MBAC FERTILIZER CORP.

Feasibility Study (FS)
Santana Phosphate Project
Pará State, Brazil

On Behalf of - **MBAC Fertilizer Corp**

Effective Date – **28 October 2013**

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1 SUMMARY

1.1 Introduction

Andes Mining Services Ltd. (AMSL), NCL Brasil Ltda. (NCL) and PegasusTSI Inc. (PegasusTSI) have been commissioned by MBAC Fertilizer Corp (MBAC) to prepare a Feasibility Study (FS) for the Santana Phosphate Project, in Pará State, Brazil.

AMSL, NCL and PegasusTSI are collectively referred to in this report as "AMSL/NCL/PegasusTSI"

The FS has been prepared under the guidelines of Canadian Institute of Mining (CIM) and National Instrument 43-101 and accompanying Form 43-101F1 Technical Report and Companion Policy 43-101CP (collectively, NI43-101).

This FS follows the Pre-Feasibility Study completed by AMSL/NCL/PegasusTSI dated June 7, 2012, as amended August 27, 2012, and titled "Pre-Feasibility Study (PFS) Santana Phosphate Project Pará State, Brazil, As Amended and Restated", which is abbreviated in this report as (AMSL/NCL/PegasusTSI PFS).

MBAC is developing the Santana Phosphate Project in the southeast of Pará State near the state border of Mato Grosso and Pará States in Northern Brazil.

The Santana Phosphate Project consists in one project site with two operational facilities: a Mine Facility and Industrial Facility. The Mine Facility is anticipated to include an open pit phosphate mine to produce an average of 1,500,000 tonnes per year of phosphate ore with an average P_2O_5 grade of 12.86%. The expected mine life for the project is 32 years besides the pre stripping period. The Industrial Facility consider a beneficiation plant to produce 300,000 tonnes per year of a 34% P_2O_5 concentrate, and a Single Super Phosphate (SSP) plant expected to produce 500,000 tonnes per year and a Sulfuric Acid Plant with a production capacity of 230,000 Tonnes per year (700 tpd). The phosphate concentrate will be transported by trucks between the mine and industrial facilities.

1.2 Project Description and Location

The Santana Phosphate Project is located in the southeast of Pará State near the state border of Mato Grosso and Pará (figure 1.2_1). The project site is located approximately 230 km west from Santana do Araguaia (Pará State) in the São Felix do Xingu County.The topographical coordinates of the project are 417882 East and 8967784 North (Datum SAD69 Zone 22 South).

Figure 1.2_1
Project Location

1.3 Ownership

MBAC, through its 100% owned Brazilian subsidiaries Itafos Mineracao Ltda. (Itafos) and MBAC Fertilizantes, is the sole registered and beneficial holder of 23 exploration properties, 1 of these a Mining Permit under application and additional 5 exploration permits under application for a total of 28 claims and 235,150ha.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access to the Santana Phosphate Project is via approximately 125km of all weather gravel roads and 80km of paved roads from the regional city centres of Santana do Araguaia, in ParáState (northeast of Santana) or via Vila Rica city in Mato Grosso State (southwest of Santana).

The climate is tropical with an annual rainfall of around 2,000mm and seasonal variations with a drier period between June and November and a wetter period between December and May. The average annual temperature is approximately 27.5ºC with minimal month to month variations. Exploration is undertaken at the project all year round. Drilling activities are intensified during the drier season, from May to November, and continues with adequate equipment during the rainy season.

Santana is an exploration camp utilising the Fazenda Santa Luzia cattle station basic infrastructure which includes a house and sheds. All electricity is supplied by diesel generators and all consumables are brought in from the nearest major cities located over 200km away.

The original forest vegetation across the Santana project area was drastically reduced and substituted by grassland for cattle pasture; as a consequence, the hydrography of the region was seriously affected by the reduction of water streams. The Project region is completely occupied by cattle farms with no social infrastructure. The closest village (80 km) is Garimpinho which has only 200 inhabitants and absolutely no social infrastructure. Vila Mandi (141 km) with approximately 1,000 inhabitants has very poor infrastructure.

Based on the above considerations, there is very little availability of human resource with basic technical knowledge. As well as this, no professional schools were identified. As a recommendation, a plan to create manpower for the project should be considered, utilizing professionals to set up a “training program” for the operation of each specific plant.

The mine and industrial facilities at Santana Project site will utilize approximatedly 420 workers including direct and indirect people. This will require the building of accommodations, including a basic infrastructure to enable a normal living. This housing area will be built in the outer limits of the project site.

The Project site is about 370 km from the closest high tension facility located in the city of Miracema. The electric power consumption in the mine and industrial facilities is estimated to be 10 -12 MWh. The Sulfuric Acid Plant has a cogeneration capacity of 7.5 MWh. CELPA sent a reference term to ANEEL with the request of a Power line (238 kV) to Santana de Araguaia. EPE (Energy Research Company from Brazilian govern) is the responsible to verify the demand and approve the Power line. It will allow MBAC the launching of a high tension power line (138 kV) from Santana do Araguaia up to the project site. CELPA belongs to GRUPO REDE which also controls CELTINS in Tocantins State, which is working with MBAC on the Itafos-Arraias Project.

The Project will take water for the mine site from Capivara river, which is at approximately 11km southwest from the mine site. In addition, a drilling survey to evaluate the potential for ground water will be undertaken in the near future.

1.5 History

In June 2010, MBAC through its wholly owned Brazilian subsidiary, Itafos Mineração Ltda. (Itafos), acquired, via an option agreement, two Exploration Licenses numbers 850.335/2010 and 851.090/2008 from Mr. Natanael Rodrigues da Silva. These claims have been transferred to MBAC by the Federal Mines Department (DNPM). In the same year, MBAC applied for nine Exploration Licenses, of which six were granted by the DNPM, and the remaining three have priority.

Mr. Natanael Rodrigues da Silva initially claimed the first area for lateritic nickel, but had identified the area as prospective for phosphate via discussions with local farmers who had returned anomalous phosphate levels in agricultural soil sampling (Figure 1.5_1).

Figure 1.5_1
Santana Phosphate Deposit - Typical Site Drilling Location

Prior to MBAC involvement there had been no material exploration or mining activities undertaken on the project or the region.

1.6 Geological Setting

The Santana property is located within the Iriri Group of hydrothermally-altered volcaniclastic and carbonate rocks of Precambrian age. The hydrothermally-altered volcaniclastic rock consists of lapilli and crystal tuff, with clasts varying considerably in size from very coarse sand-size clasts to cobble-sized particles (agglomerate, volcanic breccia).

The phosphate mineralization occurs in a hydrothermally altered (or more correctly metasomatic) lapilli tuff, consisting mainly of carbonate and/or silica. A higher grade supergene enriched zone has been interpreted to lay sub-horizontally within close proximity to the saprolite / fresh rock boundary. Lower grade mineralization has been intercepted in the deeper fresh rock drilling but has not been interpreted due to lack of data.

1.7 Exploration

MBAC commenced exploration across the Santana Project area in December 2010.

Exploration activities to-date have included project scale mapping, 20 line kilometres of ground magnetic surveys, airborne radiometrics survey, a ground penetrating radar test survey along with Auger, Reverse Circulation (RC) and Diamond Core (DC) drilling.

1.8 Mineralization

The phosphate mineralization occurs in a hydrothermally altered (or more correctly metasomatic) lapilli tuff, consisting mainly of carbonate and/or silica. MBAC refer to this as a hydrothemalite. There is a basalt unit identified in the footwall, but this has not been intercepted in drilling to date.

The highest grade mineralization occurs in the soil and saprolite domains. This residual mineralization was formed by the downward and lateral movement of meteoric waters that have weathered the crystalline carbonate bedrock to material that is often gritty in texture at the base, becoming more argillaceous vertically. However, the gritty passages may occur in any interval where the weathering has been less intense. There are manganiferous units, and units consisting of buff-coloured clay, interpreted as having once been more carbonatic.

The saprolite overlies the bedrock forming essentially tabular bodies, the geometry of which is controlled by the bedrock topography.

The thickness of the saprolite varies considerably from less than a metre overlying bedrock to greater than 75m. Drilling suggests that bodies are thicker near the base of the volcanic massifs becoming shallower outwards (15 to 20 m) towards the central parts of the alluvial plains.

1.9 Drilling

MBAC have undertaken substantial programs of auger, reverse circulation (RC) and diamond drilling (DC) across the Santana Phosphate Project, as summarized below in Table 1.9_1.

Table 1.9_1
MBAC Drilling Summary - Santana Phosphate Project

Drilling Technique	Company	Number of Drillholes	Metres Drilled
Auger	MBAC	132	1,051.60
Reverse Circulation	Servitec Sondagens	277	13,108
Diamond Core	Geosonda Sondagens	12	883.64
Diamond Core	Servitec Sondagens	302	17,493.27

All drilling has been undertaken and/or supervised by MBAC technical personnel.

RC and DC drilling and data collection methods applied by MBAC have been reviewed by AMSL during the site visits.

1.10 Sampling and Analysis

A summary of the current drilling completed by MBAC along with laboratories utilised for each phase of drilling is shown in Table 1.10_1 below.

MBAC is now utilizing a combination of lithium tetraborate fusion followed by ICP-MS and XRF for the most recent diamond drilling program completed in 2012.

Table 1.10_1 Laboratories Used in Analysing MBAC Drilling					
Year	**Company Name**	**Type of Drilling**	**Number of Holes**	**Meters Drilled**	**Lab Used**
2010	MBAC	Auger	97	832.3	ALS
2011	MBAC	Auger	35	219.3	ALS
2011	Servitec	RC	277	13,549	ALS
2011	Geosonda	DC	12	862.06	ALS
2011	Servitec	DC	102	5,406.83	ALS
2012	Servitec	DC	4	223.90	ALS
2012	Servitec	DC	196	11,879.12	SGS Geosol

1.11 Security of Samples

Core is currently transported directly from the Santana Project to Vila Mandi (town with core preparation and storage base). After logging, core samples are marked for splitting and sampling by MBAC geologists. Each core sample is placed in a plastic bag which in turn is placed in a nylon bag for transporting via truck to the ALS sample preparation laboratory located in Campos Belos, or to the SGS preparation laboratory in Paráuapebas, Pará State. Vila Mandi processing facilities and storage base is approximately 110km by road from the Santana Project area.

AMSL considers the core sampling security to meet current industry best practice.

1.12 Mineral Processing and Metallurgical Testing

MBAC has commenced metallurgical testwork late in 2010. For the FS it has considered the testwork done until July 2012 that was used to define a preferred process route that is described in this report.

Testwork covered mineralogical characterizations of different samples, bench tests that had the intention to verify the variability of samples from different portion of the deposit to the flotation process and the grindabillity characteristics, verify the performance of the process in continuous tests in pilot plant, verify the High Intensity Magnetic Separation to reduce the iron grade of concentrates produced with samples of high iron grade. The samples utilized in the study came from the drill cores from holes DD-01 to RC -118 and trench samples. The testwork has been undertaken at the MBAC's process facilities located at their Itafos phosphate mine in Brazil, in the Technological Characterization laboratory facilities of the São Paulo State University for mineralogical and cominution studies, in the FUNMINERAL mineral processing laboratories of Goiais State for complementary flotation tests, in Goceix Foundation pilot plant of Minas Gerais state for continuous grinding and desliming evaluations, in CDTN laboratories pilot plant tests at in Minas Gerais state for continuous evaluation of column flotation, in the ERIEZ laboratory in São Caetano-São Paulo for magnetic separation tests and in Andritz and Outotec for concentrate dewatering evaluations.

Mineralogical identification of representative samples indicated that main phosphate mineral is apatite and the main gangue minerals are quartz, iron oxides and clay minerals.

Flotation tests showed that most of the the Santana samples evaluated responded well to the process with high grade concentrates easily obtained.

The following preferred process route has been selected by MBAC:

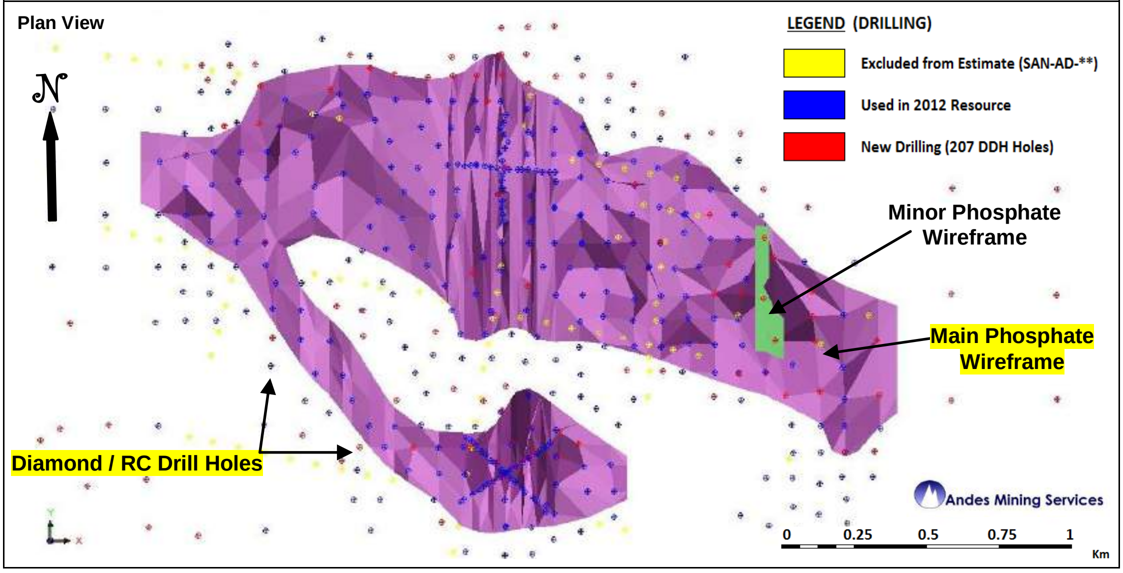
- Primary crushing of the ROM material;
- Classification of the ore into two size fractions, +6mm and -6mm;
- Grinding of the coarse product to 90% passing 106 microns;
- Low intensity magnetic separation to remove magnetite;
- Desliming of the fine product in hidrocyclones to discard the ultra-fines,
- Flotation of the grinded product mixed with the deslimed material;
- Regrind of the concentrate to 95% passing 75 microns;
- High intensity magnetic separation of the final concentrate; and
- Concentrate dewatering in thickener and filter.

1.13 Mineral Resources

The Santana indicated and inferred mineral resource estimate is based on 314 diamond holes (18,376.91m) and 277 RC holes (13,108m) drilled at a spacing of approximately 100m by 100m. MBAC have stopped all infill and extension drilling programs pending results from the FS currently underway. Only data received as at 17th January 2013 has been used in this estimate.

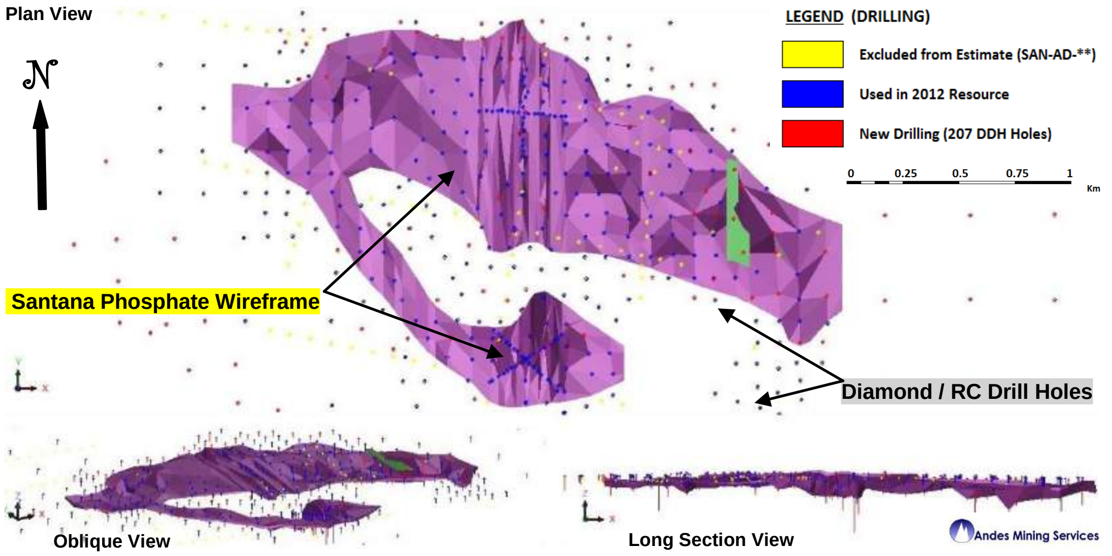
Figure 1.13_1

Phosphate Domain and Drilling (3% P_2O_5 cut-off) - MBAC Santana Project (AMSL, 28th October 2013)



The mineral resource estimate has focused on the main supergene enriched oxide mineralization with 2 flat lying mineralized domains defined using the saprolite and fresh geological boundary along with a 3% P_2O_5 grade cut-off to guide the wireframing process (Figure 1.13_1 to Figure 1.13_2).

Figure 1.13_2
Phosphate Domain and Drilling (3% P_2O_5 cut-off) - MBAC Santana Project (AMSL, 28th October 2013)



An independent mineral resource has been estimated comprising an indicated mineral resource of 60.36 Mt at 12.04% P_2O_5 and an inferred mineral resource of 26.59 Mt at 5.56% P_2O_5 (using a 3% P_2O_5 cut-off) (Figure 1.13_3 and Table 1.13_1 below).

The mineral resource statement has been classified by Qualified Person Bradley Ackroyd (MAIG) in accordance with NI 43-101. It has an effective date of 28th October 2013.

Figure 1.13_3
Grade Distribution for Santana Resource Estimate - MBAC Santana Project (AMSL, 28th October 2013)

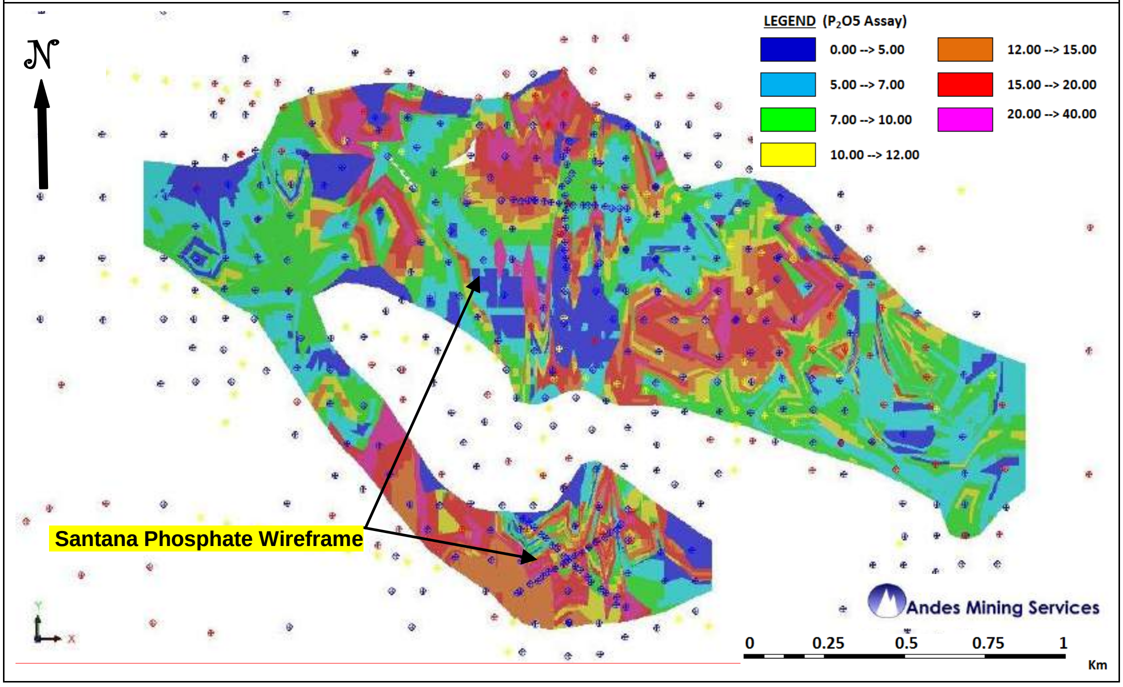
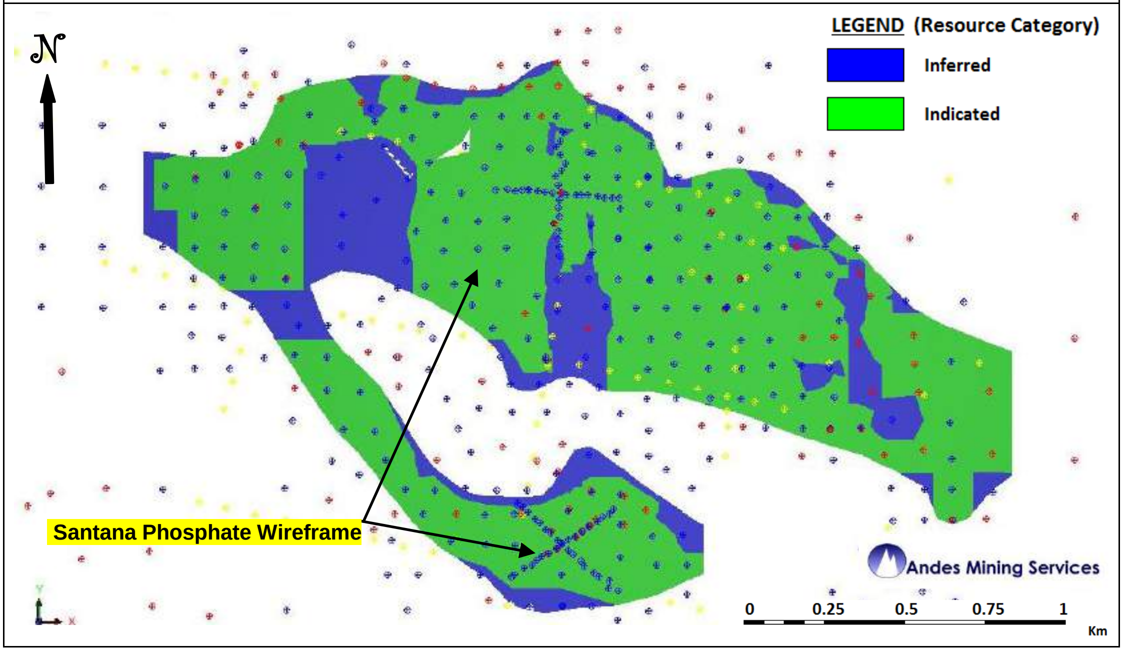


Figure 1.13_4
Indicated and Inferred Resource Categories - MBAC Santana Project (AMSL, 28th October 2013)



Mineral resources that are not mineral reserves do not have demonstrated economic viability.

AMSL and MBAC are not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that may materially affected the Mineral Resource Estimate.

Table 1.13_1 MBAC - Santana Phosphate Project Indicated and Inferred Mineral Resource Grade Tonnage Report - 28th October 2013 Ordinary Kriging (OK) (Block Model – 25mE X 25mN X 4mRL - Cut off 3% P_2O_5 utilised)										
Domain	**Cut-Off (% P_2O_5)**	**Tonnes (Mt)**	**P_2O_5**	**CaO**	**Fe_2O_3**	**LOI**	**MnO_2**	**Al_2O_3**	**SiO_2**	**TiO_2**
Inferred Mineral Resource										
Soil	3.0	0.21	9.21	9.58	19.68	7.30	1.60	14.15	34.25	1.00
Saprolite	3.0	4.90	10.10	14.92	14.89	5.88	1.85	8.00	39.77	0.70
Fresh	3.0	21.48	4.49	44.75	3.61	32.60	0.91	1.22	7.38	0.12
TOTAL INFERRED -->		**26.59**	**5.56**	**38.97**	**5.82**	**27.47**	**1.08**	**2.58**	**13.57**	**0.23**
Indicated Mineral Resource										
Soil	3.0	1.53	10.70	10.13	22.61	10.02	3.61	16.45	21.42	1.06
Saprolite	3.0	58.83	12.08	18.86	16.22	7.72	2.63	7.15	30.89	0.65
TOTAL INDICATED -->		**60.36**	**12.04**	**18.64**	**16.38**	**7.78**	**2.65**	**7.39**	**30.65**	**0.66**

Appropriate rounding has been applied to Table 1.13_1.

1.14 Mining Operations

NCL studied the Santana Project as a conventional open pit operation, producing 300 kt per year of phosphate concentrate at a grade of 34% P_2O_5.

Whittle Four-X pit optimization software was used to generate an optimal pit shell. Table 1.14_1 shows the parameters used on the pit optimization.

Table 1.14_1

Pit Optimization Parameters

Super Simple Phosphate (SSP) Price	350	USD/tonne SSP
Recovery	55%	P_2O_5
Concentrate Grade	34%	P_2O_5
SSP Grade	19%	P_2O_5
Processing		
May be selected for processing Sand-Ore, Saprolite-Ore and Fresh-Ore where :		
- CaO/P_2O_5 >= 1		
- Al2O3 + Fe2O3 < 30%		
- Only Measured and Indicated resources were used.		
Costs		
Mining	1.63	USD/tonne mined
Processing	10.91	USD/tonne processed
G&A	1.86	USD/tonne processed
Others	3.58	USD/tonne processed
Sulphuric Acid	55.30	USD/tonne SSP
Granulation	43.30	USD/tonne SSP
Slope Angle	34	degrees

The economic shell generated at the reference SSP price (350 USD/t SSP) is the basis of the operational pit design. Twenty intermediate phases were designed to prepare a LOM schedule.

The Santana Project mine schedule shows production of 300 kt per year of phosphate concentrate at a grade of 34% P_2O_5. Total material movement rate varies from 3.2 Mtpy at the start of the project up to a maximum of 9 Mt in later years. This scenario results in the processing of an average of 1.5 Mtpy of ore with an average P_2O_5 grade of 12.86%. The expected mine life for the project is 32 years besides the pre stripping period.

The mine is scheduled to work seven days per week or 356 days per year. Each day will consist of three 8-hour shifts. Four mining crews will cover the operation.

A diesel mining fleet will be used on the open pit. 97% of the material within the open pit can be removed without the use of explosives by a fleet of 5 m^3 hydraulic excavator that will load

32 t trucks. Ore will be hauled to the primary crusher for processing while waste will stored in the waste dumps surrounding the open pit or will be dumped in areas of the pit already exhausted.

The initial mining capital expenditure will be US$12.41 million. This includes US$ 7.74 million in mine equipment, US$ 4.21 million in operational expenditures during the prestripping period and US$ 0.47 million in other expenses.

Sustaining capital amounts to US$37.79 million. This includes all capital expenditures from Y01 onwards, and comprises equipment replacement and fleet increases needed to maintain a yearly production of 300 kt of P_2O_5 concentrate.

Total capital expenditures during the life of the project, including both initial capital and sustaining capital is US$ 50.20 million.

All the above-mentioned cost are inclusive of taxes.

Mine operating cost during commercial production is US$ 294.30 million or US$ 1.57/t moved.

1.15 Mineral Reserves

NCL prepared the Mineral Reserves Estimate under the supervision of Mr Carlos Guzmán BSc Mining Engineering, RM Chilean Mining Commission 119, FAusIMM 229036, Director of NCL and Qualified Person as defined in the CIM Guidelines.

Based on the whole body of work prepared for the Santana Project FS, it is the opinion of NCL that the mine production schedule defines the mineral reserve for a mining project.

Table 1.15_1 reports the mineral reserve of the Santana Project based on the production schedule used for this study.

The Mineral Resources stated in Section 14 are inclusive of the Mineral Reserves.

Table 1.15_1

Mineral Reserve Summary

Ore Reserves	Mass (tonnes ´000)	P_2O_5 (%)	Phosphate Concentrate (tonnes ´000)
Proven Mineral Reserve	-	-	-
Probable Mineral Reserve	45,481	12.86	9,459
Total Reserve	45,481	12.86	9,459

Notes for the Mineral Reserves Statement:

(1) Based on a price for SSP (Super Simple Phosphate) of US$350.00 per tonne;

(2) At a 3% P_2O_5 cut off grade;

(3) Open pit reserves assume complete mining recovery;

(4) Open pit reserves consider the inherent dilution of the resource block model;

(5) Waste tonnes within the open pit amount to 143.3 Mt at a strip ratio of 3.15:1 (waste to ore);

(6) Numbers may not add due to rounding;

(7) Plant Recovery: 55%;

(8) The mineral reserves for the Project were estimated by Carlos Guzmán, RM Chilean Mining Commission 119, FAusIMM 229036 and Director of NCL Brasil Ltda. in accordance with the Canadian Securities Administrators National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and generally accepted Canadian Institute of Mining, Metallurgical and Petroleum "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines ("CIM Guidelines");

(9) Reserves Effective Date: October 28, 2013.

(10) All mining modifiers, including aspects relating to metallurgy, processing, infrastructure and/or mining have been included in the Mineral Reserve determination. Environmental, permitting, legal, title, taxation, socio-economic, marketing, and or political factors have also been considered, where relevant, and are discussed in various sections of this report

1.16 Indicative Economics

This document provides a report on the results of mineral resources, mining, processing and a FS economic analysis of the potential project development. This FS and its economic analysis are based on the mineral reserve estimate for Santana as of 28th October 2013.

