar 12 Project, please refer to Golder's geotechnical report "Focus Ventures Bayovar 12 Pre-Feasibility Report – Open Pit Design Recommendations"

IMC completed a suite of conformation floating cone algorithm runs to assure that the IMC pit design respected the economics of the project. The cost inputs are summarized in Table 15-3 and the cone geometry closely matched the final pit limits when a 70% factor was applied to revenue from the product sales.

Table 15-3: Inputs to IMC Pit Deign Confirmation Cone Runs

Description	Value (US$)
Average mining cost (all materials)	$1.79 per dry tonne mined
Process Plant cost	$8.01 per dry tonne of product
G&A cost	$2.38 per dry tonne of product
Product Transport	$10.09 per dry tonne of product

15.2.3 Mineral Reserve Estimation Statement

The Phosphate Mineral Reserves expressed as mined phosphorite dry tonnes and phosphate concentrate (product in dry tonnes) are shown in Table 15-4 and Table 15-5. The Mineral Reserve Estimate is based on a minimum capa thickness of 30 cm and a 6% P_20_5 cutoff grade for the run of mine (ROM) phosphorite. IMC considers the criteria used to define the 20-year mineral reserve to be reasonable for public reporting. This assumes the reserve would be exploitable using open-pit mining methods.

Table 15-4: Proven and Probable Reserves Expressed as ROM Mined Phosphorite

Capa	Proven Reserves	Probable Reserves	Total Reserves	P_2O_5 (%)
	Million Tonnes (dry)			
PH01	1.28	4.00	5.28	12.33
PH02	2.42	7.57	10.00	11.15
PH03	1.25	4.02	5.26	17.52
PH04	0.73	1.03	1.76	13.59
PH05	0.00	0.26	0.26	10.01
PH06	1.65	4.76	6.41	13.63
PH07	0.58	2.88	3.46	10.27
PH08	0.70	1.49	2.19	10.69
PH09	1.15	3.29	4.43	12.39
PH10	0.08	1.43	1.51	10.46
PH11	0.82	2.87	3.69	12.85
PH12	1.22	3.80	5.03	13.93
PH13	2.47	7.03	9.50	13.73
TOTAL	14.35	44.42	58.77	12.93

Notes: ROM tonnes within the updated Pre-Feasibility Study pit design

Table 15-5: Proven and Probable Reserves Expressed as Phosphate Concentrate Product

Capa	Proven Reserves	Probable Reserves	Total Reserves	P_2O_5 (%)
	Million Tonnes (dry)			
PH01[1]	0.56	1.69	2.25	23.85
PH02[1]	1.03	3.23	4.26	22.50
PH03[2]	0.57	1.90	2.47	29.06
PH04[2]	0.28	0.38	0.66	29.66
PH05[1]	0.00	0.08	0.08	25.70
PH06[2]	0.44	1.40	1.84	28.68
PH07[1]	0.17	0.87	1.04	25.49
PH08[1]	0.19	0.41	0.61	26.05
PH09[1]	0.42	1.25	1.67	25.92
PH10[1]	0.02	0.44	0.46	26.01
PH11[2]	0.23	0.80	1.02	28.20
PH12[2]	0.37	1.16	1.53	28.05
PH13[2]	0.69	2.11	2.80	27.81
TOTAL	4.97	15.72	20.70	26.24

Notes (1) Product A (24% P_20_5) (2) Product B (28% P_20_5) within the updated Pre-Feasibility Study pit design

The Measured and Indicated Resource Estimates are inclusive of the Resources comprising the Proven and Probable Reserve Estimates described under Mineral Resource Estimates. The Mineral Reserve is limited by the 20 year production schedule and pit design. There are mineral resources outside of the pit design which meet the economic criteria to be Mineral Reserves if the time frame was extended beyond 20 years. Figure 15-1 shows the final pit geometry without the mine waste rock backfill.

15.2.4 Discussion of Potential Impacts of Factors on Mineral Reserve Estimate

The potential impacts to the mineral reserve estimate in the previous Pre-Feasibility Study are still valid and are included here.

Golder used gridded models of density for the resource and reserve estimates where available and applied the beneficiation model developed by Mr. Glenn Gruber (Gruber) to model expected mass yields and phosphate concentrate tonnages due to beneficiation. Gruber's evaluation and recommendations were based on the results of the metallurgical testing performed on the geological drill hole samples drilled in 2014 and 2015.

A basic assumption of this Report is that the estimated phosphorite 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.

Golder estimated a roof mining dilution gain of 7.5 cm, a floor dilution gain of 7.5 cm, and a geology and recovery factor of 100 percent. These assumed dilution and mining loss factors are based on extracting the phosphoritic beds using surface miners with closed spaced drilling to define bed horizons and integrated GPS to control dilution and recovery.

A market for the product at current and forecasted prices of US$145/tonne and US$185/tonne on a dry basis for Product A (24% P_20_5) and Product B (28% P_20_5) phosphate concentrate product respectively at the Bayovar 12 Phosphate Project was used for the reserve estimate.

15.2.5 Potential for Future Reserve Expansion

As stated in Section 15.2.4, the Mineral Reserves Estimate is based on the 20-year mine plan open-pit design with highwall laybacks and a target production rate of 1.0 M tonnes (dry) of phosphate concentrate per year. Although Mineral Resources exist outside the 20-year mine plan pit, the mine schedule was limited to the 20-year mine life. Mineral Resources outside of the designed pit were not considered in the Mineral Reserves Estimation. However, there is strong indication of future mineral reserve expansion through further economic evaluation.

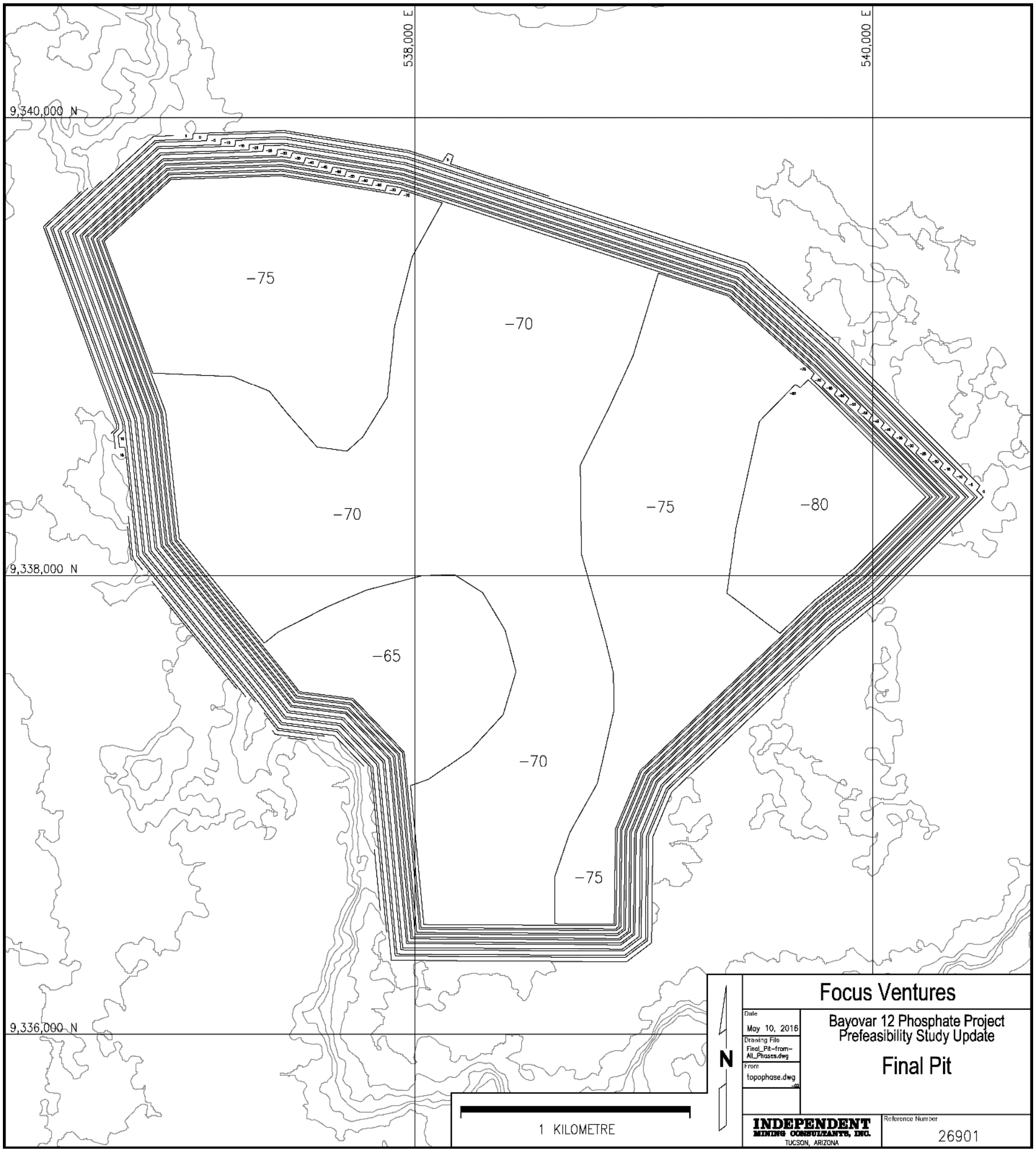


Figure 15-1: Final Pit Design

16 MINING METHODS

16.1 INTRODUCTION AND SUMMARY

The mine plan and production schedule are updated from the Pre-Feasibility Study dated December 18, 2015 to include:

- A change in the plant configuration to a single line plant which will produce two direct application phosphate rock fertilizer (DAPR) products: Product A with a target grade of 24% P_2O_5 and Product B at 28% P_2O_5;
- An increase in the plant production in Year 1 to 760,000 dry tonnes of product and to 1 million tonnes of dry product starting in Year 2;
- An adjustment to the mine sequence to maximize the in-pit backfill with waste rock;
- Changes in the capacity of loading and hauling fleets and a switch to an owner-operated fleet;
- An accelerated program of pre-production stripping
- The inputs to the mine design and plan related to the geotechnical parameters and mining model are the same as the previous Pre-Feasibility Study and are included in this report for completeness.

The annual mine production schedule is summarized in Table 16-1. The mine schedule has a target of approximately one million dry tonnes of the combined two products per year with stockpiling of the products at the plant and port load out area to absorb the fluctuations in the mine schedule and shipping schedule. Figure 16-1 is a general arrangement of the pit area, plant, and tailings storage facility (TSF). The pit is shown at the end of mining which includes a major portion of it backfilled with waste. Waste rock not placed in the pit backfills will be stored on surface to the west and south of the pit or used for embankments. The TSF embankments will be constructed with mine waste in two stages.

Table 16-1: Mine Production Schedule

Annual	Plant Feed																	
	Product A		Product B		Total Plant Feed		Product A		Product B		Total Product		Waste					
Year	Seams 1,2,5,7,8,9,10		Seams 3,4,6,11,12,13				Seams 1,2,5,7,8,9,10		Seams 3,4,6,11,12,13				Ovbrdn	Interbrdn	Total Wst		
							Dry Product		Dry Product		Dry Product					Mill tonne	Waste BCM
	dry		dry		dry								dry	dry	dry	per tonne	per mill
	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	tonnes	tonnes	product	tonne
PP	363,875	10.91	0		363,875	10.91	137,189	23.85	0		137,189	23.85	19,694,417	4,280,198	23,974,614	2.65	65.89
1	1,084,105	10.54	541,308	15.21	1,625,413	12.10	423,430	22.90	199,722	28.92	623,152	24.83	12,419,658	10,340,074	22,759,732	2.61	14.00
2	1,168,006	10.60	1,916,795	13.88	3,084,801	12.64	414,805	24.09	608,043	28.51	1,022,848	26.72	10,842,823	9,923,638	20,766,461	3.02	6.73
3	1,522,259	11.22	1,708,948	13.49	3,231,207	12.42	594,446	23.71	529,539	28.40	1,123,985	25.92	8,111,781	12,254,579	20,366,361	2.87	6.30
4	1,161,639	11.30	1,778,600	13.45	2,940,239	12.60	413,059	24.77	545,303	28.58	958,362	26.94	8,072,246	12,217,219	20,289,464	3.07	6.90
5	1,403,848	10.92	1,537,343	14.43	2,941,191	12.75	503,264	24.38	503,937	28.43	1,007,201	26.41	10,972,880	9,591,991	20,564,871	2.92	6.99
6	1,087,484	11.02	1,551,597	14.54	2,639,081	13.09	425,824	23.30	525,219	28.58	951,043	26.22	13,004,702	8,099,318	21,104,020	2.77	8.00
7	1,347,987	11.31	1,636,829	14.06	2,984,816	12.82	500,046	24.23	520,133	28.32	1,020,179	26.32	10,821,599	10,545,758	21,367,358	2.93	7.16
8	1,724,288	11.43	1,496,719	15.07	3,221,007	13.12	681,517	23.75	533,293	28.55	1,214,810	25.86	7,690,390	12,545,176	20,235,566	2.65	6.28
9	968,626	11.74	1,833,260	12.78	2,801,886	12.42	345,906	25.29	503,403	28.13	849,309	26.97	9,703,071	10,353,034	20,056,105	3.30	7.16
10	1,644,078	11.05	1,084,600	15.77	2,728,678	12.93	609,698	23.92	420,170	29.00	1,029,868	25.99	11,303,971	8,802,827	20,106,798	2.65	7.37
11	1,497,045	11.96	1,555,063	14.24	3,052,108	13.12	605,687	24.03	491,520	28.11	1,097,207	25.86	8,010,245	12,107,596	20,117,842	2.78	6.59
12	1,455,609	11.26	1,602,800	14.38	3,058,409	12.90	543,078	23.94	569,830	29.11	1,112,908	26.59	13,402,273	8,092,090	21,494,363	2.75	7.03
13	1,283,124	12.00	1,751,438	13.41	3,034,562	12.81	499,468	24.65	507,993	27.95	1,007,461	26.31	9,087,878	11,611,066	20,698,943	3.01	6.82
14	1,529,449	10.84	1,317,341	15.68	2,846,790	13.08	574,110	23.52	493,766	28.83	1,067,876	25.98	10,554,953	9,744,350	20,299,303	2.67	7.13
15	1,576,647	11.97	1,647,228	13.90	3,223,875	12.96	594,771	25.00	495,618	27.99	1,090,389	26.36	6,917,891	13,353,914	20,271,804	2.96	6.29
16	1,263,887	11.64	1,760,097	15.27	3,023,984	13.75	543,676	22.83	645,086	28.75	1,188,762	26.04	15,536,870	7,474,622	23,011,492	2.54	7.61
17	1,376,064	11.78	1,667,333	14.00	3,043,397	13.00	473,187	25.79	502,742	28.02	975,929	26.94	10,416,509	11,112,207	21,528,716	3.12	7.07
18	1,232,940	11.99	1,184,029	14.87	2,416,969	13.40	546,495	22.93	422,440	28.77	968,935	25.48	9,544,279	8,820,262	18,364,540	2.49	7.60
19	1,609,196	11.62	1,569,700	14.41	3,178,896	13.00	627,779	24.03	509,659	28.23	1,137,438	25.91	23,575	13,922,420	13,945,996	2.79	4.39
20	824,101	11.50	2,506,656	14.35	3,330,757	13.64	308,758	24.44	802,038	28.29	1,110,796	27.22	0	11,216,241	11,216,241	3.00	3.37
Total	27,124,257	11.39	31,647,684	14.26	58,771,941	12.93	10,366,193	24.03	10,329,454	28.46	20,695,647	26.24	206,132,011	216,408,579	422,540,589	2.84	7.19

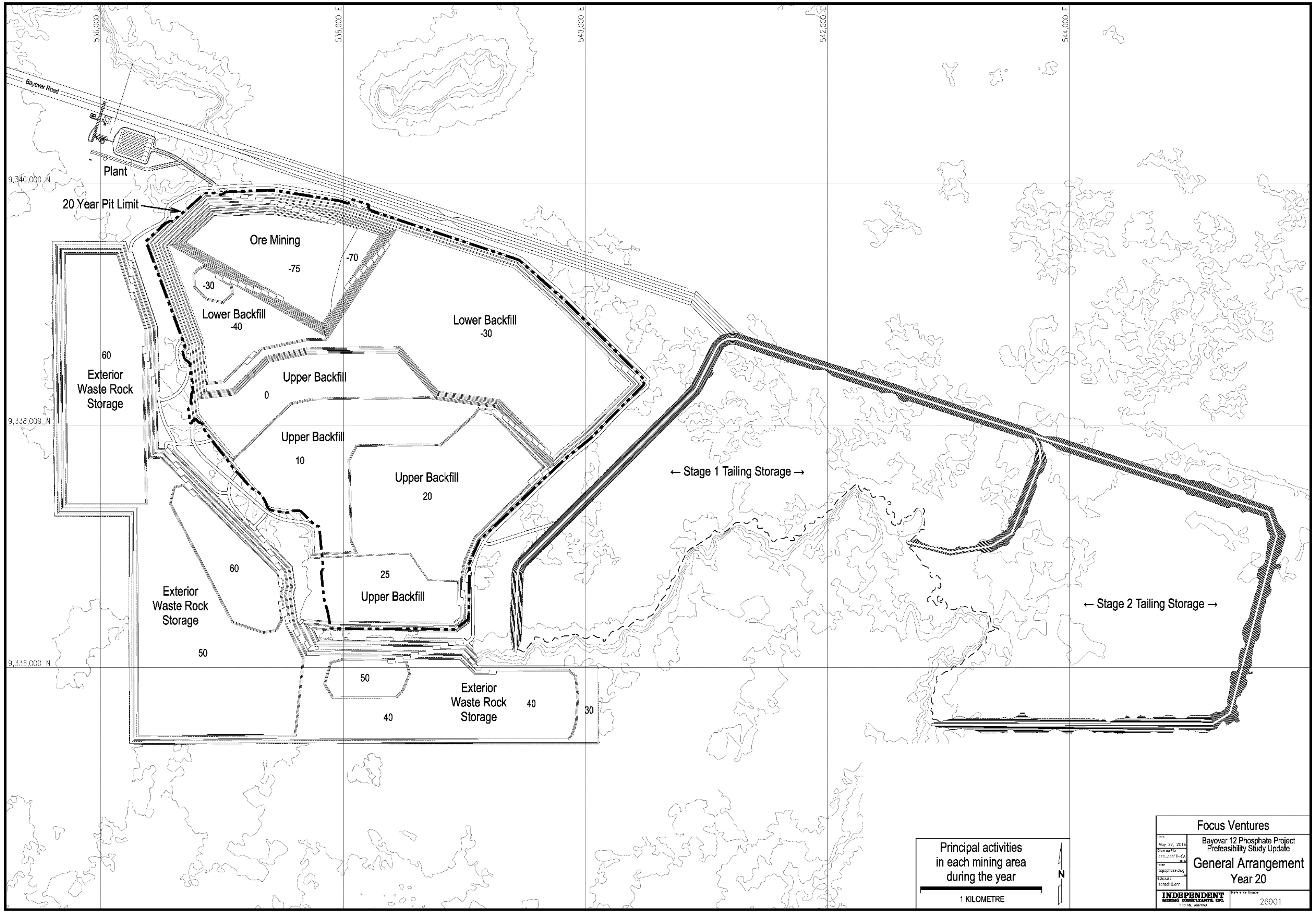


Figure 16-1: Mine General Arrangement

16.2 MINING OVERVIEW

The Project Site is contained within a low-lying, open, generally flat area, with elevations varying from 0 m to 30 m above mean sea level (amsl). The Bayovar 12 phosphorite deposit is a shallow-sea sedimentary deposit dipping between 1°and 2° toward the north-northeast with sixteen primary capas, thirteen of which will be mined. Due to the geological and topographic characteristics of the deposit, the overburden and a portion of the inter-bed waste will be mined with front end loaders (FEL) and the phosphorite ore and the remaining inter-bed waste with selective surface miners. The waste will be hauled with 89 tonne capacity trucks to the waste storage areas and phosphorite will be hauled the plant using the same size trucks. The trucks are equipped with 110 cubic meter boxes to maximize the carrying capacity of the trucks due to the low density of the materials being hauled.

The phosphorite extraction will be executed by a multiple bench open-pit mine progressing from south to north which will minimize the haul distance for the waste tonnages throughout the mine life (waste to ore ratio including pre-production stripping and ore stockpiling is 7.19). The pit is sub-divided into 13 mining phases each with independent access for mining the ore and waste and hauling it to the respective locations. The sequence is similar to the previous study which evaluated the pit sequence to start the mine in the lower waste stripping area of the deposit. Once a sufficient volume has been excavated, the waste rock is back-hauled into the mined-out area. Incorporating the in-pit backfill (IB) design slopes and required mined-out area necessary to allow waste rock to be backfilled within the pit, the in-pit backfilling will begin during Year 3 and continue for the balance of the 20 year schedule. During pre-production through Year 2, waste will be delivered to ex-pit dumps to the south and west of the pit. Waste will be delivered to the ex-pit dumps in subsequent years when the haul distance is shorter to an adjacent ex-pit dump than to a pit backfill area. This occurs when stripping of the upper benches in a new mining phase is started. The 423 million dry tonnes (511 million bcm's) of waste removed over life of the production schedule is divided between the backfill within the pit limits (73%) and the ex-pit dumps (26%) and TSF embankment construction (1%) on a bcm basis.

The front end loader (FEL) buckets (31 cubic meter) and the 89 tonne truck beds (110 cubic meter) are oversized due to the low density of the materials. The overburden and interburden will be mined with the FEL, assisted by dozers where the interburden is thin. To minimize mining dilution and maximize recovery and production capabilities, the phosphorite will be mined with GPS controlled selective surface miners. The mined phosphorite will then be loaded into the 89-tonne trucks either directly from the surface miner discharge conveyor, or using the FEL from the surface miner windrow.

The mine plan strives to maximize the number of available production faces for the various capas throughout the plan life in order to accommodate blended plant feed requirements. Production equipment is included in the plan to source ore from multiple capas and maintain the blend and feed tonnage to the plant. Additional flexibility is planned with ROM stockpile "fingers" to enable separation of individual or similar capas to provide feed during planned and unplanned mine shutdowns. Phosphorite delivered from the stockpile will be loaded by a dedicated FEL to the mill feed hopper to blend ROM ore as best as possible to meet the plant's ROM % P_2O_5 grade requirements and product shipping schedule.

Figure 16-2 shows a typical pit arrangement with overburden stripping, phosphorite mining and progressive closure using in-pit back-fill in the south part of the pit.

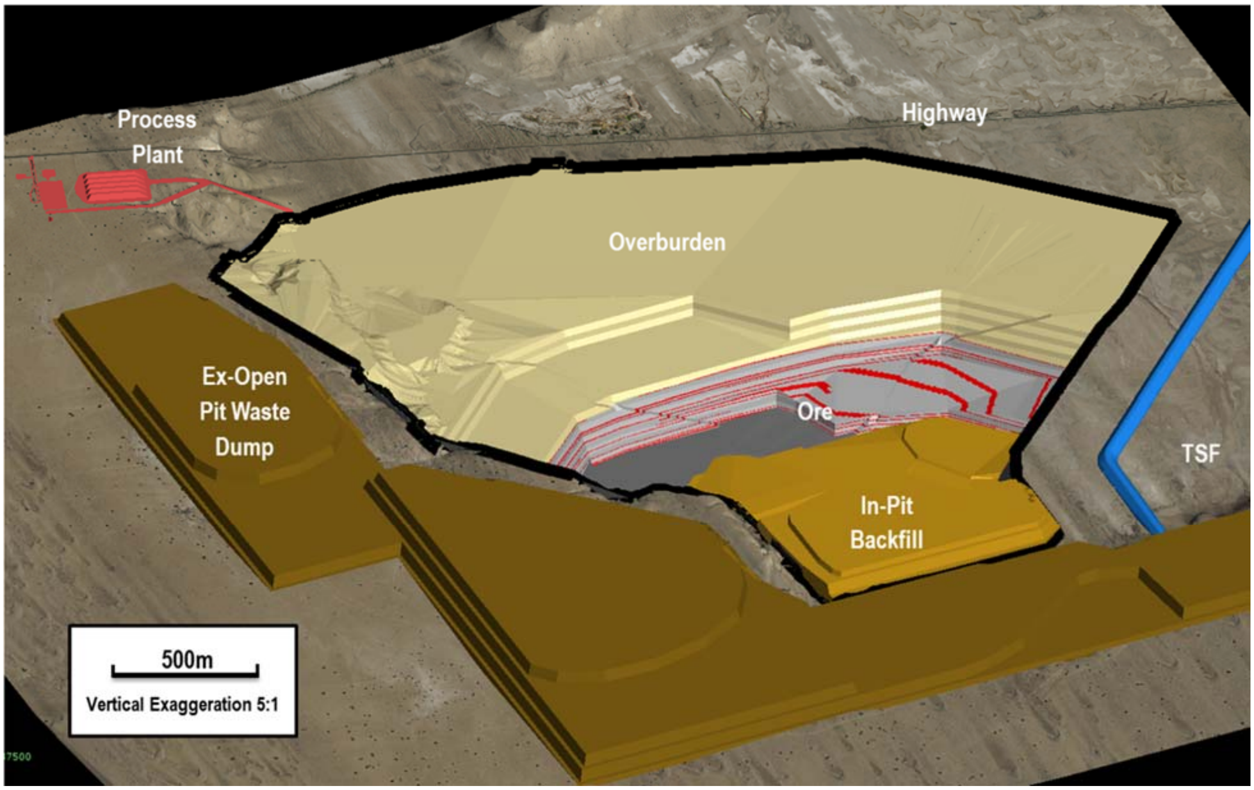


Figure 16-2: Year 7 Isometric Open Pit Arrangement

16.3 SURFACE WATER AND GROUNDWATER CONSTRAINTS

Precipitation in the region is strongly influenced by the El Niño phenomenon. El Niño is characterized by above normal water temperatures in the equatorial Pacific Ocean that results from periodic changes in current patterns. These events occur cyclically, on average every 10 years. Not all El Niño warm current events result in high precipitation in the project area. The total annual rainfall without the presence of a strong El Niño is very low with a mean of approximately 23.2 mm. Extreme precipitation events that have the potential to cause substantial flooding occur every 50 years on average. During the strong El Niño event that occurred in December 1997 to April 1998, total rainfall amounted to 1,238.8 mm. During this event, water inundated the low lying Sechura basin, including the planned Open Pit Area (OPA) and tailing storage facility (TSF), to an elevation of approximately 3m amsl forming the second-largest lake in Peru (Figure 16-3). The flood level achieved during the 1997-1998 El Niño event is the highest in the historical record and is applied as the design El Niño Event. The limits of this lake are used as the basis for protection berms that will be constructed around the facilities. A 5 m high flood protection berm will be constructed along the north margin of the OPA that will tie-in to the TSF embankment at the east end. The flood protection berms have a crest elevation of 5 m, with a 20 m crest width and 5H to 1V side slopes.

Storm water that collects in the open pit from direct precipitation will need to be managed. In addition, the topography of the area results in offsite run-on of surface water from the tablazo west of the open pit, into the pit. As the ex-pit waste dumps are built to the south and west of the open pit, the upper surface of them can be sloped away from the pit to divert the run off away from the pit. Some areas of offsite run-on can be diverted and prevented from flowing directly into the pit. The flat surface on the south and east sides of the pit does not allow storm water to be channeled and directed offsite. Crest berms along the pit crest are proposed to prevent surface water flows into the pit from these flat areas. Storm water that falls directly on the pit or that cannot be diverted will be collected in sumps in the pit that will be pumped to the TSF.

During the phased development of the pit, the area of the open pit and areas of offsite runoff that will contribute to the pit inflows will vary. The largest open pit area that will collect storm water is at the end of Year 2 prior to any pit backfill. The occurrences of pit flooding due to storm runoff will be infrequent, thus the planned pumping system would consist of portable diesel-powered pumps. This would provide the flexibility needed to assure a rapid response, as required and reduce non-productive periods. The water would accumulate in a sump in the lowest areas of the pit, and the option exists for discharge of water to either the TSF or other storm-water storage facility. Pumping capacity should be provided to accommodate the 20 year event with limited disruption to operations with a pumping capacity of 120 l/sec to 220 l/sec with an 85 m head gain. This pumping capacity should be in addition to the predicted seepage inflows. The mining activity during a year will be in multiple phases at differing elevations so ore mining can occur on higher benches if the pit bottom is flooded for short periods of time.

It is expected that sections of the surface water diversions will become in-filled with drifting sand and so it is proposed only to construct these when dictated by the phased open pit mining plan and in advance of a predicted El Niño event. The diversion channels will be unlined and typically have average bottom width of 10 m, a depth of 2 m and 2H to 1V side slopes. Total excavation required will be of the order of 110,000 m^3.

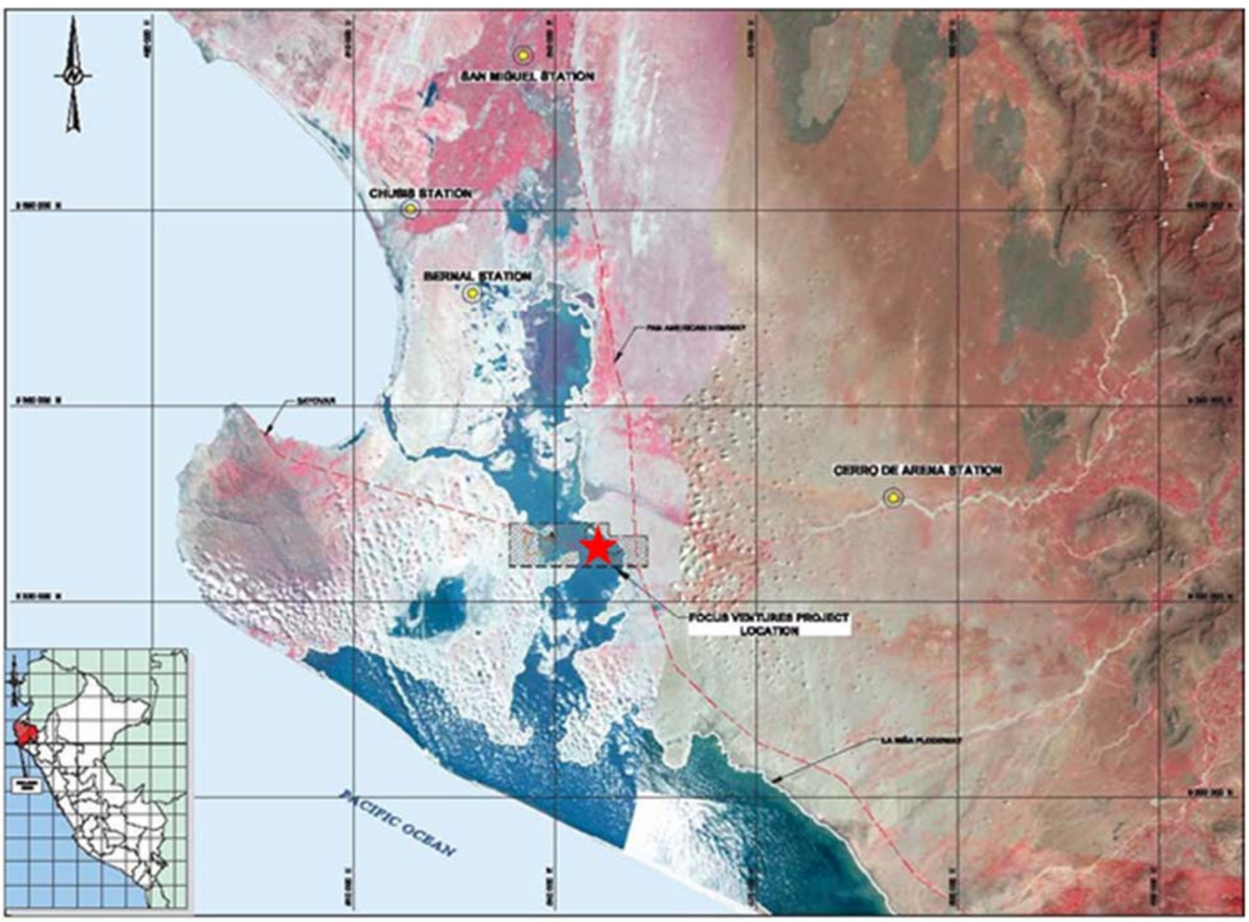


Figure 16-3: Lake formed by El Niño Extreme Event of 1997-1998

Available information indicates that the static water table is close to the ground surface in the OPA. The pit will mine down to approximately 80 m below the water table. The permeability of the Overburden and Interburden is based on 17 packer tests completed in geotechnical drill holes and the test results indicate low hydraulic conductivities of the Overburden and Interburden materials (Table 16-2). Due to the low hydraulic conductivities, dewatering wells were considered impractical for the pit development. The groundwater conditions are considered to be fully saturated for the full pit depth below mean sea level.

Table 16-2: Values of Hydraulic Conductivity (K) for Units Below Water Table

Unit	Hydraulic Conductivity (cm/sec)		
	Max.	Min.	Mean
Overburden	4.59E-05	9.00E-08	2.19E-06
Interburden	2.44E-05	1.87E-06	8.69E-06

The open pit will be backfilled as mining progresses. Based on the current mine plan, the maximum open area will occur when Year 2 is complete. The approximate area of the Year 2 pit is 1.76 km^2 which is similar to the Phase 1 pit presented in the previous Pre-Feasibility Study which is approximately 1,200 m wide by 1,300 m long with a total area of approximately 1.56 km^2. The seepage flow model was not re-examined for the new pit design and the results of the previous work are included below.

A preliminary transient seepage flow model was set up using the seepage module in SLIDE™. The transient analysis was run for a period of three years at three month time steps. The three year time frame generally corresponds with the rate that the pit phases will be developed to their full depth.

The combined seepage discharge from the floor and the side slopes of the pit at the end of Year 3 was estimated to be 7,550 m^3/day (87 l/sec). Most of the seepage will evaporate from the moist walls of the pit. Average daily evaporation rates (4.45 mm/day) applied over the open pit area are 6,950 m^3/day (80 l/s); slightly less than the estimated seepage inflows, suggesting that some management of seepage inflows should be anticipated. Anticipated seepage rates are estimated to be in the order of 2,400 m^3/day (28 l/sec).

Recently excavated faces are anticipated to generate seepage and seepage inflows from the working face will be managed through trenches and channels to convey water to sumps at the lowest level of the pit where solutions will be pumped to the TSF. Pit seepage is estimated to add approximately 4.0% to the tailings inflow rate and the tailings pond is sized to accommodate these additional inflows. Periods of lower evaporation and contributions from storm water will result in greater pumping rates.

Refinement of the water balance for the TSF sizing and evaluating the feasibility of pumping the seepage inflows to the tailings impoundment will be important for the feasibility level study under consideration.

The pit will extend, at its deepest, to approximately 80 m below the phreatic surface. Due to the low hydraulic conductivity of the materials, it is not anticipated that the pit walls will depressurize significantly, and geotechnical stability modelling for the pit slopes has been carried out with this assumption.

Golder evaluated the geologic, hydrologic, and geotechnical conditions for the design basis pit as detailed in Section 16.5. For prefeasibility planning Golder recommends a 26° overall wall angle. The open pit depth will reach approximately 80 m below the ground water table. Based on hydrogeological test data and indicated low hydraulic conductivities, dewatering wells are considered impractical for the pit development. The groundwater conditions are considered to be fully saturated for the full pit depth below mean sea level. Numerical seepage analyses and experience at similar nearby mines indicates that dry open-pit mining will be feasible and predicted groundwater seepage inflows will be manageable allowing for mining of the deposit with 65˚ temporary dig face angles. The bench design allows the bench faces to ravel as they dry and weather to angles as flat as 35° while maintaining 8.5 m safety benches.

Based on measured in situ material density and moisture-density (proctor test) laboratory results, Golder estimates an average waste rock swell factor of 30% from “in-situ” to loaded into a truck. IMC has reviewed documented swell factors for loosely consolidated materials and selected a 20% swell for both truck calculations and waste storage volumes for this Updated Pre-Feasibility Study (see Table 16-24) . Waste rock will be stacked in an external WSF early in the mine life and backfilled into the mined-out pit when pit advancement provides sufficient room for backfilling. External WSF are designed to an overall slope of 3.5H to 1V and the internal backfill (IB) are designed to an overall slope of 2.5H to 1V.

Mining recovery of the phosphorite was estimated based on an anticipated 7.5 centimetres of mining roof dilution gain and 7.5 cm of floor dilution gain. These at-grade dilution parameters were used to allow 100% recovery of the phophorite beds and were incorporated into the resource block model.

The mining method for the project will require mine haulage trucks. Excavator/truck mining will require stable haul roads and mine working surfaces for all pit levels and for all materials, including the extraction of the phosphorite. Furthermore, the excavator/truck method will require the construction and maintenance of permanent rock haul roads to the ex-pit WSF, maintenance facility, and ROM stockpile storage area adjacent to the processing plant.

Table 16-3: Summary of Mine Plan Parameters

Description	Value
Permanent inter-ramp wall angle	20° above -30 elevation, 26° below -30 elevation
Permanent wall operational FOS	1.3
Waste rock outslope angle	1V:3.5H
Waste rock spoil swell factor for storage volumes	20%
Minimum mineable matrix thickness	30 cm
Mining roof dilution	7.5 cm
Mining floor dilution	7.5 cm
Buffer between pit and flood protection berm	180m
Buffer between pit and ex-pit WSF	70m on south and plus 180m on west
Active mining months per year	12
Target average product phosphate rock grade (P_2O_4)	Two Products: A (24%) and B (28%)
Mine dewatering possible	No
Road base material to support truck traffic	Yes
Spoil Stackability	Yes

16.4 GEOTECHNICAL PARAMETERS

16.4.1 Open Pit Ground Investigation

This Updated Pre-Feasibility Study relied on the work completed by Golder for the previous PFS and the discussion of that work is included here for completeness. Limited geotechnical characterization has been carried out in the open pit area. Golder reviewed the geotechnical core logging database for exploration core holes JPQ-14-1 through 20 and JPQ-15-21 through 62. Geotechnical core logging was performed by Focus geologists collecting rock mass rating (RMR) parameters (Bieniawski, 1989). Rock mass ratings were assigned for each core interval.

Two geotechnical drill holes (JPQ2015-61 and 62) were completed in the OPA from May 11th to 20th, 2015. An additional geotechnical borehole (SPF-GT-01) was completed in the TSF in September 2015. Table 16-4 lists the borehole location, depth, and orientation. The borehole locations are shown on Figure 16-3. Borehole SPF-G-01 was relatively near the OPA and based on the horizontal continuity of the geologic strata, the geotechnical information from this borehole was considered relevant for the open pit recommendations.

Drilling was performed using wireline diamond coring methods, collecting HQ core. Standard penetration testing (SPT) was performed to collect blow count information and collect samples in the split spoon sampler. Additional material was collected from the core samples stored in Piura from the May 2015 drilling program, at the time of the September field program. Sampling of SPF-GT-1 included split spoon samples at typically 5 m spacing and undisturbed core intervals.

Table 16-4: Geotechnical Borehole Locations

Borehole	Location		Collar Elevation	Depth
	Easting	Northing		
JPQ2015-61	536518	9339505	25	105.0
JPQ2015-62	539722	9338698	2	93.0
SPF-GT-1	542401	9337342	10	40.0

In-situ hydraulic conductivity (Packer) testing was performed over typically 10 m intervals. Table 16-2 lists the results of the packer testing. The hydraulic conductivity of the Interburden is generally higher than the overburden and the geometric mean is four times higher. This higher permeability in the Interburden could be attributed to preferred flow along the capas.

Table 16-5 summarizes the laboratory testing that has been completed. Testing is mostly limited to the Overburden and Clambore units. Only one sample (which was disturbed) of the Interburden material has been collected and tested. No testing or drilling information is available for the underburden that underlies the mineralized zone. All geotechnical testing was performed at Golder's testing laboratory in Lima, Peru.

Table 16-5: Geotechnical Laboratory Testing

Test	Method	No. of Tests
Sieve/Hydrometer	ASTM D422	20
Moisture Content	ASTM D2216	19
Atterberg Limits	ASTM D4318	17
Specific Gravity of Soil Solids	ASTM D854	19
Standard Proctor	ASTM D698	4
Triaxial Shear	ASTM D2850	1
Direct Shear	ASTM D3080	2
Flex. Wall Permeability	ASTM D5084	2
California Bearing Ratio	ASTM 1883	3

Notes:
One sample of core was prepared and preserved in an undisturbed state for shipment to Golder's laboratory in Lima where consolidated, undrained triaxial tests (CU) were carried out to evaluate strength parameters for the overburden unit.

16.4.2 Open Pit Geology

The geological sequence defined by the block model and identified in drillholes comprises the following lithological units from top to bottom:

- Quaternary Alluvium/Aeolian deposits – Present at surface from 0 to 10 m thick. Unconsolidated sand. The Quaternary deposits occur at the surface and are anticipated to be dry.
- Clambore Sandstone – Present on tablazo above the water table 10 to 20 m thick
- Overburden – Present on the lower approximate 10 to 20 m of the tablazo above the water table and extending 35 m bmsl (below mean sea level). Overburden is composed of diatomaceous silt. The beige and gray Overburden are considered to have similar geotechnical properties based on available data and were combined into the same unit.
- Interburden and Capas – Present from 35 m bmsl to the base of the mineralized sequence at approximately 75 m to 85 m bmsl. Interbedded phosphorite seams (capas) occur throughout the sequence.
- Underburden – Present below the floor of the open pit. Composed of tuffaceous diatomite with numerous thin beds of tuff

The Clambore Sandstone will have very limited exposure in the open pit. The open pit will be developed almost entirely in the diatomaceous Overburden and Interburden units below the water table.

16.4.3 Strength Parameters

The available geotechnical information from the Bayovar 12 OPA is not sufficient to develop site specific strength parameters for the geotechnical units that are present. A detailed geotechnical program will be needed to support future feasibility slope design recommendations.

The geologic units that will comprise the majority of the pit walls are the same geologic units that are being successfully mined at the nearby Vale, Bayovar, Miski Mayo open pit and that were evaluated at a feasibility level at the Fosfatos Bayovar 9 projects. The ground water level at the Miski Mayo mine is near the ground surface, similar to Bayovar 12. The groundwater level at Bayovar 9 is approximately 50 m below the ground surface.

Comparison of the laboratory index testing from Bayovar 12 collected during the previous Prefeasibility study to the testing from Fosfatos Bayovor 9 deposit indicate the geotechnical properties are very similar and similar strength parameters may be anticipated.

The Clambore Sandstone is a non-plastic, fine sand to silt and classified as poorly graded sand with silt and a sandy silt. SPTs at shallow depth (5 m) met refusal indicating the material is cemented.

The moisture content, Atterberg limits, and grain size distribution for the samples from the May and September 2015 drilling programs indicate the Overburden and Interburden materials classify as high plasticity clay (CH) or elastic silt (MH). The moisture contents and liquid limits of the Overburden and Interburden samples are extremely high reflecting the highly porous nature of the diatomaceous materials. The moisture content of the Overburden was approximately 50% and as much as 83%. The Overburden and Interburden are fine grained with nearly 100% passing the #200 mesh screen. The fines are 65% to 75% silt size particles.

One triaxial shear test was completed from an undisturbed sample from borehole SPF-GT-01. This sample was taken from the gray diatomaceous Overburden unit. The sample was handled and transported to the laboratory to limit sample disturbance. Three tube samples were collected from the HQ core sample at the laboratory. Consolidated undrained triaxial tests were completed at confining pressures of 320 kPa, 640 kPa, and 1,280 kPa. Triaxial test results are provided in Golder 2015b and Mohr-Coulomb drained and undrained strength parameters are summarized in Table 16-6.

Table 16-6: Triaxial Test Results

Material	Undrained Strength		Drained Strength		Residual Drained Strength	
	Cohesion (kPa)	Friction Angle (°)	Cohesion (kPa)	Friction Angle (°)	Cohesion (kPa)	Friction Angle (°)
Overburden	414	12	417	21	284	24.7

The triaxial test stress-strain plots show a significant reduction in strength from the peak strength at failure to the residual strength after failure. Table 16-6 includes an estimate of the residual Mohr-Coulomb strength parameters after initial failure.

The Interburden is a pure diatomite and the Overburden and Underburden are a silty (tuffaceous) diatomite. Diatomite is an organogenetic sedimentary rock which is composed almost entirely of diatom fossils. Diatoms are microscopic single celled plants that grow in marine and fresh water. The diatom skeletons are composed primarily of silica. Diatomite has unusual geotechnical properties including high porosity, high water content, and low unit weight. The strength of the diatomite is transitional between a very weak rock and hard clay.

The strength of a clay is typically related to its prior stress history. The diatomaceous sediments of the Lower Zapallal Formation were likely subjected to a considerably greater depth of burial and are over-consolidated with respect to their current depth. Past lower ground water levels may have also contributed to pre-consolidation stress.

Studies carried out elsewhere on the behavior of diatomite indicate that unlike typical clay deposits, the undrained shear strength of diatomite in the pre-yield state (stress state lower than the yield stress) is independent of the consolidation stress level. When the consolidation stress is higher than the yield stress the compressibility of the diatomite increases dramatically. The microstructure of the diatomite remains unchanged in the pre-yield state but changes significantly in the vicinity of the yield stress. The breakage of diatom particles and the compression of inter-particle pores between the diatom particles in the post yield state contribute to the high compressibility of the diatomite.

The review of the literature indicates that diatomite does have properties that are distinct from typical clay or silt soils but general estimates of yield stress and strength of diatomite require site specific testing.

The triaxial test result obtained from borehole SPF-GT-01 is consistent with the behavior of diatomite in which the diatomite has a much higher peak strength than would be predicted by its stress history. However, when the peak strength is exceeded, the strength decreases to a residual value and the compressibility increases. This compression and reduction of pore space will cause an increase pore pressure and further reduce the shear strength. This type of soil behavior is capable of producing large scale slope failures if the peak yield strength is exceeded.

Often when designing with soils that have a have a strain weakening behavior, engineers will design using the residual strength. Alternatively, designs may be based on peak strength values applying higher target factors of safety. Typically, pit slopes are designed to target factors of safety of 1.3 for static conditions and 1.0 for pseudostatic loading. Appropriate target factors of safety when designing with peak strengths must consider the level of confidence in the geotechnical model, potential for local areas of weakness or structures, and potential triggering mechanisms such as high water pressures or earthquakes.

16.4.4 Stability Analyses

The geologic units that will comprise the majority of the pit walls are the same geologic units that are being successfully mined at the nearby Vale, Bayovar, Miski Mayo open pit and that were evaluated at a feasibility level at the Fosfatos Bayovar 9 projects. The ground water level at the Miski Mayo mine is near the ground surface, similar to Bayovar 12. The groundwater level at Bayovar 9 is approximately 50 m below the ground surface.

Assessment of ability to successfully develop stable slopes for the Bayovar 12 project at this prefeasibility level is based in large part on the experience at these similar nearby properties.

The available geotechnical information from the Bayovar 12 OPA is not sufficient to develop site specific strength parameters for the geotechnical units that are present. To evaluate the feasibility of development of stable slopes for the Bayovar 12 project based on available information, the geotechnical properties that were developed for the Bayovar 9 project were applied to the Bayovar 12 design cross sections and water table condition. Overall slope angles of 26° were analyzed considering the slope recommendations at Bayovar 9 ranged from 26° to 30° and the higher water table condition at Bayovar 12 would imply flatter slope angles. Information on the Underburden is lacking and the potential presence of weaker materials below the floor of the pit could influence stability. The Underburden strength was assumed to have the same strength as the Interburden and Overburden materials.

A generalized geotechnical model was developed that placed the top of the Overburden at an elevation of 0 m amsl. The Overburden extends to a depth of 35 m (35 m bmsl). The Interburden and capas extend from a depth of 35 m to 75 m to the pit floor. A second generalized model was developed for the pit slopes that mine back onto the tablazo on the southwest side of the design pit where the overall slope height will be up to 100 m. Materials exposed in the upper slopes on the tablazo, above the water table, include 10 m thick beige Overburden and 15 m thick Clambore Sandstone. Undrained shear strength properties were applied to the soil units below the water table in a total stress analysis based on the low permeability of the materials and expectation that the rate of mining and backfill will be rapid relative to the recovery of negative pore pressures developed through unloading. Effective stress parameters were applied above the water table. Peak strength parameters have been applied targeting higher factors of safety than are typically applied for pit slope design. The parameters applied in the stability models are summarized in Table 16-7.

Table 16-7: Parameters Applied in Bayovar 12 Stability Analysis

Material	Unit Weight (kN/m^3)	Cohesion (kPa)	Friction Angle (º)
Clambore Sandstone	17.5	0	35
Overburden Above Water Table	17.1	60	45
Overburden Below Water Table	17.7	400[1]	6[1]
Interburden Below Water Table	11.8	400[1]	6[1]
Underburden Below Water Table	11.8	400[1]	6[1]

Factors of safety were computed using Spencer's method of slices. Stability analyses were carried out for static conditions as well as pseudostatic loading to simulate earthquake shaking. A pseudostatic coefficient of 0.22 was applied based on one-half of the peak acceleration resulting from an earthquake with a return period of 475 years (10% probability of exceedance in 50 years) of 0.43g, estimated for the Bayovar 9 site. Stability analysis results are summarized in Table 16-8.

Table 16-8: Overall Pit Slope Stability Analysis Results

Stability Cross Section	Static FOS	Pseudostatic FOS
Section 1	2.33	1.25
Section 2	1.91	1.11

Stability analyses applying peak strength parameters developed for similar rock types at a nearby project area resulted in high factors of safety.

Therefore, pending additional site specific geotechnical characterization to support feasibility level design recommendations, overall 26-degree inter-ramp slopes are recommended. The wall design will be constructed in 10 m benches with bench face angles and catch bench widths varying by the geotechnical units described in Table 16-9. A summary of these bench face angles by geotechnical unit is provided in Table 16-9 below.

Table 16-9: Recommended Prefeasibility Slope Design Parameters

Geotechnical Unit	Interbench Height (m)	Dig Face Angle (°)	Final Bench Face Angle (°)	Catch Bench Width (m)	Inter-ramp Angle (°)
Quaternary Alluvium	Full thickness of Alluvium	30	30	10	20
Clambore Sandstone	10	65	40	8.5	26
Overburden (above water table)	10	65	45	8.5	28
Overburden (below water table)	10	65	35	6.5	26
Interburden (below water table)	10	65	35	6.5	26

16.4.5 Surface and Groundwater Management

16.4.5.1 Groundwater

As discussed in Section 16.2, the hydraulic conductivity of the Overburden and Interburden materials is low and dewatering in advance of mining is not considered practical. The average seepage flux rate into the pit at its maximum depth was estimated to be of the order of 4.84 mm/day while the average daily evaporation rate is 4.45 mm/day; slightly less than the estimated seepage inflows suggesting that some management of seepage inflows will be necessary. Seepage inflows will be pumped to the TSF.

The pit will extend at its deepest part to approximately 80 m below the phreatic surface. Due to the low hydraulic conductivity of the materials and the rate of mining and backfilling, it is not anticipated that the pit walls will depressurize significantly, and geotechnical stability modelling for the pit slopes has been carried out with this assumption.

16.4.5.2 Surface Water

The higher ground surface elevations on the tablazo along the west side of the open pit results in several areas that cannot be diverted and will contribute run-on to the open pit. The ground surface is essentially flat with the exception of the slope from the tablazo at an elevation of 25 m to the east where the ground surface is at sea level elevation. Storm water that falls on the flat surface areas is assumed to be retained in surface depressions and will not generate runoff. Precipitation that falls on the slope along the west side of the open pit will runoff toward the pit. Diversion channels can convey this runoff around the north and south ends of the pit area. However there are offsite run-on areas that cannot be diverted and will need to be managed with storm water that falls directly onto the pit. Storm water diverted around the north side of the pit may be channelled across or under the highway north of the OPA. Storm water diverted around the south side of the open pit will accumulate in the flat areas southeast of the open pit and west of the tailings due to the lack of surface gradient to convey surface water offsite. For planning purposes, diversion channels should have 10 m wide bottom width, 2H to 1V side slopes, 2 m deep, with a minimum 1% gradient. The diversion channels would be unarmored. Channels will require maintenance to remove accumulated windblown sand prior to storm events and potentially repair following storm events if significant damage occurs.

Where surface water cannot be diverted and offsite runoff does not flow directly into the pit, Crest berms are proposed at the crest of the open pit to prevent surface water from flowing into the pit. The berms should be constructed as dictated by the phased pit construction. Recommended berm dimensions are to an elevation of 1 m amsl and will have a nominal 3 m top width and 3H to 1V side slopes. The crest berm costs are assumed to be included in mining costs.

Figure 16-1 indicates aspects of Site-wide surface water management. The design is based on the previous PFS pit which is similar to the Updated PFS pit with the new design extending farther south. The site water management approach will be very close to the one presented in Figure 16-4.

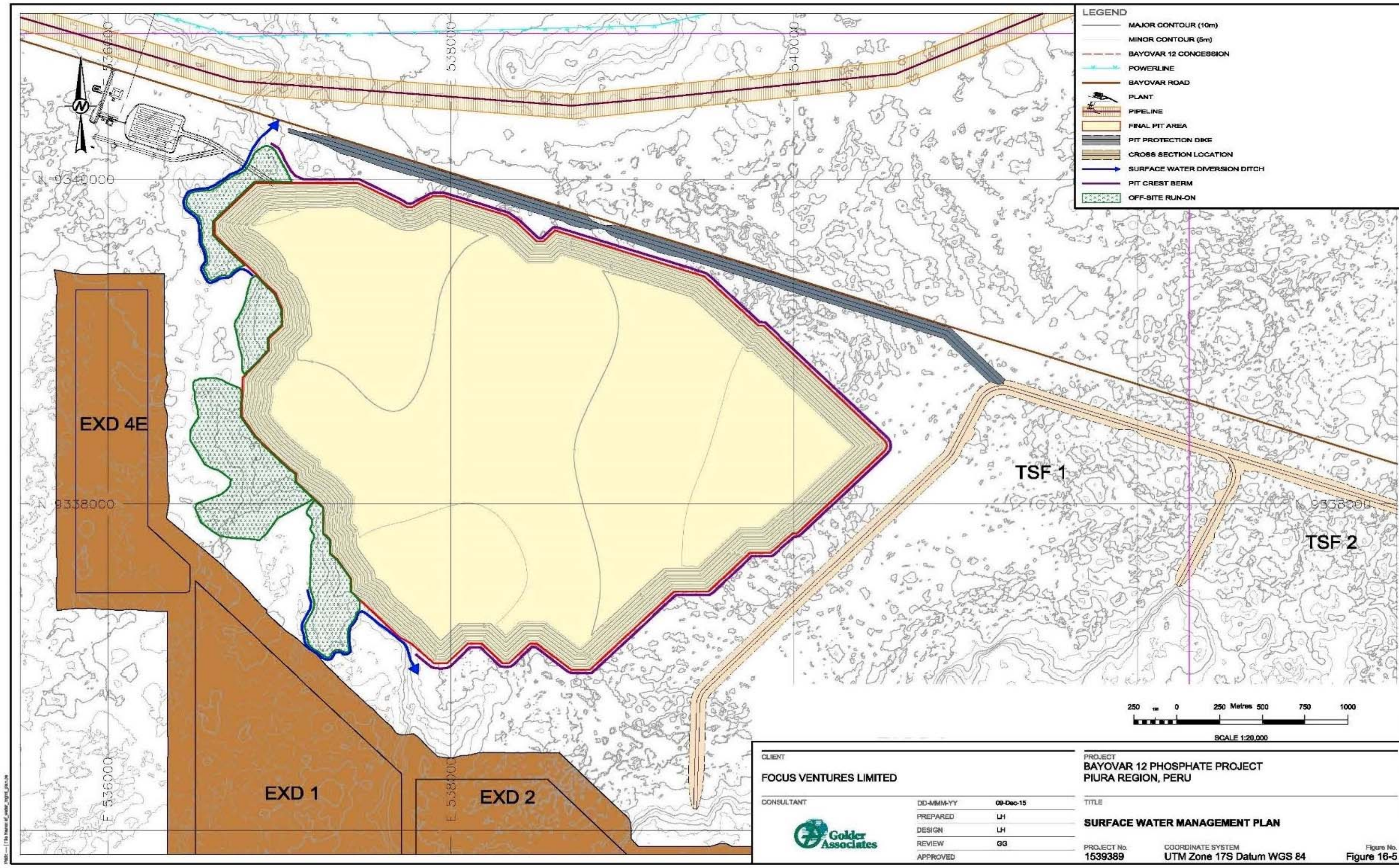


Figure 16-4: Site Water Management

16.4.6 Open-pit Trafficability

Haul roads will be required within the open pit to achieve the following actions:

- Removal of waste rock to the waste storage facility
- Removal of phosphorite to the beneficiation plant
- Removal of waste rock to the in-pit backfill
- Removal of waste rock to the tailings embankments
- The size and specification of the haul roads is dependent on the largest anticipated vehicles required to use the haul roads.

One California Bearing Ratio (CBR) test at three degrees of compaction was completed on a sample of Interburden material collected from bore hole JPQ2015-61. Testing was performed at compaction levels of 90%, 95%, and 100% of the maximum standard Proctor density under saturated conditions. The test results are summarized in Table 16-10 for various degrees of compaction.

Table 16-10: California Bearing Ratio Test Results

	CBR for 2.5 mm Penetration		
Geotechnical Unit	90% Max. Proctor Density	95% Max. Proctor Density	100% Max. Proctor Density
Interburden	2.1%	3.1%	3.7%

Table 16-11 provides typical values of CBR for a range of soil materials. The value obtained from the Interburden suggests the material will provide a poor road base. However, substantial increase in the CBR can result when fine grained soils are tested in an unsaturated condition. And well compacted dry diatomaceous Overburden and Interburden soils may achieve CBR values of 10% to 15%. Also, conclusions on trafficability cannot be made based on the results of one test. Therefore, it is recommended that additional testing be performed on the Overburden, Clambore Sandstone, Quaternary sands and other potential materials that may provide a more suitable road base. Testing should be performed for saturated and unsaturated conditions.

Table 16-11: Typical Values of CBR

Material	CBR (%)
Crushed Stone	20 – 100
Sandy Soils	5 – 40
Silty Soils	3 – 15
Clayey Soils	3 – 10
Organic Soils	1 – 5

Source: Washington State Department of Transportation Pavement Guide

Based on the information currently available and knowledge of the character of the diatomaceous materials that will provide the subgrade for the haulage roads and the anticipated seepage inflows that will occur as a result of mining below the water table, trafficability is anticipated to be an issue that could reduce haulage productivity or result in additional costs for import of suitable road base.

16.4.7 Open-pit Monitoring

Monitoring of the excavations will be a routine aspect of the operation and will require a high degree of visual inspection and monitoring of ground conditions. Monitoring may include the following actions:

- Inspections of slope condition
- Inspections of seepage drainage systems
- Installation and inspection of monitoring piezometers to confirm groundwater conditions
- Surveying to monitor settlement
- Installation and inspection of slip indicators
- Undertaking and monitoring the timely remediation of identified defects.

16.5 MINING MODEL

16.5.1 ROM Model

The Run Of Mine (ROM) model used for this updated PFS is the same model that Golder used for the previous PFS and the development of the ROM model is included here. IMC carried out random checks to verify its accuracy. The grid-based surface stratigraphic model used to estimate resources (geological model) was used as the basis for developing a ROM model for mine planning, design and scheduling. A ROM block model is distinguished from a geological model in that the ROM model identifies reserves to be mined whereas the geological model describes the in-situ resource. Mining surfaces were created to account for an anticipated 7.5 cm roof dilution and 7.5 cm floor dilution gain where the phosphorite capas were greater than the minimum mining thickness of 30 cm. Phosphorite capas less than 30cm were excluded from the mine plan. The anticipated dilution at the roof and floor should allow mining with essentially no losses based on the accuracy of extracting the phosphorite capas using GPS controlled surface miners. ROM quality surfaces were developed to account for the mining dilution gains. Dilution samples from the drillhole cores were used to model P_2O_5 dilution quality surfaces for each modelled capa using the same methodology used to model in-situ qualities. Due to insufficient dilution density data, the dilution material (interburden) was assumed to have a dry and wet density of 0.76 and 0.98 g/cm3 respectively.

The effects of dilution result in a reduction of P_2O_5 quality from in situ to ROM; given a constant roof and floor dilution gain, the overall reduction of P_2O_5 grade is dependent on capa thickness. The consistency and quality of a blended plant feed will have a significant effect on plant recovery and performance. It follows therefore that opportunities to increase the overall recovery of phosphorite exist by carefully managing the mining process.

16.5.1.1 Sources of Loss and Dilution

Phosphorite dilution will occur primarily at the seam roof and floor, however this occurs through a range of complex mechanisms. Given that the interburden material will be stripped using backhoe and subsequently mined using a selective surface miner, allocating loss and dilution to each capa to modify the in situ resource to a run of mine ore is necessary. Management of interburden stripping and phosphorite mining operating practices to manage loss and dilution will be important to delivering to the plant a consistent feed. The economics of phosphorite mining depend on the ability to use highly productive equipment and also to minimize dilution and maximize recovery. This is particularly true since the thickness of the capas varies within the deposit. Therefore, the incentive to take advantage of the economies of scale means that a compromise must be reached between rate of progress and the care required to control the resulting ROM phosphorite quality. The issue of losses and dilution with respect to mining accuracy, edge losses and effects of water are further discussed below.

16.5.1.2 Mining Accuracy

The estimation of mining recovery and dilution depends on the concept of mining accuracy. Whenever an attempt is made to separate materials at an interface, it must be accepted that sometimes the actual separation point will be high, and sometimes it will be low. The resulting cut surface will therefore be an approximation of the actual interface and will likely be a function of the surface miner which has an accuracy reported by the manufacturer of 7.5 cm. Based on this understanding, Golder developed a ROM model with a 7.5cm dilution gain at the roof and floor for mine planning purposes. The average interburden thickness (material overly the phosphorite capa) is 1.5m. To manage roof and floor dilution during mining, Golder envision a pre-drilling exercise on a dense grid (50m x 50m) to define the roof and floor surfaces of the phosphorite capas. These trial data would be used to develop a 3D mining surface representing the roof and floor of each capa which will be delivered to the surface-miner equipped with GPS to control the surface miner excavation to maintain overall recovery of the phophorite bed. Based on the accuracy of the surface miner and the pre-established 3D data grid which delineates the location of the capa roof and floor in x-y-z, IMC believes that 7.5cm roof and floor dilution is reasonable as the unit operations proposed to strip the interburden is envisioned as follows;

- Step 1 – strip interburden material (with truck shovel) to an approximate 0.3m height above phosphorite capa
- Step 2 – finish stripping with surface-miner to the pre-determined 7.5cm above capa roof
- Step 3 – mine phosphorite capa with the surface-miner calibrated to excavate the capa with the number of passes dependent on mining thickness.

16.5.1.3 Other losses and effects of water

In areas where the surface miner cannot recover all of the phosphorite capa, front end loaders or backhoes could be used. In addition, wet working conditions may lead to additional levels of loss. However, the amount of such losses will only be determined once mining has started and no accounting for such losses has been included in the ROM model.

16.5.2 Vulcan Product Model

Golder applied the beneficiation model provided by Gruber in November 2015 to the Vulcan ROM model to simulate the effects of beneficiation and predict phosphate concentrate product tonnages and qualities. Gruber based his recommendations on the results from the chemical and metallurgical testing performed on the geological drillhole samples drilled from May to December 2014 by Focus. Golder relied upon the expertise of QP Glenn Gruber's evaluation of the testing data and did not review the accuracy of the results provided in the metallurgical test database. Gruber provided his evaluation and recommendations to Golder in a technical report in November 2015. Gruber issued a revised version of this report (Rev B) in November 2015. Golder relied upon the expertise of Gruber's beneficiation model reports to estimate the recovered tonnes of concentrated phosphate and product qualities.

16.5.2.1 Beneficiation Model Overview

The beneficiation model developed by Gruber simulates the unit operations proposed for the full-scale beneficiation plant. A simplified diagram of this process has been provided by Gruber and is presented in Figure 16-4 below.

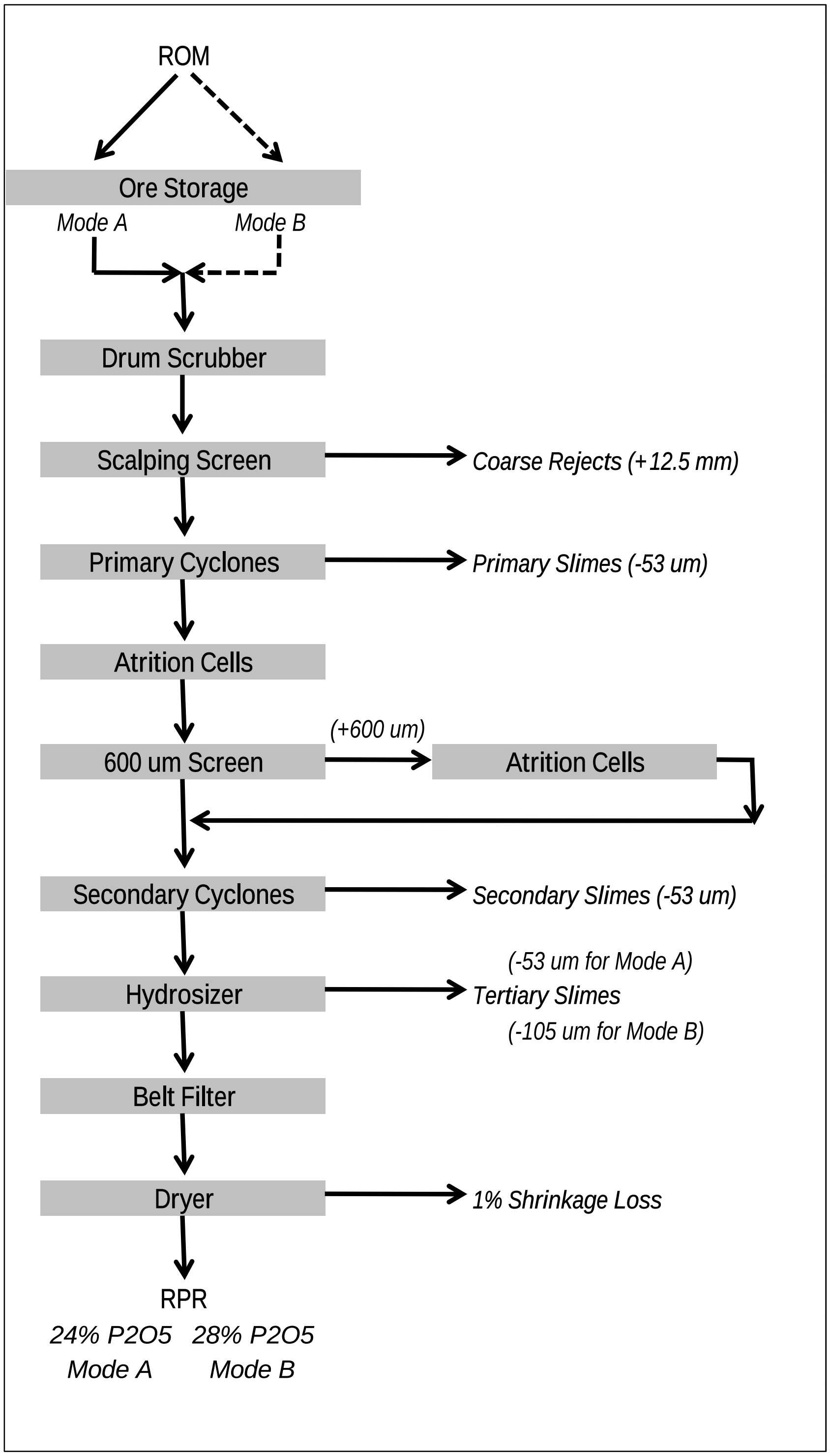


Figure 16-5: Beneficiation Process Flow Diagram (after Gruber 2015)

Data from the testwork were used to develop a flowsheet for producing 24% (Product A) and 28% (Product B) DAPR phosphate product since Focus intends to develop the deposit for the production of direct application fertilizer to take advantage of its high reactivity characteristics and for which it is suitable for use on acidic soils. To generalise the beneficiation process, Gruber segregated the scrubbing and attrition process schemes by capa based on a capa's susceptibility through processing to reach the required P_2O_5 concentrate grade. Capas were assigned to a Product using the majority process required to reach an acceptable concentrate P_2O_5 quality of 24% and or 28%.