The indicative economics for the production of 300Ktpa of phosphate concentrate at a grade of 34% P_2O_5, equivalent to 500 Ktpa of SSP, is presented in Table 1.16_1 below. The financial model indicates robust project economics with a Net Present Value (NPV) of US$396 million at a discount rate of 10% and an IRR of 19.9%. These results are based on a start of construction in Q3 2014 and a start of production in Q2 2016.

In summary the following economic results have been obtained from the feasibility project phase engineering work and economic analysis:

- Capital Cost of US$427 million, including US$50 million of contingencies.
- Initial working Capital of US$9 million.
- Sustaining Capital of US$209 million, including:
    - Environmental Compensation: US$2.5 million expended one time at the first year of operation (2016)
    - Mine equipment: US$38 million, starting in 2016 with funds expended during the whole project life.
    - Tailing dam: US$21 million, starting in 2019 and expended at a rate of US$2.1 million every 3 years.
    - Industrial site: US$148 million, starting in 2017 and expended during the project life at a rate of:
        - 2017 – 2020: US$2 million per year
        - 2021 – 2025: US$4 million per year
        - 2026 – 2045: US$6 million per year

- Closure cost: US$20 million, expended in the last 2 years of the operational life, (US$10 million per year).

- Operating costs of $113 per metric tonne of Granulated Single Super Phosphate (GSSP) in 2017 (expected to be the first year of full production), including a 5% contingency.

- Initial projected selling price of GSSP in 2016 estimated at US$345 per metric tonne (first year of sales).

- Project life evaluation based on a mine life of 32 years of operation with mineral reserves delivering an average grade of 12.9% P_2O_5 in the ore.

- Internal Rate of Return (IRR): 19.9% in 2013.

- Net present Value of US$396 million @ 10% WACC (Weighted Average Cost of Capital)

- Payback of 5 years (from start of operation in 2016).

Additionally a sensitivity analysis for NPV was carried out based on the following scenarios:

- SSP Price (±20%) versus Sulphur Price (±20%)

- SSP Price (±20%) versus Capex (±20%)

- SSP Price (±20%) versus Opex (±20%)

- SSP Price (±20%) versus WACC (8% to 12%)

See section 22.2.3 detailed results of the above mentioned sensitivity analysis scenarios.

Table 1.16_1
Santana Phosphate Project
Indicative Economics

Economic Indicators	Unit	
Currency Exchange Rate	BRL/USD	2.84 (avg. 2013 - 2016)
Weigthed Average Cost of Capital (WACC)	%	10
Mine Life	Years	32
SSP Sales Price (2016)	USD/t	345
Operating Costs (2017)	USD/t	113
Initial Capex (2013 – 2015)	USD MM	377
Working Capital (2015)	USD MM	9
Contingency	USD MM	50
Total Capex	USD MM	436
Sustaining Capital (2016 – 2046)	USD MM	209
Closure Costs (2046 – 2047)	USD MM	20
NPV @ 10% (2013)	USD MM	396
IRR (2013)	%	19.9
Payback	Years	5

1.17 Exploration and Development

Drilling and studies completed to date have defined Indicated and Inferred mineral resources at Santana. The data collected is considered to be of moderate to high quality and suitable for resource estimation.

Further scope exists to improve the geological and mineral resource estimation confidence. AMSL makes the following specific recommendations:

- Increase the drill density of a 400m by 400m area to 50m by 50m spacing to allow measured resources to be defined. DC is the preferred method of sample recovery for this infill drilling program, however RC drilling is suitable if dry samples can be procurred.

- To utilize airborne radiometrics surveys as a first pass exploration tool to help guide regional exploration drilling programs.

A recent airborne radiometrics survey has highlighted the location of the Santana Phosphate Project quite effectively. Based upon a recent radiometrics survey completed across the mineralized domain(s) at Santana, AMSL would suggest the opportunity for lateral extensions to mineralization in the vicinity of the currently defined resource is limited.

Further step-out exploration drilling is not warranted at this stage of the project, and further exploration drilling should be focused on infill drilling portions of the existing resource to a 50 x 50m drill spacing to allow Measured resource category to be defined.

Infill drilling a portion of the existing resource to a 50m x 50m drill spacing was a key recommendation presented in the previous resource estimate completed by AMSL in April 2012, and in the more recent PFS completed by AMSL/NCL/PegasusTSI in June 7, 2012, as amended August 27, 2012.

Using the updated mineral resource estimation, which will include a significant proportion of higher confidence Indicated category resources, and utilizing the results from a previous Pre-Feasibility Study completed, an updated mineral reserve estimation has been reported.

With these results, more detailed mine planning and production schedules should be generated for the open pit.

1.18 Conclusions and Recommendations

The qualified persons from AMSL/NCL/PegasusTSI consider that the proposed exploration and development strategy is entirely appropriate and reflects the potential of the Santana Phosphate Project.

Further scope exists to improve the geological and mineral resource estimation confidence.

The qualified persons from AMSL/NCL/PegasusTSI have made a number of recommendations within this report to increase the mineral resource confidence and project development.

AMS would suggest the drilling of a small number of 50 x 50m spaced infill drill holes across the highest grade portions of the orebody (near surface) in order to define a measured mineral resource category.

The cost estimate for the recommended exploration and evaluation work program is shown in Table 1.18_1 below

Table 1.18_1
Santana Phosphate Project
Proposed Resource and Evaluation Expenditure

Activity	**Total (US$)**
DC and RC Drilling	$ 1,000,000
Assaying and Characterization	$ 200,000
Geology	$ 50,000
Travel and Accommodation	$ 30,000
Field Supervision and Support	$ 150,000
Administration	$ 70,000
Sub-Total	**$1,500,000**

The Santana Phosphate Project demonstrates encouraging economics based on the scoping study concepts, cost projections and price assumptions as presented in this FS.

The qualified persons from AMSL/NCL/PegasusTSI recommend that the Project be advanced in one phase of work to the design and implementation level.

PegasusTSI has worked closely with MBAC during the Santana Project Feasibility Study Phase in order to implement industry best-practices in fertilizer plant design as well as capturing some benefits based on the very recent design and construction experience from the Itafos SSP Project. In this way a more cost competitive product can be produced on a consistent basis. Based on an intimate familiarity with the Santana Project, PegasusTSI recommends that further engineering and procurement “lessons learned” activities from the Itafos project should be included as part of the detailed engineering and procurement implementation plan. Throughout this next pivotal phase of work this valuable source of experience from the Itafos Project will be used, including specific experience from the (Pre-) Commissioning and Start-Up/Operations Team.

The contracting approach for the Santana project should also be revisited. It is recommended that MBAC retain an experienced Engineering, Procurement and Construction Management (EPCM) contractor to insure the project implementation in strict accordance with the design basis documents and execution philosophy. Managing the entire project under this mandate will provide greater assurance to MBAC, its investors and its stakeholders that the Santana

Project will be delivered in strict accordance with the design basis, in the most cost effective manner possible and in accordance with the project schedule.

2 INTRODUCTION

2.1 Scope of Work

Andes Mining Services Ltd. (AMSL), NCL Brasil Ltda. (NCL) and PegasusTSI Inc. (PegasusTSI) have been commissioned by MBAC Fertilizer Corp (MBAC) to prepare a Feasibility Study (FS) for the Santana Phosphate Project, in Pará State, Brazil.

AMSL, NCL and PegasusTSI are collectively referred to in this report as “AMSL/NCL/PegasusTSI”

The PFS has been prepared under the guidelines of Canadian Institute of Mining (CIM) National Instrument 43-101 and accompanying documents Form 43-101F1 Technical Report and Companion Policy 43-101CP (collectively NI43-101).

This FS follows the Pre-Feasibility Study completed by AMSL, NCL and PegasusTSI dated June 7, 2012, as amended August 27, 2012, and titled “Pre-Feasibility Study (PFS) Santana Phosphate Project Pará State, Brazil, As Amended and Restated”, which is abbreviated in this report as (AMSL/NCL/PegasusTSI PFS).

2.2 Forward-Looking Information

This report contains "forward-looking information" within the meaning of applicable Canadian securities legislation. Forward-looking information includes, but is not limited to, statements related to the capital and operating costs of the Santana Phosphate Project, the price assumptions with respect to phosphate materials, production rates, the economic feasibility and development of the Santana Phosphate Project and other activities, events or developments that the MBAC and the authors of this PFS expect or anticipate will or may occur in the future. Forward-looking information is often identified by the use of words such as "plans", "planning", "planned", "expects" or "looking forward", "does not expect", "continues", "scheduled", "estimates", "forecasts", "intends", "potential", "anticipates", "does not anticipate", or "belief", or describes a "goal", or variation of such words and phrases or state that certain actions, events or results "may", "could", "would", "might" or "will" be taken, occur or be achieved.

Forward-looking information is based on a number of factors and assumptions made by the authors and management of MBAC, and considered reasonable at the time such information is made, and forward-looking information involves known and unknown risks, uncertainties and other factors that may cause the actual results, performance or achievements to be materially different from those expressed or implied by the forward-looking information. Such factors include, among others, obtaining all necessary financing, licenses to explore and develop the project; successful definition and confirmation based on further studies and additional exploration work of an economic mineral resource base at the project; as well as those factors disclosed in MBAC’s current Annual Information Form and Management's Discussion and Analysis, as well as other public disclosure documents, available on SEDAR at www.sedar.com.

Although MBAC and the authors of this PFS have attempted to identify important factors that could cause actual actions, events or results to differ materially from those described in forward-looking information, there may be other factors that cause actions, events or results not to be as anticipated, estimated or intended. There can be no assurance that forward-looking information will prove to be accurate. The forward-looking information contained herein are presented for the purposes of assisting investors in understanding MBAC's plan, objectives and goals and may not be appropriate for other purposes. Accordingly, readers should not place undue reliance on forward-looking information. MBAC and the authors of this PFS do not undertake to update any forward-looking information, except in accordance with applicable securities laws.

2.3 Principal Sources of Information

In addition to a site visit undertaken by Mr. Bradley Ackroyd (AMS) to the Santana Phosphate Project between 20th and 23rd of November 2012, and Mr. Robert Alexander (PegasusTSI) between 29th and 31st of October 2012, the authors of this report has relied extensively on information provided by MBAC along with discussions with MBAC technical personnel and prior consultants. The authors believe that their inspections can be considered current as there was no new material scientific or technical information about the property that would have required another personal inspection subsequent to the dates of their most recent inspections.

A full listing of the principal sources of information is included in Section 27 of this report and a summary of the main documents is provided below:

- MBAC (June 2011) - Internal Technical Report on Santana Phosphate Project, Pará State, Brazil.
- MBAC (15th August 2011) - Internal Metallurgy Report on Santana Phosphate Project, Pará State, Brazil.
- Letter dated 20 September 2011 - Magma Servicos de Mineracao Ltda., of Brasilia, Brazil
- Geomma Consulting (2011) - Preliminary Environmental Assessment
- ATS Engineering (2011) - Infrastructure review
- Amazon/NCL PEA
- AMSL/NCL/PegasusTSI PFS
- PegasusTSI Basic Engineering Book for Feasibility Study

AMSL/NCL/PegasusTSI have made enquiries to establish the completeness and authenticity of the information provided and identified. AMSL/NCL/PegasusTSI have taken all appropriate steps in their professional judgement, to ensure that the work, information or advice contained in this report is sound and Amazon/NCL/PegasusTSI do not disclaim any responsibility for this report.

2.4 Qualifications and Experience

The “qualified persons” (as defined in NI 43-101) for this report are Mr. Bradley Ackroyd (AMSL), Mr. Carlos Guzman (NCL) and Mr. Robert Alexander (PegasusTSI).

Mr. Ackroyd is the principal consulting geologist for AMSL with 12 years experience in exploration and mining geology. Mr. Ackroyd is also a Member of the Australian Institute of Geosciences (MAIG) and is responsible for sections 4 to 12, 14; and jointly responsible for sections 1 to 3 and 23 to 27.

Mr. Guzman is the principal consulting mining engineer for NCL with 18 years experience in Mining engineering. Mr. Guzman is a registered Member of the Chilean Mining Commission and is responsible for Sections 15 and 16; and jointly responsible for Sections 1 to 3, 21 and 23 to 27.

Mr. Alexander is a registered Professional Engineer for the State of Florida, USA, and Manager of Engineering for PegasusTSI. Inc. His technical responsibility includes, primarily, the production facilities for ore beneficiation and SSP production and is responsible for Sections 13, 17 to 20, and 22; and jointly responsible for Sections 1 to 3, 21 and 23 to 27.

Neither AMSL/NCL/PegasusTSI nor the authors of this report has or has had previously any material interest in MBAC or related entities or interests. Our relationship with MBAC is solely one of professional association between client and independent consultant. This report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.5 Units of Measurements and Currency

Metric units are used throughout this report unless noted otherwise. Currency is United States dollars ("US$").

2.6 Abbreviations

A full listing of abbreviations used in this report is provided in Table 2.6_1 below.

Table 2.6_1
List of Abbreviations

	Description		Description
$	United States of America dollars	$l/hr/m^2$	litres per hour per square metre
"	Inches	M	Million
µ	Microns	m	Metres
3D	three dimensional	Ma	thousand years
AAS	atomic absorption spectrometry	Mg	Magnesium
Au	Gold	ml	Millilitre
bcm	bank cubic metres	mm	Millimetres
CC	correlation coefficient	Mtpa	million tonnes per annum
cm	Centimetre	N (Y)	Northing
Co	Cobalt	Ni	Nickel
CRM	certified reference material or certified standard	NPV	net present value
Cu	Copper	NQ_2	Size of diamond drill rod/bit/core
CV	coefficient of variation	ºC	degrees centigrade
DDH	diamond drillhole	OK	Ordinary Kriging
DTM	digital terrain model	P80 -75µ	80% passing 75 microns
E (X)	Easting	Pd	Palladium
EDM	electronic distance measuring	ppb	parts per billion
Fe	Iron	ppm	parts per million
G	Gram	psi	pounds per square inch
g/m^3	grams per cubic metre	PVC	poly vinyl chloride
g/t	grams per tonne of gold	QC	quality control
HARD	Half the absolute relative difference	QQ	quantile-quantile
HDPE	High density poly ethylene	RC	reverse circulation
HQ_2	Size of diamond drill rod/bit/core	RL (Z)	reduced level
Hr	Hours	ROM	run of mine
HRD	Half relative difference	RQD	rock quality designation
ICP-AES	inductivity coupled plasma atomic emission spectroscopy	SD	standard deviation
ICP-MS	inductivity coupled plasma mass spectroscopy	SG	Specific gravity
ISO	International Standards Organisation	Si	Silica
kg	Kilogram	SMU	selective mining unit
kg/t	kilogram per tonne	t	Tonnes
km	Kilometres	t/m^3	tonnes per cubic metre
km^2	square kilometres	tpa	tonnes per annum
kW	Kilowatts	UC	Uniform conditioning
kWhr/t	kilowatt hours per tonne	w:o	waste to ore ratio

3 RELIANCE ON OTHER EXPERTS

AMSL/NCL/PegasusTSI has relied on the independent lawyers Magma Servicos de Mineracao Ltda., of Brasilia, Brazil for their opinion on the title for the Santana mineral permits and AMSL/NCL/PegasusTSI have received a memorandum from them dated 15 October 2013, supporting MBAC's claims.

4 PROPERTY DESCRIPTION AND LOCATION

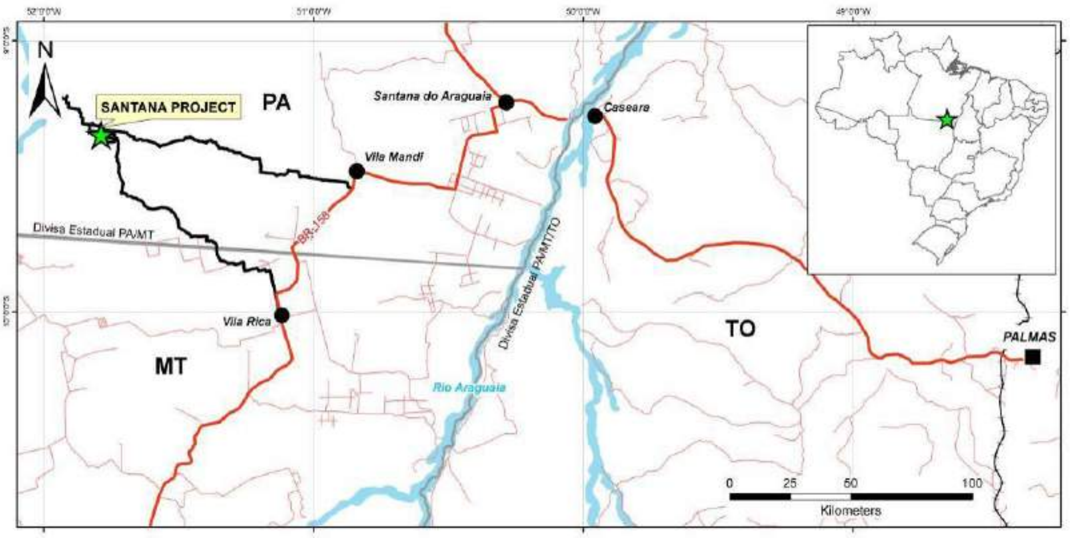
4.1 Project Location

The Santana Phosphate Project is located in the southeast of Pará State near the state border of Mato Grosso and Pará.Santana is located approximately 200km from the cities Santana do Araguaia, (Pará State) and Vila Rica (Mato Grosso State) (Figure 4.1_1).

The topographical coordinates of the project are 417882 East and 8967784 North (Datum SAD69 Zone 22 South).

Palmas city in Tocantins is the most convenient major city with commercial flights and light aircraft from Palmas takes approximately 1 hour to reach Santana.

Figure 4.1_1
Project Location



4.2 Tenement Status

MBAC, through its 100% owned Brazilian subsidiaries Itafos Mineracao Ltda. (Itafos) and MBAC Fertilizantes, is the sole registered and beneficial holder of 21 exploration properties with an additional 6 exploration permits under application for a total of 235,150ha.

Details of MBAC permits in the Santana region are found in Table 4.2_1 and Figure 4.2_1 below.

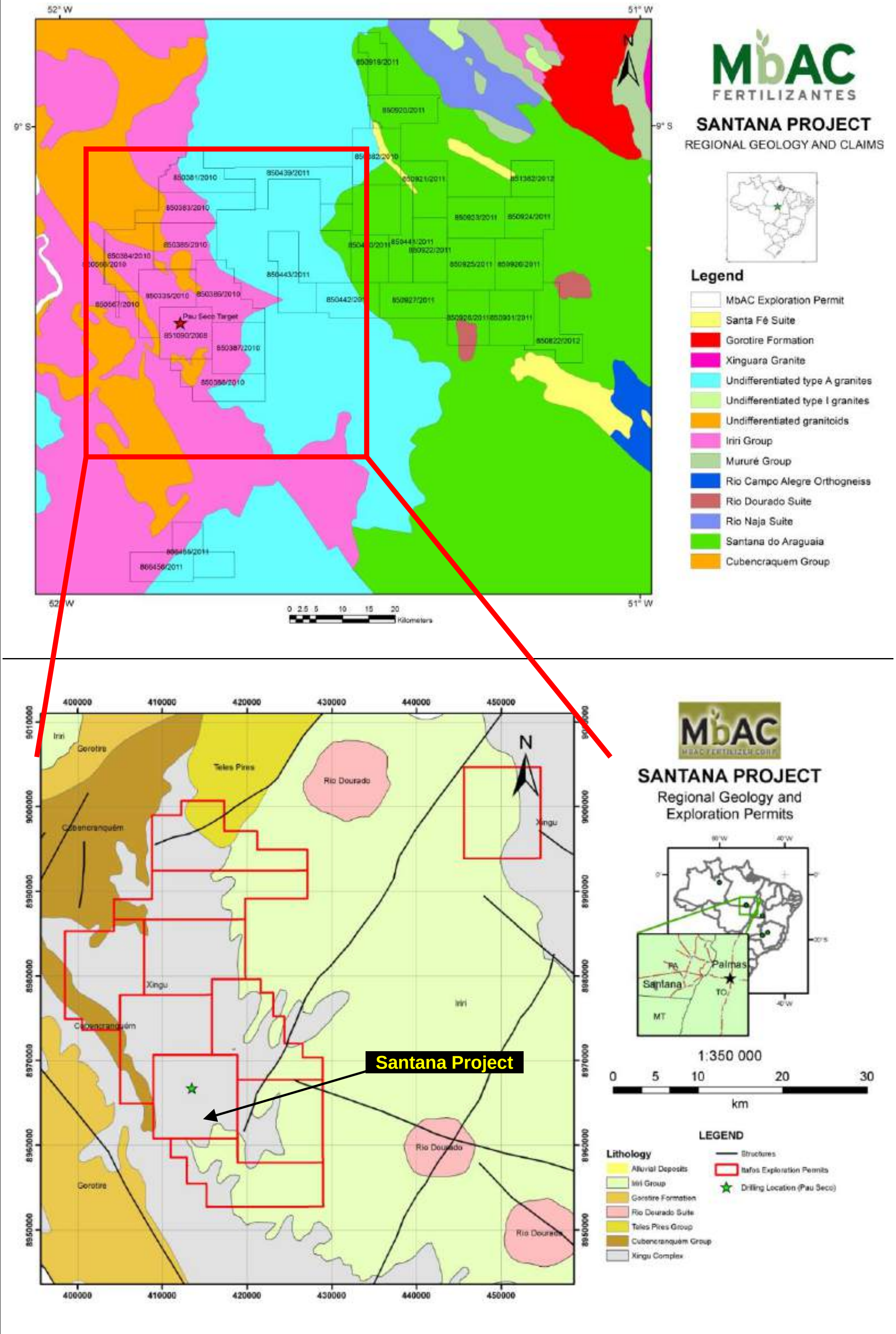
The exploration permits are valid for three years and are renewable for an equal period. MBAC is required to pay $2.36 Brazilian Reais per hectare per year to the Federal Mines Department (DNPM) for permit annual maintenance fee.

Table 4.2_1

Summary of MBAC Permits Status in the Santana Region

Permit Type	Permit No.	Date Granted	Holder	Area (Ha)	Comments
Search Request	850.381/10	Under Application	ITAFÓS	8667.90	Priority
Exploration	850.383/10	15.12.10	MBAC	9850.34	-
Exploration	850.384/10	15.12.10	MBAC	9946.74	-
Exploration	850.385/10	15.12.10	MBAC	9900.64	-
Exploration	850.386/10	15.12.10	ITAFÓS	9995.82	-
Exploration	850.387/10	15.12.10	ITAFÓS	9857.65	-
Exploration	850.388/10	15.12.10	ITAFÓS	9862.03	-
Mining Permit	851.090/08	Under Application	MBAC	9828.91	-
Exploration	850.335/10	Renewal Requested	MBAC	9932.69	-
Exploration	850.439/11	21.06.11	MBAC	9949.54	-
Exploration	850.440/11	21.06.11	MBAC	9820.21	-
Exploration	850.441/11	21.06.11	MBAC	6364.61	-
Exploration	850.442/11	21.06.11	MBAC	7811.24	-
Exploration	850.443/11	21.06.11	ITAFÓS	9116.48	-
Search Request	850.567/10	Under Application	ITAFÓS	2195.97	Priority
Search Request	850.919/11	Under Application	ITAFÓS	4719.28	-
Search Request	850.920/11	Under Application	ITAFÓS	7239.86	-
Exploration	850.921/11	06.07.12	ITAFÓS	8137.99	-
Exploration	850.922/11	04.12.12	ITAFÓS	7978.11	-
Exploration	850.923/11	04.12.12	ITAFÓS	8618.96	-
Search Request	850.924/11	Under Application	ITAFÓS	7033.98	-
Exploration	850.925/11	04.12.12	ITAFÓS	8952.82	-
Exploration	850.926/11	04.12.12	ITAFÓS	8934.87	-
Exploration	850.927/11	04.12.12	ITAFÓS	8711.74	-
Exploration	850.928/11	04.12.12	ITAFÓS	8309.51	-
Exploration	850.931/11	04.12.12	ITAFÓS	8979.09	-
Exploration	850.822/12	05.04.13	MBAC	8799.61	-
Exploration	851.382/12	03.07.13	MBAC	5634.13	-
Total Held / Under Consideration				***235,150.72***	

Figure 4.2_1
Santana Tenement Location Map *(MBAC, 28th October 2013)*



4.3 Royalties and Agreements

Based on information supplied to AMSL/NCL/PegasusTSI by independent lawyers Magma Servicos de Mineracao Ltda., of Brasilia, MBAC has the following royalties and agreements in place:

- Brazilian government royalty for sold phosphate is 2% Net Smelter Return (NSR). This rate is currently under review by the DNPM.
- MBAC has a verbal agreement with three landowners in which MBAC has been granted full access to the properties in exchange for ongoing minor maintenance including fences, gates and repairs to a farm house.
- MBAC has exercised an option agreement with Natanael Rodrigues da Silva (dated 2nd June 2010) to acquire 100% of permit numbers 851090/2008 and 850335/2010. This agreement includes the payment of 1% Net Smelter Return (NSR) per sold phosphate.

Ongoing obligations are:

- Biannual progress reports describing work completed and results obtained.

4.4 Environmental Liabilities

AMSL/NCL/PegasusTSI is unaware of any environmental liabilities to which the Santana Phospate Project is subject.

4.5 Permitting

No additional permits are required at the current stage of exploration.

AMSL/NCL/PegasusTSI is unaware of any other factors risking the development of the exploration works.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Project Access

Access to the Santana Phosphate Project is via approximately 125km of all weather gravel roads and 80km of paved roads from the regional city centres of Santana do Araguaia, in Pará State (northeast of Santana) or via Vila Rica city in Mato Grosso State (southwest of Santana)

The Project is located on a large cattle station called Fazenda Santa Luzia. A nearby cattle station has a well maintained gravel airstrip which MBAC utilise with small airplanes requiring approximately a one hour flight from Palmas city in Tocantins state.

5.2 Physiography and Climate

The climate is tropical with an annual rainfall of around 2,000mm and seasonal variations with a drier period between June and November and a wetter period between December and May. The average annual temperature is approximately 27.5ºC with minimal month to month variations.

Exploration is undertaken at the Project all year round. Drilling activities are intensified during the drier season, from May to November, and continues with adequate equipment during the rainy season. At the peak of the rainy season drilling and mapping are paralysed for two to three weeks, when floods cover the bridges.

The region dominated by igneous massifs separated by alluvial lowlands. The lowlands lie at about 300m ASL, with the tops of the massifs and tablelands at about 480m ASL.

Figure 5.2_1
Panoramic View Looking North over the Santana Project Area (April, 2012)

5.3 Local Infrastructure and Services

Santana is an exploration camp utilising the Fazenda Santa Luzia cattle station basic infrastructure which includes a house and sheds. All electricity is supplied by diesel generators and all consumables are brought in from the nearest major cities located over 200km away.

MBAC has moved the sample preparation and geological base to the village called Vila Mandi, which has electricity and medical services available.

The original forest vegetation across the Santana project area was drastically reduced and substituted by grassland for cattle pasture; as a consequence, the hydrography of the region was seriously affected by the reduction of water streams. Livestock is the economic driver for this region (Pará State), having the highest cattle concentration in Brazil. The Project region is completely occupied by cattle farms with no social infrastructure. The closest village (80 km) is Garimpinho which has only 200 inhabitants and absolutely no social infrastructure. Vila Mandi (141 km) with approximately 1,000 inhabitants has very poor infrastructure. It is the place where the farmers meet for cattle negotiation. Vila Mandi is likely too far from the mine, and cannot be considered as a support for human resources and services.

Based on the above considerations, there is very little availability of human resource with basic technical knowledge. As well no professional schools were identified. As a recommendation, a plan to create manpower for the Project should be considered, utilizing professionals to set up a “training program” for the operation of each specific plant.

The mine and industrial facilities at Santana Project site will utilize approximatedly 420 workers including direct and indirect people. This will require the building of accommodations, including a basic infrastructure to enable a normal living. This housing area will be built in the outer limits of the project site.

The Project will take water for the mine site from Capivara river, which is at approximately 11km southwest from the mine site. In addition, a drilling survey to evaluate the potential for ground water will be undertaken in the near future.

A comprehensive review of local infrastructure and services is included in section 18 of this report.

6 HISTORY

In June 2010, MBAC through its wholly owned Brazilian subsidiary, Itafos, acquired, via an option agreement, two Exploration Licenses numbers 850.335/2010 and 851.090/2008 from Mr. Natanael Rodrigues da Silva. These claims have been transferred to MBAC by the DNPM.

In the same year, MBAC applied for nine Exploration Licenses, of which six were granted by the DNPM, and the remaining three have priority.

Mr. Rodrigues da Silva initially claimed the first area for lateritic nickel, but had identified the area as prospective for phosphate via discussions with local farmers who had returned anomalous phosphate levels in agricultural soil sampling.

Prior to MBAC involvement there had been no material exploration or mining activities undertaken on the project or the region.

6.1 Recent Exploration (MBAC)

MBAC commenced exploration activities across the Santana Project in December 2010 with exploration activities continuing to the present day.

In mid 2011, Amazon Geoservices estimated an inferred mineral resource based upon 53 diamond holes (2,386m) and 6 RC holes (238m) drilled at a spacing of approximately 200m by 200m. Only data received as at 31st July 2011 was used in this estimate. (Preliminary Economic Assessment, Santana Phosphate Project, Pará State, Brazil. Beau Nicholls (Amazon) / Carlos Guzman (NCL), September 2011).

The mineral resource estimate was focused on the main supergene enriched oxide mineralization with 2 flat lying mineralized domains defined using the saprolite and fresh geological boundary along with a 3% P_2O_5 grade cut-off to guide the wireframing process.

An independent inferred mineral resource was estimated comprising 33.5Mt with an average P_2O_5 content of 12.39% (using a preferred 3% P_2O_5 cut-off) (Table 6.1_1 below).

Table 6.1_1
Grade Tonnage Report Santana Phosphate Project
Inverse Distance Weighted Power 2 – 31st July 2011
(Block Model – 12.5mE X 12.5mN X 3mRL)

Lower Cut-off Grade (% P_2O_5)	Tonnes (Mt)	P_2O_5 %	CaO %	Fe_2O_3 %	Al_2O_3 %	SiO_2 %
3	**33.5**	**12.4**	**16.7**	**17.6**	**8.8**	**27.5**
10	17.2	17.6	23.8	15.3	6.4	22.2
20	5.8	24.3	35.1	10.6	4.1	11.8

Appropriate rounding has been applied to the Table 6.1_1

In early 2012, AMSL estimated an indicated and inferred mineral resource based upon 114 diamond holes (5,894.7m) and 274 RC holes (12,936m) drilled at a spacing of approximately 100m by 100m. Only data received as at 8th February 2012 was used for the mineral resource estimate.

The mineral resource estimate completed by AMSL focused on the main supergene enriched oxide mineralization, with 2 flat lying mineralized domains defined using the saprolite and fresh geological boundary along with a 3% P_2O_5 grade cut-off to guide the wireframing process.

An independent resource was estimated comprising an indicated mineral resources of 66.1Mt at 10.5% P_2O_5 and an inferred mineral resource of 21.8Mt at 7.9% P_2O_5 (using a 3% P_2O_5 cut-off) (Table 6.1_2 below). (Mineral Resource Estimate, Santana Phosphate Project, Pará State, Brazil. Beau Nicholls / Ian Dreyer, February 2012).

Table 6.1_2
Grade Tonnage Report Santana Phosphate Project
Ordinary Kriging (OK) Estimate – 29th February 2012
(Block Model – 25mE X 25mN X 4mRL - Cut off 3% P_2O_5 utilised)

Material Type	Indicated		Inferred	
	Tonnes (Mt)	P_2O_5 (%)	Tonnes (Mt)	P_2O_5 (%)
Saprolite	47.4	11.6	8.8	9.8
Fresh Rock	18.6	7.7	13.0	6.7
Total	**66.1**	**10.5**	**21.8**	**7.9**

Appropriate rounding has been applied to the Table 6.1_2

The statement was classified by Qualified Person Ian Dreyer (BSc (Geo) AusIMM (CP)) in accordance with the NI 43-101.

7 GEOLOGICAL SETTING AND MINERALIZATION

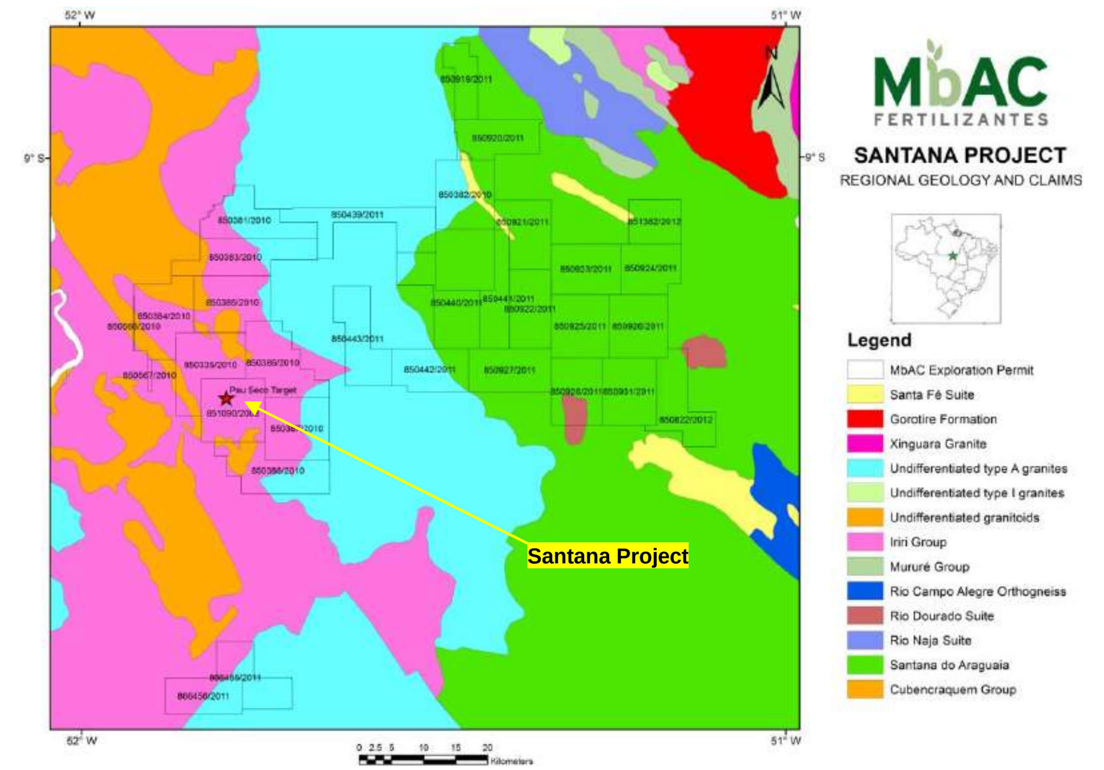
7.1 Regional Geology

The Santana Phosphate Project is located within the Tapajós District (Figure 7.1_1) situated in the south-central portion of the Amazon Craton which became tectonically stable at the end of the Late Proterozoic period. The Craton is generally divided into the Guyana Shield north of the Amazon River and the Brazil Shield south of the Amazon River. The provinces have a northwest trend across the shields. The Brazil Shield has, as its nucleus, the Archaean granitoid - greenstone terranes of the Carajás-Imataca province in the east. The structural provinces become younger towards the west and are dominantly granitic rocks of Paleoproterozoic age. There is a general agreement that in this region, initial oblique collision tectonism was associated with crustal shortening linked to subduction and or accretion of magmatic arcs and early continental nucleation.

The main units that form the basement of the Tapajós Gold Province are the Paleoproterozoic Cuiú-Cuiú Metamorphic Suite (2.0 to 2.4Ga old), and the Jacareacanga Metamorphic Suite, also of possible Paleoproterozoic age (>2.1Ga), regionally mapped as Xingu complex (Figure 7.1_1).

Figure 7.1_1

Regional Geology Map - Santana Phosphate Project ***(MBAC, 28th October 2013)***



The Cuiú-Cuiú Suite comprises gneisses, migmatites, granitoid rocks and amphibolites. The Jacareacanga Suite comprises a supra-crustal sedimentary-volcanic sequence, which has been deformed and metamorphosed to greenschist facies. Both Suites are intruded by granitoids of the Parauari Intrusive Suite consisting of a monzodiorite dated at 1.9 to 2.0Ga.

These form the basement of the extensive felsic to intermediate volcanic rocks of the Iriri Group, dated at 1.87 to 1.89Ga, including co-magmatic and anorogenic plutons of the Maloquinha and Rio Dourado Suite with intrusive events dated at 1.8 to 1.9Ga. The Iriri-Maloquinha igneous event is associated with a strong extensional period.

Regional structural analysis in the Tapajós area has identified important lineaments that trend mainly northwest to southeast with a less well defined transverse east to west set.

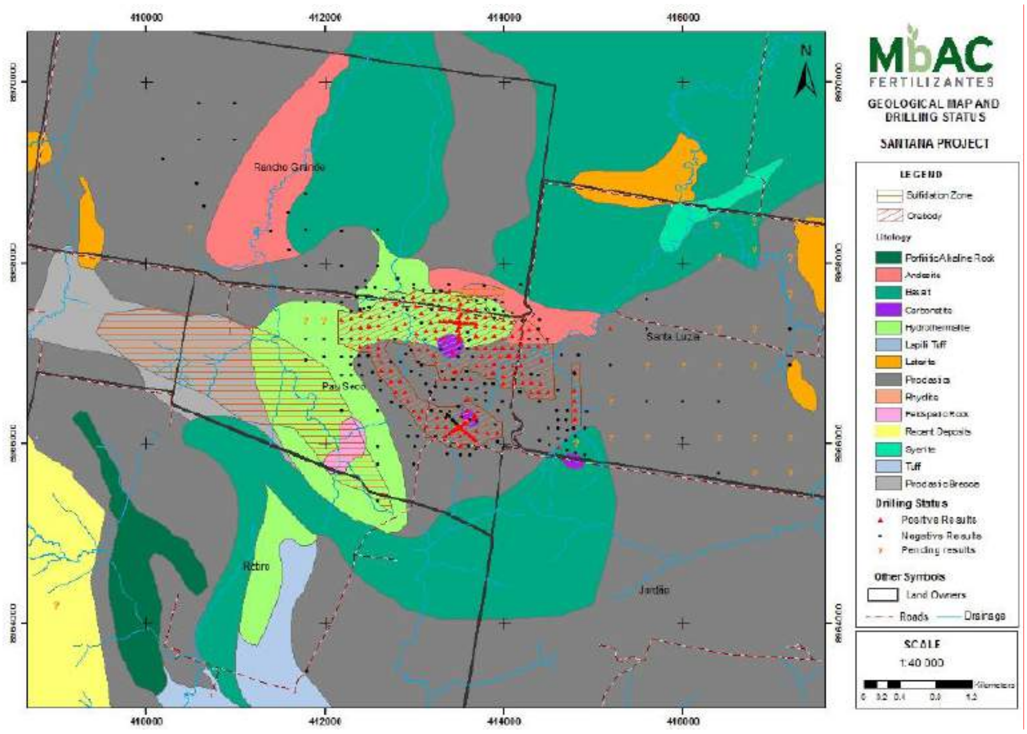
The Santana Phosphate Project represents the first phosphate occurrence in the region.

7.2 Local and Property Geology

The Santana property lies within the domain of hydrothermally-altered volcaniclastic and carbonate rocks, associated with peridotite and calcite carbonatite of post Iriri Group age (Precambrian) (Figure 7.2_1).

The hydrothermally-altered volcaniclastic rock consists of lapilli and crystal tuff, with clasts varying considerably in size from very coarse sand-size clasts to cobble-sized particles (agglomerate, volcanic breccia). The above-mentioned rocks are believed to represent the proximal crater facies of the extrusion of a carbonatite. Abundant sedimentary features such as cross-bedding and well-rounded fragments suggest that a part of the volcaniclastic sequence was deposited under water.

Figure 7.2_1
Project Scale Surface Geology - Santana Phosphate Project *(MBAC, 28th October 2013)*



The rock has undergone weak to very strong carbonatization. Where carbonate has not been the principal hydrothermal process, then the rock may have been silicified (and/or albitized). Less extensive, but important locally are hematitization, chloritization, the introduction of manganese and possibly sericitization. Disseminated pyrite may occur up to an estimated 15%.

Karstification and solution weathering has resulted in the development of dolines (sink holes) that may be seen at surface and expressed as large depressions at least 15m in diameter.

Meteoric weathering of this hydrothermalite led to the formation of a phosphate resource in the saprolite overlying bedrock (Pau-Seco target). Additional large widths of fresh hydrothermally-altered carbonate rock have been intersected in deeper drill holes down to 200m, however, this part of the deposit has not been adequately drilled yet.

Figure 7.2_2
Outcrop Showing Main Hydrothermal Unit (Carbonatite) with Overlying Volcanic Tuff

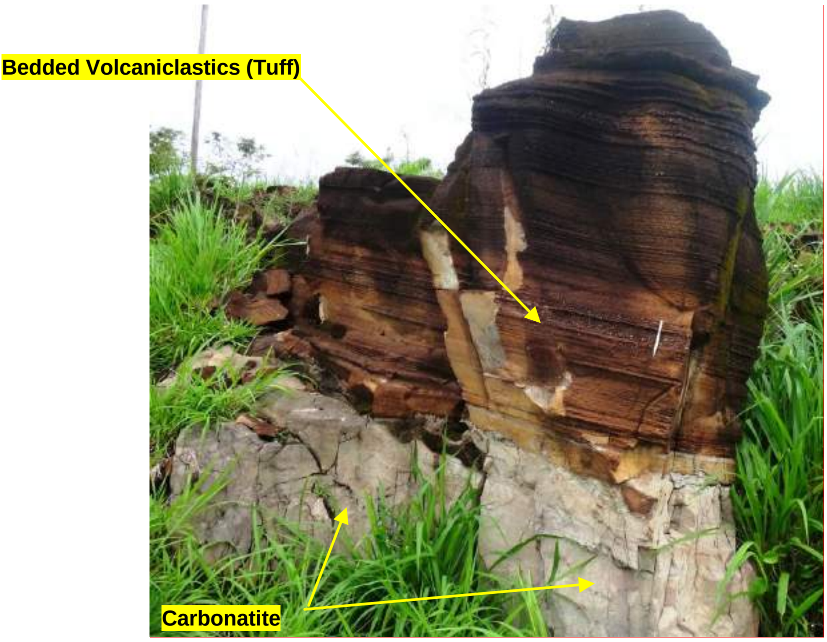


Figure 7.2_3
Diamond Core - Approximately 5m of fresh Hydrothermalite contained within the Saprolite Unit

7.3 Mineralization

The phosphate mineralization occurs in a hydrothermally altered (or more correctly metasomatic) lapilli tuff, consisting mainly of carbonate and/or silica. MBAC refer to this as a hydrothemalite. There is a basalt unit identified in the footwall, but this has not been intercepted in drilling to date.

The highest grade mineralization occurs in the soil and saprolite domains. This residual mineralization was formed by the downward and lateral movement of meteoric waters that have weathered the crystalline carbonate bedrock to material that is often gritty in texture at the base, becoming more argillaceous vertically. However, the gritty passages may occur in any interval where the weathering has been less intense. There are manganiferous units, and units consisting of buff-coloured clay, interpreted as having once been more carbonatic.

The saprolite overlies the bedrock forming essentially tabular bodies, the geometry of which is controlled by the bedrock topography.

The thickness of the saprolite varies considerably from less than a metre overlying bedrock to greater than 75m. Drilling suggests that bodies are thicker near the base of the volcanic massifs becoming shallower outwards (15 to 20 m) towards the central parts of the alluvial plains.

Limited X-ray diffraction studies of 15 core samples from eight of the first 12 drill holes showed the most common phosphate-bearing mineral species in the saprolite to be Hydroxyapatite $Ca_5(PO_4)_3(OH)$ followed by Crandallite $CaAl_2(OH)_6(PO_3(O_{0.5}(OH)_{0.5}))_2$. Hydroxyapatite also occurs in the bedrock. Wavellite $Al_2(PO_4)_2(OH)_3(H_2O)_5$ and Goiazite $SrAl_3(PO_4)_2(OH)_5$ are present in minor amounts and are characteristic of the soil cover.

Electron microprobe analyses revealed other minerals containing phosphorous including apatite, kulanite, gorceixite, childrenite, trolleite, cheralite and monazite.

Figure 7.3_1
Apatite Mineralization along Fracture within Diamond Core

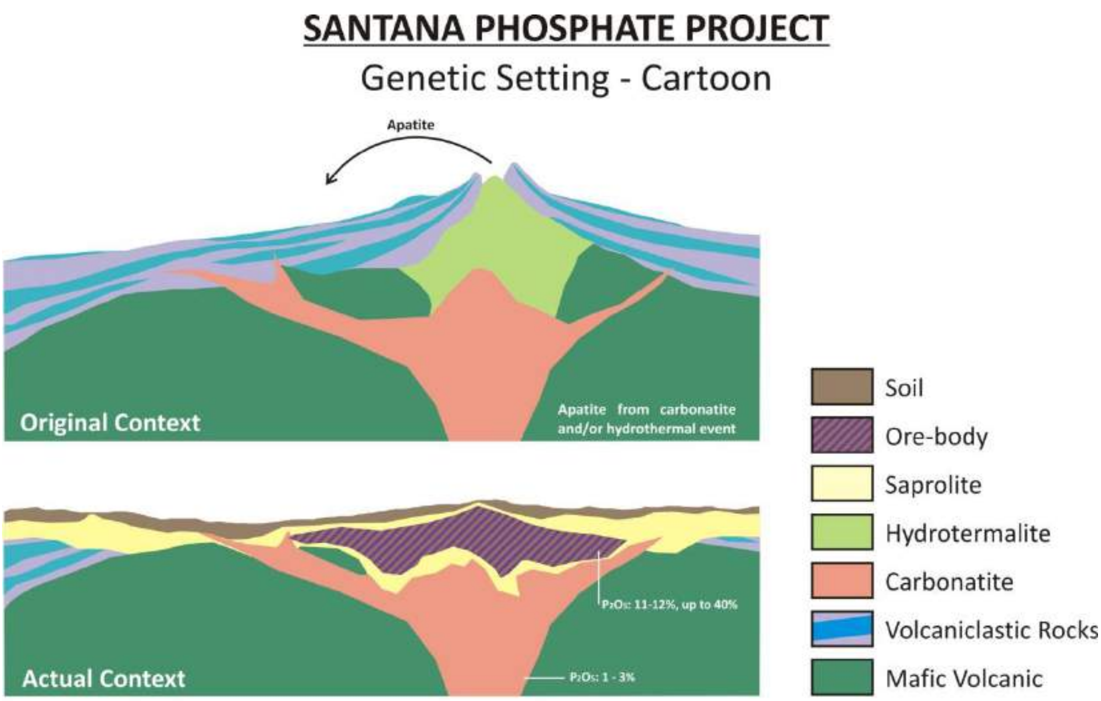
8 DEPOSIT TYPES

The Santana Geological model which is described in more detail in section 14 of this report is a post-tectonic hydrothermal related phosphate deposit with supergene enrichment. P_2O_5 mineralization is believed to be reconcentrated in a meteoric environment, whereby P_2O_5 is released in the oxidising near-surface environment, and then re-precipitated near the water table where conditions become reducing near the REDO boundary.

AMSL is unaware of any similar occurrences of phosphate in Brazil.

The basis of which the exploration program is being planned by MBAC is covered in section 9 of this report.

Figure 8_1
Genetic Setting for Santana Phosphate Deposit



Saprolite: P_2O_5 reconcentrated in meteoric environment. P_2O_5 released in the oxidizing near-surface environment and then reprecipitated near the water table where conditions become reducing near the REDOX boundary. Fe + Mn oxides and hydroxides and associated biogenic agents may participate.

9 EXPLORATION

MBAC commenced exploration across the Santana Project area in December 2010.

Exploration activities to-date have included project scale mapping, 20 line kilometres of ground magnetic surveys, airborne radiometrics survey, a ground penetrating radar test survey along with Auger, Reverse Circulation (RC) and Diamond Core (DC) drilling which is covered in more detail within Section 10 of this report.

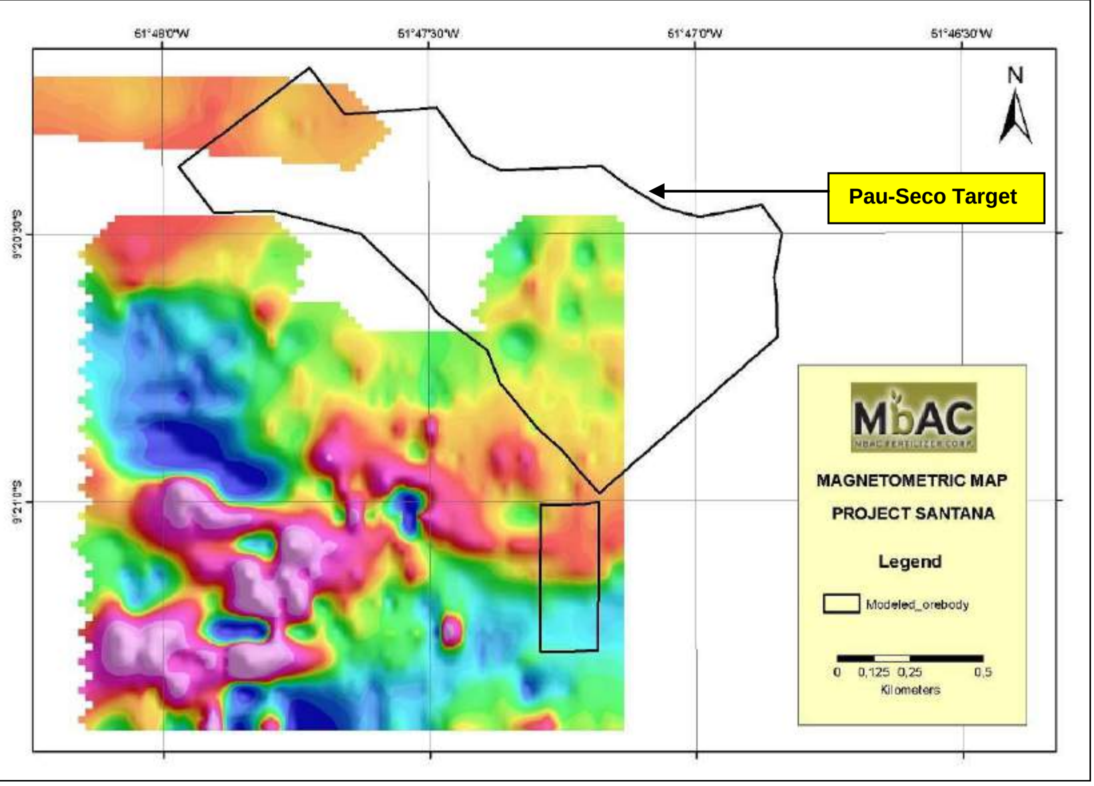
9.1 Ground Magnetic Survey

MBAC undertook a total of 20km of Ground Magnetics at 100m spacing on east-west lines. The survey is displaced to the southwest of the current mineralization.

The E-W direction of the lines is also incorrect at this latitude, due to the proximity of the magnetic equator. Given these two factors, the result of the magnetic survey was considered to be of low quality and did not assist in interpretation or target definition.

The figure below shows the results of the survey relative to the location of the modelled ore body at the Pau-Seco target.

Figure 9.1_1
Ground Magnetic Survey *(Total Magnetic Intensity - TMI)* - Santana Phosphate Project



9.2 Ground Penetrating Radar

A 5km trial ground penetrating radar survey was undertaken by Groundradar Measured Resources (Canada) using the Ultra GPR developed in-house. The aim of the trial was to verify if the equipment was capable of defining the base of saprolite (which represents the higher grade material defined by drilling).

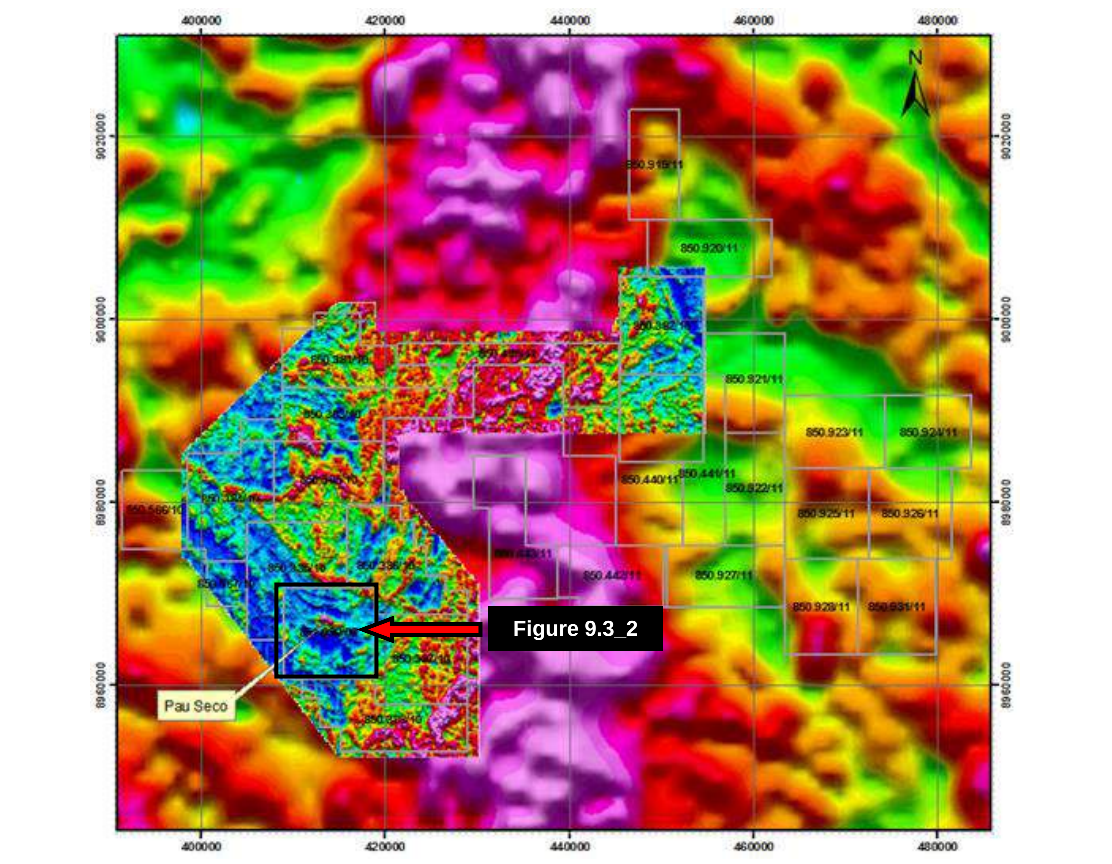
The survey was partially successful in defining this boundary. It correlated well with the drilling where the GPR signal managed to penetrate the saprolite, but there are several gaps where the signal was 100% absorbed before reaching the target . It is believed that this is due to the local compact nature of the clay. The trial survey covers only parts of two lines in the target area.

9.3 Airborne Magnetic and Radiometrics Survey

MBAC completed a detailed airborne magnetics and radiometric survey in November 2012.

Airborne radiometics data was very effective in highlighting the location of the Santana phosphate project as can be seen in Figures 9.3_1 and 9.3_2 below.

Figure 9.3_1
Airborne Radiometrics Survey - Santana Project (November 2012)



Airborne magnetics detailed a number of NE-SW trending structures across the concessions held by MBAC, with extensive folding noted across the northern portion of the Santana project area (Figure 9.3_3).