Gruber concluded that a 24% P_2O_5 DAPR product through tertiary desliming at 53µm and a higher grade (28% P_2O_5 DAPR) product by coarsening the tertiary desliming to 105µm to reject lower grade near size material.

Table 16-12 below shows the process scheme to which each phosphate capa has been assigned.

Table 16-12: Beneficiation Process by Phosphate Capa

Phosphorite Capa	Product
PH01	A – 24%
PH02	A – 24%
PH03	B – 28%
PH04	B – 28%
PH05	A – 24%
PH06	B – 28%
PH07	A – 24%
PH08	A – 24%
PH09	A – 24%
PH10	A – 24%
PH11	B – 28%
PH12	B – 28%
PH13	B – 28%

Each beneficiation product was modelled in Vulcan for every capa shown in Table 16-12.

16.5.2.2 Concentrate Quality and Yield

Gruber developed equations for predicting the concentrate mass yields according to size fraction for each respective capa. Recoveries for each Product are presented in Table 16-13.

Table 16-13: Phosphate Recoveries for 24% P_2O_5 (Mode A) and 28% P_2O_5 Product (Mode B)

Operating Mode A												
	Resource			+28 mesh		28/270 mesh		+270 Concentrate		-270 mesh		P2O5
Capa	Metric Tons	% P2O5	% WSS	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	Recovery
PH01	5,278,423	12.33	6.51	0.88	21.92	42.14	23.90	43.02	23.86	50.47	4.09	83%
PH02	9,999,977	11.15	7.71	3.33	27.95	39.57	22.07	42.90	22.52	49.39	3.02	87%
PH05	824,845	12.13	6.93	3.11	28.63	35.68	25.44	38.79	25.70	54.28	3.98	82%
PH07	3,261,593	10.28	5.84	1.04	27.83	28.89	25.75	29.93	25.83	64.23	3.98	75%
PH08	2,185,906	10.69	5.43	0.98	30.25	26.95	25.99	27.93	26.14	66.64	5.08	68%
PH09	4,433,704	12.39	4.48	0.88	31.37	37.09	25.81	37.97	25.94	57.55	4.41	79%
PH10	1,509,066	10.46	4.68	1.03	31.89	29.60	26.00	30.63	26.20	64.69	3.77	77%
Combined	27,493,514	11.43	6.37	1.87	27.91	36.73	23.93	38.60	24.12	55.03	3.85	81%

Operating Mode B												
	Resource			+28 mesh		28/150 mesh		+150 Concentrate		-150 mesh		P2O5
Capa	Metric Tons	% P2O5	% WSS	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	Recovery
PH03	5,264,531	17.52	5.23	6.59	31.12	40.64	30.33	47.23	30.44	47.54	6.62	82%
PH04	1,757,167	13.59	5.35	4.04	30.58	33.24	29.48	37.27	29.60	57.38	4.47	81%
PH06	6,044,550	13.56	4.26	1.29	29.45	27.54	28.88	28.82	28.90	66.92	7.82	61%
PH11	3,685,348	12.85	4.13	2.20	30.93	25.71	27.95	27.90	28.18	67.97	7.34	61%
PH12	5,027,668	13.93	2.98	2.80	31.78	27.71	27.86	30.50	28.22	66.52	8.00	62%
PH13	9,499,165	13.73	2.93	1.05	30.92	28.68	27.66	29.73	27.77	67.34	8.13	60%
Combined	31,278,429	14.26	3.86	2.61	30.98	30.22	28.65	32.83	28.83	63.31	7.56	66%

16.6 PIT AND PHASE DESIGN

The grid-based stratigraphic mining model described in Section 16.5 was converted to a sub-blocked model comprising of 40-m by 40-m by 1.6-m parent blocks and 5-m by 5-m by 0.2-m sub-blocks for pit optimisation purposes. The block model contained all information needed to assign a block value, including recovered waste rock volumes, ROM tonnages and % P_2O_5 by capa, and recovered product tonnages and % P_2O_5 by capa. The block model was exported to IMC for the development of the updated mine production schedule and Mineral Reserve estimate. The goal of the resource optimisation analysis was to determine the optimised Resources that will satisfy a 20-year mine production plan. The target annual production rate for the mine plan commences at 760Kt ramping up to 1.0 Mt (dry basis) of processed concentrate for a total of 20.7 Mt of DAPR for the planned life of mine. The optimised resource was defined as the phosphorite with the best product yield and lowest resultant strip ratio.

Optimisation was conducted on Measured and Indicated Resources only; Inferred Resources were not included in the mine plan. To prevent the optimised pits from encroaching on the Bayovar road a 180-m offset buffer zone was established from the Bayovar road.

The final pit was sub-divided into 13 mining phases which would be scheduled over the pre-production period and the 20 year mine production schedule. The pit design parameters presented in Table 16-14 are used in all the pit phase designs. All phase designs have independent haul roads for the removal of the mined materials to their destinations: plant, in-pit backfill, ex-pit dumps or the TSF embankment. The sequence of the phase designs progresses from south to north across the pit. This approach has shorter waste hauls for the early years when all the waste is placed in the ex-pit dumps to the south and west of the pit. The waste tonnage is significantly more than the ore tonnage with the waste to ore ratio being 7.19 / 1 dry tonnes.

The ore, product and waste tonnages are shown in Table 16-15 and the phase sequence is illustrated on Figure 16-6. The mining schedule progresses through the phases in the order they are numbered, but at any point in time, more than one phase will be mined (with the exception of pre-production and Year 20).

Table 16-14: Summary of Mine Design Parameters

Description	Value
Pit Wall inter-ramp slope angle	20 degrees above -30 elevation; 26 degrees below
Bench Height	5m, double benched to 10m
Bench face design angle	30 degrees above -30 elevation; 35 degrees below
Bench dig face angle	~65 degrees
Pit Haul roads	25m wide, 8% maximum grade
Minimum mineable thickness	30 cm
Mining roof dilution gain	7.5 cm
Mining floor dilution gain	7.5 cm
Mining Recovery	100%
Pit Buffer from Bayovar Road	180 m
Target Average Product Grade	Product A (24+% P2O5) and Product B (28+% P_2O_5)

[1]For a complete discussion of the geotechnical units in the Bayovar 12 Project, please refer to Golder's geotechnical report "Updated NI 43-101 Mineral Resource Technical Report on the Bayovar 12 Phosphate Project, Piura Region, Peru.

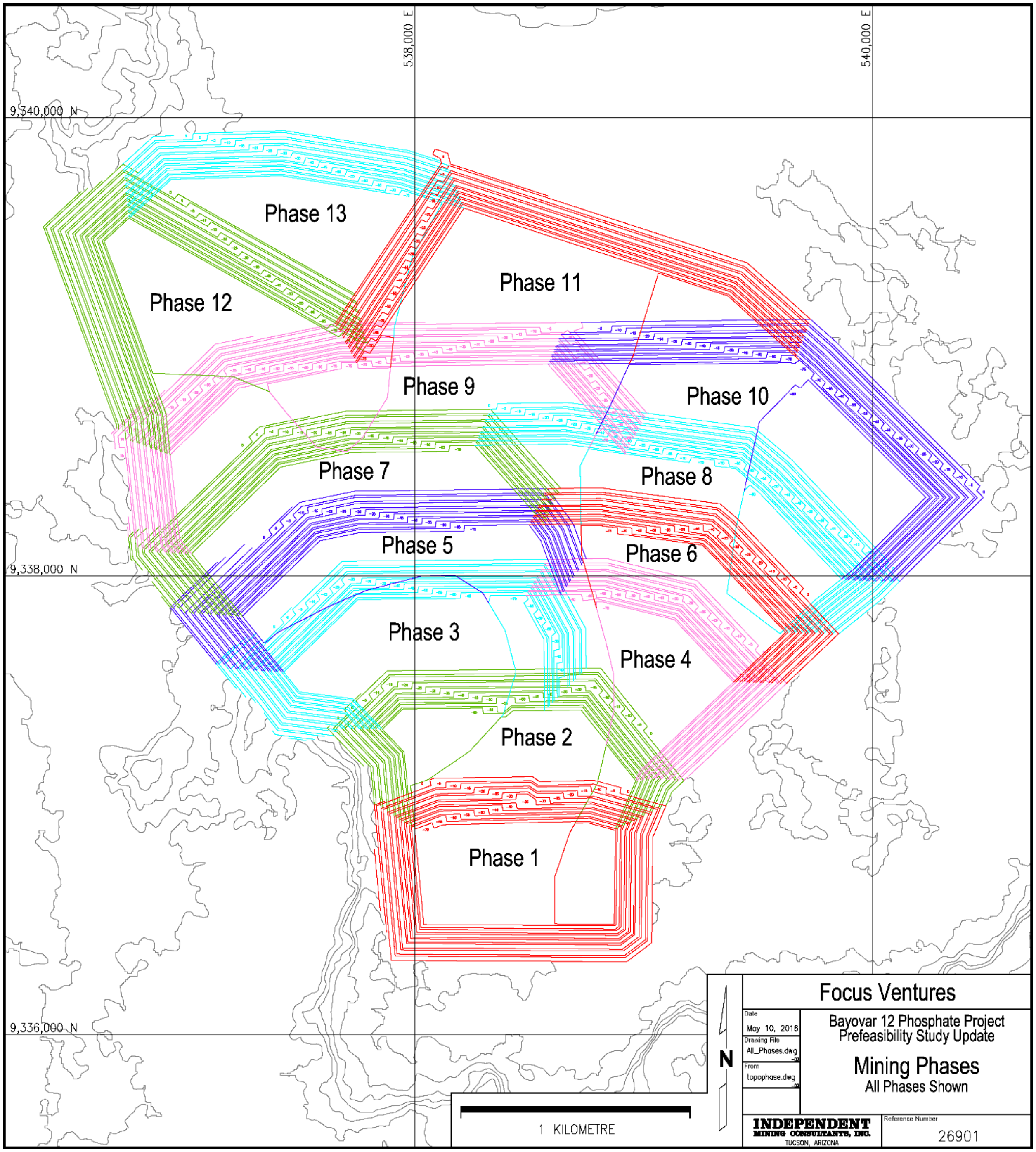


Figure 16-6: Mining Phase Layout

Table 16-15: Pit Phase Tonnages & Grades

	Dry Ore Tonnages				Dry Product Tonnages				P2O5 recovery		Total Product		Ore to Mill		Waste				
													Dry Tonnes		Ovbrdn	Interbrdn	Total Wst		
Mining	Product A		Product B		Product A		Product B				Dry Product							Mill tonne	Waste tonne
Phase	Capas 1,2,5,7,8,9,10		Capas 3,4,6,11,12,13		Capas 1,2,5,7,8,9,10		Capas 3,4,6,11,12,13								Dry	Dry	Dry	per tonne	per mill
	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	P_2O_5 %	Prod. A	Prod. B	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	tonnes	tonnes	product	tonne
1	1,502,959	10.61	1,622,235	13.89	545,470	23.90	508,454	28.39	81.75%	64.06%	1,053,924	26.07	3,125,194	12.31	19,675,563	14,788,267	34,463,830	2.97	11.03
2	1,545,135	10.85	2,160,969	13.75	578,117	23.79	675,630	28.45	82.04%	64.69%	1,253,747	26.30	3,706,104	12.54	12,423,739	14,807,218	27,230,957	2.96	7.35
3	1,919,190	11.22	2,753,626	13.51	724,895	23.95	862,972	28.58	80.63%	66.30%	1,587,867	26.47	4,672,816	12.57	13,297,456	15,919,500	29,216,956	2.94	6.25
4	1,626,225	10.88	1,598,443	14.94	590,062	24.19	541,289	28.41	80.67%	64.40%	1,131,351	26.21	3,224,668	12.89	12,446,605	12,114,076	24,560,681	2.85	7.62
5	1,718,859	11.28	2,069,324	13.78	645,929	24.05	653,880	28.46	80.12%	65.26%	1,299,809	26.27	3,788,183	12.65	12,284,480	13,328,691	25,613,171	2.91	6.76
6	1,380,704	11.14	1,416,057	14.99	510,864	24.23	481,267	28.40	80.48%	64.39%	992,131	26.25	2,796,761	13.09	10,793,864	10,291,693	21,085,557	2.82	7.54
7	1,901,024	11.52	2,380,746	13.65	740,635	23.94	745,099	28.46	80.96%	65.25%	1,485,734	26.21	4,281,770	12.70	14,090,190	16,031,762	30,121,952	2.88	7.03
8	2,359,239	11.29	2,503,644	14.72	873,581	24.28	847,636	28.47	79.63%	65.48%	1,721,217	26.34	4,862,883	13.06	16,376,423	15,722,444	32,098,867	2.83	6.60
9	3,124,812	11.55	3,603,442	13.98	1,203,006	24.08	1,172,795	28.57	80.26%	66.51%	2,375,801	26.30	6,728,254	12.85	19,294,894	22,042,765	41,337,659	2.83	6.14
10	3,118,750	11.52	3,298,145	14.74	1,189,870	24.22	1,099,322	28.36	80.21%	64.13%	2,289,192	26.21	6,416,895	13.18	22,461,142	21,309,102	43,770,244	2.80	6.82
11	3,203,988	11.81	3,300,503	14.61	1,256,060	24.14	1,104,976	28.45	80.13%	65.19%	2,361,036	26.16	6,504,491	13.23	17,453,730	23,852,570	41,306,300	2.75	6.35
12	2,012,879	11.50	2,188,918	14.40	794,762	23.75	721,811	28.46	81.54%	65.17%	1,516,573	25.99	4,201,797	13.01	22,612,486	17,202,658	39,815,144	2.77	9.48
13	1,710,492	12.01	2,751,632	14.59	712,940	23.67	914,324	28.39	82.15%	64.66%	1,627,264	26.32	4,462,124	13.60	12,921,439	18,997,833	31,919,272	2.74	7.15
Total	27,124,256	11.38	31,647,684	14.26	10,366,191	24.03	10,329,455	28.46	80.68%	65.13%	20,695,646	26.24	58,771,940	12.93	206,132,011	216,408,579	422,540,590	2.84	7.19

16.7 MINE PRODUCTION SCHEDULE AND MINING SEQUENCE

The mine production schedule targets the production of 760,000 tonnes of combined Products A and B during Year 1 and approximately 1 million tonnes of combined Products A and B each year for the remaining pit life through Year 20. The tonnage of product and the split between Products A and B vary, but between mining and the plant stockpiles there will be sufficient flexibility to maintain the production schedule. The tonnages of product, ROM ore and waste are all shown in dry tonnes. Table 16-1; the summary of the mine schedule is repeated as Table 16-16 .

For the Updated Pre-Feasibility Study, the pit and phase designs have horizontal 5m high benches in the phosphorite capas and interbeds, even though the beds dip 1 to 2 degrees to the east – northeast. The tabulation of the tonnages and grades are summarized by capas on the 5m benches for the production schedule. The mining of the capas and some of the interbeds will be mined with GPS controlled selective surface miners which can closely follow the dip of the beds. The annual schedule shown in Table 16-17 does not try to target the exact 1 million dry tonnes of product each year, but comes close based on the percent of a 5m bench that is mined at the end of each mining period and over the 20 year mine schedule produces slightly more product than targeted. For the Feasibility Study, the mine and plant schedules will be refined to reflect the product production tonnages more closely. For mine development planning, the short range mining plan and schedule will incorporate the dip of the beds in the daily, weekly and monthly production schedules. As well, for the short range plans, the pit phases will be sub-divided with one area mining ore and an adjacent area mining the interburden waste being removed to expose the next ore capa.

The mine schedule begins in Phase 1 at the south end of the pit during pre-production and the overburden, interburden above Capa 1 and the majority of the Capa 1 are mined from Phase 1. The 137,189 tonnes of ore are stockpiled at the plant for processing in Year 1. During Year 1, ore is mined from Phase 1 and the overburden stripping and mining in the upper capas is being done in Phase 2. During Year 2, mining is completed in Phase 1 (down to capa 13), the middle and lower capas are mined in Phase 2 and the overburden stripping has begun in Phase 3. This progression through the mining phases continues for the remaining years, with three phases being mined in most years. Table 16-17 shows the active mining phases in each year (the term 'yes' in the table indicates that the overburden is being mined in the phase during the particular year). Figures 16-6 through 16-15 illustrate the mining schedule at the end of pre-production and years -1, 1, 2, 3, 4, 5, 7, 10, 15 and 20.

The production schedule has both Products A and B being produced in each year in varying amounts. The products are related to the capas being mined per Table 16-17. In some years, the capas being mined for Product A (24% P_2O_5) do not produce the target +24% grade based on the test work to date (for example, Capa 1 produces a 23.85% product and Capa 2 a 22.50%). Blending of the plant products or adjustments to the hydro-sizer in the plant will be done to achieve the target product grade. Additional test work is planned prior to the Feasibility Study to develop a more detailed grade – recovery curve by capa.. On a 12 month basis, it is shown in Table 16-18 that this can be achieved, but it may not always happen on a weekly or monthly basis due to mining locations and the mining sequence through the phases. In these cases, some blending of product will be required at the plant depending on the product sales and shipping schedules.

An example of a monthly schedule was done for Year 5 and is summarized in Table 16-18. During Year 5, Phases 3, 4 and 5 are being mined, with Phase 3 in the lower capas, Phase 4 in capas 2 through 13 and Phase 5 in overburden stripping for the majority of the year and ore from Capa 1 at the end of the year. In this example, the ore and waste materials are in BCM's and the ore and waste rates are uniform each month and the tonnage of product varies depending on the yields from the various capas. In reality, the mining rates will be adjusted based on product orders and blending requirements.

In the three phases mined during Year 5, each one is tabulated on 5m benches and a bench is completed for the entire phase before progressing to the next bench below. During months 1 through 4, the +24% P_2O_5 specification for Product A is not met because the source for Product A is Capa 2 which has a 22.50% P_2O_5 product yield. In the later months, Product B does not always meet its 28% specification because its source is Capa 13 during these months which has an average yield of 27.81% P_2O_5. It is envisioned that by allowing the ore rate to the plant to fluctuate and adjustments to the yield cut in the plant based on what Capa is providing the ore, the product specifications can be met in shorter time frames than annually.

Table 16-16: Mine Production Schedule

Annual	Product 24		Product 28		Total Product		Ore to Plant		Waste				
							Dry Tonnes		Ovbrdn	Interbrdn	Total Wst		
	Dry Product		Dry Product		Dry Product							Mill tonne	Waste tonne
Year									Dry	Dry	Dry	per tonne	per mill
	tonnes	P_2O_5 %	tonnes	P_2O_5 %	tonnes	P_2O_5%	tonnes	P_2O_5%	tonnes	tonnes	tonnes	product	tonne
PP	137,189	23.85			137,189	23.85	363,875	10.91	19,694,417	4,280,198	23,974,614	2.65	65.89
1	423,430	22.90	199,722	28.92	623,152	24.83	1,625,413	12.1	12,419,658	10,340,074	22,759,732	2.41	14
2	414,805	24.09	608,043	28.51	1,022,848	26.72	3,084,801	12.64	10,842,823	9,923,638	20,766,461	2.95	6.73
3	594,446	23.71	529,539	28.4	1,123,985	25.92	3,231,207	12.42	8,111,781	12,254,579	20,366,361	2.86	6.3
4	413,059	24.77	545,303	28.58	958,362	26.94	2,940,239	12.6	8,072,246	12,217,219	20,289,464	3.05	6.9
5	503,264	24.38	503,937	28.43	1,007,201	26.41	2,941,191	12.75	10,972,880	9,591,991	20,564,871	2.92	6.99
6	425,824	23.30	525,219	28.58	951,043	26.22	2,639,081	13.09	13,004,702	8,099,318	21,104,020	2.7	8
7	500,046	24.23	520,133	28.32	1,020,179	26.32	2,984,816	12.82	10,821,599	10,545,758	21,367,358	2.93	7.16
8	681,517	23.75	533,293	28.55	1,214,810	25.86	3,221,007	13.12	7,690,390	12,545,176	20,235,566	2.64	6.28
9	345,906	25.29	503,403	28.13	849,309	26.97	2,801,886	12.42	9,703,071	10,353,034	20,056,105	3.3	7.16
10	609,698	23.92	420,170	29.00	1,029,868	25.99	2,728,678	12.93	11,303,971	8,802,827	20,106,798	2.65	7.37
11	605,687	24.03	491,520	28.11	1,097,207	25.86	3,052,108	13.12	8,010,245	12,107,596	20,117,842	2.78	6.59
12	543,078	23.94	569,830	29.11	1,112,908	26.59	3,058,409	12.89	13,402,273	8,092,090	21,494,363	2.74	7.03
13	499,468	24.65	507,993	27.95	1,007,461	26.31	3,034,562	12.81	9,087,878	11,611,066	20,698,943	3.01	6.82
14	574,110	23.52	493,766	28.83	1,067,876	25.98	2,846,790	13.08	10,554,953	9,744,350	20,299,303	2.65	7.13
15	594,771	25.00	495,618	27.99	1,090,389	26.36	3,223,875	12.96	6,917,891	13,353,914	20,271,804	2.96	6.29
16	543,676	22.83	645,086	28.75	1,188,762	26.04	3,023,984	13.75	15,536,870	7,474,622	23,011,492	2.44	7.61
17	473,187	25.79	502,742	28.02	975,929	26.94	3,043,397	13	10,416,509	11,112,207	21,528,716	3.12	7.07
18	546,495	22.93	422,440	28.77	968,935	25.48	2,416,969	13.4	9,544,279	8,820,262	18,364,540	2.46	7.6
19	627,779	24.03	509,659	28.23	1,137,438	25.91	3,178,896	13	23,575	13,922,420	13,945,996	2.79	4.39
20	308,758	24.44	802,038	28.29	1,110,796	27.22	3,330,757	13.65	0	11,216,241	11,216,241	2.99	3.37
Total	10,366,193	24.03	10,329,454	28.46	20,695,647	26.24	58,771,941	12.93	206,132,011	216,408,579	422,540,589	2.84	7.19

Table 16-17: Active Mining Phases by Year

Phase		Pre-Prod.	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
1	Overburden Stripping	Yes																				
	Capas Mined	1	1 - 6	6 - 13																		
2	Overburden Stripping		Yes																			
	Capas Mined		1 - 2	2 - 8	6 - 13																	
3	Overburden Stripping			Yes	Yes																	
	Capas Mined				1 - 4	2 - 13	11 - 13															
4	Overburden Stripping				Yes	Yes																
	Capas Mined					1 - 3	2 - 13	11 - 13														
5	Overburden Stripping					Yes	Yes															
	Capas Mined						1	1 - 8	7 - 13													
6	Overburden Stripping							Yes	Yes													
	Capas Mined								1 - 8	3 - 13												
7	Overburden Stripping							Yes	Yes	Yes												
	Capas Mined									1 - 6	6 - 13											
8	Overburden Stripping									Yes	Yes											
	Capas Mined										1	1 - 10	7 - 13									
9	Overburden Stripping											Yes	Yes									
	Capas Mined												1 - 4	2 - 10	8 - 13	12 - 13						
10	Overburden Stripping													Yes	Yes							
	Capas Mined														1 - 3	1 - 10	7 - 13	12 - 13				
11	Overburden Stripping															Yes	Yes					
	Capas Mined																1	1 - 7	6 - 13	12 - 13		
12	Overburden Stripping																	Yes	Yes			
	Capas Mined																		1	1 - 6	6 - 13	
13	Overburden Stripping																		Yes	Yes	Yes	
	Capas Mined																				1 - 4	2 - 13

Table 16-18: Year 5 – Example Monthly Schedule

Month	Ore	Product B - 28%		Product A - 24%		Total Product		Waste Mining - bcm			Total
	bcm	Tonnes	% p2o5	Tonnes	% p2o5	Tonnes	% p2o5	Overbrdn	Interbrdn	Total	bcm
1	214,475	33,878	28.06	61,324	22.50	95,202	24.48	1,035,193	1,235,907	2,271,100	2,485,575
2	214,475	34,082	28.03	61,324	22.50	95,406	24.48	2,051,100	220,000	2,271,100	2,485,575
3	214,475	77,895	28.78	25,792	22.50	103,687	27.22	1,836,317	434,783	2,271,100	2,485,575
4	214,475	75,993	29.05	37,919	22.50	113,912	26.87	1,554,229	716,871	2,271,100	2,485,575
5	214,475	51,924	28.55	25,141	25.70	77,065	27.62	823,542	1,447,558	2,271,100	2,485,575
6	214,475	63,124	28.69	28,937	25.54	92,061	27.70	1,285,533	985,567	2,271,100	2,485,575
7	214,475	7,440	27.81	42,866	25.62	50,306	25.94	1,457,794	813,306	2,271,100	2,485,575
8	214,475	35,524	27.88	32,964	25.57	68,488	26.77	1,626,501	644,599	2,271,100	2,485,575
9	214,475	16,492	27.81	62,727	25.93	79,219	26.32	387,901	1,883,199	2,271,100	2,485,575
10	214,475	13,011	27.81	62,463	25.95	75,474	26.27	0	2,271,100	2,271,100	2,485,575
11	214,475	51,524	28.18	17,578	25.59	69,102	27.52	0	1,622,155	1,622,155	1,836,630
12	214,461	43,049	27.98	44,229	23.86	87,278	25.89	0	345,995	345,995	560,456
Total	2,573,686	503,936	28.42	503,264	24.38	1,007,200	26.40	12,058,110	12,621,040	24,679,150	27,252,836

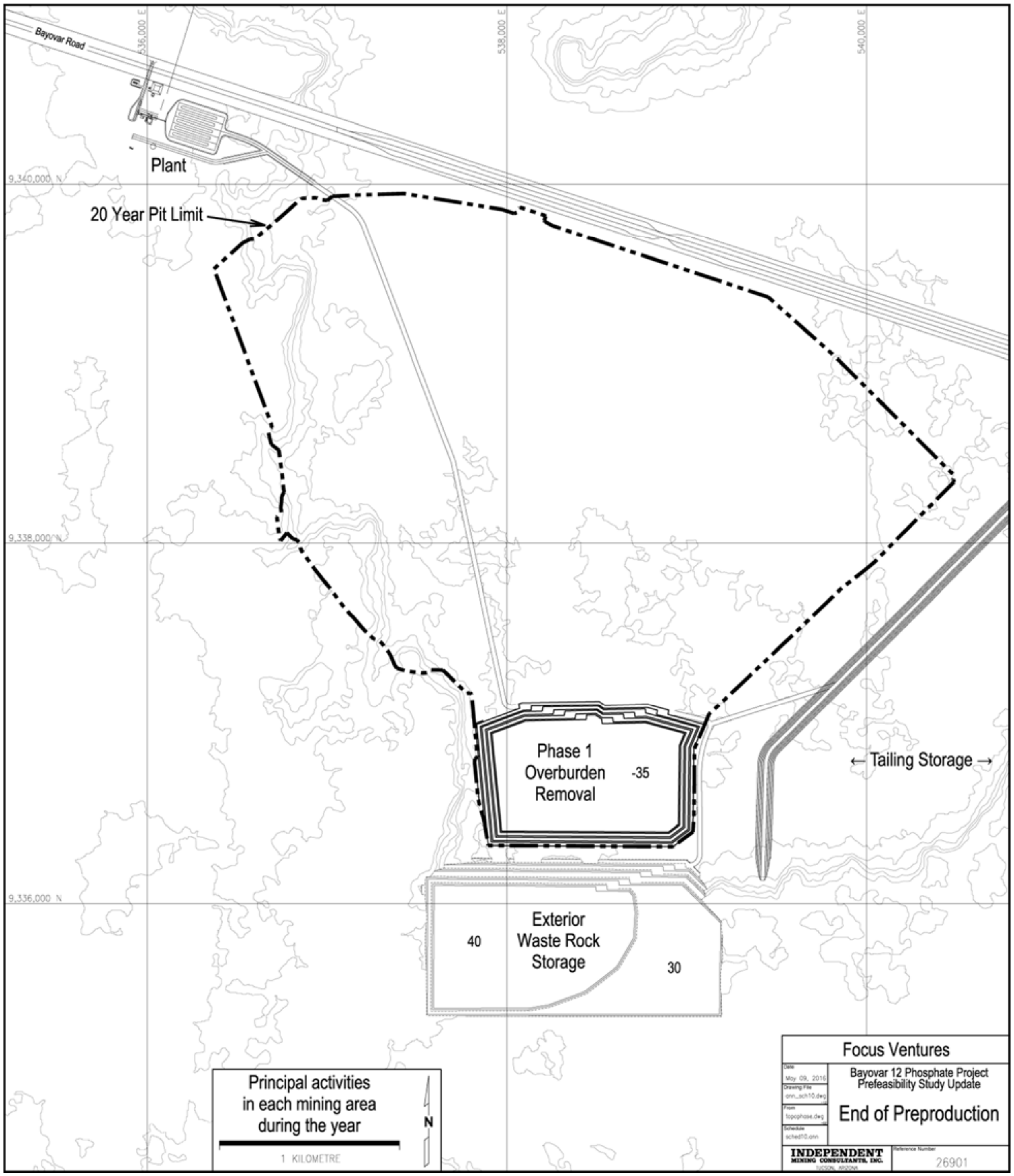


Figure 16-7: Mine Plan at end of Pre-Production

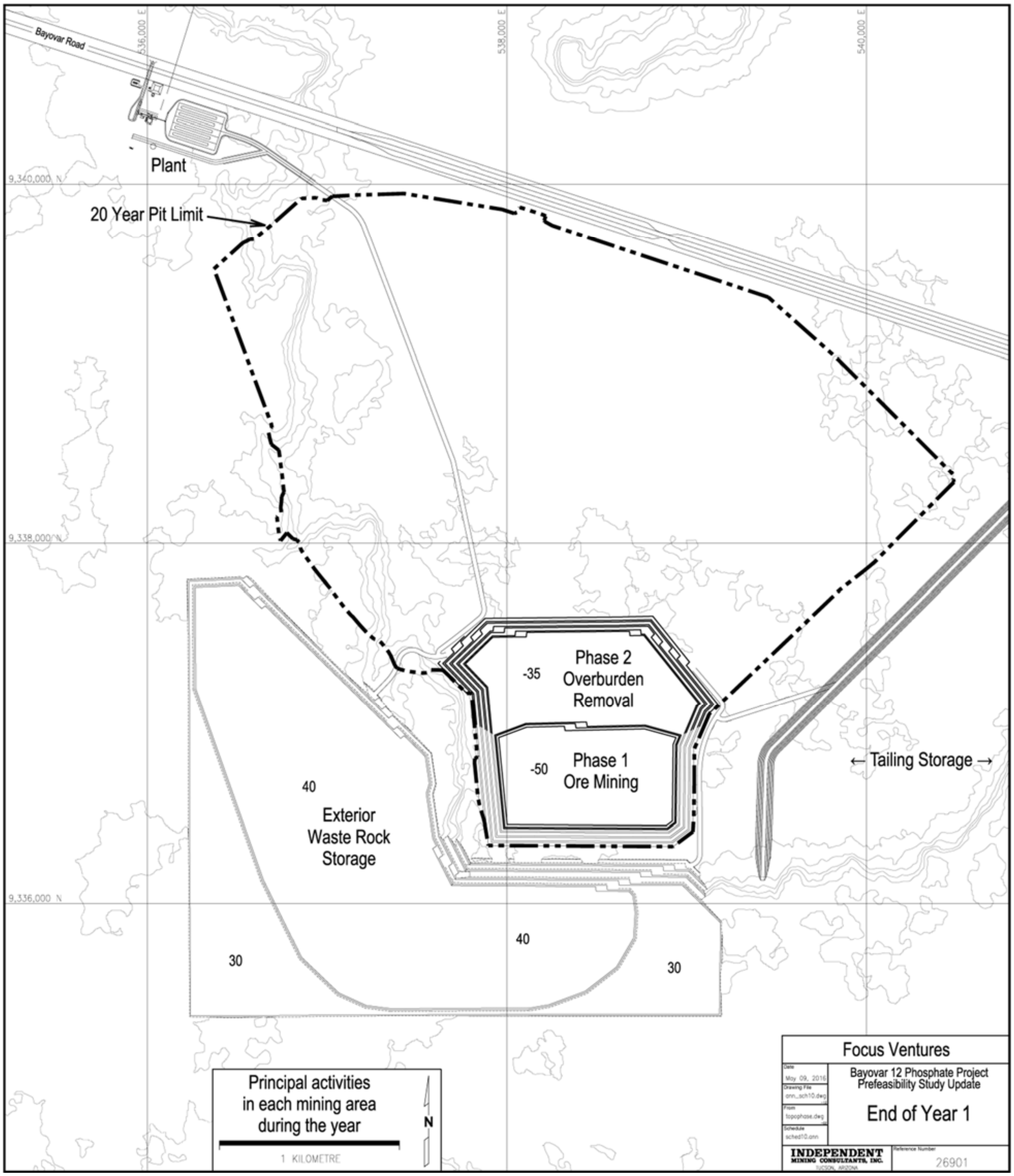


Figure 16-8: Mine Plan at end of Year 1

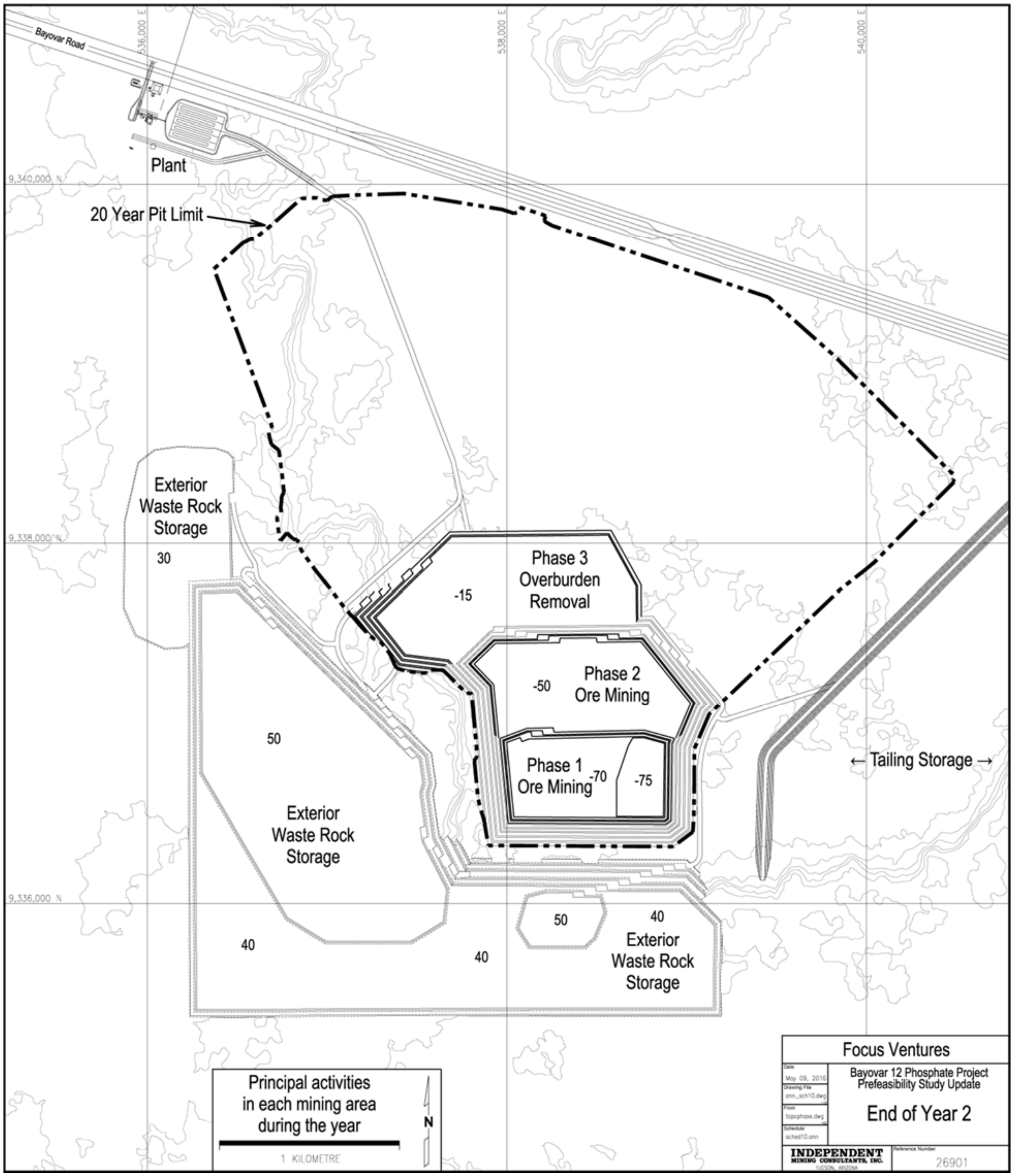


Figure 16-9: Mine Plan at end of Year 2

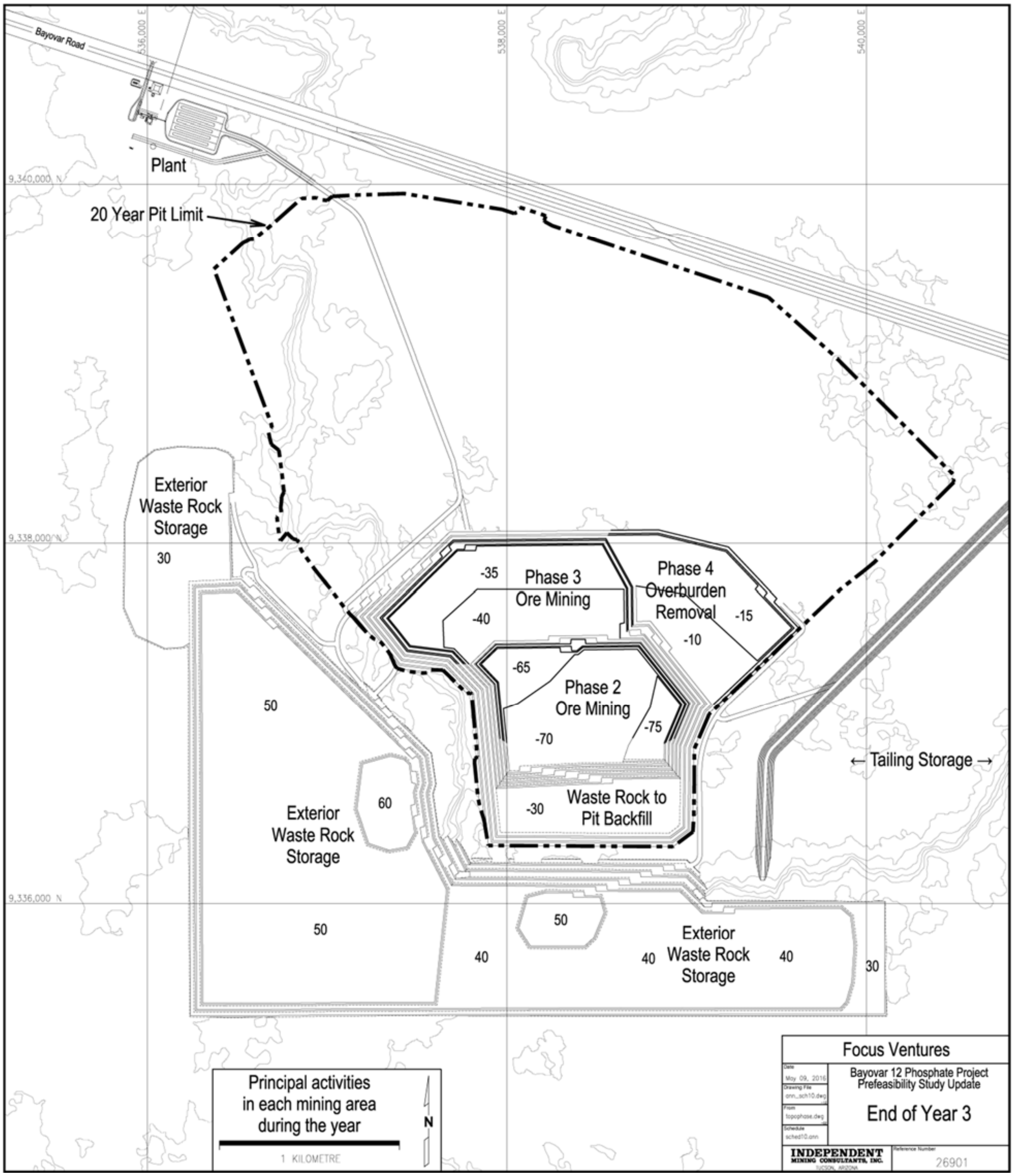


Figure 16-10: Mine Plan at end of Year 3

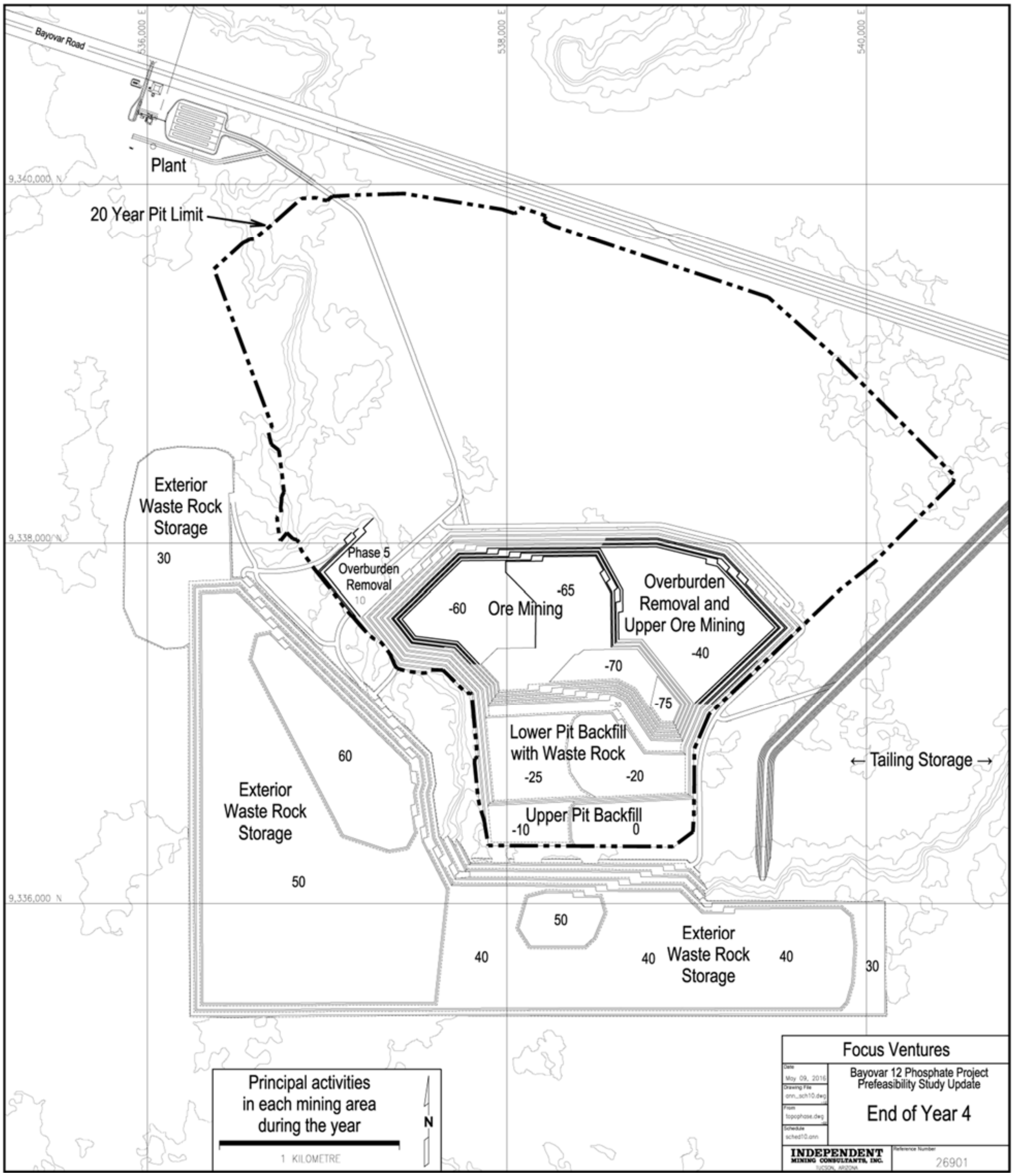


Figure 16-11: Mine Plan at end of Year 4

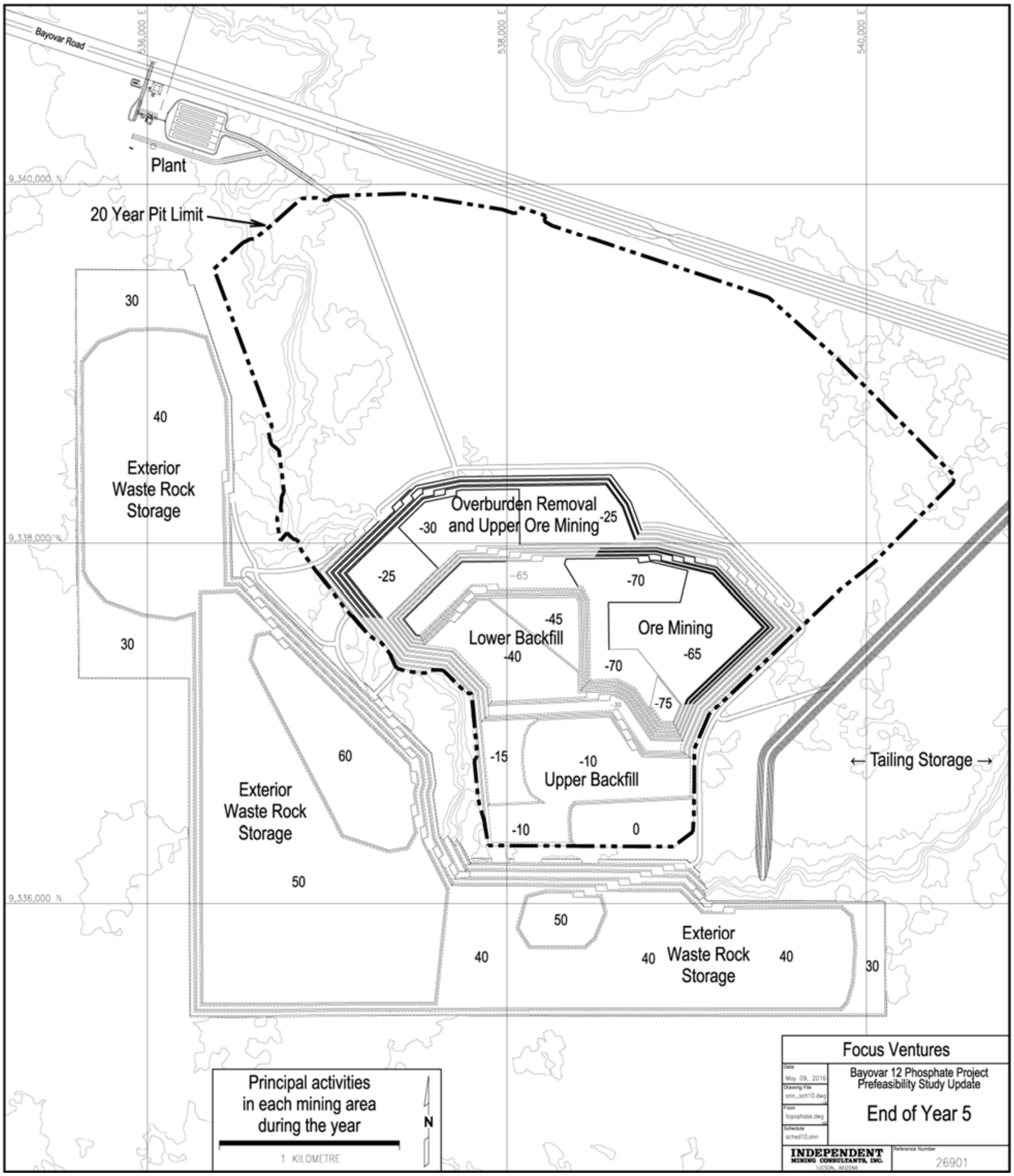


Figure 16-12: Mine Plan at end of Year 5

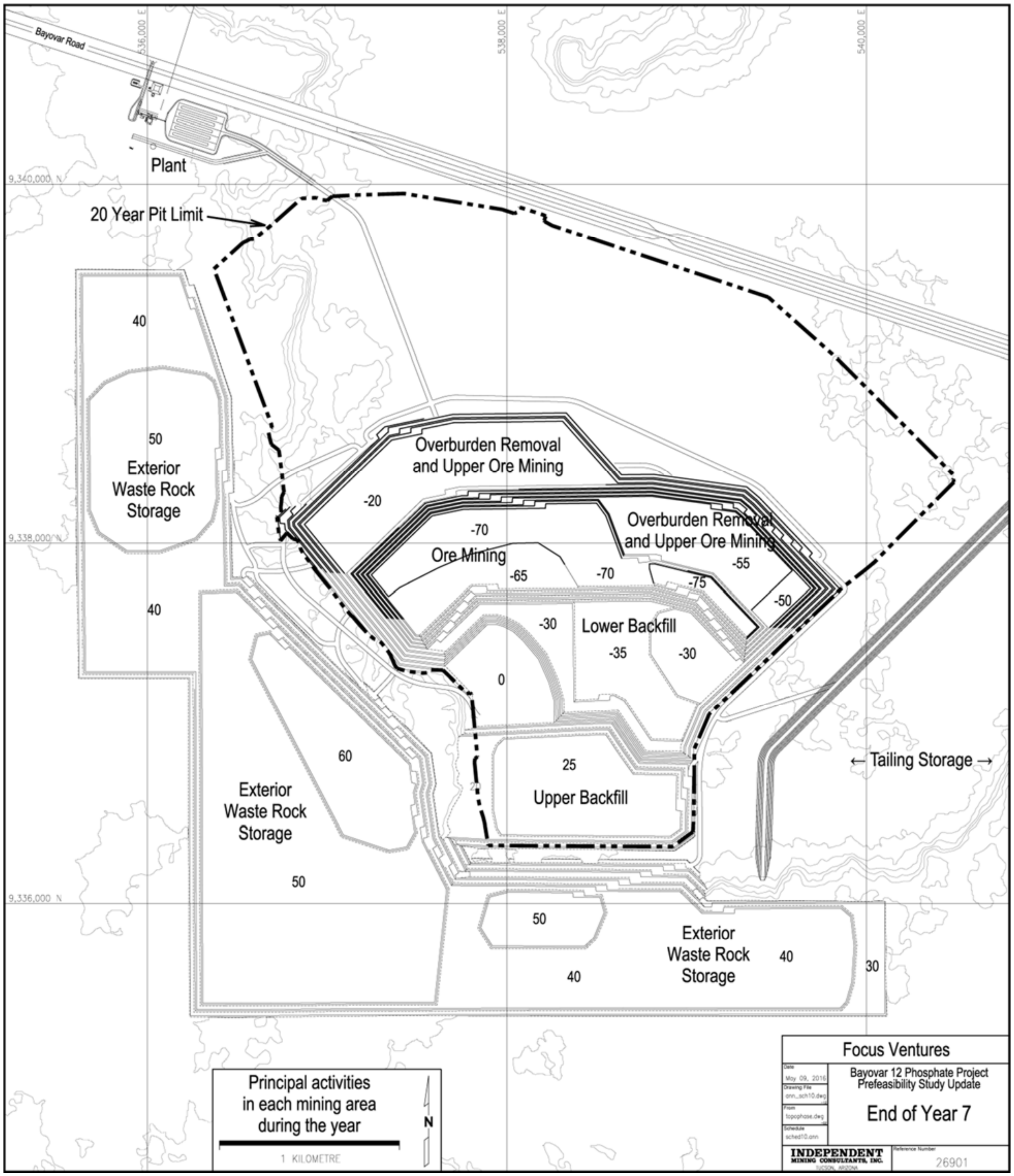


Figure 16-13: Mine Plan at end of Year 7

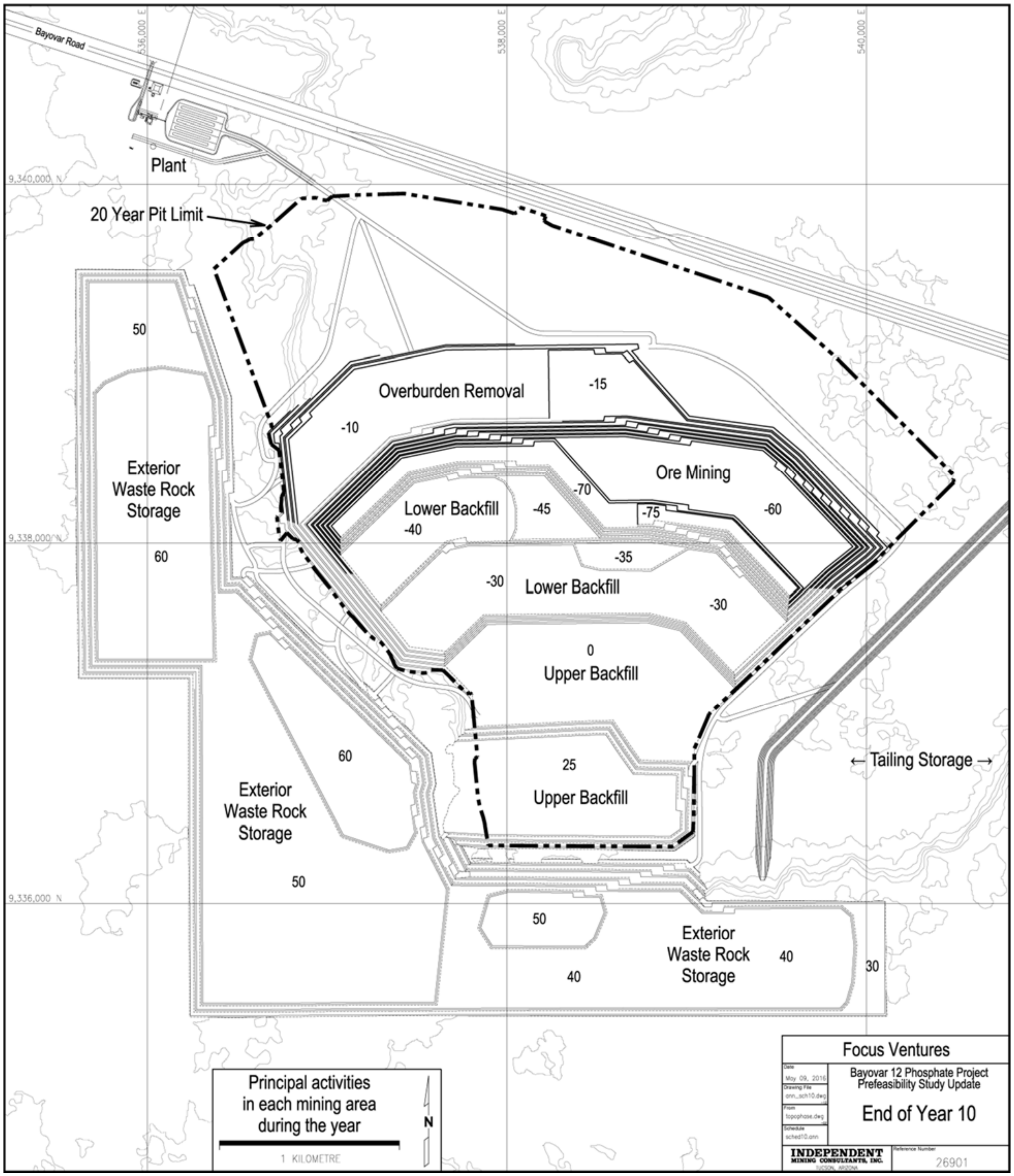


Figure 16-14: Mine Plan at end of Year 10

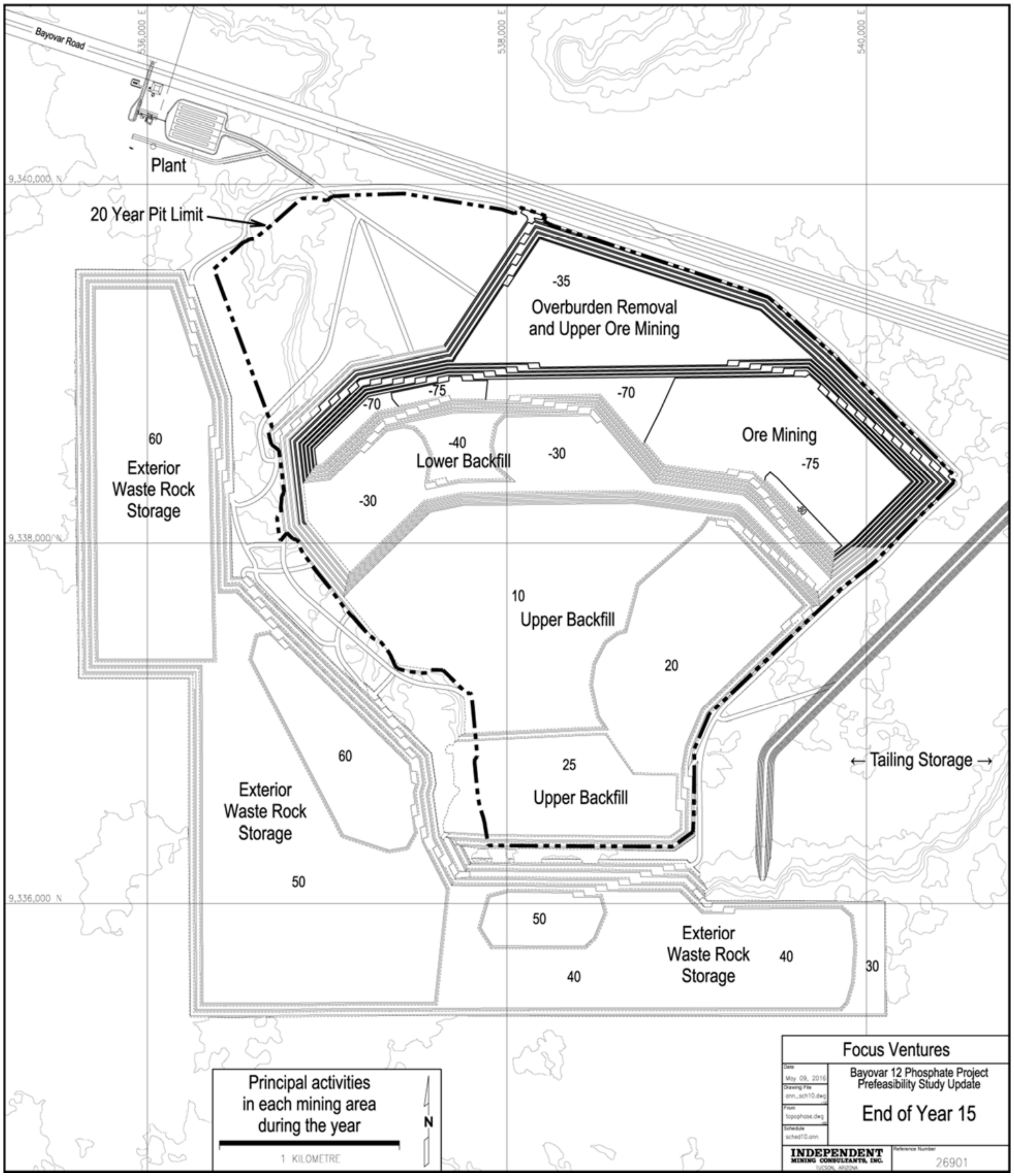


Figure 16-15: Mine Plan at end of Year 15

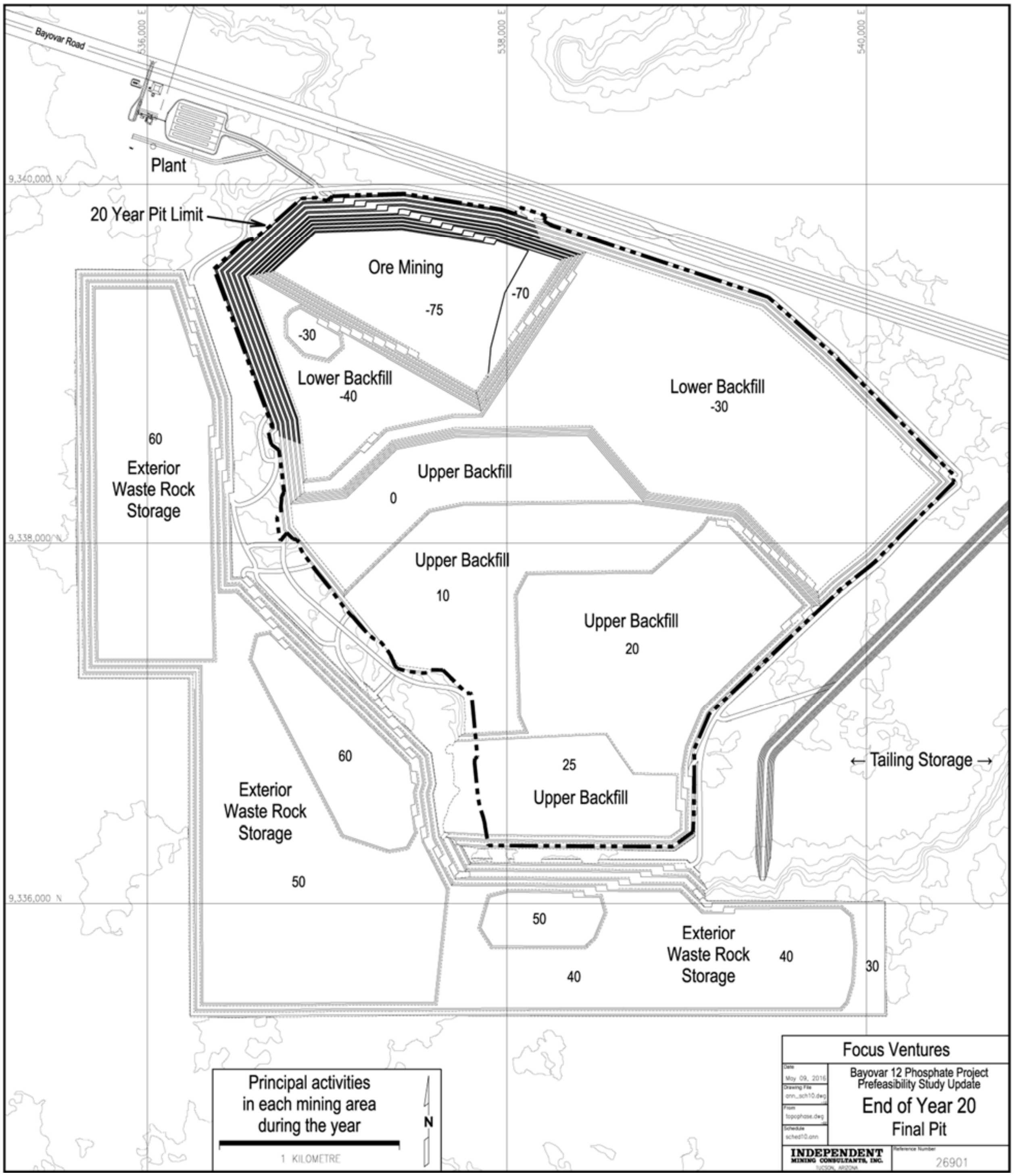


Figure 16-16: Mine Plan at end of Year 20

16.7.1 Waste Rock Storage Facilities

The mine waste (overburden and interburden) will be stored or used in three areas: in ex-pit dumps to the south and west of the pit, as backfill in the mined-out pit areas and for the construction of TSF embankment and berms (haulage costs of which are included in mining costs). Controls around the pit for water runoff during El Niño events are not allocated at this time in the waste placement schedule and would be re-allocated from the ex-pit dumps. All of the waste is placed outside of the pit through the end of Year 2, at which time mining Phase 1 is completed. Pit backfilling with waste begins during Year 3 with 75m offset from the active mining areas. The pit backfill is constructed in three phases: lower backfill (up to the -30m elevation), upper backfill (from -30m to pit rim elevation) with a 50m setback from the crest of the lower backfill and waste stacked within the pit limit above the upper backfill. The volume of all waste dumps is based on a 20% swell from bcm to in-dump lcm. The design criteria for the various dumps are presented in Table 16-19.

The waste storage schedule is shown on Table 16-20 in bcm with the conversion from dry tonnes to bcm being 1.32 (bcm/dmt) for interburden and 1.10 for overburden. Once the in pit backfill volume becomes available during Year 3, the majority of the waste is put back into the pit during Years 3 through 11. After Year 11, all of the waste is placed either in the pit as backfill or within the pit outline as stacked waste above the backfill.

Table 16-19: Waste Dump Design Parameters

Dump	Face Angle	Overall Slope Angle	Placement lift height	Catch bench set back
Lower Backfill (pit floor to -30m elevation)	1.8 : 1.0 (29 degrees)	2.5 : 1.0 (22 degrees)	10 metres	7 metres
Upper Backfill (-30 elevation to pit rim elevation)	1.8 : 1.0 (29 degrees)	2.5 : 1.0 (22 degrees)	30 metres (crest dumped)	50 metres
Stack Dump above backfill	1.6 : 1.0 (32 degrees)	3.5 : 1.0 (16 degrees)	10 metres	19 metres
ExPit Dumps	1.6 : 1.0 (32 degrees)	3.5 : 1.0 (16 degrees)	10 metres	19 metres

Mine waste will be used to construct the TSF containment embankment in two phases. The Phase 1 TSF embankment will be constructed as part of upfront capital expenditures from pre-stripping and will require approximately 2.6 million bcm of overburden placed in engineered lifts to 95% compaction. The Phase 2 embankment is constructed during Years 5 and 6 and will require 3.40 million bcm. The material for the Phase 2 will be delivered to the Phase 2 embankment area late in year 4 and again in early year 6; this smooths the operating truck requirements. The engineered fill will need to be placed at near optimum moisture contents. Overburden stripped from above the water table will require addition of water to reach optimum moisture and material stripped from below the water table may need to be spread and dried to reach optimum moisture. The updated PFS tailings design is smaller than the original design and during a wet year, there may be some release of decant water from the facility. This will be further addressed during the Feasibility Study as the tailings volumes may adjust with further design enhancements in the plant, notably making use of re-cycled processed water.