Figure 9.3_2
Airborne Radiometrics Survey (Pau-Seco Target) - Santana Project (November 2012)

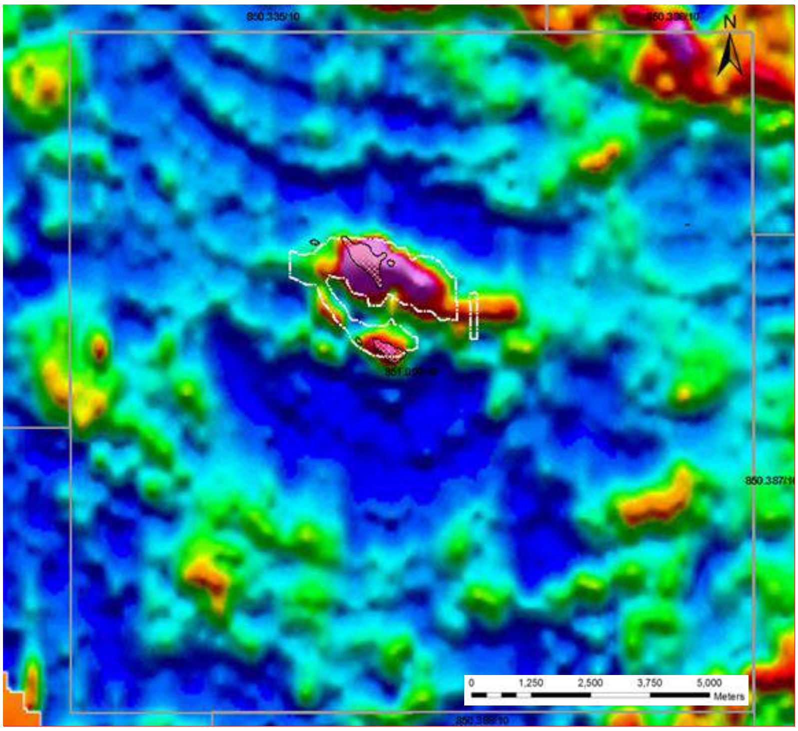
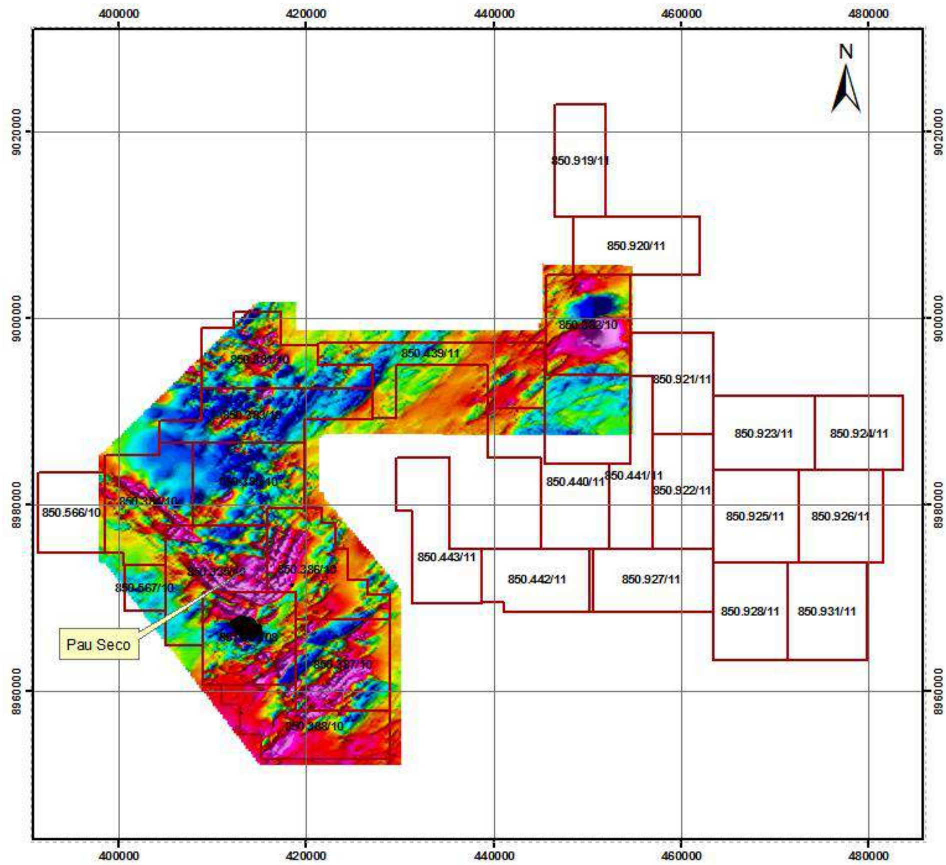


Figure 9.3_3
Airborne Magnetics Survey - Santana Project (November 2012)



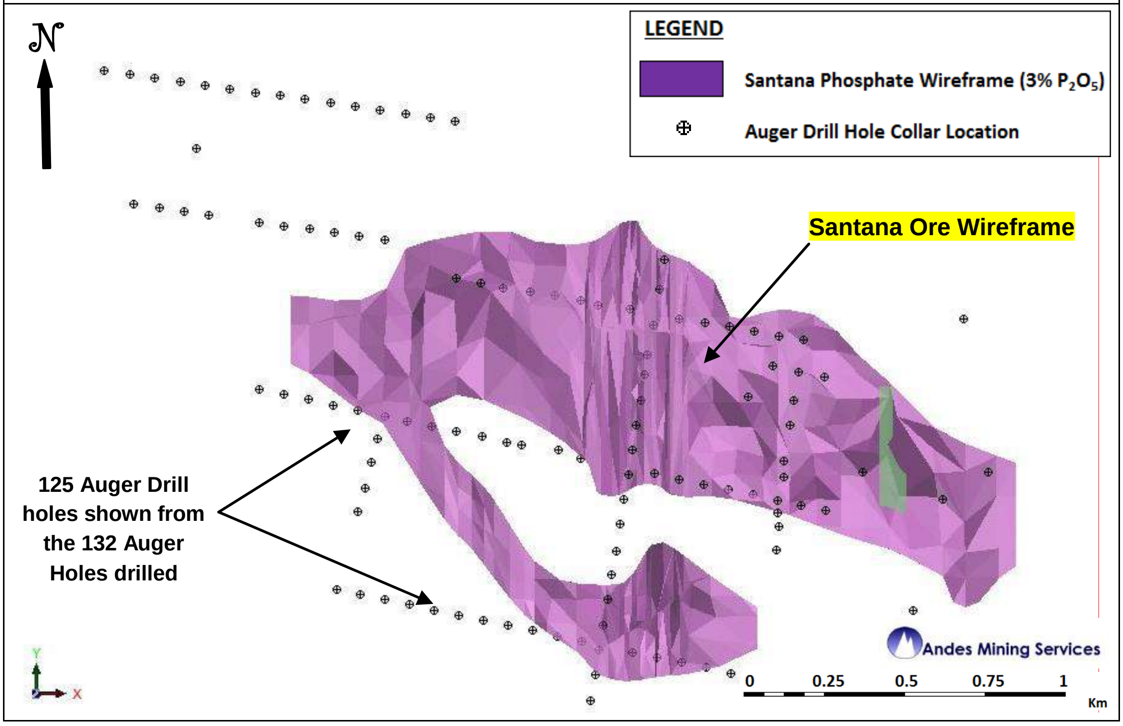
9.4 Auger Drilling

MBAC has completed a total of 132 Auger holes for 1051.6m (Figure 9.4_1). Vertical holes were completed on a 50m by 700m grid utilising a "trado" Auger unit which takes a 30 – 50cm sample which is then recovered by pulling out all the rods. Minor contamination from wall material falling to the base of the hole occurs using this method of drilling. Holes were drilled between 5 and 10m in depth.

The Auger drilling was effective in defining a regionally anomalous zone which was then utilised for planning follow-up RC and DC drilling.

RC and DC drilling has been covered in Section 10 of this report.

Figure 9.4_1
MBAC Auger Drilling across the Santana Phosphate Project (5th February 2013)



9.5 Bulk Density Determinations

MBAC have taken a total of 2,978 bulk density determinations from weathered and fresh diamond core samples as part of the recent drilling campaigns completed across the Santana Project area (Figures 9.5_1 and 9.5_2).

Of the 2978 bulk density measurements collected, a total of 1335 'humid / wet' bulk density measurement were made which have been disregarded by AMSL as part of the total dataset. These humid / wet bulk density measurements do not give an accurate reflection of the true bulk density for the mineralized material and have been collect in order to determine the moisture content of material for mining / feasibility studies.

AMSL have only considered samples which were initially dried in a small oven before bulk density measurements were made. A total of 1,643 samples were available for consideration.

Figure 9.5_1 **Fresh Core Prepared for Bulk Density Testwork**	**Figure 9.5_2** **Bulk Density Measurements (Procedure)**
	

Of the 1,643 samples, AMSL have only considered a total of 702 samples based on the following criteria;

- Samples less than 10cm in length deleted from database;
- Outliers with a bulk density value less than 1.25 disregarded;
- Only 'dry' samples with dry weight and dry immersion weight have been considered.

Bulk density measurements (used by AMSL for the mineral resource estimation) were undertaken by MBAC technicians using the following procedure;

- >10cm full core is wrapped in plastic film on the drill rig;
- Sample is weighed wet and then dried in a small oven;
- Dry core sample is weighed on electronic scale to determine mass of dry core and then weighed immersed in water to determine the volume (Archimedes principle);
- Both wet and dry bulk densities are then determined.

The dry bulk densities used in the mineral resource estimation are summarized in Table 9.5_1 below:

Table 9.5_1
Dry Bulk Density Measurements for Santana Phosphate Project

Material Type	Number of Samples	Density (g/cm^3)
Sand / Soil	35	1.50
Saprolite	605	1.64
Fresh	62	2.69

Samples collected for bulk density measurements are consider by AMSL to be representative of the various geological boundaries defined for the Santana Project, with no sampling bias evident. AMSL note a very sharp boundary between saprolite and fresh rock material across the Santana project area (Figure 9.5_3).

Figure 9.5_3

Sharp Boundary between Saprolite and Fresh Rock (Significant Density Difference)

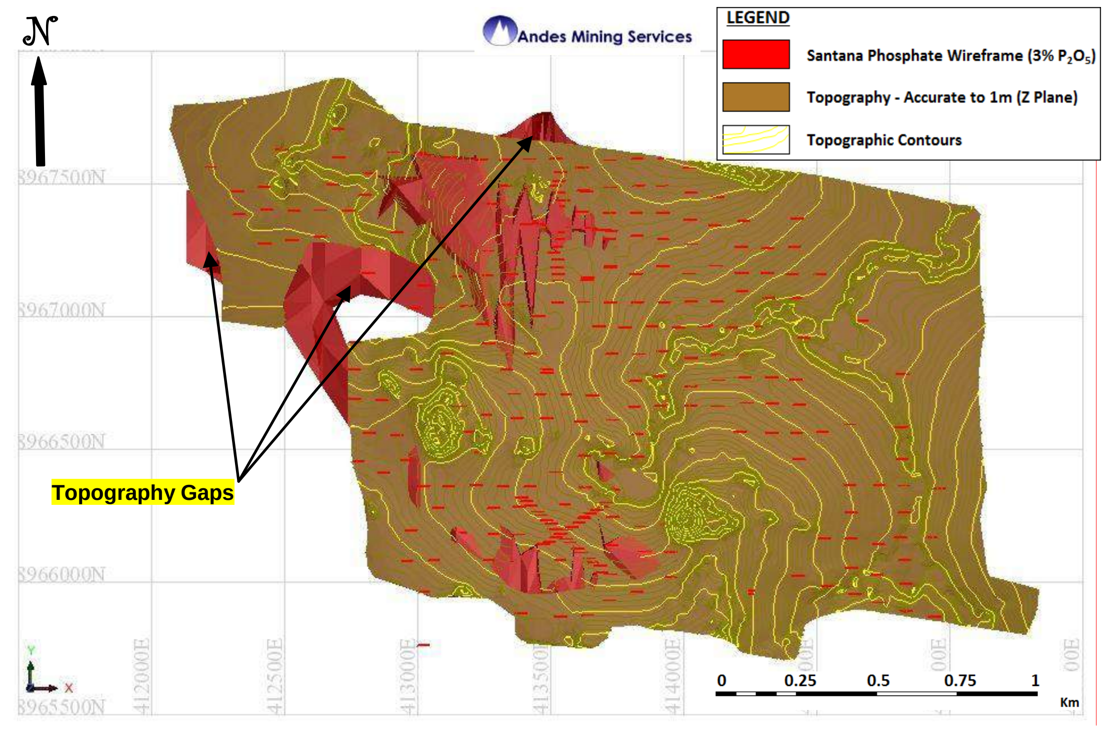
9.6 Topographical Survey

A moderately detailed topographic survey has been completed by MBAC across the Santana Project area (using drill hole collar locations) which is accurate to within 1m through use of a base / total station and survey equipment (Figure 9.6_1).

AMSL note that the current topographic surface does not cover the entire area of phosphate mineralization as illustrated in Figure 9.6_1 below.

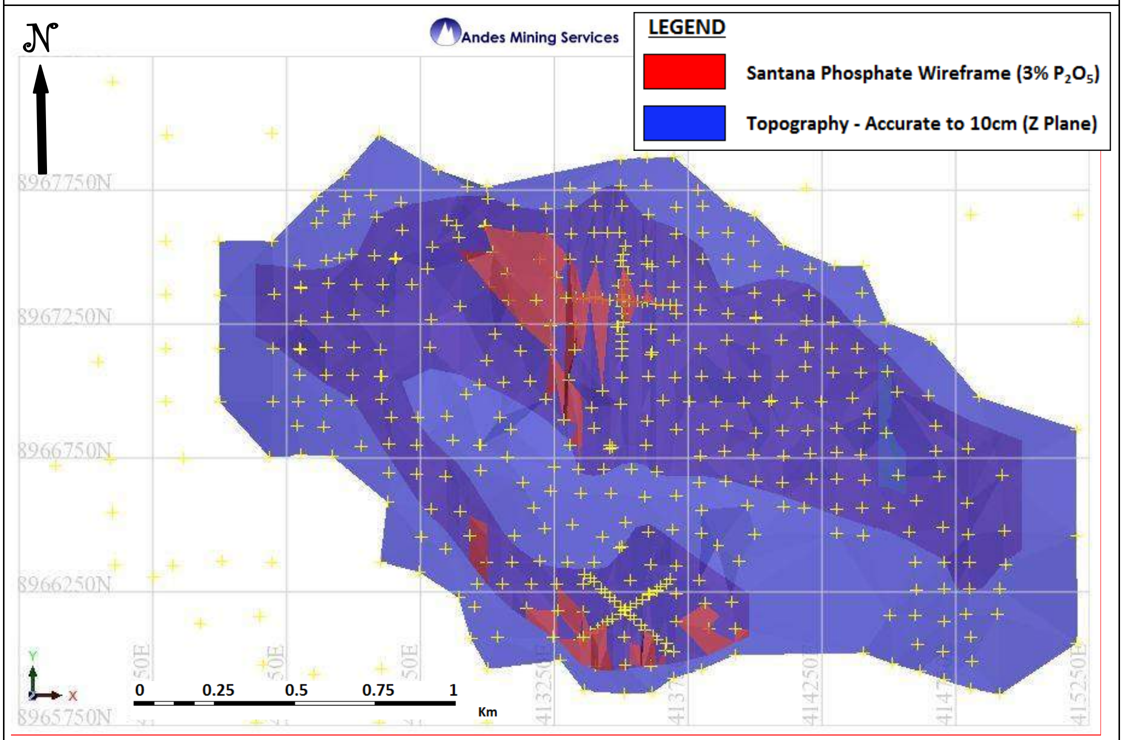
AMSL have subsequently generated a new topographic surface based on a recent detailed survey of drill hole collars (accurate to within 10cm vertically on the Z plane, and 2-3cm along the X and Y planes).

Figure 9.6_1
Historical Topographic Survey - Santana Phosphate Project



A final topographic surface was generated for the Santana Project area (Figure 9.6_2) based on drill hole collar locations which have been utilized to generate a surface DTM in Surpac, and subsequently utilized as an upper bounding surface for the wireframe generation.

Figure 9.6_2
Detailed Topographic Survey - Santana Phosphate Project (28th October 2013)



A detailed airborne topographic survey was due to be flown by MBAC in early 2013, however delays in having the survey flown meant that AMSL were unable to utilize this topographic survey for the mineral resource estimation.

10 DRILLING

MBAC have undertaken substantial programs of auger, reverse circulation (RC) and diamond drilling (DC) across the Santana Phosphate Project, as summarized below in Table 10_1.

Table 10_1
MBAC Drilling Summary - Santana Phosphate Project

Drilling Technique	Company	Number of Drillholes	Metres Drilled
Auger	MBAC	132	1,051.60
Reverse Circulation	Servitec Sondagens	277	13,108
Diamond Core	Geosonda Sondagens	12	883.64
Diamond Core	Servitec Sondagens	302	17,493.27

All drilling has been undertaken and/or supervised by MBAC technical personnel.

RC and DC drilling and data collection methods applied by MBAC have been reviewed by AMSL during the site visits.

Drilling included within the Santana Phosphate Project resource is listed below in Table 10_2 and illustrated in Figures 10_1 and 10_2.

Table 10_2
Santana Phosphate Project Resource - Drilling Summary Statistics

Year	Drilling Technique	Summary
2010	Auger	97 Holes (834.8m Total)
2011	Auger	35 Holes (216.8m Total)
	RC	277 Holes (13,108m Total)
	DDH	116 Holes (6,431.49m Total)
2012	DDH	198 Holes (11,945.42m)

Figure 10_1

Plan View of Mineralized Domain Outline with Drillholes by Year *(AMSL, 28th October 2013)*

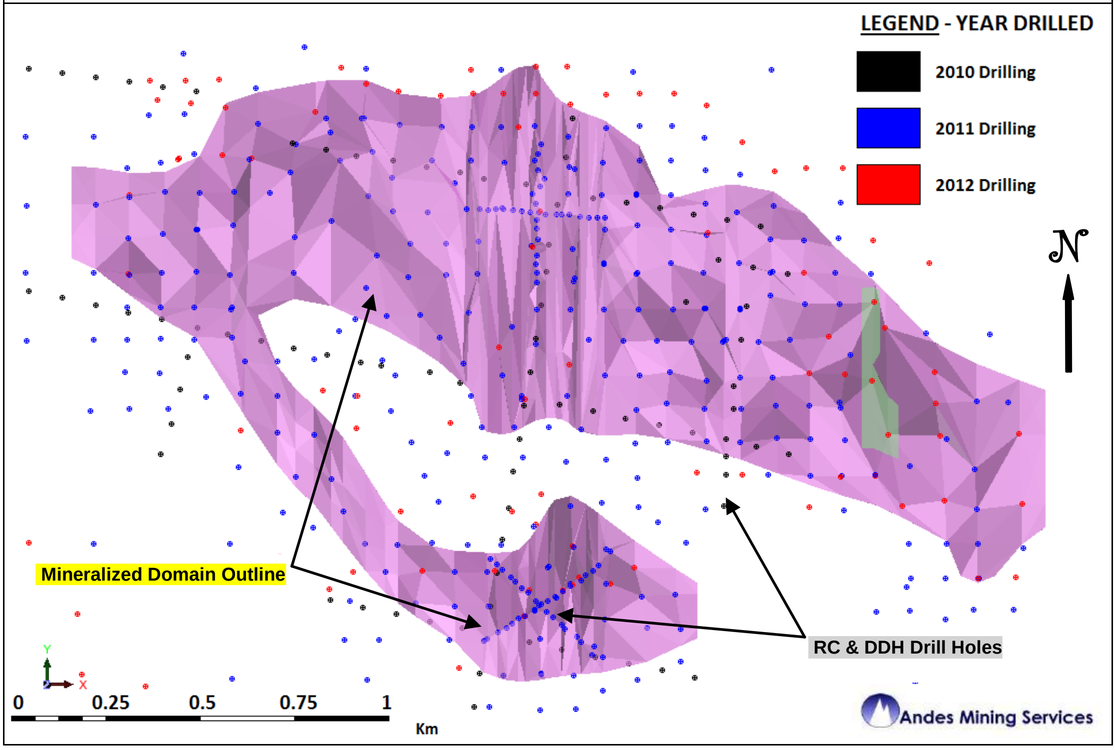
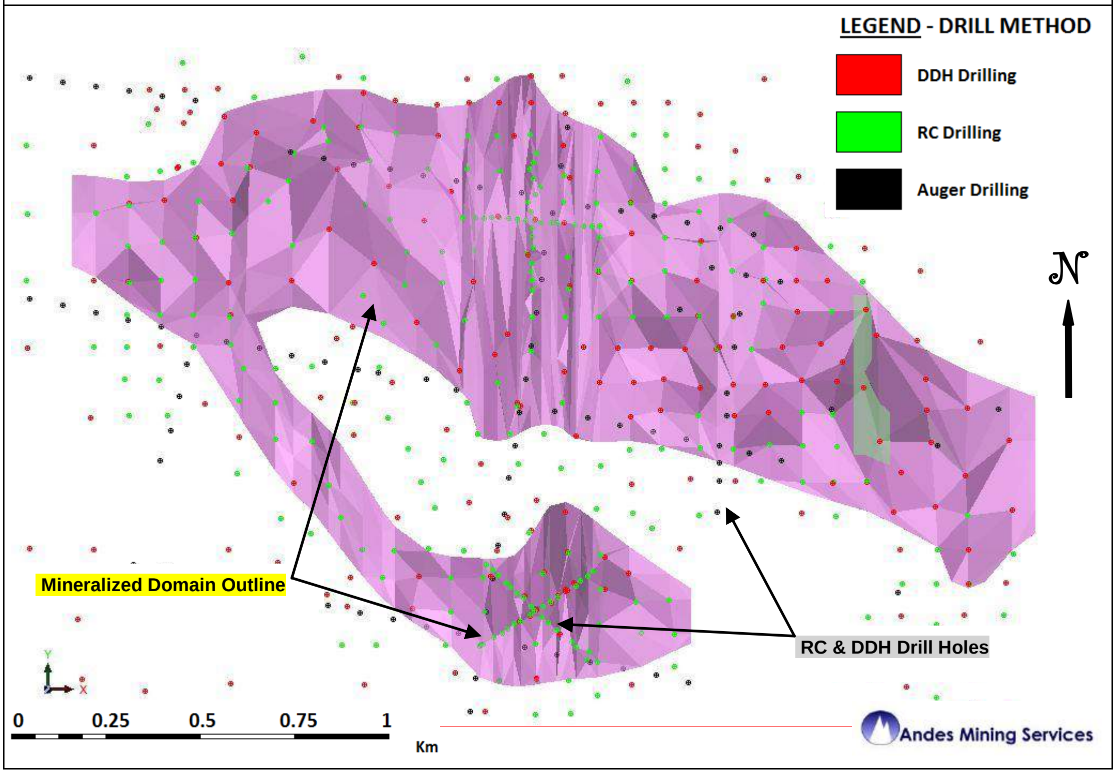


Figure 10_2

Plan View of Mineralized Domain Outline with Drillholes by Type *(AMSL, 28th October 2013)*



10.1 Auger Drilling

MBAC completed auger drilling in September 2011. A total of 132 auger holes were drilled for a combined length of 1,051.60m. Holes were drilled vertical between 1 and 16m in depth, with an average depth of 7.97m. Assaying was carried out by ALS laboratory, which is independent of MBAC.

It is not uncommon for this method of drilling to have minor contamination in the recovered samples from wall material falling to the base of the hole but is an effective tool in weathered and dry material.

Auger results were not used as part of the current mineral resource estimate; however results were used to guide the wireframing process for the current mineral resource modelling. Initial Auger Drilling provided a vector for the reverse circulation and diamond drilling programs recently undertaken by MBAC in 2011 / 2012.

10.2 Reverse Circulation (RC) Drilling

Drilling was contracted to Servitec Sondagens. RC Drilling equipment used on the project included a Explorak R50 RC rig and a 5.5 inch face sampling hammer.

Holes were drilled vertical and have not been surveyed downhole. AMSL does not consider that these holes would deviate enough to make a material difference.

Observations:

- Samples taken and weighed on meter by meter basis;
- Cyclone is cleaned on a rod by rod basis;
- Samples split to around 3kg via a single tier splitter;
- Logging of alteration, lithology and weathering;
- Hole collar coordinates picked up utilising a hand held GPS (accuracy +/- 10m).

10.2.1 Reverse Circulation Drilling Results and Quality

During the initial site visit in 2011, AMSL noted the drilling of hole SAN-RC-0050. The hole was in dry material at 50m but the intervals 10 to 25m were saturated.

On further inspection of the bag farm it was noted that a large number of RC samples were saturated (Figures 10.2.1_1 and 10.2.1_2). A review of the sampling sheets shows that approximately 25% of all samples are saturated.

AMSL considers that wet RC samples could be a material issue and that MBAC should search for a solution that allows the RC drilling to produce dry material. This was recommended in the Amazon/NCL PEA.

Figure 10.2.1_1
Wet RC Drilling - Santana Phosphate Project (2011)

Figure 10.2.1_2
Wet RC Samples at Bag Farm - Santana Phosphate Project (2011)

This could be a material issue as wet RC samples in the saprolitic material can ultimately create a sample bias. AMSL has seen examples of phosphate being washed out of the sample (such as apatite mineralisation along fractures) which ultimately can result in underestimation of the phosphate grade.

A total of 7 twin holes (RC and DC holes within 5 meters) have been undertaken to allow a comparative analysis of the results to determine the precision of the RC versus the DC, with results presented Section 11 of this report.

AMSL considers the wet RC samples should be avoided in RC drilling as this causes both down hole smearing and also washing. It was recommended at the time that MBAC should undertake additional dry RC twin holes next to wet RC holes to determine the precision and any potential wet RC bias.

In 2012, MBAC decided to no longer utilize RC drilling for resource definition drilling across the Santana Phosphate Project, and have instead opted to use diamond drilling for all further resource definition drilling programs.

10.3 Diamond Core (DC) Drilling

Drilling was initially conducted by Geosonda who utilised a Chinese rotary drill called Drill XY-4. Productivity was poor after less than 900m drilled, so MBAC changed contractors to Servitec Sondagens (Figure 10.3_1). Servitec utilised a Boart Longyear DB-525 and a Maquesonda FS-320. Both Geosonda and Servitec drilling is dominantly HQ sized core with minor NQ sized core utilised on holes greater than 100m in depth and HW utilised to collar some holes.

Figure 10.3_1
Servitec Sondagens Drill Rig - Santana Project (April 2012)

Holes were drilled vertical and have not been surveyed downhole. AMSL does not consider that these holes would deviate enough to make a material difference. Core has not been oriented as all holes are vertical.

Observations:

- Storage of all core in wooden core boxes at drill site and then transported to the base for logging and sampling;
- Run markers with metal tags indicating drilled depth and recovery;
- Measurement and recording of core recovery for each drilling run;
- Photography and detailed logging of core before splitting;
- Detailed logging of alteration, lithology, structures and sulphides;
- Hole collar picked up utilising a hand held GPS (accuracy +/- 10m).

10.3.1 Diamond Drilling Results and Quality

From observations of drilling in progress and drillholes reviewed during the site visits, AMSL noted that MBAC DC procedures are of high quality with >90% recovery returned in both saprolite and fresh material (Figures 10.3.1_1 to 10.3.1_4).

AMSL considers the DC drilling procedures to be of an acceptable industry standard.

Figure 10.3.1_1
Core Storage at Rig - SAN-DD-208

Figure 10.3.1_2
Core Storage at Base Camp

Figure 10.3.1_3 Field Inspection of Drill Collar (SAN-DD-164)	Figure 10.3.1_4 Drill Core – Sharp Saprolite to Fresh Boundary
	

10.4 Drilling Results

Significant drill results have not been individually reported, but they are summarised in the mineral resource section of this report.

Drilling was orientated to enable perpendicular intercepts of the main trend of mineralisation but once again the three dimensional modelling has ensured that this is accounted for.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

A summary of the current drilling completed by MBAC along with laboratories utilised for each phase of drilling is shown in Table 11_1 below.

MBAC is now utilizing a combination of lithium tetraborate fusion followed by ICP-MS and XRF for the most recent diamond drilling program completed in 2012.

Table 11_1
Laboratories Used in Analysing MBAC Drilling

Year	Company Name	Type of Drilling	Number of Holes	Meters Drilled	Lab Used
2010	MBAC	Auger	97	832.3	ALS
2011	MBAC	Auger	35	219.3	ALS
2011	Servitec	RC	277	13,549	ALS
2011	Geosonda	DC	12	862.06	ALS
2011	Servitec	DC	102	5,406.83	ALS
2012	Servitec	DC	4	223.90	ALS
2012	Servitec	DC	196	11,879.12	SGS Geosol

11.1 MBAC Sampling Method

11.1.1 Reverse Circulation

MBAC geologists supervised all RC sampling undertaken. RC samples were taken on 1m intervals. MBAC initially utilised a single tier riffle splitter of poor design and passed the dry sample approximately 3 times to get a 3kg sample.

MBAC purchased a 3 tier riffle splitter (following recommendations from Amazon Geoservices in a previous site visit). This has reduced the work load and reduces potential error.

Figure 11.1.1_1
Single Tier Splitter Previously Used

Figure 11.1.1_2
Typical 3 Tier Splitter Recommended

11.1.2 Diamond Drill Core

MBAC geologists supervised all core sampling undertaken. Core samples were normally taken on 1m intervals with some taken between 0.75m and 1.25m intervals based on the geological logging. Core is split in half using a blade in the weathered material and via a diamond saw in the fresh material (Figures 11.1.2_1 and 11.1.2_2).

Figure 11.1.2_1 **Core Saw Utilized for Collecting 1/2 Core Samples**	**Figure 11.1.2_2** **Half Core Examples via Core Saw and Blade**
	

The ½ core is bagged and sent for preparation and the remaining ½ core is returned to the core box and a ply wood lid is nailed on and the box is stored for future reference.

AMSL recommends that the practice of irregular sample intervals should not be continued. The mineralization in Santana is often subtle with no clearly defined visible geological controls and as such a regular 1m sample interval for all the drillcore is recommended.

11.2 Sample Security

Core is currently transported directly from the Santana Project to Vila Mandi (town with core preparation and storage base). After logging, core samples are marked for splitting and sampling by MBAC geologists. Each core sample is placed in a plastic bag which in turn is placed in a nylon bag for transporting via truck to the ALS sample preparation laboratory located in Campos Belos, or to the SGS preparation laboratory in Paráuapebas, Pará State. Vila Mandi processing facilities and storage base is approximately 110km by road from the Santana Project area (Figures 11.2_1 and 11.2_2).

AMSL considers the core sampling security to meet current industry best practice.

Figure 11.2_1 Diamond Drill Core Samples Ready for Despatch	Figure 11.2_2 Diamond Drill Core Storage - Vila Mandi
	

11.3 ALS Laboratory Sample Preparation and Analysis

Initial sample preparation was performed by ALS Minerals Ltd. (ALS) Sample Preparation Laboratory located in Campos Belos, Goias State, Brazil. In late 2011, MBAC changed laboratory to SGS-Geosol Ltda. (SGS), which prepares the sample at Paráuapebas in Pará state, Brazil, before sending sample pulps to Belo Horizonte for analysis. Sample preparation and analysis procedures are the same for both Laboratories:

- Drying and weighing of whole sample;
- Crushing of sample to -2mm;
- Sample homogenization and splitting to a 1kg sub-sample;
- Pulverization to 95% passing -150 mesh;
- Splitting of pulverized material to 50 gram pulp.

Sample pulps are analysed for phosphate using a lithium tetraborate fusion followed by XRF analysis (0.01% detection limit).

The ALS and SGS analytical procedures are ISO 9001 certified and are in accordance with ISO/IEC 17025. Both ALS and SGS are independent of MBAC.

11.4 Adequacy of Procedures

The sampling methods, chain of custody procedures, and analytical techniques are all considered appropriate and are compatible with accepted industry standards.

12 DATA VERIFICATION

12.1 Geological Database

MBAC provided AMSL with an excel database, complete with collar, survey, geology and assay information. AMSL have validated the database using the Surpac Database Audit tool, with no material inconsistencies noted. In addition, AMSL have made a manual check of the database, and any minor inconsistencies noted were promptly rectified by MBAC personnel.

The following checks were performed;

- Holes that had no collar data;
- Overlaps in sample intervals;
- Gaps in sample intervals;
- Matching the geological logging length to the drill hole sample length.

There were no material errors noted within the database as a whole, however more care should be taken in future to remove overlapping geological and sample intervals within the database which have now been corrected.

The excel database was converted into an Access format database which is compatible with Surpac software, and allows key relationship based changes / modifications to be easily made (for example – application of average density grades across geological boundaries).

Hardcopy assay data from SGS and ALS was made available to AMSL, and a comparison of these results with the data supplied in the MBAC database was completed as part of the validation checks. AMSL checked a total of 10% of the MBAC drillholes for validation purposes. No material errors were identified with the original log and the digital database.

12.2 QA/QC

MBAC has set in place a Quality Assurance and Quality Control (QA/QC) programme that included the submission of blanks, standards, field duplicates and umpire assays.

MBAC undertakes quality control on approximately 10% of the total samples prepared. This includes three blanks, three duplicates and three certified standards for every 90 rock samples submitted. The first QAQC sample in each batch is a blank to check that the system was clean.

12.2.1 Certified Standards and Blanks

MBAC utilized a total of 3 standards for ALS laboratory sample submissions (these standards have been prepared and certified by SGS) and one blank material (locally sourced and not certified).

The results for ALS have been analysed by AMSL and are summarised in Tables 12.2.1_1 and Figures 12.2.1_1 to 12.2.1_4 below.

Generally the results show excellent precision for ALS blanks and standards submitted.

Table 12.2.1_1 Standards and Blank Submitted by MBAC to ALS								
Standard Name	**Expected Value (EV) (%)**	**+/- 2SD (EV) (%)**	**Date Range**	**No of Analyses**	**Minimum (%)**	**Maximum (%)**	**Mean (%)**	**% Within +/- 2 SD of EV**
MBAC Submitted Blanks								
Blank	0.01	0.009 to 0.011	Jan 2011 to Jan 2012	654	0.005	1.96	0.03	95
SGS Certified Standards								
PFA	12.99	12.69 to 13.28	Apr 2011 to March 2012	39	12.9	13.2	13.05	100
PFB	2.49	2.44 to 2.54	Apr 2011 to March 2012	100	2.48	2.65	2.53	**99**
PFM	11.09	10.84 to 11.33	Apr 2011 to March 2012	379	10.8	11.65	11.2	98

The blank material was sourced by MBAC from un-mineralized granite material. There are results for blanks up to 1.96% P_2O_5. The causes for the low level values returned from the blank material are potentially related to the actual un-certified granite material utilised containing low levels of phosphate (Figure 12.2.1_1).

Figure 12.2.1_1
Blank Submissions - ALS

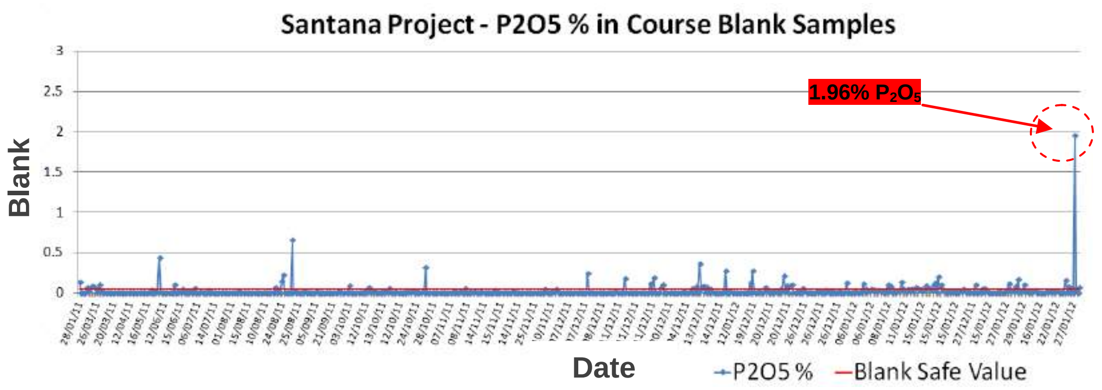


Figure 12.2.1_2
Standard PFA - ALS

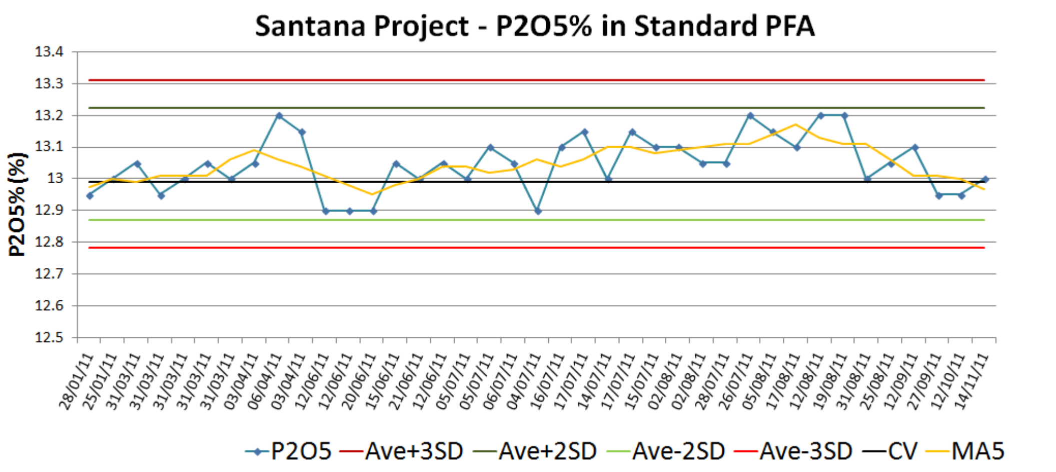


Figure 12.2.1_3
Standard PFB - ALS

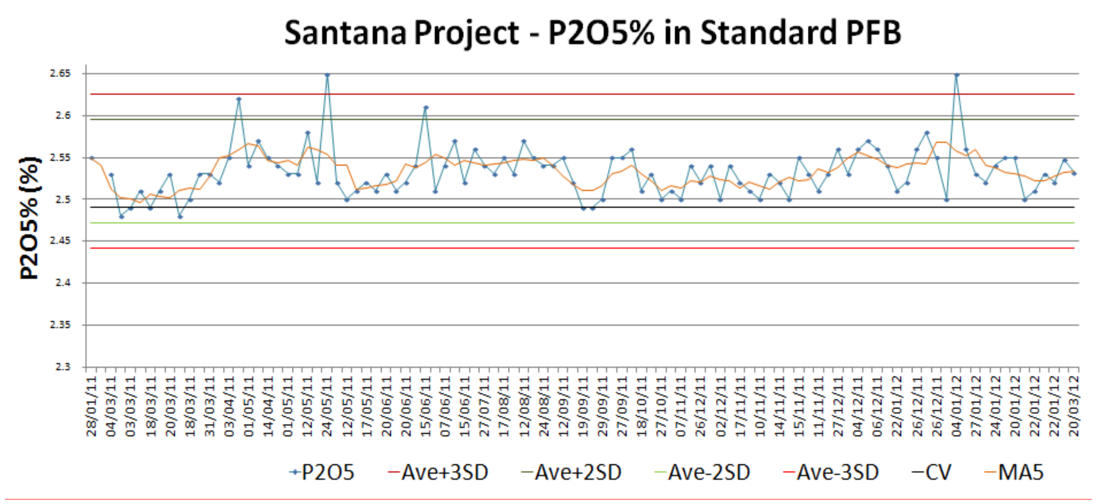
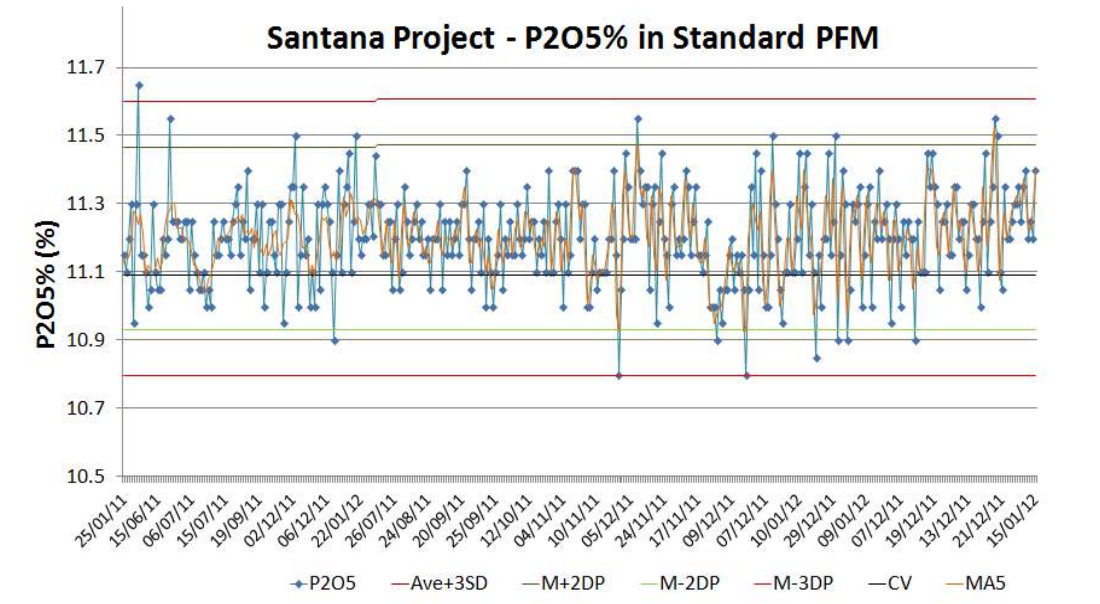


Figure 12.2.1_4
Standard PFM - ALS



Standards submitted to ALS have shown excellent precision.

MBAC utilized a total of 6 standards for SGS laboratory sample submissions (three of these standards have been prepared and certified by SGS) and one blank material (locally sourced and not certified).

The results for SGS have been analysed by AMSL and are summarised in Tables 12.2.1_2 and Figures 12.2.1_5 to 12.2.1_10 below. The graph for GRE-03 has not been shown given the limited number (3) of standards analysed.

Generally the results show excellent precision for SGS blanks and standards submitted.

Table 12.2.1_2 Standards and Blanks Submitted by MBAC to SGS								
Standard Name	**Expected Value (EV) (%)**	**+/- 2SD (EV) (%)**	**Date Range**	**No of Analyses**	**Minimum (%)**	**Maximum (%)**	**Mean (%)**	**% Within +/- 2 SD of EV**
MBAC Submitted Blanks								
Blank	0	0.05	Jan 2012 to Sep 2012	683	0.005	0.784	0.02	97
SGS Certified Standards								
PFA	12.99	12.69 to 13.28	Jul 2012 to Sep 2012	10	12.74	13.19	13.04	100
PFB	2.49	2.44 to 2.54	Jan 2012 to Sep 2012	141	2.42	2.65	2.52	**99**
PFM	11.09	10.84 to 11.33	Jan 2011 to Sep 2012	223	10.81	11.64	11.15	100
GPO-11	9.72	9.52 to 9.92	Jun 2012 to Oct 2012	15	9.54	9.92	9.72	100
GPO-13	4.94	4.77 to 5.04	Jun 2012 to Sep 2012	14	4.79	5.01	4.91	**100**
GRE-03	15.37	14.85 to 15.35	Oct 2012	3	14.96	15.18	15.10	100

Figure 12.2.1_5
Blank Submissions - SGS

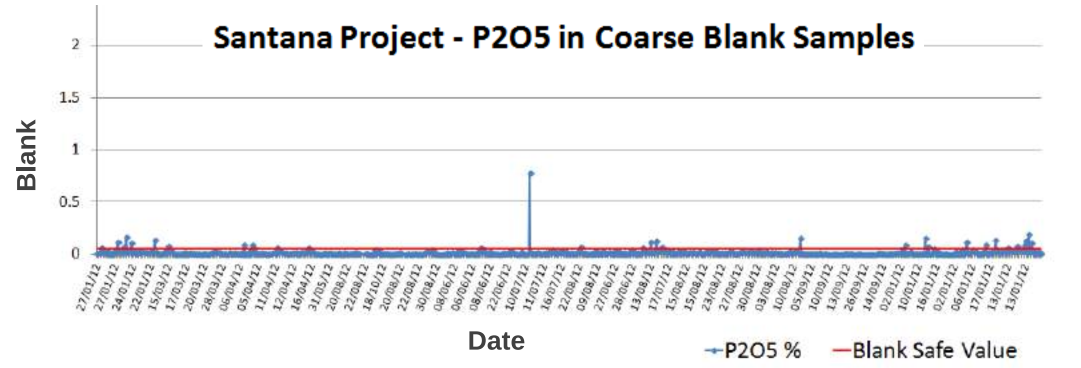


Figure 12.2.1_6
Standard PFA - SGS

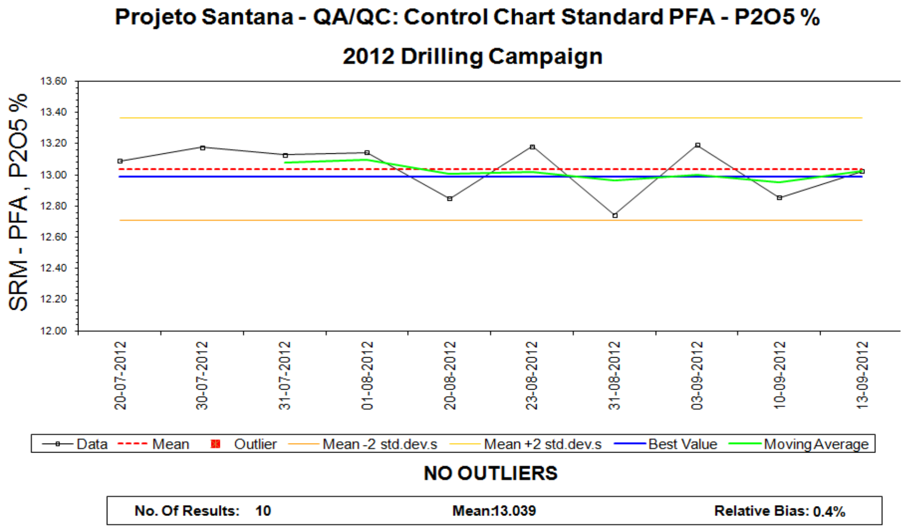


Figure 12.2.1_7
Standard PFB - SGS

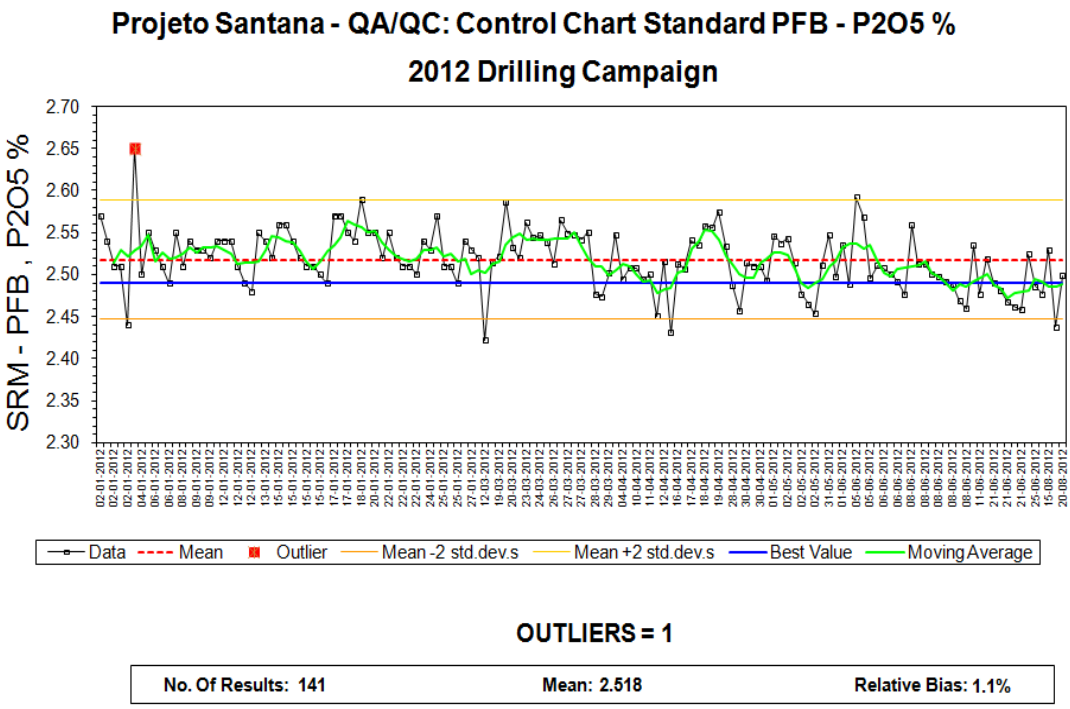


Figure 12.2.1_8
Standard PFM - SGS

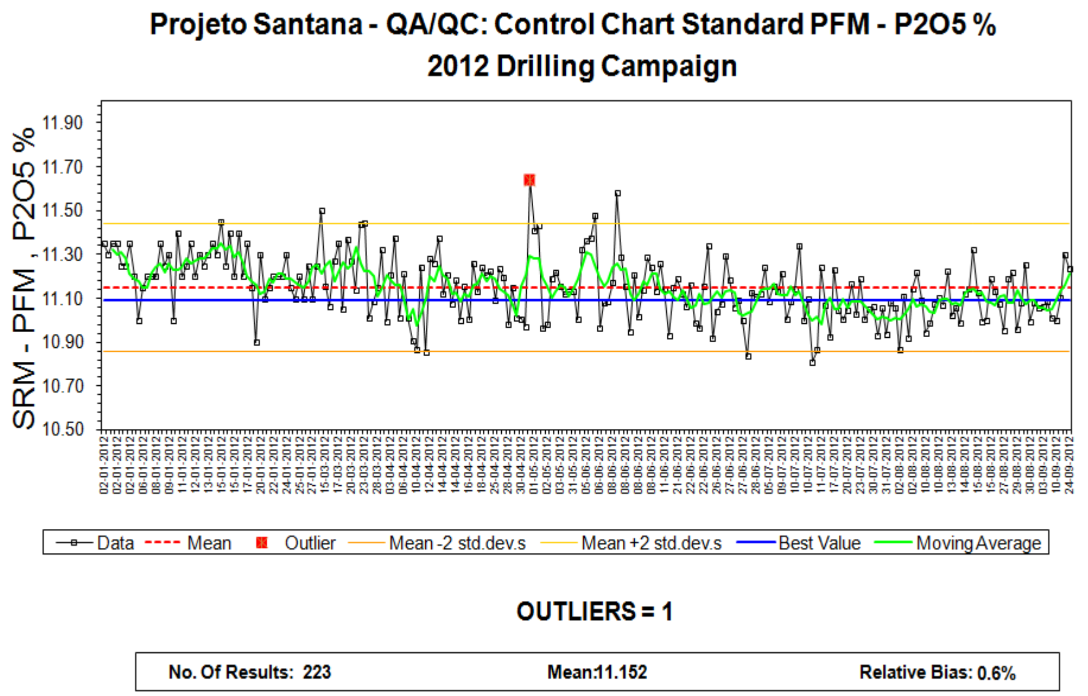


Figure 12.2.1_9
Standard GPO-11 - SGS

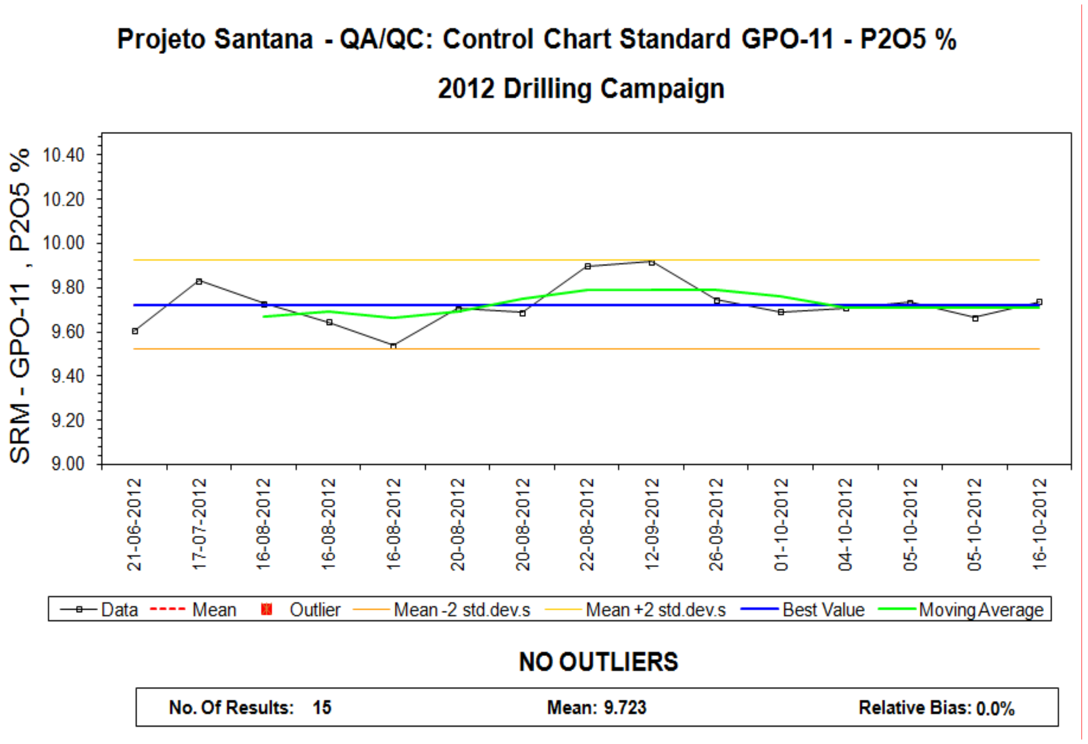
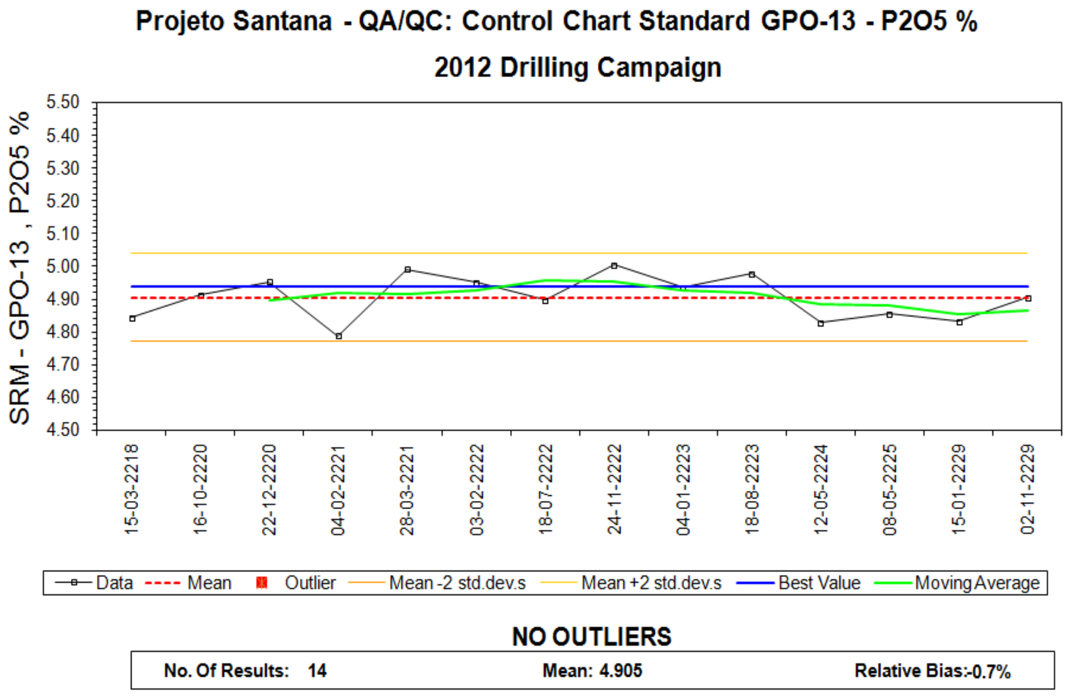


Figure 12.2.1_10
Standard GPO-13 - SGS



Standards submitted to SGS have shown excellent precision.

12.2.2 Field Duplicates

The field duplicate data has been analysed by AMSL using a number of graphical comparative analyse methods.

The objective of this is to determine relative precision levels between various sets of assay pairs and the quantum of relative error. This directly reflects on the precision of the sampling technique utilised.

ALS Field Duplicates

MBAC completed field duplicate by splitting the crush reject and resubmitting for analysis. A total of 606 field duplicates were submitted to the ALS laboratory.

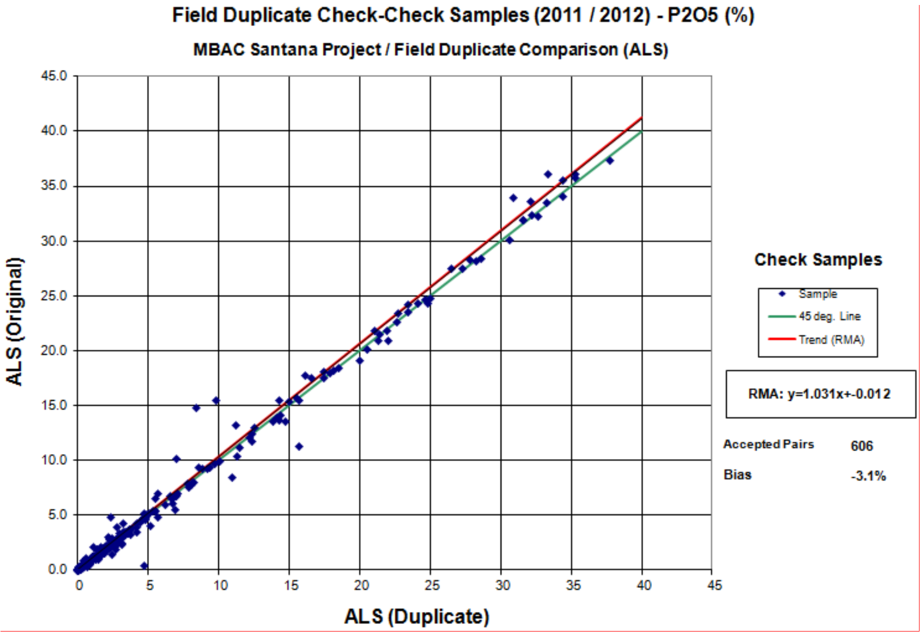
Based on the analysis, AMSL concludes that the precision of field duplicates submitted for analysis is acceptable for P_2O_5 as shown in Table 12.2.2_1 and Figures 12.2.2_1 and 12.2.2_2 below.

Table 12.2.2_1

Field Duplicate Data Statistics - ALS (P_2O_5)

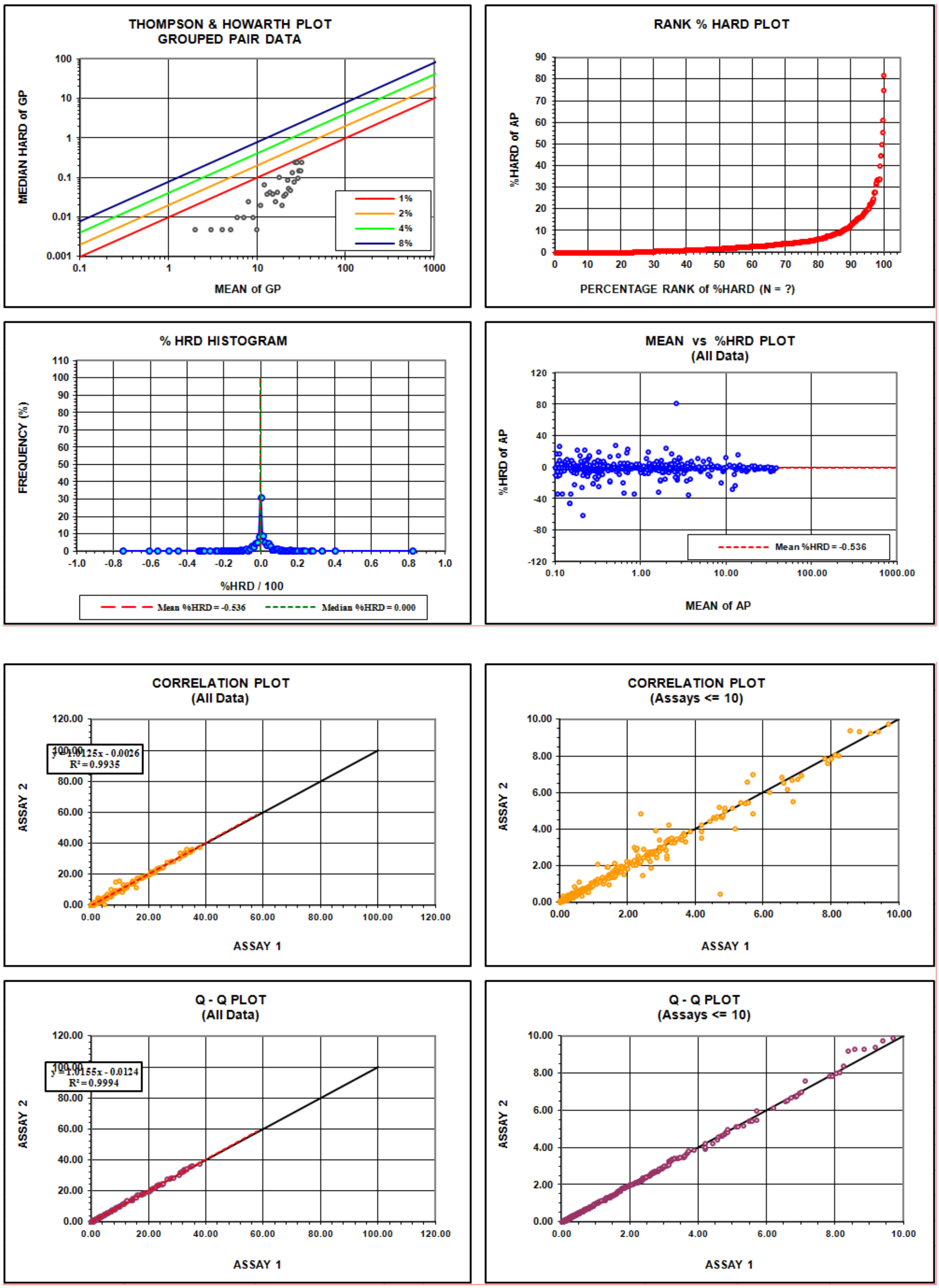
MBAC Santana Phosphate Project - ALS Duplicate Data (P2O5)							
	P2O5 (Original)	P2O5 (Duplicate)	Units		Result	95% C.L.	Units
No of Pairs:	606	606		Pearson CC:	0.997		
Minimum:	0.010	0.010	%	Spearman CC:	0.992		
Maximum:	37.700	37.400	%	Mean %HARD:	4.689	....	%
Mean:	3.279	3.317	%	Median %HARD:	1.695	....	%
Median:	0.500	0.495	%	Precision at 90%:	12.668	....	%
Std Dev:	6.879	6.988	%	Mean %HRD:	-0.536	0.789	%
CV:	2.098	2.107		Median %HRD:	0.000		%

Figure 12.2.2_1
Field Duplicate Data - ALS (P_2O_5)



MBAC completed diamond core field duplicate by splitting the crush reject and resubmitting for analysis. Reverse circulation field duplicates were prepared by re-splitting the bulk reject.