Table 16-20: Waste Rock Storage Facilities Schedule

Annual	MINING SCHEDULE						DUMP SCHEDULE												
	Waste			Waste			Inpit Backfill							Expit Requirements					Inpit/Expit
	Ovbrdn	Interbrdn	Total Wst	Ovbrdn	Interbrdn	Total Wst	To Backfill							To Dumps		To Tails	Total		Combined
							Lower		Upper		Stacker		Combined	Overburden	Interburden	Overburden	Combined		Total
Year	Dry	Dry	Dry				Ovbrdn	Interbrdn	Ovbrdn	Interbrdn	Ovbrdn	Interbrdn							
	tonnes	tonnes	tonnes	BCM	BCM	BCM	BCM	BCM	BCM	BCM	BCM	BCM	BCM	BCM	BCM	BCM	BCM		BCM
PP	19,694,417	4,280,198	23,974,614	21,642,216	5,631,839	27,274,055	-	-	-	-	-	-	-	19,027,216	5,631,839	2,615,000	27,274,055		27,274,055
1	12,419,658	10,340,074	22,759,732	13,647,976	13,605,360	27,253,336	-	-	-	-	-	-	-	13,647,976	13,605,360	-	27,253,336		27,253,336
2	10,842,823	9,923,638	20,766,461	11,915,190	13,057,418	24,972,608	-	-	-	-	-	-	-	11,915,190	13,057,418	-	24,972,608		24,972,608
3	8,111,781	12,254,579	20,366,361	8,914,046	16,124,447	25,038,492	-	9,621,000	-	-	-	-	9,621,000	8,914,046	6,503,447	-	15,417,492		25,038,492
4	8,072,246	12,217,219	20,289,464	8,870,600	16,075,288	24,945,888	-	14,210,000	4,094,833	1,865,288	-	-	21,847,121	1,421,767	-	1,667,000	4,775,767		24,945,888
5	10,972,880	9,591,991	20,564,871	12,058,110	12,621,040	24,679,150	-	7,423,708	-	4,647,885	-	-	12,071,593	12,058,110	549,448	-	12,607,558		24,679,150
6	13,004,702	8,099,318	21,104,020	14,290,882	10,656,997	24,947,879	-	4,048,639	2,141,167	4,508,827	7,048,094	2,099,531	18,169,258	5,101,621	-	1,677,000	5,101,621		24,947,879
7	10,821,599	10,545,758	21,367,358	11,891,868	13,875,998	25,767,865	-	13,875,998	5,231,345	-	1,429,178	-	20,536,520	5,231,345	-	-	5,231,345		25,767,865
8	7,690,390	12,545,176	20,235,566	8,450,978	16,506,810	24,957,788	-	15,326,655	3,576,468	-	3,753,197	-	22,656,320	1,121,313	1,180,155	-	2,301,468		24,957,788
9	9,703,071	10,353,034	20,056,105	10,662,715	13,622,413	24,285,128	-	13,622,413	10,662,715	-	-	-	24,285,128	-	-	-	-		24,285,128
10	11,303,971	8,802,827	20,106,798	12,421,947	11,582,667	24,004,614	-	11,582,667	-	-	-	-	11,582,667	12,421,947	-	-	12,421,947		24,004,614
11	8,010,245	12,107,596	20,117,842	8,802,468	15,931,048	24,733,516	-	4,792,920	5,979,339	11,138,128	-	-	21,910,387	2,823,129	-	-	2,823,129		24,733,516
12	13,402,273	8,092,090	21,494,363	14,727,772	10,647,487	25,375,260	-	10,647,487	7,561,910	-	7,165,862	-	25,375,260	-	-	-	-		25,375,260
13	9,087,878	11,611,066	20,698,943	9,986,679	15,277,718	25,264,397	-	9,190,513	-	6,087,205	9,986,679	-	25,264,397	-	-	-	-		25,264,397
14	10,554,953	9,744,350	20,299,303	11,598,850	12,821,513	24,420,362	-	5,870,718	2,585,623	6,950,795	9,013,227	-	24,420,362	-	-	-	-		24,420,362
15	6,917,891	13,353,914	20,271,804	7,602,078	17,570,939	25,173,017	-	17,570,939	-	-	7,602,078	-	25,173,017	-	-	-	-		25,173,017
16	15,536,870	7,474,622	23,011,492	17,073,484	9,835,029	26,908,513	-	9,835,029	15,073,484	-	2,000,000	-	26,908,513	-	-	-	-		26,908,513
17	10,416,509	11,112,207	21,528,716	11,446,713	14,621,325	26,068,039	9,383,197	14,621,325	2,063,517	-	-	-	26,068,039	-	-	-	-		26,068,039
18	9,544,279	8,820,262	18,364,540	10,488,218	11,605,608	22,093,826	10,488,218	11,605,608	-	-	-	-	22,093,826	-	-	-	-		22,093,826
19	23,575	13,922,420	13,945,996	25,907	18,318,974	18,344,881	25,907	15,879,059	-	2,439,915	-	-	18,344,881	-	-	-	-		18,344,881
20	0	11,216,241	11,216,241	0	14,758,212	14,758,212	-	14,758,212	-	-	-	-	14,758,212	-	-	-	-		14,758,212
Total	206,132,011	216,408,579	422,540,589	226,518,693	284,748,130	511,266,823	19,897,322	204,482,890	58,970,399	37,638,043	47,998,314	2,099,531	371,086,499	93,683,658	40,527,666	5,969,000	140,180,324		511,266,823

16.8 MINE EQUIPMENT REQUIREMENTS

The equipment selection for the project is dependent on a variety of factors including annual material movement requirements, bench height, pit configuration, number of mining faces and the required selectivity of the mining equipment in waste rock and phosphorite. Based on these factors, 1,000 tph continuous surface miners were selected as the primary loading fleet for phosphorite. These machines are large enough to produce the annual tonnages required and are able to efficiently load the 90-tonne (110 cubic metre bed) class of trucks selected to transport phosphorite to the plant or ore stockpile.

Overburden stripping and interburden waste will be mined with a 31 cubic metre front end loader with thinner interburden zones mined with the surface miner. The large FELs are used to efficiently expose phosphate concentrate leaving a temporary face angle of approximately 65˚. Several 405-horsepower (hp) dozers are used to prepare the working surface and to create access to the work area, and provide support for the excavators at mining faces. Waste rock haulage is accomplished with a fleet of 90-tonne capacity end-dump trucks. The 31 m^3 FEL can load these 90-tonne trucks with overburden in three passes and four passes for the inter-burden. The exposed phosphorite is mined with surface miners, and is hauled directly to the plant hopper or to a ROM stockpile using 90-tonne capacity end-dump trucks. The surface miners can continuously load the trucks.

The mine is scheduled to operate 720 twelve-hour shifts per year using four crews. Table 16-21 illustrates the available operating minutes per shift. Availability and utilization factors, as shown in Table 16-22 were applied to calculate number of units required for the major mining equipment. Table 16-23 is a summary of the major mining equipment utilized in the initial fleet during pre-production and the maximum number of units.

The reference to specific manufacturers' equipment is to illustrate the size and type of mining equipment and is not intended to be a recommendation by IMC of a specific manufacturer.

Table 16-21: Operating Minutes Per Shift

Summary of Operating Time Per Shift	
Scheduled Time Per Shift (min)	720
Less Scheduled Nonproductive Times	
Travel Time/Shift Change/Blasting (min)	10
Equipment Inspection (min)	10
Lunch/Breaks	30
Fueling, Lube, & Service (min)	10
Net Scheduled Productive Time (Metered Operating Time) (min)	660
Job Efficiency (50 Minutes Productive Time Per Metered Hour)	83.3%
Net Productive Operating Time Per Shift (min)	550

Table 16-22: Equipment Availability and Utilitzation

Equipment Type	Mechanical Availability	Utilization of Availability	Maximum Utilization	Manpower Based on
WIRTGEN 2500SM Continuous Miner (2.5 M)	0.90	0.80	0.720	# of Units
CAT 994 Front End Loader (31 CuM)	0.90	0.90	0.810	# of Units
CAT 777 Coal Haul Truck (90 tn)	0.90	0.90	0.810	Utilization
CAT D9/D10 Track Dozers	0.90	0.75	0.675	Utilization
CAT 834 Wheel Dozer (450 HP)	0.90	0.75	0.675	Utilization
CAT 16M Motor Graders (297 HP)	0.90	0.75	0.675	# of Units
CAT 770-W Water Truck (30,000 Ltr)	0.90	0.75	0.675	Utilization
CAT 336 Aux Loader (1 CuM)	0.90	0.75	0.675	Utilization
CAT CS-56 Compactor (147 HP)	0.90	0.95	0.855	Utilization

Table 16-23: Summary of Major Mining Equipment Units

Equipment	Initial Purchase for Pre-Production	Maximum Units in Fleet
WIRTGEN 2500SM Continuous Miner (2.5 M)	1	2
CAT 994 Front End Loader (31 CuM)	4	4
CAT 777 Coal Haul Truck (90 tn)	34	39
CAT D9/D10 Track Dozers	2	4
CAT 834 Wheel Dozer (450 HP)	1	1
CAT 16M Motor Graders (297 HP)	2	3
CAT 770-W Water Truck (30,000 Ltr)	1	2
CAT 336 Aux Loader (1 CuM)	1	1
CAT CS-56 Compactor (147 HP)	1	1

The ore and waste haulage represents the number of units in the mining fleet, 53% of the initial mine equipment capital and 56% of the life of mine operating cost, thus attention has been given to minimizing haul distances and maximizing the carrying capacity of each truck. The truck beds are 110 cubic meter (a coal bed option) due to the low density of the materials. Table 16-25 summarizes the hauling capacity of the trucks for each material: ore, interburden and overburden. Due to the high moisture content, only the interburden fills the truck in both tonnage capacity and volume capacity. The ore and overburden fill the truck to tonnage capacity, but not to volume capacity.

Table 16-24: Haul Truck Capacity

	Units	Ore	Interburden	Overburden
Material Characteristics				
Dry Bank Density	(mt/bcm)	1.22	0.76	0.91
Wet Bank Density	(mt/bcm)	1.60	0.98	1.42
Dry Loose Density	(mt/lcm)	1.02	0.63	0.76
Wet Loose Density	(mt/lcm)	1.34	0.82	1.18
Swell	(%)	20.0%	20.0%	20.0%
Moisture Content	(%)	31.5%	29.0%	56.0%
Truck Capacity By Weight				
Tonnes Per Truck (Dry)	(mt)	68.2	69.5	57.5
Tonnes Per Truck (Wet)	(mt)	89.7	89.7	89.7
Truck Rated Payload	(mt)	89.7	89.7	89.7
Percent Full By Weight	*(%)*	*100%*	*100%*	*100%*
Truck Capacity By Volume				
Truck Rated Volume Capacity	(lcm)	110.0	110.0	110.0
Truck Volume Utilized	(lcm)	67.1	109.7	75.8
Percent Full By Volume	*(%)*	*61%*	*100%*	*69%*

Mine haul roads will require upgrading with crushed aggregate to support the weight of the haul trucks given the load?? bearing capacity of the road surfaces. As such, haul roads were classified in two categories: in-pit haul roads that provide access to the mining faces are required in each mining phase; and ex-pit haul roads, which provide access from the pit crest to the beneficiation plant, waste rock storage facilities and other surface facilities. Both the in-pit and ex-pit haul roads were designed at 25 m width, including berms and ditches.

The in-pit haul roads within each mining phase are temporary and those on the leading wall of a mining phase will be mined out by the subsequent mining phase. The roads and ramps on the in-pit backfill and the ex-pit dumps are also somewhat temporary, but will require surfacing due to the nature of the dumped material. Ex-pit haul roads are permanent throughout the mine life and are extended or shortened incrementally to match the advancing mine phases.

Pit centroids by mining phase, in-pit backfill centroids, ex-pit WSF centroids, and overfill centroids were approximated for each year using the face and waste dump advances and the weighted average elevation. Haul profile routes from the annual pit centroids to the corresponding in-pit backfill, ex-pit WSF, and overfill centroids were created by increments of uniform road gradients. A maximum grade of 8 percent was used based upon the truck specifications and anticipated ground conditions

The haul profile segments of distance, grade and rolling resistance were entered into an excel file and then transferred to a database as input to a haulage simulation program based on Caterpillar©'s Fleet Production and Cost Analysis (FPC) software. The simulation results provide the estimate waste rock and phosphorite haul times and the required truck shifts to move the ore and waste materials from sources to destinations. Up-hill speeds were regulated by the rim pull curves, grade of the segment and the rolling resistance assigned to the segment with the maximum speed limited to 56 kph on long flat hauls. Down-hill maximum speed was restricted to 29 kph on the 8% gradient. The inputs to the simulation runs are shown in Table 16-25 and the truck capacities are previously shown in Table 16-24. The trucks used for the Updated Pre-Feasibility Study are 90 mt trucks with a 110 cubic metre coal bed being loaded with a 31 cubic meter FEL or the surface miner.

Table 16-25: Haul Simulation Assumptions

Rolling Resistance Parameters		Value
1st segment after being loaded		8.0%
All remaining segments		5.5%
Last 500m (or less) on fresh waste dump		8.0%
Combined Fixed Times for Loading and Dumping in Minutes		
Material	Loaded with FEL	Loaded with surface miner
Ore	4.40	8.37
Interburden	5.10	11.67
Overburden	4.20	9.83

The average haul times (not including the fixed times of loading and dumping) by year and material type are shown in Table 16-26. The haul times vary depending on the depth in the pit and the distance to the plant or waste destination. The shorter haul times for Interburden reflect that much of this material is placed as in-pit backfill, while the longer haul times for Overburden are indicative of placement as both backfill and in the ex-pit dump. Once the backfill operations for interburden waste begin in Year 3, the waste haul times become fairly uniform.

Table 16-26: Average Haulage Time in Minutes

Year	Ore to Plant	Interburden to Dump	Overburden to Ex-Pit Dump	Overburden to TSF
Pre-Production	35.36	23.96	22.96	35.21
1	26.85	28.40	25.30	
2	39.26	31.67	24.09	
3	36.28	19.40	28.89	
4	34.48	13.45	27.89	
5	40.46	18.52	25.53	
6	33.71	22.10	25.42	58.35
7	37.30	15.72	26.51	
8	35.46	16.41	27.78	
9	32.09	17.72	25.07	
10	35.92	19.86	26.75	
11	33.20	19.60	31.43	
12	28.95	20.11	27.74	
13	30.98	23.23	29.34	
14	33.90	21.54	26.74	
15	35.10	17.75	33.16	
16	33.04	22.09	27.90	
17	33.48	25.27	29.35	
18	24.70	20.64	22.83	
19	24.90	26.85	28.31	
20	23.31	15.29	-	

The surface miner shift requirements for ore and some interburden are based on the shift productivity of 4,894 bcm for ore and 5,108 bcm for interburden. The front end loader shift requirements for overburden and interburden are based on the shift productivity of 10,658 bcm in overburden and 12,044 bcm in interburden. The truck shift requirements are based on the haulage simulation results. The balance of the major equipment shift requirements are based on the number of work areas and type of work for each piece of equipment. Table 16-27 summarizes the operating shifts by year for each equipment type.

Table 16-27: Schedule Shifts for Major Mining Equipment by Year

	Units	-1	1	2	3	4	5	6	7	8	9	10
SCHEDULED SHIFTS FOR MAJOR EQUIPMENT:												
WIRTGEN 2500SM Continuous Miner (2.5 M)	Shifts	78	350	665	697	634	634	569	644	695	604	588
CAT 994 Front End Loader (31 CuM)	Shifts	2,491	2,377	2,139	2,109	2,107	2,119	2,172	2,207	2,098	2,074	2,071
CAT 777 Coal Haul Truck (90 tn)	Shifts	19,730	20,371	21,320	18,600	16,111	18,487	20,694	18,316	17,143	16,751	18,442
CAT D9/D10 Track Dozers	Shifts	972	1,215	1,458	1,701	1,701	1,701	1,701	1,701	1,701	1,701	1,701
CAT 834 Wheel Dozer (450 HP)	Shifts	122	243	243	365	365	365	365	365	365	365	365
CAT 16M Motor Graders (297 HP)	Shifts	778	826	1,069	1,118	1,118	1,118	1,118	1,118	1,118	1,118	1,118
CAT 770-W Water Truck (30,000 Ltr)	Shifts	486	608	729	851	851	851	851	851	851	851	851
CAT 336 Aux Loader (1 CuM)	Shifts	122	243	243	365	365	365	365	365	365	365	365
CAT CS-56 Compactor (147 HP)	Shifts	215	431	431	646	646	646	646	646	646	646	646
Total Equipment Shifts	Shifts	24,993	26,665	28,298	26,450	23,897	26,285	28,479	26,212	24,980	24,474	26,147

	Units	11	12	13	14	15	16	17	18	19	20	Total
SCHEDULED SHIFTS FOR MAJOR EQUIPMENT:												
WIRTGEN 2500SM Continuous Miner (2.5 M)	Shifts	658	659	654	614	695	652	656	521	685	718	12,673
CAT 994 Front End Loader (31 CuM)	Shifts	2,086	2,203	2,144	2,095	2,106	2,357	2,226	1,898	1,459	1,157	43,696
CAT 777 Coal Haul Truck (90 tn)	Shifts	18,633	20,180	19,731	21,468	18,329	22,481	21,424	14,947	13,367	8,162	384,687
CAT D9/D10 Track Dozers	Shifts	1,701	1,701	1,701	1,701	1,701	1,701	1,701	1,701	1,701	1,701	34,263
CAT 834 Wheel Dozer (450 HP)	Shifts	365	365	365	365	365	365	365	365	365	365	7,169
CAT 16M Motor Graders (297 HP)	Shifts	1,118	1,118	1,118	1,118	1,118	1,118	1,118	1,118	1,118	1,118	22,793
CAT 770-W Water Truck (30,000 Ltr)	Shifts	851	851	851	851	851	851	851	851	851	851	17,132
CAT 336 Aux Loader (1 CuM)	Shifts	365	365	365	365	365	365	365	365	365	365	7,169
CAT CS-56 Compactor (147 HP)	Shifts	646	646	646	646	646	646	646	646	646	646	12,712
Total Equipment Shifts	Shifts	26,422	28,088	27,573	29,222	26,175	30,535	29,351	22,411	20,555	15,082	542,293

16.9 MINE PERSONNEL REQUIREMENTS

The mine personnel requirements are summarized on Table 16-28 for the supervision, engineering and geology personnel, and Table 16-29 for the hourly mine operations and maintenance personnel. The personnel operating the surface miner, the front end loader and the motor graders are assigned based on the number of units in the fleet, and the balance of the equipment operating personnel are based on the total number of units operating. Some cross training of operators is assumed for the truck drivers, dozer operators and service crew. For the total manpower requirements on the payroll, a 10% VSA allocation to cover vacations, sick leave and absenteeism is included. The hourly labor is based on four crews, so only one quarter of the personnel listed on the bottom of Table 16-29 would be on the property during a particular shift. The salaried staff would work days only with the exception of the shift supervisors who would rotate with their assigned crew.

Table 16-28: Mine Supervision, Engineering and Geology Personnel

JOB TITLE	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
MINE OPERATIONS:																					
Mine Operations Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Shift Supervisors	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Environmental Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Operations Total	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	5	5
MINE MAINTENANCE:																					
Mine Maintenance Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Shift Supervisors	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Planner/Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Warehouse Supervisor	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Mine Maintenance Total	6	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	5	2
MINE ENGINEERING:																					
Chief Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Senior Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
AutoCad Tech	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Planning Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Mine Engineering Total	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	2	2
MINE GEOLOGY:																					
Senior Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Sr Geotechnical Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Topography Personnel	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Geo Tech - Sampler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Mine Geology Total	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	1
TOTAL PERSONNEL	22	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	23	17	10

Table 16-29: Mine Operations and Maintenance Labor

JOB TITLE	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
MINE OPERATIONS:																					
Shovel Operator	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Loader Operator	20	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	12	8
Haul Truck Driver	130	134	140	122	106	121	136	120	113	110	121	122	133	130	141	120	148	141	98	88	54
Track Dozer Operator	7	8	10	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11
RTD Operator (Wheel Dozer)	1	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Grader Operator	8	8	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Water Truck Operator	3	4	5	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Utility Equip Operator (Service Crew)	3	5	5	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Mine Dispatcher	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Laborer	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	0
Operations Total	184	193	206	194	178	193	208	192	185	182	193	194	205	202	213	192	220	213	170	156	114
MINE MAINTENANCE:																					
Senior Maintenance Mechanics	53	56	59	55	51	55	59	55	53	52	55	55	58	59	62	55	63	60	48	44	33
Maintenance Technicians	27	28	30	28	26	28	30	28	27	26	28	28	29	30	31	28	32	30	24	22	17
Welder / Mechanic	24	26	27	25	23	25	27	25	24	24	25	25	27	27	29	25	29	28	22	20	15
Warehouse Attendant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Warehouse Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Fuel & Lube Crew	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Tire Crew	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Laborer Mnt	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Maintenance Total	126	132	138	130	122	130	138	130	126	124	130	130	136	138	144	130	146	140	116	108	85
VS&A at 10.0%	31	33	34	32	30	32	35	32	31	31	32	32	34	34	36	32	37	35	29	26	20
TOTAL LABOR REQUIREMENT	341	358	378	356	330	355	381	354	342	337	355	356	375	374	393	354	403	388	315	290	219
Maint/Operations Ratio	0.68	0.68	0.67	0.67	0.69	0.67	0.66	0.68	0.68	0.68	0.67	0.67	0.66	0.68	0.68	0.68	0.66	0.66	0.68	0.69	0.75

17 RECOVERY METHODS

17.1 PROCESS (BENEFICIATION) DESCRIPTION

The Bayovar 12 process plant has been designed to produce two phosphate concentrate products totaling 1 million mtpa, having minimum assays of 24 or 28% P_2O_5 DAPR in roughly equal proportions. The process plant is designed for batch operation so that it can easily switch from one phosphate concentrate product to the other with a minimum of down-time, according to the product demand and the mine extraction schedule.

The plant feed consists of phosphorite beds (Capas) consisting of phosphate pellets hosted in diatomite. The plant feed consists of 13 separate capas numbered 1 through 13. The lowest number capa is the highest in the mine stratigraphy. A testwork program has demonstrated that some of the capas are more suitable for producing 24% P_2O_5 DAPR while others are more suitable to produce 28% P_2O_5 DAPR. The mine plan offers considerable flexibility for the selective mining and blending of various capas to optimize process plant design with particular reference to P_2O_5 recoveries and throughput rates for each of the products. These features, when combined with 85% plant availability, offer the opportunity to optimize the plant design parameters presented below at feasibility level.

The beneficiation process consists of washing, scrubbing, and three stages of desliming, concentrate dewatering and concentrate drying to a target of four percent moisture. Unit operations consist of drum washing, size classification, attrition scrubbing, hydraulic classification with hydrocyclones and a hydrosizer, belt filtration and rotary drying. The two DAPR products use identical process unit operations, except that the cut point for the 28% P_2O_5 DAPR is higher, eliminating lower grade fines. The beneficiation process will utilize seawater and some recycled process water. The final DAPR products will not rinsed with fresh water to remove salts so that approximately 6,000 ppm NaCl will remain in the product.

Samples for metallurgical testing were collected by Focus Venture's geology team in Peru. Jacobs Engineering performed testwork described in the following reports to determine the plant's design parameters.

Belt weigh scales and samplers will be located throughout the plant for metallurgical accounting and product quality control.

Each process plant consists of the following unit operations as shown in Figure 17-1.

- Run-of-Mine (ROM) Ore Handling
- Drum Washing, Size Classification and Cyclone Classification
- Primary Attrition Scrubbing
- Secondary Attrition Scrubbing, Size Classification, Cyclone Classification and Hydraulic Sizing
- Belt Filtration
- Concentrate Drying and Product Storage
- Tailing Impoundment and Water Systems

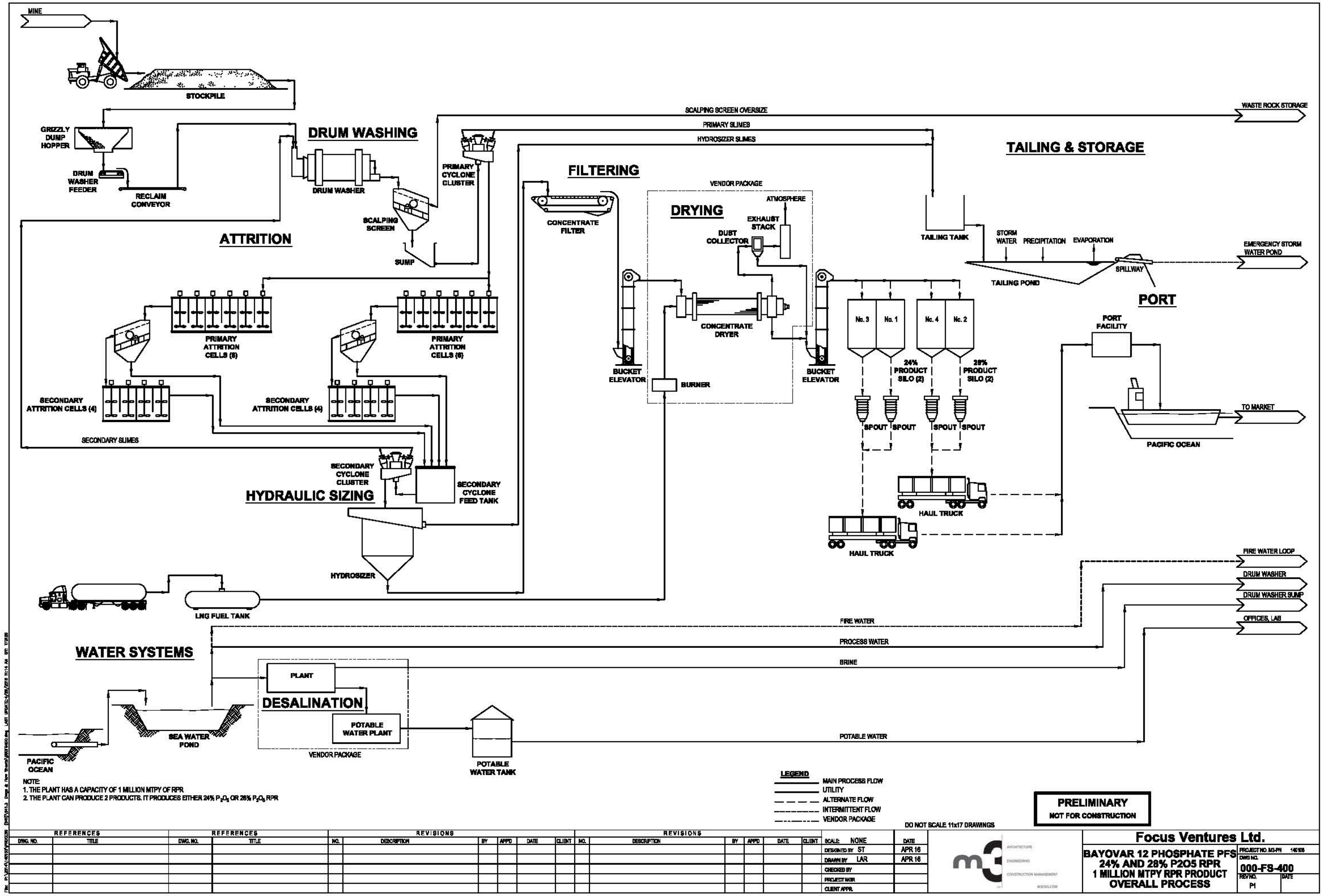


Figure 17-1: Bayovar 12 Plant Flowsheet

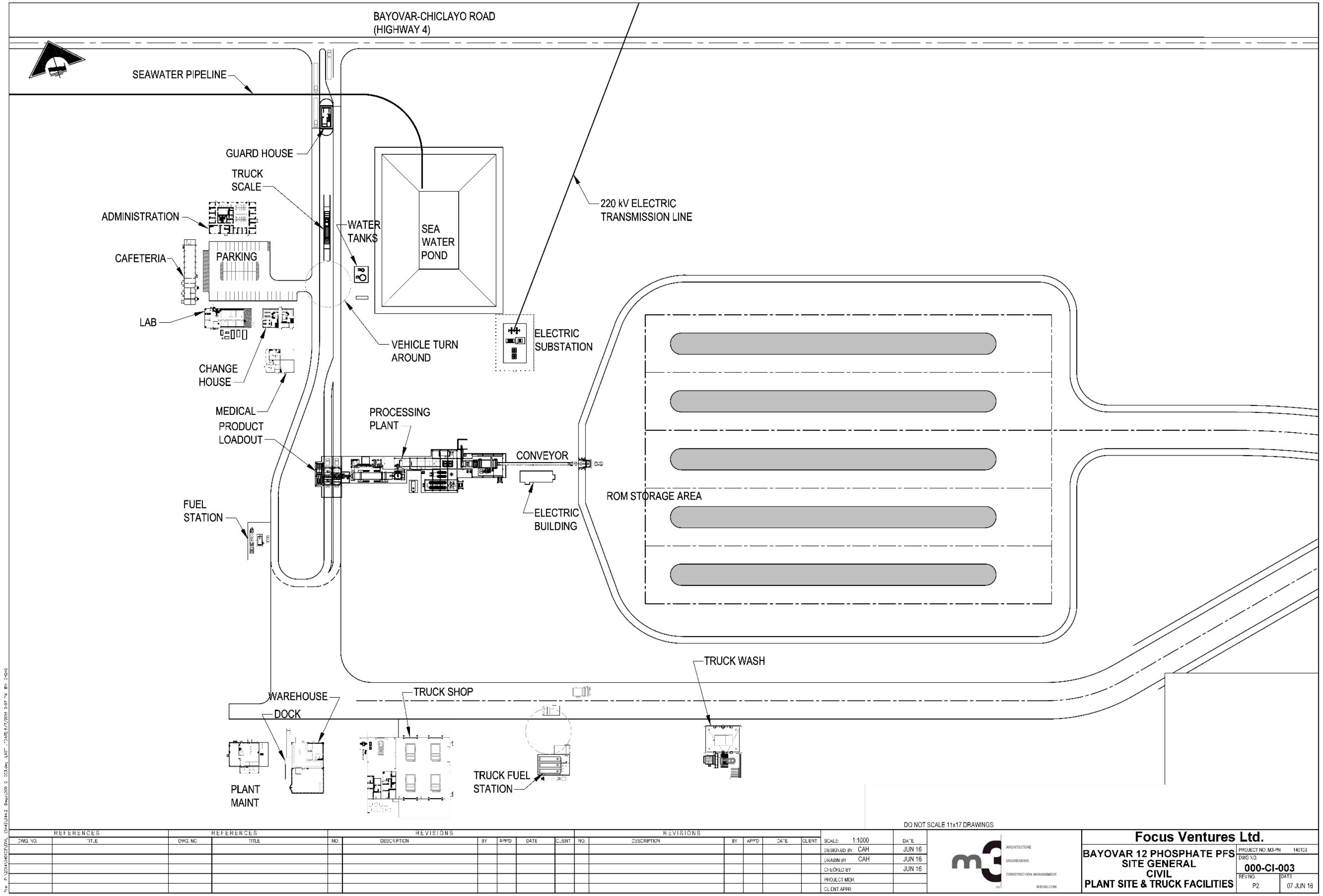


Figure 17-2: General Arrangement for Bayovar 12 Plant

17.2 GRADE AND RECOVERIES

Applying the test results for each ore layer indicated that the phosphorite recovery for Operating Mode A is 81.5% with a grade of 24.1% P_2O_5 and for Operating Mode B it is 66.4% with a grade of 28.8% P_2O_5 (Table 17-1).

In operation it is likely that the desliming cut point would be slightly less than 105 um so as to reduce the grade to 28% P_2O_5 and increase phosphorite recovery to more than 70%. For example, desliming at 74 um (200 mesh) in Operating Mode B is projected to produce a concentrate containing 27.7% P_2O_5 at 82% P_2O_5 recovery.

Table 17-1: Laboratory P_2O_5 Recovery for 24 and 28 Percent P_2O_5 DAPR Concentrate

Operating Mode A

	Resource			+28 mesh		28/270 mesh		+270 Concentrate		-270 mesh		% P_2O_5
Capa	Metric Tons	% P2O5	% WSS	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	Recovery
PH01	5,278,423	12.33	6.51	0.88	21.92	42.14	23.90	43.02	23.86	50.47	4.09	83.2
PH02	9,999,977	11.15	7.71	3.33	27.95	39.57	22.07	42.90	22.52	49.39	3.02	86.6
PH05	824,845	12.13	6.93	3.11	28.63	35.68	25.44	38.79	25.70	54.28	3.98	82.2
PH07	3,261,593	10.28	5.84	1.04	27.83	28.89	25.75	29.93	25.83	64.23	3.98	75.2
PH08	2,185,906	10.69	5.43	0.98	30.25	26.95	25.99	27.93	26.14	66.64	5.08	68.3
PH09	4,433,704	12.39	4.48	0.88	31.37	37.09	25.81	37.97	25.94	57.55	4.41	79.5
PH10	1,509,066	10.46	4.68	1.03	31.89	29.60	26.00	30.63	26.20	64.69	3.77	76.7
Combined	27,493,514	11.43	6.37	1.87	27.91	36.73	23.93	38.60	24.12	55.03	3.85	81.5

Operating Mode B

	Resource			+28 mesh		28/150 mesh		+150 Concentrate		-150 mesh		% P_2O_5
Capa	Metric Tons	% P2O5	% WSS	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	% Wt	% P2O5	Recovery
PH03	5,264,531	17.52	5.23	6.59	31.12	40.64	30.33	47.23	30.44	47.54	6.62	82.0
PH04	1,757,167	13.59	5.35	4.04	30.58	33.24	29.48	37.27	29.60	57.38	4.47	81.1
PH06	6,044,550	13.56	4.26	1.29	29.45	27.54	28.88	28.82	28.90	66.92	7.82	61.4
PH11	3,685,348	12.85	4.13	2.20	30.93	25.71	27.95	27.90	28.18	67.97	7.34	61.2
PH12	5,027,668	13.93	2.98	2.80	31.78	27.71	27.86	30.50	28.22	66.52	8.00	61.8
PH13	9,499,165	13.73	2.93	1.05	30.92	28.68	27.66	29.73	27.77	67.34	8.13	60.1
Combined	31,278,429	14.26	3.86	2.61	30.98	30.22	28.65	32.83	28.83	63.31	7.56	66.4

17.3 PROCESS PLANT DESIGN

17.3.1 Design Basis

The plant is designed to produce up to 1,000,000 dry mtpa of combined 24% and 28% P_2O_5 DAPR concentrate.

The plant equipment is sized for the 28% P_2O_5 product because it has a lower yield, requiring more feed and producing more tailings.

For both products, the first two desliming steps are performed with hydrocyclones to reject the -53 micron particles. The third desliming step is performed with a hydrosizer. For 24% P_2O_5 product, the hydrosizer cut-point is adjusted to remove the remaining slimes less than 53 microns. When 28% P_2O_5 product is produced, the hydrosizer cut-point is adjusted to remove particles less than 105 microns.

Product recovery losses are due to phosphate (P_2O_5) reporting to slimes during attrition scrubbing and classification. There is an additional 1 percent loss of phosphate concentrate to dust and shrinkage. The plant operating availability is 85% based on 7,446 operating hours per year.

The plant performance for each type of product is shown in Table 17-2.

Table 17-2: Plant Products and Plant Performance

	24% P_2O_5	28% P_2O_5
Plant Products and Plant Performance		
Yearly Production of DAPR, mtpy	Up to 1,000, 000	Up to 1,000, 000
Product Grade	24% P_2O_5	28% P_2O_5
Recovery of P_2O_5 %	81.5	66.4
Product size	Greater than 270 mesh (53 micron)	Greater than 150 mesh (106 micron)
Calculated Residual Salt in Product ,%	0.6	0.6
Plant Availability (Run Time), %	85	85
ROM Ore and Seawater		
Capa #	1, 2, 5, 7, 8, 9, 10	3, 4, 6, 11, 12, 13
Feed Rate, mtph	354.7	373.9
P_2O_5 in ROM Ore, %	11.43	14.26
Water soluble Salt in ROM, %	6.37	3.86
Moisture in ROM ore, %	30	30
Seawater Required, m^3/h	1,298	1,591
Plant Operation		
Hydrocyclone cut size, mesh (micron)	270 mesh (53 micron)	270 mesh (53 micron)
Hydrosizer cut size, mesh (micron)	270 mesh (53 micron)	150 mesh (106 micron)
Solid Tailings (Excluding Dissolved Salt), mtph	193	220
Tailings slurry, m^3/h	1494	1,802

The P_2O_5 yearly and hourly and yearly production rates for the 24% and 28% P_2O_5 DAPR and product grades are shown in Tables 17.3 and 17.4

Table 17-1: 24% P_2O_5 DAPR: Product Mass Balance

Feed	Tonnes/yr.	Tonnes/hr.	% P_2O_5
Salt	168,540	22.6	
Scalp O'Size (+18mm)	24,722	3.3	
+270 mesh (+53 µm)	1,010,101	135.7	
-270 mesh (-53 µm)	1,437,377	193.0	
ROM	2,640,740	354.7	11.50
Tailings			
Salt	168,540	22.6	
1st -270 mesh (-53 µm)	934,295	125.5	4.02
2nd -270 mesh (-53 µm)	452,774	60.8	4.02
3rd -270 mesh (-53 µm)	50,308	6.8	4.02
Total Tailings	1,605,917	215.7	4.02
Scalp O'Size (+18mm)	24,722	3.3	8.0
Total Waste	24,722	219.0	4.02
Product			
+28 mesh (600 µm)	51,132	6.9	27.5
28/100 mesh (150 to 600 µm)	375,825	50.5	28.68
100/270 mesh (53 to 150 µm)	583,144	78.3	20.89
Total Concentrate	1,010,101	135.7	24.14
Shrinkage	-10,101	-1.4	24.14
Shipped Concentrate	1,000,000	134.3	24.14

Table 17-2: 28% P_2O_5 DAPR: Product Mass Balance

Feed	Tonnes/yr.	Tonnes/hr.	% P_2O_5
Salt	103,366	13.9	
Scalp O'Size (+18mm)	26,804	3.6	
+270 mesh (+53 µm)	1,010,101	135.7	
-270 mesh (-53 µm)	1,643,462	220.7	
ROM	2,783,733	373.9	14.50
Tailings			
Salt	103,366	13.9	
1st -270 mesh (-53 µm)	1,012,965	136	6.86
2nd -270 mesh (-53 µm)	490,899	65.9	6.86
3rd -270 mesh (-53 µm)	139,598	18.7	6.86
Total Tailings	1,746,828	234.5	6.86
Scalp O'Size (+18mm)	26,804	3.6	8.0
Total Waste	1,773,632	238.1	6.86
Product			
+28 mesh (600 µm)	64,848	8.7	31.02
28/100 mesh (150 to 600 µm)	429,929	57.7	30.22
100/270 mesh (53 to 150 µm)	515,325	69.2	26.93
Total Concentrate	1,010,102	135.6	28.59
Shrinkage	-10,101	-1.4	28.59
Shipped Concentrate	1,000,001	134.2	28.59

Note:The Primary and Secondary Cyclones remove material that is finer than 270 mesh. The Hydrosizer cut point is 150 mesh, therefore the first, second and third wastes are reported as -150 mesh. For a more detailed explanation see Section 13.6 in Mineral Processing and Metallurgical Testing.

17.3.2 Run-of-Mine (ROM) Ore Handling

The ROM handling area is designed to stockpile sufficient phosphorite beds from different capas to accommodate to have sufficient material on hand to blend a consistent feed to the plant for either DAPR product.

Haul trucks deliver ore from the open pit mine to designated areas on the stockpile. The stockpile has a total storage capacity of 14 days (104,030 dry tonnes); seven days of storage for 50,644 dry tonnes of ore for 24 percent P_2O_5 DAPR and seven days of storage for 53,383 dry tonnes of ore for 28 percent P_2O_5 DAPR. Front end loaders transfer the ore at a rate of 373.5 dry mtph to the Feed Bin at the head of the Plant. The feed travels from the Feed Bin via a belt feeder to the Drum Washer Feed Conveyor that delivers the ore to the Drum Washer.

The ROM handling area consists of stockpile, feed bin, belt feeder, belt conveyor, and assorted chutes.

The ROM metallurgical sampler, located across the Drum Washer Feed Conveyor, samples wet, salt-bearing feed material entering the plant. A belt scale will determine the weight of the ore entering the plant and a self-cleaning magnet will remove tramp iron before it enters the Drum Washer.

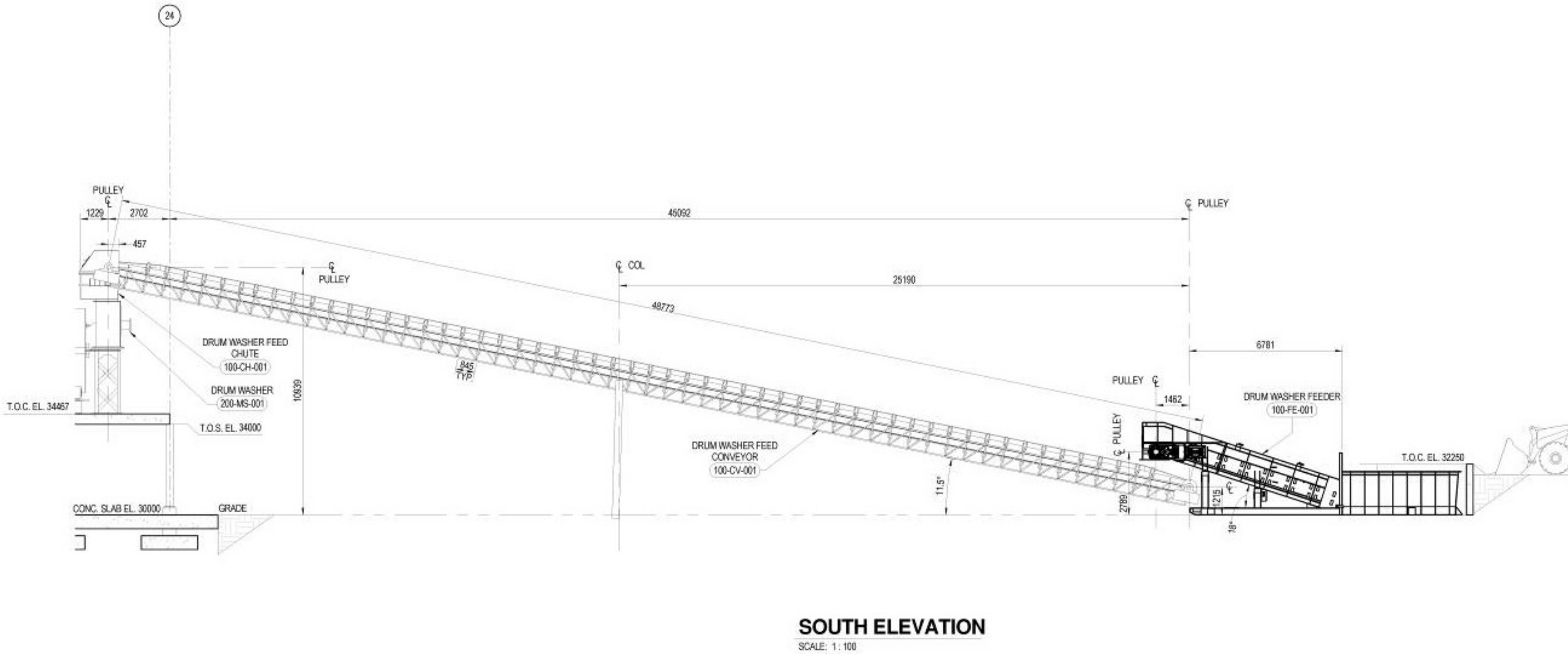


Figure 17-3: General Arrangement of ROM Handling Area

17.3.3 Drum Washing, Scalping, and Cyclone Classification

The phosphorite feed discharges from the Drum Washer Feed Conveyor into the Drum Washer (Figure 17-4), where it is mixed with a proportional amount of process water to form a slurry with a density of 25% solids. The drum washing equipment is designed to disaggregate the unconsolidated phosphate pellets from the diatomite/clay gangue and water soluble salts by tumbling and washing action. Testwork shows that the optimum drum washer retention time is 3 minutes.

The drum washer slurry discharges onto a scalping screen with 18 mm screen openings. The screen oversize is estimated to be approximately 1% of the drum washer solids and consists of low phosphate grade chunks of diatomite with an average phosphate grade of 8% P_2O_5. The scalping screen is equipped with wash sprays that keep the material moving across the screen deck. The oversize material is rejected and conveyed to a bunker where it is periodically loaded from the bunker onto trucks and disposed of at the Waste Rock Storage Facility. The screen undersize material and water pass through the screen into the Drum Discharge Washer Sump.

The washed phosphate pellets and slimes pass through the scalping screen and collect in a sump that is then pumped to the primary cyclone cluster, where slimes are separated as cyclone overflow from phosphate concentrate in the cyclone underflow. Water is added at the sump to dilute the slurry to a density of 20% solids. A primary cyclone feed pump sends the slurry from the drum washer discharge sump to the primary cyclone cluster. The metallurgical shift sampler is located ahead of the primary cyclone cluster. It collects a slurry sample that represents what the scrubbing process will “see”. Most of the soluble salt is removed in the drum washing phase and the “clots” of diatomite from interburden will be removed by the scalping screen, both ahead of the sampler.

The primary cyclones are set to separate at 53 microns (-270 mesh). The cyclone overflow stream flows by gravity to the tailing tank from where it is pumped to the Tailings Storage Facility.

The cyclone underflow, with a slurry density of 50% to 55% solids, is the product stream. This stream flows to the primary attrition scrubbers for further disaggregating and concentrating.

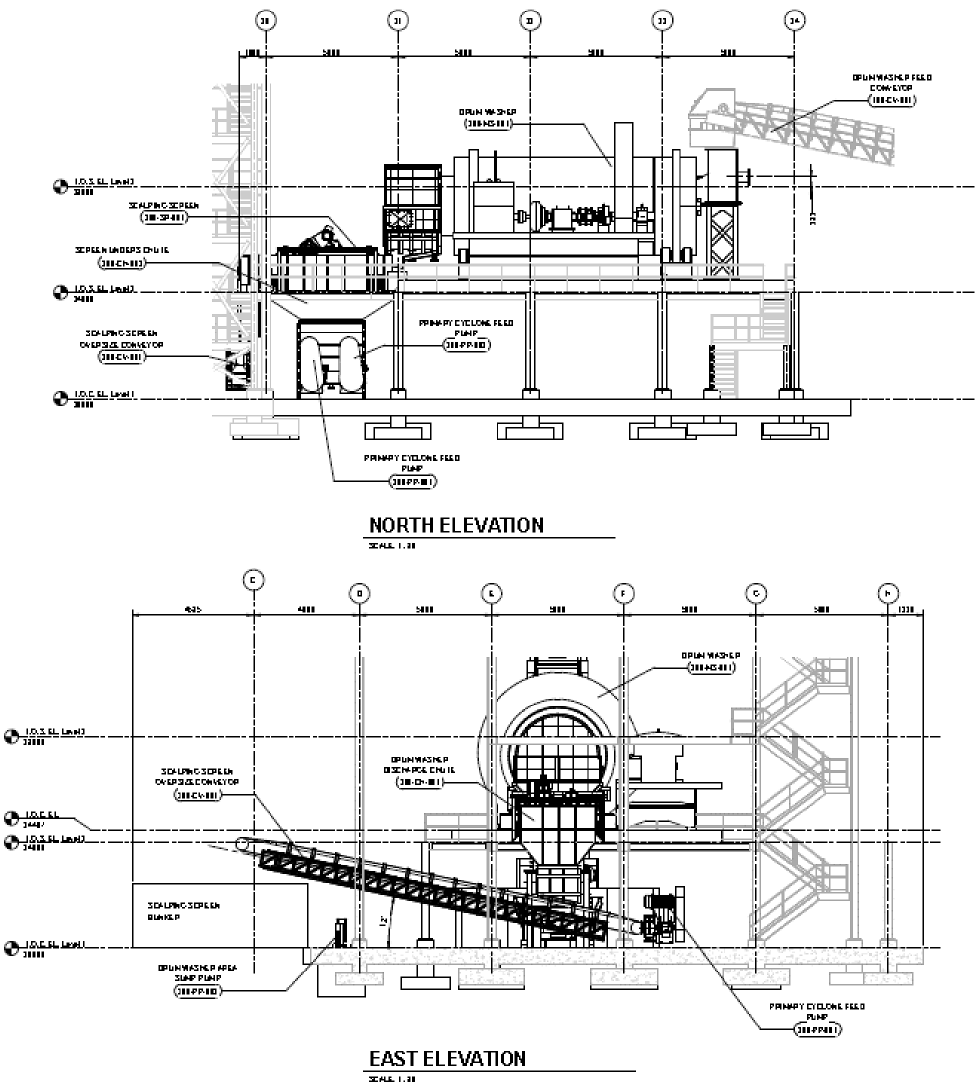


Figure 17-4: General Arrangements of Drum Washing and Scalping

17.3.4 Primary Attrition Scrubbing

The primary attrition scrubbers (Figure 17-5) are arranged in pairs of intense scrubbing machines that disaggregate the low grade phosphate concentrate from more diatomite and clay gangue material. The attrition scrubbers operate at a high slurry density of 50% - 55% solids. Testwork showed that a retention time of 12 minutes was required.

In this circuit, the primary attrition scrubbers are arranged into two parallel rows of six 5-cubic meter attrition cells, each. Each bank of primary attrition scrubbers discharges into an attrition screen feed tank, which then is pumped to an attrition screen. The attrition screens remove phosphate pellets that are larger than 600 micron (28 mesh) for additional scrubbing via the secondary attrition scrubbers. The attrition screen underflow is directed to the secondary cyclone feed tank.

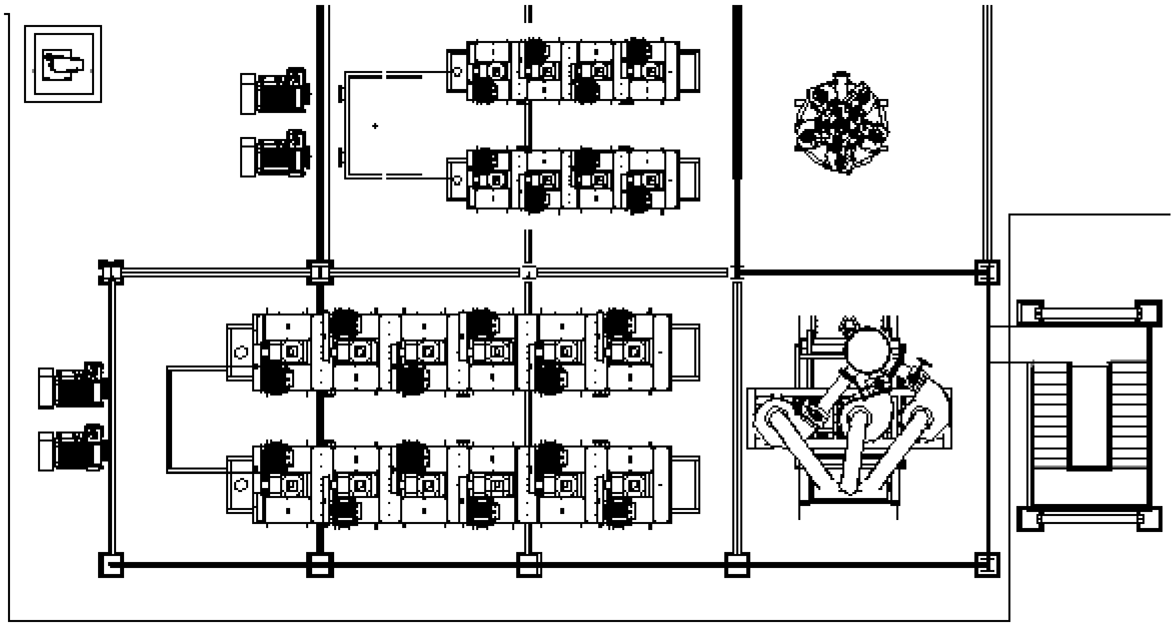


Figure 17-5: General Arrangement of the Attrition Section

17.3.5 Secondary Attrition Scrubbing, Size Classification, Cyclone Classification, and Hydraulic Sizing

The secondary attrition scrubbers, shown on Figure 17-5 are required to beneficiate the +600 micron phosphate pellets via intense scrubbing. Testwork indicates that the retention time in secondary attrition scrubbing is also 12 minutes at a slurry density of 50 - 55 percent solids. The current arrangement is to have two parallel banks of four 2.1-cubic meter attrition cells, each.

The secondary attrition scrubbers discharge to the secondary cyclone feed tank where it is combined with the undersize from attrition screens, where both streams are diluted to 20% solids. The secondary cyclone feed pump pumps the slurry to the Secondary Cyclone cluster where the cut size of 53 microns separates slimes from phosphate pellets. The slimes report to the tailings tank while the underflow from the secondary cyclones reports to the hydrosizer for further classification at a slurry density of 55% solids.

The hydrosizer in the redesigned plant will consist of two 3.1-m x 3.1-m cones that will handle a flow of 239 mtph of phosphate concentrate and additional “teeter” water. The hydrosizer utilizes an upward flow of water to fluidize light-density particles into an overflow stream and allow the coarser, denser particles to settle in the underflow stream. For 24% P_2O_5 DAPR, the overflow stream will normally contain particles sizes of -53 microns (-270 mesh). For 28% P_2O_5 DAPR overflow stream normally contains -106 microns particles (-150 mesh). The hydrosizer overflow will gravity flow to the tailing tank from where it will be pumped to the TSF. The hydrosizer underflow is the final washed product stream at a slurry density of 65% to 70% solids. This stream of approximately 212 mtph of product will flow by gravity to the concentrate belt filter. A metallurgical sample will be taken between the hydrosizer underflow and the belt filter.

17.3.6 Belt Filtering

The hydrosizer underflow discharges into the distribution box above the concentrate belt vacuum filter. An auger inside the distributer box spreads a uniform thickness of slurry across the width of the belt filter. The belt vacuum filter dewaters the phosphate concentrate and returns the filtrate solution to the process solution as make-up water at the secondary cyclone feed tank. The filter cake drops from the belt filter onto a bucket elevator that loads the concentrate dryer feed bin.

The belt vacuum filter reduces the concentrate moisture from 35 percent to 15 percent. The tests indicated that rinsing the filter cake with seawater water does not reduce the cadmium content, therefore the filter was sized for dewatering without a rinsing. The Metsim simulation calculated the remaining salt content at 0.6 percent in both the 24% the 28% P_2O_5 DAPR final products.

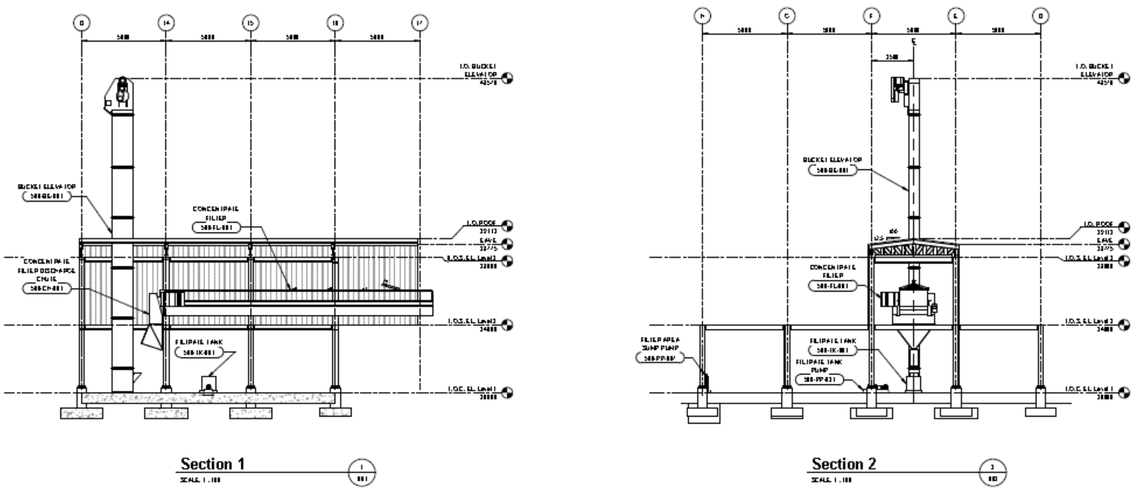


Figure 17-6: General Arrangement for Concentrate Belt Vacuum Filter

17.3.7 Concentrate Drying and Product Storage

The dewatered filter cake is stored in the concentrate dryer feed bin. A screw feeder meters the filtered cake into the rotary concentrate dryer at a rate of 160 mtph at 15% moisture.

The dryer heats the concentrate from 25° C to 120° C. The target product moisture is 4%. The dryer fuel is liquefied natural gas (LNG). A blower supplies air to the burner, which supplies approximately 80 million BTU/h. The rotary dryer has replaced the fluid bed dryer that was quoted in the original PFS because fluid bed dryers start having problems with fines below 100 microns. The rotary dryer in the current design is 4-m diameter by 32-m long. It is a tire driven unit powered by a 200 HP motor. The off-gas handling includes a combustion chamber, an off-gas cyclone and a baghouse dust collector. The baghouse dust is collected and returned as product to the system. LNG fuel for drying accounts approximately 40% of the plant operating cost. This study accounts for LNG being delivered daily by truck.

The dry phosphate concentrate discharges from the rotary dryer to a discharge conveyor and onto a bucket elevator that lifts the concentrate to the top of four product silos. The phosphate concentrate is routed to individual silos via a pair of diverter gates and tubular drag conveyors. Each product silo has a 7-hour retention time and a volume of approximately 438 cubic meters of storage. Each silo is fitted with a bin vent and an unloading spout.

After storage the final product is loaded into haul trucks, delivered to the port facility and loaded onto ships for market. The port facility is owned and operated by a Juan Paulo Quay (JPQ,) a third party and part Owner of the Bayovar 12 concession.

17.3.8 Metallurgical Sampling and Product Control

Grade control is essential to the success of the Bayovar 12 plant. The current plant flowsheet uses a belt crosscut sampler on the drum washer feed conveyor to determine the head grade, and metallurgical slurry samplers ahead of the primary cyclones for process control, on the discharge of the hydrosizer ahead of the belt filter to measure the product grades and on tailings stream to determine recovery and losses. Shift samples will be assayed at the onsite laboratory. Future need for an onstream analyzer will be evaluated once the plant is up and running.

The single process line will produce two products in a batch operation of the plant. Switching from 28% P_2O_5 DAPR to 24% P_2O_5 DAPR will not be a problem since the grade of the initial 24% product after the switchover will have a slightly higher grade due to the remaining 28% product in the dryer. Switching from 24% to 28% product will require extra care since plant will not want to dilute the 28% product with 24% product. The residence time in the 32-m long concentrate dryer will be approximately 20 minutes.

17.3.9 Tailings Storage Facility

Overflow from the primary and secondary clusters and the hydrosizer are collected in a tank and pumped directly to the TSF. Due to the very slow settling rate, it is impractical to reclaim water in a thickener. The tailing slurry contains approximately 11 percent solids.

The TSF consists of a tailing tank, 800 HP slurry pumps, an 8.6 km 20" pipeline, and tailings storage impoundments.

17.3.10 Water Systems

The Bayovar 12 plant site uses 1,591 cubic meters per hour of seawater to beneficiating the ore. Seawater Pumps deliver water via a pipeline from the Port of Bayovar to a seawater pond located at the Bayovar mine site. The seawater pond is large enough to accommodate 24 hours of water for the plant operation and maintain a reserve for the firewater system. Seawater is used at the scalping and attrition screens, cyclone feed tanks, hydrosizer and for cleaning the concentrate filter cloth.

A small seawater desalination plant produces desalinated water for potable use at the plant, offices and ancillary facilities. The desalination plant uses a reverse osmosis unit that is capable of producing 2.9 m^3/h of desalinized water.

Four 250 HP, horizontal centrifugal pumps deliver the seawater via a 32" HDPE pipeline. The Seawater Pond measures of 75 m by 100 m and a depth of 9.8 m. The seawater pond, the seawater pond pumps, firewater system, desalination plant and desalinized water tank, potable water plant and tank and associated equipment are located at the Bayovar 12 plant site.

The current plant design recycles overflow from the secondary cyclones in order to decrease the make-up water demand and increase the tailings slurry density, which in turn reduces the volume of tailings storage required. Recycled water is piped to the drum washer, scalping screen spray and primary cyclone feed tank.

It is not currently possible to reclaim water with a tailing thickener because the settling rate is too slow to be practical. The tailings stream removes most of the phosphorite feed's original salt content to the TSF, where the tailings eventually reach saturation and salt precipitates. Recovering water from the tailings pond is not practical due to slow settling solids and salt saturation of supernatant water in the TSF. Consequently, with the exception of some recycled water from the secondary cyclone overflow, the plant uses the seawater once and disposes of it in the TSF.

17.3.11 Capital Equipment for Plant and Infrastructure

The Bayovar 12 plant consists of two planned process lines, each having a capacity of producing 1,000,000 tonnes per year of DAPR concentrate. Each process line has identical equipment from the ROM feed section (Areas 100 & 105) to the concentrate drying loadout section (Areas 550 & 555). The tailings lines from the two process lines come together in a large pump box prior to pumping to the TSF.

Table 17-5Table is a summary of the major equipment required for the two process lines. This list includes all of the significant process equipment as well as the major equipment components that support the plant infrastructure. Costs for the equipment mainly reflect budgetary pricing that was provided by reputable manufacturers for the Bayovar 12 project. These costs were used to build up the capital cost estimate described in Section 21.

Table 17-5: Summary of Major Capital Plant and Infrastructure Equipment

Tag Numbers	Item	Description	HP	Qty per plant	Unit Cost	Total Plant
100-FE-001	ROM Feed Station	90 Tonnes Truck Cap., Pocket Cap. 180 Tonnes, 36" wide belt feeder	100	1	$ 431,250	$ 431,250
100-CV-001	Drum Washer Feed Conveyor	36" belt, 160 ft/sec; 11.7 degrees lift; 330 MTPH of 1842 kg/m³ phosphate rock, 25° surcharge	30	1	$ 202,500	$ 202,500
200-MS-001	Drum Washer w/mtrs	Rotary Drum, Gear Driven; 16 ft [4.9m] dia. x 40 ft [12.2m] long. Inching Drive	700	1	$ 2,125,500	$ 2,125,500
200-SR-001	Scalping Screen	Linear 4'x10' wet screen; high intensity; 18-mm opening	25	1	$ 108,911	$ 108,911
200-PP-001&002 /	Primary Cyclone Feed Pumps	12x10 Horizontal Centrifugal Pump, one operating, one standby	250	2	$ 83,652	$ 167,304
200-CY-001	Primary Cyclones	5-place cyclone manifold with 20" diameter cyclones; 4 installed, 1 spare;	N/A	1	$ 140,118	$ 140,118
300-MS-001 thru 006 & 300-MS-007 thru 012	Primary Attrition Scrubbers	Two banks of six 5-m³ tanks with dual impeller per shaft	75 HP each	12	$ 83,250	$ 999,000
350-CY-001	2nd Cyclone Cluster	2-place cyclone manifold with 20" diameter cyclones	N/A	1	$ 79,760	$ 79,760
350-MS-001	Hydrosizer	Two cones: 3.2-m x 3.2-m each with a separation section, HDPE lined cone sections, epoxy-coated C.S.,	N/A	1	$ 252,000	$ 252,000
350-MS-021 thru 024 & 350-MS-025 thru 028	2nd Attrition Scrubber	Two banks of four 2.125 m³ tanks with dual impeller per shaft	40 HP each	8	$ 74,875	$ 599,000
350-PP-003 & 004	2nd Cyclone Feed Pump	8x6 Horizontal Centrifugal Pump, one operating, one standby	200 HP each	2	$ 56,358	$ 112,716
350-SR-001	Attrition Screen	4' wide by 10' long; high capacity long life polyurethane screen panels with 600 micron openings	25 HP each	2	$ 108,911	$ 217,822
500-FL-001	Concentrate Filter	64 m² filter area; includes 350 HP vacuum pump, filtrate tanks, vacuum receiver, distributor box and auger	50 HP belt drive + 350 HP vacuum pump	1	$ 890,000	$ 890,000
500-BE-001	Dryer Bucket Elevator	25-m height; 160 mtph capacity; loads concentrate dryer	40	1	$ 125,000	$ 125,000
550-BE-001	Silo Bucket Elevator	27.5m height; 160 mtph capacity; loads product silos	40	1	$ 150,000	$ 150,000
550-BN-001 &002	Product Silo	14' Dia. X 55' H bolted steel silo; four silos x 6-hour storage each; includes bin vent & spouts	N/A	4	$ 181,273	$ 725,092

Tag Numbers	Item	Description	HP	Qty per plant	Unit Cost	Total Plant
550-DC-001	Dust Collector	Cartridge Style Dust Collector, fan, and rotary valve	40	1	$ 130,152	$ 130,152
550-KN-001	Concentrate Rotary Dryer	4-m dia. x 32-m long; 2° inclination; LNG-fired; includes off-gas combustion & handling, and gas cyclone/dust collection	N/A	1	$ 2,135,000	$ 2,135,000
600-PP-001/002 & 003/004	Tailing Pump	8x10 Rubber-Lined Horizontal Centrifugal Pump, one operating, one standby	800 HP each	2	$ 280,000	$ 560,000
650-EQ-001 & 650-EQ-002	Seawater Intake Structure	Low velocity 360-degree intake; 1,140 m^3/hr each max flow	N/A	2	$ 141,000	$ 282,000
650-PP-001 thru 004	Sea Water Supply Pumps	10 x 8 Horizontal centrifugal pumps; 316 SS,	250 HP each	4	$ 86,931	$ 347,742
650-MS-002 / 655-MS-002	Seawater Pond Barges	316 Stainless Steel Pump Barge Capable of Supporting (2) Pumps/Motors; Standoff legs for pump protection on dry land. Fiberglass handrail with (1) gate.	N/A	1	$ 162,220	$ 162,220
650-PP-003/004 655-PP-003/004	Seawater Pond barge pumps	14/ x 12 Vertical turbine pumps; 1-stage, 316 SS, one operating, one standby	200 HP ea.	2	$ 120,570	$ 241,140
670-WT-001 & 002	RO Desalination Plant	4.5m^3/hr. capacity	40	1	$ 154,244	$ 154,244
700-SG-001	15 kV Switchgear	7 circuits	N/A	1		$ 400,000
700-TX-001	Main Substation Transformer	240 kV/13.8 kV; 16 mVA	N/A	1		$ 500,000
710-CS-001	Circuit Switcher	On incoming power line	N/A	1		$ 127,000
907-EQ-001	Manual Truck wash Package	High pressure and high flow pumps, oil separator, tanks, spray monitors, and pressure sprayers				$ 168,560

17.4 PLANT RAW MATERIALS

17.4.1 Ore

The plant will produce 1,000,000 mtpy of DAPR with a grade of either 24 percent or 28 percent P_2O_5. The product grade will be selected based on the availability of mined ore or market conditions. Both grades will be produced by using different blends of ore and by using the same equipment at different operating conditions. A stockpile will hold 14 days of feed material.

For 1,000,000 mtpy of 24 percent P_2O_5 DAPR 2,640,740 mtpy of dry ore will be processed having a grade of 11.43 percent P_2O_5, 6.37 percent water soluble salts (WSS) and 30 percent moisture.

For 1,000,000 mtpy of 28 percent P_2O_5 DAPR 2,783,733 mtpy of dry ore will be processed having a grade of 14.26 percent P_2O_5, 3.86 percent water soluble salts (WSS) and 30 percent moisture.

17.4.2 Water

The plant will require 1,591 cubic meters per hour of seawater. It will be stored in a pond with 24 hours of retention time. A desalination plant on site will produce 2.9 cubic meters per hour of desalinated water for the laboratory and employees' personal use.

Recycling secondary cyclone overflow to the drum washer instead of sending it to the tailing pond lowers the required seawater by 400 cubic meters per hour.

It is not practical to reclaim tailing water with a thickener due to the low settling rate of the very light fine particles. The soluble salt is dissolved and pumped with the tailings to the TSF preventing the process water from becoming saturated with salt.

17.4.3 Electrical

Electric power is delivered from a nearby 138 kV transmission line to a substation on site.

17.4.4 Fuel for Dryer

Diesel fuel and gasoline arrives on site by truck. Liquefied natural gas (LNG) for the dryer arrives on site by truck.

17.4.5 Air

A plant air compressor will deliver 400 ACFM at 125 psi of plant air and instrument air for the plant.

18 PROJECT INFRASTRUCTURE

The infrastructure for the Bayovar 12 Project includes site access and concentrate haulage roads, a port facility for shipping concentrate overseas, power supply and a new power transmission line, a seawater pipeline for process water supply, ancillary building facilities, a reverse osmosis water treatment plant to deliver desalinated and potable water for human consumption, fire protection and sanitary septic facilities, site communications, and the Tailings Storage Facility (TSF).

Several of these project components are described in more detail in other sections of this report, and only a general description of the relevant aspects of the project infrastructure-related components is given here.

18.1 TRANSPORTATION

18.1.1 Site Access and Site Roads

The Bayovar 12 mining concession is crossed by the Pan American Highway on its eastern side (Figure 18-1) which connects the city of Piura to the north with Chiclayo to the south. The Chiclayo-Bayovar Highway is a two-lane paved road that connects to the Pan American Highway to the Port of Bayovar to the west and this road runs diagonally across the concession. Access to the Bayovar 12 plant is from the Chiclayo-Bayovar Highway, which is located approximately 17 km west of the Pan American Highway. Administration facilities are located 100 meters south of the Chiclayo-Bayovar Highway.

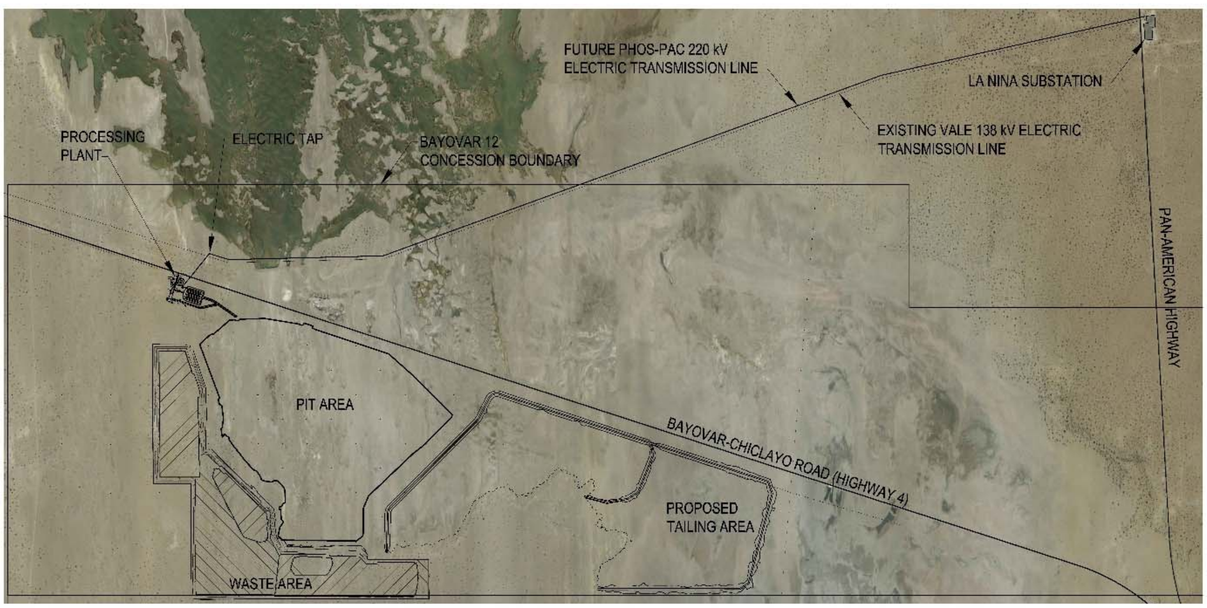


Figure 18-1: Overall Site Plan

Transportation to and around the site will be unpaved roads developed to accommodate the demands of the project. The soft overburden lends itself to low cost road building with graders, dozers, and scrapers without any need for blasting or supporting MSE walls. Culverts will need to be installed where washes cross the roads.

18.1.2 Concentrate Haulage

The proposed plan for the Bayovar 12 mine is to haul concentrate to the Port of Bayovar along the Chiclayo-Bayovar road over a distance of 45 km to the port facility owned by JPQ shareholders. Currently, JPQ hauls gypsum from the

Bayovar 12 concession using trucks to its port for shipment to plants in Ecuador. JPQ currently produces 80,000 tonnes of gypsum per year from the concession.

A dedicated contract concentrate hauler has estimated the haulage rate of $5.09 per tonne. The trucks that will be 35-tonne end-dumping trailers. The round-trip to the Port of Bayovar including loading, transporting, unloading, returning, and driver breaks is estimated to require 3 hours, based on an over-the-road speed of 45 km/hr.. For 1 million tonnes of concentrate per year, 28,575 round-trips will be required annually which translates to 78 truckloads per day, based on a 365-day year. Currently, FCV has established that concentrate hauling will be conducted during during daylight hours on a12-hour day shift basis, only. For this reason, 100 percent truck availability, since truck maintenance will be handled on the off-shift. At this rate, FCV will need to ship 6.5 truckloads of phosphate concentrate per hour which extends to approximately 20 trucks running.

18.2 PORT FACILITIES

The project is planning to use the port facilities owned by its partner, JPQ, located in the Port of Bayovar (Figure 18-2). The facility is a medium depth port that handles small Handysize ships that have a capacity of 20,000 long tons deadweight (DWT) – 28,000 DWT and Handysize ships, with a capacity of 28,000 – 40,000 DWT. The port has been actively shipping gypsum from open stockpiles.

Figure 18-2: Port Image

JPQ solicited an engineering report from Consultora e Inmobilaria Volcan (CIVSA) of Arequipa, Perú for improvements to JPQ's port facility so that it could handle DAPR phosphate concentrates delivered from the Bayovar 12 mine. The improved port facility would include a scale house, a 45,000-tonne warehouse for keeping phosphate concentrates dry and protected from the wind, a feed bin, a loading conveyor, a ship loader, a truck wash for exiting concentrate trucks, ancillary buildings, and sanitary sewer facilities (Figure 18-3). The concept for the port is that a 35-tonne truck passes the security control gate, is weighed on the incoming scale and dumps into the warehouse. Upon exiting the port, the empty truck is weighed on the outgoing scale to record the weight of the load delivered. A front end loader scoops phosphate concentrate into a 15-tonne capacity dump truck which transfers the material to the feed bin. The loading conveyor has a capacity of 500 tonnes per hour.

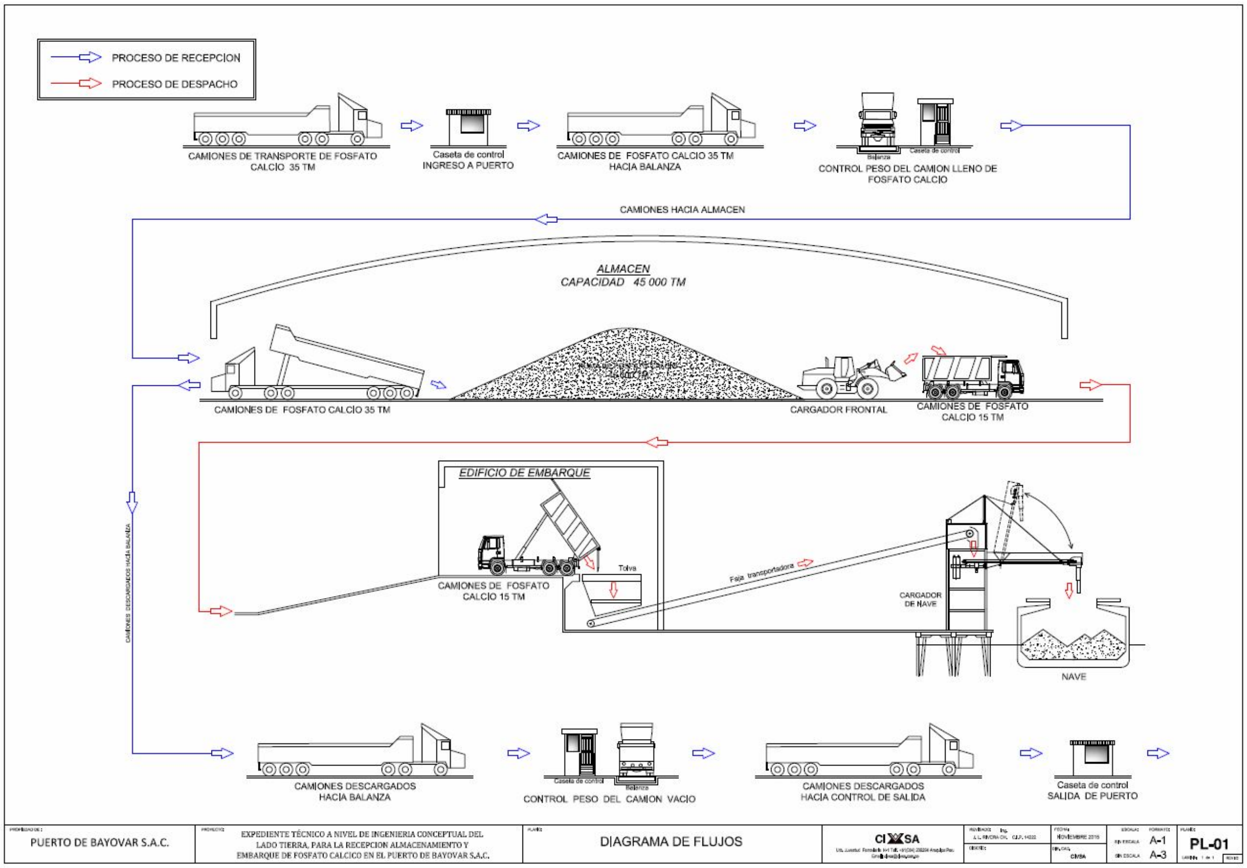


Figure 18-3: Port Facilities Flow Diagram

Figure 18-4 is a conceptual general arrangement prepared by CIVSA for the study. CIVSA's design can handle reception of 210 tonnes per hour equaling 6 trucks per hour. The warehouse will be a covered facility that is equipped with dust collection. The feed bin will be housed at the base of a small building into which the 15-tonne trucks will dump the concentrate. The feed belt conveyor to the ship loader will be covered to eliminate phosphate dust from falling into the harbor. The feed belt conveyor will dump onto the ship loader belt that will load the Handy size ships. The ship will have to be moved manually to spread the load in the ship's hold, as the ship loader stinger, or loading spout has a fixed position.

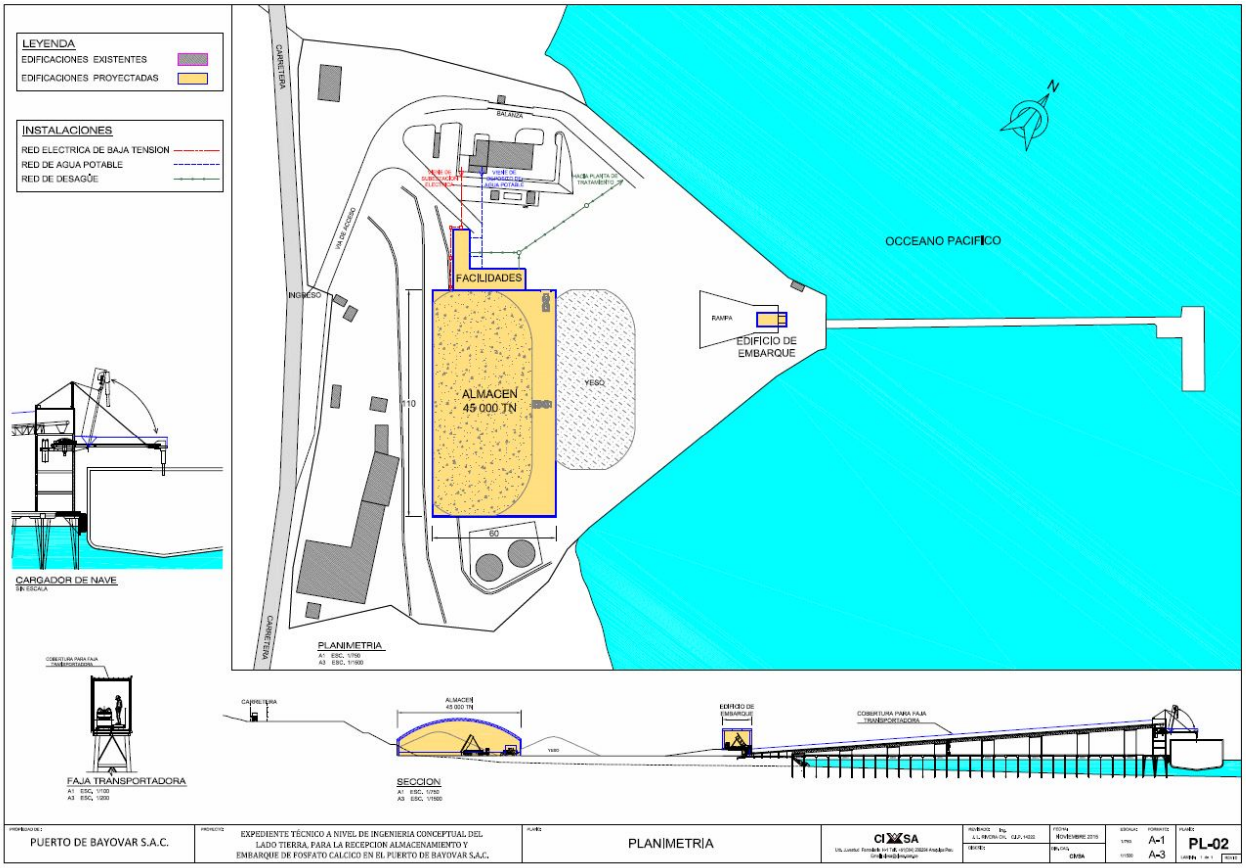


Figure 18-4: Proposed Improvements to Port Facilities

Port power distribution will tap into an existing 22.9 kV power line and transform to low voltages for operations.

The port improvement execution schedule is estimated to take 12 months of construction, the cost will be borne by JPQ and recovered through port usage costs.

18.3 POWER TRANSMISSION LINE AND SITE POWER

Utility power is available from a 220 kV power transmission line which runs parallel to the route of the Pan American Highway. To supply power to the Bayovar 12 plant, a 138 kV 16-kilometer power transmission line from the La Niña substation to the Bayovar 12 main substation will need to be constructed (Figure 18-1). Power would be stepped down at the plant site to the distribution voltage of 13.8 kV.

The Bayovar 12 power transmission line route will parallel the transmission line that supplies power to Vale's Bayovar mine (Figure 18-5), skirting the southern edge of the El Niño floodway embankment and then crossing the Bayovar-Chiclayo highway to the plant.

Figure 18-5: Existing Power Transmission Line to Vale's Bayovar Mine

An opportunity exists for FCV to share a 138 kV power transmission line needed to serve the FOSPAC plant. FCV has had preliminary discussions with FOSPAC who are amenable to sharing costs for infrastructure. The power line would run along the same route as the proposed Bayovar 12 power line. In this case, FCV would tap the FOSPAC 138 kV line at a disconnect switch, and install a 700 meter transmission line to the Bayovar 12 substation. The savings from sharing the power transmission line with FOSPAC would be on the order of $2 million in initial capital.

18.4 SEAWATER SUPPLY PIPELINE

The Bayovar 12 process plant operates on seawater that will be pumped at a flow rate of 1,591 m^3/h. The delivery point for the seawater is the plant seawater pond from which process water will be pumped to the various unit processes. A small stream from the seawater pond is taken to a reverse osmosis (RO) desalination plant, whose brine product will be discharged back into the seawater ponds. Filtrate from the vacuum belt filter will also be discharged to the seawater ponds. There are no plans at this time to reclaim supernatant water from the TSF.

The seawater inlets will be located along the ship loading wharf at JPQ's facility in the Port of Bayovar. The seawater intakes will be omni directional heads fitted to the upstream end of the intake system pipeline located some distance into the sea from the shoreline. Entrance velocity is expected to be lower than 0.091 m/s. Eddies through the intake are eliminated thus head loss at the seawater intake is reduced to a negligible level – < 0.2 millimeters.

The seawater intakes will be piped to a bank of four, 250 HP, corrosion resistant horizontal centrifugal pumps that will be mounted on shore at a pump station at the port loadout. The seawater intakes will be mounted from a platform from which piping will run to the pump station. Intake pipes will be coupled to a header that that is downstream of the pump station.

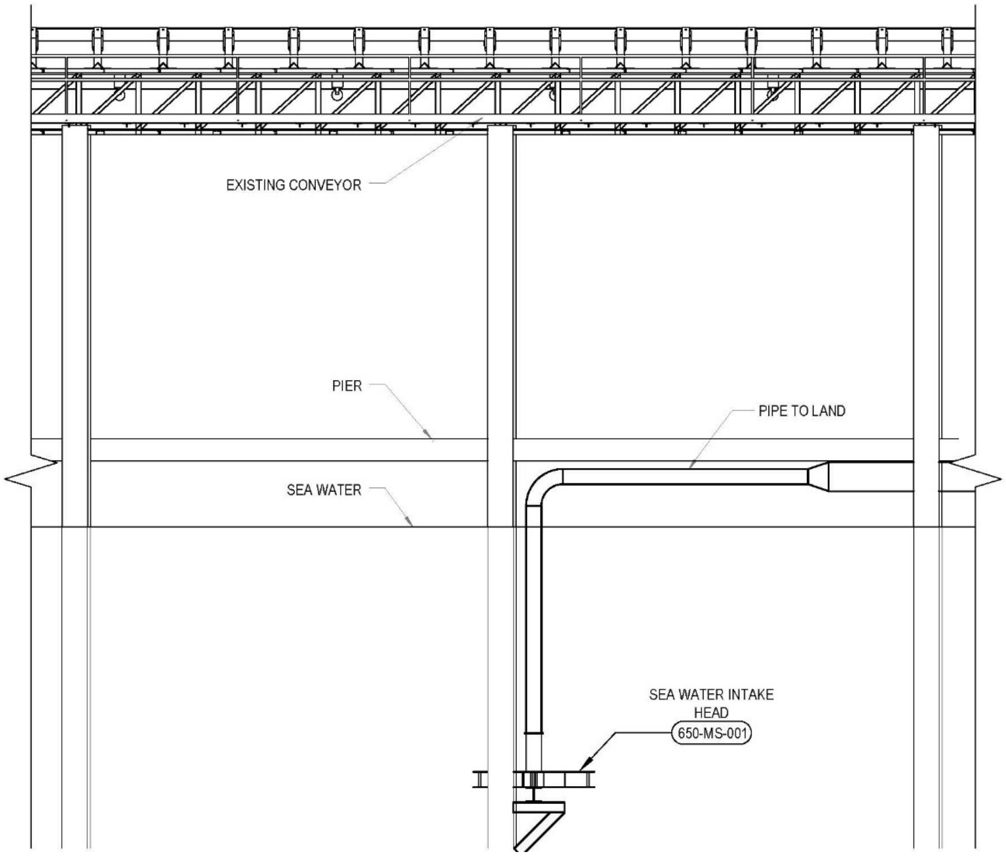


Figure 18-6: Seawater Intake

The overland route for the seawater supply pipeline will follow the Bayovar-Chiclayo highway along the south side of the road. The route, which is 45 kilometers long, is relatively flat (Figure 18-7) reaching a maximum elevation of 62 meters amsl. The triangle indicates the location of the proposed plant site on the tablazo at an elevation of 32 meters amsl.

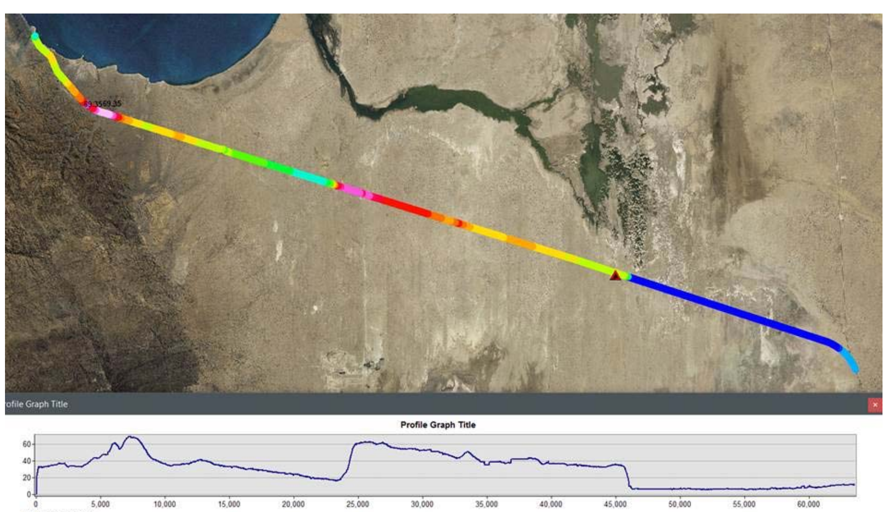


Figure 18-7: Elevation Profile of the Proposed Seawater Supply Pipeline to Bayovar 12 Plant

The seawater supply pipeline will consist of 32-inch HDPE pipe using the appropriate pressure ratings (SDR 11, 13.5 and SDR 17) that correspond to elevations. The pipeline will be laid in a trenchand covered. The design will include expansion capacity and air vacuum release valves to prevent pipeline collapse.

FCV is investigating using TUBI technology, self-contained, mobile pipe extrusion station to fabricate HDPE pipe onsite or along the pipeline route as it is laid. A TUBI system can produce lengths of pipe up to 1000 feet that are longer than commercially available 40 and 50-foot lengths. Benefits of TUBI technology include: reducing freight, reducing the number of field welds which leads to reduced handling steps and installation time, and includes 100% X-ray quality control technology.

The seawater supply pipeline will initially discharge into the Seawater Pond which measures 100-m long by 75-m wide by 9.8-m deep. The pond will be designed with a partition so that maintainance and cleaning can be performed on on side of the Seawater Pond while the other side is operationg.

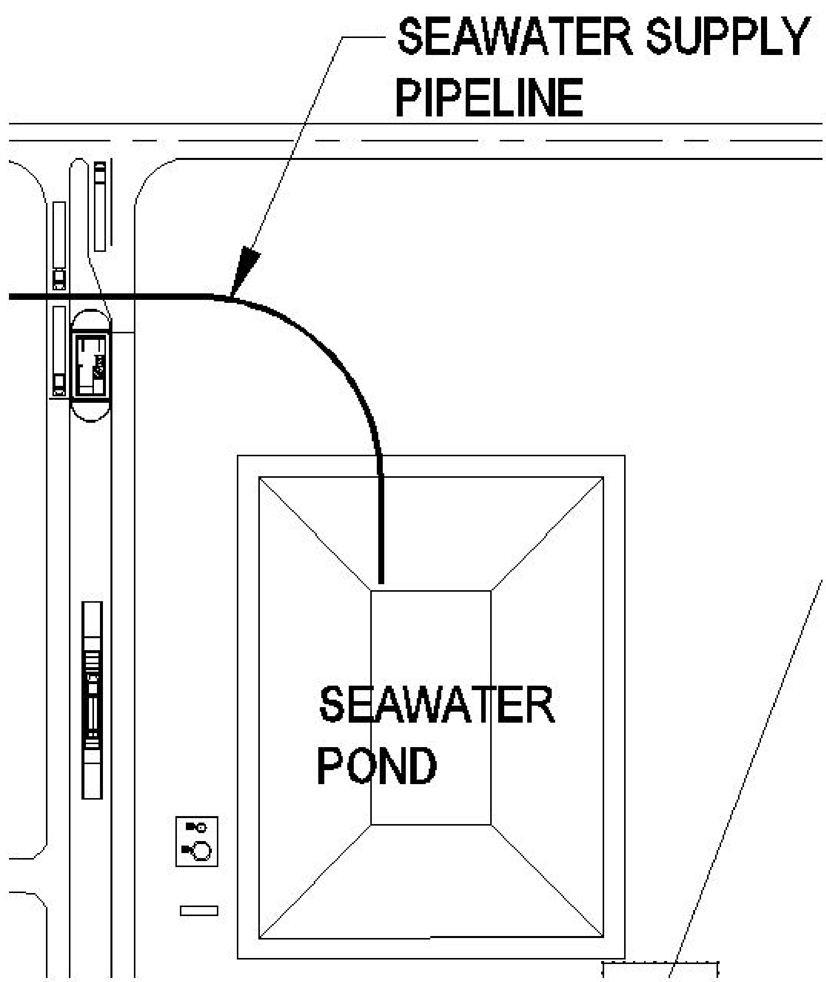


Figure 18-8: Seawater Ponds

18.5 SITE LAYOUT & ANCILLARY BUILDINGS

Bayovar buildings and facilities are divided into four functional areas: Mine Facilities, Process Facilities, Administration, and Tailings Storage.

The mine facilities include the following:

- Truck Shop
- Truck Wash
- Warehouse
- Truck fuel storage and fueling station
- ROM Storage Pad

The process facilities include the following:

- Dump Pocket & Feed Conveyor
- Drum Washing
- Attrition Scrubbing and De-sliming
- Concentrate Belt Filtration
- Concentrate Drying and LNG Storage
- Concentrate Storage, Truck Loadout and Truck Scales
- Tailings Pumping
- Plant Maintenance Building
- Analytical Laboratory
- Process water (seawater) supply pond, RO treatment, and potable water storage area
- Electrical substation

The administrative facilities are located near the main entrance and include the following:

- Security Building
- Administration building
- Cafeteria
- Medical/Emergency Building
- Change House

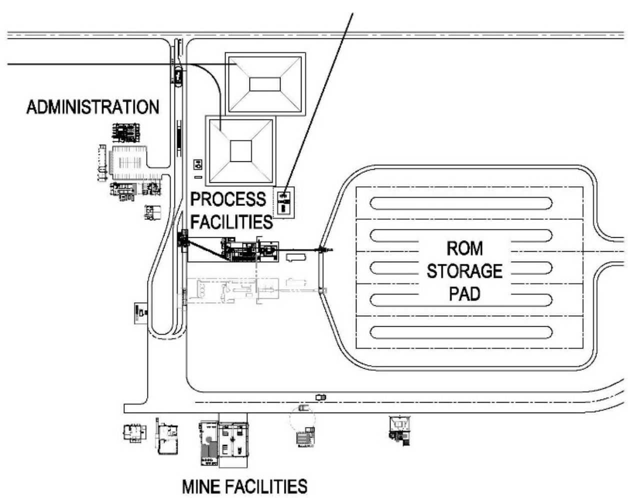


Figure 18-9: Plant Site Overview

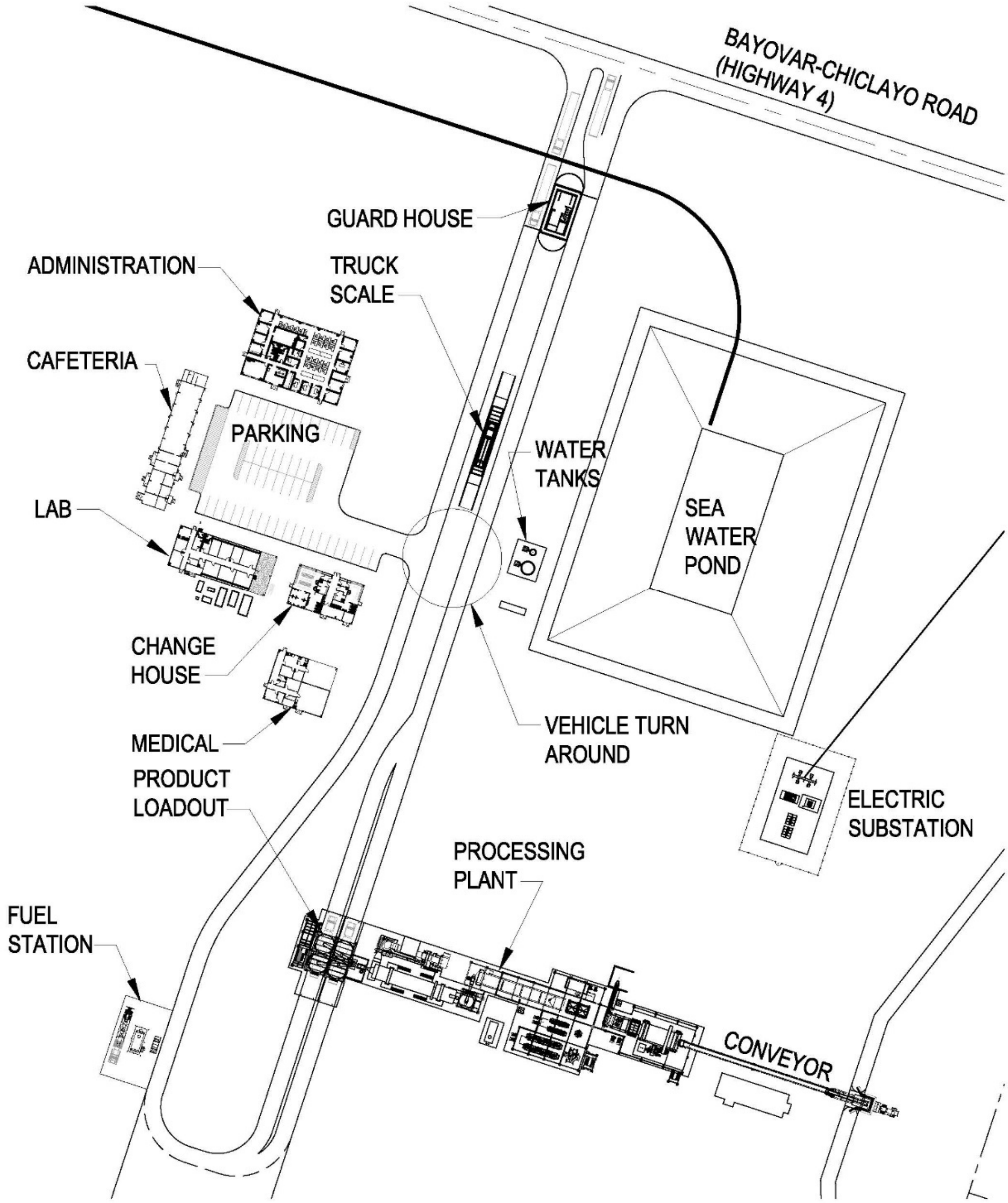


Figure 18-10: Plant Area Detail

18.5.1 Mine Services Facilities

Mine service facilities area will be located south of the main plant facilities and west of the Bayovar 12 open pit. This area includes the Truck Shop, Truck Wash, and Fuel Facility. This area will provide services and support to the mining operation and equipment fleet. The ROM pad is located directly east of the Bayovar 12 plant.

The Truck Shop building is a pre-engineered metal building that will house four haul truck maintenance bays equipped with overhead bridge cranes, a separate bay for heavy equipment such as dozers and other tracked equipment, a bay on a slab outdoors for light vehicle maintenance, offices, break/lunch room, electrical and mechanical rooms, lube room, electrical repair shop, welding/repair area, and a tank farm with containment for lubricants, fluids, and waste products. The foundation will be concrete spread footing, piers and grade beams. Concrete floor slabs will be provided over the entire area of the building, with thicknesses suitable for the offices, warehousing, and mine truck support and aprons.

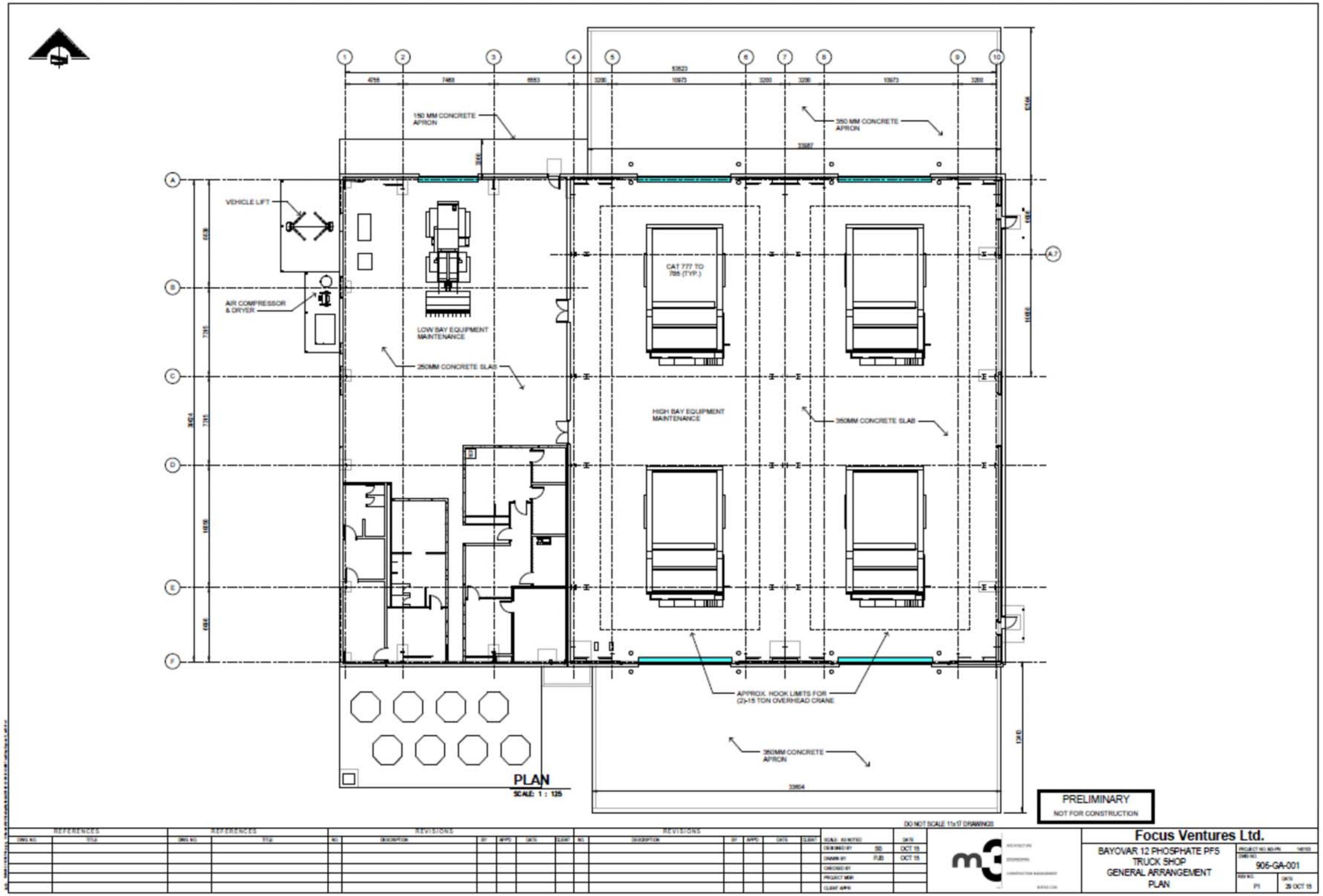


Figure 18-11: Truck Shop

An uncovered Truck Wash area is located adjacent to the Truck Shop. It will include a manual wash system for mine haul trucks; with 100% recycling system with makeup water connection. The facility will include a concrete settling basin and dirty water sump. Truck wash equipment will include a high flow pump, a high pressure pump, three high flow wash monitors, the two high pressure lances. It will include a sump pump, solids separation, oil skimmer, holding tank, water level controls, makeup water controls and air injection.

18.5.2 Process Facilities

The process plant facilities include a dump pocket & feed conveyor, drum washing, attrition scrubbing and de-sliming, concentrate belt filtration, concentrate drying and LNG storage, concentrate storage, truck loadout and truck scales, tailings pumping, plant maintenance building, an analytical laboratory, process water (seawater) supply ponds, desalination RO treatment, and potable water storage area, and electrical substation. The process areas are described in Section 17. The plant ancillary facilities are briefly described as follows.

18.5.2.1 Plant Maintenance Building

The maintenance workshop building will be a 19.0 m x 19.4 m, pre-engineered metal building with insulated roofing, siding, and a reinforced concrete mat foundation. There are areas for welding, mechanical repairs, electrical repairs, a tool crib, and a machine shop. A bridge crane is located along the mechanical bay

18.5.2.2 Analytical Laboratory

The laboratory building will be a modular structure on a reinforced concrete slab foundation. Laboratory facilities include sample receiving and storage, sample drying, sample preparation, analytical equipment, a balance room, electrical and mechanical rooms, a computer room, men's and women's restrooms, a break room, a loading dock, and two offices. The building will be air conditioned, and fume extraction and dust collection equipment is provided.

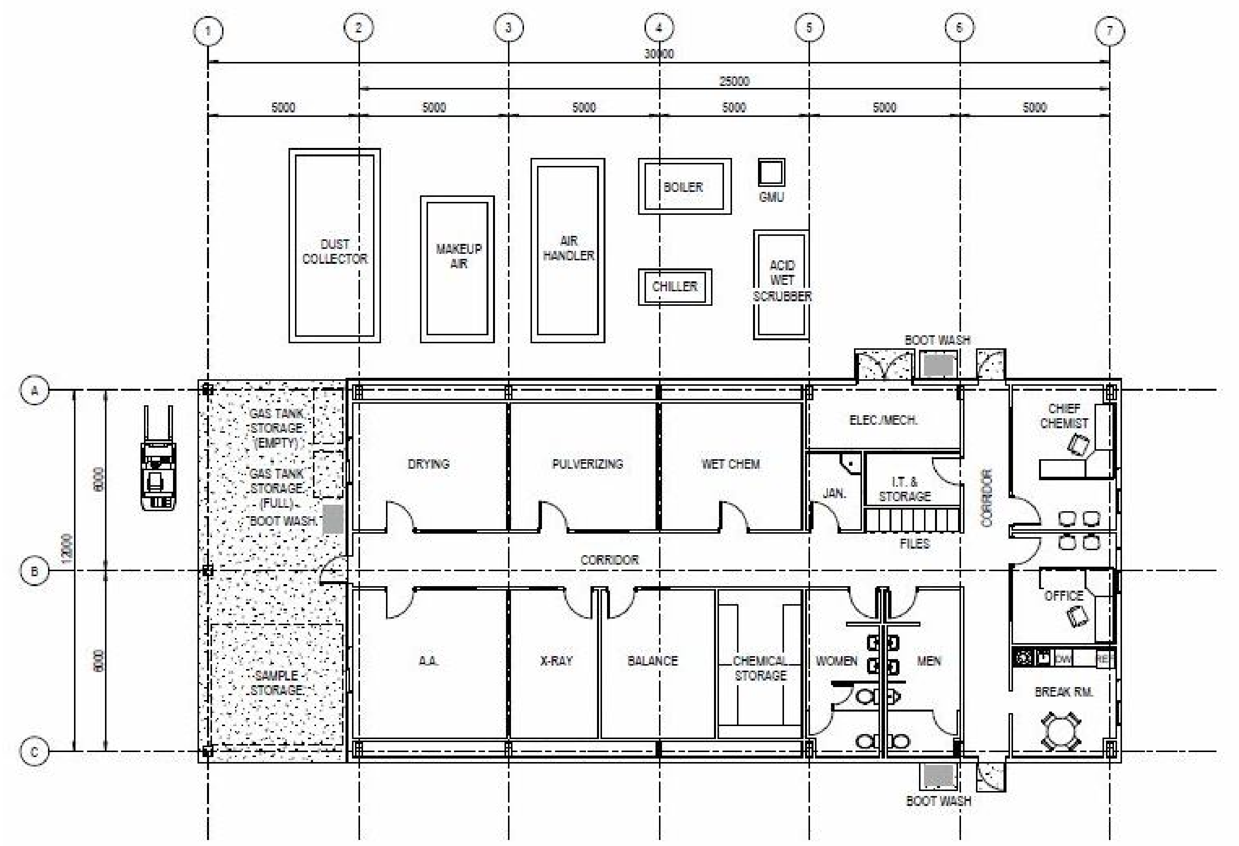


Figure 18-12: Laboratory

18.5.3 Administration Facilities

The administration facilities are located near the main entrance. The administration facilities include the main gate and security building, the Administration building, the cafeteria, the medical emergency building and the change house.

18.5.3.1 Security Building

The main mine security office will be located at the entrance to the mine site about 40 meters south of the Bayovar-Chicalyo Highway. The guard house will be a prefabricated, 12-m x 6-m metal building contain a security office, a truck scale office, a restroom, a small break room area, and utility areas.

A truck scale is located nearby to weigh loads of supplies coming into the site and concentrate trucks leaving the site. The truck scale is 21 meters long, not including 10.5 meters of apron and ramp on each end of the scale. The scale is a steel deck that is capable of weighing the 35-tonne concentrate trucks. Personnel in the security building will monitor the full and empty weights of concentrate and delivery trucks to the site.

18.5.3.2 Administration Building

The Administration Building is planned to be a modular 30-m x 20-m complex of offices, break and conference rooms an area for work stations, restrooms and rooms for utilities and communications. This facility will be air-conditioned and insulated and installed on a concrete slab foundation. A gravel-surfaced parking area for employees and visitors is provided. The offices will be used for senior personnel, records and archives, accounting, and engineering.

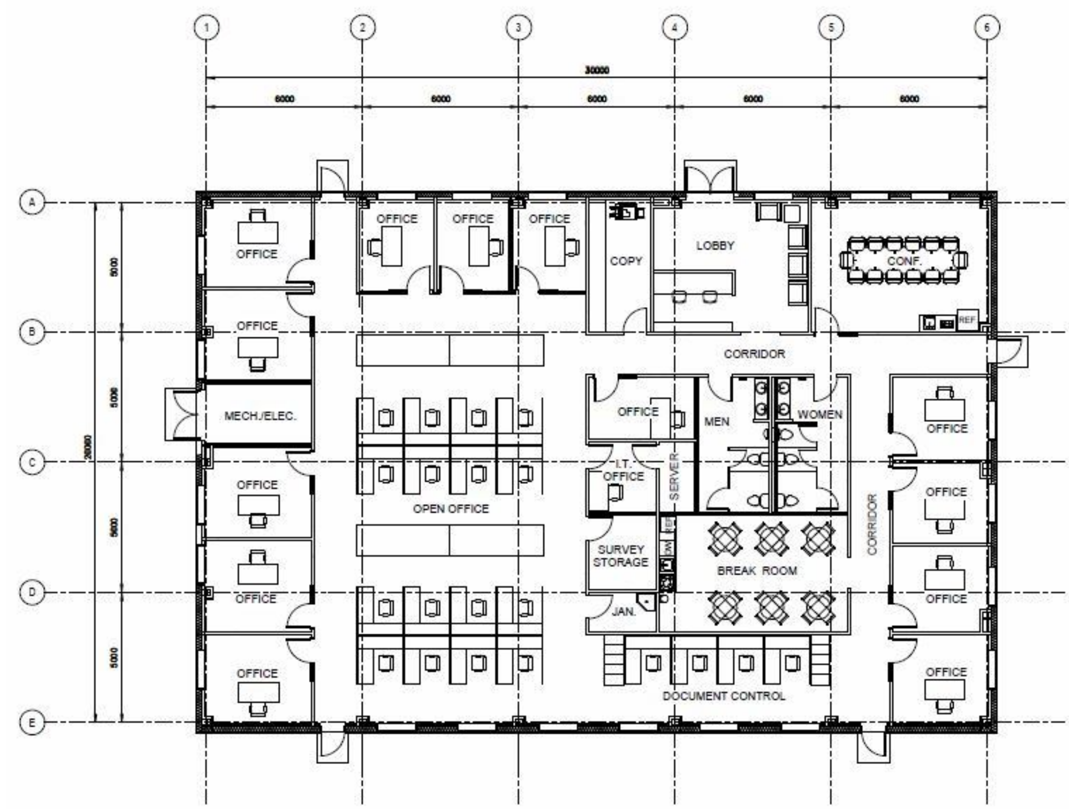


Figure 18-13: Administration Building

18.5.3.3 Cafeteria

The operating plan for the Bayovar 12 mine includes utilizing four crews to cover 12-hour shifts, 24 hours per day, 360 days per year. While no camp is necessary for housing employees, a full service cafeteria has been included to serve miners, plant personnel, and administration personnel. The cafeteria is located adjacent to the Administration Building and contains seats for 120 people. The building contains a kitchen, cold and dry storage, restrooms and utility rooms.

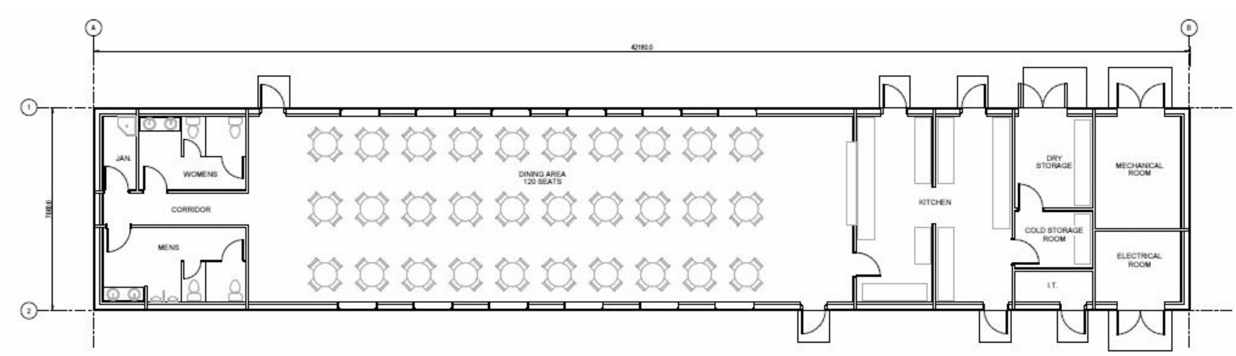


Figure 18-14: Cafeteria Building

18.5.3.4 Medical/Emergency Building

The Medical/Emergency Building is an 18-m x 15-m modular building with first aid, exam, and nursing rooms. There is a slab for parking the site ambulance. This building also houses the health/safety department with offices for supervisor, a trainer, PPE storage and a training room with seating for 30 site personnel.

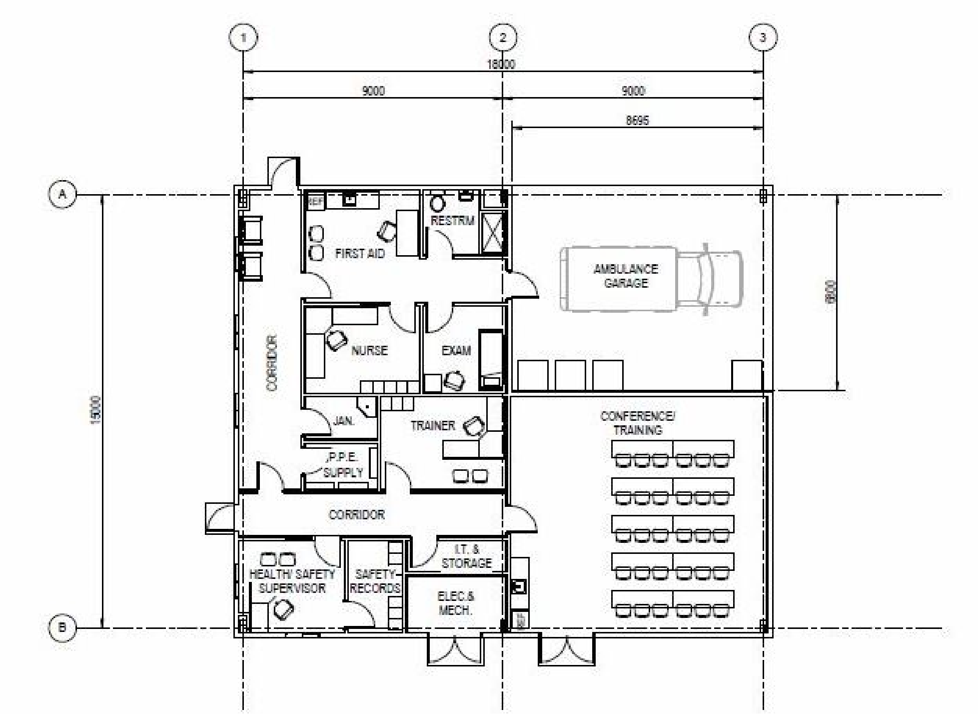


Figure 18-15: Medical Building

18.5.3.5 Change House

The Change House is a 20-m x 12-m modular building with locker rooms for men and women, lockers for mine and plant supervisors, restrooms and showers, and uniform service.

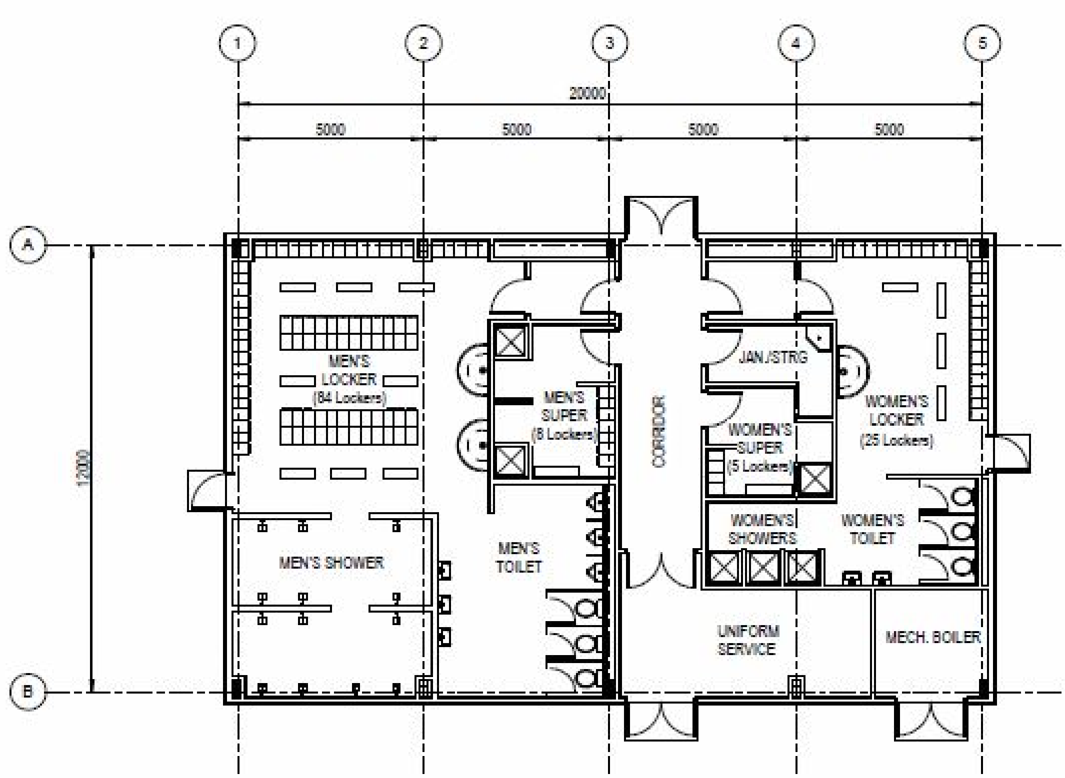


Figure 18-16: Change House

18.6 REVERSE OSMOSIS DESALINATION PLANT

A packaged 4.5 m3/h (20 gpm) reverse osmosis plant has been included to supply desalinated potable water for human use for the operation. This plant can produce 108 m^3/d. The sizing of the RO plant is based on a consumption of 100 gallons per day per person for approximately 300 people on site per day at full production. Each RO plant is a single pass, 8-membrane system based on a fresh water (permeate) recovery of 50% and a salt rejection of 99.5%.

Fresh water will be stored in a potable water tank after it has been treated chemically and with ultraviolet light for human consumption. Fresh water will be used for cooling water for the vacuum pumps for the belt filter. Cooling water for plant equipment will be cooled with a chiller utilizing a closed water-glycol circuit running through a heat exchanger.+

Figure 18-17: Reverse Osmosis Unit

18.7 FIRE PROTECTION & SANITARY SEPTIC FACILITIES

Fire protection will be achieved with a skid mounted, dual pump fire water system. The pumps, one electric and one diesel (back-up pump) are sized to deliver 227 m3/h (1,000 gpm) for 120 minutes. The firewater pumps will draw directly from the seawater ponds. A buried firewater loop serving process facilities, mine facilities, and ancillary facilities will provide water to hydrants at a spacing of approximately 100 meters.

Two septic sewer facilities have been designed for the plant site. Each will consist of a septic tank and a leach field. The larger system (3,000 gal tank) will serve the Administrative Facilities. The second (1,000 gal tank) will serve the Mine Facilities and the plant control room.

18.8 SITE COMMUNICATIONS

Site telephone and internet services will be provided by local providers. It is anticipated that the Bayovar 12 site will use physical cable for telecommunications, and for internet communications. The hub for site communications will reside in the Administration Building with fiber optic communications to the plant facilities.

The mine radio system will include one base station and a control-tower station at the mine from which all mining equipment and haul trucks will be dispatched and controlled, and a number of repeater stations will be installed. One station will provide coverage to the tailing area, and others are required to extend coverage throughout the mine site and to the Port of Bayovar.

Telephone and data communications including voice, data and internet communications will be provided for the mine site and the Port of Bayovar. The communications system will connect to a central communications center, which will include a telephone/fax PBX and network servers for email, internet and data services. Other network servers to manage site operations and for data storage will also be located in the central communications center, with the exception of the process servers which will be located at the processing facility. The mine site telephone system will link all essential areas of the site together, and through the satellite system, to outside of the project site.

All vehicles will be equipped with radios and essential personnel will have hand-held radios. Key personnel will also be equipped with mobile telephones. Cellular phones will have coverage to the Port of Bayovar as a safety precaution.

18.9 TAILINGS DISPOSAL

The Bayovar 12 TSF will consist of approximately 9.8 Mm2 of available storage area as shown in Figure 18-18. The TSF is designed so that tailings can be impounded to a maximum height of 6.0 m, with spillway discharge and 1.0 meter freeboard.

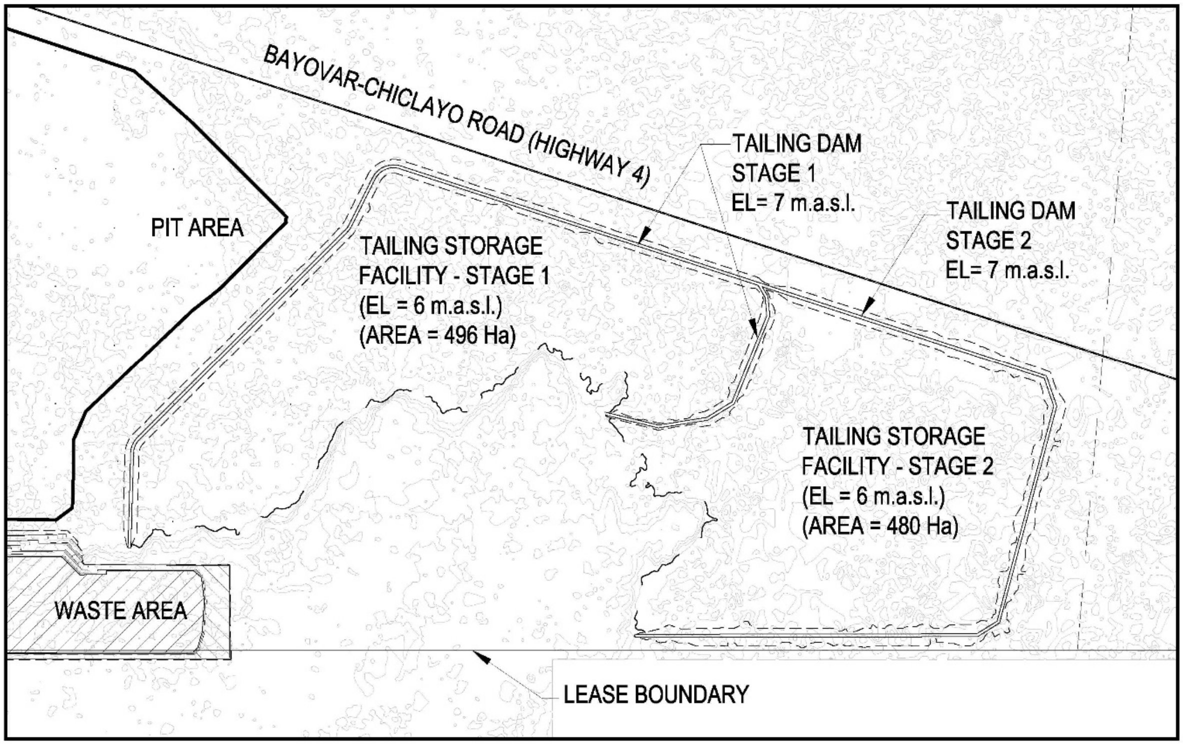


Figure 18-18: Tailing Storage Facility

The containment embankment cross section (refer to Figure 18-19) consists of the open pit waste spread and compacted with 5:1 (H: V) side slopes, and a 20.0-meter wide crest, and a geotextile lined downstream face for erosion control. The TSF is expected to have the capacity for 58.8 million cubic meters of tailings, an average loading rate of approximately 2.5 million cubic meters per year (Mm3/yr), and an operating life of 20 years.

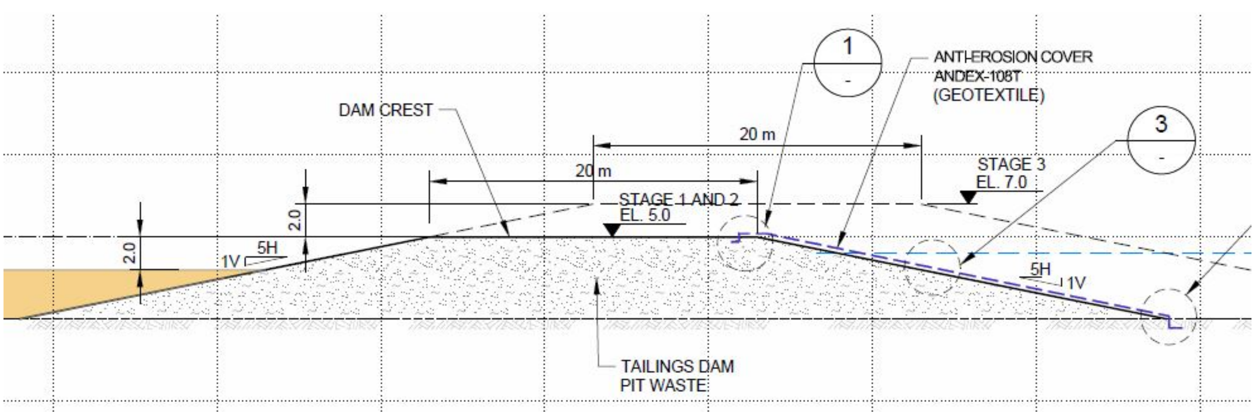


Figure 18-19: Tailing Embankment Cross Section

The TSF is located partially in the floodway of a potential El Niño extreme event so the dam for the TSF will be constructed to the planned height shown on the drawings.

Operational criteria applied as a basis for the TSF layout and sizing included a tailing delivery rate of 2.5 Mm3/yr for a total projected tailing production volume of 49.8 Mm3. The tailings are delivered in a slurry with 11% solids. The containment accounts for natural passive evaporation.

A Stage 1 TSF with a surface area of 5.0 million square meters (Mm^2) is designed for containment and evaporation of six years of production. The Stage 2 TSF adds 4.8 Mm^2 of surface area for subsequent year's production.

The dam foundation design will be supported on cohesive silts and clays. The groundwater level was recorded at 5 m below the ground surface,

<u>Tailings Water Management System</u>

The proposed surface water management system is that the runoff generated in the upstream catchment area discharges into the TSF. Considering the large evaporation rate and the large extent of the TSF, it is expected that this runoff can be evaporated to the environment.

During an El Niño event, the runoff generated can be discharged into the TSF. In this case, a spillway across the crest of the north dike is proposed to control overflows during extreme events.

18.10 SURFACE WATER MANAGEMENT

The climate in the Sechura desert in the Bayovar 12 area is dry and windy. Surface water diversions will be necessary for occasional rains and during El Niño years. Most surface water will report to the open pit where it will be channeled to collection sumps from where it will be pumped to the seawater or tailings ponds.

Diversion ditches and associated culvert systems, as well as ponds, sumps and pipelines, will be designed to address the majority of surface water flow at the project site. Water diversions will also be needed along the length of the south side of the Bayovar Chiclayo highway to protect the berms of the TSF from erosion to prevent breaching of the slimes in the impoundment

19 MARKET STUDIES AND CONTRACTS

19.1 OVERVIEW

The Bayovar 12 project, located within the Sechura basin in northern Peru, hosts a major deposit of what is arguably the most reactive sedimentary phosphate rock fertilizer in the world (Reactive Phosphate Rock or “RPR”). This project’s relatively simple mine plan and flow sheet will produce a natural phosphate fertilizer that is suitable for use as direct application phosphate rock (DAPR) that can be applied directly to many soils without the need for conventional acid pre-treatment of the rock or the addition of other manufactured chemical compounds. DAPR is most effective when used on acid soils such as those which form the bulk of tropical South American soils[1]. Agronomic research has demonstrated that Sechura RPR, due to its high reactivity, can outperform more expensive and non-organic single- and triple-superphosphate fertilizers when used in tropical soil and climatic conditions (*e.g.*Bolland and Gilkes, 1981[2]).

The deposit is located near tidewater in an established phosphate mining district in Northern Peru on the doorstep of a large and growing market for fertilizer products. The region is witnessing a rapid expansion of farming particularly for oil palms -an increasingly important cash crop - and organically cultivated food products. The favorable dissolution kinetics of DAPR have already made it the preferred fertilizer for oil palm plantations in Malaysia and Indonesia and it is also a natural fit for plantations of the region where it is located.

The project will produce a highly sought-after, organic DAPR fertilizer. This product will service the growing oil palm market in the Americas, especially the organic palm oil plantations of Colombia and Ecuador, including the local Peruvian market where phosphate nutrient needs are currently met entirely by imports. In addition, by achieving organic certification of its products, Focus aims to become the supplier of choice of natural phosphate fertilizer for organic producers throughout the Americas where 18% of the world's organic farmland is located.

19.2 INTRODUCTION

This preliminary market analysis is based on:

1. Marketing reports commissioned during 2015 by Focus Ventures;
2. Internal research into recent sales prices achieved for DAPR in the Americas;
3. A short study commissioned by Focus on making blended fertilizer products using Bayovar RPR, and;
4. An internal report on the Peruvian fertilizer market based on Peruvian government statistics.

Concentrate production will be phased in over three years with initial production of 300k tpa in year 1, rising to 1,000k tpa by year 4 for a mine life of 20 years. 249 M tonnes of Measured and Indicated Resources remain for extraction after the initial 20 year mine life.

The process plant has been designed to produce two natural phosphate rock fertilizer products: a 24% P_2O_5 RPR for use as an organic DAPR and a higher grade + 28% P_2O_5 product suitable for blending with other natural nutrients to make customized multi-nutrient organic fertilizers.

[1] http://nelson.wisc.edu/sage/data-and-models/atlas/maps/soilph/atl_soilph_sam.jpg
[2] *Bolland, M., and Gilkes, R., The agronomic effectiveness of reactive phosphate rocks 2. Effect of phosphate rock reactivity. Australian Journal of Experimental Agriculture, 37, 937-46*

The project is located close to the tropical regions of western Central and South America, on the doorstep of a large and growing market for fertilizer products. It is anticipated from the results of published research studies that both of these products will be able to outperform the highly-soluble manufactured chemical fertilizers in the acid soils of the targeted markets.

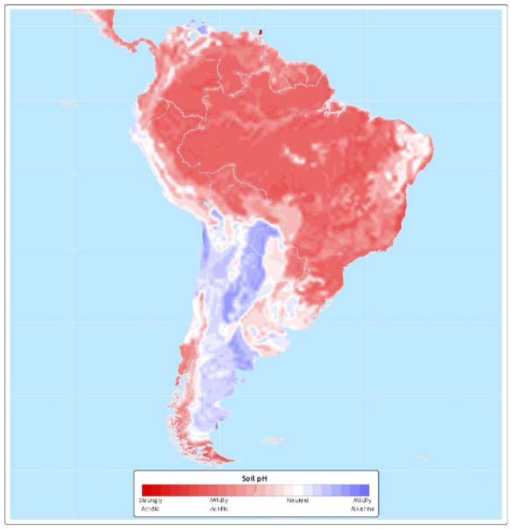


Figure 19-1: Distribution of Acid Soils in South America

19.3 KEY MARKETS

19.3.1 Oil Palms

Palm oil recently displaced soya as the most consumed edible oil worldwide. In 2014 over 50 million tonnes of palm oil were produced. The growth in palm oil production has been rapid because it is a high-yield, low-cost crop which produces over 5 times as much oil per hectare as other oleaginous species. The oil also has a greater variety of uses than competing crops and is used in the production of soaps, cosmetics, ice cream, chocolate bars *etc*.

Oil palms only grow in tropical regions and thrive on acidic soils which are typically low in nutrients and require significant fertilizer use[3]. In an extensive study by Ng *et al* (2013) of Indonesian and Malaysian plantations -where the majority of the world's oil palm plantations are located the application of Bayovar DAPR optimized palm nutrition and oil yields. In Southeast Asia it remains the preferred oil palm plantation fertilizer and commands premium pricing. Depending on plant maturity and soil characteristics optimum annual application rates of natural phosphate rock concentrate range from 300 kg to 1 tonne per hectare.