Figure 12.2.2_2
Field Duplicate Data Plots - ALS (P_2O_5)



SGS Field Duplicates

MBAC completed field duplicate by splitting the crush reject and resubmitting for analysis. A total of 404 field duplicates were submitted to the SGS laboratory.

Based on the analysis, AMSL concludes that the precision of field duplicates submitted for analysis is acceptable for P_2O_5 as shown in Table 12.2.2_2 and Figures 12.2.2_3 and 12.2.2_4 below.

Table 12.2.2_2

Field Duplicate Data Statistics - SGS (P_2O_5)

MBAC Santana Phosphate Project - SGS Duplicate Data (P2O5)

	Assay 1	Assay 2	Units		Result	95% C.L.	Units
No of Pairs:	404	404		Pearson CC:	0.994		
Minimum:	0.005	0.012	%	Spearman CC:	0.981		
Maximum:	33.200	33.500	%	Mean %HARD:	5.625	………	%
Mean:	1.296	1.311	%	Median %HARD:	2.977	………	%
Median:	0.242	0.240	%	Precision at 90%:	14.403	………	%
Std Dev:	3.825	3.931	%	Mean %HRD:	0.985	1.014	%
CV:	2.952	2.999		Median %HRD:	0.000		%

MBAC completed diamond core field duplicate by splitting the crush reject and resubmitting for analysis. Reverse circulation field duplicates were prepared by re-splitting the bulk reject.

Figure 12.2.2_3
Field Duplicate Data - SGS (P_2O_5)

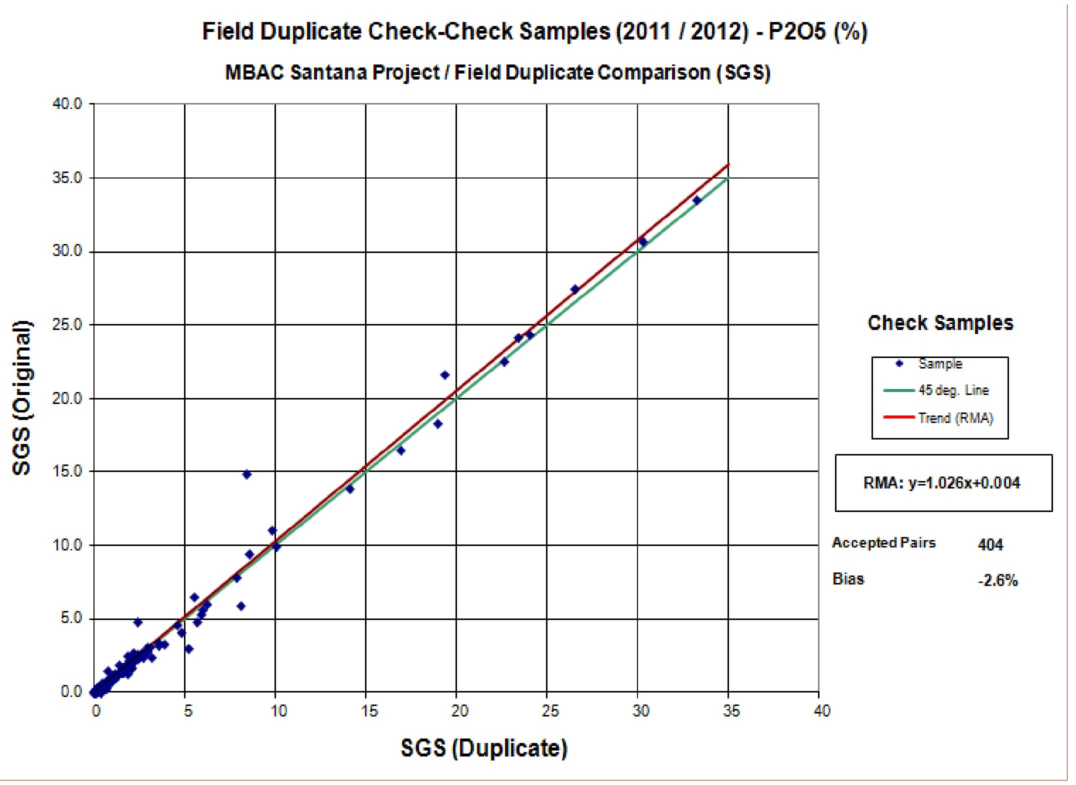
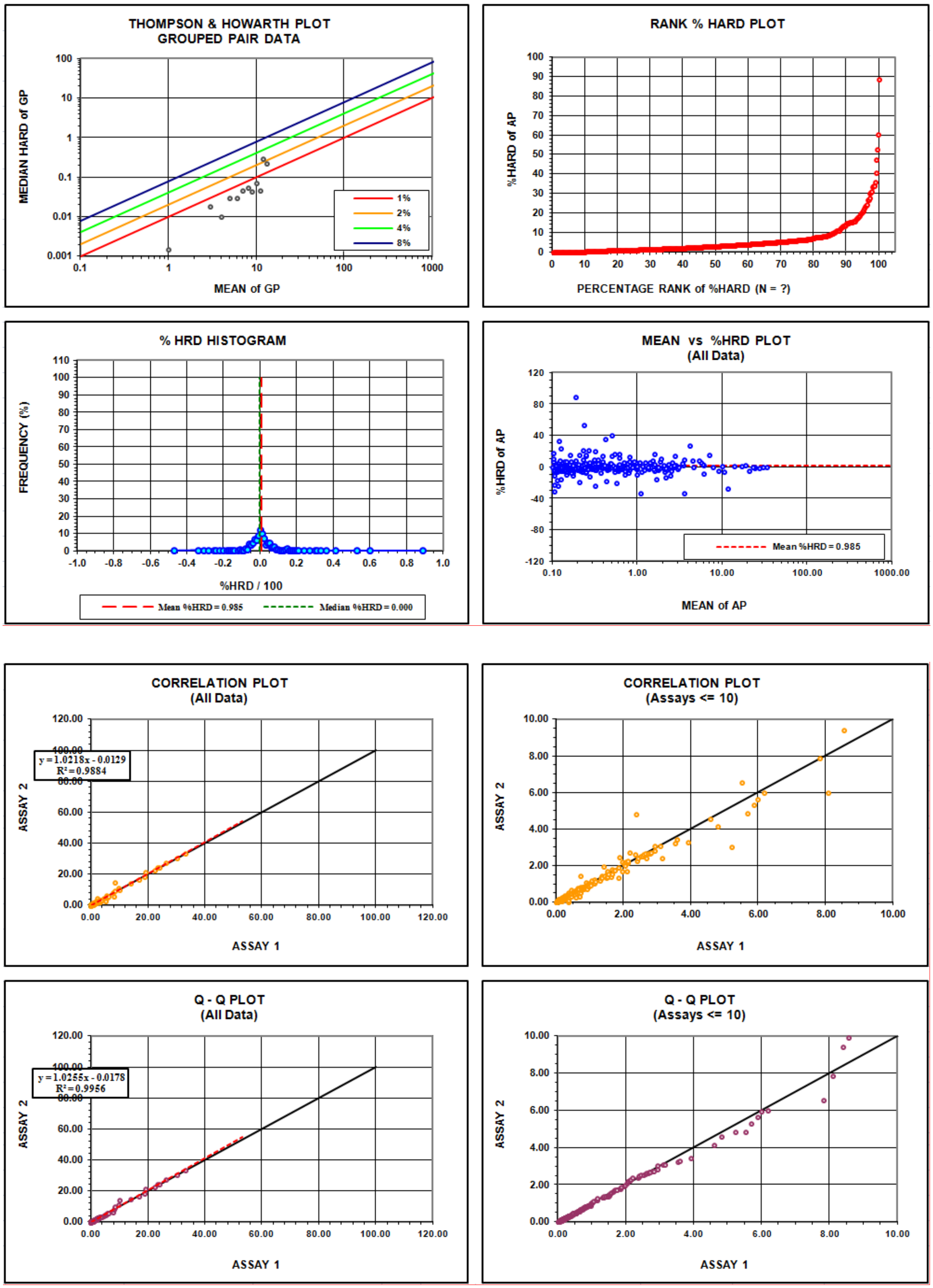


Figure 12.2.2_4
Field Duplicate Data Plots - SGS (P_2O_5)



12.2.3 Twin Hole Comparison (RC vs DC Drilling)

The 10 twin holes (RC versus DC) completed by MBAC has been analysed by AMSL using a number of graphical comparative analyse methods. The objective of this is to determine relative precision levels between various sets of assay pairs and the quantum of relative error. This directly reflects on the precision of the sampling technique utilised.

The diamond core was not entirely sampled on 1m intervals as recommended in the prior NCL PEA report so this is not a true twin hole comparison, however the results returned from this exercise indicate poor precision between the two sets of data. Despite the poor precision between datasets, there is no apparent bias between RC and DC drilling results.

DC vs RC comparison results are presented below in Table 12.2.3_1 and Figures 12.2.3_1 and 12.2.3_2.

Table 12.2.3_1

Twin Hole Comparison Statistics - DC vs RC Drilling (P_2O_5)

MBAC Santana Phosphate Project - Twin Hole Comparison (P2O5)

	Assay 1	Assay 2	Units		Result	95% C.L.	Units
No of Pairs:	460	460		Pearson CC:	0.737		
Minimum:	0.030	0.020	%	Spearman CC:	0.753		
Maximum:	38.200	37.600	%	Mean %HARD:	30.480	....	%
Mean:	4.787	4.867	%	Median %HARD:	22.808	....	%
Median:	1.407	1.400	%	Precision at 90%:	68.198	....	%
Std Dev:	8.623	8.328	%	Mean %HRD:	-1.710	3.607	%
CV:	1.801	1.711		Median %HRD:	0.207		%

Figure 12.2.3_1
Twin Hole Comparison Graph - DC vs RC Drilling (P_2O_5)

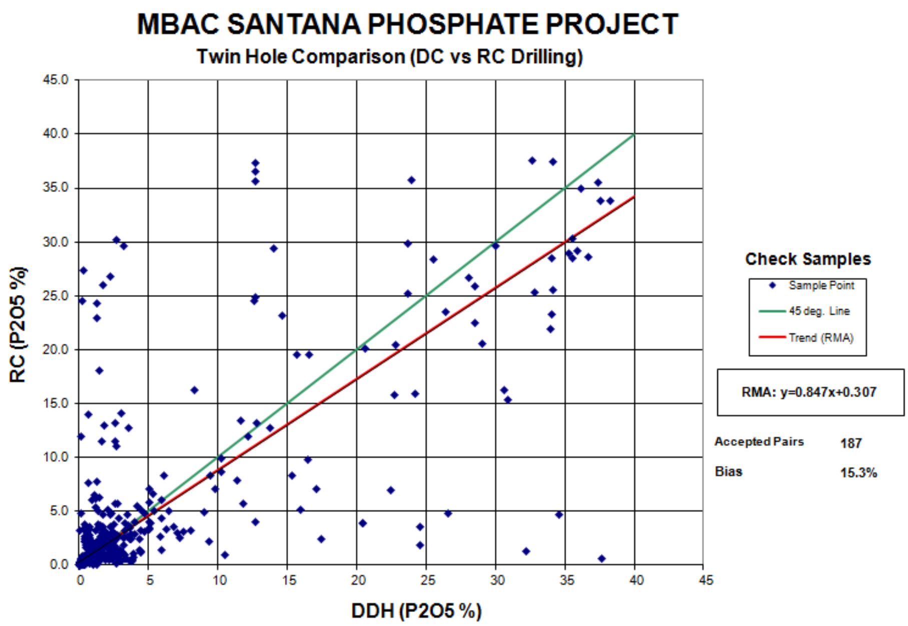
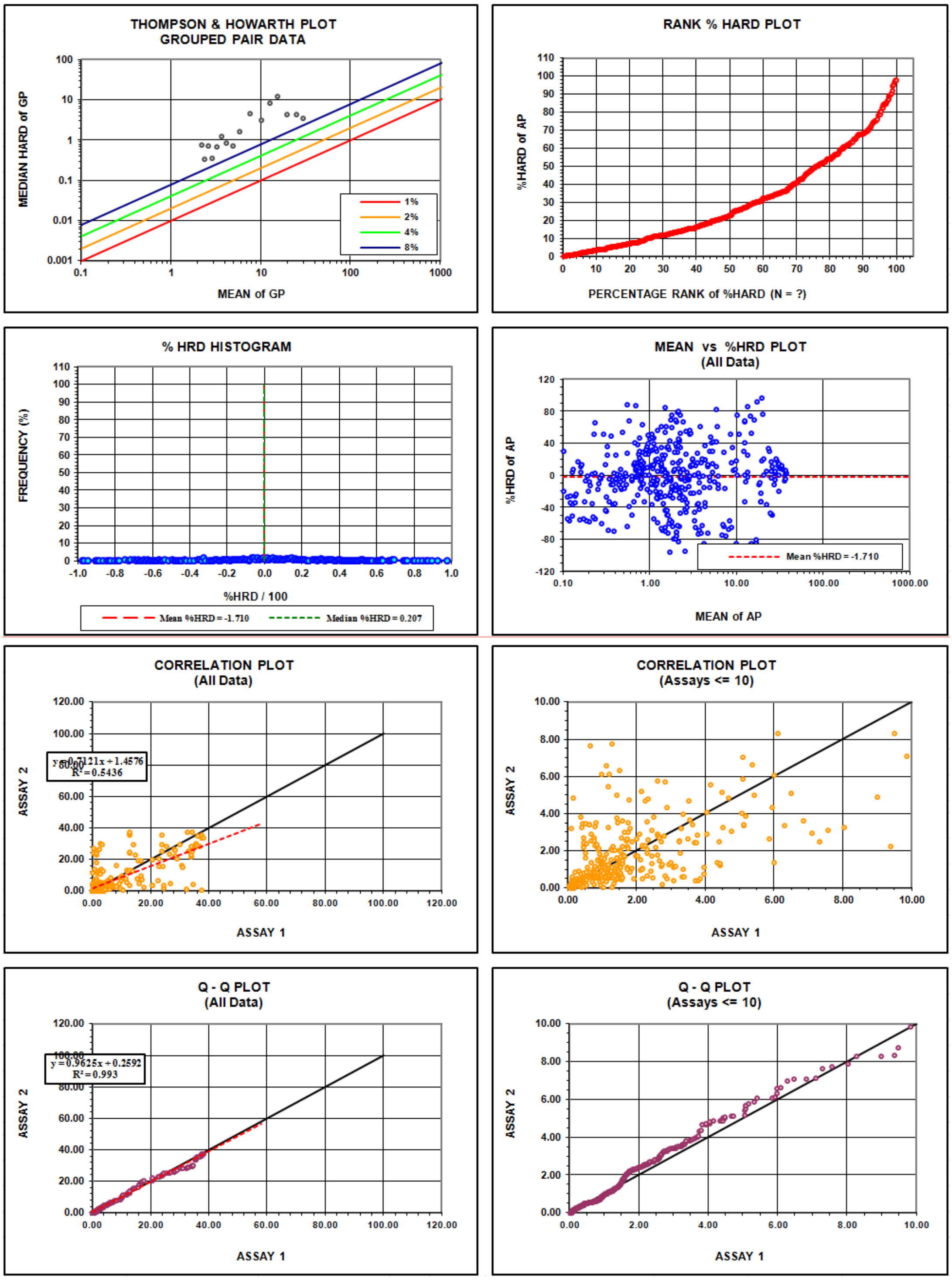


Figure 12.2.3_2
Twin Hole Comparison Data Plots - DC vs RC Drilling (P_2O_5)



12.2.4 Umpire Assays

A total of 1,608 umpire assay samples have been submitted (early 2013) by MBAC based on the recent reverse circulation and diamond drilling program.

Umpire assays were submitted in two separate batches in an effort to test both the SGS and ALS laboratories based on a switch in laboratories in early 2012. Details for each submission are listed below;

- 1,015 Umpire Assays Submitted - Original ALS samples submitted to SGS.
- 593 Umpire Assays Submitted - Original SGS samples submitted to ALS.

Based on the analysis, AMSL concludes that the precision of assay results between laboratories is acceptable as shown in Figures 12.2.4_1 and 12.2.4_2 below;

Figure 12.2.4_1
Umpire Assay Comparison - Original ALS vs Check SGS Pulps (P_2O_5)

Santana Project - RMA Parameters - Check Samples/Drill Core

Element	R^2	N (total)	Pairs	m	Error (m)	b	Error (b)	RMA Ecuation	Vies
P2O5 (%)	0.970	1015	1015	0.994	0.005	-0.012	0.136	RMA: y=0,994x+-0,012	0.6%
Al2O3 (%)	0.998	1015	1015	0.988	0.001	-0.010	0.025	RMA: y=0,988x+-0,01	1.2%
CaO (%)	0.990	1015	1015	0.996	0.003	-0.052	0.135	RMA: y=0,996x+-0,052	0.4%
Fe2O3 (%)	0.997	1015	1015	0.995	0.002	0.015	0.047	RMA: y=0,995x+0,015	0.5%
MgO (%)	0.992	1015	1015	0.982	0.003	0.010	0.017	RMA: y=0,982x+0,01	1.8%
MnO2 (%)	0.979	1015	1015	0.997	0.005	-0.007	0.030	RMA: y=0,997x+-0,007	0.3%
SiO2 (%)	0.961	1015	1015	1.009	0.006	-0.122	0.267	RMA: y=1,009x+-0,122	-0.9%
TiO2 (%)	0.997	1015	1015	1.029	0.002	-0.005	0.002	RMA: y=1,029x+-0,005	-2.9%

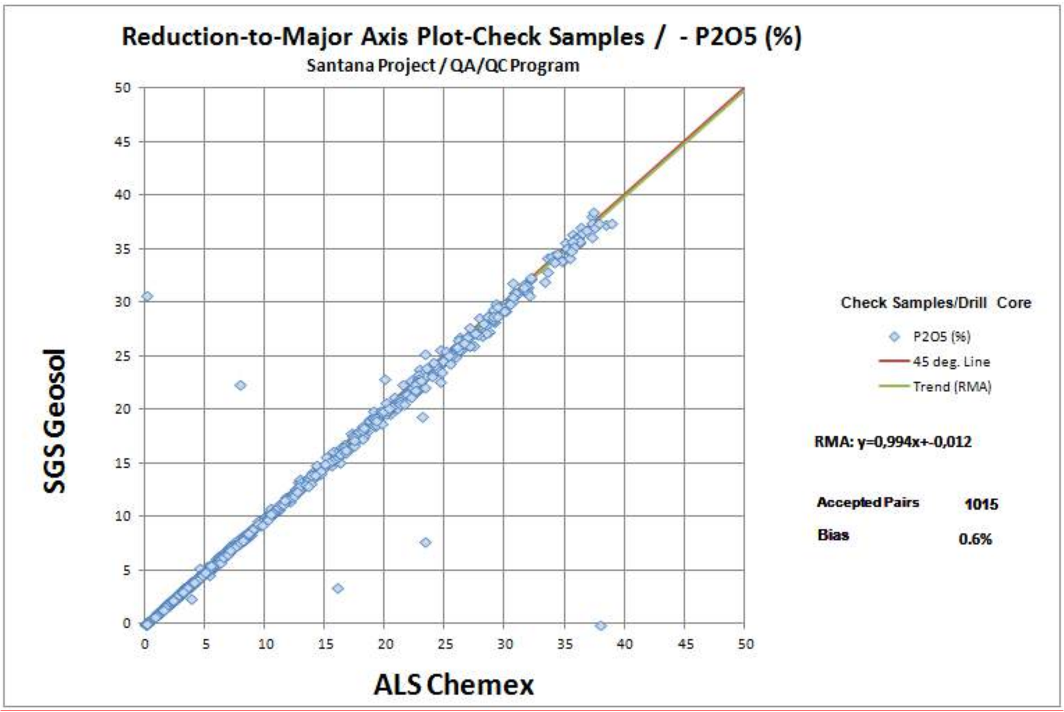
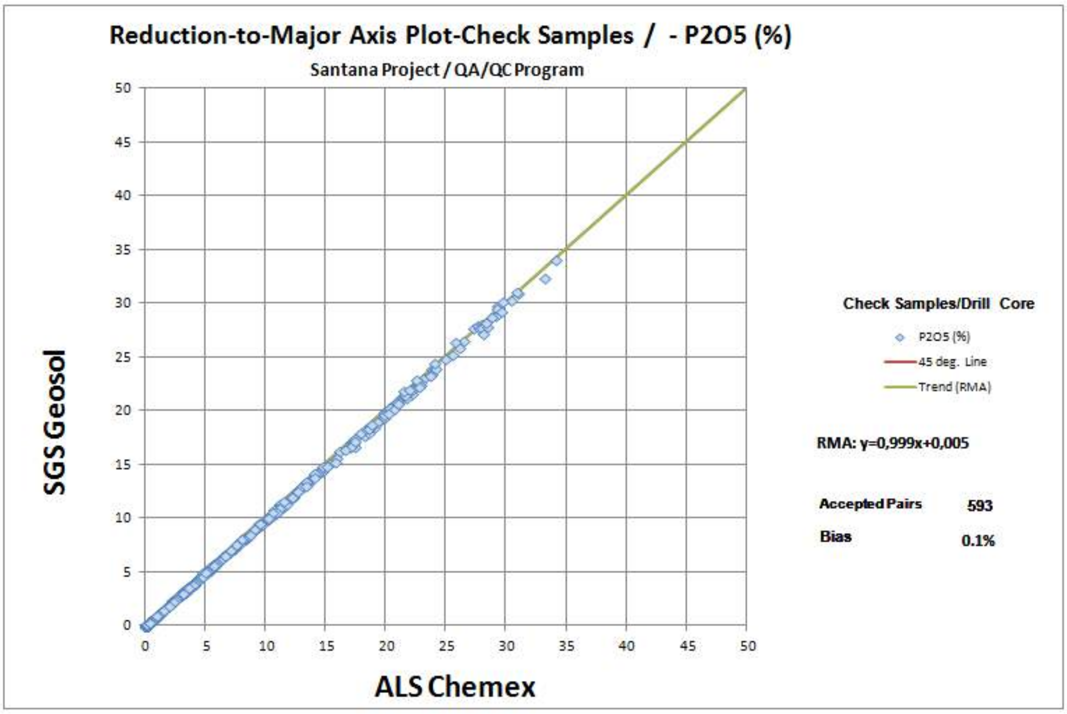


Figure 12.2.4_2
Umpire Assay Comparison - Original SGS vs Check ALS Pulps (P_2O_5)

Santana Project - RMA Parameters - Check Samples/Drill Core									
Element	R^2	N (total)	Pairs	m	Error (m)	b	Error (b)	RMA Ecuation	Vies
P2O5 (%)	1.000	593	593	0.999	0.001	0.005	0.015	RMA: y=0,999x+0,005	0.1%
Al2O3 (%)	0.999	593	593	1.003	0.001	0.025	0.023	RMA: y=1,003x+0,025	-0.3%
CaO (%)	1.000	593	593	0.998	0.001	0.057	0.030	RMA: y=0,998x+0,057	0.2%
Fe2O3 (%)	0.998	593	593	1.009	0.002	-0.066	0.036	RMA: y=1,009x+-0,066	-0.9%
MgO (%)	0.999	593	593	1.002	0.001	-0.012	0.009	RMA: y=1,002x+-0,012	-0.2%
MnO2 (%)	0.997	593	593	1.040	0.002	0.022	0.014	RMA: y=1,04x+0,022	-4.0%
SiO2 (%)	0.999	593	593	0.997	0.001	0.086	0.046	RMA: y=0,997x+0,086	0.3%
TiO2 (%)	0.997	593	593	0.984	0.002	0.008	0.002	RMA: y=0,984x+0,008	1.6%



All umpire assay results are inside a 5% tolerance limit.

12.2.5 Data Quality Summary

The standards data has shown a high accuracy (within 2 standard deviations) as reported by the SGS laboratory for PFB and PFM. Standard PFA was not utilised due to QA/QC issues.

The field duplicate data for both RC and DC has returned acceptable precision suggesting no issues with precision with the sampling method repeatability.

Umpire assays have returned acceptable precision suggesting no bias between laboratories.

RC versus DC twin holes has returned poor precision. The reason for this could be attributed to the fact that 25% of the RC holes are wet. Another likely factor may be water washing out phosphate mineralization in the DC drilling and sampling procedure.

AMSL noted this may be a material issue for the mineral resource estimate, and recommended steps be taken by MBAC to address this issue. Since late 2011, only DC drilling has been completed across the Santana project area, and AMSL note suitable recoveries and sample procedures employed..

AMSL considers the data of sufficient accuracy and precision for the current mineral resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Beneficiation Process Testing

The beneficiation process for this FS was developed based on previous pieces of information presented in the PEA and PFS report and in more test works that will be presented in this section; which included geological data, characterization studies, and metallurgical test works. Based on this information, the beneficiation process was delineated and defined. This section of the FS will describe the unit operations involved, handling of products, and auxiliary infrastructure, Process Flow Diagrams (PFDs), material balance, equipment list, and layouts. Finally, recommendations for additional information required for the detailed study phase will be given.

13.1.1 Technological Characterization Studies and Results

This study was done by the Technological Characterization laboratory facilities of the São Paulo State University. The mineralogical characterization studies were carried out using two composite samples from the first 22 diamond drill core holes, that were classified according the chemical composition as Normal (CaO/P_2O_5 ratio >1,18) and Low relationship (CaO/P_2O_5 ratio <1,18). These characterization studies included physical and chemical characterization, particle size distributions (PSD), complete head chemical analyses, Screen (Size) Assays, mineralogical studies, liberation studies and sink and float tests.

For these purposes, Screen (Size) Assays, minerals separation by heavy media and high intensity magnetic separation, and QEMSCAN with X-Ray Diffraction (XRD) and X-Ray Fluorescence (XRF) for analysis were used. After the screen analysis of the sample at 1.68 mm, 0.60 mm, 0.30 mm, 0.150 mm, 0.074 mm, 0.037 mm, 0.020 mm, and deslimed at 0.010 mm; the sub-samples of each size fraction up to 0.020 mm were submitted to heavy media separation . Then, magnetic separation of each size fraction of the sink material was carried out. In the case of Normal sample 0.3A (4.8KGauss) was used, but for the Low Relation (LR) sample 0.1A (1.9KGauss), 0.2A (3.3KGauss), and 0.3A were used. All fractions produced were studied using QEMSCAN, XRD and XRF. The results obtained showed that both the Normal sample and the LR sample were representative of the Group studied, and in agreement with previous evaluations. The Normal sample showed a CaO/P_2O_5 ratio of 1.3 with 40.9 wt% above 0.020 mm containing 33% of P_2O_5, and the -0.010-mm size fraction (slimes) 29.4 wt% containing 25% of P_2O_5; whereas, the LR sample showed a CaO/P_2O_5 ratio of 0.79 with 40.8 wt% above 0.020mm containing 43.5% of P_2O_5, and the -0.010-mm material (slimes) 47 wt% containing 39.8% of P_2O_5.

Table 13.1.1_1 presents the mineralogical composition for the 0.600x0.010-mm size fraction of both Normal and LR samples. This table showed that apatite corresponded to 54% of the material, with quartz corresponding to 29%, clayed minerals to 7.2%, iron bearing minerals to 2.4% (1.3% being magnetite and 1.1% goethite), and other aggregates of several minerals corresponding to 7.4% of the mass. In the case of LR sample only 12% of the material was apatite, 30% of the weight was quartz, 8.9% were clayed minerals, 10% iron bearing minerals (3.9% magnetite and 6.1% goethite); the aggregate of several minerals corresponding to 39.1% including 3.4% of psilomelane (manganese mineral).