A rapid expansion is underway in the Latin American oil palm sector. For example, in 1999 there were roughly 150,000 hectares under cultivation in Colombia which had grown to 477,000 by 2013[4]. The Colombian government is targeting continued expansion to 3-million hectares of plantations by the year 2020 – a + 6 -fold increase over 2013 levels.

In Ecuador, oil palm cultivation has grown by 7% per year for the past 10 years with over 250,000 hectares in production by 2012. In Peru, oil palms were first planted at Tochache in the Department of San Martin in the Peruvian Amazon in 1973. By the year 2000, nearly 15,000 hectares were planted with oil palm[5]. By 2012, the area under cultivation had

[3] *Patrick Ng, Kah Goh and Zaharah, 2010, Direct applications of phosphate rocks on sustainability of oil palm plantations.*
[4] *http://www.cifor.org/publications/pdf_files/OccPapers/OP-122.pdf*
[5] *http://www.cifor.org/publications/pdf_files/OccPapers/OP-122.pdf*

grown to just under 60,000 hectares. Today, another 100,000 to 200,000 hectares of plantations are proposed or planned for the region of Loreto in northeastern Peru.

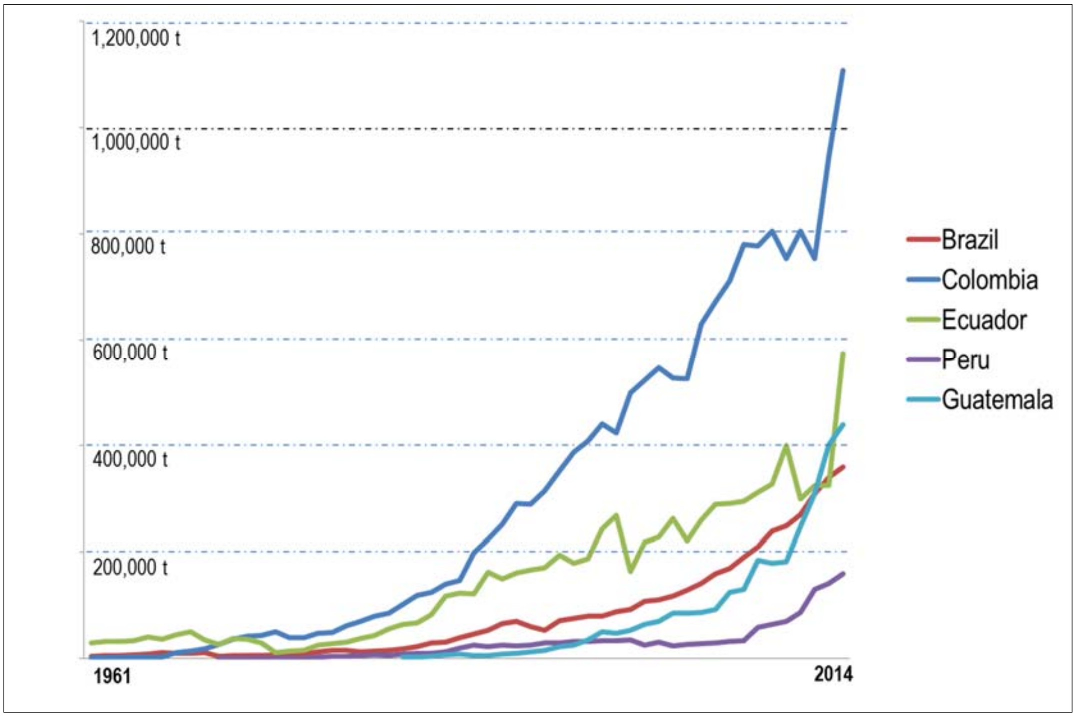


Figure 19-2: Latin American Palm Oil Production 1961-2014[6]

19.3.2 Organic Farms

Much of the expansion in oil palm cultivation in Central and South America focused on the production of organic palm oil. This will be an important part of Focus' long term marketing strategy.

Phosphate rock from Bayovar is a natural mineral product. This study contemplates that it will be mined and processed without the addition of, or reaction with, any other chemical compounds which will allow Focus to obtain organic certification for its products.

Organic farming represents an increasingly large share of the world's agricultural industry. Since 1990 the market for organic food and other products has grown rapidly, reaching $72 billion worldwide at the end of 2013[7]. The area of organically managed farmland globally grew at a compound rate of 8.9% per annum between 2001-2011. As of 2013, approximately 43 million hectares worldwide were farmed organically. Consumer demand for organically produced goods has grown by double digits during most years since the 1990s, providing market incentives for U.S. farmers

[6] *Food and Agriculture Organization of the United Nations http://ensia.com/features/can-latin-america-do-palm-oil-right/ http://www.fibl.org/en/media/media-archive/media-release/article/growth-continues-global-organic-market-at-72-billion-us-dollars-with-43-million-hectares-of-organic.html*

[7] *http://www.fibl.org/en/media/media-archive/media-release/article/growth-continues-global-organic-market-at-72-billion-us-dollars-with-43-million-hectares-of-organic.html*

across a broad range of products, and now represents between 4-5% of U.S. food sales: close to a US$30 billion[8] market. Eighteen percent of the world's organic farmland is in Latin America[9].

Focus plans to produce large quantities of product with a consistent and predictable chemical composition and P_2O_5 percentage, which demonstrates highly-soluble chemical behavior in acidic soils. It is one of only a few sources of natural phosphate nutrient available for large scale organic farms and plantations. Numerous papers that have highlighted the excellent agronomic effectiveness of phosphate rock from the Sechura district are listed on the following link; http://www.focusventuresltd.com/i/pdf/Sechura_RPR_References.pdf

Focus will be seeking organic certification of its products as soon as possible after production has commenced, and will be targeting west coast Latin American and North American organic producers.

19.4 PERUVIAN FERTILIZER CONSUMPTION

The fertilizer market in Peru is relatively immature; only a small percentage of Peru's farmland is regularly fertilized. As a result, Peruvian agricultural productivity is relatively low but as investment drives modernization, fertilizer use should increase accordingly. Peru does not produce fertilizers in any significant quantities; all chemical fertilizers consumed in the country - more than 900,000 tonnes of fertilizer per year are imported from the USA, Canada, Russia and Colombia. Figure 19-3 shows how Peruvian consumption of fertilizers has risen steadily over the last few years. Up to mid-July 2015, the country imported 596,196 tonnes of fertilizers implying annual consumption in excess of 1 million tonnes for the year. Ironically, despite being one of the largest producers of phosphate rock in the Americas, with one of if not the most natural phosphate rock known, more than 99% of Peru's production of Sechura phosphate rock is exported abroad for the production of soluble chemical fertilizers.

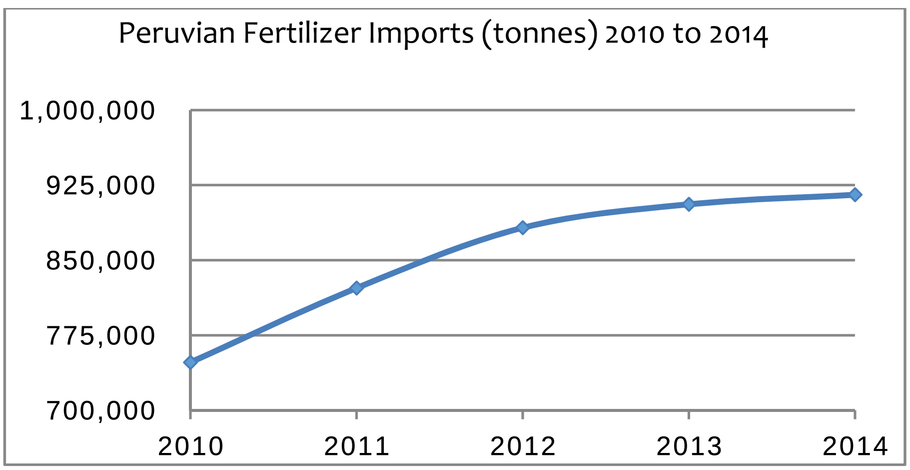


Figure 19-3: Peruvian Fertilizer Imports 2010-2014

19.5 SECHURA REACTIVE PHOSPHATE ROCK: CHARACTERISTICS & AGRONOMIC EFFECTIVENESS

RPR can be defined as natural phosphate rocks that release their phosphorus into plant-available form at a fast enough rate, even on only mildly acidic soils, to maintain crop and pasture production over time as well as soluble phosphorus fertilizer does. In general the effectiveness of DAPR is greater with long-term and perennial crops than for short-term

[8] *http://orgprints.org/25172/1/willer-lernoud-2014-world-of-organic.pdf*
[9] *http://orgprints.org/25172/1/willer-lernoud-2014-world-of-organic.pdf*

or annual crops, due to its slower release into the soil. DAPR has been used successfully for many tree crops in Asia including rubber, palm oil and tea.

Sechura RPR has been shown to be one of the most reactive[10] in the world, and an effective slow-release phosphate fertilizer under favorable conditions[11] Table 19-1 below, shows the solubility spectrum developed by the International Fertilizer Development Center (IFDC) for phosphate rocks. The test values quantify the natural ability of phosphate rock to breakdown under ambient conditions.

Table 19-1: The Solublity Spectrum of Phosphate Rocks

	Potential response	Solubility (%P_2O_5)	
	NAC	2% Citric acid	2% Formic Acid
High	>5.4	>9.4	>13.0
Medium	3.2-4.5	6.7-8.4	7.0-10.8
Low	<2.7	<6.0	<5.8

The neutral ammonium citrate solubility of Sechura rock has been measured by the International Fertilizer Association (IFA) at between 5 to 9% putting at the upper end of the highly reactive classification and making it ideally suited for use as a direct application phosphate rock[12] [13]. A full range of solubility tests were carried out for Focus by Jacobs on four processed samples from Bayovar 12 during bench scale metallurgical test work consisting of Plant Line 1 and Plant Line 2 product and two composites. All samples returned excellent solubilities described below.

Table 19-2: Available P_2O_5 Analysis, Bayovar 12

				Solubility (%P_2O_5)	
		% P_2O_5	NAC	2% Citric acid	2% Formic Acid
Line 1 - 24% P_2O_5		23.87	6.45	9.42	14.33
Line 2 - 28% P_2O_5		28.23	6.99	10.46	16.03
Composite PH02-06		23.55	6.06	10.79	16.47
Composite PH11-13		24.86	8.9	14.68	18.67

The whole rock solubility reported in Table 19-2 indicate high solubilities, and there is little doubt that Sechura RPR is highly-reactive and one of the world's best RPRs for use as DAPR.

Numerous studies have demonstrated that Sechura RPR is a *highly effective source of phosphorus that can equal or outperform the agronomic performance of manufactured, water soluble fertilizers* in acid soils with elevated rainfall. The soil and climatic conditions in large parts of Central and South America (acid soils with high rainfall) are ideal for maximizing the agronomic performance of Sechura RPR and the rock should perform as well as manufactured fertilizers such as single superphosphate (SSP).

An important consideration for farmers is that DAPR usually costs much less per tonne than SSP and other manufactured fertilizers. Also, DAPR's slower phosphorus-release characteristic doesn't overwhelm the phosphorus

[10] *Application strategies for Sechura phosphate rock use on permanent pasture. Gregg, Mackay, Currie & Syers, 1988.*
[11] *A. G. Sinclair , P. W. Shannon & W. H. Risk (1990) Sechura phosphate rock supplies plant-available molybdenum for pastures, New Zealand Journal of Agricultural Research, 33:3, 499-502*
[12] *Use of phosphate rocks in sustainable agriculture. FAO Fertilizer & Plant Nutrition Bulletin 13. 2004.*
[13] *Van Kauwenbergh, S.J. 1995. Mineralogy and characterization of phosphate rock. In K. Dahanayake, S.J. Van Kauwenbergh & D.T. Hellums, eds. Direct application of phosphate rock and appropriate technology fertilizers in Asia – what hinders acceptance and growth, pp. 29–47. Kandy, Sri Lanka, Institute of Fundamental Studies.*

capacity of the soils. DAPR releases phosphorus over a 3-4 year profile providing consistent long-term nutrients to the crops.

As discussed above, Sechura RPR is particularly well suited for use on palm oil plantations and has been used on plantations in Indonesia and Malaysia for some years. It is advertised by large distributors of fertilizer products as THE premium phosphate fertilizer. The rapid expansion underway in the Latin American oil palm sector represents a major opportunity for Focus' DAPR production. This market will be an important part of Focus' long-term marketing strategy.

19.6 PRICES PAID FOB BAYOVAR FOR DAPR

Two mines close to Focus' Bayovar project have been producing and selling DAPR for several years. Fosyeiki S.A.C. and Corporación Agrosechura Peru S.A.C. produce and export ~22-24% P_2O_5 rock by truck and ship to Ecuador, Nicaragua, Bolivia and Colombia. Fosyeiki's product is claimed to range from 22-24% P_2O_5 and its sales volumes appear to be limited by its production capacity. FOB prices by ship to Central America have ranged from US$163/T to US$195/T. FOB prices achieved by Agrosechura by truck to Bolivia and Ecuador have ranged up to US$220/T.

19.7 PRICES PAID BY PERUVIAN FARMERS FOR DAPR

The Peruvian agricultural ministry (MINAGRI) compiles pricing information for direct application phosphate rock from across Peru, broken down by region for the years 2013 to 2015. Quantities sold throughout Peru are not reported by MINAGRI. Generally DAPR is used by organic farmers who cannot use chemical fertilizers or pesticides on their crops. The average retail price paid by the farmer in Peru for DAPR has remained fairly constant at about US$320/T. Prices are consistently lowest (US$200-240/t) in the regions of Cajamarca, Amazonas and San Martin closest to Piura with lower transport costs. Farmers in Loreto are paying the most for DAPR, currently about US$510/T. Junín is the region with the largest number of organic farmers in Peru. The price of DAPR in Junín is US$210/T currently.

19.8 OTHER DAPR SALES IN THE AMERICAS

Recent sales of DAPR in North and South America are summarized below.

- Fertoz, Wapiti mine, BC, Canada: C$200/T (US$150/T) for 20% P_2O_5 rock
- Dusolo Fertilizers, Bomfim mine, Brazil, C$93 – C$110/T (US$70-83/T) low grade 15% P_2O_5 rock
- B&A Mineracao, Bonito Mine, Brazil, US$185/T for 20% P_2O_5 rock.

19.9 PRICE ASSUMPTION FOR 24% P_2O_5 DAPR PRODUCT

Retail prices of DAPR to organic farmers in Peru have remained fairly constant at about US$320 per tonne for the past few years. In Malaysia, Bayovar rock is currently being marketed by Union Harvest for US$160-170/T. This Pre-Feasibility Study assumes a wholesale price of US$145/t for Focus' 24% P_2O_5 product.

19.10 PRICE ASSUMPTION FOR 28% P_2O_5 DAPR PRODUCT

Several studies conducted by governmental organizations over the past decade have shown that on plantations located in areas with acidic soils and high rain fall, Sechura RPR competes agronomically with SSP, achieving similar or higher crop yields. Consequently RPR and SSP tend to compete directly and prices are similar, as seen in countries such as Brazil.

Focus will therefore be looking to displace SSP sales in South and Central America with a cheaper but more effective product. This Pre-Feasibility Study assumes a wholesale price of US$185/t for Focus' higher grade product, which would represent approximately a 25% discount to SSP prices in Latin America.

19.11 BLENDED RPR PRODUCTS

Focus has retained a consulting soil scientist and fertilizer expert to advice on opportunities to produce simple RPR-based blended products. RPR can easily be blended with other nutrients. From a spreading perspective, it is easiest if these are in fine granular form to ensure even spreading and avoid segregation on transport. The most commonly deficient nutrients that need addition are sulphur, which can be added as gypsum* or fine elemental sulphur (S), potassium (K) which can be added as standard grade potash or sulphate of potash, and magnesium which can be added as dolomite or magnesium oxide, and of course nitrogen (N). Any form of fertilizer N can be blended, but preferably in fine or prilled form. In late 2015 Focus was investigating carrying out field trials of Sechura RPR blends at a palm oil plantation in Peru to compare its efficacy versus other fertilizers.

**There is a substantial resource of high quality gypsum on the Bayovar12 concession which forms an evaporite layer approximately 0.4 - 1m thick within 0 - 2m of the surface. There is a small mining operation on the concession exploiting the gypsum and producing about 80kt per year.*

19.12 SALES CONTRACTS

No sales contracts or off-take agreements exist at the time of writing of this report. Focus has had preliminary discussions with a number of potential consumers of DAPR in South and Central America and will be running agronomic trials with these potential customers during 2016.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 GENERAL REGULATORY AND INSTITUTIONAL FRAMEWORK

Peruvian legislation requires that an Environmental Impact Assessment (EIA) be completed and approved when projects of public or private investment are undertaken that are likely to cause significant and negative environmental impacts. This requirement also extends to any modifications, extensions or diversification of existing projects if these changes in size, scope or circumstances could generate new or increased negative environmental impacts.

The competent authority that will be in charge of the review and approval of the Detailed Environmental Impact Assessment (dEIA) is the National Service for Environmental Certification for Sustainable Investment (SENACE), created by Law N° 29968; however, some dEIAs are expressly excluded from SENACE review by Supreme Decree, Article 1.3 Law N° 29968. SENACE is in the process of implementing functions and will start its competences on December 28th, 2015.

This section provides a synthesis of the current regulations framework related to authorizations and permits for the current Project description. Table 20-1 includes the competent authorities these procedures are related to.

Table 20-1: Applicable Regulations to the Project

Title	Norm	Regulatory Body
General Regulations		
Political Constitution of Peru of 1993		Several authorities
National Environmental Policy	S.D. Nº 012-2009-MINAM	MINAM and other authorities
General Environmental Law and its amending provisions	Law Nº 28611	Several authorities
Organic Law for the Sustainable Use of Natural Resources	Law Nº 26821	Several authorities
Law on the National Environmental Management System, its amending provisions and Regulations thereof	Law Nº 28245 S.D. Nº 008-2005-PCM	MINAM and other authorities
Law on the National Environmental Impact Assessment System, its amending provisions and Regulations thereof	Law Nº 27446 S.D. Nº 019-2009-MINAM Ministerial Resolution (M.R.) N° 157-2011-MINAM	MINAM and other authorities
Criminal Code – Title XIII	Legislative Decree (L.D.) Nº 635, amended by Law N°29263	Government Attorney General's Office / The Judiciary
Law Establishing the Obligation to Prepare and File Contingency Plans	Law Nº 28551	Several authorities
Regulation of Environmental Emergencies Report of the activities under the purview of the Agency for Assessment and Environmental Control - OEFA	Board Resolution N° 018-2013-OEFA-CD	OEFA
Regulations of the Mining Subsector		
Consolidated Text of the General Mining Law and its amending provisions	S.D. Nº 014-92-EM	MINEM

Title	Norm	Regulatory Body
Regulation of Environmental Protection and Management for Operational Activities, Benefit, General Labor, Transportation and Storage Mining and Terms of Reference	S.D. N° 040-2014-EM M.R. N° 116-2015-MEM-DM	MINEM
Approval of Online Environmental Assessment System to obtain the environmental certification for medium and larger mining projects, and related norms	M.R. N° 270-2011-MEM/DM M.R. Nº 358-2013-MEM-DM M.R. N° 314-2014-MEM-DM	MINEM
Mine Closure Regulations	Law Nº 28090; S.D. Nº 033-2005-EM	MINEM
Approved Provisions for Mining Procedures that Promote Investment Projects	S.D. N° 001-2015-EM	MINEM
Special Provisions for Implementation of Administrative Procedures and Technician Criterias Modifying regulating or Extensions and Components Miners Technological improvements in Mining Projects Units Exploration and Exploitation No Significant Environmental Impacts that Count on Environmental Certification	S.D. N° 054-2013-PCM M.R. N° 120-2014-MEM/DM	Several authorities
Special Provisions for Implementation of Procedure Administrative and Other Measures to Promote Investment Projects Public and Private and Specific Provisions Detailed Study for Environmental Impact Mining and Energy Sector	S.D. N° 060-2013-PCM M.R. N° 092-2014-MEM/DM	Several authorities
Regulation for protection and environmental management of exploitation, production, labor, transportation and storage activities; and its terms of reference	S.D. N° 040-2014-EM M.R. N° 116-2015-MEM-DM	MINEM
Cases in which the approval of the Environmental Impact Assessment and Environmental Management Programs requires Technical Opinion of the Agriculture Sector	S.D. N° 056-97-PCM	MINAGRI, MINEM
Mining environmental regulation for mineral concentrate storage	L.D. N° 1048	Several authorities
Other Specifically Adjusted Aspects		
Water		
Water Resources Law	Law Nº 29338	National Water Authority (ANA)
Regulations of the Water Resources Law and its amendments	D.S. Nº 001-2010-AG; D.S. N° 005-2013-AG D.S. N° 023-2014.MINAGRI	ANA
Rules of Administrative Procedures for Granting Water Use Rights and Authorization Works Execution in Natural Water Sources	Executive Resolution (E.R.) N° 007-2015-ANA	ANA
Environmental Quality Standards for Water	S.D. Nº 002-2008-MINAM	MINAM
Approval for the implementation of National Environmental Quality Standards (ECA by Spanish acronym) for water	S.D. N° 023-2009-MINAM	MINAM

Title	Norm	Regulatory Body
Maximum Permissible Levels for National or Municipal Wastewater Treatment Plants Effluents	S.D. Nº 003-2010-MINAM	MINAM
Classification of surface water and marine – coastal and its updating	E.R. N° 202-2010-ANA; E.R. N° 489-2010-ANA	ANA
National Protocol to water quality of surface water bodies monitoring	E.R. Nº 182-2011-ANA	ANA
Procedure for the opinion to be issued by the National Water Authority in the process of evaluation of environmental impact studies related to water resources	E.R. Nº 106-2011-ANA	ANA
Air and Noise		
Environmental Quality Standards Regulations for Air and Update of Standards for Lead (Pb)	S.D. Nº 074-2001-PCM S.D. N° 069-2003-PCM	MINAM
Maximum Permissible Levels of compound elements present in gaseous emissions from mining and metallurgical units	M.R. Nº 315-96-EM/VMM	MINEM
Environmental Quality Standards for Air and Complementary Dispositions	S.D. Nº 003-2008-MINAM S.D. N° 006-2013-MINAM	MINAM
Environmental Quality Standards Regulations for Noise	S.D. Nº 085-2003-PCM	MINAM
Flora, Fauna and Biological Diversity		
Forest and Wild Fauna Law and Regulations	Law N° 27308 ; Law N° 29763; S.D. N° 014-2001-AG	MINAM
Law on the preservation and sustainable use of biological diversity, Regulations and its amending provisions	Law Nº 26839 S.D. Nº 068-2001-PCM	MINAM and other authorities
Approval of an updated list of classification and categorization of endangered wildlife species legally protected and its amending	S.D. N° 004-2014-MINAGRI D.S. 002-2015-MINAGRI	MINAGRI
Endangered Species of Wild Flora Categorization Approved	S.D. N° 043-2006-AG	MINAM
Protected Natural Areas		
Protected Natural Areas Law, Regulations and its amending provisions	Law Nº 26834 S.D. Nº 038-2001-AG	SERNANP
Legislative Decree establishing measures to ensure the heritage of protected natural areas and it Regulation	L.D. N° 1079 S.D. N° 008-2008-MINAM	SERNANP
Solid Waste and Hazardous Waste and Material Transport		
General Law on Solid Waste, Regulations and its amending provisions	Law N° 27314; S.D. Nº 057-2004-PCM L.D. N° 1065	Several authorities
Regulating of Material and Hazardous Waste Transportation Law, Regulations, and Modifications	Law Nº 28256 S.D. Nº 021-2008-MTC	Several authorities
Radioactive Material		
Law Regulating the Use of Sources of Ionizing Radiation and Regulations	Law N° 28028 S.D. N° 039-2008-EM	

Title	Norm	Regulatory Body
Radiation Safety Requirements for Industrial Radiography and its Modifications	Presidential Resolution N° 147-09-IPEN/PRES Presidential Resolution N° 234-10-IPEN/PRES	IPEN
Physical Security Requirements of Radioactive Sources	Presidential Resolution N° 131-11-IPEN/PRES	IPEN
Consumption of Hydrocarbons		
Regulations for the Commercialization of Liquid Fuels and other Hydrocarbon-derived products, Safety Regulations for the Storage of Hydrocarbons, and its amending provisions	S.D. N° 045-2001-EM S.D. Nº 052-93-EM S.D. N° 036-2003-EM	MINEM/OSINERGMIN
Approval of the Regulation of Oil Record and Modifications	Board Resolution N° 191-2011-OS-CD	OSINERGMIN
Occupational Safety and Health		
General Health Law and its amending provisions	Law Nº 26842	Several authorities
Occupational Health and Safety Law, Modification and Regulations	Law N° 29783; Law N° 30222 S.D. N° 005-2012-TR	MINEM/OSINERGMIN
Regulation of Occupational Safety and Health and Other Complementary Measures in Mining	S.D. N° 055-2010-EM	MINEM/OSINERGMIN
Cultural Heritage		
National Cultural Heritage Law	Law Nº 28296	MC
Modification of National Cultural Heritage Law and its Regulations	L.D. N° 1003 S.D. N° 011-2006-ED	MC
Law amending articles 226 and 228 of the Criminal Code against the National Cultural Heritage	Law Nº 28567	MC
Archaeological Interventions Regulations and Amended and Special Procedures for Implementation	S.D. N° 003-2014-MC	MC
Regulation of Archaeological Research and Special Provisions for the Procedure of Issuance of Certificate of Absence of Archaeological Remains (CIRA) and the Adoption of Archaeological Monitoring Plan	Supreme Resolution (S.R.) N° 004-2000-ED S.D. N° 054-2013-PCM	MC
Public Participation		
Law of Transparency and Access to Public Information	Law N° 27806	Several authorities
Consolidated Text of Law N° 27806, Law of Transparency and Access to Public Information and it Regulation	S.D. N° 043-2003-PCM S.D. N° 072-2003-PCM	Several authorities
Regulation on Transparency, Public Access to Environmental Information and Public Participation and Consultation in Environmental Matters	S.D. N° 002-2009-MINAM	MINAM
Prior commitment as a prerequisite for development of mining activities and its amending provisions	S.D. Nº 042-2003-EM; S.D. N° 052-2010-EM	MINEM

Title	Norm	Regulatory Body
Rules of Participation in the Mining Subsector	S.D. N° 028-2008-EM	MINEM
Rules governing the Public Participation Process in Subsector Mining	R.M. N° 304-2008-MEM-DM	MINEM
Soil		
Approved Land Classification Regulations for its use capacity	S.D. N° 017-2009-AG	MINAGRI
Approved Environmental Quality Standards (ECA) for Soil	S.D. N° 002-2013-MINAM	MINAM
Complementary provisions for the implementation of the Environmental Quality Standards (ECA) for Soil are approved	S.D. N° 002-2014-MINAM	MINAM
Approved Guidelines for Soil Sampling and Guide for the Preparation of Plans for Soil Decontamination	M.R. N° 085-2014-MINAM	MINAM
Approved Sampling Protocol for Environmental Emergency	R.M. N° 125-2014-MINAM	MINAM
Private Investment Law for the development of economic activities in national territory and peasant and native communities lands and its amending provisions	Law Nº 26505; S.D. N° 011-97-AG	Several authorities
Supervision		
Law on the National System of Environmental Assessment and Supervision and it Modification	Law Nº 29325 Law N° 30011	OEFA and several authorities
Regulations and Functions of OEFA	S.D. N° 022-2009-MINAM	OEFA
Common Regime of Environmental Control	M.R. N° 247-2013- MINAM	MINAM
Regulation of Direct Supervision Agency Assessment and Environmental Control - OEFA.	Board Resolution N° 007-2013-OEFA-CD	OEFA
Regulation of energetic and mining activities supervision – Osinergmin	Board Resolution N° 171-2013-OS/CD	OSINERGMIN
Legal rules for the application of Art. 17 of the Law of the National System of Evaluation and Environmental in the field of environmental enforcement mining control	Board Resolution N° 031-2014-OEFA-CD	OEFA
Rules of citizen participation in environmental monitoring actions by the Agency for Assessment and Environmental Control - OEFA	Board Resolution N° 032-2014-OEFA-CD	OEFA
Environmental Liabilities		
Law Regulating Environmental Liabilities Mining Activity, its Regulations and Modifications	Law N° 28271; S.D. N° 059-2005-EM; L.D. Nº 1042; S.D. Nº 003-2009-EM.	MINEM
Ports		
National Port System Law; modified by the L.D. N° 1022	Law N° 27943; L.D. N° 1022	MTC and other authorities
Regulations of the National Port System Law	S.D. N° 003-2004-MTC; S.D. N° 027-2008-MTC	MTC

20.1.1 Permits Required for the Mining Project

The main permits required for the Project are presented in Table 20-2.

Table 20-2: Permits Required for the Project

Permit	Legal Basis	Preparation Estimated Time	Approval Estimated Time	Authority that Approves
Authorization of the Start of Mining Activities - Approval of the Mining Plan	• Regulation of Environmental Protection and Management for Operational Activities, Benefit, General Labor, Transportation and Storage Mining (S.D. Nº 040-2014-EM) • Requirements to be taken into account for the Development of Mining Activities in Construction Material Quarries (M.R. Nº 188-97-EM-VMM, modified by S.D. N° 020-2012-EM) • Approved dispositions to promote mining investments projects (S.D. N° 001-2015-EM) • General Administrative Procedure Law (Law Nº 27444)	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 4 months 2. Information unavailable or incomplete: 4 months	MINEM
Certificate of Mining Operation (COM)	• Law of firearms, ammunition, explosives, fireworks and related materials for civilian use (Law N°30299) • Regulation of the Emergency Declared for the Use of Explosives (S.D. N° 086-92 PCM) • General Administrative Procedure Law (Law Nº 27444) • Regulations of Occupational Safety and Health and other Complementary Measures in Mining (S.D. N° 055-2010-EM) • Formalization and Promotion of the Small Mining and Artisan Mining Law (Law N° 27651)	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 1 months	MINEM
Granting of the Process Plant Concession	• Ordered Only Text of the General Mining Law (S.D. N° 014-92-EM) • Regulation for the Environmental Protection in the Metallurgical Mining Activity (S.D. N° 016-93-EM) • Law that Establishes Periods for the Previous Evaluation of Certain Administrative Procedures Processed in the Ministry of Energy and Mines (Law N° 27798) • Regulation of Environmental Protection and Management for Operational Activities, Benefit, General Labor, Transportation and Storage Mining (S.D. Nº 040-2014-EM)	1. Information available and complete: - Phase A: For the evaluation of the petition and publication of the notice, two months are estimated. - Phase B: Preparation of the technical dossier to obtain the authorization for construction, the time estimated would be one month. MINEM time of approval has been estimated in five months. In that regard, the estimated time to obtain the authorization would be six months. - Phase C: For the follow-up inspection and entitlement, one month is estimated. *The total estimated time for the obtainment of the Process Plant Concession is nine months without taking into consideration the construction duration.*		MINEM

Permit	Legal Basis	Preparation Estimated Time	Approval Estimated Time	Authority that Approves
		2. Information unavailable or incomplete: - Phase A: to prepare the technical dossier to obtain the construction authorization, the estimated time would be three months. MINEM time approval has been estimated in six months. - Phase B: for the evaluation of the petition and the publication of notice, two months have been estimated - Phase C: for the follow-up inspection and entitlement, one month is estimated (after construction). *The total estimated time to obtain the Process Plant Concession is 12 months.*		
Water Use License	• Water Resources Law (Law Nº 29338) • Regulations to the Law of Water Resources (S.D. Nº 001-2010-AG) • Rules of Administrative Procedures for Granting Water Use Rights and Authorization Works Execution in Natural Water Sources (R.J. N° 007-2015-ANA)	The water use license is not really a procedure, but is granted automatically after 5 days, after field verification of approved works is performed.		Local Water Authority (ALA)
Favorable Technical Report on Direct Consumers of Liquid Fuels	• General Administrative Procedure Law (Law N° 27444) • Rules of Security for the Storage of Hydrocarbons (S.D. Nº 052 93 EM) • Rules of Security for the Commercialization of Oil By-products. (S.D. N° 054-93-EM). • Regulations for the Commercialization of Liquid Fuels and Other Oil By-products (S.D. N° 045- 2001-EM) • Incorporates requirements for different supervision administrative procedures in charge of OSINERG. (D.C.R. N° 162-2005-OS/CD) • Regulations for the Environmental Protection in the Hydrocarbon Activities (S.D. Nº 015-2006-EM; modified by S.D. N° 039-2014-EM) • Rules of Security for the Hydrocarbon Activities and modifies different regulations (S.D. N° 043-2007-EM)	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 4 months 2. Information unavailable or incomplete: 4 months	OSINERGMIN

Permit	Legal Basis	Preparation Estimated Time	Approval Estimated Time	Authority that Approves
	• Board Resolution N° 191-2011-OS-CD and its modifications B.R. N° 125-2014-OS-CD			
Definitive Electric Concession	• Law of Electric Concessions (D.L. N° 25844) • Regulation of the Electric Concessions (S.D. N° 009-93-EM) • Authorizing the Architects Association of Peru and the Engineering Associations of Peru for Supervising the Engineering and Architecture Professionals of the Republic (Law N° 16053) • General Administrative Procedure Law (Law N° 27444) • Modification of the Rules of the Regulation of Electric Concessions (S.D. N° 025-2006-EM)	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 8 months 2. Information unavailable or incomplete: 8 months	MINEM
Use of Right of Way/Crossroads		1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 8 months 2. Information unavailable or incomplete: 8 months	MTC
Regulation of Archaeological Research and Special provisions for the procedure of issuing the Certificate of Absence of	• General Law of Cultural Patrimony of the Nation (Law N° 28296; modified by L.D. N° 1003) and Regulations • Title VIII of the Penal Code, Crimes Cultural Heritage • Regulations for Archaeological Interventions (S.D. N° 003-2014-MTC)			MC

Permit	Legal Basis	Preparation Estimated Time	Approval Estimated Time	Authority that Approves
Archaeological Remains (CIRA)				
Sanitary Authorization of Drinking Water Treatment Systems	• General Health Law (Law N° 26842)	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 10 months 2. Information unavailable or incomplete: 10 months	DIGESA
Multidisciplinary Civil Defense Technical Inspection (INDECI) - Region (DCM00)	• Supreme Decree approving the Regulation of Technical Safety Inspections in Civil Defense, S.D. N° 058-2014-PCM, which modified S.D. Nº 066-2007-PCM • Safety Regulations for Oil Liquefied Gas Facilities and Transportation (S.D. Nº 027 94-EM) • Regulations for the Commercialization of Liquid Fuels and Other Hydrocarbon By-Products (S.D. Nº 045-2001-EM and amendments)	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 2 months 2. Information unavailable or incomplete: 2 months	INDECI
Detailed Civil Defense Technical Inspection (INDECI) - Region	• Regulation of Technical Safety Inspections in Civil Defense (S.D. N° 058-2014-PCM) • Safety Regulations for Oil Liquefied Gas Facilities and Transportation (S.D. Nº 027-94-EM) • Regulations for the Commercialization of Liquid Fuels and Other Hydrocarbon By-Products (S.D. Nº 045-2001-EM and amendments)	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 2 months 2. Information unavailable or incomplete: 2 months	INDECI
Hydrocarbons Registry of Direct Consumers	• S.D. N° 045-2012 PCM	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 3 months	1. Information available and complete: 1 month 2. Information unavailable or incomplete: 1 month	OSINERGMIN
License for Stationary and Portable Nuclear Meters (MR00)	• Regulatory Law for the Use of Ionizing Radiation Sources (Law Nº 28028) • Regulations to the Regulatory Law for the Use of Ionizing Radiation Sources (S.D. Nº 039-2008-EM).	1. Information available and complete: 15 days 2. Information unavailable or incomplete: 1 month	1. Information available and complete: 2 months 2. Information unavailable or incomplete: 2 months	IPEN
Mine Closure Plan	• Law Nº 28090 Mine Closure Regulations (S.D. Nº 033-2005-EM)		10 months	MINEM

20.1.2 Sequence of Permitting Application to Develop the Mining Project

The permit application sequence for developing mining projects in Peru is presented in Figure 20-1.

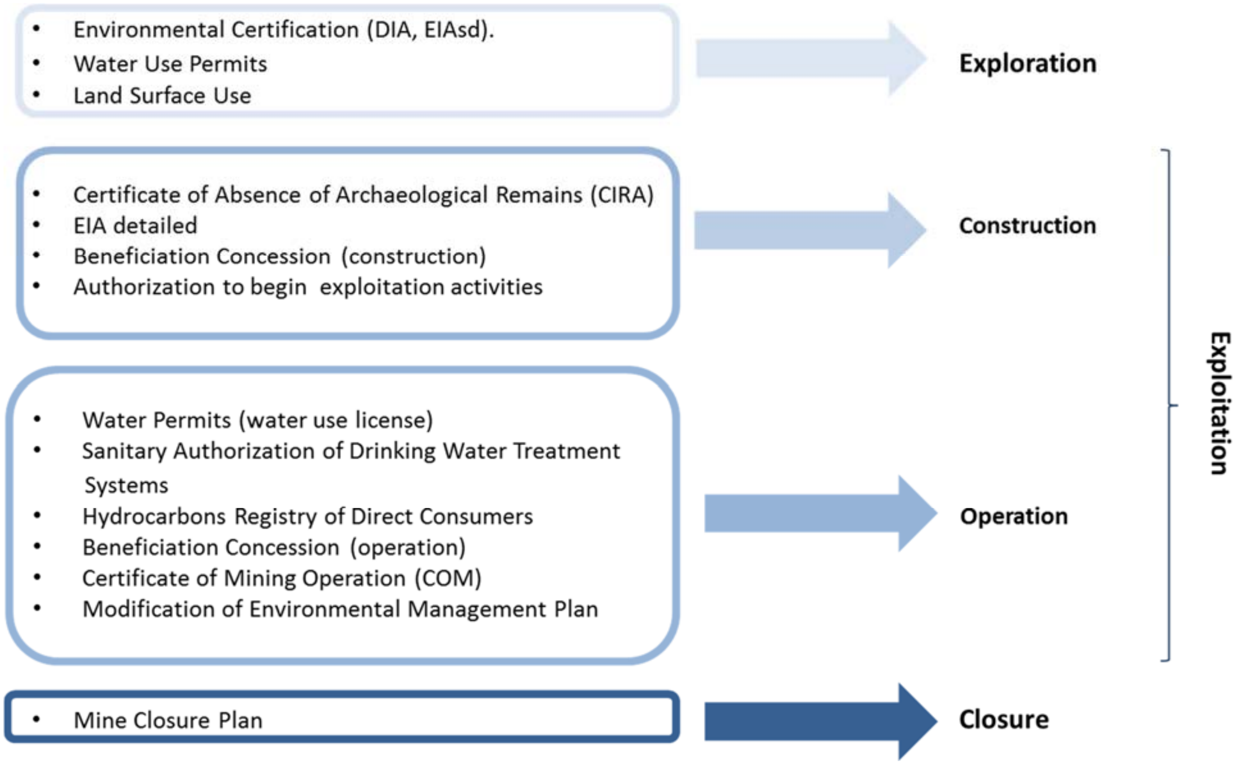


Figure 20-1: Mining Project Permit Application Sequence

20.2 DESCRIPTION OF PHYSICAL COMPONENTS

This section contains information regarding the physical components of the area of assessment (Figure 20-2), defined in the area which the Project will be located. This information proceeds from studies performed in the area, considering the data available representative for the Bayovar 12 Phosphate Project. The studies reviewed are the following:

- EIA of Bayovar Phosphates Mine (Golder 2007), owned by Compañía Minera Miski Mayo (CMMM).
- First Modification of the EIA of Bayovar Phosphates Mine (Golder 2011).
- EIA of Phosphates Project (BISA 2013) owned by Fosfatos del Pacífico (FOSPAC).
- Second Modification of the EIA of Bayovar Phosphates Mine (Golder 2014).
- Semi detailed Exploration Environmental Impact Assessment (EIAsd) of Bayovar N° 12 prepared by Asesores y Consultores Mineros S.A. (ACOMISA 2014).

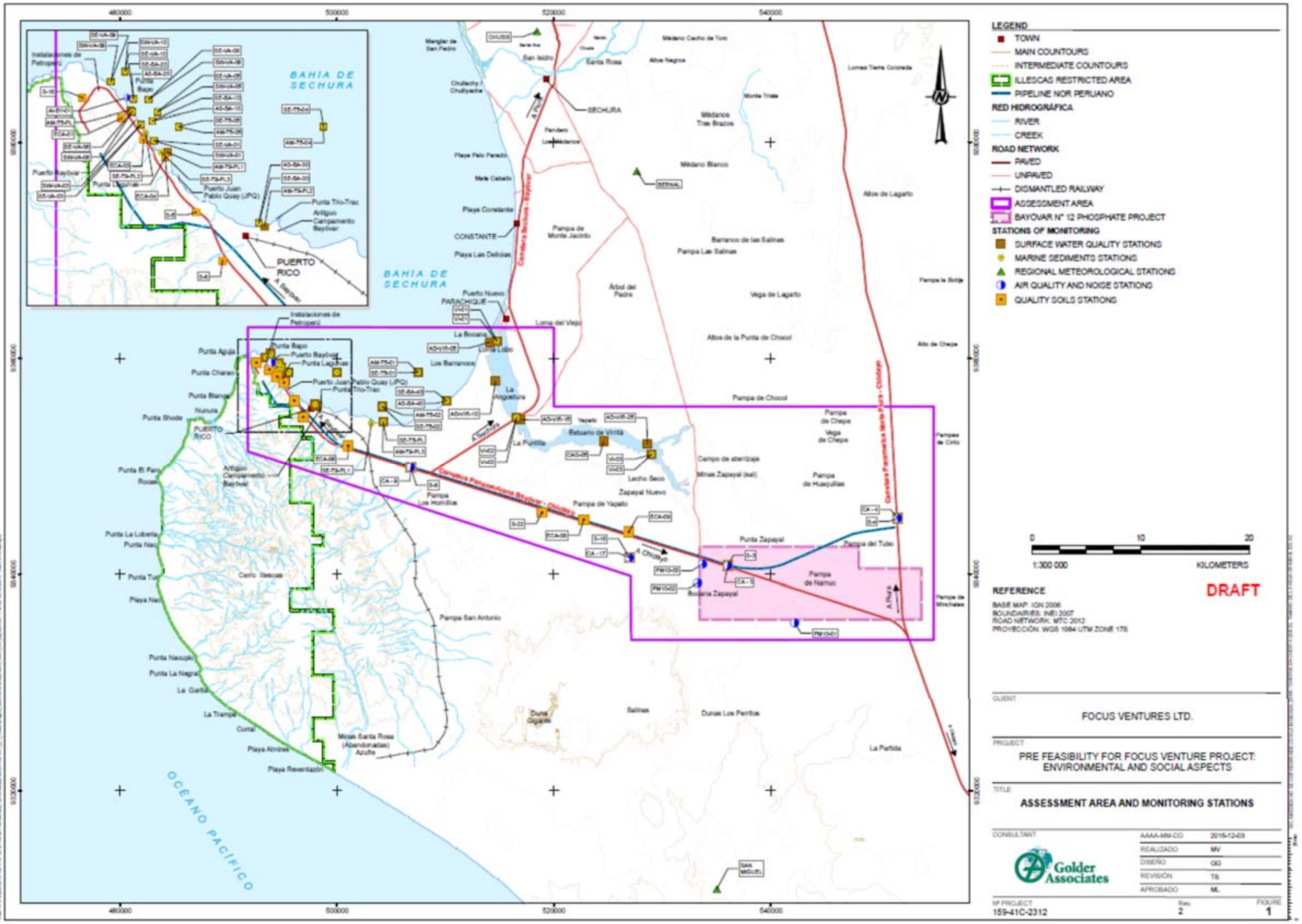


Figure 20-2: Assessment Area and Monitoring Stations

Climate and Meteorology

The analysis of the meteorological conditions was performed based on the official records available from three regional meteorological stations: San Miguel; Chusis; and Bernal, located in the Piura Region, near the Project, and operated by the National Meteorological and Hydrological Service of Peru (SENHAMI).

The climate in the study area is influenced by variations in the equatorial belt and by the changes of direction (east to west) of the cold and hot water currents. These characteristics result in high temperatures and low rainfall, except for short periods of sporadic income of warm water current (El Niño) in the southern hemisphere or other extraordinary events, as occurred in 1982-1983 and 1997-1998. The meeting of the cold water Humboldt Current (14 °C - 19 °C) with the warm El Niño current (22 °C - 27 °C) at Sechura Bay, in the south of Piura, causes some variety and climate mergers that give unique characteristics to this region. Factors influencing the climate are the Andes Mountains that cross the western zone of the South American continent, and act as a natural barrier preventing the passage of the humid air masses from the Atlantic to the Pacific, the South Pacific Anticyclone and the Inter-tropical Convergence Zone which condition the Alisios winds blowing from the southeast and northeast sectors. According to SENAMHI (2008) the climate classification of the study area, based on the Thornthwaite method (1931), is arid, semi warm, with deficient rains in all seasons (normal years) and relatively high humidity. Key characteristics of the study area climate are:

- Temperature: The annual air temperature in the study area is characterized by maximum values presented between January and March, with the highest average maximum temperature recorded in February (33.1 °C); and minimum air temperature values recorded from July to September, with the lowest average minimum temperature recorded in August (16.6 °C).

- Humidity: During an annual cycle, the highest average relative humidity values were recorded between June and August. The average monthly relative humidity fluctuates between 67.5% in January to 76.0% in June at San Miguel station; and from 70.0% in February to 77.2% in August at Chusis station.

- Wind: The annual course of wind speed in the area of study is not only due to the direct influence of solar radiation but also to the exchange of large-scale global atmospheric circulation. The average wind speed in the area is 4.4 m/s; this velocity is considered as weak breezes according to the Beaufort scale (OOM 2008). The predominant wind direction is from the south (65.3%) at San Miguel station and from the southeast (48.8%) at Chusis station.

- Solar Radiation: According the Atlas of Solar Radiation (SENAMHI, 2003), the northern coast of Peru (between 3° S and 6° S latitude), presents high values of solar energy during the austral summer; however, the maximum values are recorded in spring, from October to November. At a regional level, the average daily values of solar radiation oscillate in the range from 5,000 W/m^2 to 6,500 W/m^2.

- Precipitation: The rains usually occur in the summer months, from January to April; the rest of the year is commonly dry. The average annual precipitation at Chusis was 77.7 mm and 58.6 mm at Bernal.[14]

Hydrography

The Project area is located on the interbasin between the Piura River and the Cascajal River. The hydrographic basin has an area of approximately 12,216 km^2.

[14] This average is strongly influenced by extraordinary precipitation that occurred in 1983 and 1998; reaching values of 1,036.1 mm at Chusis and 1,200.5 mm at Bernal.

The Piura River is formed from a diverse system of headwaters derived from the Andes Mountains that converge in the Las Lomas and Chulucanas districts to the east, limiting possibilities to install reservoirs or other facilities for regulation and use of water. Ramón, Ñapique and Mala Vida are ephemeral lagoons, located close to the southeast extreme of the Lower Piura Valley, approximately 30km south of Piura city. Once the flow provided by the Piura River fills the Ramón Lagoon during extreme El Nino events, this lagoon starts its overflow towards Ñapique Lagoon, and then drains towards the town of Sechura. The defenses built to prevent this from happening, including a dam at Ñapique Lagoon, force the water out through Ramón Lagoon to the Pampas Las Salinas by an extension called Bravo Ramon River, forming the La Niña Lake some 12km further south. In extreme cases, La Niña Lake connects with the Virrilá Estuary to reach the sea.

The Cascajal River Basin has 5,350 km^2 of area and is 154 km long; this river is considered a weak river because it does not reach the sea, except during years with substantial rain when the Cascajal River reaches the sea through the Virrilá Estuary. The endorheic Cascajal River Basin contributes with the Piura River forming La Niña Lake, and eventually contributes to the aquifer recharge. The records of gauging water flow on the Cascajal River show that this river normally does not have exceedances into its agricultural area, but in extreme years of El Nino events, its flow increases to seven or eight times compared to normal flow. The annual volume of the Cascajal River in a normal year reaches 79 Mm^3.

Geology, Geomorphology and Geodynamic

Regional Geology

Peru is located on the edge of two tectonic plates, Sudamericana and Nazca. Regionally, the study area is located in the Sechura basin; the western limit of this basin is delineated by Cerro Illescas, which comprises Precambrian and lower Paleozoic basement rocks, and the Andes mountains are at the eastern limit.

The regional lithostratigraphy includes metamorphic and igneous rocks that outcrop in Cerro Illescas. The metamorphic rocks consist of gneisses and tonalite from the Precambrian period and migmatites, granites, schists, phyllites and shales from the Lower Paleozoic period. The Quaternary deposits are represented by the Hornillos Formation, Talara and Lobitos shoals, whose main outcrops are located near Cerro Illescas and the Virrilá Estuary.

Local Geology

The following soil units are present in the area of assessment:

- Aeolian deposits, such as barchan dunes, around some vegetation named Algarrobal-Sapotal; comprising fine sand with loose, rounded grains; these are mainly located in the Duna Gigante.

- Lake and estuary deposits, with laminated beige fine sand and clay interbedded with coquina beds; these are located in Salina Grande depression.

- Alluvial deposits that contain sub angular unconsolidated conglomerates in a beige colored quartz and feldspar matrix, located next to Cerro Illescas.

- Lobitos and Talara terraces, consists of poorly lithified sub angular fossil conglomerate in a sandy or bioclastic matrix (Lobitos); and medium to coarse poorly consolidated arkose lenses, and medium conglomerate interbedded with coquina sequences (Talara).

Likewise, the rock units present in the assessment area are the following:

- Paleozoic Socle, sedimentary sequence of 2 m to 5 m thick light grey quartzite, interbedded with siltstone schist and phyllite.

- Verdún Formation, that consists of a terrace formation series; its base is composed of cemented reddish conglomerate, which contains rounded clasts. Facies higher in the formation are beige in color and are predominantly bioclastic dolomitic-calcareous sandstone.

- Chira Formation, that has a thin beige colored fossil-bearing sandstone at the base, interbedded with siltstone and tuffaceous sandstone. The middle member of the formation comprises fine grey fissile shale.

- Montera Formation, thick yellowish grey feldspathic sandstone beds, interbedded with silty sandstone and conglomerate lenses, transitioning gradationally in the upper members to conglomerate interbedded with sandstone and mudstone at the top of the sequence.

- Zapallal Formation, of the Lower-Middle Miocene, deposited as a result of a sedimentation process of deep subsidence, and has the following three members: (i) lower member with three levels; (ii) middle member (Clambore Sandstone); and (iii) upper member that includes five levels.

- Miramar Formation, that contains alluvial conglomerate, poorly consolidated in a sandy matrix interbedded with sandstone lenses.

Structural Geology

Structurally there are two distinct sectors:

- Cerro Illescas Sector represents a regional metamorphism deformation event, with faulting and intense folding and foliation. There are two fault populations: (i) longitudinal faults: the main regional scale structural feature are NW-SE oriented faults with dextral movement; (ii) transverse faults: Less frequent sinistral NE-SW and E-W faults, activated during Cenozoic during extensional tectonics.

- Sechura Basin Sector, that includes three deformation stages: (i) after base Zapallal diatomites sedimentation, with direction between 340° and 350°, 1.0% to 1.4% gradient, and folds and microfaults; (ii) happened after the last sedimentation of Zapallal Formation diatomites, with 10°- 20° direction; 0.6% to 0.8% gradient and its beds are poorly folded and have faults; (iii) occurred after the Pliocene and shows faults that cut Peistoncene and Holocene beds.

Geomorphology

The main geomorphological regional features in the Bayovar deposit are the Cerro Illescas and the Sechura Desert (Sechura Basin). They are located between the Pacific Ocean and west of the Andes Mountains. The landforms are the result of Cenozoic tectonism, sedimentary processes and erosion (aeolian, pluvial eustatic changes or El Niño phenomenon). The uprising occurred during the Pleistocene and Holocene, and the tilting of the plain generated other landforms such as Virrilá Estuary, Salina Grande Depression and the Duna Gigante.

The main characteristics of the geomorphological units are presented in Table 20-3.

Table 20-3: Main Characteristics of Geomorphological Units

Geomorphological Units	Symbol	Gradient (%)	Composition	Description	Geodynamic Processes
Coastal Strip	LFL	1 – 20	Gravel, sand	Comprise the shore and 200 m inland, there are beaches and cliffs	Marine erosion, aeolian erosion
Coastal Plain	LLC	1 – 3	Sand, gravel	Located between coastal strip and the Andes Mountains, flat or slightly inclined relief, consisting of loose or unconsolidated materials	Aeolian erosion, pluvial erosion and occasionally alluvial erosion (El Niño Phenomena)
Dissected plains	LCD	0 – 15	Gravel, sand, fine material and coquina	Terraces more lifted than the Coastal Plain, composed of sand, gravel and coquina, slightly eroded	Aeolian erosion, pluvial erosion and occasionally alluvial erosion (El Niño Phenomena)
Dunes	LD	0 – 3	Sand	Sand deposit moving from south to north, barchans and longitudinal dunes (Duna Gigante)	Aeolian erosion
Depression Relief	DEP	0 – -5	Sand, coquina, pour lithified sedimentary rocks	Concave surface formed by subsidence of sedimentary sequences, slightly folded and deepened by climatic events (El Niño)	Aeolian erosion, pluvial erosion and occasionally alluvial erosion (El Niño Phenomena)
Sedimentary origin Hills and Terraces	CLS	5 – 25	Sandstones, siltstones	Trapezoidal hills shapes, low elevation, rounded, with flattened ridges	Slope erosion
Igneous and Metamorphic Hills	CLM	10 – 45	Garnets, tonalites, phyllites, schists, gneiss, migmatites	Conic hills with steep slopes, narrow and rounded ridges	Slope erosion

External Geodynamic

External geodynamic features in the area are controlled by exogenous factors like weathering, erosion, landslides, among others. These processes may mean geological risks, which could affect human activities. The identification of dynamic processes was based mainly on satellite images. The main processes identified are presented in Table 20-4.

Table 20-4: Main Geodynamic Processes

Geodynamic Process	Description	Affected Areas
Marine erosion	It occurs in the coastal strip (waves and tides) and anomalous tsunamis and tides. The resulting landforms are cliffs, bays and minor structures	Punta Aguja, Punta Bapo and Punta Lagunas, in metamorphic and igneous outcrops
Slope erosion	Ongoing process in steep slopes; its intensity increases when natural conditions vary, like the lack of vegetation or the concentration of surface runoff channels that intersect (fractures and faults)	Cerro Illescas (dry streams, gullies).This process is activate during summer and when El Niño phenomena occur
Rockfalls	It occurs on slopes and hill steep slopes, where rock outcrops show moderate to intense fracturing. Landslides caused by weathering and the opening of fractures allow the rock falls	Cerro Illescas and Bayovar (road cuts and slopes)
Eolian erosion	Transport and accumulation of fine material, mainly silt and sand building dunes. It is an ongoing process that produces abrasion, total and partial coverage	-
Flooding	Not common, specially related to El Niño phenomena. Consist of the accumulation of runoff water caused by heavy rainfall in low-lying areas, flat or zero slope	Salina Grande depression, mainly in south and southwest of Bayovar deposit

Soils

The soils in the area of assessment are undeveloped mineral soils because this area normally presents low rainfall and shallow vegetation, which causes low incorporation of organic material in the soil.

Parental materials are residual (rock weathering) and transported material (deposited material by water action, colluvial, wind and ocean). The soil temperature regimen is isohyperthermic; average annual soil temperature is equal to or greater than 22 °C and the difference between summer temperatures and winter temperatures is approximately 6 °C y. The soil moisture regimen is arid, dry in all parts for more than half of the accumulative days per year, and wet in some or in all its parts for less than 90 consecutive days.

Taxonomic Classification of Soils

Soils in the Project area are classified into two distinct landscape types; hilly landscape and plain landscape. According to an ACOMISA study performed in the assessment area in 2014, there are seven soils units identified with a common name for easy classification, and have been grouped into two subgroups corresponding to the Order Aridisols. The order includes arid and ochric soils. Table 20-5 shows Classification of the seven study area soils according to the taxonomic classification of soil (USDA 2006).

Table 20-5: Taxonomic Classification of Soils

Order	Suborder	Large Group	Subgroup	Common Name
Aridisols	Calcids	Haplocalcids	Aquic haplocalcids	East of the Yapato Pampas
				Northeastern of the Perritos dunes
				South of the Huaquillas Pampas
		Petrocalcids	Acuic Petrocalcids	Access road
				Northeastern of the Yapato Pampas
				Concession North limit
				South of the Salt Mine

Source: EIAsd of Bayovar N° 12 (ACOMISA 2014).

Subgroups identified in the soils of the Project area have been defined by means of cartographic units, grouped into two associations: Yapato – Huaquillas and Namuc – Access road.

Capacity Major Land Use

In accordance with D.S. N° 017-2009-AG, land use is the natural capacity for production under continuous treatment and specific uses. The goal of this technical system is to assign the most appropriate ground use.

Table 20-6: Cartographic Units – Consociations and Associations of Soils

Unit	Description
X	Severe agrological deficits and unsuitable for farming.
X-C3si	Land protection, with severe deficiencies of agricultural quality, and also presenting flooding problems with river water.

Source: EIAsd of Bayovar N° 12 (ACOMISA 2014).

Current Land Use

Classification of current land use within the Project area according to the International Geographical Union (UGI) identifies two distinct land use categories:

- 1st category: Urban area, government and private facilities. This category includes Puerto Rico urban area.
- 7th category: Forest Land. This category includes Algarrobal-Sapotal vegetation, represented by trees named Algarrobo (Prosoposis pallida); woody species as Sapote (Colicodendron scabridum), Faique (Acacia macracantha) and Vichayo (Capparis avicennifolia) are present too. There are shrubby strata as Encelia canescens and Galvezia fructicosa and herbaceous strata as Tiquilia dichotoma, Tiquilia paronychioides and Alternanthera peruviana.

Soil Quality

The soil quality assessment was carried out using information from 14 monitoring stations. Results were compared with Environmental Quality Standards (ECA) for soils, approved in D.S. N° 002-2013-MINAM, and with Guide Values of Canadian Council of Ministers of the Environment (CCME 2007). The most relevant results are the following:

- Arsenic (As): None of the results exceeded the ECA for agricultural or industrial soil use.

- Cadmium (Cd): All the results for this parameter were below the industrial ECA for soils or industrial CCME standard (22.0 mg/kg). At six stations (S-3, S-16, S-18, S-22, ECA-08 and ECA-09), the content of cadmium were above the ECA and CCME standards for agricultural soils (22 mg/kg), may be because cadmium compounds are part of the parent material (rock) or are effects of operating phosphate activities around.

- Selenium (Se): Content of selenium exceeded agricultural CCME standard (1.0 mg/kg) at one station; it could be related to the presence of others metals which it is linked to. Selenium does not have ECA for agricultural or industrial soil use.

Furthermore, the content of radioactive elements was analyzed using information from eight stations established in the EIAsd Bayovar N° 12 (ACOMISA 2014). Radioactive elements were compared with Reference Values of Radioactive Substances Act 1993 (RSA 93) in units Becquerels per gram (Bq/g). None of the results of uranium or thorium exceeded the RSA 93 and the concentrations of these parameters were far below reference values.

Water and Sediment Quality

The characterization of surface water quality (marine and continental) and marine sediments was developed considering information from 27 monitoring stations distributed in three sectors: Bayovar Port (11), South of Sechura Bay (8) and Virrilá Estuary (8); as well as 23 sediments stations, located in Bayovar Port Sector (11), south of Sechura Bay (8) and Virrilá Estuary (3).

The results of water monitoring were compared with ECA for surface water, Category 2 (Coastal Marine Activities) and Category 4 (Conservation of Aquatic Environment) parameters, stablished through D.S. N° 002-2008-MINAM. The results of sediments quality were compared with international standards, the Canadian Sediment Quality Guidelines for the Protection of Aquatic Life, established by the Canadian Council of Ministers of the Environment for the protection of aquatic life (CCME, 2002).

Marine and Continental Water Quality

In general, most of evaluated parameters were within the range of values reported in previous periods, without exceeding the standards of environmental quality for water.

- pH reported ranged from 6.0 to 9.0 units; being generally within the range of the ECA.

- Phosphates ranged between 0.006 mg/L to 0.49 mg/L, exceeding the ECA Cat. 2-C2 (0.03 mg/L – 0.09 mg/L), Cat. 2-C3 (0.1 mg/L) and Cat. 4-AM (0.031 mg/L – 0.093 mg/L) in all stations evaluated. Phosphates are naturally present in the Peruvian north coast given the high productivity of the waters. These high concentrations could be linked to the areas of upwelling, where the cold waters of seabed and rich in nutrients, reach the surface generating higher productivity.

- Microbiological parameters presented a maximum concentration of 790 NMP/100 mL for fecal coliform, and 2,200 MPN/100 mL for total coliform exceeding the ECA for water. These results may be related to human activities near the area of assessment, as sewers, shell fish farming, among others. The values for total coliform that exceeded standards were also present in the EIA (Golder 2007), although less frequently.

- Phenols, total petroleum hydrocarbons (TPH) and oils and fats, were reported below the limit of detection.

- Metals were reported, in general, within the standards or below the detection limit of the laboratory test method, except the values for nickel (0.00007 mg/L to 0.029 mg/L), copper (average concentration of 0.0067 mg/L and maximum value of 0.082 mg/L); mercury (average value of 0.00044 mg/L and maximum value of 0.055 mg/L); lead (0.0017 mg/L to 0.0350 mg/L) and zinc (0.0228 mg/L to 0.0865 mg/L), that exceeded standard values.

Marine Sediments

- pH values ranged from 7.7 to 8.7, similar to those reported for seawater.

- TPH (C10 - C28) reported values below the detection limit of laboratory except at the Port Bayovar sector where reached values up to 22 mg/kg.

- Total organic carbon (TOC) ranged from 2.4 mg/kg to 26.9 mg/kg. On the other hand, the average concentration of nitrate was 1.1 mg/kg at the Port Bayovar sector and 0.6 mg/kg at Sechura Bay. The values of phosphates ranged from 2.2 mg/kg to 110.0 mg/kg.

- Arsenic and cadmium were recorded above the CCME guide values. Likewise, mercury, lead, copper and zinc values were reported below environmental standards for marine sediments.

Hydrogeology

The following three important aquifers are present within the Project area:

- Alluvial aquifer of the Piura River Valley, located in the Piura river bed and extending regionally throughout the valley to the river mouth on the sea. This aquifer is characterized as clastic (fluvial and alluvial) and is typically a few meters deep.

- Overburden aquifer that extends from the foot of the Andes to the east, to the eastern base of the slopes of Cerro Illescas to the west. It is delimited by two regional faults, and made up of layers of calcareous sandstone of the Montera Formation. The metamorphic rocks of Cerro Illescas constitute the local basement. The aquifer deepens eastward, becoming a confined aquifer because the Zapallal Formation overlying and layers of sandstones intercalated with clay.

- Zapallal aquifer, that extends from north of the Piura Valley to Minchales Cascajal area to the south; consists of Tertiary sandstone of Zapallal Formation, which overlies Montera Formation, and is covered by fine sediments of wind and alluvial origin recently. This aquifer is confined by an impermeable clay horizon brackish preventing downward leakage and lower clay strata that make up most of the column of Zapallal Formation.

According to chemical analyses of existing groundwater, pH fluctuated between 6.8 and 7.8, and electrical conductivity, between 1.41 dS/m and 3.65 dS/m, representing a slight decrease of high mineralization. The salinity of the groundwater estimated from the conductivity is linked to the accumulation of salts in the first horizons of the soil and its subsequent washing during periods of significant rainfall in which there is infiltration. The evaporation and the small rains carry salts due to proximity to the sea.

The EIA of Phosphates Project (BISA 2013) indicates that groundwater levels ranged from 0.26 m to 0.56 m, from September 2004 to October 2006; and it went down to an average of 0.91 m, from October 2006 to October 2011. In the Hydrogeological Model (KCB 2013), historical records of water levels show a decrease of approximately 5 m water level, from June 2012 to September 2012. As of this date, levels remain constant, but without a recovery. In this way, it is possible to infer that groundwater levels present a consistent trend slightly downward and the flow direction is south-southwest to north-northeast, showing that the groundwater flow direction is towards the ocean.

The main mechanism that contributes to the recharge of aquifers is precipitation. The precipitation of the annual regime, discharged mainly in the Andean foothills (Alto Piura, Olmos, others), discharging in the river and the alluvial aquifer of the Piura River, which in turn, discharges in the Zapallal aquifer in their headwater. Rains associated with El Niño phenomena at the Tablazo, discharged underground of the Sechura desert, including the pampa Minchales-Cascajal and some areas of Ramon, Bayovar and Illescas. It is worth mentioning that, in normal years and typical rainfall events, the average evaporation rate is much greater than medium precipitation; therefore a significant overflow due to precipitation is not expected to occur.

On the other hand, the sources of underground water discharge from the entire aquifer system are formed by the few springs of the lower Piura Valley, wells for groundwater extraction and evaporation; the surpluses go to the Pacific Ocean according to the groundwater flow modeling.

Oceanography

This assessment summarizes the oceanographic conditions nearby the Sechura Bay where the Project will develop the shipment of phosphate concentrates at the existing JPQ Port facilities. The oceanographic study was conducted in April and June 2013, and consisted in the measurement of oceanographic parameters, tides and marine currents. 16 sampling stations were established in order to measure temperature and salinity in the water column. Additionally, an acoustic Doppler current profiler (ADCP) was installed to measure tides, waves and current marines.

- Temperature registered an average of 15.89 °C, in a range from 15.43 °C to 17.11 °C. The average temperature decreased based on the depth; the thermocline was identified between 5 m and 12 m of depth, which it is typical in areas of coastal upwelling along the Peruvian coast (IMARPE 2007).

- Salinity had an average value of 35.13 UPS, with a range from 35.09 UPS to 35.20 UPS. The analysis of the results indicates conditions of Cold Coastal Waters and Subtropical Surface Waters mix.

- Wind had a southwest predominant direction, with an average speed of 4.7 m/s.

- Tides, in the north location of the ADCP, the sea surface had 2.10 m of amplitude, while in the south location this amplitude was 0.88 m.

- Wave height oscillates between 0.19 m and 0.41 m (average height of 0.29 m) in the north, and from 0.18 m to 0.35 m in the south (average height of 0.26 m). The standard deviation was 0.05 m and 0.04 m, respectively. This variation could be explained by the influence of intermittent waves caused by atmospheric disturbances that take place in the Southern Hemisphere. The predominant wave propagation direction was northwest, north-northwest and north.

- Marine currents at the sea surface north of assessment area, had maximum speed of 33 cm/s, while bottom/depth reached maximum values of 18 cm/s. In the south, these values were 27 cm/s y 11 cm/s, respectively. The surface currents flowed mainly to the northwest and presented a synchrony between the decreased flow velocity and the change in its predominant direction. The bottom currents exhibit greater dispersion, with a southwest direction trend.

Air Quality

The characterization of air quality and noise in the assessment area was carried out using information from eight stations for air quality and five stations for noise. The results of air quality monitoring were compared with the ECA for air, established through D.S. Nº 074-2001-PCM, D.S. Nº 003-2008-MINAM and R.M. Nº 315-96-EM/VMM, or with Ontario's Ambient Air Quality Criteria (Ontario Ministry of the Environment, Canada) in absence of the first.

- Particulate matter: none of the results exceeded the Air ECA for PM10 and maximum values were recorded in CA-3 station in November 2011 (40.0 µg/m^3) and April 2012 (41.0 µg/m^3), probably because of removal of soil particles by wind erosion. Air ECA for PM2.5 was exceeded in AI-BY-01 station for two of the seven days of monitoring period, mainly by emissions from ships in Petroperu Port, concentrated phosphate shipment and sea salt (aerosols) from the ocean by action of northwest local winds.

- Gases: none of the results exceeded the Air ECA for H_2S, N_2O, CO and O_3; most results of SO_2 were below the limit of quantification and only a value of this parameter, registered in AI-BY-01 (41.4 µg/m^3), exceeded the Air ECA, probably for the traffic exhaust in Petroperú Port.

- Metals monitored were lead, arsenic, cadmium and copper, using a filter of PM10, none of the values exceeded Air ECA, respectively.

- Organic compounds: none of the results exceeded the Air ECA for total hydrocarbons expressed as hexane; and the maximum value was recorded in AI-BY-01 (70.8 µg/m^3) station; benzene results were below limit of quantification.

Noise

The information of environmental noise available for the assessment area was compared with ECA for noise regarding industrial areas (D.S. Nº 085-2003-PCM); these standards establishes a set of maximum noise levels in the environment, which must not be exceeded in order to protect human health. These values correspond to the values of weighted equivalent continuous A pressure (L_{AeqT}), in decibels units (dBA).

Data of noise stations is from the monitoring performed in November 2011, April 2012, July 2012 and June 2013, during day and night time. All noise levels did not exceed the ECA both in day and night times. Maximum values expressed in L_{AeqT} were related to traffic sounds or ocean waves, according the location of monitoring stations.

Biological Components

This section contains information regarding the biological components of the area in which the Project is located. This information is based on studies performed in the area, therefore considering the data available representative for the Bayovar 12 Phosphate Project.

The Project is located in the ecoregion Coastal Desert (Brack 1986), an environment characterized by a dry semiarid weather with scarce annual precipitation and relatively high temperatures. The area in which the Project is situated comprises two life zones defined by the Peruvian Ecological Map (INRENA 1995):

- Tropical Premontane – Super arid Desert, located in the warm temperate latitudinal strip along the southern coastal desert of Peru. The mean annual temperatures range from a maximum of 24 °C to a minimum of 19.7 °C. Annual precipitation oscillates between 5.4 mm and 59.6 mm. The landscape varies from flat to slightly curved, typical of elevated plains of the southern coastal region; to rough or abrupt, in the steep slopes of the occidental Andes basin. Vegetation is very scarce and characterized by xerophytes, primarily *Prosopis pallida (*algarrobo), *Capparis angulat*a (sapote) and *Gynerium sagitatum (*caña brava). The use of this life zone is limited to agriculture along irrigated alluvial valleys, as well as livestock activities.

- Tropical – Super Arid Desert, located in the warm temperate latitudinal strip along the southern coastal desert of Peru. The mean annual temperature is 24 °C, with an annual precipitation oscillating between 62.5 mm and 125 mm. The landscape varies from flat to slightly curved and is subject to intense eolic erosion. Vegetation is scarce and limited to disperse xerophyte type of shrubs and grasses used for occasional pasture; the predominant species include *Prosopis* sp. (algarrobo) y *Capparis* sp. (sapote).

20.2.1 Conservation Interest Areas

Throughout the Sechura province, there are four conservation interest areas at different stages of the evaluation process and categorization by Peruvian environmental authorities.

- Virrilá Estuary: located approximately 30 km from the Project and constituted by an area rich in aquatic resources and hosts a high diversity of migratory and resident birds (CDC-UNALM 1992). In 1998, Wetlands International included Virrilá as a wetland of high biological value with significant benefits for human population with low impacts and threats. Furthermore, this area was declared as an Important Bird Area for Conservation (Birdlife 2012), as well as meets the requirements to become a Ramsar site and also form part of the Hemispheric Network of Shorebird Reserves.
- Reserved Zone Illescas: created in 2010 through the R.M. N° 251-2010-MINAM and located approximately 50 km from the Project. It is characterized by several vegetation formations that provide special habitats adapted to the extreme conditions of aridity and humidity, characteristics of the coastal desert of Peru. It is an important refuge for threatened endemic wildlife, especially bird life, as well as resting and feeding place for migratory birds from Nearctic and Austral.
- San Pedro Mangroves: included on June 2008 in the International Wetlands List from the Ramsar Convention, issued for the protection of wetlands. This fragile ecosystem is located approximately 60 km from the Project and is home for a great number of migratory shorebirds, especially the species *Calidris alba* (Sanderling); additionally, 23 species of shorebirds were registered here. This area was also defined as an area of regional importance of the Shorebird Reserve Hemispheric Network.
- Regional Community Conservation Area: in 2010, under the Participatory Biodiversity Conservation Dry Forest in the north Coast of Peru Project, SWISSCONTACT prepared the "Diagnosis and Elaboration of Technical Records in the Priority Areas for Conservation in the Dry Forests in Tumbes, Piura and Lambayeque". In this technical record, the proposal was to create the Regional Community Conservation Area of Virrila – Ramon and Ñapique was included. The Regional Government of Piura, through the Regional Council Agreement N° 672-2010/GRP-CR, December 23rd, declared it as regional interest, ratifying its ecological importance to create such regional community conservation area.

20.2.2 Terrestrial Biota

20.2.2.1 Flora and Vegetation Types

The assessment area has a total of 54 species of flora was registered, distributed among 27 families. The vegetation has a low population rate, due to the extreme conditions of aridity. However, the species registered in the area adapted to such conditions and developed seasonal growth.

Although the Bayovar 12 concession is located primarily in a desert (Figure 20-3), the following vegetation types are identified regionally within the Project Area:

- Coastal Desert: Occupies sandy soils along the coast and on occasions forming small dunes. Characterized by severe drought with non-dominant vegetation; however in some cases some types of individuals near and adjacent to vegetation species such as halophytes and xerophytes can be observed.
- Halophytic Vegetation: Restricted to areas near beach banks and some sectors of the Virrilá Estuary. It is situated along the coast, where the occasional influx of seawater allows the development of plants adapted to high salt quality soils, forming discontinuous vegetation with small patches of both halophytes and coastal grasses. In the assessment area, this type of vegetation is distributed as a discontinuous coastal fringe in front of the Cerro Illescas, as well as the banks of the Virrilá Estuary. The main species identified are succulent plants such as Sesuvium portulacastrum, Batis maritime and in some areas there is presence of Distichlis spicata and Sporobolus virginicus.

Figure 20-3: Typical Landscape, Bayovar 12 Concession

- Algarrobal – Sapotal: This vegetation comprises most of the area surrounding the Project occupying sand and silt soils with scarce water availability. The slopes are light with formation of small shoals that lies close to the sea and dunes. This vegetation is dominated by *Prosopis pallida* (Algarrobo), *Colicodendron scabridum* (Sapote), *Acacia macracantha* (Faique) and *Capparis avicennifolia* (Vichayo); shrub species include *Encelia canescens* and *Galvezia fruticosa;* and an herbaceous layer composed of *Tiquilia dichotoma, Tiquilia paronychioides* and *Alternanthera peruviana.*

- Xerophytic Vegetation: Confined to the rocky areas, particularly throughout the Cerro Illescas, constitutes an important source of water collection and storage through seasonal drizzle from the ocean. The soil is poor with few nutrients. The representative species include *Cryptocarpus pyriformis, Maytenus octogona, Parkinsonia aculeate, Prosopis pallida, Haageocereus* aff. *Pacalaensis* and *Melocactus peruvianus.*

- Mangrove Vegetation: Observed in the San Pedro Mangrove and composed essentially of woody vegetation, where the main vegetation is a dense forest on a muddy and brackish ground. The main species found in the San Pedro Mangrove are *Laguncularia racemosa* (jeli de mangle) and *Avicenia germinans (Jeli salado)*; with halophytic vegetation in patches on the borders.

- Aquatic Vegetation: This vegetation is scarce in the areas surrounding the Project, but found in the Virrilá Estuary, and Ñapique and Ramon Lakes. The vegetation are rooted submerged hydrophytes such as *Ruppia maritima, Potamogeton striatus* and *Potamogeton* sp*.* along the banks of the Piura River and some of its canals, were registered *Typha domingensis*.

20.2.2.2 Fauna

The fauna in the assessment area is represented by mammals, birds, amphibians and reptiles.

- Mammals: The summary of mammals registered in the assessment area is composed of 24 species, 18 families and 8 orders; from which 20 species are terrestrial and 4 are marine species. The terrestrial species are divided into three main groups: small, medium and large mammals. The groups of small terrestrial mammals most represented in the area are bats (Chiroptera order), with five species composing the groups, followed by the mice group (Rodentia) with four species. The large terrestrial mammals group is represented by *Lycalopex sechurae* (Sechura fox) and *Conepatus semistriatus* (small fox). The *Lycalopex sechurae* is included in the IUCN Red List of Threatened Species in the category of Near Threatened. Four marine mammals were registered in the 2014 evaluation, through visual siting's (*Delphinus capensis* and *Otaria flavescens)* skeleton records (*Ziphius cavirostris)* and interviews (*Megaptera novaeangliae)*.

- Birds: Birds registered in the assessment area comprise 128 species, 44 families; distributed as 76 in aquatic environment and 55 in terrestrial environment. Among the species found in the terrestrial environments were *Synallaxis stictothorax, Piezorhina cinerea, Conirostrum cinereum, Amazilia, Pyrocephalus rubinis, Geositta peruviana, Pseudelaenia leucospodia and Polioptila plumbea.* Flocks of *Phoenicopterus chilensis, Ardea alba, Egretta thula, Larus dominicanus, Larus belcheri, Chroicocephalus cirraphalus, Sula variegate, Mycteria Americana, Himantopus mexicanus, Flaco peregriunus, Pelecanus thagus, Phalacrocorax brasilianus and Fulica ardesiaca* were among the species observed in the aquatic environments. In the area, 9 species are considered Nearctic migratory marine birds that visit the sea near the Project area in the summer months. The presence of the following species is highlighted: Charadriidae family, Scolopacidae family and the Laridae family. During the site visit conducted by Golder in May 2015, a large number of *Phoenicopterus chilensis* (Flamingos) were observed in the Niña Lake, close to the Project site.

- Amphibians and Reptiles: The summary of amphibians and reptiles registered is composed of 14 species, 8 families and 3 orders; from which 2 species are amphibians and 12 species are reptiles. Among the reptiles, the dominant families, represented by three species each, were Tropiduridae (*Microlophus occipitalis, Microlophus peruvianus* and *Microlophus thoracicus)*, Teiidae (Callopistes flavipunctatus, Dicrodon guttulatum *and* Dicrodon heterolepis) and Phyllodactylidae (*Phyllodactylus kofordi, Phyllodactylus microphyllus* and *Phyllodactylus reissii)*. The other 3 families only registered one species each: Elaídae (*Micrurus tschudii),* Colubridae (*Oxyrhopus fitzingeri)* and Cheloniidae (*Chelonia mydas).* There is only one marine registered reptile, *Chelonis mydas* (Green turtle).

The socio-economic value is determined by the uses identified, which include food, furs, crafts, medicine and rituals, harmful, as pets and as guano species. The species considered to have a socio-economic value in the assessment area are 5 species of mammals (*Lycalopex sechurae*, Sechura fox; *Conepatus semistriatus,* small fox; *Leopardus colocolo*, among others), 7 species of birds and 5 species of reptiles (*Callopistes flavipunctatus,* Iguana; *Dicrodon heterolepis*, Lagartija de Cabeza Colorada; *Chelonis mydas,* Green turtle; geckos *Phyllodactylus kofordi* and *Phyllodactylus microphyllus*).

A total of 53 species of conservation concern were identified in the Project area: 8 mammals, 40 birds and 5 reptiles. Additionally, 18 endemic species were registered, out of which 2 are mammals, 13 birds and 3 reptiles. From the total, 9 species of birds are endemic to the Humboldt Current. The species with the highest conservation category are: *Phytotoma raimondii, Sternula lorata, Sterna hirundinacea* and *Chelonis mydas.* Much of this area comprise the area of environmental indirect influence.

20.2.3 Aquatic Biota

The evaluation of the aquatic biota in the inland water bodies, include the Piura River, the Virrilá Estuary, San Pedro Mangroove, San Ramon and Ñapique Lakes, as well as temporary wetlands Las Salinas, Chocol, Zapayal and Ñamuc.

- Virrilá Estuary: A total of 41 phytoplankton species were registered, distributed in 3 groups. The Bacillariophyta division was the most representative with 25 species, followed by Cyanophyta with 14 species and Chlorophyta with 2 species. A total of 17 zooplankton species were registered, distributed in 6 taxas: Arthropod (6 species), Rotifer (5 species), Protozoa (3 species), Nematode (2 species) and Ciliate (1 specie).

Likewise, a total of 31 macrobentos species were registered distributed in 3 groups: Annelida (22 species), Arthropod (7 species) and Mollusc (2 species). In regard to fish, a total of 12 species were registered, belonging to Phylum Chordata; *Mugil* sp., *Eugerres periche* (mojarra) and *Albula vulpes* (zorro) are the most representative.

- La Niña Lake is the largest wetland within the Piura River aquatic system; however it is a temporary wetland, occurring only during the El Niño Phenomenon. Along the lake there are numerous sandbanks, of great importance for birds. A total of 18 phytoplankton species have been identified in this lake, the Bacillariophyta is the most representative group (9 species), followed by Cyanophyta (6 species) and Chlorophyta (3 species). A total of 4 zooplankton species were identified, distributed in 4 groups Rotifer, Arthropoda, Nematode and Protozoa. In the case of macrobentos, 3 species were identified, all from the Gastropod class; with *Heliosoma* sp. as the most representative. The fish in this wetland are represented by 7 species of the Phylum Chordata: *Mugil cephalus, Oreochromis* sp., *Cyprinus carpio, Dormitator latiforns, centropomus* sp., *Trichomycterus* sp. and *Poecilia reticulata.* This lake is used temporary by artisanal fishermen.

- San Ramón and Ñapique Lakes: A total of 58 phytoplankton species were identified, distributed in 5 divisions: Cyanophyta (28 species), Bacillariophyta (15 species), Chlorophyta (10 species), Euglenophycota (4 species) and Pyrrophycophyta (1 species). A total of 25 zooplankton species were identified, distributed in 3 taxa: Rotifer (17 species), Arthropod (6 species) and Protozoa (2 species). The macrobentos group was composed of 24 species distributed in 2 groups: Insecta (18 species) and Gastropoda (6 species). A total of 13 species fish of the Chordata Phylum were identified, with the most representative being *Bryconamenicus breviscostris, Bryconamenicuas peruanus, Poecilia reticulate* and *Oreochromis niloticus niloticus.*

- San Pedro Mangrove: Previous studies as EIA (Golder 2007) identified 45 phytoplankton species: Bacillariophyta (30 species) as the most diverse, followed by Cyanophyta (6 species), Chlorophyta (6 species), Euglenophycota (2 species) and Pyrrophycophyta (1 species). A total of 15 zooplankton species were identified, distributed in 4 taxas: Rotifer (7 species), Arthropod (5 species) and Nematode (2 species) and Ciliophora (1 specie). In regard to macrobentos, a total of 16 species were identified, distributed in 6 groups: Insecta (5 species), Gastropoda, Oligochaeta and Polychaeta *(*3 species each), and Malacostraca and Hirundima (1 species each). In the case of fish, 12 species of the Chordata Phylum were identified; the most representative are *Mugil* sp., *Centropomus* sp., and *Poecilia reticulata.*

20.2.4 Habitat and Marine Biology

The assessment area for marine biology covers the intertidal and subtidal zones of the southern coast of the Sechura Bay in Piura. In regard to the habitat, the intertidal zona presents 3 types of habitats: sandy beaches, mixtes beaches (composed of sedimentary rocks and sand) and rocky shores. The subtidal zone presented five types of substrate: sand, fine mix, silt, coarse mix with sand, and silt with organic remains. The sea water column was well mixed and did not present marked variations between surface and bottom; however, the average values of temperature and dissolved oxygen decreased slightly with depth, while values indicated the predominance salinity water mixture formed by the Peruvian Coastal Current and the warm Equatorial Surface Waters.

A total of 45 phytoplankton taxa were identified. In relation to zooplankton >150 µm, a total of 54 taxa were identified. As well as to zooplankton >3,000 µm, a total of 51 taxa was identified, where the dominant phylum was Arthropod, the predominant taxa was *Calanus* sp. (Calanoid), *Acartia tonsa* (Calanoid) and *Pinnotheridae* (Decapod).

Seaweed was distributed in two zones: subtidal and intertidal zones. The seaweed identified in the intertidal belongs to two divisions, Chlorophyta division (4 taxa) and Rhodophyta (5 taxa). In the subtidal zone, the 2 divisions identified were also Chlorophyta division (4 taxa) and Rhodophyta (5 taxa), with the highest biomass belonging to *Caulerpa filiformis, Rhodymenia flabellifolia* and *Ulva lactuca.* The high biomass of *Caluerpa filiformis* is a consequence of the favorable conditions for its development and serves as an adequate substrate for the settlement of *Argopecten* *purpuratus* (Peruvian scallop locally named "concha de abanico"). Peru has become the third largest exporter of "concha de abanico" and Piura region contributed with 78.6% of national production in 2012; currently, there are 181 active aquaculture concessions and authorizations in Sechura Bay (National Aquaculture Cadastre, Produce that are shown in Figure 20-4. These species could be affected by organisms producing algal blooms, as it happened in 2007, when red tide caused by the non-toxic dinoflagellate Akashiwo blood for one month occurred and caused the death of "conchas de abanico".

The macrobentos community identified in the intertidal zone can be divided into 3 substrates: sandy intertidal, mixed intertidal and rocky intertidal. In the sandy intertidal zone, the macrobentos community identified was composed of 7 taxa belonging to 3 phyla: Annelid (2 taxa), Arthropod (4 taxa) and Nemertea (1 taxon). In the mixed intertidal zone the macrobenthos community was composed of 22 taxa belonging to 5 phyla: Annelid (8 taxa), Arthropod (6 taxa), Cnidaria (1 taxon), Mollusc (6 taxa) y Nemertea (1 taxon). In the rocky intertidal zone, a total of 17 taxa were identified, belonging to 4 phyla: Annelid (1 taxon), Arthropod (4 taxa), Cnidaria (2 taxa) and Mollusc (10 taxa). The macrobenthos identified in the subtidal zone was a total of 106 taxa, belonging to 7 phyla, with the most important in relation to the total abundance were Arthropod, Annelid and Mollusc.

The fish community was composed of a total of 10 taxa, with an abundance of 235 individuals and a biomass of 17.63 kg. The species most frequently caught were *Paralabrax humeralis* ("cabrilla"), *Auchenionchus* sp. ("trambollo") and *Mugil cephalus* ("lisa").

An important resource in the Sechura Bay, *Argopecten purpuratus,* (Peruvian scallop), was evaluated at adult and larval population level. The greatest adults and larvae density was observed within the natural bank in the Sechura Bay, while near the port facilities this species was observed in lower numbers. As bycatch species the most frequent species were the Mollusc *Crucibulum spinosum* and *Nassarius dentifer.*

In regard to bioaccumulation organisms, 6 species of invertebrates and 5 fish were selected to analyze the concentration of 32 metals. The Mollusc collected, except *Thais chocolate* (black snail) and *Crossata ventricosa* (pink snail), presented cadmium concentrations slightly exceeding FAO (1983) values. Likewise, *Loligo gahi* (Squid) and *Solenosteira cf. fusiformis* (Snail), presented copper concentrations slightly exceeding FAO (1983) values. In regard to the Arthropod, *Hepatus chilensis* (cangrejo puñete) collected near the Port of Miski Mayo, presented higher copper and cadmium concentrations in relation to FAO (1983) values.

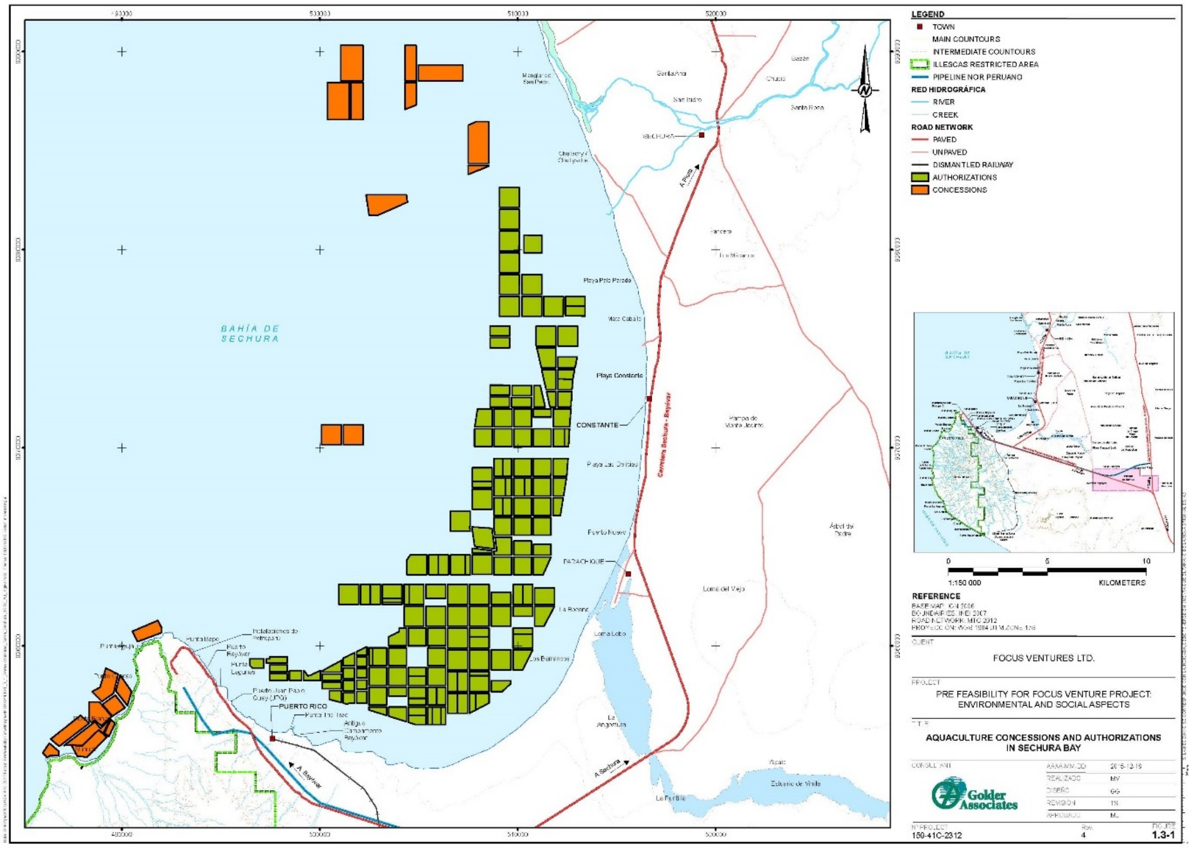


Figure 20-4: Aquaculture Concessions and Authorizations in Sechura Bay

20.3 DESCRIPTION OF THE SOCIOECONOMIC AND CULTURAL ENVIRONMENT

- Location and Territorial Division: The Province of Sechura is located in the Piura Region, approximately 55 km southwest of the city of Piura, in the area called Bajo Piura. It extends over 6,369.9 km^2 and it is politically divided into six districts: Bellavista de La Unión, Bernal, Cristo Nos Valga, Rinconada Llicuar, Sechura and Vice. The Sechura Desert is located in the province (considered the largest in the country) where Bayóvar Depression is extended, an area rich in brine, phosphates and other non-metallic mineral deposits. Sechura province hosts the Farmers' Community San Martín de Sechura (CSMS, by Spanish acronym) established on 1544, that comprises an approximate extension of 796,000 ha, from which only around 300,000 ha have been recorded. CSMS, surface title holder of more than 226,000 ha of the community, was created in 2004 under the Cooperation Master Agreement for the Bayóvar Project Sustainable Development. The contribution received by the community from the extraction companies for surface, beneficial use and easement rights are administered by the Community Foundation.

- Demography: According to the population forecasts from the National Institute of Statistics and Information (INEI, by its Spanish acronym) the Province of Sechura has 75,652 inhabitants/year as at 2015, representing 4.1% of the regional population. The province has experienced, between 1981 and 2015, a significant population growth, getting to 4% per year. This growth was fostered, among other factors, by immigration in the District of Sechura (mainly in Sechura village) since the 80's due to the expectation of job opportunities created by fisheries and the development of non-metallic mineral resources activity. One third of the population is under 15 years old and 62% between 15 and 64 years old. The latest age group corresponds to the economically-active population, supporting the economic burden expressed in the dependence ratio of

61.2 for every 100 individuals. The masculinity index in the province is 99.6 male/100 female while in Sechura, this index is 93.8 male/100 female.

- Education: In the Province of Sechura, the reading and writing skilled population over 15 years old represents around 93%, and respect to the educational level attained, the elementary-school level population (42% average) is dominant, and the further studies population (technical and university) percentage is under 10% average.

- Health: The main health services in the province are supplied by the Ministry of Health (MINSA) recording a total of 18 facilities, from which 4 are health centers and 14 are health posts. Density of medical doctors per every thousand inhabitants is 0.3 reflecting a deficit in health-care coverage in the province. The most frequent reasons for consultation in the region were acute respiratory infections, illnesses in the buccal cavity and intestine illnesses[15]. On the other hand, chronic malnutrition affects 29.2% of children under 5 years old in the province, being considered a health problem of medium prevalence according to the World Health Organization (WHO). Most population in the province (70.4%) is affiliated to the Health Integral System (SIS, by Spanish acronym), a system of public insurance which gives priority to health-care for populations in poverty and extreme poverty.

- Housing and utilities: Most dwellings in the province are independent houses; more than 90% are owned. The dominant material of external house walls is brick or cement block but it is also used the plywood. The floor is earth or cement; while the roof material is mainly calamine sheets. Coverage of public network water utilities inside the house is 73.5% average, but more than 95% of houses do not have this service 24 hours daily nor the 7 weekdays. To compensate the lack or deficiency of this utility, the population resort to other water supply systems, such as community supply tanks, tank trucks and informal connections from the neighborhood. More than 90% of houses have power supply service and 60% use gas as fuel for cooking.

- Transportation and communications: The main transportation means in the province is land transportation. Road infrastructure comprises 215.7 km of national roads and 294.6 km of neighboring roads. The surface of the national network is in good working conditions being dominant the types of asphalt and semi-paved surfaces. Port infrastructure comprises the Port of Bayóvar and the artisanal fishing docks of Las Delicias, Parachique-La Bocana and Puerto Rico coves grouped under the Association Front of Artisanal Aquiculture Fishermen of Puerto Rico Bayóvar (AFREPAAC) located in the Sechura Bay. Regarding the information technologies and communication (ITC), mobile phone and TV sets are the ITC with the largest number of users in the Province (87% average); the radio is also a communication means with a significant representation, mainly in Sechura town.

- Land resource: The Province of Sechura lies in the coastal littoral subspace having an extensive marine-coastal area with important hydrobiological diversity, where fisheries and mariculture are developed. Also, Sechura Desert is located in the Province comprising one third of the land where the largest Peruvian phosphates reservoir is located, currently developed by mining companies. There are also salt flats in Sechura Desert used by CSMS[16]. The agricultural area comprising only 5% of the land.

- Water resources: The Río Chira-Piura basin supplies water for population, agricultural, industrial and mining use. The water for agricultural use is under the irrigation system of Medio and Bajo Piura, which integrates to the irrigation sector of Sechura. This sector has four irrigation users committees comprising more than 11,000 users in a total area of 13,067.25 ha, from which 98% approximately are under irrigation.17 Currently,

[15] MINSA 2013. Available in: http://www.minsa.gob.pe/estadisticas/estadisticas/Morbilidad/CEMacros.asp?08.

[16] 2013 Piura Regional Government. Diagnosis of Sechura Coastal Marine Area.

[17] Group INCLAM 2012. Diagnosis of Water Resources Management in Chira-Piura Basin.

the water service for population consumption is handled by the Consortium Pro-Gestión and the source of water supply is the well Altos Negros, located 11 km from CP Sechura. Drinking water service in CP Puerto Rico, Playa Blanca cove and Illescas hamlet is in charge of Bayóvar Water Supply Center owned by the Piura Regional Government, which has the Bayóvar Well as the water supply source located at Illescas hamlet.

- Economic Profile: Is basically primary with dominance of fisheries (artisanal and industrial) and agriculture (small agricultural producers), occupying approximately 53% of the economically-active population (PEA, by Spanish acronym). The commercial and services activities concentrate in the District of Sechura, as it is the capital of the Province and headquarters of most non-metallic mining companies extracting phosphates, salt and aggregate material. Entrepreneurial mining employs less than 1% of PEA in the Province. 79% of households have an income over the minimum salary, indicating that families can cover the household expenditures and satisfy their needs in terms of money.

- Agriculture: This activity is developed in north of the Province of Sechura; production concentrates in crops oriented to the internal market, being the main ones cotton, paddy rice and hard yellow maize. In general, Sechura producers sell their products to the collection centers located in the District of La Unión and the city of Piura with the expectation of receiving better prices.

- Cattle raising: Extensive-type cattle raising is mainly developed in two areas, southwest the Province of Sechura, from the Virrilá Estuary up to Cerro Illescas foothills, and in the District of Cristo Nos Valga, northwest the Province of Sechura. Goat cattle raising is the most important, but also beef cattle and sheep cattle are raised in a lower rate, as well as work animals, such as horses, mules and donkeys. CC San Martín de Sechura has granted land to Empresa Comunal de Servicios Agropecuarios San Cayetano S.R.L. (Ecomusa) for cattle rising, that is the beneficial user of 30,000 ha in the area of Illescas grouping 80 families working in extensive raising of beef, sheep, goats and pork cattle. These families do not live permanently in the area and, generally, combine this activity with agriculture or fisheries. Water supply for cattle is from the Bayóvar Water Supply Center.

- Fisheries: Fishing is one of the main production activities developed in the Province, mainly in the District of Sechura. Approximately 29% of the population employed work in fisheries and mariculture, developed in the Sechura Bay. In CP Sechura this percentage represents 39% of the population. Most fishermen choose fishing mackerel and kelp bass, due to their large demand in the national market and the canning industry. Another important hydrobiological resource is scallops rising, due to the important economic returns. More than 50% of hydrobiological resources obtained are for sale and most of them traded in the District of Sechura. A large number of farmers from CSMS work in artisanal fisheries in the Virrilá Estuary and the lakes Ñapique, Ramón Grande, Ramón Chiquito, Peñal, Sánchez, La Mariposa, El Tisal, La Niña and others, where they catch a variety of fish, as well as white crayfish, crabs and shrimps.

- Mining, Oil and Manufacture: The Province of Sechura is an area rich in non-metallic minerals with a very special value for agriculture and modern industry, such as phosphates, brine, dolomite, gypsum, sulfur, clay and others. Likewise, sulfur deposits are located south of Cerro Illescas, 4 km from the shoreline of the area Reventazón in Sechura Desert; also, brine deposits are located in Zapayal, Namuc and Ramón Grande basins, with estimated reserves of over 1,400 million tons. The most important activities are the following:

    - Bayóvar Phosphates Mine of Compañia Minera Miski Mayo, owned by Vale S.A., Mosaic and Mitsui & Co Ltd., this non-metallic mine is located 60 km southwest the town of Sechura and started operations in 2010, with a total estimated reserves of 597 million tons.
    - Fosfatos del Pacífico S.A. (Fospac) company with a non-metalic phosphate mining project located close to kilometer 30 of the Bayóvar-Chiclayo highway. This company is owned by Cementos Pacasmayo S.A.A. (70%) and Mitsubishi Corporation (30%). Fospac has also a modern brick plant in the Sechura Desert that produces 50 000 tons/year of diatomite bricks.

- o Juan Paulo Quay S.A.C. (JPQ), owner of the Bayovar 12 concession, undertakes exploitation of gypsum within the claim. Shareholders of JPQ own port facilities for the exportation of mining products, located 40 km from the Bayovar N° 12 Project.
- o Americas Potash (Growmax) is a non-metallic mining company with surface rights of 82,000 ha located in the Sechura Desert. The company acquired the concession for brine development and entered into an agreement with CSMS for surface access right during 33 years with the possibility for an agreement extension of up to 100 years.
- o UEA Fosyeiki holds a group of non-metallic mining concessions totaling 1 600 000 ha. At present the mining authority has only approved exploration of phosphates on an area of 907 ha. This group on concessions are overlapping the concessions Bayóvar 13 and Bayóvar 16.
- o The final section of North Peruvian Oilpipeline crosses the Sechura Desert; it supplies oil to the Bayóvar Terminal located in Sechura Bay, from Loreto, in the Peruvian jungle.

- Social development: The level of social development attained in the Province of Sechura is measured using the Human Development Index (HDI) prepared by the United Nations Development Program (UNDP) and the Unsatisfied Basic Needs (UBN) approach developed by the Economic Commission for Latin America and The Caribbean (ECLAC). HDI in the Province of Sechura has positively evolved between 2007 and 2012, going up 15 positions in the national ranking for above period (from position 53 to 68 at the Peruvian provinces level). Regarding UBN, allowing determining non-monetary poverty based on the basic needs covered by the population, it can be noticed that incidence of total poverty (households with at least one UBN) and extreme poverty (households with two or more UBN) is 32.2% and 22.2% respectively). The most compelling UBN in the Province of Sechura are the unsuitable physical characteristics of housing (32.2%) and the lack of excreta disposal system (23.9%).
- Social and political organization: Currently, the Province of Sechura is mainly driven by three institutions representing the interests of most of the provincial population, namely the Sechura Provincial Municipality, CCSMS and the fishermen and mariculture farmers' unions working in Sechura Bay. The organizations representing minor groups with specific interests are also present such as the Association of Artisanal Fishermen of the Ñapique and La Niña Lakes, located 40-km from the project.

- Cultural heritage: The Province of Sechura has a varied cultural and tourist heritage which combines architectonic, historic, religious and natural elements, resulting in a Mestizo profile inherent to the province. Pre-Hispanic archaeological and colonial infrastructure (churches) remains have been recorded. With regards to natural richness, the province has beautiful beaches, lakes, mangrove areas and dunes which are tourist attractions.

- Perceptions: Regarding the mining activity, the population in the Province of Sechura perceives it as an activity creating positive and negative impacts. The mining activity is considered positive because it offers job opportunities and contributes to streamlining the local economy through the intensification of the commercial activity and the proliferation of different business lines. On the other hand, mining is perceived as an activity likely to affect the environment and could entail certain damages to other pre-existing activities in the province, mainly those referred to local production activities such as agriculture, cattle raising and fishing.

20.4 ENVIRONMENTAL ISSUES

The main Project facilities will include: pit area, tailing disposal, waste storage, process plant and auxiliary facilities (port facilities, seawater intake, and camps). The environmental components related with these facilities are: air, noise and marine water.

The most important environmental aspects in the mine area are related to the generation of particulate matter (dust) and noise generation. Main activities of the Project include material transport and water management. In port area, project facilities will be related with marine environmental components (aquatic biology, oceanography).

The following potential social and environmental impacts related to the Project were identified:

- Noise generation and air quality change: Expected due to the increased traffic, land movement and personnel. As mitigation measures, the Project shall maintain an adequate vehicle fleet, access roads from and to the Project to avoid the generation of particulate material and gases. During operation of the Project, atmospheric emissions will be generated from the plant facilities, access road, these sources include sulphur dioxide (SO2), oxides of nitrogen (NOX), carbon monoxide (CO), particulate matter (PM) with a diameter of less than 2,5 µm (PM2,5), particulate matter with a diameter of less than 10 µm (PM10) and greenhouse gases (GHGs). Emissions from other sources are expected to be minimal, such as material transport and ship loading, which will be minimized by best management practices.

- The project involves a series of activities that generate noise. The main ones are the construction of some infrastructure metalworking, transport and quarrying, and vehicle back-up sirens.

- Loss of Soil: Expected due basically to the construction of tailing pond; however, the Project foresees soil separation into piles to avoid soil lost or degradation

- Loss of Habitat: decreased diversity and affected protected species and/or migratory birds due to the Project facilities and activities in the area. This shall be minimized by the Project when considering the location options for their facilities, in places presenting less biodiversity.

- Sensitive areas (national or international conservation status areas) do not exist in the Project area. However, in the study area the existence of the Laguna La Niña has been recorded, a body of water whose characteristics vary during the El Niño event, occupying areas with different extent. This lake is used as a rest area for some species of migratory birds, and where the amount of water increases considerably, can become temporary source of aquatic resources for the residents of the area.

- Although flora and fauna are scarce and the project will have less interaction with these environmental components; two species of flora are important “algarrobo” Prosopis pallida y “sapote” Colicodendron scabridum. These species are included in National Category List (D.S N° 043-AG 2006) as Vulnerable (“algarrobo”) and Critically Endangered (“sapote”).

- Demographic effects of worker migration; to be mitigated by contracting personnel from within the area of influence of the project.

- Traffic change in the main access roads: due to the increased road infrastructure and increased number of trucks and vehicles which shall occur mainly during the construction phase, but during the operation phase it shall be minimized.

- Perceptions on disturbed water quality and quantity that might arise in the minds of the local population in the province of Sechura, which shall be mitigated by providing suitable explanation to the local people of the influenced area, regarding the activities of the Project.

- Change in job opportunities and community infrastructure: due to the increased demand of workers and services in Project stages.

- Increased budget of the district, province and regional governments: due to increased incomes related to tax collection to be paid by the Project.

- The port activities could impact traditional fishing activities and mariculture. This activity is one of the main economic activities in the Bay of Sechura, specifically in Puerto Rico. This activity is performed on the seabed by use of cages. Currently, in Puerto Rico marine farmers associations and population exploit natural resources located in the bay.

20.5 REQUIREMENTS AND MANAGEMENT PLANS

Focus will implement prevention, mitigation and rehabilitation measures, related with the features and activities of the project and potential impacts. All these measures will be included in Environmental Management Plan (EMP). The EMP comprises individual plans that outline the scope of environmental management in order to compliance with applicable regulatory requirements and Focus policies.

For the design, construction, operation and closure of Project, it will be applied the following criteria:

- Archaeological Sites: Where feasible, avoidance of terrestrial and marine archaeological monuments. If monuments cannot be avoided, adherence to regulatory protocols for protection and/or rescue.

- Protected Areas: Avoidance of direct Project footprint effects to nearby protected areas and avoidance or minimization of indirect effects (e.g. marine water quality, air quality).

- Wildlife Species of Conservation Status: Avoidance of direct mortality, destruction of habitats and indirect effects to species with conservation status, including Peruvian and International Union for the Conservation of Nature (IUCN) designated species.

- Key Wildlife Habitats: Avoidance or minimization of direct and indirect impacts to key wildlife habitats (e.g., reproductive and migratory habitats).

- Visual Aesthetics: Avoidance or minimization of visual aesthetic impacts to sensitive viewing receptors, including nearby protected areas and populated areas.

- Natural and Industrial Risks: to establish Project design parameters, to minimize impacts on environmental, public health and safety, associated with natural and industrial hazards.

- Oceanography and Coastal Morphology: to the extent practical, the seawater intake will be buried to minimize effects to coastal morphology.

- Freshwater: Water is recycled into the process where practical and appropriate. The discharge of untreated wastewater, industrial water, spill, drainage, residual industrial liquids and solids into the marine environment, is prohibited unless it meets applicable water quality standards for discharge. The RO brine is recycled into the Seawater Supply Ponds and is not discharged to the surface.

An environmental quality standard (EQS) is defined in Peru as the measure that sets the concentration or degree of elements, substances or physical, chemical and biological parameters present in the air, water or soil, when acting as a receptor body, which does not represent a significant risk for the health of persons or the environment. Depending on the particular parameter being referred to, the concentration or degree can be expressed as maximums, minimums or ranges.

According to Peruvian law, EQSs are mandatory for use in the design of legal standards and public policies. In addition, EQSs are for mandatory reference in the design and application of all environmental management instruments (such as environmental impact assessments).

Likewise, Peruvian law provides that environmental certification cannot be granted through the National Environmental Impact Assessment System, in any case in which the related environmental impact assessment concludes that the implementation of the activity involved will lead to non-compliance with an EQS.

Environmental Management Measures

- It's estimated to be used 23 trucks making 4 loads per day in a 12-hour shift. Thus, a particulate matter emission control will be performed, through irrigation in areas with potential dust emission. Also it will be implemented humid management and coverage systems for concentrate and quarries materials transportation, and for concentrate shipment.

- Noise damping systems will be implemented for power equipment.

- Dewatering volumes will be maintained within design forecasts, so as not affect groundwater levels in the surrounding area.

- Residual waters from pit dewatering will be discharged in tailing facilities. Rain water collected in the open pit will be collected in one or several in-pit pump(s), until there is enough quantity to pump it out to the surface where it will be piped into the tailings line and out to the tailings pond.

- Runoff management for the Project will consider enabling protection dam (to protect the pit area) and derivation channels.

- In order to avoid impacts on groundwater quality, it should consider factors of soil permeability for liquid effluents disposal in the facilities like tailings deposit.

- To control, collect, and treat water that comes into contact with project industrial activity.

- Water intake facilities will include electric vertical turbine pumps that are mounted on JPQ's port ship loading conveyor, and process waste ("brine" o salmuera") will be disposal inside tailing pond.

- Waste generated from mining will be stored in both external pit (ex-pit) and internal (in-pit) waste storage facility (WSF). An ex-pit waste storage facility WSF adjacent to the mine pit has been designed to store mine waste as the mine pit is developed.

- The in-pit backfilling process will be implemented when the WSF reaches maximum capacity and or when sufficient space within the mined out pit is created.

Monitoring Plan

Focus will develop a monitoring plan in order to monitor and control environmental performance of the Project on compliance with Peruvian environmental standards. In order to control emissions and effluents related with air, noise and water, it must comply with the maximum permissible limits (MPL) for Mining-Metallurgical activities.

The monitoring plan will include:

- Air and Noise: environmental quality standards in main receptors.

- Marine Water Quality: environmental quality standards in port area.
- Flora and Fauna: abundance, richness and diversity for key species.
- Marine Biology: abundance, richness and diversity for key species.

20.6 SOCIAL REQUIREMENTS, PLANS AND AGREEMENTS WITH LOCAL COMMUNITIES

According the "Guide of Community Relations" released by the Ministry of Energy and Mining (MINEM, by Spanish acronym), it is necessary design a Community Relations Plan (CRP), that contents all the measures required for the social management in a project. The main objective of this plan is to regulate and fortify the relations between communities and enterprises, and to manage potential social issues.

The measures of social management should consider the development of communities, beyond the useful life of mine, and the principles of participation, sustainability and synergy. The programs proposed will be developed, if possible, through agreements with local communities and authorities, in order to guarantee their sustainability. On the other hand, the participation of population is also a requirement for CRP to improve the local capabilities.

20.6.1 Social Programs

Focus will implement a set a social programs that includes communication, training, education and health as main action lines, during the construction and operation phases of the Project.

The preliminaries social programs are the following:

- Program of communication and information.
- Program of environmental training.
- Program of fishermen training.
- Program of support sporting activities.
- Program of education support.
- Program of health support.

The estimated amount of inversion for those programs is US$ 3,150,000, which will be distributed over 20 years.

20.6.2 Agreements with Local Communities

The Comunidad Campesino San Martín de Sechura (CCSMS) owns the surface rights of Bayovar N° 12 Concession. An agreement signed in 2009 between Juan Paulo Quay S.A.C. (JPQ) and this community grants surface land use and access rights to JPQ, in order to develop the Project. The agreement includes the respective payment and other duties from JPQ, and the commitment of CCSMS to provide and maintain an adequate social context to Project development, as well as to work together with the Company promoting a sustainable development for both community and Project.

20.7 CONCEPTUAL CLOSURE PLAN

The conceptual closure plan will be prepared according the requirements of the Law N° 28090, Law that regulate the mining closure and its regulation (D.S. N° 033-2005-EM).

The closure plan considers decommissioning and demolition of facilities that can be removed completely from the site (buildings, tubes, etc.), and not going to play any role in the post-closure. For those permanent facilities that will be on the site but without any further use, the plan includes the closing and conditioning in order to minimize the risks to public health and safety and to the environment.

20.7.1 Objectives

The main objectives of the closure plan, according the Peruvian regulation, are the following:

- Safeguarding of the health and safety of individuals;
- physical, chemical and biological stability of the areas disturbed by mining operations;
- future land use;
- protection of courses and bodies of surface and groundwater;
- minimizing landscape and visual impacts; and
- Minimizing socioeconomic impacts of mine closure.

20.7.2 Closure Criteria

The conceptual closure considers the following general criteria:

- To comply with current legal requirements and best international practices.
- Remnant process solutions post-closure will be left in a state that requires minimal maintenance and passive care.
- Where possible, it will be prioritized the transfer of usable facilities to communities or to the Government, at time of final closure.
- Where possible, a progressive closure of facilities will be undertaken, during the operation phase of the Project.

Additionally, it is necessary to includes some specifically closure criteria regarding safety, geotechnical stability in the long-term (security factors), design of hydraulic infrastructure, chemical stability, quality water and soil, biological stability, future land use, among others aspects.

20.7.3 Components of Closure

According the components of Project, the following main facilities will be closed:

- Open pit.
- Waste storage facility (WSF).
- Tailings storage facility (TSF).
- Processing plant.

- Water treatment plant.
- Energy facilities.
- Water intake facilities.
- Port.

20.7.4 Activities of Closure

Activities of Open Pit Closure

- Demolition and dismantling: All the infrastructure presents in the pit will be removed or demolished. The waste disposal will be according a plan of waste management.
- Physical stability: The pit will be largely infilled with the waste material extracted during its exploitation, to reflect where possible, the initial topography of the area.
- Hydrological stability: The drainage channels will not be dismantled at the closure; during the post-closure phase the hydrological assessment of these channels will continue in order to determine the possibility of remediation.
- Land use: The land use will be similar to the extent practical to pre-mining conditions, as will the final topography will be also like before mining.
- Revegetation: It is not planned any measure of revegetation, due to the scarcity of vegetation in the area.

Activities of Tailings Storage Facility Closure

- Demolition and dismantling: Pipelines and other installations will be removed or demolished. The waste will disposal according a plan of waste management.
- Physical stability: Rehabilitation measures include the drainage and filling of this facility with selected surface material on level surfaces and shaped. The purpose of placing the cover material is selected to minimize the potential for surface water infiltration. Additional or alternative rehabilitation measures could also be implemented in the detailed closure plan.
- Hydrological stability: The drains or sewers in the area of TSF will be dismantled during the closure, in order to restore the natural flow of rain water.
- Chemical stability: The materials stored in the pit are not potentially acid-rock generating, so they will not need a waterproof cover.
- Land use: The land use will be similar to the initial condition, because the topography will be also like before mining to the extent practical.
- Reclamation: It is not planned any measure of revegetation, due to the vegetation is scarce in the area.

Activities of Waste Storage Facility Closure

- Demolition and dismantling: Not applicable.

- Physical stability: Contouring the surface of the dump, with an adequate slope and considering the security factors of design.

- Chemical stability: The waste is not Potential Acid Generator (No PAG), therefore the chemical stability will be guaranteed only with the corresponding physical stability.

- Land use: The land use will be similar to the initial condition.

- Reclamation: It is not planned any measure of revegetation, due to the vegetation is scarce in the area.

Activities of Processing Plant and Water Treatment Plant Closure

- Demolition and dismantling: The plants will be dismantled; this shall include the removal of equipment and material from the main and auxiliary facilities in order to fulfill the closure objectives. In general, the dismantling of facilities will consist of activities that are described below:

  - Removal, transfer or sale of equipment and concentrator plant materials. The necessary equipment will be left to implement the post-closure activities.

  - Removal of buildings or structures. The necessary buildings and structures will be left to implement the post-closure activities.

  - Purging, cleaning and removal of tanks, pipelines and process systems.

  - De-energized and removal of power lines that are not necessary for the post-closure.

  - All water management facilities such as pumping systems, piping, tanks, etc., will be dismantled.

  - Bypass channels will be not dismantled at the end: during the post-closure phase a hydrological assessment thereof will determine its dismantling.

  - Withdrawal, transfer or sale of all reactive chemicals or substances that are in laboratories or respective warehouses.

- Physical stability: demolition and dismantling activities will leave a stable ground physically. Only is considered a leveling off the ground areas.

- Chemical stability: as part of the decommissioning, there will be potentially areas that could have been impregnated with oil. These areas will be recovered. To do this, once removed the structures, sampling and testing shall be carried out to determine the condition of the soils. Impregnated soils will be excavated and disposed in places authorized through EPS-RS. The excavated areas will be filled with clean natural soils.

- Land use: at the end of the closing activities, the area where the plant was will have similar conditions to the environment and the use of the land would be similar to the environment condition.

Activities of Water Intake Facilities Closure

- Demolition and dismantling: Pipelines, pumps and installations that are part of the seawater intake system will be purged and then dismantled.

- Physical and chemical stability: These conditions will be feasible after the dismantling and demolition of the bases, the terrain will be physically and chemically stable.

- Land use: The area of these facilities will recover the similar condition of the environment.

Activities of Electricity Facilities Closure

- The transmission line will be not dismantled at the closure; during the post closure an evaluation will determine if is required its dismantling.

Social Programs

Social programs will be developed as part of the closure plan, and shall be framed within community relations projects. These programs could be the following:

- Temporary work program, to promote and generate job opportunities with the aim of improving the income of the local families located within the area of direct influence of the closure plan, and contributing to improve their living conditions. The population also notes that the hiring of unskilled labor must be maximized locally and through communal policy for more transparent information on available jobs and its corresponding duration.

- Training program for environmental monitoring, this program seeks to create local community awareness in about the importance and need to manage and conserve natural resources. If communities are aware of environmental issues from its scope it can motivate them to implement policies and actions to preserve the environment during operations of project closure and rehabilitation of areas disturbed by mining activities.

20.7.5 Maintenance and Monitoring Post-Closure

In the phase of post-closure it is necessary to assure that, where possible, the closure measures recover the initial conditions of the Project area; therefore, the maintenance and monitoring programs will permit to assess the effectivity of closure activities and to identify potential issues in order to apply corrective actions if will be required.

Maintenance Post-Closure

In the post-closure the facilities are subject to conditions of technical abandonment, passive or active care.

- The activities of passive and active care that are considered in the post-closure are as follows:
- Inspections of stability waste dump and tailings storage facility, according to a timetable and procedures defined;
- inspections of the physical stability of the pit;
- inspections of buildings and infrastructure remaining in the post-closure;
- inspections water supply systems;
- access control to areas to prevent disruption of operations and post-closure to protect the public; and
- Schedule and procedures of inspection and control will be defined during the final rehabilitation.

Additionally, specific contingency plans will be developed, as following:

- instability detection for waste dump and tailings storage facility;

- fault detection channels or water pipe systems;
- detection of changes in the quality of surface and groundwater; and
- Leak detection.

Monitoring Post-Closure

For those components that are not an anticipated risk, it will be defined a preliminary post-closure monitoring, estimated by five years, according to the minimum period stablished by Peruvian legislation.

This program could include the following aspects:

- Monitoring of physical stability, for example in order to verify periodically, the possible differential settlements on the slopes of the waste rock dump.
- Biological monitoring, to look after the integrity of the vegetation and soil.
- Social monitoring, to evaluate the results concerning the activities implemented as part of social programs, according to the indicators in each of the programs and mainly based on an assessment of conditions in the post-closure.

21 CAPITAL AND OPERATING COSTS

The estimated capital expenditure or capital costs (CAPEX) for the Bayovar 12 Project consists of two components:

(1) The initial CAPEX to design, permit, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, and utilities. The initial CAPEX also includes indirect costs for engineering, construction management, and Owner's costs.

(2) The sustaining CAPEX for facilities expansions, mining equipment replacements, expected replacements of process equipment and ongoing environmental mitigation activities;

The capital cost estimates reported in this section address the construction of a phosphate beneficiation plant capable of producing one million tonnes of RPR concentrate (dry basis) from two process lines at full production.

Table 21-1: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Total CAPEX ($000s)
Capex	Mine	$59,387	$160,802	$220,189
	Processing Plant	$95,567	$20,000	$115,567
	TSF	$10,255	$12,760	$23,015
Owner's Costs		$2,508		$2,508
Total CAPEX with Contingency		$167,716	$193,562	$361,279

Life-of-Mine (LOM) operating costs have been developed for mining, processing and general & administrative (G&A) costs. Operating costs include labor, equipment operation, power, fuel, reagent, and consumable consumption, maintenance and repairs, and outside services. Operating cost build-ups are described in the sections below. Table 21-2 summarizes the LOM operating costs that were derived from the financial model.

Table 21-2: Life-of-Mine Operating Costs based on 20,696,000 tonnes of DAPR Concentrate

Item	Unit Cost ($USD per Product Tonne)	LOM Cost ($000s)
Mining	$39.72	$821,932
Process Plant	$8.01	$165,747
G&A	$2.38	$49,350
Transportation	$10.09	$208,819
Total Cost	$60.20	$1,245,848

21.1 MINE CAPEX AND OPEX

IMC estimated the mine operating and capital costs of phosphorite ore production, overburden and interburden waste mining to produce 1.0 mtpy of DAPR products on an annual basis. The mine cost model assumes the pre-production to be at Year -1 and that all pre-mining overburden stripping, interburden waste mining and phosphorite ore mining (stockpiled and processed during Year 1) are capitalized. All mining activities will be performed by Focus using company-owned equipment and company employees. The IMC estimates encompassed all costs associated with mining activities, phosphorite and overburden handling, phosphorite stockpile processing, and other mine support services required for the delivery of phosphorite ore to the beneficiation plant. Capital expenditures incorporate all mobile mining equipment costs, but exclude costs associated with infrastructure development such as ROM stockpile, maintenance facilities, offices, wash house, worker camps, warehouse and storage, fuel storage and islands, and so forth. These infrastructure capital expenditures are provided by M3. The cost estimates associated with processing and other activities after the phosphorite is delivered to the hopper are provided by other parties (M3/Focus) as well as overhead, or indirect, processing operating costs. The cost model included costs for ongoing back-filling and reclamation of the open pit during the 20 years of operation, but did not take into account the expenses after closure (e.g. final reclamation and re-vegetation, building and infrastructure demolition, and haul road re-grade).

The project economic analysis was conducted by M3 with inputs from Focus and other experts and consultants identified earlier, including Gruber for beneficiation yield and M3 for estimates of infrastructure design and OPEX/CAPEX costs of all activities past the plant feed hopper. IMC did not independently verify the accuracy of these third party estimates and costs.

The cost model reflects zero-based principles for each year of production. The annual mine operating costs are estimated by combining the annual production statistics from the mine plan with the estimated equipment productivities, utilizations, and mine operating schedules. All operating and capital cost estimates assume Focus owning 100 percent equity of the Project. The estimate therefore represents the costs that Focus would incur in purchasing the required equipment and staffing the mining operation to meet the mine plan and production schedule.

For the purposes of this Study, the following costs to supply as input into the financial model:

- Direct mine operating costs: Labor, materials, and supplies
- Indirect mine costs (overhead): To support mining only
- Capital expenditures: All capital costs required to purchase the equipment necessary to operate the mine.
- Non-cash costs: Non-cash costs were not provided and will be developed by M3

Specific equipment suppliers are referenced within the capital cost estimates. The use of these equipment suppliers in the model does not represent a recommendation from IMC that Focus use these suppliers. Equipment suppliers' names represent only the equipment sizes and capacities typically used within the mining industry, simplifying cost estimation and documentation.

All costs and dollar amount referenced in this Study are expressed in terms of First Quarter 2016 US dollars (USD).

LOM unit costs for each of the for each of the three mined volumes and the LOM mining cost per tonne ore are summarized in Table 21-3.

Table 21-3: Average LOM Cash Mining Cost per Dry Tonne of Ore Mined

	$/t Mined	$/t Ore
Ore	2.20	2.20
Overburden	1.92	6.09
Interburden	1.56	5.63
Total		13.92

21.1.1 Direct Mining Costs

21.1.1.1 Direct Mine Operating Costs – Overview

Direct mine-operating costs include the required labor, and supply, and materials costs based on the mine plan schedule. Labor costs include wages for annual production, maintenance and support employees, and salaries for mine administration and supervisory staffs.

IMC used the Golder developed wage information (from the December 18, 2015 Pre-Feasibility Study) for labor calculations, including payroll burdens (e.g. payroll taxes and fringe benefits) that are typical for this region in Peru. Supply and materials costs include expenditures necessary for operating equipment and infrastructure, including costs for consumables, tires, repair parts, and other miscellaneous operating supplies. A zero-based budgeting approach estimated labor and materials costs in developing the cost model. 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.

For this Study, operating costs associated with initial overburden pre-stripping and ore mined during Year -1, and costs associated with material placement for the pit protection berm and TSF embankments are capitalized.

The direct operating costs are developed for the following unit operations of the mining activities:

- Loading: operating, maintenance and labor to load overburden and interburden with front end loaders, and interburden and phosphorite ore with surface miners
- Hauling: operating, maintenance and labor to haul ore to the plant; overburden and interburden to the ex-pit and in-pit WSFs; overburden to TSF embankment construction
- Auxiliary: operating, maintenance and labor to operate support equipment in the pit, WSFs and road construction and maintenance
- General Mine: includes truck dispatch and general labor personnel, cost for pit dewatering, road base materials, software licences, minor equipment (eg pickup trucks, light plants, etc.) operating costs and departmental costs,
- General Maintenance: includes warehouse, fuel and lube, tire and general maintenance personnel, small equipment repair and general departmental costs,
- Mine G&A: includes supervision, engineering and geology staff costs and allocation for VS&A.

Drilling and blasting costs are not included as the adjacent operating mines in the same rock formations have demonstrated that the material is free digging and blasting is not required. Pit dewatering, surface water controls, and pumping activities are included under general mine operating expenses even though their occurrence is intermittent as a response to major events such as El Niño; these costs are not incurred as daily operational expenses.

Phosphorite ore processing encompasses the costs to handle ore between the point haul trucks deliver plant feed to the plant feed hopper and/or the ROM stockpile and are the responsibility of M3. The costs associated with all activities downstream of the plant feed hopper (i.e., beneficiation process, water and power supply costs, transporting the product from the plant, load-out to an offsite location, port costs, and so forth) are the responsibility of M3 and Focus.

21.1.1.2 Direct Mine Operating Costs – Labor

IMC estimated mine operating labor requirements based on the level of equipment usage dictated by the mine plan. IMC allotted maintenance labor, support labor, mine administration, and supervisory staff to adequately support production activities and to facilitate effective mine operations. Manpower requirements necessary for the operation of primary production equipment (such as surface miners, front end loaders, 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.

The mine operations are scheduled on a seven-day per week, two twelve (12) hour shifts per day basis using four crews. IMC incorporated non-working day assumptions for paid holiday and vacation and shutdown due to bad weather. The mine is assumed to operate 360 days per year (365 total days per year less an assumed 5 days of mine shutdown due to the rain events).

Total labor cost comprises wages for mine operation employees and salaries for supervisory and administrative personnel. Annual wage rates and salaries were based on typical labor rates in Peru's mining industry (collected by Golder for the previous Pre-Feasibility Study). Higher pay-grade categories were assigned to maintenance personnel and equipment operators having greater skill level or work responsibility. General laborers or lower responsibility personnel filled lower pay grade categories. Total costs including burden and benefits formed the yearly equivalent of the base rate charges. Staff salaries were estimated using base salaries deemed competitive within the region. Both hourly labor and salaried staff have a 35% burden applied to the base rate in order to develop the total labor cost to the project. Table 21-3 lists the annual wage rates for salaried mine administration and supervisory staff for Peru. Table 21-4 lists the annual salaries for mine operators and maintenance staff.

Table 21-4: Summary of Annual Salaries for Salaried Staff

Position	Annual Base Salary	Total Annual Salary (including Burdens)
Mine Superintendent	$122,302	$187,122
Mtc Superintendent	$120,490	$184,350
Chief Geologist	$110,400	$168,912
Geologist	$70,364	$107,657
Chief Engineer	$63,103	$96,548
Planning Engineer	$46,633	$71,348
Senior Engineer	$41,991	$64,246
Surveyor	$31,861	$48,747
Clerk	$31,605	$48,356
Mtc Shift Supervisor	$31,605	$48,356
Opr Shift Supervisor	$31,605	$48,356
Warehouse Supervisor	$31,605	$48,356
Lead E&I	$21,937	$33,564
HR Representative	$21,937	$33,564
AutoCad Technician	$19,397	$29,677
Topography Personnel	$14,501	$22,187
Sampling Technician	$11,491	$17,581

Table 21-5: Summary of Annual Salaries for Hourly Labor

Job Classification	Annual Base Salary	Total Annual Wage Rate (including Burdens)
Operations Labor		
C Miner Operator	$16,800	$25,704
Haul Truck Operator	$16,800	$25,704
Dozer Operator	$16,800	$25,704
Grader Operator	$16,800	$25,704
Fuel/Lube Operator	$16,800	$25,704
FEL Operator	$16,800	$25,704
Misc. Eq. Operator	$10,800	$16,524
Assistant Operator	$10,800	$16,524
Helpers	$9,600	$14,688
Maintenance Labor		
Lube Truck Drivers	$10,800	$15,120
Tireman	$10,800	$15,120
Heavy Duty Mechanics	$14,400	$20,160
Light Duty & Welders	$10,800	$15,120
Tool Crib Attendant	$9,600	$13,440
Warehouse Attendant	$9,600	$13,440
Warehouse Clerk	$18,000	$25,200
Apprentices	$9,600	$13,440

21.1.1.3 Direct Mine Operating Costs – Material & Supply

The material and supply component of the direct mine operating cost represent the expenses incurred for equipment such as fuel, lubricants, rubber tires, repair/replacement parts and non-equipment operating supplies including maintenance supplies plus other miscellaneous general mine items.

Annual equipment operating supply requirements were estimated on a cost per machine's engine-hour basis. Note that an engine-hour is herein defined as a scheduled hour adjusted for non-consuming mechanical and scheduled non-working time to reflect the portion of total scheduled time that a piece of equipment is consuming operating supplies. The unit cost for diesel fuel (US$0.75/litre) was supplied by Focus. The cost for tires, lubricants, wear parts and repair parts are based on the costs of running the same or similar machines from the IMC data base.

21.1.1.4 Direct Mine Operating Costs – Equipment Hourly Rates

Equipment hourly operating costs are a function of the estimated hourly consumption or usage of fuel, lubricants, rubber tires, filters, and repair/replacement parts. Estimated consumption rates of fuel and lubricants for individual pieces of equipment were based on manufacturer/dealer specifications and guidelines, engineering estimates, and actual operating data on file at IMC. 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 are based on the estimated tire lives. Equipment hourly repair/replacement and filter costs reflect manufacturer/dealer cost information and engineering estimates based on IMC's experience.

The annual operating costs for major mining equipment was estimated by multiplying the operating shifts derived for a particular piece of equipment (see Section 16.8) in a given year by the respective machine operating cost per shift. Operating hours for major production equipment (e.g. front end loaders, surface miners, 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 21-6 summaries the estimate of the operating cost per shift for the major mining equipment. Table 21-7 shows the development of these costs for the front-end loader, surface miner and haul trucks as these are the majority of the mine mobile equipment fleet.

Table 21-6: Major Mining Equipment – Operating Cost per Shift

Equipment	US$/shift
WIRTGEN 2500SM Surface Miner	2,514
Cat 994 Front End Loader (31 cubic metre)	3,045
Cat 777 Coal Haul Truck (91 tonne)	1,027
Cat D9/D10 Dozers	797
Cat 834 Wheel Dozer	622
Cat 16M Motor Graders	545
Cat770-W Water Truck (30,000 liters)	799
Cat 336 Aux. Loader	350
Cat CS-56 Compactor	246

Table 21-7: Equipment Operating Cost Detail

Category	WIRTGEN 2500SM	Cat994 Loader	Cat777 Coal Haul Truck
Machine Life (metered hours)	75,000	45,000	
Cost/metered hour:			
Fuel Cost ($/liter)	0.75	0.75	0.75
Fuel Consumption (liters/hr)	119	145	62
Fuel Cost	$89.44	$108.75	$46.50
Number of Tires	-	4	6
Cost per tire		$80,057	$15,667
Tire life (hours)		4,000	5,000
Tire Cost / hour		$80.06	$18.80
Lube-Oil-Filters-Grease	$22.36	$34.33	$13.92
Repairs	$98.00	$51,82	$14.15
Wear Items	$18.77	$1.84	-
Total Cost / hour	$228.57	$276.80	$93.37
Metered minutes per shift	660	660	660
Total Cost Per Shift	$2,514	$3,045	$1,027

21.1.1.5 Direct Mine Operating Costs – Summary

The LOM mine operating costs are summarized in Table 21-8 by the major areas of parts and consumables and labor. Table 21-8 shows the costs by the cost categories described in section 21.1.1. The costs for loading, hauling and auxiliary include the cost of the machines assigned to the category for fuel, tires, repair parts, wear items, the maintenance labor to maintain the machine and the operator. Table 21-8 shows the total cost for each year. The Year -1 (pre-production) is included in both tables but the costs will be capitalized. The direct mine operating costs do not include the contractor's cost for the compaction of the TSF embankments, but does include the haulage of the overburden waste from the pit to the embankment. The cost of final reclamation, other than the on-going dump management during the delivery of the waste to the storage locations is not included in the direct mine operating costs.

Table 21-8: Life of Mine Cost by Major Areas

Cost Area	$ x 1000	% of Total Cost
Parts and Consumables		
Diesel Fuel	292,200	33.85
Tires	122,564	14.20
Lubricants, Repairs, Wear Parts	208,741	24.18
Gen. Mine / Gen. Maint/ Pumping	48,791	5.65
Total Parts & Consumables	672,296	77.89
Labor		
Salaried Staff	32,290	3.74
Hourly Labor	158,525	18.37
Total Labor	190,815	22.11
Total Mining Cost	863,111	

Table 21-9: Summary of Mine Operating Costs

	Totals By Dry Bank Cubic Meters (x1000)				Totals By Dry Bank Tonnes					Mine Operating Cost - Total Dollars ($USx1000)							Totals By Dry Tonne		Totals By Bank Cubic Meter		Totals By Dry Product Tonne
Mining Year	Ore Dry (k bcm)	Interburden Dry (k bcm)	Overburden Dry (k bcm)	TOTAL Dry (k bcm)	Ore Dry (k tonne)	Interburden Dry (k tonne)	Overburden Dry (k tonne)	TOTAL Dry (k tonne)	Product Dry (tonne)	Loading ($x1000)	Hauling ($x1000)	Auxiliary ($x1000)	General Mine ($x1000)	General Maint. ($x1000)	Mine G&A ($x1000)	TOTAL ($x1000)	Cost Per ORE ($/kt)	Cost Per MINED ($/kt)	Cost Per ORE ($/bcm)	Cost Per MINED ($/bcm)	Dry PRODUCT ($/dry prod tne)
-1	298	5,632	21,642	27,572	364	4,280	19,694	24,338	137,189	8,566	24,893	2,446	1,714	1,318	2,239	41,175	113.158	1.692	138.052	1.493	300.135
1	1,332	13,605	13,648	28,586	1,625	10,340	12,420	24,385	623,152	8,912	25,685	3,073	1,744	1,358	2,317	43,089	26.509	1.767	32.342	1.507	63.886
2	2,529	13,057	11,915	27,501	3,085	9,924	10,843	23,851	1,022,848	8,983	26,867	3,714	1,712	1,334	2,360	44,970	14.578	1.885	17.785	1.635	42.933
3	2,649	16,124	8,914	27,687	3,231	12,255	8,112	23,598	1,123,985	8,970	23,430	4,357	1,717	1,328	2,310	42,113	13.033	1.785	15.901	1.521	37.310
4	2,410	16,075	8,871	27,356	2,940	12,217	8,072	23,230	958,362	8,806	20,329	4,366	1,707	1,305	2,255	38,769	13.185	1.669	16.086	1.417	40.254
5	2,411	12,621	12,058	27,090	2,941	9,592	10,973	23,506	1,007,201	8,841	23,289	4,360	1,699	1,310	2,308	41,806	14.214	1.779	17.341	1.543	41.508
6	2,163	10,657	14,291	27,111	2,639	8,099	13,005	23,743	951,043	8,835	26,072	4,357	1,700	1,323	2,360	44,647	16.918	1.880	20.639	1.647	45.740
7	2,447	13,876	11,892	28,214	2,985	10,546	10,822	24,352	1,020,179	9,139	23,081	4,360	1,733	1,343	2,305	41,961	14.058	1.723	17.151	1.487	41.131
8	2,640	16,507	8,451	27,598	3,221	12,545	7,690	23,457	1,214,810	8,932	21,631	4,364	1,715	1,319	2,280	40,241	12.493	1.716	15.242	1.458	32.998
9	2,297	13,622	10,663	26,582	2,802	10,353	9,703	22,858	849,309	8,627	21,127	4,364	1,684	1,285	2,269	39,356	14.046	1.722	17.136	1.481	46.339
10	2,237	11,583	12,422	26,241	2,729	8,803	11,304	22,835	1,029,868	8,576	23,246	4,361	1,674	1,284	2,308	41,449	15.190	1.815	18.532	1.580	40.192
11	2,502	15,931	8,802	27,235	3,052	12,108	8,010	23,170	1,097,207	8,800	23,468	4,358	1,704	1,314	2,310	41,954	13.746	1.811	16.770	1.540	38.237
12	2,507	10,647	14,728	27,882	3,058	8,092	13,402	24,553	1,112,908	9,166	25,430	4,356	1,723	1,343	2,349	44,367	14.506	1.807	17.698	1.591	39.776
13	2,487	15,278	9,987	27,752	3,035	11,611	9,088	23,734	1,007,461	8,973	24,908	4,367	1,719	1,342	2,345	43,655	14.386	1.839	17.551	1.573	43.332
14	2,333	12,822	11,599	26,754	2,847	9,744	10,555	23,146	1,067,876	8,716	27,080	4,365	1,689	1,321	2,383	45,554	16.002	1.968	19.522	1.703	42.392
15	2,643	17,571	7,602	27,816	3,224	13,354	6,918	23,496	1,090,389	8,959	23,096	4,361	1,721	1,332	2,305	41,774	12.958	1.778	15.808	1.502	38.311
16	2,479	9,835	17,073	29,387	3,024	7,475	15,537	26,035	1,188,762	9,624	28,326	4,356	1,768	1,404	2,405	47,882	15.834	1.839	19.318	1.629	38.632
17	2,495	14,621	11,447	28,563	3,043	11,112	10,417	24,572	975,929	9,226	26,980	4,353	1,743	1,369	2,376	46,047	15.130	1.874	18.459	1.612	47.183
18	1,981	11,606	10,488	24,075	2,417	8,820	9,544	20,782	968,935	7,866	18,854	4,366	1,609	1,197	2,225	36,117	14.943	1.738	18.231	1.500	36.743
19	2,606	18,319	26	20,951	3,179	13,922	24	17,125	1,137,438	6,819	16,868	4,365	1,515	1,091	1,716	32,374	10.184	1.890	12.425	1.545	28.404
20	2,730	14,758	0	17,488	3,331	11,216	-	14,547	1,110,796	5,868	10,322	4,375	1,352	915	975	23,808	7.148	1.637	8.720	1.361	21.390
TOTAL	48,174	284,748	226,519	559,441	58,772	216,408	206,133	481,313	20,695,647	181,204	484,982	87,742	35,342	27,135	46,701	863,108	14.686	1.793	17.917	1.543	39.72
									PERCENT	21.0%	56.2%	10.2%	4.1%	3.1%	5.4%	100.0%					

21.1.2 Indirect Mining Costs

Indirect mine operating costs are those costs incurred by the mining operation and not directly attributable to the production of phosphorite. Indirect costs usually include the following: property and liability insurance, permitting fees, bonding, engineering consulting fees, exploration drilling, legal and auditing fees, freight and postage fees, communications fees, government and environmental relations fees, lab sampling and quality control, employee-related training, industry dues, royalty costs, and other miscellaneous expenses. These costs are not included in the mine operational estimates.

21.1.3 Mine Capital Expenditure

IMC estimated the mine capital expenditures for mobile mine equipment required to achieve the mine production schedule. Table 21-9 is a summary of the mine capital and operating costs by year. The initial capital for Year -1 includes $91.06 million of equipment purchases and $41.18 million for the pre-production mining.

The unit cost for the equipment was provided by EMG Mining Consultants from its global pricing database as of 2016 quarter 1. EMG is a major equipment procurement company and has global pricing agreements with most mining equipment manufacturers. Table 21-10 shows the unit prices for the equipment included in the mine capital cost estimate. The costs include freight, assembly, training and tires for the mobile equipment.

The detail for the capital expenditures for the major and support equipment by year is shown on Table 21-11. The total capital by year includes 3% of the major equipment purchases allocation for shop tools, a 5% allocation for initial spare parts and a 5% contingency of the prices for all mining equipment.

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. This is necessary when equipment eventually becomes unserviceable and/or non-functional during the normal course of operations. Where possible, IMC used major equipment rebuilds to extend the effective lives of the loaders, haul trucks and, water truck. The front-end loaders were replaced at the end of the first round of useful life (Year 8), but re-built at the second round in Year 16. The dozers and graders were always replaced. Table 21-12 shows the major mining equipment units being purchased, replaced or re-built (rebuild cost assumed to be half of the purchase price). The upper table is the purchased or replaced equipment. After Year 4, the purchases are replacements with the exception of the two haul trucks in Year 16 which are additions to the fleet. In Year 18, there are 21 haul trucks being re-built, this will be spread over several years in the Feasibility Study. The dozer replacement schedule continues to Year 19 at the request of FOCUS in order to have sufficient dozer capacity to complete the reclamation at closure.

Table 21-10: Summary of Mine Capital and Operating Costs
($USx1000)

Year	Mine Equipment			Operating Costs			Total
	Initial Capital Cost	Sustaining Capital Cost	Total Mine Capital	Operating Consum. Cost	Operating Labor Cost	Total Operating Cost	Total Cost
-1	91,058		91,058	32,182	8,993	41,175	132,234
1		6,990	6,990	33,727	9,362	43,089	50,078
2		5,699	5,699	35,127	9,843	44,970	50,668
3		1,568	1,568	32,820	9,293	42,113	43,681
4		473	473	30,079	8,689	38,769	39,242
5		255	255	32,541	9,265	41,806	42,062
6		0	0	34,806	9,841	44,647	44,647
7		0	0	32,725	9,237	41,961	41,961
8		23,662	23,662	31,278	8,963	40,241	63,903
9		3,463	3,463	30,516	8,840	39,356	42,819
10		0	0	32,184	9,265	41,449	41,449
11		3,411	3,411	32,661	9,293	41,954	45,365
12		473	473	34,648	9,718	44,367	44,840
13		602	602	33,984	9,671	43,655	44,257
14		0	0	35,458	10,096	45,554	45,554
15		0	0	32,537	9,237	41,774	41,774
16		17,032	17,032	37,550	10,332	47,882	64,914
17		1,986	1,986	36,027	10,020	46,047	48,034
18		21,050	21,050	27,768	8,349	36,117	57,168
19		1,291	1,291	25,032	7,343	32,374	33,665
20		0	0	18,643	5,165	23,808	23,808
Total	91,058	87,956	179,014	672,293	190,815	863,108	1,042,121

Table 21-11: Mine Major Equipment

Mine Major Equipment	Total Unit Price $USx1000
WIRTGEN 2500SM Continuous Miner (2.5 M)	2,932.0
CAT 994 Front End Loader (31 CuM)	4,279.0
CAT 777 Coal Haul Truck (90 tn)	1,413.0
CAT D9/D10 Track Dozers	1,142.5
CAT D10T Track Dozer	1,276.0
CAT D9T Track Dozer	1,009.0
CAT 834 Wheel Dozer (450 HP)	1,028.0
CAT 16M Motor Graders (297 HP)	848.0
CAT 770-W Water Truck (30,000 Ltr)	698.0
CAT 336 Aux Loader (1 CuM)	307.0
CAT CS-56 Compactor (147 HP)	226.0
Mine Support Equipment	
Fuel Truck 5,000 gal	281.0
Lube Truck	352.0
Pickup Truck (4x4)	40.0
Light Plants	13.4
CAT IT62 - Integrated Tool Carrier	239.0
Grove TR600E Crane (50 ton) - Road Machinery	539.0
Man Van	48.0
Tractor & Lowboy (off-highway)	1,228.0
Haul Truck Retriever	1,200.0
Wenco Mine Communications Network	1,079.2
Welding Truck	206.0
Mechanics Truck	231.0
Mine Dispatch System	1,500.0
Spare Loader Bucket	245.4
Cat 988 with Tire Handler	897.0
RT Forklift (Sellick S160-4)	126.0
Water Pipe - Dewatering	75.0
Mine Pumps (Diesel Trailer Mounted)	189.0
Mine Planning Software	241.0
Shop Jacks	42.0

Table 21-12: Mine Capital by Year

	Unit Cost	Life	-1		1		2		3		4		5		6		7		8		9	
	($1000)	Hours	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)
MINE MAJOR EQUIPMENT:																						
WIRTGEN 2500SM Continuous Miner (2.5 M)	2,932	75,000	1	2,932	1	2,932	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 994 Front End Loader (31 CuM)	4,279	45,000	4	17,116	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4	17,116	0	0
CAT 777 Coal Haul Truck (90 tn)	1,413	100,000	34	48,042	1	1,413	2	2,826	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT D9/D10 Track Dozers	1,143	35,000	2	2,285	1	1,143	0	0	1	1,143	0	0	0	0	0	0	0	0	2	2,285	1	1,143
CAT 834 Wheel Dozer (450 HP)	1,028	35,000	1	1,028	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 16M Motor Graders (297 HP)	848	35,000	2	1,696	0	0	1	848	0	0	0	0	0	0	0	0	0	0	0	0	2	1,696
CAT 770-W Water Truck (30,000 Ltr)	698	75,000	1	698	1	698	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 336 Aux Loader (1 CuM)	307	45,000	1	307	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT CS-56 Compactor (147 HP)	226	25,000	1	226	0	0	0	0	0	0	0	0	1	226	0	0	0	0	0	0	1	226
Subtotal Major Equipment				74,330		6,186		3,674		1,143		0		226		0		0		19,401		3,065
MINE SUPPORT EQUIPMENT:		Years																				
Fuel Truck 5,000 gal	281	8	2	562		0		0		0		0		0		0		0	2	562		0
Lube Truck	352	8	1	352		0		0		0		0		0		0		0		0		0
Pickup Truck (4x4)	40	4	7	280		0		0		0	7	280		0		0		0	7	280		0
Light Plants (*)	13	8	2	27		0		0		0	2	27		0		0		0	2	27		0
CAT IT62 - Integrated Tool Carrier	239	18	1	239		0		0		0		0		0		0		0		0		0
Grove TR600E Crane (50 ton) - Road Machinery	539	18	1	539		0		0		0		0		0		0		0		0		0
Man Van	48	4	3	144		0		0		0	3	144		0		0		0	3	144		0
Tractor & Lowboy (off-highway) (*)	1,228	18		0		0	1	1,228		0		0		0		0		0		0		0
Wenco Mine Communications Network	1,079	18	1	1,079		0		0		0		0		0		0		0		0		0
Welding Truck	206	8	1	206		0		0		0		0		0		0		0	2	412		0
Mechanics Truck (*)	231	8	1	231		0		0		0		0		0		0		0	1	231		0
Mine Dispatch System	1,500	18	1	1,500		0		0		0		0		0		0		0		0		0
Spare Loader Bucket	245	8		0		0	1	245		0		0		0		0		0		0		0
Cat 988 with Tire Handler	897	18	1	897		0		0		0		0		0		0		0		0		0
RT Forklift (Sellick S160-4)	126	18	1	126		0		0		0		0		0		0		0		0		0
Water Pipe - Dewatering	75	18	1	75		0		0	1	75		0		0		0		0		0		0
Mine Pumps (Diesel Trailer Mounted)	189	18	1	189		0		0	1	189		0		0		0		0		0		0
Mine Planning Software	241	18	1	241		0		0		0		0		0		0		0		0		0
Shop Jacks	42	18	1	42		0		0		0		0		0		0		0		0		0
Subtotal Mine Support Equipment				6,729		0		1,473		264		451		0		0		0		1,656		0
Shop Tools (3% of Major Equipment)		3.0%		2,230		186		110		34		0		7		0		0		582		92
Initial Spare Parts (5% of Major Equipment @ New Purchase Price)		5.0%		3,717		309		184		57		0		11		0		0		970		153
Contingency (5% of All Mine Equipment)		5.0%		4,053		309		257		70		23		11		0		0		1,053		153
TOTAL EQUIPMENT/FACILITIES CAPITAL				91,058		6,990		5,699		1,568		473		255		0		0		23,662		3,463

	10		11		12		13		14		15		16		17		18		19		20		Project
	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	Total
MINE MAJOR EQUIPMENT:																			0				
WIRTGEN 2500SM Continuous Miner (2.5 M)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5,864
CAT 994 Front End Loader (31 CuM)	0	0	0	0	0	0	0	0	0	0	0	0	4	8,558	0	0	0	0	0	0	0	0	42,790
CAT 777 Coal Haul Truck (90 tn)	0	0	0	0	0	0	0	0	0	0	0	0	2	2,826	0	0	21	14,837	0	0	0	0	69,944
CAT D9/D10 Track Dozers	0	0	1	1,143	0	0	0	0	0	0	0	0	2	2,285	1	1,143	0	0	1	1,143	0	0	13,710
CAT 834 Wheel Dozer (450 HP)	0	0	1	1,028	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	2,056
CAT 16M Motor Graders (297 HP)	0	0	1	848	0	0	0	0	0	0	0	0	0	0	0	0	2	1,696	0	0	0	0	6,784
CAT 770-W Water Truck (30,000 Ltr)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	349	1	349	0	0	0	0	2,094
CAT 336 Aux Loader (1 CuM)	0	0	0	0	0	0	1	307	0	0	0	0	0	0	0	0	0	0	0	0	0	0	614
CAT CS-56 Compactor (147 HP)	0	0	0	0	0	0	1	226	0	0	0	0	0	0	1	226	0	0	0	0	0	0	1,130
Subtotal Major Equipment		0		3,019		0		533		0		0		13,669		1,718		16,882		1,143		0	144,986
MINE SUPPORT EQUIPMENT:																							
Fuel Truck 5,000 gal		0		0		0		0		0		0		0		0		0		0		0	1,124
Lube Truck		0		0		0		0		0		0		0		0		0		0		0	352
Pickup Truck (4x4)		0		0	7	280		0		0		0	7	280		0		0		0		0	1,400
Light Plants (*)		0		0	2	27		0		0		0	2	27		0		0		0		0	134
CAT IT62 - Integrated Tool Carrier		0		0		0		0		0		0		0		0		0		0		0	239
Grove TR600E Crane (50 ton) - Road Machinery		0		0		0		0		0		0		0		0		0		0		0	539
Man Van		0		0	3	144		0		0		0	3	144		0		0		0		0	720
Tractor & Lowboy (off-highway) (*)		0		0		0		0		0		0		0		0		0		0		0	1,228
Wenco Mine Communications Network		0		0		0		0		0		0		0		0		0		0		0	1,079
Welding Truck		0		0		0		0		0		0		0		0		0		0		0	618
Mechanics Truck (*)		0		0		0		0		0		0		0		0		0		0		0	462
Mine Dispatch System		0		0		0		0		0		0		0		0		0		0		0	1,500
Spare Loader Bucket		0		0		0		0		0		0		0		0		0		0		0	245
Cat 988 with Tire Handler		0		0		0		0		0		0		0		0		0		0		0	897
RT Forklift (Sellick S160-4)		0		0		0		0		0		0		0		0		0		0		0	126
Water Pipe - Dewatering		0		0		0		0		0		0		0		0		0		0		0	150
Mine Pumps (Diesel Trailer Mounted)		0		0		0		0		0		0		0		0		0		0		0	378
Mine Planning Software		0		0		0		0		0		0		0		0		0		0		0	241
Shop Jacks		0		0		0		0		0		0		0		0		0		0		0	42
																							-
Subtotal Mine Support Equipment		0		0		451		0		0		0		451		0		0		0		0	11,475
Shop Tools (3% of Major Equipment)		0		91		0		16		0		0		667		62		962		34		0	5,072
Initial Spare Parts (5% of Major Equipment @ New Purchase Price)		0		151		0		27		0		0		1,111		103		1,603		57		0	8,454
Contingency (5% of All Mine Equipment)		0		151		23		27		0		0		1,134		103		1,603		57		0	9,028
TOTAL EQUIPMENT/FACILITIES CAPITAL		0		3,411		473		602		0		0		17,032		1,986		21,050		1,291		0	179,014

Table 21-13: Mine Equipment Purchase, Replacement or Rebuild Schedule

Purchased Equipment	Unit Cost	Total	Time Period - Units Purchased																				
Equipment Type	($1000)	Purchased	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
WIRTGEN 2500SM Continuous Miner (2.5 M)	2,932	2	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 994 Front End Loader (31 CuM)	4,279	8	4	0	0	0	0	0	0	0	4	0	0	0	0	0	0	0	0	0	0	0	0
CAT 777 Coal Haul Truck (90 tn)	1,413	39	34	1	2	0	0	0	0	0	0	0	0	0	0	0	0	0	2	0	0	0	0
CAT D9/D10 Track Dozers	1,143	12	2	1	0	1	0	0	0	0	2	1	0	1	0	0	0	0	2	1	0	1	0
CAT 834 Wheel Dozer (450 HP)	1,028	2	1	0	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0
CAT 16M Motor Graders (297 HP)	848	8	2	0	1	0	0	0	0	0	0	2	0	1	0	0	0	0	0	0	2	0	0
CAT 770-W Water Truck (30,000 Ltr)	698	2	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 336 Aux Loader (1 CuM)	307	2	1	0	0	0	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0
CAT CS-56 Compactor (147 HP)	226	5	1	0	0	0	0	1	0	0	0	1	0	0	0	1	0	0	0	1	0	0	0

Rebuild Equipment	Unit Cost	Total	Time Period - Units Rebuilt																				
Equipment Type	($1000)	Rebuilt	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
WIRTGEN 2500SM Continuous Miner (2.5 M)	1,466	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 994 Front End Loader (31 CuM)	2,140	4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4	0	0	0	0
CAT 777 Coal Haul Truck (90 tn)	707	21	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	21	0	0
CAT D9/D10 Track Dozers	571	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 834 Wheel Dozer (450 HP)	514	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 16M Motor Graders (297 HP)	424	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT 770-W Water Truck (30,000 Ltr)	349	2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	1	0	0
CAT 336 Aux Loader (1 CuM)	154	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAT CS-56 Compactor (147 HP)	113	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

21.1.3.1 Non-Cash Costs – Depreciation and Final Reclamation Accrual

Non-cash costs include the depreciation charges expensed, in accordance with cost accounting practices, to compensate for the decline in value of capital items over time and these will be developed by M3 and Focus.

For final reclamation, the costs are cover the re-grading of the areas affected by the mining operation to a stable configuration. These costs were not developed by IMC and are the responsibility of M3.

21.2 PLANT CAPEX AND OPEX

21.2.1 Plant Capex

The initial capital cost includes the development of the mine, the engineering and construction of the process plant, the ancillary facilities, a 138 kV power transmission line from La Niña substation, the 45-km seawater supply line for process water, ponds, a small desalination facility, and the first phase of the Tailings Storage Facility (TSF).

The process plant has been designed to produce 1,000,000 tonnes per annum of combined 24% P_2O_5 DAPR and 28% P_2O_5 DAPRstarting in Year 1. The initial plant capital cost is estimated to be $95.6 million which includes 20% contingency but does not include mining capital, pre-stripping costs, TSF embankment costs, Owner's cost or IGV.

Initial CAPEX includes an estimate of contingency based on the accuracy and level of detail of the cost estimate. The purpose of the contingency provision is to make allowance for uncertain cost elements which are predicted to occur, but are not included in the cost estimate. These cost elements include uncertainties concerning completeness and accuracy of material takeoffs, accuracy of labor and material rates, accuracy of labor productivity expectations, and accuracy of equipment pricing.

Battery limits for the process plant starts at the ROM stockpile to TSF pipeline discharge, including power transmission to the site and power distribution at the site, the seawater supply pipeline, and pumping costs. M3's scope also includes ancillary buildings such as the security building, administration building, cafeteria, change house, truck shop, truck wash, plant maintenance building, warehouse, laboratory, medical building and fuel stations.

The primary assumptions used to develop the CAPEX are provided below:

- All cost estimates were developed and are reported in United States of America (US) dollars.
- Qualified and experienced construction contractors will be available at the time of project execution.
- Borrow sources are available within the Project boundary.
- Weather related delays in construction are not accounted for in the estimate.

21.2.1.1 Currency

The estimate is expressed in 1st quarter 2016 United States dollars. No provision has been included to offset future escalation. No funds have been allocated in the estimate to offset potential currency fluctuations. No provision has been made for currency fluctuations.

21.2.1.2 Estimate Exclusions

Items not included in the M3 capital estimate are as follows:

- Sunk costs;
- Allowance for special incentives (schedule, safety, etc.);
- Reclamation costs (included in financial analysis);
- Escalation beyond 1st Quarter 2016;

- Foreign currency exchange rate fluctuations;
- Interest and financing cost.

Risk due to political upheaval, government policy changes, labor disputes, permitting delays, weather delays or any other force majeure occurrences are also excluded.

21.2.1.3 Accuracy

The estimate has been developed to a level sufficient to assess/evaluate the project concept, various development options and the overall project viability. After inclusion of the recommended contingency, the capital cost estimate is considered to have a level of accuracy in the range of -20%/+20%.

21.2.1.4 Plant Equipment

All major plant equipment was investigated for this prefeasibility study. Flowsheets and Metsim mass balance reports in conjunction with a variety of testwork results, led to a process design criteria (PDC) for the Bayovar 12 Project. Pricing was solicited from qualified vendors for the following equipment:

- ROM feed bin and reclaim feeder
- Plant conveyors
- Drum washer
- Attrition cells
- Hydrocyclones and hydrosizer
- Vibrating screens
- Concentrate belt vacuum filter
- Concentrate rotary dryer, off-gas handling, and dust collection
- Bucket elevators for concentrate handling
- Slurry and seawater pumps
- Desalination reverse osmosis plant
- Pump barge
- Water intake system
- Main substation transformer and switchgear
- Field erected and shop fabricated tankage and silos
- Fuel storage for diesel and LNG

Other ancillary pieces of equipment at the site have been costed using historical equipment pricing of similar size and duty from recent M3 studies and EPCM projects.

- Dust collectors
- Air compressors, receivers, and air dryers
- Slurry samplers

Commodity pricing was solicited from several suppliers for the following:

- Structural steel
- Mechanical steel and platework
- Concrete

- HDPE pipe materials

21.2.1.5 Material Quantities

Discipline engineers developed material take-off quantities (MTO's) for earthworks, concrete, steel, architectural, long piping runs, and electrical based on general arrangement drawings, civil site plans and single line drawings developed for the Project. In-plant piping and instrumentation and controls were factored for the estimate according to historical factors.

21.2.1.6 Pricing Methodology

The capital estimate is built up by cost centers as defined by the project Work Breakdown Structure (WBS) and by prime commodity accounts, which include earthwork, concentrate, structural steel, mechanical equipment (including plate work), piping, electrical and instrumentation.

The estimate is based on the assumption that equipment and materials will be purchased on a competitive basis and installation contracts will be awarded in defined packages on either a time and materials basis or as lump sum contracts.

Below is a discussion of how the estimating methodologies have been applied within the commodity groups.

21.2.1.7 Labor Productivity

Installation hours are based on United States standard rates for the lower 48 states and have been adjusted with productivity factors for working in the Peruvian Coastal region at low altitude. The productivity factors were developed using historical data from similar projects in the region, as well as comparing man-hours provided by local contractors with the U.S. standards.

Overall, the labor man-hours reflect a 2.0 times decrease in productivity from U.S. standards to account for longer workday/workweek, general workforce skill level, the extent of manual production and the climate at the site.

21.2.1.8 Labor Rates

Labor rates were provided for various trades and skill levels using information from recent historical projects in Peru. Construction rates were examined by Humberto Montes, an in-country project consultant to FCV and then modified to include overtime, supervision, and other overhead considerations.

The wage rates used reflect a 50-hour work week with construction 7 days off. Labor rates do not cover contractor field indirect costs including: mobilization and demobilization, temporary facilities, temporary utilities, testing services, and construction equipment. These items are included with the construction indirect cost.

Average construction crew rates have been developed for each commodity type from the labor information by blending appropriate labor and skill levels to derive reasonable crew mixes.

21.2.1.9 Buildings

The structural components (civil, concrete & steel) for the process ancillary buildings have been based on material take-offs (MTOs). Architectural finishes, plumbing, and electrical additions were factored on a square meter basis.

Process plant building costs are included in the overall cost of the plant area in which they fall. Table 21-14 lists the capital costs of onsite ancillary facilities.

Table 21-14: Onsite Ancillary Facilities CAPEX

Onsite Auxiliary Facilities	Direct Cost Plant ($000s)
Ancillaries – General	271
Security Building	130
Administration Building	960
Cafeteria	479
Truck Scale	509
Assay Lab	1,012
Warehouse	616
Truck Shop	4,708
Truck Wash	926
Change House	460
Plant Maintenance Building	1,423
Medical/Emergency Building	324
Fuel Station + Fuel Depot	401
Direct Onsite Auxiliary Facilities	$12,220

21.2.1.10 Power Transmission Capital Costs

M3 developed a cost for a 15 km 138 kV transmission line and its interconnection from the La Niña substation to the Bayovar 12 substation. Components for the power transmission system and substation were itemized and costed by M3. The cost estimate for the power line and substation is summarized in Table 21-15 below. The detailed breakdown is presented in the complete Bayovar 12 Capex.

Table 21-15: Power Transmission Line & Main Substation CAPEX

Item	Direct Cost ($000s)
15 km Overhead 220 kV power line including 50 Transmission towers, cable, hardware & switching equipment	2,957
Bayovar 12 Plant Substation including stepdown transformer, equipment, and cable	2,202
Total Cost	$5,158

21.2.1.11 Seawater Supply System

The seawater supply pipeline and storage includes an intake station, an overland HDPE pipeline, seawater ponds at the plant site, and seawater barge pumps and barges. The single largest capital impact to plant development is the installation of the seawater pipeline at nearly $17.1 million direct cost. It is also the longest duration schedule activity for the project and will require more detailed analysis during feasibility studies.

Table 21-16 lists the cost items in the seawater supply system for the Bayovar 12 plant.

Table 21-16: Seawater Supply Cost

Item	Direct Cost ($000s)
Seawater Intake	335
Seawater Supply Pumps	406
Seawater Pipeline – 32"	14,301
Seawater Ponds & other civil	742
Seawater Pond Pumps & Barges	401
Firewater pumps & Firewater Piping	474
Electrical & Instrumentation	421
Other	22
Total Cost	$17,102

21.2.1.12 Indirect Costs

Indirect costs are those costs that can generally not be tied to a specific work area, as summarized in Table 21-17. This category includes "other direct costs" that are related to construction that can't be assigned directly to a work area including the following:

- Quality assurance testing is included at 2% of total direct costs for civil, concrete, piping, steel, and electrical costs;
- survey is included at 1% of total direct costs for civil, concrete, and steel costs;
- mobilization of contractors is 0.5% of total direct cost without mine & mobile equipment and including quality assurance;
- pipe spooling detail is included at 3% of piping materials; and
- Programming included at 0.2% of direct costs.

Table 21-17: Indirect Capital Cost Summary

Indirect Cost Items	Plant Indirects ($000s)
Quality Assurance Testing	707
Surveying	88
Pipe Spooling	442
Programming	124
Mobilization	708
Freight + Customs + Export Packing	4,604
EPCM Costs	9,207
Vendor Erection Supervision, Start-up, and Commissioning	666
Capital and Commissioning Spares, First Fills	1,467
Other Indirect Costs	1,009
Total Indirect Costs	$19,022

21.2.1.13 EPCM Costs

EPCM cost estimates break down into various categories that total approximately 15.1% of direct constructed field cost excluding mining pre-strip and mine equipment costs, as shown in Table 21-18.

Table 21-18: EPCM Capital Cost Summary

EPCM Components	Percentage of Total Direct Field Cost	Cost ($000s)
Project Services	1.0%	605
Project Control	0.75%	454
Management & Accounting	0.75%	454
EPCM Fee Fixed	1.5% of EPCM cost	136
Engineering	6.0%	3,931
Construction Management	6.5%	3,629
EPCM Total	15.1%	$9,209

Table 21-18 summarizes capital cost estimate for the plant and infrastructure for the Bayovar 12 plant. This capital cost summary excludes owner and contractor mining, pre-stripping, tailings development and Owner's Cost. Indirect costs are 31% of direct costs for the plant. A contingency of 20% of the direct and indirect construction cost has been added to the Capex total.

Table 21-19: Initial Capital Cost Summary for Process Plant

Area	Plant Cost ($000s)
General Site Costs	2,548
ROM Dump Pocket and Feed Conveyor	2,044
Drum Washing and Desliming	4,394
Attrition Scrubbing	5,159
Concentrate Filtration	2054
Concentrate Drying and Loadout	6,978
Tailings Line	2,182
Seawater Supply	17,102
Desalination RO Plant & Firewater Supply	966
Power Transmission Line and Main Substation	5,158
Ancillaries	12,220
Direct Cost	$60,815
Contractor Indirects	3,039
EPCM Services	9,209
Commissioning and Vendor Reps	666
Capital & Commissioning Spare Parts & Initial Fills	1,467
Freight, Duties	4,604
Indirect Cost	$19,024
Contingency (Process Plant) at 20%	15,728
Total	$95,567

Process Plant Operating Cost Summary

This section addresses the following costs:

- Process Plant Operating & Maintenance Cost

The process plant operating costs are summarized by area and then by cost element of labor, electric power, reagents, maintenance parts and supplies and services. Below in Table 21-19 is a summary of the average annual operating cost by Area for the Life-of-Mine (LOM) at full production of 1 million tonnes of DAPR concentrate.

Table 21-20: LOM Process Operating Cost Summary

	LOM Average Process Plant Production	
Product – 24% DAPR	518,300	
Product – 28% DAPR	516,500	
	Annual Cost	Unit Cost per Tonne
ROM Stockpile	$314,883	$0.30
Washing & Scrubbing	$1,796,027	$1.74
RO Plant	$139,060	$0.13
Concentrate, Tailings	$4,983,381	$4.82
Ancillary	$1,053,984	$1.02
Total	$8,287,335	$8.01

21.2.1.14 Process Labor and Fringes

The process plant operating and maintenance labor costs were derived from a staffing plan and are based on labor rates from an industry survey for this region and modified where necessary. The annual salaries include overtime and benefits for both salaried and hourly employees. The benefit rate used is 40%. A summary of the labor annual cost is shown in Table 21-21.Table 21-21: Process Labor Summary

Table 21-21: Process Labor Summary

	Personnel	Annual Cost
Operations	43	$994,560
Maintenance	36	$591,360
Total	155	$1,585,920

21.2.1.15 Maintenance Parts

An allowance was made to cover the cost of maintenance for the plant facilities and all items not specifically identified. The allowance made as a percent of the direct capital cost of equipment for each area; the rate used was 5%. For ancillary areas, a maintenance rate of 1% of direct capital cost was used. The annual cost is estimated to be $1.4 million.

21.2.1.16 Electrical Power

Electrical power costs were based on current pricing at a rate of $0.07 per kWh. The electric power consumption was based on the equipment list connected kW, discounted for operating time and the anticipated operating load level. The estimated annual power cost is $2.4 million on approximately 33.8 million kWh.

21.2.1.17 Process Supplies and Services

An annual allowance was estimated for items such as lubricants, diesel fuel, safety items and tools. The allowances were estimated from historical information or from other operations and projects. The annual cost is estimated to be $3.5 million from the consumption of truck delivered natural gas for the concentrate dryer at a rate of $8 per 1000 cubic feet.

21.2.1.18 G&A

The General and Administrative area includes Administration, Controllers, HR, Purchasing/warehousing, Safety, Security, Environmental, Community Relations. The total staff is 35 employees. The typical year is estimated to have operating expenses of $2.5 million. Presented below in Table 21-21 is a typical year of G&A operating expenses:

Table 21-22: General and Administration Operating Expenses

Cost Item	Annual Cost
Labor & Fringes	$1,158,000
Property & Business Interruption Insurance	$750,000
Offices Expenses	$30,000
Communications	$60,000
Community Relations	$150,000
Contractors & Consultants	$200,000
Employee Related	$120,000
Total General and Administration	$2,468,000

21.2.1.19 Tailings Storage Facility Capex

The TSF embankment will be constructed with ROM waste material from the open pit excavation and overburden waste will be hauled and dumped at the TSF as part of the mine operating costs. Thus the only costs attributed to the tailings TSF will be the spreading, moisture conditioning and compaction of the fill. Other costs for the TSF construction will be the protection of the downstream slope of the TSF to prevent erosion during flood events and the cost of constructing spillways.

The capital cost of the 20-year mine life TSF layout as shown on Figure 16-1 are provided in Section 16. To reduce the initial capital cost, the TSF will be constructed in two stages. Stage 1 has a surface area of 4.4 Mm^2 and accommodates the tailings deposition rate during Years 1 through 5. Stage 2 has a surface area of 4.4 Mm^2 providing a total surface area of 8.8 Mm^2 and accommodate tailings deposition, storm water and pit seepage through Year 20. The existing topographic base survey has a contour interval of 5 meters and a more detailed topographic survey will be required when the earthwork quantity estimates are updated at feasibility level.

Earthwork costs were developed based on a budget cost estimate provided by Stracom GyM (a Peruvian contractor) dated October 2015. The unit rates of $0.27/m2 for foundation preparation and $2.77/m3 for the embankment construction were applied to the two stages of the embankment construction. The total cost included a contractor indirect cost of 18.70% and allocations have been added for camp costs plus mobilization and de-mobilization of the contractor's equipment. An erosion control geotextile will be placed on the downstream out slope of the tailings TSF. Costs for the erosion control, spillways and monitor wells have not been included in the updated PFS but will be addressed in the Feasibility Study.

Table 21-23: TSF Capital Cost Estimate

Item	Description	Year	Quantity	Unit	Unit Cost	Cost ($000s)
Stage 1	Embankment Fill	-1	2,615,000	M3	$3.92	$10,255
	Over-Haulage	-1				Included in Mine Op Costs
	Pit Protection Berm	-1	923,000			Included in Mine Op Costs
	Total Stage 1 Cost					$10,255
Stage 2	Embankment Fill	5-6	3,354,000	M3	$3.80	$12,760
	Over-Haulage	5-6				Included in Mine Op Costs
	Total Stage 2 Cost					$12,760
Total TSF Cost						$23,015

21.3 OWNER'S COST

The current capex includes an estimate for Owners Costs. These costs include estimates for Owners staffing during preproduction, site communications, Owners living expenses, administrative and construction offices, operator training, Owner's commissioning, construction insurance, environmental compliance, community development, consultants, and legal expenses. Table 21-22 lists the categories of Owner's costs anticipated during capital construction of the mine and plant. There are no Owner's costs incurred as sustaining capital since the plant will already in operation and all operating costs will then be expensed.

Table 21-24: Owner's Cost Estimate

Item	Sub Section	Unit of Measure	Units	Unit Cost ($)	Total ($)
Staff Build-up	G&A	# personnel	4	80,000	320,000
Owner's Team Camp Costs	Food & Housing	man-months	60	750	45,000
Temporary Sanitation	Portable toilets (daily)	each	10	250	2,500
Offices	Temp on site (months)	months	12	1,500	18,000
Admin Equipment, Office Furniture,		lot	1	50,000	50,000
Light vehicles and equipment including	Light Vehicles	each	4	2,500	10,000
Medical, Security & Safety	Medical Station Supplies		1	25,000	25,000
	Safety Supplies	# staff	250	150	37,500
Job Specific Training	New Hires for operations	# staff	250	1,000	250,000
Owner Commissioning Team		# personnel	3	50,000	150,000
Insurance		lot	1	750,000	750,000
Environmental	Permitting Indirects	lot	1	500,000	500,000
Community Development		lot	1	150,000	150,000
Addition Consultants		lot	1	100,000	100,000
Legal, Permits & Fees		lot	1	100,000	100,000
Total					2,508,000

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine (LOM) based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of phosphate ore. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

22.1.1 Mine Production Statistics

Mine production is reported as ore and overburden from both the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report.

The Life-of-Mine ore quantities and ore grades are presented Table 22-1.

Table 22-1: Life of Mine Ore, Overburden Quantities, and Ore Grade

	Dry k Tonnes	Grade %
Phosphate Ore	58,772	12.9%
Overburden	206,133	
Interburden	216,408	
Total Waste Mined	422,541	

22.1.2 Plant Production Statistics

The process plant has two processing lines capable of producing 1,370 tonnes per day each of concentrate at 85% availability. The product is categorized into two grade 24% and 28%. Presented below is the life of mine production for both products.

Table 22-2: Life of Mine Production

	Dry k Tonnes
Phosphate concentrate - 24% DAPR	10,366
Phosphate concentrate - 28% DAPR	10,329
Total Phosphate Concentrate	20,696

22.1.2.1 Marketing Terms

The phosphate production is assumed to be shipped to end consumer and the terms are negotiable at the time of the agreement. The financial analysis presented here does not consider any deductions or penalties are being accessed. The product is priced to be shipped FOB, Port of Bayovar, Peru.

22.1.3 Capital Expenditure

22.1.3.1 Initial Capital

The financial indicators have been determined with equity financing for the initial capital. Any acquisition cost or expenditures prior to start of the full project period have been treated as "sunk" cost and have not been included in the analysis.

The total initial capital carried in the financial model for new construction and pre-production mine development is expended over a 2-year period. The initial capital includes Owner's costs and contingency. The cash flow will be expended in the years before production.

The initial capital is presented in Table 22-3.

Table 22-3: Initial Capital

	$ in millions
Mining (includes preproduction)	$59.4
Process Plant	$95.6
Owner's Cost	$2.5
TSF	$10.3
Total	$167.7

22.1.3.2 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. Included in the sustaining is capital for the expansion of the process plant to add Process Line #2 for $40.2 million. Mining sustaining capital includes the replacement of the mine fleet and overburden stripping, totaling $100.4 million. The total LOM sustaining capital is estimated to be $140.6 million. This capital will be expended during an 18-year period.

Table 22-4: Sustaining Capital

	$ in millions
Mining	$160.8
Process Plant	$20.0
TSF	$12.8
Total	$193.6

22.1.3.3 Working Capital

A 45-day delay of receipt of revenue from sales is used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model. In addition, working capital allowance of $2.5 million for plant consumable inventory is estimated in Year -1 and $2.5 million in Year 2. Also included is the IGV payments and refunds which is refunded on a 90-day cycle. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.1.3.4 Salvage Value

An allowance for salvage value has been included in the cash flow analysis of approximately $10.0 million. Most of the salvage value is tied to salvaging the seawater and tailings pipelines for re-use of HDPE, for resale of ancillary buildings, and for steel salvage. The process equipment itself will have limited value after 20 years of continuous usage.

22.1.3.5 Revenue

Annual revenue is determined by applying phosphate prices to the annual product by grade for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. Prices used in the evaluation are as follows:

- Phosphate Ore - 24% DAPR $145.00/tonne
- Phosphate Ore – 28% RPF $185.00/tonne

22.1.4 Total Production Cost

The life of mine Production Cost is estimated to be $69.83 per tonne of product being sold, excluding the cost of the capitalized pre-stripping. The Production Cost includes mine operations, process plant operations, general administrative cost, corporate overhead, shipping charges, royalties, and closure/reclamation and salvage value. Table 22-5 shows the estimated production cost by area per metric ton of product sold.

Table 22-5: Life of Mine Operating Cost

Operating Cost	US$/tonne product sold	LOM Cost ($millions)
Mining	$39.72	$821.9
Process Plant	$8.01	$165.7
General & Administration	$2.38	$49.4
Transportation	$10.09	$208.8
Total	$60.20	$1,245.8
Royalty	$9.07	$187.8
Interest	$0.98	$20.3
Reclamation/Closure	$0.06	$1.3
Salvage Value	($0.48)	($10.0)
Total Production Cost	$69.83	$1,445.2

22.1.4.1 Royalty and Export Duties

The royalty basis is 3.5% of gross revenues to the Peruvian government plus 2% to Radius Gold. A royalty is estimated at $187.8 million for the life of the mine.

22.1.4.2 Reclamation and Closure

Much of reclamation is going to be concurrent with mining from the backfilling of the open pit with waste. An allowance for reclamation and closure was included in the cash flow of $1.26 million for the life of the mine to cover monitoring.

Reclamation will be limited to removal of structures that will be offset by sale of equipment and structures from the plant.

22.1.4.3 Salvage Value

At end of the mine life an estimated salvage value was shown of $10.0 million has been included mainly from salvaging the large diameter HDPE pipe used for the seawater supply line, and the tailings line.

22.1.5 Taxation

22.1.5.1 Depreciation

Ten year straight line method for depreciation has be used for both initial and sustaining capital.

22.1.5.2 Income Tax

A corporate income tax rate of 26% was included in the economic model. This is applied to net profits of the company. Income taxes paid are estimated to be $418.1 million.

22.1.5.3 Value Added Tax

Value added tax (IGV) is levied on the supply of goods and services subject to the tax. The financial model applies an 18% rate and IGV tax is also reimbursed and it is assumed that the IGV paid and IGV recovered are the same; it is shown in the working capital section.

22.1.6 Project Financing

The project financing in the financial model is based on a combination of equity and debt financing.

22.1.7 Net Income after Tax

Net Income after Tax amounts to $1,189.5 million.

22.1.8 NPV and IRR

The NPV calculation includes Years 1 through 20 and adds the pre-production capital in Years -2 and -1. The economic analysis indicates that the project has an after-tax Internal Rate of Return (IRR) of 26.3% with a payback period of 3.9 years and a Net Present Value at 7.5% of $ 457.7million.

The sensitivity analysis table below compares the base case when the commodity prices, initial capital and operating cost are varied from the base case. The project is most sensitive to variation to the metal prices; while the initial capital and operating costs are similar.

Table 22-6: Sensitivity Analysis

Commodity Price Sensitivity after Taxes (costs in $000's)			
	NPV @7.5	IRR	Payback
20%	$693,384	34.7%	2.9
10%	$575,562	30.6%	3.3
Base Case	$457,741	26.3%	3.9
-10%	$339,919	21.9%	5.0
-20%	$222,098	17.2%	6.2
Operating Cost Sensitivity after Taxes			
	NPV @7.5	IRR	Payback
20%	$362,258	22.5%	4.8
10%	$409,999	24.4%	4.3
Base Case	$457,741	26.3%	3.9
-10%	$505,482	28.2%	3.6
-20%	$553,223	30.1%	3.3
Initial Capital Sensitivity after Taxes			
	NPV @7.5	IRR	Payback
20%	$432,749	23.5%	4.4
10%	$445,245	24.8%	4.2
Base Case	$457,741	26.3%	3.9
-10%	$470,237	28.0%	3.7
-20%	$482,733	30.0%	3.4

22.1.9 Financial Model Tabulation

Table 22-7 shows a tabulation of the base case financial model.

Table 22-7: Base Case Financial Model (US$ in Thousands)

	Total	-3	-2	-1	1	2	3	4	5	6	7
Mining Operations											
Phosphate Ore											
Beginning Inventory (kt)	58,772	58,772	58,772	58,772	58,408	56,783	53,698	50,467	47,526	44,585	41,946
Mined (kt)	58,772	-	-	364	1,625	3,085	3,231	2,940	2,941	2,639	2,985
Ending Inventory (kt)	-	58,772	58,772	58,408	56,783	53,698	50,467	47,526	44,585	41,946	38,961
Phosphate Grade (%)	12.9%	0.0%	0.0%	10.9%	12.1%	12.6%	12.4%	12.6%	12.8%	13.1%	12.8%
Contained Phosphate (kt)	7,601	-	-	40	197	390	401	370	375	345	383
ROM OB Volume (kbcm)	226,519	-	-	21,642	13,648	11,915	8,914	8,871	14,291	12,058	11,892
ROM IB Volume (kbcm)	284,748	-	-	5,632	13,605	13,057	16,124	16,075	12,621	10,657	13,876
Waste (kbcm)	511,267		-	27,274	27,253	24,973	25,038	24,946	26,912	22,715	25,768
Total Material Mined (kt)	570,039	-	0	27,638	28,879	28,057	28,270	27,886	29,853	25,354	28,753
					16.77	8.10	7.75	8.48	9.15	8.61	8.63
					40.4	23.8	22.2	25.9	24.5	25.6	25.3
Process Plant											
Phosphate Ore - 24% RPR											
Product (kt)	10,366			-	561	415	594	413	503	426	500
Product Grade (%)	24.02%			23.9%	22.9%	24.1%	23.7%	24.8%	24.4%	23.3%	24.2%
Recovered Phosphate (kt)	2,490				128	100	141	102	123	99	121
Payable Product (kt)	10,366	-	-	-	561	415	594	413	503	426	500
Phosphate Ore - 28% RPR											
Beginning Inventory (kt)	10,329	10,329	10,329	10,329	10,329	10,130	9,522	8,992	8,447	7,943	7,418
Mined (kt)	10,329	-	-	-	200	608	530	545	504	525	520
Ending Inventory (kt)	-	10,329	10,329	10,329	10,130	9,522	8,992	8,447	7,943	7,418	6,898
Phosphate Grade (%)		0.0%	0.0%	0.0%	28.9%	28.5%	28.4%	28.6%	28.4%	28.6%	28.3%
Contained Phosphate (kt)	2,939	-	-	-	58	173	150	156	143	150	147
Product (kt)	10,329				200	608	530	545	504	525	520
Product Grade (%)	28.46%				28.9%	28.5%	28.4%	28.6%	28.4%	28.6%	28.3%
Recovered Phosphate (kt)	2,939				58	173	150	156	143	150	147
Payable Product (kt)	10,329	-	-	-	200	608	530	545	504	525	520
Total Product 24%+ 28%	20,696		-	-	760	1,023	1,124	958	1,007	951	1,020
Income Statement ($000)											
Metal Prices											
Phosphate Price ($t)											
24%RPR	$145.00	$0.00	$0.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00
28%RPR	$185.00	$0.00	$0.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00
Revenues											
24%RPR	$1,503,098	$0	$0	$0	$81,290	$60,147	$86,195	$59,894	$72,973	$61,744	$72,507
28%RPR	$1,910,949	$0	$0	$0	$36,949	$112,488	$97,965	$100,881	$93,228	$97,166	$96,225
Total Revenues	$3,414,047	$0	$0	$0	$118,238	$172,635	$184,159	$160,775	$166,202	$158,910	$168,731
Operating Cost											
Total Mine Costs	$821,932				$43,089	$44,970	$42,113	$38,769	$41,806	$44,647	$41,961
Process Plant	$165,747	$0	$0	$0	$6,539	$8,209	$8,863	$7,791	$8,107	$7,744	$8,191
G&A	$49,350	$0	$0	$0	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468
Transportation	$208,819	$0	$0	$0	$7,672	$10,321	$11,341	$9,670	$10,163	$9,596	$10,294
Total Operating Cost	$1,245,848	$0	$0	$0	$59,767	$65,966	$64,785	$58,697	$62,544	$64,454	$62,914
Mining Cost per tonne	$39.72				$56.67	$43.97	$37.47	$40.45	$41.51	$46.95	$41.13
Process Plant per tonne	$8.01				$8.60	$8.03	$7.89	$8.13	$8.05	$8.14	$8.03
G&A per tonne	$2.38				$3.25	$2.41	$2.20	$2.57	$2.45	$2.59	$2.42
Transportation per tonne	$10.09				$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09
Op Cost per product tonne	**$60.20**				$78.61	$64.49	$57.64	$61.25	$62.10	$67.77	$61.67
Royalty	$187,773	$0	$0	$0	$6,503	$9,495	$10,129	$8,843	$9,141	$8,740	$9,280
Salvage Value	($10,000)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation/Closure	$1,264	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Interest Payments	$20,286				$3,642	$3,263	$2,748	$1,990	$1,135	$225	$97
Total Production Cost	$1,445,171	$0	$0	$0	$69,913	$78,724	$77,662	$69,529	$72,820	$73,419	$72,291
Operating Income	$1,968,876	$0	$0	$0	$48,326	$93,911	$106,498	$91,245	$93,382	$85,491	$96,440
Depreciation											
Initial Capital	$240,563				$24,056	$24,056	$24,056	$24,056	$24,056	$24,056	$24,056
Sustaining Capital	$120,715				$799	$1,469	$1,726	$1,873	$2,645	$3,375	$3,475
Total Depreciation	$361,279	$0	$0	$0	$24,855	$25,525	$25,782	$25,929	$26,702	$27,431	$27,531
Net Income after Depreciation	$1,607,597	$0	$0	$0	$23,470	$68,386	$80,716	$65,316	$66,680	$58,061	$68,909
Income Taxes	$418,140	$0	$0	$0	$6,102	$17,780	$20,986	$16,982	$17,337	$15,096	$17,916
Net Income	$1,189,458	$0	$0	$0	$17,368	$50,605	$59,730	$48,334	$49,343	$42,965	$50,993

	8	9	10	11	12	13	14	15	16	17	18	19	20
Mining Operations													
Phosphate Ore													
Beginning Inventory (kt)	38,961	35,740	32,938	30,210	27,158	24,099	21,065	18,218	14,994	11,970	8,927	6,510	3,331
Mined (kt)	3,221	2,802	2,729	3,052	3,058	3,035	2,847	3,224	3,024	3,043	2,417	3,179	3,331
Ending Inventory (kt)	35,740	32,938	30,210	27,158	24,099	21,065	18,218	14,994	11,970	8,927	6,510	3,331	(0)
Phosphate Grade (%)	13.1%	12.4%	12.9%	13.1%	12.9%	12.8%	13.1%	13.0%	13.8%	13.0%	13.4%	13.0%	13.6%
Contained Phosphate (kt)	423	348	353	400	394	389	372	418	416	396	324	413	454
ROM OB Volume (kbcm)	8,451	10,663	12,422	8,802	14,728	9,987	11,599	7,602	17,073	11,447	10,488	26	-
ROM IB Volume (kbcm)	16,507	13,622	11,583	15,931	10,647	15,278	12,822	17,571	9,835	14,621	11,606	18,319	14,758
Waste (kbcm)	24,958	24,285	24,005	24,734	25,375	25,264	24,420	25,173	26,909	26,068	22,094	18,345	14,758
Total Material Mined (kt)	28,179	27,087	26,733	27,786	28,434	28,299	27,267	28,397	29,932	29,111	24,511	21,524	18,089
	7.75	8.67	8.80	8.10	8.30	8.33	8.58	7.81	8.90	8.57	9.14	5.77	4.43
	20.5	28.6	23.3	22.5	22.7	25.1	22.7	23.1	21.7	26.7	22.5	16.1	13.3
Process Plant													
Phosphate Ore - 24% RPR													
Product (kt)	682	346	610	606	543	499	574	595	544	473	546	628	309
Product Grade (%)	23.8%	25.3%	23.9%	24.0%	23.9%	24.7%	23.5%	25.0%	22.8%	25.8%	22.9%	24.0%	24.4%
Recovered Phosphate (kt)	162	87	146	146	130	123	135	149	124	122	125	151	75
Payable Product (kt)	682	346	610	606	543	499	574	595	544	473	546	628	309
Phosphate Ore - 28% RPR													
Beginning Inventory (kt)	6,898	6,364	5,861	5,441	4,949	4,379	3,871	3,378	2,882	2,237	1,734	1,312	802
Mined (kt)	533	503	420	492	570	508	494	496	645	503	422	510	802
Ending Inventory (kt)	6,364	5,861	5,441	4,949	4,379	3,871	3,378	2,882	2,237	1,734	1,312	802	-
Phosphate Grade (%)	28.6%	28.1%	29.0%	28.1%	29.1%	28.0%	28.8%	28.0%	28.8%	28.0%	28.8%	28.2%	28.3%
Contained Phosphate (kt)	152	142	122	138	166	142	142	139	185	141	122	144	227
Product (kt)	533	503	420	492	570	508	494	496	645	503	422	510	802
Product Grade (%)	28.6%	28.1%	29.0%	28.1%	29.1%	28.0%	28.8%	28.0%	28.8%	28.0%	28.8%	28.2%	28.3%
Recovered Phosphate (kt)	152	142	122	138	166	142	142	139	185	141	122	144	227
Payable Product (kt)	533	503	420	492	570	508	494	496	645	503	422	510	802
Total Product 24%+ 28%	1,215	849	1,030	1,097	1,113	1,007	1,068	1,090	1,189	976	969	1,137	1,111
Income Statement ($000)													
Metal Prices													
Phosphate Price ($t)													
24%RPR	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00	$145.00
28%RPR	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00	$185.00
Revenues													
24%RPR	$98,820	$50,156	$88,406	$87,825	$78,746	$72,423	$83,246	$86,242	$78,833	$68,612	$79,242	$91,028	$44,770
28%RPR	$98,659	$93,130	$77,731	$90,931	$105,419	$93,979	$91,347	$91,689	$119,341	$93,007	$78,151	$94,287	$148,377
Total Revenues	$197,479	$143,286	$166,138	$178,756	$184,165	$166,402	$174,593	$177,931	$198,174	$161,619	$157,393	$185,315	$193,147
Operating Cost													
Total Mine Costs	$40,241	$39,356	$41,449	$41,954	$44,367	$43,655	$45,554	$41,774	$47,882	$46,047	$36,117	$32,374	$23,808
Process Plant	$9,451	$7,085	$8,254	$8,690	$8,792	$8,109	$8,500	$8,646	$9,283	$7,905	$7,859	$8,950	$8,778
G&A	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468	$2,468
Transportation	$12,257	$8,570	$10,391	$11,071	$11,229	$10,165	$10,775	$11,002	$11,995	$9,847	$9,777	$11,477	$11,208
Total Operating Cost	$64,417	$57,478	$62,562	$64,182	$66,855	$64,397	$67,297	$63,889	$71,627	$66,267	$56,221	$55,269	$46,262
Mining Cost per tonne	$33.13	$46.34	$40.25	$38.24	$39.87	$43.33	$42.66	$38.31	$40.28	$47.18	$37.28	$28.46	$21.43
Process Plant per tonne	$7.78	$8.34	$8.01	$7.92	$7.90	$8.05	$7.96	$7.93	$7.81	$8.10	$8.11	$7.87	$7.90
G&A per tonne	$2.03	$2.91	$2.40	$2.25	$2.22	$2.45	$2.31	$2.26	$2.08	$2.53	$2.55	$2.17	$2.22
Transportation per tonne	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09	$10.09
Op Cost per product tonne	$53.03	$67.68	$60.75	$58.50	$60.07	$63.92	$63.02	$58.59	$60.25	$67.90	$58.02	$48.59	$41.65
Royalty	$10,861	$7,881	$9,138	$9,832	$10,129	$9,152	$9,603	$9,786	$10,900	$8,889	$8,657	$10,192	$10,623
Salvage Value	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation/Closure	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Interest Payments	$28	$955	$916	$709	$630	$398	$156	$90	$53	$696	$608	$1,253	$694
Total Production Cost	$75,307	$66,314	$72,615	$74,722	$77,614	$73,947	$77,055	$73,765	$82,580	$75,852	$65,486	$66,715	$57,578
Operating Income	$122,172	$76,972	$93,522	$104,033	$106,551	$92,454	$97,538	$104,166	$115,594	$85,768	$91,908	$118,600	$135,569
Depreciation													
Initial Capital	$24,056	$24,056	$24,056	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital	$5,941	$6,387	$6,487	$6,129	$5,607	$5,510	$5,463	$4,790	$5,864	$6,063	$5,802	$5,585	$35,728
Total Depreciation	$29,997	$30,443	$30,543	$6,129	$5,607	$5,510	$5,463	$4,790	$5,864	$6,063	$5,802	$5,585	$35,728
Net Income after Depreciation	$92,175	$46,529	$62,979	$97,904	$100,944	$86,944	$92,075	$99,376	$109,730	$79,705	$86,106	$113,016	$99,841
Income Taxes	$23,966	$12,098	$16,375	$25,455	$26,245	$22,606	$23,939	$25,838	$28,530	$20,723	$22,388	$29,384	$25,959
Net Income	$68,210	$34,431	$46,604	$72,449	$74,699	$64,339	$68,135	$73,538	$81,200	$58,981	$63,718	$83,632	$73,882

Operating Income	$1,968,876	$0	$0	$0	$48,326	$93911	$106498	$91,245	$93,382	$85,491	$96,440	$122,172	$76,972	$93,522
Working Capital														
Accounts Receivable	$0	$0	$0	$0	($14,577)	($6,706)	($1,421)	$2,883	($669)	$899	($1,211)	($3,544)	$6,681	($2,817)
Accounts Payable	$0	$0	$0	$0	$4,912	$510	($97)	($500)	$316	$157	($127)	$124	($570)	$418
Supply Inventory	$0	$0	$0	($5,000)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
IGV Payment	($321,449)	$0	($6,007)	($35,083)	($15,497)	($16,663)	($16,358)	($15,554)	($18,117)	($14,726)	($12,402)	($13232)	($12254)	($12,978)
IGV Refund	$321,449		$0	$6,007	$35,083	$15497	$16663	$16,358	$15,554	$18,117	$14,726	$12,402	$13,232	$12,254
Total Working Capital	$0	$0	($6,007)	($34,076)	$9,921	($7,363)	($1,212)	$3,187	($2,917)	$4,447	$987	($4,251)	$7,089	($3,123)
Initial Capital														
Mine	$59,387	$0	$0	$59,387	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$95,567	$0	$23,892	$71,675	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TSF	$10,255	$0	$0	$10,255	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Cost	$2,508	$0	$627	$1,881	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sub Total	$167,716													
Sustaining Capital														
Mine	$160,802	$0	$0	$0	$14,581	$15994	$16736	$17,565	$18,464	$2,544	$1,380	$5,128	$4,244	$4,143
Process Plant	$20,000	$0	$0	$0	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000
TSF	$12,760				$0	$0	$0		$6,470	$6,290	$0	$0	$0	$0
Owner's Cost	$0	$0	$0	$0	$0		$0		$0	$0	$0	$0	$0	$0
Sub Total	$193,562													
Total Capital	$361,279	$0	$24,519	$143,198	$15,581	$16994	$17736	$18,565	$25,933	$9,835	$2,380	$6,128	$5,244	$5,143
Cash Flow before Taxes	$1,607,597	$0	($30,526)	($177,274)	$42,666	$69554	$87549	$75,867	$64,532	$80,104	$95,047	$111,792	$78,817	$85,256
Cumulative		$0	($30,526)	($207,800)	($165,134)	($95,581)	($8,031)	$67,836	$132,367	$212,472	$307,518	$419,311	$498,127	$583,383
Income Taxes	$418,140	$0	$0	$0	$6,102	$17780	$20986	$16,982	$17,337	$15,096	$17,916	$23,966	$12,098	$16,375
Cash Flow after Taxes	$1,151,629	$0	($30,526)	($177,274)	$36,563	$51,773	$66,563	$58,885	$47,195	$65,009	$77,130	$87,827	$66,719	$68,881
Cumulative		$0	($30,526)	($207,800)	($171,237)	($119,463)	($52,900)	$5,985	$53,180	$118,188	$195,319	$283,146	$349,865	$418,746
Financial Indicators before Taxes														
NPV @ 0%	$1,607,597													
NPV @5%	$869,108													
NPV @7.5%	$652,729													
NPV @ 10%	$494,911													
IRR	32.5%													
Payback	3.1				1.0	1.0	1.0	0.1	-	-	-	-	-	-
Financial Indicators after Taxes														
NPV @ 0%	$1,189,458													
NPV @5%	$623,712													
NPV @7.5	$457,741													
NPV @ 10%	$336,550													
IRR	26.3%													
Payback	3.9				1.0	1.0	1.0	0.9	-	-	-	-	-	-

Operating Income	$104,033	$106,551	$92,454	$97,538	$104,166	$115594	$85768	$91,908	$118,600	$135,569
Working Capital										
Accounts Receivable	($1,556)	($667)	$2,190	($1,010)	($412)	($2,496)	$4,507	$521	($3,442)	($966)
Accounts Payable	$133	$220	($202)	$238	($280)	$636	($441)	($826)	($78)	($740)
Supply Inventory	$0	$0	$0	$0	$0	$0	$0	$0	$0	$5,000
IGV Payment	($13,402)	($13,999)	($13,761)	($13,267)	($12,310)	($14,680)	($13,832)	($12,664)	($13,146)	($11,517)
IGV Refund	$12,978	$13,402	$13,999	$13,761	$13,267	$12310	$14680	$13,832	$12,664	$13,146
Total Working Capital	($1,846)	($1,044)	$2,226	($278)	$265	($4,229)	$4914	$863	($4,003)	$4,923
Initial Capital										
Mine	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
TSF	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Cost	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sub Total										
Sustaining Capital										
Mine	$4,985	$5,107	$5,452	$1,313	$739	$4,182	$3743	$8,139	$12,487	$13,874
Process Plant	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000
TSF	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owner's Cost	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sub Total										
Total Capital	$5,985	$6,107	$6,452	$2,313	$1,739	$5,182	$4743	$9,139	$13,487	$14,874
Cash Flow before Taxes	$96,202	$99,400	$88,228	$94,947	$102,692	$106,183	$85939	$83,631	$101,110	$125,618
Cumulative	$679,585	$778,985	$867,213	$962,160	$1,064,852	$1,171,035	$1,256974	$1,340,605	$1,441,715	$1,567,333
Income Taxes	$25,455	$26,245	$22,606	$23,939	$25,838	$28530	$20723	$22,388	$29,384	$25,959
Cash Flow after Taxes	$70,747	$73,154	$65,623	$71,007	$76,854	$77,653	$65216	$61,244	$71,726	$99,659
Cumulative	$489,493	$562,647	$628,270	$699,277	$776,131	$853,785	$919,000	$980,244	$1,051,970	$1,151,629
Financial Indicators before Taxes										
Payback	-	-	-	-	-	-	-	-	-	-
Financial Indicators after Taxes										
Payback	-	-	-	-	-	-	-	-	-	-

23 ADJACENT PROPERTIES

The Bayovar-Sechura phosphate deposit is host to a number of phosphate operations and projects in various stages of the development cycle including:

i. Current Producers:

   a. Vale – Miski Mayo Bayovar Mine

   b. Fosyeiki Mine

ii. Feasibility Studies/ Detailed Design:

   a. FOSPAC (Cementos Pacasmayo / Mitsubishi / Zuari)

iii. Exploration/ Preliminary Economic Assessment/ Prefeasibility Studies

   a. Focus Ventures

   b. GrowMax/Americas Potash Peru

Figure 4-2 in the report outlines concession boundaries.

The two most significant operations or projects in the area are Vale's Bayovar Mine and the FOSPAC Bayovar 9 Concession Project (completed Feasibility Study in 2014).

Vale's currently producing Bayovar Mine located on the Bayovar 2 concession, 15 km west of the Focus Bayovar 12 Concession, is one of the largest phosphate deposits in South America. Vale sold minority stakes in the project to Mosaic (35%) and Mitsui (25%) for $660,000,000 in 2010 (Vale 2010 Annual Report).

FOSPAC (Cementos Pacasmayo / Mitsubishi / Zuari) is developing a phosphate deposit on the Bayovar 9 concession, located immediately west of the Focus Bayovar 12 Concession and north of the Vale Bayovar Mine. FOSPAC completed a Feasibility Study on the project in early 2014. The project contemplates a mine life of 20 years based on 130 Mt (dry-density) of measured and indicated resources grading 17.5 wt. % P_2O_5 (FOSPAC Environmental Impact Assessment Report, October 2013).

Fosyeiki operates a small open pit phosphate mining operation on a narrow Concession between the Vale and FOSPAC concessions (approximately 200 m wide by 2,000 m long) located to the southwest of the Focus Bayovar 12 Concession. The operation includes stripping of overburden and mining of the PH01 PH02 and PH03 phosphorite beds by dozer and excavator. Basic processing is performed on site using a coal fired dryer that removes the moisture and some of the fines, resulting in a slight P_2O_5 product grade increase, prior to being bagged and sold as a direct application fertilizer in the domestic Peruvian market as well as abroad.

The GrowMax/ Americas Potash Peru project includes three concessions (Bayovar 6, Bayovar 7 and Bayovar 8) situated to the north of the Focus Bayovar 12 Concession. GrowMax released an initial NI 43-101 phosphate Mineral Resource technical report on the project in April 2015. GrowMax is currently conducting additional phosphate and potash exploration and evaluation activity on their three Bayovar concessions.

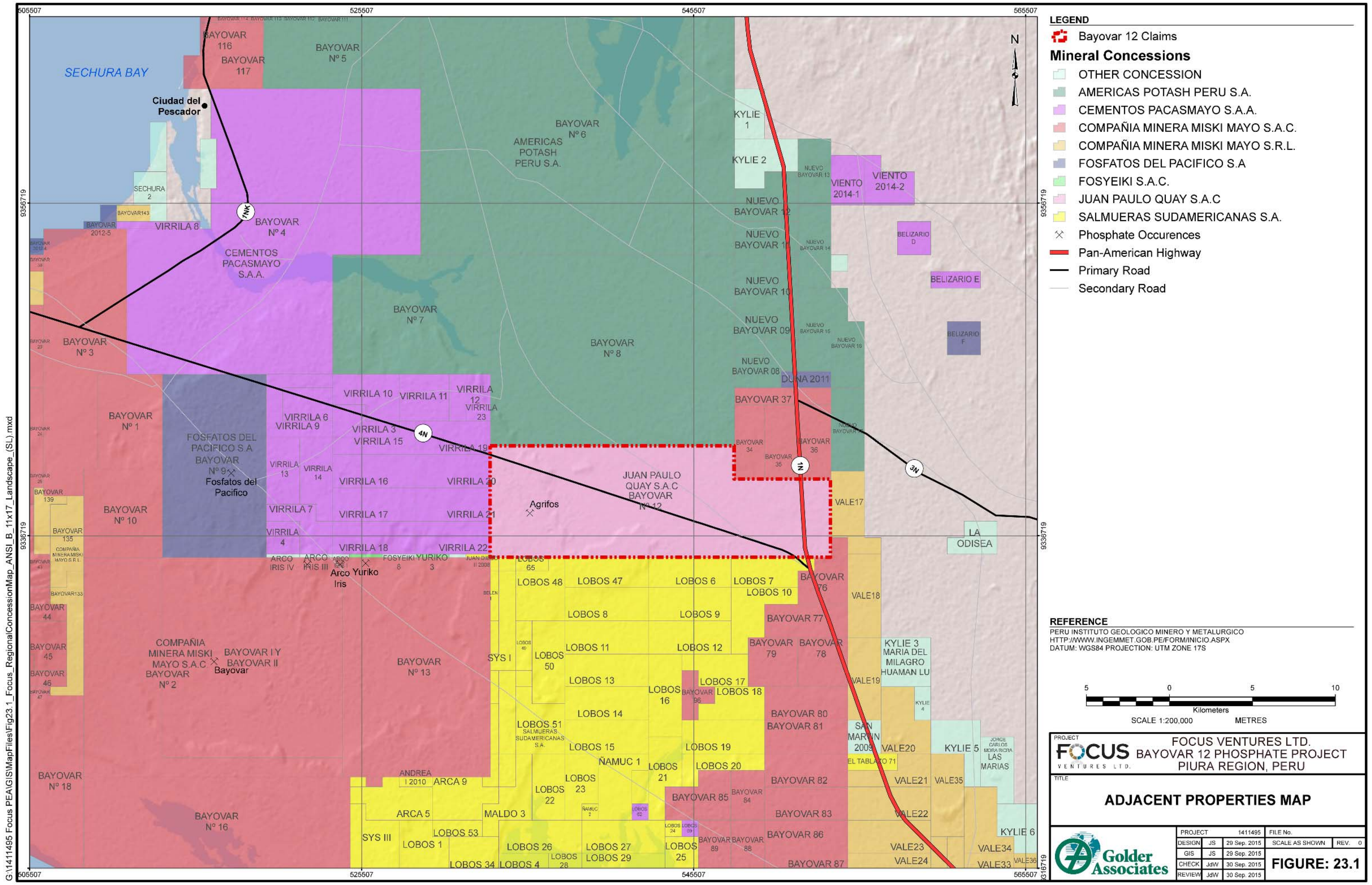


Figure 23-1: Adjacent Properties Map

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION PLAN

24.1.1 Description

The Project Execution Plan describes, at a high level, how the project will be carried out. This plan contains an overall description of what the main work focuses are, project organization, the estimated schedule, and where important aspects of the project will be carried out.

The project execution proposed incorporates an integrated strategy for engineering, procurement and construction management (EPCM). The primary objective of the execution methodology is to deliver the project at the lowest capital cost, on schedule, and consistent with the project standards for quality, safety, and environmental compliance.

24.1.2 Objectives

The project execution plan has been established with the following objectives:

- To maintain the highest standard of safety so as to minimize incidents and accidents;
- To design and construct a process plant, together with the associated infrastructure, that is cost-effective, achieves performance specifications and is built to high quality standards;
- To design and operate the mine using proven methodologies and equipment;
- To optimize the project schedule to achieve an operating plant in the most efficient and timely manner within the various constraints placed upon the project; and
- To comply with the requirements of the conditions for the construction and operating license approvals.

24.1.3 Plan of Approach

24.1.3.1 Philosophy

This section describes the execution plan for advancing the Bayovar 12 Project from the current Prefeasibility Report stage to production. The project execution plan will ensure that key project processes and procedures are in place that will:

- Develop a Project Schedule beginning with the Feasibility Study through Permitting, Construction and Commissioning;
- Consider significant project logistics;
- Develop and implement site communications, construction infrastructure, and water supply for an early and efficient startup;
- Plan for early construction mobilization;
- Develop a Health and Safety Plan that is comprehensive yet concise so that contractors, construction managers, and members of Focus Venture’s development team are safe during the field construction phase of the project;

- Develop and execute project control procedures and processes;
- Perform constructability reviews;
- Implement project accounting and cost control best practices;
- Issue a cost control plan and a control budget; and
- Oversee project accounting.

Focus Ventures intends to utilize an Engineering, Procurement and Construction Management (EPCM) approach utilizing multiple hard money and low unit cost prime contracts for CM, as the recommended method for executing the project. The capital cost estimate is based on this methodology. Mine development pre-production work activities as well as the water diversion tunnel, the site access road construction and power transmission line will be performed by contractors selected through a pre-qualification and pre-tending process. Because the project is located approximately one hour from the population centers of Sechura and Piura, FCV plans to hire construction workers from these areas, wherever possible.

Some sections affecting the project are:

- Ability to start work that does not require engineering;
- Availability of construction and engineering resources;
- Experience of the qualified firms considered and their typical and proposed approach; and
- An approach that utilizes the best resources available (matching contractors to the size of each contract).

As previously mentioned, M3 utilized an EPCM approach as the basis for the capital cost estimate. This approach provides for contracts that would include civil, concrete, structural steel, mechanical, piping, electrical and instrumentation.

The majority of mechanical and electrical equipment required for the project will be procured within the western hemisphere and fabricated items will be sourced in Peru. Concrete and building construction materials will be sourced locally, wherever possible. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment will be sourced within Peru, and to the extent practical, within the region.

24.1.3.2 Engineering

The PFS level EPCM schedule is based on financing and the Notice to Proceed by Peru permitting agencies to be granted in Year -3. Engineering will be done to match the plant protocol for drawing titles, equipment numbers and area numbers. Design will produce drawings in the Imperial System of Units (English) format. Drawings will be prepared in Spanish while specifications will be written in English.

A site conditions specification has been prepared and will be updated to ensure that vendors are aware of the site conditions and project specifications. Individual equipment specifications will be prepared for all procurement packages that are not off-the-shelf items.

Engineering controls will be maintained through drawing lists, specification lists, equipment lists, pipeline lists, cable schedule, and instrument lists. Control of Engineering Requisitions for Quote (ERFQ) will be performed through an anticipated purchase orders list. Progress will be tracked through the use of the lists mentioned.

Concrete reinforcing steel drawings will be done using customary bar available in Peru. Reinforcing bar will be fully detailed to allow either site or shop fabrication.

Structural steel will be detailed using TEKLA software. Mechanical steel will be dictated utilizing either Inventor or TEKLA. This will allow fabrication of steel prior to the award of steel installation contracts.

Owner review of engineering progress and design philosophy will be an ongoing process.

24.1.3.3 Procurement

Procurement of long delivery equipment is the main driver for the project schedule. Most major equipment for the Bayovar 12 project has lead times in the 20 to 36 weeks ARO timeframe not including overseas shipment to the site, with a few items in the 40 week delivery schedule. M3 has added 8 weeks for overseas shipping, customs clearance, and delivery to the site in the EPCM schedule. Submittal drawing review has been included in the overall delivery schedule.

Procurement of major process equipment will be conducted by the EPCM contractor, acting as Agent for FCV through the use of owner-approved purchase order forms. This will include all of the equipment in the equipment list as well as all of the instruments in the instrument list. Some instruments will be part of vendor equipment packages. In addition, structural steel, electrical panels, electrical lighting, major cable quantities, specialty valves and special pipe will be purchased. Contractors will be responsible for the purchase of common materials only.

Equipment and bulk material Suppliers will be selected via a competitive bidding process. Similarly, construction contractors will be selected through a pre-qualification process followed by a competitive bidding process. It is envisaged that the project will employ a combination of lump sum and unit price contracts as appropriate for the level of engineering and scope definition available at the time contract(s) are awarded.

It is intended that equipment will be sourced on a world-wide basis, assessed on the best delivered price and delivery schedule, fit-for-purpose basis.

Equipment will be purchased FOB at the point of manufacture or nearest shipping port for international shipments. A logistics contractor will be selected to coordinate all shipments of equipment and materials for the project and arrange for ocean and overland freight to the job site.

The EPCM contractor will be responsible for the receipt of the major equipment and materials at site. The equipment and materials will be turned over to the installation contractor for storage and safe keeping until installed. Bulk piping and electrical materials and some minor equipment will be made part of the construction contracts, and as such will be supplied by the various construction contractors. It is expected that each construction contractor provide for the receipt, storage, and distribution of materials and minor equipment they purchased.

The EPCM contractor will establish a list of recommended pre-qualified vendors for each major item of equipment for approval by FCV. The EPCM contractor will prepare the tender documents, issue the equipment packages for the bid, prepare a technical and commercial evaluation, and issue a letter of recommendation for purchase for approval by FCV. FCV through the assistance of the EPCM contractor will conduct the commercial negotiations with the recommended vendor and advise the EPCM contractor of the negotiated terms for preparation of the purchase documents. When approved, the EPCM contractor will issue the purchase order, track the order, and expedite the engineering information and delivery of the equipment to the site.

24.1.3.4 Inspection

The EPCM contractor will be responsible to conduct QA/QC inspections for major equipment during the fabrication process to ensure the quality of manufacture and adherence to specifications. Levels of inspection for major equipment will be identified during the bidding stage, which may range from receipt and review of the manufacturer's quality control procedures to visits to the vendor's shops for inspection and witnessing of shop tests prior to shipment of the equipment. Where possible, inspectors close to the point of fabrication will be contracted to perform this service in order to minimize the travel cost for the project. Some assistance may also be provided by the EPCM engineering design team.

24.1.3.5 Expediting

The EPCM contractor will also be responsible to expedite the receipt of vendor drawings to support the engineering effort as well as the fabrication and delivery of major equipment to the site. An expediting report will be issued at regular intervals outlining the status of each purchase order in order to alert the project of any delays in the expected shipping date or issue of critical vendor drawings. Corrective action can then be taken to mitigate any delay.

The logistics contractor will be responsible to coordinate and expedite the equipment and material shipments from point of manufacture to site, including international shipments through customs.

24.1.3.6 Project Services

The EPCM contractor will be responsible for management and control of the various project activities and ensure that the team has appropriate resources to accomplish FCV's objectives.

24.1.4 Construction

24.1.4.1 Construction Methodology

The construction program is scheduled to start in Year -2. The work includes civil site preparation of the plant site, the seawater pipeline construction, construction of the first seawater pond to provide water for construction, and the power line. Concrete foundations for the process buildings and other support structures will be constructed beginning in Year -2. The grinding-flotation building and autoclave buildings are planned to be a bridge-frame metal, moment frame structures. The truck shop, plant maintenance shop, and warehouse buildings are currently planned as pre-engineered metal buildings. Most of the ancillary buildings on the Bayovar 12 site are planned to be modular buildings including the Admin office, cafeteria, lab, security building and Medical/Emergency building.

Construction work is scheduled for approximately 18 months from mobilization to the commencement of commissioning. Earthworks associated with the well field and related facilities will commence after the permits have been released as soon as the contractor can be mobilized to the field. This work will include completion of the surface diversions, process building foundations and process ponds.

24.1.4.2 Construction Management

Construction Management will be done as Agent for the Owner using prime contracts for civil/concrete and structural/mechanical/electrical/piping/instrumentation. The contracting plan is based on utilizing local contractors to execute the construction work packages to minimize mobilization and travel costs. The EPCM contractor will pre-qualify local contractors and prepare tender documents to bid and select the most qualified contractor for the various work packages. Some work packages will include the design, supply, and erection for specific facilities which are specialized in nature. The EPCM team will be comprised of individuals capable of coordinating the construction effort, supervising and inspecting the work, performing field engineering functions, administering contracts, supervising warehouse and material management functions, and performing cost control and schedule control functions. These activities will be under the direction of a resident construction manager and a team of engineers, and locally hired supervisors, and technicians. There would also be a commissioning team to do final checkout of the project.

Construction progress will be measured by using quantity ledgers for construction quantities to develop percent completion and earned hours by contractors. Quantity surveyors will measure the amount of civil quantities, yards of concrete placed, tons of steel erected, and similar measures for architectural, piping and electrical quantities. Mechanical installations will be measured based on the estimated installation hours from the control estimate developed during detailed engineering.

Some site services will be contracted to third party specialists, working under the direction of the resident construction manager. Construction service contracts identified at this time include field survey and QA/QC testing services.

24.1.5 Contracting Plan

Contracting is an integral function in the project's overall execution. Contracting for the Bayovar 12 Project will be done in full accord with the provisions of the FCV/EPCM contract.

A combination of vertical, horizontal, and design-construct contracts may be employed as best suits the work to be performed, degree of engineering and scope definition available at the time of award. A concrete batch plant will be located on site that will use screened colluvial and alluvial materials native to Meadow Creek. There will not be a dedicated construction camp at the Bayovar 12 site.

The civil contract will cover all clearing, grubbing, bulk excavation, engineered fill, grading, and construction of TSF berms, ponds and pipe trenches.

The concrete contract will include all concrete forming, rebar, placement and stripping. If possible, the batch plant will be tied to the concrete placement contract to leverage the economy of having one management for both functions.

As part of the contracting strategy, a list of proposed contract work packages has been developed to identify items of work anticipated to be assembled into a contract bid package. Depending upon how the project is ultimately executed and the timing, several work packages may be combined to form one contract bid package. Table 24-1 represents the Proposed Contract Work Package list:

Table 24-1: Proposed Contract Work Package List

No.	Bid Packages:	Comments
1	Materials Testing	Soils, Concrete & Structural Materials
2	Survey	Confirm Existing Terrain. Create Topo of Roadway, Heap Leach & Plant Site Areas
3	138 kV Power Transmission Line	
4	Main Substation	Includes Emergency Generator Installation & Testing
5	Field Electrical Distribution - Sub Station to Process Areas, Camp & Water Pumping	Overhead lines and duct banks from switch gear
6	Seawater Supply System	Includes Pipeline; seawater intake, and pump installation
7	Septic System - Sewer Piping, Plant & Leach Field	Two septic systems required: process plant area and camp area
8	Clearing, Grubbing, Site Excavation, Engineered Backfill, Grading, Trenching, - all Areas	
9	Concrete Work - All Areas	
10	Structural Steel Buildings & Platforms	From foundation bolts. Includes roofing and siding installation.
11	Architectural Finishes	In offices and larger frame structure buildings including the lab building
12	Field Erected Tanks	Typically part of design-supply-erect contract.
13	Mechanical Equipment	Drum washer, attrition cells, cyclones, belt filter, concentrate dryer, dust collector, conveyors, & pumps
14	Process Piping & Field Instrumentation	
15	Instrumentation & Controls Programming	PLC programming, HMI screen development; I/O & communications.

24.1.6 Project Schedule

A sequence of effort has been developed from this study with a prospective schedule by which the project will likely proceed. The schedule includes Owner Activities, Engineering, Contracts, Procurement, Construction, Remaining Site Work, Site Pre-Commissioning, and Site Commissioning activities and is presented as Figure 24-1.

The 220 kV power transmission line will commence during Year -2.

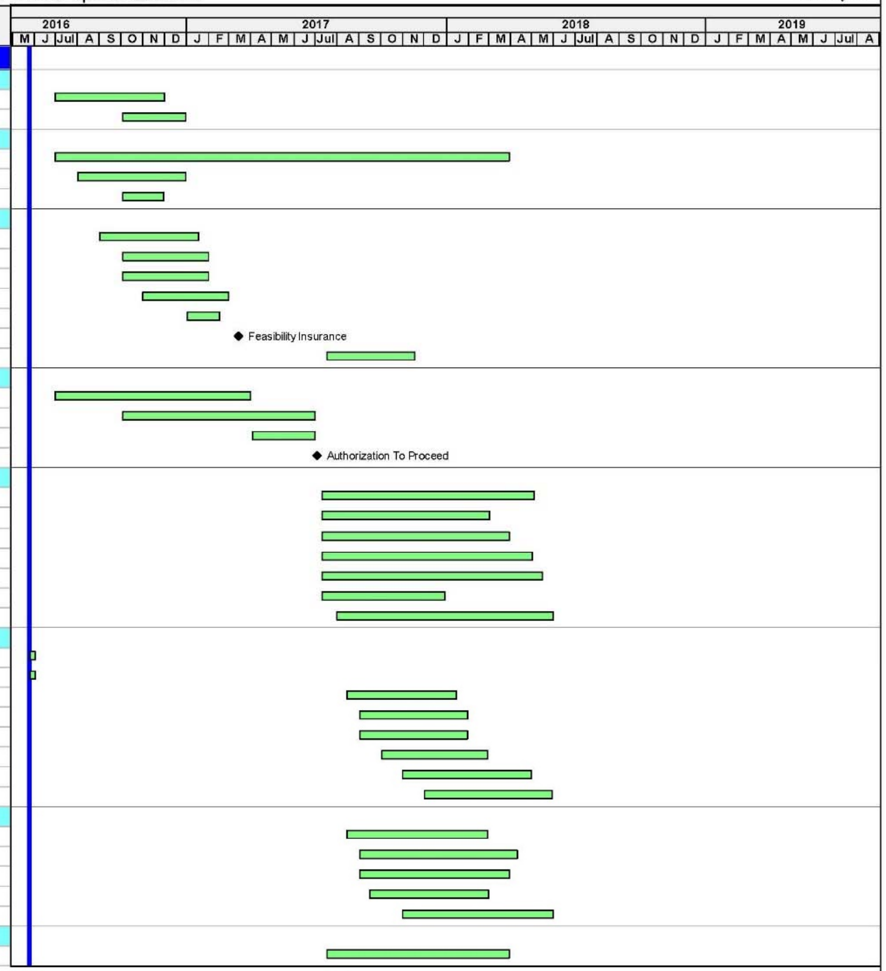
Construction Completion and Turn-over Procedure

The Construction Completion Procedure is part of the Construction Quality Plan as well as the project specific Commissioning Plan. Contractors are to enter into contractual agreements with Focus to perform certain portions of the work, which includes quality control of their work.

The Commissioning Plan will be developed and implemented to insure a step-by-step, documented process and procedure for all mechanical, process, electrical/instrumentation completion, checkout and pre-operational testing. Pre-operational testing and commissioning will take place concurrent with mechanical completion. Pre-operational testing is currently scheduled to commence in Year -1 and wet commissioning and start-up is scheduled to commence in Year -1.

PN# 140103 | **Bayovar 12 PFS Update Schedule** | 27-May-16

Activity ID	Activity Name	Original Duration	Start	Finish
Bayovar 12 PFS Update Schedule		37m	26-May-16	28-Jun-19
Geotechnical Studies		6m	01-Jul-16	30-Dec-16
GS-000-CI-000	Geotechnical Site Investigation	5m	01-Jul-16*	01-Dec-16
GS-000-CI-010	Geotechnical Analysis	3m	03-Oct-16*	30-Dec-16
Pilot Testing		21m	01-Jul-16	30-Mar-18
PT-000-OP-020	Marketing/Off-Take Agreemements	21m	01-Jul-16*	30-Mar-18
PT-000-OP-000	Pilot Plant Operation	5m	01-Aug-16*	30-Dec-16
PT-000-OP-010	Tailing Testwork	2m	03-Oct-16*	30-Nov-16
Feasibility Study		15m	01-Sep-16	17-Nov-17
FS-000-MP-000	Mine Planning	4m	01-Sep-16*	18-Jan-17
FS-000-CI-010	Infrastructure	4m	03-Oct-16*	31-Jan-17
FS-000-SS-000	Process Plant Design	4m	03-Oct-16*	31-Jan-17
FS-000-CI-000	TSF & Waste Dumps	4m	01-Nov-16*	28-Feb-17
FS-000-SU-000	Estimating & Financial Analysis	2m	03-Jan-17*	16-Feb-17
FS-000-OA-000	Feasibility Insurance	0m		15-Mar-17*
FS-000-BE-000	Basic Engineering	4m	17-Jul-17*	17-Nov-17
Environmental Assessment & Permitting		12m	01-Jul-16	03-Jul-17
EP-000-OA-000	Develop EIA	9m	01-Jul-16*	31-Mar-17
EP-000-OA-020	Community Relations	9m	03-Oct-16	30-Jun-17
EP-000-OA-010	Government Review	3m	03-Apr-17	30-Jun-17
EP-000-OA-030	Authorization To Proceed	0m	03-Jul-17	
Long Lead Procurement		11m	10-Jul-17	31-May-18
PO-EXXX-000	Substation Transformer	10m	10-Jul-17*	04-May-18
PO-MXXX-000	Drum Washer	8m	10-Jul-17*	02-Mar-18
PO-MXXX-010	Concentrate Dryer	9m	10-Jul-17*	30-Mar-18
PO-MXXX-020	Belt Filter	10m	10-Jul-17*	01-May-18
PO-MXXX-030	Attrition Cells	10m	10-Jul-17*	15-May-18
PO-QXXX-000	Mining Fleet	6m	10-Jul-17*	29-Dec-17
PO-MXXX-040	Slurry & Seawater Pumps	10m	01-Aug-17*	31-May-18
Detailed Engineering		24m	26-May-16	30-May-18
BE-000-JJ-000	I&C Design	0m	26-May-16	02-Jun-16
BE-000-ME-000	Mechanical Design	0m	26-May-16	02-Jun-16
BE-000-AR-000	Ancillary Buildings	5m	15-Aug-17*	15-Jan-18
BE-000-CI-000	Civil Grading & Pond Design	5m	01-Sep-17*	31-Jan-18
BE-000-CN-000	Concrete Design	5m	01-Sep-17	31-Jan-18
BE-000-SS-000	Structural Steel Design	5m	02-Oct-17	28-Feb-18
BE-000-PP-000	Piping Design	6m	01-Nov-17	30-Apr-18
BE-000-EL-000	Electrical Design	6m	01-Dec-17	30-May-18
Procurement		10m	14-Aug-17	31-May-18
PO-MXXX-070	Desal Plant	6m	14-Aug-17*	28-Feb-18
PO-MXXX-050	ROM Feeder Station	7m	01-Sep-17*	10-Apr-18
PO-PXXX-000	Tankage & Silos	7m	01-Sep-17	30-Mar-18
PO-MXXX-060	Conveyors	5m	15-Sep-17*	01-Mar-18
PO-SXXX-000	Structural Steel	7m	01-Nov-17	31-May-18
Contracts		8m	17-Jul-17	30-Mar-18
CO-CXXX-000	Contracts	8m	17-Jul-17*	30-Mar-18



Data Date: 26-May-16 | Actual Work | Work | Milestone | Page 1 of 2

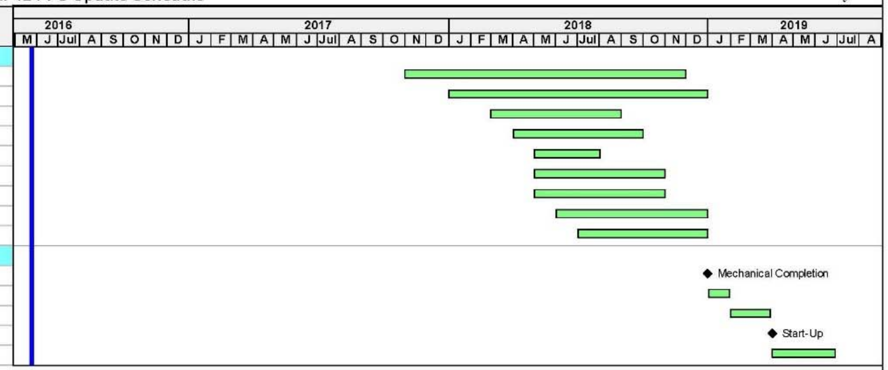
Figure 24-1: EPCM Schedule

PN# 140103

Bayovar 12 PFS Update Schedule

27-May-16

Activity ID	Activity Name	Original Duration	Start	Finish
Construction Management		14m	01-Nov-17	31-Dec-18
CM-000-CI-000	Pre-Stripping	13m	01-Nov-17*	30-Nov-18
CM-000-PP-000	Seawater Pipeline	12m	02-Jan-18	31-Dec-18
CM-000-CI-010	Site Grading	6m	01-Mar-18*	31-Aug-18
CM-000-CN-000	Concrete Placement	6m	02-Apr-18	01-Oct-18
CM-000-CI-020	Seawater Pond	3m	01-May-18	01-Aug-18
CM-000-ME-000	Mechanical Equipment Installation	6m	01-May-18	31-Oct-18
CM-000-SS-000	Structural Steel Erection	6m	01-May-18	31-Oct-18
CM-000-PP-010	Piping & Instrumentation Installation	7m	01-Jun-18	31-Dec-18
CM-000-EL-000	Electrical Installation	6m	02-Jul-18	31-Dec-18
Commissioning		6m	31-Dec-18	28-Jun-19
COM-000-SU-000	Mechanical Completion	0m		31-Dec-18
COM-000-SU-010	Precommissioning	1m	02-Jan-19	31-Jan-19
COM-000-SU-020	Commissioning	2m	01-Feb-19	29-Mar-19
COM-000-SU-030	Start-Up	0m	01-Apr-19	
COM-000-SU-30	Ramp-Up	3m	01-Apr-19	28-Jun-19



Data Date: 26-May-16 | Actual Work | Work | Milestone | Page 2 of 2

Figure 24-1: EPCM Schedule (Continued)

24.1.7 Quality Plan

A project specific, Quality Plan will be developed and implemented on the site. The Quality Plan is a management tool for the EPCM contractor, through the construction contractors, to maintain the quality of construction and installation on every aspect of a project. The plan, which consists of many different manuals and subcategories, will be developed during the engineering phase and available prior to the start of construction.

24.1.8 Commissioning Plan

The Commissioning Plan will also be project specific and is characterized as the transition of the constructed facilities from a status of "mechanically" or "substantially" complete to operational as defined by the subsystem list that will be developed for the project. The commissioning group will systemically verify the functionality of plant equipment, piping, electrical power and controls. This test and check phase will be conducted by discrete facility subsystems. The tested subsystems will be combined until the plant is fully functional. Start-up, also a commissioning group responsibility, will progressively move the functional facilities to operational status and performance.

In addition to these activities, the commissioning portion of the work will also include coordination of facilities operations training, maintenance training and turnover of all compiled commissioning documentation in an agreed form.

24.1.9 Health and Safety Plan

The Health and Safety Plan (HASP) will be established for the construction of the Bayovar 12 Project and any other authorized work at the project site. The HASP covers all contractor personnel working on the project and any other authorized work for the project.

The HASP specifies regulatory compliance requirements, training, certifications and medical requirements necessary to complete the project for all personnel and contractors involved in the project. Along with the Operations Procedures, the HASP is to be followed by all Contractor personnel working at the site.

24.1.10 Project Organization

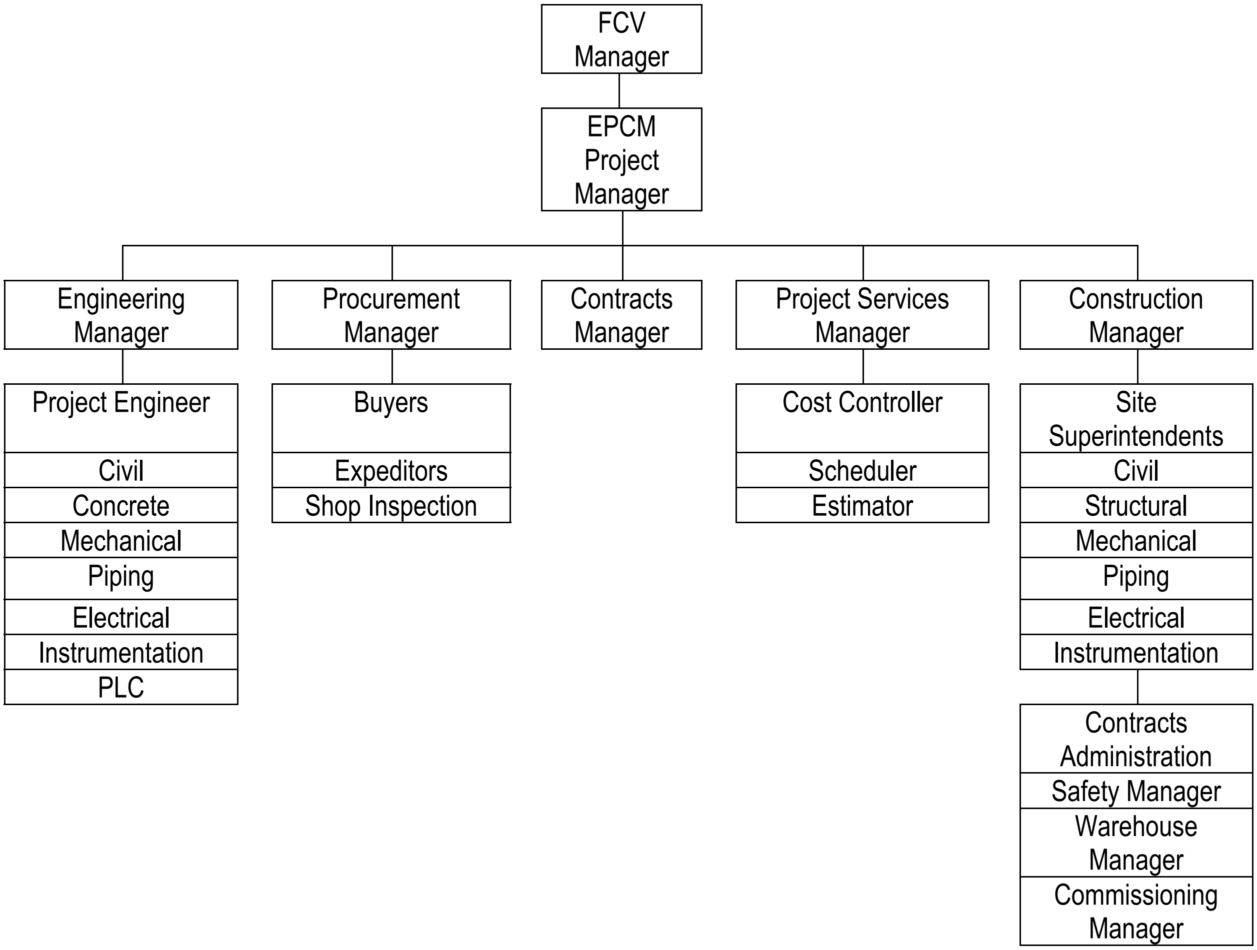


Figure 24-2: Project Organization Block Diagram

25 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

According to CIM definition standards for Mineral Resources and Mineral Reserves prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014, a Preliminary Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the "Modifying Factors" and the evaluation of any other relevant factors that are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserve at the time of reporting. A Preliminary Feasibility Study is at a lower confidence level than a Feasibility Study. Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves; these include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

25.2 INTERPRETATION

The QPs of this Report have reviewed the data for the Project and are of the opinion that the Bayovar 12 Phosphate Project meets the requirements for a Preliminary Feasibility Study. Opinions from individual QPs on the sections of the PFS that they are responsible for (see Section 2 for responsibilities) are set out in the following subsections.

25.2.1 Surface Rights, Royalties, and Mineral Tenure

Focus Ventures possesses a 70% ownership of the Bayovar 12 concession on which the Project is located as described in Section 4 of this Report, subject to the royalties, agreements, limitations and encumbrances described in Section 4.

25.2.2 Geology and Mineralization

The understanding of the regional and local geology with regards to the lithology, structure, alteration and mineralization for each of the mineralized zones and deposit types discussed in Sections 7 and 8 of this Report is sufficient to estimate the Mineral Resources and Mineral Reserves contained herein.

25.2.3 Exploration

The previous drilling exploration programs, along with the subsurface geologic interpretation, and mineralogy research carried out to date, reasonably supports the potential for expansion of defined phosphate deposits, and the potential for discovery of new open pit mineable prospects as discussed in Section 9 of this Report.

25.2.4 Drilling and Sampling

The drilling methods, recovery, collar survey, downhole survey, and material handling for the samples used in the Mineral Resource and Mineral Reserve estimates for this Report are sufficient to support the Mineral Resource and Mineral Reserve estimates contained in this Report, subject to the assumptions and qualifications contained in Sections 10 and 11 of this Report.

25.2.5 Data Verification

The data used for estimating the Mineral Resources for the Bayovar 12 Project is adequate for the purposes of this Report and may be relied upon to report Mineral Resources and Mineral Reserves based on the conditions and limitations set out in Section 12 of this Report.

25.2.6 Metallurgy

The metallurgical testing conducted on samples from exploration cores included extensive mineralogical studies and developmental metallurgical testing on each of the phosphorite beds (capas). The developmental metallurgical testing and analysis detailed in Section 13 of this Report supports the selection of the process flow sheet, making it possible to design a plant containing two process lines that can process all phosphate ores from the Project as they are mined subject to the conditions and limitations set out in Section 13 of this Report.

25.2.7 Mineral Resources

The Mineral Resource estimates in Section 14 of this Report are accurate to within the level of estimate required for categorization as Measured, Indicated and Inferred mineral resources suitable for use in a Preliminary Feasibility Study, subject to the conditions and limitations set out in Section 14 of this Report, and these estimates were performed consistent with industry best practices and demonstrate reasonable prospects for economic extraction, as required by NI43-101.

25.2.8 Mineral Reserves

A review of the designs, schedules, risks, and constraints of the Project detailed within this Report and given that there is, in the opinion of the QP, a basis for an economically viable Project after taking into account mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, governmental factors and other such modifying factors, thereby supporting the declaration of Mineral Reserves. Subject to the conditions and limitations in this Report, this PFS demonstrates that, as of the date of this Report, extraction can reasonably be justified. The term 'Mineral Reserve' does not necessarily signify that all governmental approvals have been received; it does signify that there are reasonable expectations that such approvals will be granted.

25.2.9 Mine Plan and Schedule

The mine plan and schedule detailed in Section 16 of this Report have been developed to maximize mining efficiencies, while utilizing the current level of geotechnical, hydrological, mining and processing information available and are, subject to the conditions and limitations set out in Section 16, sufficient to support the declaration of Mineral Reserves.

25.2.10 Metallurgical Recovery

The recovery methods including the major unit operations detailed in Section 17 of this Report comprising, drum washing, attrition scrubbing, desliming with hydrocyclone, and further classification with a hydrosizer. Downstream processing of concentrate includes dewatering using vacuum belt filters followed by concentrate drying using a rotary dryer. Process slimes are sent directly to the TSF, from where no process water is planned to be reclaimed. These unit processes are sufficient to demonstrate recoveries to support the mine planning and economics detailed herein, and the declaration of Mineral Reserves.

25.2.11 Infrastructure

The on-site and off-site infrastructure detailed in Section 18 of this Report are designed and cost estimated to a level of detail that supports Project viability and the economics detailed herein.

25.2.12 Market Studies and Contracts

Market Studies for DAPR in Section 19 of this Report are consistent with industry standards and market patterns, and are similar to other industrial minerals studies for developing product markets found throughout the world. The prices selected for 24% P_2O_5 DAPR and 28% P_2O_5 DAPR in this Report represent a probable range of scenarios that support a prefeasibility economic analysis.

25.2.13 Environment, Permits, and Social and Community Impacts

Section 20 of this Report summarizes the reasonable available information on: environmental studies conducted to date and the related known environmental issues associated with the Project, the Project related social and community impacts, the Project permitting requirements, and the requirements and plans for waste rock and tailings storage.

Additionally, mine closure, reclamation and mitigation are discussed and cost estimated to a level of detail that supports Project economic and technical viability to the level of a Prefeasibility Study and the economics detailed herein.

25.2.14 Capital and Operating Costs

The capital and operating costs detailed in Section 21 of this Report, which were derived from several previous Sections, are designed and cost-estimated to a level of detail that supports Project economic and technical viability to the level of a Prefeasibility Study and the economics detailed herein.

25.2.15 Financial Analysis

The financial analysis presented in Section 22 of this Report illustrates that the Project economics, subject to the conditions and limitations in this Report, are positive and can support estimation of Mineral Reserves and the demonstration of technical and economic viability to the level of a Prefeasibility Study.

25.3 CONCLUSIONS

The financial analysis presented in Section 22 demonstrates that the Bayovar 12 Phosphate Project is technically and economically viable and has the potential to generate satisfactory economic returns based on the assumptions and conditions set out in this Report. This conclusion warrants continued work to advance the Project to the next level of study, which is a Feasibility Study.

25.4 RISKS

The QPs of this Report are not aware of any unusual, significant risks or uncertainties that could be expected to affect the reliability or confidence in the Project based on the data and information available to date.

As with most projects at the preliminary feasibility level, there continues to be risks that could affect the economic potential of the Project. Many of the risks relate to the need for additional field information, laboratory testing, or engineering to confirm the assumptions and parameters used in this Report. External risks are, to a certain extent, beyond the control of the Project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. Table 25-1 identifies what are currently deemed to be the most significant internal Project risks, potential impacts, and possible mitigation approaches. In summary, the Project-specific risks identified following the PFS include:

- Mineral Resource modelling and classification;
- geotechnical engineering;
- areal variation of mineral recoveries;
- metallurgical recoveries;
- TSF management;
- Acquisition of easements and/or use permits for offsite infrastructure.

Table 25-1: Project Risks Identified Following the PFS

Risk		Explanation / Potential Impact	Possible Risk Mitigation
General Risks Common to the Mining Industry			
GR1	CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of Project success. An increase in OPEX of 20% would reduce the after tax $NPV_{7.5\%}$ by approximately $107 M using current open pit designs. If OPEX increases, then the mining cut-off grade would increase and, all else being equal, the size of the optimized pit would reduce, yielding fewer mineable tons of phosphate ore.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the accuracy of cost estimates.
GR2	Permit Acquisition or Delay	The ability to secure all of the permits to build and operate the Project is of paramount importance. Failure to secure the necessary permits could stop or delay the Project.	The development of close relationship with local communities, other stakeholders and government regulators, along with a thorough Environmental and Social Impact Assessment and a Project design that gives appropriate consideration to the environment and local expectations is required.
GR3	Ability to Attract Experienced Professionals	The ability of Focus Ventures to attract and retain competent, experienced professionals is a key success factor for the Project.	The early search for, and retention of, professionals may help identify and attract critical people.
GR4	Falling DAPR Prices	A drop in prices for DAPR during the mine development process could have a negative impact on the profitability of the operation, especially in the critical first years.	Begin construction when the outlook is good for price improvement.
GR5	Change in Permit Standards, Processes, or Regulations	A change in standards, processes, or regulations can have a significant impact in project schedules, operation cost and capital cost.	Maintain and continue to foster relationships with key federal and local legislators and regulators.
Bayovar 12 Phosphate Project Specific Risks			
PR1	Mineral Resource Classification	Mineral Resources Classification was based on geometric drilling distance which introduces some level of risk and uncertainty.	Perform a geostatistical confirmation drilling program to verify the mathematical distances for measured, indicated, and inferred resources.
PR2	Geotechnical Engineering	The geotechnical condition of the soils under the TSF, plant, and onsite and offsite infrastructure facilities may be different than assumed and could have financial implications on the Project CAPEX and/or OPEX.	Further field investigations are required to support the Feasibility Study.

Risk		Explanation / Potential Impact	Possible Risk Mitigation
PR3	Mining Dilution	The capa/interbed interface is difficult to see visually. The current assumption for dilution include 7.5 cm of roof and floor interbed included with each capa. Actual operation using selective miners could differ from current dilution assumptions.	Perform a test mine to determine the amount of dilution that is likely to be encountered during commercial mining.
PR4	Metallurgical Recoveries	Changes to metallurgical assumptions could lead to reduced P_2O_5 recovery and revenue, increased processing costs, and/or changes to the processing circuit design, which would all negatively impact the project economics. A 5% reduction in total phosphate recovery would reduce the $NPV_{7.5\%}$ by about $51M.	Pilot plant runs should be completed to support the Feasibility Study, to increase the confidence of the recovery assumptions and overall process design.
PR5	TSF Management	TSF management is a critical component of the Project. The tailings feed is designed for a percent solids of 9%, with no water reclaim.	More settling tests are needed to determine the optimum solids density of tailings to determine if the solids density can be increased at little cost.
PR6	Fuel Prices	The cost for electrical power, diesel, and LNG have been determined from current project pricing. While the electrical demand for the project is low, the costs for diesel and LNG for drying concentrate are significant drivers of the project economics	Update the financial model periodically with up-to-date fuel pricing to keep ahead of any significant financial changes.
PR7	Development or Construction Schedule	The Project development could be delayed or extended for a number of reasons, which would impact Project economics.	If an aggressive schedule is to be followed, FS field work and critical path laboratory testing should begin as soon as possible.

25.5 OPPORTUNITIES

There are many significant opportunities that could improve the economics, and/or permitting schedule of the Project. The major opportunities that have been identified at this time are summarized in Table 25-1. Further information and assessments are needed before these opportunities could be included in the Project economics.

The opportunities are separated into general opportunities common to the mining industry, and Project-specific opportunities unique to the Bayovar 12 Phosphate Project. The Project-specific opportunities are further categorized into three broad categories of potential to improve the Project Net Present Value (NPV); the categories, and a brief listing the opportunities, are provided below:

- Potential Benefit Opportunities include:
  - Out of pit conversion of Inferred Mineral Resources to Mineral Reserves adjacent to the current Mineral Reserves;
  - Metallurgical recovery improvements that improve the Project economics;
  - An increase in DAPR prices as demand for organic fertilizers increases;
  - Government funding towards off-site infrastructure.

Table 25-2: Project Opportunities Identified Following the PFS

Opportunity		Explanation	Potential Benefit
General Opportunities Common to the Mining Industry			
GO1	Permit Acquisition	In the same way that permit acquisition is a potential risk to the Project schedule, it may also be an opportunity. Peru is characterized as having a low jurisdictional risk, and as a mining friendly country.	The opportunity to shorten the permitting schedule exists.
GO2	Rising Commodity Prices	Increases in DAPR prices increase the revenue and Project economics.	Increased revenue increases financial factors.
GO3	Reagent/Fuel Price Decreases	Reduction in reagent and consumable prices, especially diesel and LNG has the potential to decrease operating costs and enhance the Project economics.	Lower OPEX may lead to higher net revenue and enhanced Project economics.
Project Specific Opportunities with Potential Benefit			
PO1	Extension of the Mine Life by extending the Open Pit to include all Indicated Resources	Additional drilling to the north of the current pitshell have the potential of increasing the grade and tonnage of the Mineral Reserves by (a) converting above cutoff Inferred Mineral Resources to Indicated, (b) adding new above cutoff mineralization in currently poorly drilled areas.	Increases in Mineral Reserve tonnages, especially at higher grades, could improve the Project economics, especially if those improvements could be realized in the early stages of development.
PO2	Re-location of the Bayovar-Chiclayo Highway within the project area for improved project economics	Relocation of the Bayovar-Chiclayo Highway to the north would allow expansion of the current pit to exploit additional resources in the area of the lowest overburden and permit the location of the TSF into the north part of the concession	Reduction in haulage costs and reduced TSF construction costs could improve the Project economics.
PO3	Use of Continuous Mining System	Review the mine schedule to incorporate use of a compact Bucket Wheel Excavator for overburden mining, and in-pit conveyors and spreaders for waste management	Incorporating a Continuous Mining System may increase capex but could significantly reduce mining operating costs.
PO4	Metallurgical improvements that improve the Project economics	The major testing required to potentially improve Project metallurgy include: scrubbing residence time studies, mineralogical profiling, flow sheet upside investigations, follow-up pilot testing	Further metallurgical testing is needed better define these opportunities.
PO5	Government funded off-site infrastructure	Government funding programs provides a unique opportunity to invest in road, transit and port projects that promise to achieve critical national objectives.	If the Bayovar 12 Phosphate Project is deemed to be a significant impact, any funding provided by the government would move costs out of CAPEX and/or OPEX.
PO6	Utilizing used equipment	The current PFS CAPEX is based on all equipment being purchased new and, as a result, used or salvaged equipment could be used in place of the currently estimated new equipment.	Utilizing used equipment, if available when required, could significantly reduce CAPEX and/or reduce the Project development timeframe.
PO7	Infrastructure Sharing with neighboring project	The FOSPAC phosphate deposit is located 15km west of the Bayovar12 and nearing completion of a Feasibility Study.	Sharing of infrastructure costs such as the electricity transmission line and seawater pipeline could significantly reduce initial CAPEX for both projects

26 RECOMMENDATIONS

26.1 MINERAL RESOURCES

Golder recommends the following actions:

Perform a targeted geostatistical drilling and analytical program designed to evaluate short range variability in grade and thickness and to improve the database for statistical and geostatistical analyses. Golder recommends that one 800 m by 800 m block be drilled off in a cross pattern of 50 m to 100 m spaced holes. Golder recommends that the geostatistics be evaluated further once additional drilling and analytical data are available to determine if the results support a less conservative area of influence based classification for the Bayovar 12 Concession Mineral Resources.

Infill drilling within Phase 1 and Phase 2 exploration drilling areas once the geostatistical drilling and modeling are completed and Measured classification distances are confirmed for the purpose of upgrading Mineral Resources from Indicated and Inferred categories into Measured Mineral Resouces.

Perform trial down-hole geophysical surveys for evaluation of potential quantitative identification of phosphorite beds for sampling;

Continue to collect additional relative density and moisture analytical data for all phosphorite and diatomite beds to improve calculated relative density values used in converting volume into tonnes;

Proceed with additional exploration drilling on 800 & 400-meter centers to extend coverage of Measured & Indicated Resources within the current pit limits and beyond, especially to the north and north east of the Bayovar-Chiclayo road

Update the geological model and Mineral Resource Estimates based on data and observations from any additional drilling and analytical work;

Update the NI 43-101 Technical Report for additional Mineral Resources with the results of any additional exploration program results.

26.2 MINERAL RESERVES

Infill drilling is recommended to bring the reserves in Updated Pre-Feasibility Study for mining Phases 1 through 4, which covers the phosphorite production during Years 1 through 5, into the proven category

The Mineral Reserves Estimate is based solely on the 20-year mine plan open-pit design with high wall laybacks and a maximum production rate of 1.0 M tonnes (dry) of phosphate concentrate per year. Although mineral resources exist outside the 20-year mine plan pit, the mine schedule was limited to the 20-year mine life and were not considered in the Mineral Reserves Estimation.

26.3 METALLURGICAL RECOMMENDATIONS

The individual layer test work used composite samples prepared from the cores from nine drill holes. Subsequent resource drilling and chemical analyses of the drill cores indicate that the average grade of the phosphorite layers has increased from 12.7 to 13.7% P_2O_5. It is therefore recommended that additional testing be performed with core samples of phosphorite layers from different locations within the proposed mine pit to determine if the recovery assumptions and predicted concentrate grades are applicable over the entire mine pit. The proposed number of drill holes to be included in this variability testing is five for the first 5-year mining block (one for each year), and two each for the 2nd, 3rd, and 4th 5-year mining blocks, for a total of 11 holes. If the proposed test work results demonstrate that the recovery assumptions and predicted concentrate grades are valid over the entire mine pit, the variability testing may be terminated. However, if the recovery assumptions and predicted concentrate grades are not applicable over the entire mine pit, more variability testing within the problematic mining blocks would be recommended. There is lower grade P_2O_5 in several of the interbeds which may be of high enough grade to be blended with adjacent capas for plant feed and still make saleable product grade. This would simplify the phosphorite mining and lower mining costs.

Additional settling tests and filtration tests are recommended to establish unit area requirements for high rate thickeners and filtration rates for concentrates prepared according to the proposed flowsheet. This work could be performed in conjunction with the variability testing program.

Future bench-scale metallurgical testing, if practical, should be performed with samples that have not been dried. If dried samples are available for use, they should be rehydrated to approximate the in situ moisture content prior to testing.

Pilot scale testing is recommended to support future feasibility study work. If the variability testing shows uniformity it may be possible to obtain the bulk samples of phosphorite layers from 150 mm diameter core samples. If the variability testing shows a variable response to beneficiation it will be necessary to obtain the bulk samples of phosphorite layers from different locations within the proposed mine pit. Drilling large diameter core will allow for distributed sampling.

26.4 MINERAL RECOVERY RECOMMENDATIONS

Test work should continue to develop improved P_2O_5 recovery in the plant and develop grade recovery curves by capa as noted in 26.3 of the previous Pre-Feasibility Report. The Bayovar 12 plant is sized based on the current level of metallurgical testwork. The drum scrubber, attrition cells, and hydrosizer for each process line are well matched to the throughput requirements. A great deal of power and energy costs are tied up in concentrate filtration and concentrate drying. M3 recommends investigating the following plant improvements:

- Optimize belt filtration to determine the best size and vacuum pressure to remove the maximum amount of moisture from the filter. Based on additional testwork outlined in Section 26.3, Focus should reinvestigate the best filtration technology and filter size to handle 1 million MTPA per annum.
- Optimize concentrate drying to determine the appropriate technology to dry the concentrates. The cost of drying concentrate to 4% moisture accounts for approximately one third of the processing cost. A rotary dryer is more forgiving for a wide range of particle sizes but is 20% less efficient for drying. The cost of a rotary dryer is higher than using fluid bed dryers that handle the same capacity. However, fluid bed dryers are limited to a narrow range of particle sizes and do not handle fines well. Drying testwork should be considered to solve the question.
- Investigate the methodology for onsite blending of DAPR concentrates with additives to make custom DAPR fertilizers. M3 investigated the mixing and storage equipment for DAPR fertilizer amendments and product storage as part of this study. However, this section of the plant is currently not included or costed into the plant design. In conjunction with more advanced market studies, Focus should investigate the requirement for onsite where rehandling will be less expensive versus constructing a mixing plant offsite near the place of final product delivery.

26.5 MINING EQUIPMENT AND APPROACH

Mining represents the largest component of cost to produce the product in the Updated Pre-Feasibility Study. Hauling and loading represents 56% and 21% of the mining cost, respectively. Approaches to lower these costs should be evaluated.

Overburden – The overburden represents 44% of the total waste by volume and in the Updated Pre-Feasibility study it is removed by front end loader – truck mining system. Alternative systems should be evaluated to potentially lower operating costs, such as;

- Bucket Wheel Excavator (BWE) and conveyor-stacker system
- Rotary cutter head and conveyor-stacker system
- Dozing to a trap (pocket) and conveyor-stacker system
- Load and truck haul to pit rim and convey to dumps

Interburden – The interburden represents the balance of the waste and is in thinner geometries than the overburden. The BWE and rotary cutter have better application to thicker mining geometries such as the overburden, but dozing the interburden to a dozer trip or to piles for loading may reduce cost and maintain the selectivity of interburden versus phosphorite.

26.6 INFRASTRUCTURE RECOMMENDATIONS

Recommended additional work related to the geotechnical design of the TSF includes:

- Development of detailed surface topography at an accuracy of at least 0.6 m is needed to improve volumetric estimates for the impounded tailings and tailings dam.
- Install a weather station, including a Pan Evaporation Class A, on the Project site to start collecting local data in order to refine the evaporation and precipitation estimates made in the present evaluation.
- Perform field investigation and laboratory testing program in footprint of TSF embankment and WSF.
- Develop detailed water balance model that accounts for monthly variations in evaporation and simulates a range of annual climate patterns.

Perform laboratory slurry consolidation testing of samples of the tailings to determine achievable tailings placed density and volume of retained solution.

Next to the TSF, the seawater supply pipeline is the largest cost driver for the process plant. M3 developed a PFS-level design for the seawater delivery system. However, because of cost implications, it will be important to develop a detailed design for the pipeline that includes:

- Anchoring details for seawater intake structure;
- Pipeline trench dimensions and depth;
- Develop detailed line pressure ratings for the pipeline route to confidently determine the length of SDR 11 pipe vs SDR 13.5 pipe vs SDR 17 pipe;
- Design and cost the pipeline to handle HDPE expansion and contraction with varying temperature and design and cost a pipeline anchoring strategy;
- Determine that an easement for the proposed pipeline route is available and the distance that will be required from the Bayovar-Chiclayo Highway to run the pipeline.

The power transmission line is a relatively short distance from the La Niña substation to the plant. Focus will have to investigate the easement of the power line route to the site that does not interfere with Vale's power line.

Geotechnical conditions for the foundations for power line towers along the power line route will need to be investigated further.

The daily number of concentrate trucks will test the integrity of the Bayovar-Chiclayo Highway. Focus needs to obtain permits and approvals to use the Bayovar-Chiclayo Highway for the long-term delivery of concentrates to the Port of Bayovar.

In order to expand the mineral resource and tailings impoundment to the north, the Bayovar-Chiclayo Highway will need to be relocated on the Bayovar 12 concession. Focus will need to commence a study to move and rebuild the highway to the north of its current location.

26.7 OPA RECOMMENDATIONS

- Additional geotechnical investigation and laboratory testing is needed to develop reliable material parameters for use in the stability analyses for the open pit involving approximately 5 to 10 geotechnical drill holes. Geotechnical drill holes should be extended approximately 20 m into Underburden to adequately characterize those materials that may influence global stability.
- Additional hydraulic packer testing should be carried out to verify lateral continuity of the hydraulic properties of Overburden and Interburden and to collect data on the Underburden materials.
- More robust numerical seepage models should be carried out to improve estimates of seepage inflow rates.
- Water balance models should be refined to evaluate monthly evaporation and statistically developed rainfall patterns to evaluate the ability to manage process, seepage, and storm water for a range of climatic and operating conditions.
- Additional CBR testing should be completed

26.7 COST ESTIMATE FOR RECOMMENDED WORK

The estimated budget to carry out the recommended work is summarized in Table 26-1. Costs are based on experience with similar projects or from hours estimates for each recommended task.

Table 26-1: Recommended Work

Recommended Additional Work Tasks	Cost Estimate (US$)
Targeted geostatistical drilling & analysis (1,800 m)	$300,000
Additional resource definition drilling (Indicated & Measured categories) & analyses (up to 4,000 m)	$940,000
Additional Geotechnical Characterization of TSF	$100,000
OPA Geotechnical Investigation and Design	$175,000
Additional bench scale testwork for grade determination and variability definition across the extent of open pit	$250,000
Additional bench scale testwork for filtration & settling	$30,000
Bulk sampling by large diameter core and pilot scale metallurgical testwork	$1,500,000
Hydrogeological Studies	$205,000
Filtration & dryer optimization	$30,000
DAPR amendment design study	$100,000
Seawater supply pipeline engineering and easement determination	$50,000
Power transmission line easement definition geotechnical assessment for foundations	$60,000
Concentrate haulage permit and highway relocation study	$65,000
Total Cost	$3,805,000

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APPENDIX A – CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE of QUALIFIED PERSON

I, Conrad E. Huss, P.E., Ph.D., do hereby certify that:

1. I am Senior Vice President and Chairman of the Board of:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Rd., Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I graduated with a Bachelor's of Science in Mathematics and a Bachelor's of Art in English from the University of Illinois in 1963. I graduated with a Master's of Science in Engineering Mechanics from the University of Arizona in 1968. In addition, I earned a Doctor of Philosophy in Engineering Mechanics from the University of Arizona in 1970.

3. I am a Professional Engineer in good standing in the State of Arizona in the areas of Civil (No. 9648) and Structural (No. 9733) engineering. I am also registered as a professional engineer in the States of California, Illinois, Maine, Minnesota, Missouri, Montana, New Mexico, Utah, and Wyoming.

4. I have worked as an engineer for a total of forty-three years. My experience as an engineer includes over 37 years designing and managing mine development and expansion projects including material handling, reclamation, water treatment, base metal and precious metal process plants, industrial minerals, smelters, special structures, and audits.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am the principal Qualified Person for the preparation of the technical report titled "Bayovar 12 Phosphate Project NI 43-101 Updated Pre-feasibility Study, Department of Piura, Peru", (the "Technical Report") with an effective date of May 11, 2016, prepared for Focus Ventures Ltd. I am responsible for Sections 1, 2, 3, 4, 5, 6, 18, 19, 21, 22, 23, 24, 25, 26, and 27.

7. I have not visited the Bayovar 12 Concession property.

8. I was the principal Qualified Person for the previous prefeasibility study, "Bayovar 12 Phosphate Project NI 43-101 Pre-feasibility Study, Department of Piura, Peru" with an effective date of December 18, 2015 and amended on April 4, 2016.

9. As of 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 contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of June, 2016.

Signature of Qualified Person

Conrad E. Huss, P.E., Ph.D.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

As the qualified person responsible for preparing the technical report entitled **Bayovar 12 Phosphate Project NI 43-101 Updated Prefeasibility Study, Department of Piura, Peru**, with an **effective date of May 11, 2016** and prepared for Focus Ventures Ltd., I, **Jerry DeWolfe, P. Geo.**, do hereby certify that:

a) I am an Associate and Senior Geologic Consultant at:

Golder Associates Ltd.

102, 2535 3rd Avenue S.E., Calgary, Alberta, Canada T2A 7W5

b) I am a member in good standing of the following professional associations:

- Association of Professional Engineers and Geoscientists of Alberta (APEGA)
- Association of Professional Engineers and Geoscientists of British Columbia (APEGBC)
- Association of Professional Geoscientists of Ontario (APGO).

c) I graduated with a Bachelor of Science with Honours in Geology, from Saint Mary's University, Halifax, Nova Scotia, Canada, in 2000. I graduated with a Master of Science in Geology, from Laurentian University, Sudbury, Ontario, Canada, in 2006.

I have worked as a geologist for 16 years. My experience has focused on exploration, mine geology and resource estimation of phosphate, coal, oil shale and other stratigraphically controlled deposits, base metals deposits and precious metals deposits.

As a result of my education, professional qualifications, and experience, I am a Qualified Person as defined in National Instrument 43-101.

d) I completed a personal inspection of the Bayovar 12 Concession property that is the subject of the technical report from July 2 to July 5, 2014.

e) I am responsible for the preparation of items 7, 8, 9, 10, 11, 12 and 14 as well as pertinent resource geology contributions to items 1, 25 and 26 included in this technical report.

f) I am independent of the issuer, Focus Ventures Ltd. in accordance with the guidelines and requirements presented in Section 1.5 of National Instrument 43-101.

g) I served as the Golder Associates Ltd. Qualified Person during the preparation of two previous NI 43-101 technical reports on the project titled *Updated NI 43-101 Mineral Resource Technical Report on the Bayovar 12 Phosphate Project, Piura Region, Peru* (dated October 5, 2015, effective September 10, 2015) and *NI 43-101 Mineral Resource Technical Report on the Bayovar 12 Phosphate Project, Piura Region, Peru* (dated October 23, 2014, effective August 31, 2014). I also served as the Golder Associates Ltd. Qualified Person during the preparation of the previous NI 43-101 prefeasibility study titled "Bayovar 12 Phosphate Project NI 43-101 Pre-feasibility Study, Department of Piura, Peru" with an effective date of December 18, 2015 and amended on April 4, 2016.Additionally, I served as the Golder Associates Ltd. senior resource geologist during the preparation of the field program methodology and guidelines that were prepared collaboratively by Focus Ventures Ltd. and Golder Associates Ltd.; I have had no other prior involvement with the project.

h) I have read National Instrument 43-101, Form 43-101F1 and the Companion Policy 43-101CP, and this technical report has been prepared in compliance with the guidelines presented in NI43-101, Form 43-101F1 and 43-101CP.

i) As of the effective date of this technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

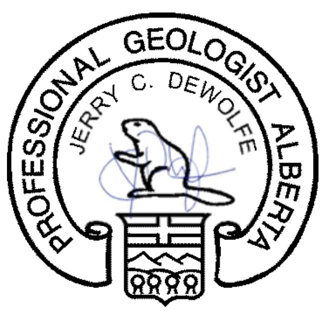
Dated at Calgary this 28th of June, 2016.

Jerry DeWolfe (signed by)

Signature of Qualified Person

Jerry DeWolfe, P. Geo.

Print name of Qualified Person



CERTIFICATE OF QUALIFIED PERSON

I, Herbert E. Welhener do hereby certify that:

1. I am a Vice President of Independent Mining Consultants, Inc. located at 3560 E. Gas Road, Tucson, Arizona, USA;;
2. This certificate applies to the technical report titled "Bayovar 12 Phosphate Project - NI 43-101 Updated Pre-Feasibility Study" (the "Technical Report"), prepared for Focus Ventures, Ltd. The Effective Date of the Technical Report is 11 May 2016;
3. I am a graduate with a Bachelor of Science in Geology from the University of Arizona in 1973 and I have practiced my profession continuously since that time. Since graduating I have worked as a consultant on a wide range of mineral projects, specializing in precious, base and industrial metals. I have undertaken many mineral resource estimations, mine evaluation technical studies and due diligence reports in a variety of settings around the world. I am a registered member of the Society of Mining, Metallurgy and Exploration, Inc. (SME RM # 3434330).
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5. I visited the Bayovar 12 property on January 31 and February 1, 2016.
6. I am co-author and reviewer of this report and have specific responsibility for the Mineral Reserve estimate and Sections 15, 16, 18.9, 21.1 and parts of 1, 25, 26 and 27 in the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1; the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th day of June, 2016.

Herbert E. Welhener, SME RM

SME
Society for Mining, Metallurgy & Exploration
Herbert E. Welhener
SME Registered Member No. 3434330
Signature ____
Date Signed 6/28/16
Expiration date 12/31/16

CERTIFICATE of QUALIFIED PERSON

I, Glenn A. Gruber, do hereby certify that:

1. I am currently the sole agent and Managing Member of Phosphate Beneficiation LLC:

   365 Rensselaer Ave
   Auburndale, FL 33823-9211

2. I am a graduate of the Haileybury School of Mines (1965). I am a graduate of Michigan Technological University and received a Bachelor of Science degree in Metallurgical Engineering in 1968.

3. I am a:
   - Qualified Professional Member of the Mining and Metallurgical Society of America (Member Number 01481QP)
   - Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 125380)

4. I have practiced metallurgical and mineral processing engineering for phosphate metallurgy, process design and project management since 1970. I have worked for International Minerals and Chemicals Company (a vertically integrated phosphate mining company) as Senior Process Engineer, Shift Supervisor, and Plant Metallurgist over a three-year period. I worked for Agrico Chemical Company (a vertically integrated phosphate mining company) as Assistant Chief Metallurgist and Beneficiation Plant Superintendent for 4 years. I was employed by Jacobs Engineering Company for 36 years as Director of Minerals Technology and Manager of Mining and Beneficiation, where I managed the phosphate laboratory and professionals that performed feasibility studies and due diligence studies of domestic and foreign phosphate operations and provided process engineering design for phosphate beneficiation plants.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 13 "Mineral Processing and Metallurgical Testing" of the technical report titled "Bayovar 12 Phosphate Project NI 43-101 Updated Pre-feasibility Study, Department of Piura, Peru" (the "Technical Report") dated effective May 11, 2016, prepared for Focus Ventures Ltd.

7. I was the Qualified Person for the previous prefeasibility study, "*Bayovar 12 Phosphate Project NI 43-101 Pre-feasibility Study, Department of Piura, Peru*" with an effective date of December 18, 2015 and amended on April 4, 2016.

8. I did not complete a personal inspection of the Bayovar 12 Concession property that is the subject of the technical report; however, I prepared some of the laboratory test

procedures, observed the samples as received by the beneficiation laboratory, and I observed several laboratory tests performed on core samples from the Focus deposit.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of June, 2016.

Glenn A Gruber

Signature of Qualified Person

Glenn A. Gruber

Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Thomas L. Drielick, P.E., do hereby certify that:

1. I am currently employed as Sr. Vice President by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Rd., Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of Michigan Technological University and received a Bachelor of Science degree in Metallurgical Engineering in 1970. I am also a graduate of Southern Illinois University and received an M.B.A. degree in 1973.

3. I am a:
   - Registered Professional Engineer in the State of Arizona (No. 22958)
   - Registered Professional Engineer in the State of Michigan (No. 6201055633)
   - Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 850920)
   - Member in good standing of AACE (Association for the Advancement of Cost Engineering) International, Inc. (No. 05031)

4. I have practiced metallurgical and mineral processing engineering and project management for 44 years. I have worked for mining and exploration companies for 18 years and for M3 Engineering & Technology Corporation for 26 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am the Qualified Person for Process Engineering for the preparation of the technical report titled "Bayovar 12 Phosphate Project NI 43-101 Updated Pre-feasibility Study, Department of Piura, Peru", (the "Technical Report") with an effective date of May 11, 2016, prepared for Focus Ventures Ltd. I am responsible for Section 17 of the Technical Report.

7. I have not visited the Bayovar 12 Concession property.

8. I was the Qualified Person for Process Engineering for the previous prefeasibility study, "Bayovar 12 Phosphate Project NI 43-101 Pre-feasibility Study, Department of Piura, Peru" with an effective date of December 18, 2015 and amended on April 4, 2016.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 28th day of June, 2016.

Signature of Qualified Person

Thomas L. Drielick

Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

As the qualified person responsible for preparing the technical report entitled **Bayovar 12 Phosphate Project NI 43-101 Updated Pre-feasibility Study, Department of Piura, Peru**, with an **effective date of May 11, 2016** and prepared for Focus Ventures Ltd., I, **Edward H Minnes, P.E.**, do hereby certify that:

a) I am a Mining Practice Leader at:

Golder Associates Inc.

44 Union Blvd, Suite 300 Lakewood, CO 80228

b) I graduated with a Bachelor of Applied Science – Mining Engineering, from Queen's University, Kingston, Ontario, in 1984.

c) I am licensed as a professional engineer in Missouri, USA and registered member of the Society for Mining, Metallurgy, and Exploration.

d) I have worked as a mining engineer for a total of 30 years since graduation from Queen's University.

e) I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

f) I am responsible for preparation of Section 20 of the Technical Report.

g) I am independent of the issuer, Focus Ventures Ltd. in accordance with the guidelines and requirements presented in Section 1.5 of National Instrument 43-101.

h) I was the Qualified Person for mine engineering for the previous prefeasibility study, "Bayovar 12 Phosphate Project NI 43-101 Pre-feasibility Study, Department of Piura, Peru" with an effective date of December 18, 2015 and amended on April 4, 2016.

i) I have read National Instrument 43-101, Form 43-101F1 and the Companion Policy 43-101CP, and this technical report has been prepared in compliance with the guidelines presented in NI43-101, Form 43-101F1 and 43-101CP.

j) As a result of my education, professional qualifications, and experience, I am a Qualified Person as defined in National Instrument 43-101.

k) As of the effective date of this technical report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Denver this June 28, 2016.

Signature of Qualified Person

Edward Minnes, P.E.
Print name of Qualified Person