Table 13.1.1_1
Mineralogical Composition for the 0.600x0.010-mm Size.
Fraction for Normal and Low Relation Samples

Minerals	Normal %	Low Relation %
Apatite	54.00	12.00
Quartz	29.00	30.00
Clayed Minerals	7.20	8.90
Psilomelane	2.00	3.40
Magnetite	1.30	3.90
Goethite	1.10	6.10
Aluminum Phosphates	0.70	8.80
Aggregate (goethite + apatite)	1.30	4.00
Aggregate (clay + sec. phos. + Fe ox.)	1.00	13.00
Aggregate (clay + sec. phos. + apatite)	0.50	6.60
Aggregate (clay + Psilom. + Fe ox.)	0.20	1.90
Others*	1.70	1.40
Total	100.00	100.00

* Others= plagioclase, anatase, titanite, Spinel, calcite, dolomite, gibsite, monazite.

Apatite for the Normal sample was mainly free with porous and fractured grains, some cemented clay-limonite, quartz, and mix particles of clay and goethite were also present. The mineralogical distribution of the 0.600x0.010-mm particles showed that 86% of the apatite was free; 12% constituted binary mixes with quartz (6.9%), clays (1.1%), psilomelane (1.3%), and other aggregates (2.7%); 1.6% with ternary mixes of quartz (10.4%), clays (0.2%), psilomelane (0.3%), and other aggregates (0.7%). For the 0.600x0.037-mm size fraction phosphate liberation was 56% to 78%; whereas, for the -0.037 mm was > 90%. Whether the area or perimeter was used to determine the liberation, the results were similar, indicating that the Normal type of samples were more suitable for beneficiation using surface based processes (see Table 13.1.1_2). Therefore, the potential recovery curves as indicated by grade as a function of recovery for apatite for the 0.600x0.010-mm size fraction showed a 95% recovery of apatite at 40% P_2O_5, but 75% for the 0.150x0.074-mm fraction, 85% for the 0.075x0.037-mm material; the overall potential recovery calculated in 83%.

In the case of the LR sample, the apatite was 67% free as determined by area analysis, and showed aggregates of aluminum phosphate + clays + goethite. The grains are generally porous and fractured. Binary mix particles were 17% including quartz (2.6%), clay (1.4%), psilomelane (2.2%), Al-phosphates (4.8%), goethite (2.6%), and other aggregates (3.4%). Ternary mix particles were 16%, containing quartz (1.1%), clay (1.2%), psilomelane (2.4%), Al-phosphates (3.3%), goethite (2.7%), and other aggregates (5.3%). The 0.074-mm size fraction reported 31%-37% apatite fee; whereas, the <0.074-mm material 50.9%. If the perimeter analysis was used, free apatite was 61% (see Table 13.1.1_2). These results indicated that the liberation by area (67%) was greater than that reported by perimeter (61%); thus more locked minerals in the edge of the particles. Consequently, the results indicated that LR type of phosphate ore was more difficult for beneficiation by surface processes than the Normal type.

Table 13.1.1_2

Liberation Grade for Apatite in the 0.600x0.010-mm Size Fraction for Normal and Low Relation Samples.

	Normal				**Low Relation**			
	Fraction of an Apatite Particle				**Fraction of an Apatite Particle**			
Grade of Liberation	≥ 85	≥ 90	≥ 95	100	≥ 85	≥ 90	≥ 95	100
By Area	91	89	86	75	76	72	60	51
By Perimeter	89	87	84	77	71	67	61	53

13.1.2 Metallurgical Test Work

Based on the information obtained from the characterization studies and the preliminary flotation tests results presented on the PEA, metallurgical testwork was conducted on the Group samples prepared from the Diamond and RC Drill cores from DD-22 to RC-188, Samples from Trench at Hole SAN-DD-0004/006 for 2m-3m and 3m-5m Composites, and High and Low Fe Composites. This testwork was aimed at determining the variability of these samples and confirm the general design parameters, such as weight distribution of products, grades and recoveries, unit operations and metallurgical processes, solids content of different products and streams, etc.

13.1.2.1 General Aspects

Metallurgical testwork was carried out by MBAC at its process facilities located at its Itafos Phosphate Mine, Campos Belos, Goias State, Brazil, and in The FUNMINERAL mineral processing laboratories from the State of Goiais. Magnetic separation tests were done by ERIEZ laboratories in São Caetano- São Paulo. The materials and methods used in the flotation testwork was the same as described in the PEA and the magnetic separation tests were done in the following equipment.

- Wet-low-intensity magnetic Rod (1,500Gauss to 7,000 Gauss)
- Wet-high-intensity magnetic separator, AC, type AL-4 (8,000 Gauss to 10,000 Gauss).

The flotation tests were conducted using a standard procedure with variations according to the variable to be investigated. For this purpose, samples were screened at a specific size; the coarse product was ground until it reached the desired particle size. Thus, this ground material was mixed with the fine product and scrubbed with the addition of NaOH in the attrition cell for 15 minutes before being deslimed in the desliming unit. The hydrocyclone underflow was floated in the laboratory machine; whereas, the hydrocyclone overflow was collected, decanted, dried weighed, and analyzed. The rougher concentrates were reground to a specified size. Eventually, some tests were conducted without the desliming step and/or without regrinding for research purposes.

Reagents were diluted to 5% weight before being used. Corn Starch and soy bean oil were prepared with a 15 wt% NaOH solution. In general, conditioning was carried out in the presence of depressants for three minutes. Then, the collector was added and conditioned for two minutes more. Other depressants and modifiers, such as Na2SiO3 (sodium silicate), ArrMaz Polymer 1111 were also used in the testwork.

Depending on the Group of composite diamond drill phosphate ore and/or type of composite prepared, the variables to be studied and the conditions of the tests (un-deslimed, deslimed, grinding mesh, regrinding mesh, pH, special reagents, etc.) changed; the rougher flotation time varied from 3.5 minutes to 14 minutes or until starvation. The first cleaner stage used 3 minutes to 9 minutes; whereas, the second cleaner stage varied from 2.5 minutes to 6 minutes. Some tests for the High and Low Fe composites were conducted using three cleaner stages, the third cleaner stage using 1.5 minutes to 4 minutes.

In the case of magnetic separation studies, cleaner concentrates from repetition tests of Test SAN 17, labeled Rep 1 to 4 were used. The material was homogenized, and split in a Jones splitter to obtain three sub-samples. Then, the sub-samples were re-pulped and submitted to 1,500 Gauss wet-low-intensity magnetic separation. A second wet-low-intensity magnetic separation tests were carried out at 7,000 Gauss. Finally, tests were performed using a wet-high-intensity magnetic separator at 8,000 Gauss and with apertures of 1.5 mm, 2.5 mm, and 3.2 mm. A test at 10,000 Gauss and 1.5 mm aperture was also carried out.

13.1.2.2 Grinding and Regrinding for Flotation Tests

Composite Bulk Samples obtained from a Trench at Hole SAN-DD-0004 at (3m-5m), November 2011 required 3 minutes of grinding time to obtain 95% passing 106 µm. No regrinding tests were conducted. However, this grinding time may be equivalent to that of grinding + regrinding in the first 32 tests. Again, the material showed to be friable, and the aggressive grinding resulted in P_2O_5 losses in the slimes (< 10-µm particle size) as high as 29% with an average of 23.9% ± 3.5%.

In the case of the confirmation flotation tests for Bulk RC Drilling Samples for High Fe and Low Fe samples Groups, only two tests were carried out using a regrinding step of 1.5 minutes for each Group type. However, High Fe Group was ground for 3.5 minutes for a P90 of 106 µm; whereas, Low Fe Group was ground for 6.75 minutes and 7 minutes. The slimes (-10-µm particles) P_2O_5 losses were 34.2% and 24.9% for High Fe and Low Fe Group, respectively, when regrinding to P95 of 106 µm was the target. High Fe Group and Low Fe Group using a single grinding step resulted in a P90 of 106 µm. The P_2O_5 losses in the slimes were 29.0% ± 0.7% for the High Fe Group and 23.2% ± 0.6% for the Low Fe Group.

These results clearly showed that Santana Phosphate ore is friable, and requires a stage comminution system to reduce P_2O_5 losses in the slimes by a controlled size reduction and sizing system.

13.1.2.3 Bench Scale Flotation Tests

The Group F, Medium Grade was chosen to study the main flotation variables. After this, the conditions defined were used to estimate the recovery and the concentrate quantities that could be gotten with other Groups.

Several conditions that influence the process were evaluated and are summarized below:

a. Effect of Grinding Size - Grinding size of 90% passing 150# showed to be the best grinding size. Desliming is necessary, but requires control in the Flotation Feed Preparation-size reduction stage.
b. Effect of pH - Results showed a strong effect of pH. P_2O_5 recovery increased from 13.6% at pH 7.5 to 79.5% at pH 11. However, P_2O_5 grade was reduced to 27.9% due to SiO2 activation and increased in silicates. Test SAN 9 conducted at pH 10 reported a lower recovery of 73.1%, but significant higher concentrate grade, 29.2% P_2O_5; thus, more selective.
c. Effect of Depressor Type and Addition - Test SAN 11 to 15 evaluated corn starch against sodium silicate in two addition levels without desliming. Better P_2O_5 grade and recovery, were obtained with sodium silicate at an addition level of 430 g/t.
d. Effect of Attrition with Soda and Concentrate Regrinding - The concentrates produced up to Test SAN 15 presented high grade of Fe2O3 and Al2O3 that could be deleterious for the acidulation in order to get a minimum of 18% solubility in citric acid (SSP specifications). Tests SAN 16 to 18 ground samples to 90% passing 65#, and they were scrubbed with NaOH before desliming. Then, the rougher concentrates were reground to P80 106 µm, before the concentrates being floated twice. It was possible to obtain concentrates with high P_2O_5 grade and lower Al2O3 and Fe2O3 grades. Under these conditions, P_2O_5 grade increased to 35.8% for a recovery of 59.0% of P_2O_5. Fe2O3 decreased to 2.1% and Al2O3 to 0.37%. However, P_2O_5 losses in the slimes increased to 27.2%.
e. Effect of Bulk Flotation and Dispersant - A dispersant, ArrMaz Polymer 1111 was tested, but after attrition scrubbing. Comparison of the tests was difficult, but it could be concluded that P_2O_5 recoveries above 50% are obtainable (even though recycling of intermediate products streams was not considered), and that P-1111 may help in obtaining low Fe2O3, Al2O3, and SiO2 concentrate with high P_2O_5 grade. Again, comminution technique was of utmost importance to minimize P_2O_5 losses in the slimes.
f. High and Low Fe Composite Flotation Tests - it was demonstrated that high Fe material resulted in low concentrate and recovery; thus, it should be selectively mined and blended at smaller portions to the ore feed to the plant.

g. Table 13.1.2.3_1 shows the influence samples presenting different chemical composition in the quality and P_2O_5 recovery in the concentrates. The main parameters that must be controlled to exploit and homogenize the ore are the CaO/P_2O_5 ratio and the Fe + Al assays. Mining plan already consider this restrictions on the ore.

Table 13.1.2.3_1

Bench scale flotation tests results showing the variability of different samples.

Samples	test	CONCENTRATES								
		Assay(%)					Weight (%)	P_2O_5 recovery (%)	kgR_2O_3 / t P_2O_5	$kgFe_2O_3$ /t P_2O_5
		P_2O_5	CaO	Fe_2O_3	Al_2O_3	SiO_2				
bulk, 2-5m, close holes DD-004/006	78	35,60	42,14	3,20	1,51	2,00	39,70	61,20	132,30	89,89
	79	34,86	40,32	3,60	1,71	2,15	38,30	57,90	152,32	103,27
	82	35,54	39,76	3,20	1,82	2,40	30,00	51,00	141,25	90,04
	83	34,20	38,22	4,80	2,20	3,19	37,20	61,20	204,68	140,35
composed from DD-029 to DD-063	REP1	31,15	39,06	6,00	0,75	9,09	26,30	59,37	216,69	192,62
	REP2	32,34	38,50	5,20	0,70	8,95	27,20	63,03	182,44	160,79
	REP3	31,10	35,86	5,60	0,68	10,85	28,70	64,17	201,93	180,06
	REP4	31,41	37,38	5,60	0,70	10,55	26,80	61,73	200,57	178,29
High Iron (>16%) composed from RC-92 to RC-118	AF-01	31,30	38,92	6,00	0,63	5,88	10,30	39,40	211,82	191,69
	AF-02	33,31	39,48	6,00	0,48	4,51	8,90	32,20	194,54	180,13
	AF-03	30,47	37,80	8,00	0,65	9,22	14,40	49,20	283,89	262,55
	AF-05	30,06	38,22	8,00	0,81	8,03	15,50	46,90	293,08	266,13
Low Iron (<16%) composed from RC-109 to RC-142	BF-01	33,36	44,68	2,01	0,42	9,79	30,75	57,40	72,88	60,23
	BF-02	31,50	40,24	2,61	0,45	14,03	36,92	64,50	96,92	82,72
	BF-03	32,47	37,02	2,27	0,45	11,51	36,60	62,08	83,53	69,78
	BF-05	34,16	40,60	2,80	0,54	5,95	29,04	51,67	97,78	81,97
Composed 21,7	28	34,15	44,81	2,51	0,44	10,40	47,00	74,10	86,48	73,65
Composed LR 12,1	29	35,65	42,28	5,20	3,38	3,12	13,00	36,70	240,67	145,86
Global 17,1	30	35,91	45,22	2,80	1,15	6,29	28,00	57,10	110,00	77,97
Composed 2 - 12,2	31	34,36	45,80	2,57	0,54	8,93	24,30	68,40	90,52	74,88
Global 2-12,4	32	35,88	45,92	3,20	1,77	5,68	17,20	49,90	138,52	89,19

13.1.2.4 Magnetic Separation Tests

Concentrate composite of the samples Rep 1 to 4 were submitted to magnetic separation both at low intensity and high intensity fields. When a permanent-low magnetic field of 1,500 Gauss was applied to the slurry, no separation of magnetic material occurred. At a 7,000 Gauss of permanent magnetic field a minimum recovery of 0.6 g or 0.23 wt% magnetic material was obtained.

Table 13.1.2.4_1 Best Magnetic Separation Test Results showed that at 8,000 Gauss high magnetic field and 1.5 mm of aperture between magnets, the best results were obtained. These results showed that Fe2O3 could drop below 5% to 4.4% with an increase in P_2O_5 grade to 32.2% from a concentrate of 31.5% P_2O_5 and a recovery of 98.71% of P_2O_5. Since the rejection of Fe2O3 was in an acceptable range for this concentrate, it was considered for the process flow sheet design. A small increment in the intensity of the magnetic field (10,000 Gauss) did not show any improvement. However, higher intensity magnetic fields (up to 14,000 Gauss) may result in even better separation since this was related to the magnetic susceptibility of Fe-Mn minerals in the phosphate ore. Thus, this type of tests was recommended.

Table 13.1.2.4_1

Best Magnetic Separation Test Result

Produt	Weight, g	Weight, %	P_2O_5 (%)	CaO (%)	Fe_2O_3(%)	Al_2O_3 (%)	SiO_2 (%)	Mn (%)	Distr P_2O_5 (%)	Distr Fe_2O_3 (%)
Conditions :		T2-A == Scavenger - 8000 gauss - matriz 1,5 mm								
Magn	10.40	3.60%	11.25%	11.27%	32.80%	0.52%	16.62%	4.24%	1.29%	21.76%
Non-magn	278.70	96.40%	32.21%	37.38%	4.40%	0.36%	9.98%	1.81%	98.71%	78.24%
Total	289.10	100.00%	31.46%	36.44%	5.42%	0.37%	10.22%	1.90%	100.00%	100.00%

13.1.2.5 Pilot Plant Testwork

Pilot Plant Testwork was done from March to June 2012 in Gorceix Foundation in Ouro Preto MG, for grinding and desliming studies, and in the CDTN column flotation facilities in Belo Horizonte MG for column flotation studies. Sample was collected at Santana site in a trench open around hole DD-004 at a depth of 2 to 3m. Total sample weight was around 20 t. After crushing in a roll crusher to minus 2mm and homogenization, sample presented the chemical composition (%) and size distribution as showed below:

Table 13.1.2.5_1 – Chemical Analysis and Size Distribution of Pilot Plant Sample

P_2O_5	Fe_2O_3	SiO_2	Al_2O_3	CaO	MnO_2	F	CaO/P_2O_5	Laboratory
17.38	15.70	18.57	7.29	15.70	6.37	0.58	0.90	Labfert

Size fraction (µm)	DISTRIBUTION (%) WEIGHT			DISTRIBUTION (%) P205		ASSAY (%)
	RETAINED	ACUMULATED	PASSING	RETAINED	ACUMULATED	P2O5
1700	**8.74**	**8.74**	**91.26**	**7.72**	**7.72**	15.60
1180	**5.41**	**14.16**	**85.84**	**4.69**	**12.41**	15.32
1000	**2.02**	**16.17**	**83.83**	**1.75**	**14.16**	15.30
850	**2.75**	**18.92**	**81.08**	**2.30**	**16.46**	14.80
600	**3.16**	**22.08**	**77.92**	**2.59**	**19.05**	14.50

Size fraction (µm)	DISTRIBUTION (%) WEIGHT			DISTRIBUTION (%) P205		ASSAY (%)
	RETAINED	ACUMULATED	PASSING	RETAINED	ACUMULATED	P2O5
450	**0.88**	**22.96**	**77.04**	**0.70**	**19.75**	14.00
300	**2.98**	**25.95**	**74.05**	**2.35**	**22.10**	13.92
212	**2.67**	**28.61**	**71.39**	**1.85**	**23.95**	12.27
150	**1.61**	**30.22**	**69.78**	**1.09**	**25.04**	11.90
106	**2.09**	**32.32**	**67.68**	**1.41**	**26.44**	11.86
75	**2.52**	**34.84**	**65.16**	**1.72**	**28.17**	12.10
53	**2.65**	**37.48**	**62.52**	**2.13**	**30.29**	14.20
45	**1.11**	**38.59**	**61.41**	**0.91**	**31.21**	14.60
38	**0.84**	**39.43**	**60.57**	**0.72**	**31.93**	15.20
0	**60.57**	**100.00**	**0.00**	**68.07**	**100.00**	19.86
TOTAL	**100.00**	**CALCULATED P2O5 GRADE (%)**				**17.67**

To grind and deslime the ore it was tested two circuits. Figure 13.1.2.5_1 below present flowsheet 1 and the Table 13.1.2.5_2 present the metallurgical balance obtained with this flowsheet.

Figure 13.1.2.5_1 - Flowsheet 1

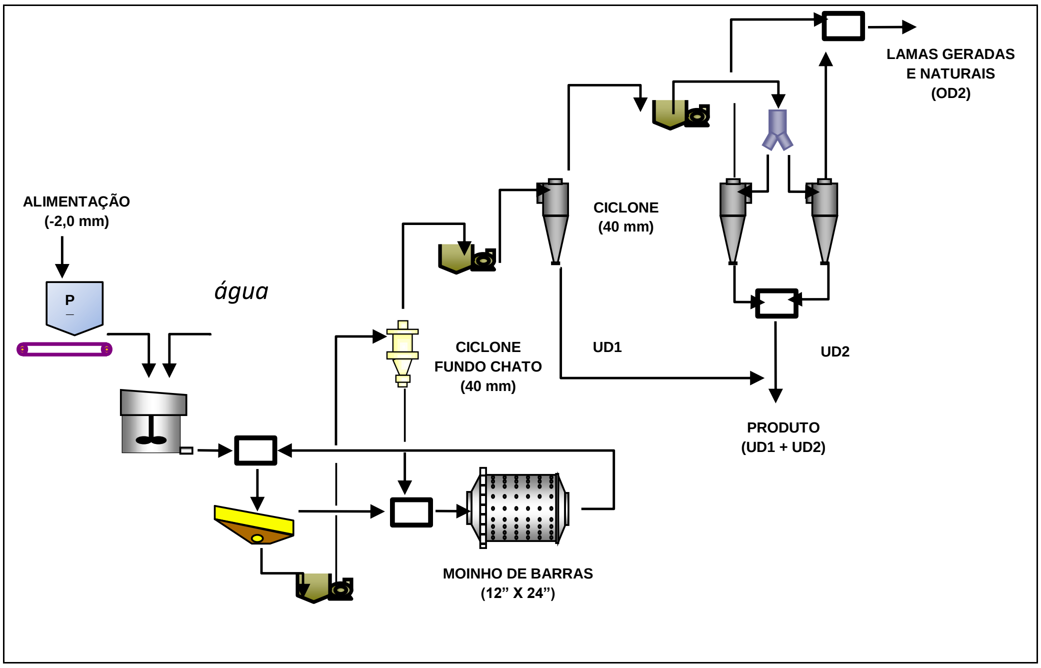


Table 13.1.2.5_2 – **Metallurgical Balance flowsheet1**

STREAMS	MEASURED WEIGHT (Kg/h)	ADJUSTED WEIGHT (kg/h)	MEASURED GRADE P2O5(%)	ADJUSTED GRADE P2O5 (%)	WEIGHT DISTRIBUTION (%)	P2O5 DISTRIBUTION (%)
AL	197.40	211.30	16.90	17.43	100.00	100.00
AP	195.00	211.30	16.60	17.43	100.00	100.00
OSP	72.30	55.44	15.00	15.00	26.24	22.58
USP	198.40	251.60	18.10	16.54	119.10	113.02
DM	75.20	95.78	13.50	13.70	45.33	35.62
OCC	202.50	211.30	18.10	17.43	100.00	100.00
UCC	43.90	40.34	11.90	11.90	19.09	13.04
UD1	87.20	91.03	16.50	16.00	43.08	39.54
OD1	119.50	120.30	18.70	18.51	56.92	60.46
UD2	57.70	57.69	22.00	21.46	27.30	33.63
OD2	60.90	62.58	16.10	15.79	29.62	26.83

Figure 13.1.2.5_2 below presents flowsheet 2 and the Table 13.1.2.5_3 present the metallurgical balance obtained with this flowsheet.

Figure 13.1.2.5_2 - Flowsheet 2

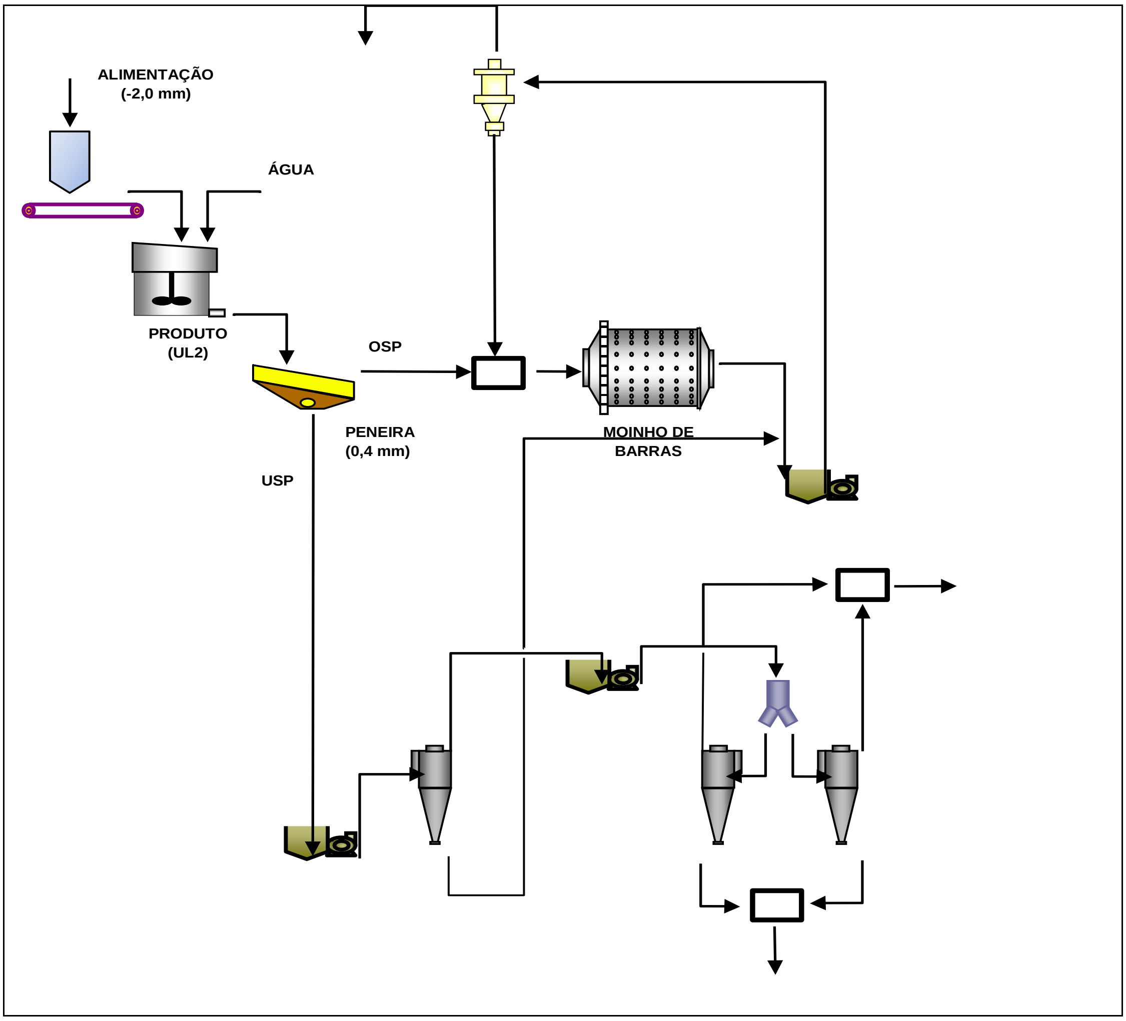


Table 13.1.2.5_3 - Metalurgical Balance Flowsheet 2

STREAMS	MEASURED WEIGHT (Kg/h)	ADJUSTED WEIGHT (kg/h)	MEASURED GRADE P2O5(%)	ADJUSTED GRADE P2O5 (%)	WEIGHT DISTRIBUTION (%)	P2O5 DISTRIBUTION (%)
AL	231.10	226.20	19.30	18.47	100.00	100.00
AP	242.40	226.20	20.40	18.47	100.00	100.00
OSP	63.60	69.72	16.20	16.27	30.82	27.15
UCC	139.70	147.00	12.70	12.39	64.96	43.58
ALMOAGEM	203.30	216.70	14.00	13.64	95.77	70.73
USP	158.20	156.50	20.10	19.45	69.18	72.85
DM	225.10	216.70	13.60	13.64	95.77	70.73
UL1	91.20	101.80	18.60	18.82	44.98	45.85
ACC	316.30	318.40	14.00	15.29	140.80	116.60
OCC	238.60	171.50	17.10	17.78	75.80	73.00
OL1	73.00	54.75	20.20	20.60	24.20	27.00
UL2	25.40	28.33	24.00	24.37	12.52	16.53
OL2	24.20	26.42	16.40	16.56	11.68	10.47

Deslimed product obtained from the two flowsheets, (UD1+UD2) from flowsheet 1 and (OCC+UL2) from flowsheet 2 were collected as obtained (slurry) and transported to CDTN laboratory for the column flotation testwork. The two samples tested presented the following size distributions:

Table 13.1.2.5_4 - Size distributions of the two samples

Size (µm)	% Acumulated	
	OCC+UL2	UD1+UD2
500.00	2.76	0.21
250.00	3.85	1.52
150.00	4.28	6.74
106.00	5.27	13.82
75.00	8.76	23.07
53.00	14.47	33.18
45.00	17.65	37.93
38.00	21.17	42.74
25.00	30.37	54.15
17.00	39.06	63.81
12.0	47.38	71.50
8.00	57.67	78.78
5.00	69.03	84.96
2.00	84.19	92.25
1.00	91.48	95.80

A summary of the results obtained in the pilot testwork is presented in Table 13.1.2.5_5 below:

Table 13.1.2.5_5 – Pilot Testwork Summary

Test	sample	Concentrate Assay (%)			P2O5 recovery (%)	
		P2O5	F2O3	SiO2	FLOTATION	OVERALL
FP-1	OCC+UL2	34.40	5.36	2.47	71.60	64. 10
FP-2		32.70	6.47	2.82	74.90	67.00
FP-3	UD1+UD2	30.45	8.27	4.08	86.50	63.30
FP-4		31.79	7.25	3.25	82.80	60.60

Results shows that the product obtained from flowsheet 2 (OCC+UL2) had a better response in the flotation column circuit. It was obtained a higher grade concentrate with better overall recovery. The conditions used in the flotation test FP-1 is presented in Table 13.1.2.5_6 below:

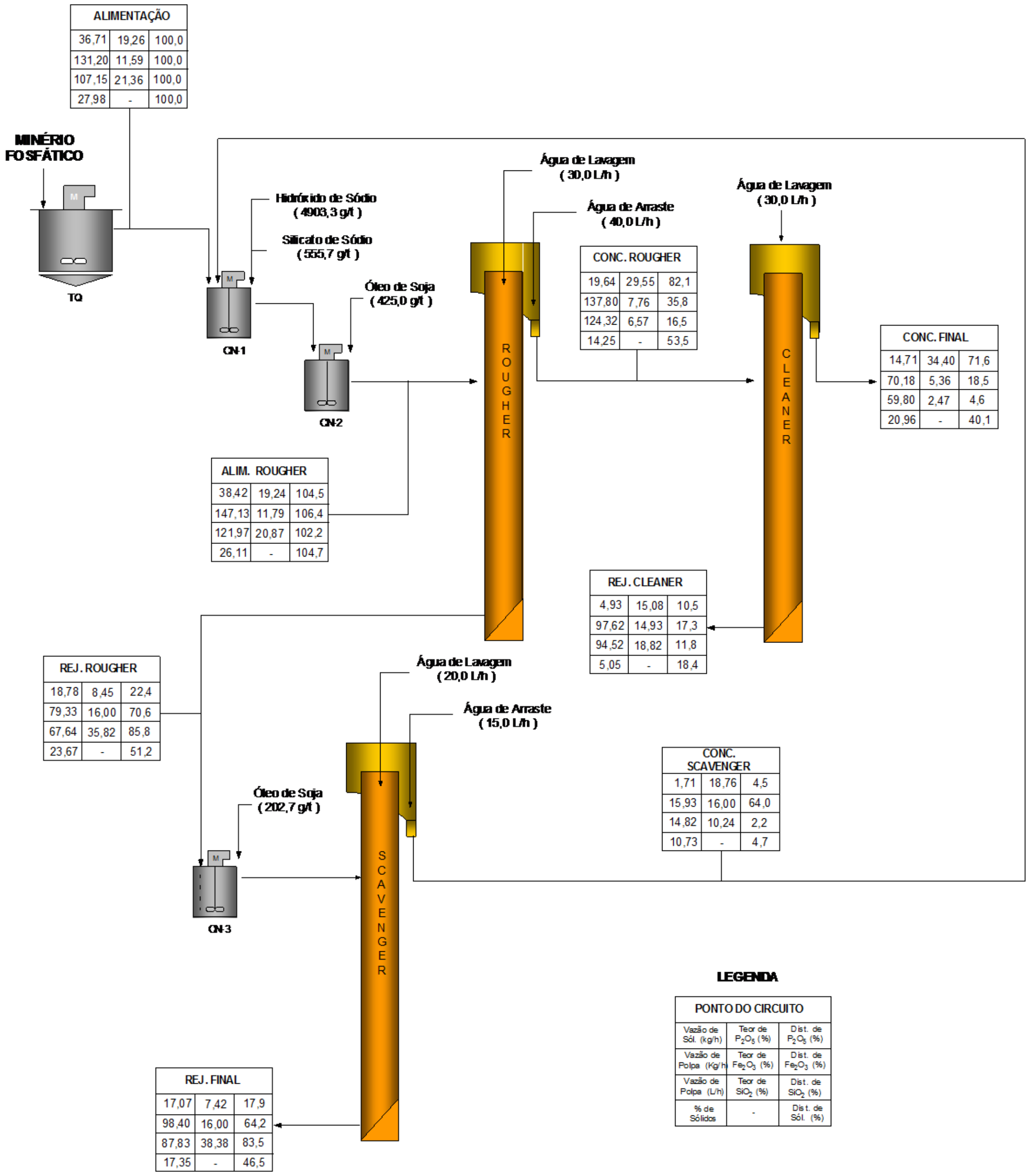
Table 13.1.2.5_6 – FP-1 Flotation Test Conditions

Equipment	**Conditions**	**Test**
		FP-1
Conditioner CN-1	• Residence time (min)	7.90
	• Percent solids	26.10
	• NaOH (g/t)	4903.30
	• Sodium silicate (g/t)	555.70
Conditioner CN-2	• Residence time (min)	5.20
	• Percent solids	26.10
	• Soy bean oil (g/t)	425.00
	• pH slurry	10.00
Conditioner CN-3	• Residence time (min)	5.40
	• Percent solids	23.70
	• Soy bean oil (g/t)	202.70
	• pH slurry	9.90
Rougher column	• Diameter (cm)	10.20
	• Total height (cm)	527.00
	• Superficial velocity of air (cm/s)	1.23
	• Superficial velocity of wash water (cm/s)	0.10

Equipment	Conditions	Test FP-1
	• Superficial velocity of slurry (cm/s)	0.23
	• Feed % solids	26.10
	• Froth height (cm)	62.00
	• Slurry residence time (min)	28.90
	• Transport capacity (g/cm^2.min)	4.00
	• *Hold up* of ar (%)	13.90
	• *Bias*	0.60
Cleaner column	• Diameter (cm)	10.20
	• Total height (cm)	430.00
	• Superficial velocity of air (cm/s)	1.03
	• Superficial velocity of wash water (cm/s)	0.10
	• Superficial velocity of slurry (cm/s)	0.32
	• Feed % solids	14.30
	• Froth height (cm)	60.00
	• Slurry residence time (min)	17.10
	• Transport capacity (g/cm2.min)	3.00
	• Hold up of ar (%)	10.50
	• Bias	0.80
Scavenger column	• Diameter (cm)	10.20
	• Total height (cm)	547.00
	• Superficial velocity of air (cm/s)	1.23
	• Superficial velocity of wash water (cm/s)	0.07
	• Superficial velocity of slurry (cm/s)	0.30
	• Feed % solids	23.60
	• Froth height (cm)	69.00
	• Slurry residence time (min)	22.10
	• Transport capacity (g/cm2.min)	0.40
	• Hold up of ar (%)	16.60
	• Bias	1.30

The material balance obtained is this test is presented below:

Figure 13.1.2.5_3 – Flotation Circuit Material Balance. Test FP-01



The sample used in the pilot plant presented Low Relation CaO/P2O5 indicating the presence of secondary phosphates which are difficult to concentrate by flotation. However the results obtained, concentrate with a grade of 34.4 % P2O5 with an overall recovery of 64% show that this type of ore can be availed.

From the pilot plant results it was possible to confirm the process flowsheet to be used in the industrial plant. The circuit removing the fines before grinding, flowsheet 2, improved metallurgical results as compared with the circuit removing the fines after grinding, flowsheet 1.

Natural fines after desliming, UL2 from flowsheet 2 did not cause major problems in flotation and must be jointed with grinded product, OCC flowsheet 2 to feed flotation circuit.

The pilot plant results were also important to adjust material balance developed in previous phases and to obtain engineering parameters to size equipment.

13.1.2.6 Comminution Studies

To correctly size the mill, MbAC contracted the Technological laboratory facilities of the São Paulo State University to do a characterization of Santana ore including Bond Work Index, Drop Weight Test, Bond Abrasion Index, Batch Grinding Test, Feed size distribution. Table 13.1.2.6_1 below shows samples identification and tests done.

Table 13.1.2.6_1 Sample Identification, Description and Testing.

Sample Identification	Sample Description	Testing				
		WI	AI	DWT	BGT	SDD
RK 72831	Drill Core Sample - Outcrop sample representing one lithotype	Yes	No	No	No	No
RK 72832	Drill Core Sample - Outcrop sample representing one lithotype	Yes	No	No	No	No
RK 72833	Drill Core Sample - Outcrop sample representing one lithotype	Yes	No	No	No	No
Santana A	Bulk sample - Fines - Sample obtained at a trench around DD-004 hole, at 2-5 m depth. Two 100 L drums	No	No	No	Yes	Yes
Santana B	Bulk sample - Relatively hard ore - Sample obtained at a trench around DD-006 hole, at 3 m depth. One 200 L drum	Yes	No	Yes	Yes	No
3-5%	Combines Drill Core Sample: DD-011, 095101 and DD-155	No	Yes	No	No	No
5-7%	Combines Drill Core Sample: DD-253 and DD-263	No	Yes	No	No	No
20-30%	Combines Drill Core Sample: DD-060, DD-061 and DD-103	No	Yes	No	No	No

The work index per sample tested is presented in Table 13.1.2.6_2.

Table 13.1.2.6_2 WI Testing Summary.

Sample	Testing Screen (mm)	Percent Passing at Testing Screen	F_{80} (mm)	P_{80} (mm)	G_{bp} (g/rev)	WI (kWh/st)	WI (kWh/t)	Value
RK 72831	0.149	0.5	1.000	0.096	5.549	4.9	5.4	Low
RK 72832	0.149	0.4376	1.260	0.112	3.040	8.5	9.4	Average
RK 72833	0.149	0.5876	1.000	0.090	8.143	3.4	3.7	Low
Santana B	0.149	0.304	2.300	0.120	5.893	4.7	5.2	Low

Sample RK 72832 resulted in the highest Bond Work Index – WI result (9.4 kWh/t), ranked as an average value. Samples RK 72831, RK 72833 and Santana B indicated WI within the 3.7 – 5.4 kWh/t range, all ranked as low WI values.

Table 13.1.2.6_3 shows the results of high energy breakage (Drop Weigh Test - DWT) and low energy (abrasion) breakage tests as obtained for the Santana B sample. It lists the obtained parameters and respective classification.

Table 13.1.2.6_3 – High Energy and Low Energy Breakage Test Results for Santana B

Sample	Impact				Abrasion		Specific Gravity
	A	*b*	BI	Classification	t_a	Classification	
Santana B	64.87	3.06	198.64	ETB	2.17	EBAb	2.33

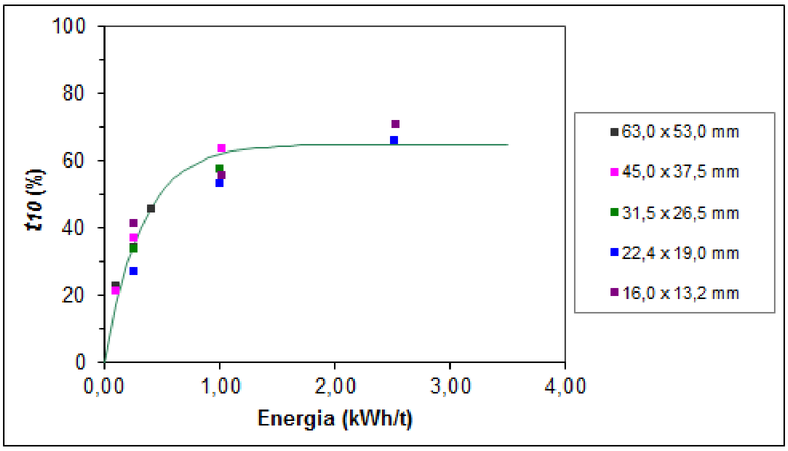
According to results listed on Table 13.1.2.6_3, the Santana B sample indicated an extremely low resistance to high energy breakage (impact) as assessed by Full Drop Weigh Test (DWT). The same sample indicated an extremely low energy breakage as assessed by abrasion test.

The abrasion breakage here represents the actual resistance to breakage by low energy provided by abrasion mechanism, as opposed to abrasiveness assessed by Bond Abrasion Index. The latter representing the characteristics of a given sample to steel wearing.

Table 13.1.2.6_3 indicates that a 2.33 specific gravity (SG) was obtained for the Santana B sample.
The A and b values listed in Table13.1.2.6_3 were determined from mathematic regression among 15 pairs of values of energy applied - Ecs, (kWh/t) and resulting fragmentation - t10 (%), according to the equation shown in section 3.2 of the above referred report.

Figure 13.1.2.6_1 shows the experimental values and parameterized curve calculated for the relationship between energy and fragmentation, for the Santana B sample.

Figure 13.1.2.6_1 – Experimental values and parameterization curve – Santana B sample



Batch grinding tests – BGT were carried out with samples Santana A and Santana B. Each test comprised of four different grinding conditions, according to three grinding periods ie. 5, 10 and 30 minutes

Figure 13.1.2.6_2 shows percent passing at 0.106 mm figures as a function of specific energy (kWh/t), for all four BGT carried out on Santana A. Accordingly, Figure 13.1.2.6_3 shows the same chart based on results obtained for Santana B samples. Both graphs thus represent the relationship between applied energy and resulted breakage.

Figure 13.1.2.6_2 – Percent passing at 0.106 mm versus energy consumption – Sample Santana A

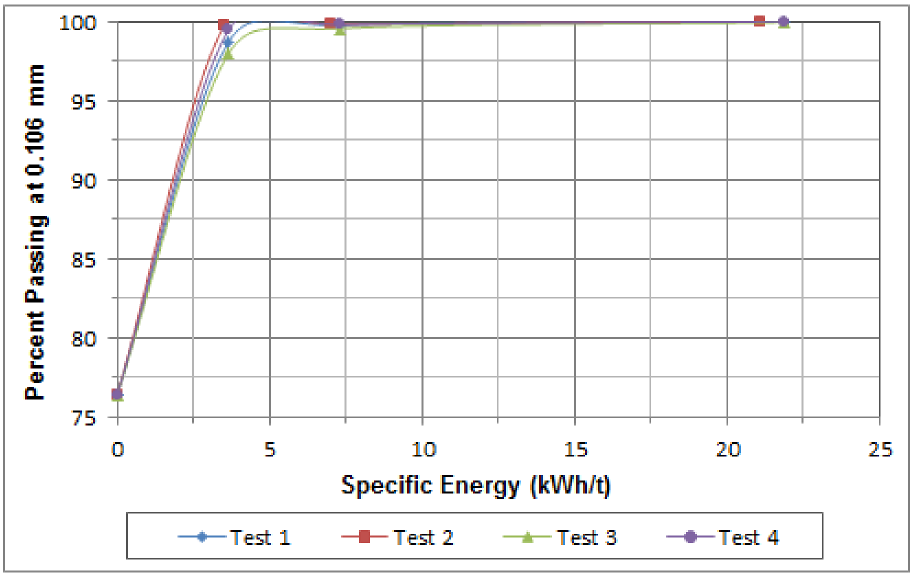
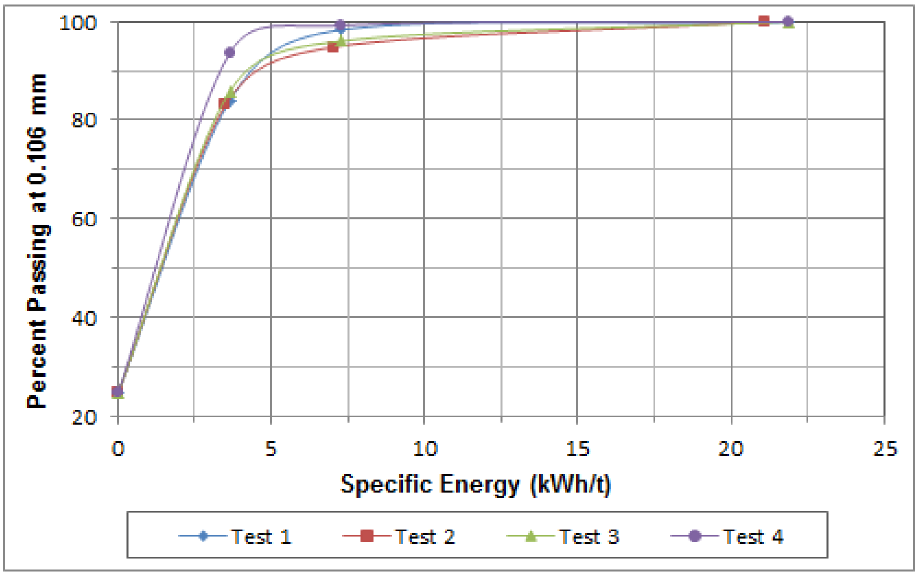


Figure 13.1.2.6_3 – Percent passing at 0.106 mm versus energy consumption – Sample Santana B



Three samples were prepared by MBAC for Bond Abrasion Index – AI determination. Testing procedures were described in detail in the report entitled "*Caracterização Tecnológica de Amostras de Minério de Santana*" written in Portuguese, document identification NT HDA MBAC Santana 02-13 – Rev 0 – 04 April 2013, issued by HDA in April 2013 to MBAC.

Table 13.1.2.6_4 shows identification and description for all three samples.

Table 13.1.2.6_4 - AI Testing Sample Identification

Sample	Description
3-5%	3-5% Holes DD-011, 095101 e 155
5-7%	5-7% Holes DD-253 e 263
20-30%	20-30% Holes DD-060, 061 e 103

Table 13.1.2.6_5 shows the results obtained in all three AI tests carried out on Santana samples.

Table 13.1.2.6_5 - Bond AI Testing Results

Sample	Initial Mass (g)	Final Mass (g)	AI
3-5%	91.441	91.426	0.015
5-7%	91.426	91.384	0.042
20-30%	92.175	92.111	0.064

According to Table 13.1.2.6_5 samples 3-5%, 5-7% e 20-30% indicated AI results respectively of 0.015, 0.042 and 0.062. The results classified all three samples as very low abrasive.

The main aspects associated with Santana samples were their very low resistance to breakage of individual particles, very low resistance to breakage by abrasion, together with their low to average WI and very low abrasive characteristics. One bulk sample (Santana A) indicated a high amount of natural fines (74.2% passing at grinding size) and low energy consumption in Batch Grinding Tests - BGT, while a second one (Santana B) resulted in a relative higher energy consumption in BGT.

The relative soft nature of Santana ore determined the use of sizers in the crushing stage.

The relative high amount of natural fines in Santana ore determined installing a desliming stage prior to the grinding circuit. The separation of natural fines from generated fines would thus not only alleviate the grinding circuit, but also avoid overgrinding of the former, which would in turn reduce its flotation performance.

The combination between high amount of fines and soft nature was considered suitable for adopting a single grinding stage for the Santana project. The coarse fraction from crushing stage would in turn contribute to grinding as grinding media in a Semi-Autogenous – SAG mill. The adopted flow sheet for Santana project was thus a Single Stage SAG milling – SSSAG.

Characterization campaign indicated no potential for critical size build-up in the SAG mill chamber. Accordingly, the high A value associated with high b indicated a low resistance to first fracture, together with a significant generation of fines after breakage. Therefore the relative low resistance to first fracture is a clear indication that there is no potential for critical size build up in the SAG mill chamber.

A primary crushing stage, followed by screening, desliming and grinding was therefore adopted for the Santana industrial beneficiation circuit, here referred as Base Case.

Project criteria for the Santana comminution circuit were as follows:

- R.O.M.: 1,545,480 t per year;

Beneficiation Plant

- Availability: 88.77%;
- Hours available for production: 365 × 24 × 0.8877 = 7776 h/year;
- Nominal feed rate (rounded): 1,545,480 / 7776 = 198.75 t/h;
- Safety factor for designing the screens: 1.20;
- Safety factor for designing the SAG mill: 1.10;
- Grinding circuit product: 90% passing at 0.105 mm;

Ore Characteristics

- Specific gravity: 2.7;
- Breakage characteristics:
- Single particle breakage – Breakage Index (A x b) = 199
- Abrasion breakage - parameter ta = 2.17
- Bond Work Index (BWI) – Ball Mill = 5.2 kWh/t

13.1.2.7 Concentrate Characteristics and Recovery

Based on the information on flotation results and magnetic separation tests, the following concentrate characteristics and average recovery was considered for the process flow sheet and mass balance as shown in Table 13.1.2.7_1.

Table 13.1.2.7_1

Chemical Composition and Average Recovery of a Typical Concentrate. Flourine Analysis by Celqa, Other Elements by Pro-solo

P2O5, %	CaO, %	Fe2O3, %	Al2O3, %	SiO2, %	F	RECOVERY
34.10	43.40	2.40	0.40	11.10	1.70	55.00

13.1.2.8 Concentrate dewatering

Samples from the concentrate obtained in the pilot plant were tested to define the settling rate and the filtering unitary area to size thickener and filter to dewater the concentrate. Two equipment suppliers did the testwork with the following results:

Due to the fine characteristic of the concentrate vacuum filtration is not recommended. The best way to dewater the concentrate is to use a thickener to increase % solids to 50-60% and then filter in a pressure filter to obtain a cake with 17% moisture.

13.2 Single Super Phosphate Production Test

This report contains production tests of single super phosphate (SSP) in a pilot plant, from phosphate rock concentrate produced in Santana do Araguaia, PA, performed in October and November, 2012 at LABFERT, a laboratory accredited by the Brazilian Ministry of Agriculture, located in Uberaba MG.

The concentrate was supplied by the Mineral Research and Process Area from flotation tests performed until July 2012.

The main purposes were to confirm the parameters from the laboratory small scale in the pilot plant scale, producing SSP (single super phosphate) mixing the rock with diluted sulfuric acid and to simulate the granulation of this SSP in pilot plant scale.

Tests contemplated all variables involved such as P_2O_5 concentration, content of Fe2O3, Al2O3, F, rock grain size and sulfuric acid concentration, always guided by parameters similar to other well-known phosphate rocks currently used by the fertilizer industry.

These tests were performed on pilot plant scale, where there are limitations related to process optimization, specially related to temperatures, which are higher in industrial scale. Nevertheless, regarding the main purpose, that is to validate the viability of SSP production using the Santana phosphate concentrate, all tests are representative.

The methodology for both sample preparation and tests execution as well as in physical-chemical analysis follows the standard used by the main SSP production companies and by official laboratory control and the Brazilian Ministry of Agriculture, which is the highest Supervisor and Legislation Controller of fertilizers in the country.

13.2.1 Chemical Reactions in SSP Achieving Process

Besides the calcium phosphates (fluorine-, hydroxy- and carbonate-apatites), in the mineralogical composition of phosphates concentrates there may be presence of secondary phosphates (barium and strontium aluminum-phosphates), carbonates (calcium and/or magnesium), iron and titanium oxides, barium sulphate and minerals with silica (quartz, micas etc.).

This material stability is affected in acid medium, resulting in its partial or total solubilization, with heat release.

During the phosphate rock acidification process in the production of SSP, the fluorine-apatite dissolution begins with sulfuric acid and goes on in a reaction medium containing a mix of sulfuric and phosphoric acids.

Fluorine-apatite solubilization may be viewed in the two basic reactions: (i) first, apatite is attacked by sulfuric acid, originating phosphoric/hydrofluoric acids and calcium sulphate (reaction 2-1); (ii) immediately, the phosphoric acid reacts with the remaining fluorine-apatite producing monocalcium phosphate (FMC) and hydrofluoric acid (reaction 2-2).

Ca10(PO4)6F2 + 10H2SO4 + xH2O → 6H3PO4 + 10CaSO4xH2O + 2HF (2-1)

Ca10(PO4)6F2 + 14H3PO4 + 10H2O → 10Ca(H2PO4)2.H2O + 2HF (2-2)

13.2.2 Experimental Procedure

13.2.2.1 Sample Preparation

Before tests execution, phosphate rock was identified, and the sample was quartered and analyzed all the main constituents and the grain sizes in meshes of 100, 200 and 325#. The necessary amount for each test (around 1.000 grams) was calculated and taken from this sample of analyzed rock.

13.2.2.2 Chemical Analyses

Chemical analyses of the rock and SSP were performed as per the official methods of Brazilian law, as per Normative Instruction n° 28. The contents were:

Table 13.2.2.2_1 – Phosphate Rock Concentrated Chemical Analyses:

P2O5 Total	CaO	MgO	Fe2O3	Al2O3	SiO2	Umidade	K2O	Na2O	MnO	F	SO3
35,36	46,64	0,17	3,87	1,59	5,72	0,02	0,10	0,28	3,23	2,04	ND

SSP: P_2O_5 Total, P_2O_5 soluble in neutral citrate solution of Ammonium + Water, P_2O_5 soluble in water, free P_2O_5 (expressed as H3PO4) e Humidity (Total water).

Sulfuric acid used in tests was PA acid with a 99% concentration.

13.2.2.3 Physical Analyzes

Physical analyses performed were: rock grain sizing at 100, 200 and 325 mesh and temperature measuring during the reaction of the SSP formation.

Table 13.2.2.3_1 – Phosphate Rock Concentrated Grain Sizing:

#100	#200	#325
0.00	0.60	5.00

13.2.2.4 Acidity Ratio (A/R)

The acidity ratio (A/R) is the result of the division of a certain quantity of acid 100% by 100 grams of rock. The higher the rock Calcium content the more sulfuric will be needed for neutralization. Evidently, other compounds also consume the acid such as iron and aluminum acids. In the small scale tests we obtained an A/R of 0.65 g/g which we used in pilot with good results.

13.2.2.5 Sulfuric Acid Concentration

Concentrated sulfuric acid (99%) must be diluted with water to allow higher reactivity in the chemical attack to the crystalline structure of the rock, since the purpose is transforming three-calcium phosphate (water insoluble) in mono and bi-calcium. The value of the ideal acid concentration varies from rock to rock, but usually it is in the range of 63 and 70%.

For these tests the acid concentration was 65%, as set out in the small scale tests.

13.2.2.6 Rock Crushing

Granulometry (grain size distribution) must be controlled to achieve the best reactivity, because the thinnest is the rock the bigger is the specific surface and the bigger the contact between liquid and solid phases, allowing a proper solubilization reaction. Usually it is assumed that rocks with 95% below 200 Mesh and 80% below 325 Mesh are ideal for the best reaction, but eventually a reduction to 95% below 325 Mesh is required. In these tests we used rock with >99% below 200 Mesh.

13.2.2.7 Raw material preparation and dosage

Raw materials, phosphate rock, sulfuric acid and water used in the tests were previously weighed in an accurate analytical scale.

The amount of Rock was around 1.000 grams. It was calculated for each test. Acid and Water varied according to the Acid/Rock (A/R) Ratio and acid concentration established. The amount of rock was added simultaneously with the diluted acid in a mixer.

13.2.2.8 Reaction

The reaction of the rock and acid in the Mixer simulates what happens inside an industrial reactor. All tests were performed in the same way, using a thermometer to record the highest temperature achieved.

Figure 13.2.2.8_1 – Rock and Acid Reaction Simulation

13.2.2.9 Granulation

The granulation of the cured SSP, after 14 days, simulates what happens inside the industrial granulation.

The granulation occurred in a plate granulator where it was added limestone to control the acidity of the GSSP. After the granulator, GSSP was dried in an oven and screened. The fines and the crushed coarse returned to improve the granulation.

Figure 13.2.2.9_1 – Granulation of Cured SSP Simulation

13.2.2.10 Evaluated Parameters

For the acidulation process, a sample of each test after reaction is completed and temperature measured was taken, to allow an instantaneous analysis of the newly produced SSP. New samples are collected after 24 hours, 7 days and 14 days of the performed test, to follow up the evolution of the conversion of insoluble P_2O_5 (three-calcium) into soluble P_2O_5, since reactions keep occurring during this period named "cure" and it is necessary in order to enable SSP to achieve its best physical (humidity, acidity) and chemical (higher content of soluble P_2O_5) characteristics.

During the "cure" the following contents were analyzed: Total P2O5, soluble in Neutral Ammonium Citrate + Water, free Acidity and Humidity. From these values, the conversion of soluble P2O5 over the total P2O5 available was determined. This conversion establishes the P205 performance yield and indicates its evolution along the "cure" period.

For the granulation process the GSSP was analyzed. Total P2O5, soluble in Neutral Ammonium Citrate + Water, free Acidity, Humidity and Hardness.

13.2.3 Results Obtained

13.2.3.1 Physical-Chemical Analysis of the Selected Tests

Table 13.2.3.1_1

Physical-Chemical Analysis of the Selected Tests. Values expressed in %

Test no. 1 - Labfert		A/R ratio = 0,65						
Rock data		Time	Total P2O5	P2O5 CNA	Conversion	P2O5 water	Free acidity	Humidity
Total P2O5	35,36%	Instantaneous	18,00	16,45	91,4%	14,10	11,95	15,00
CaO	46,64%	24 h	18,56	17,15	92,4%	14,95	11,30	11,56
Fe2O3	3,87%	7 days	19,16	18,05	94,2%	16,20	8,33	7,46
Granulometry	< 200#	14 days	19,14	18,39	96,1%	16,25	7,97	7,25
Acid Concentration	65%	Dry base	20,64	19,83	96,1%	17,52	8,59	0,00

Test no. 2 - Labfert		A/R ratio = 0,65						
Rock data		Time	Total P2O5	P2O5 CNA	Conversion	P2O5 water	Free acidity	Humidity
Total P2O5	35,36%	Instantaneous	17,81	15,90	89,3%	13,75	12,72	11,09
CaO	46,64%	24 h	18,13	16,60	91,6%	14,55	9,97	10,02
Fe2O3	3,87%	7 days	18,25	17,50	95,9%	14,50	8,97	8,81
Granulometry	< 200#	14 days	18,49	17,96	97,1%	15,30	7,58	9,34
Acid Concentration	65%	Dry base	20,39	19,81	97,1%	16,88	8,36	10,30

Test no. 3 - Labfert		A/R ratio = 0,65						
Rock data		Time	Total P2O5	P2O5 CNA	Conversion	P2O5 water	Free acidity	Humidity
Total P2O5	35,36%	Instantaneous	19,25	16,85	87,5%	14,65	11,42	9,07
CaO	46,64%	24 h	19,26	18,00	93,5%	15,10	10,55	7,42
Fe2O3	3,87%	7 days	19,30	18,70	96,9%	14,95	9,58	7,17
Granulometry	< 200#	14 days	19,20	18,76	97,7%	15,50	6,61	7,48
Acid Concentration	65%	Dry base	20,75	20,28	97,7%	16,75	7,14	8,08

Granulation Test - Labfert		Added Limestone - 7 and 4%										
SSP (14 days) - Humidity base - 2%						GSSP						
P2O5 Total	P2O5 CNA	P2O5 H2O	CNA / Total	% H3PO4	Humidity	P2O5 Total	P2O5 CNA	P2O5 H2O	CNA / Total	% H3PO4	Humidity	Hardness (7 < x < 8)
20,61	20,07	17,77	97,38	10,88	6,00	19,63	19,06	14,75	97,10	0,90	1,00	1,05
20,32	19,88	16,42	97,83	8,72	9,18	20,00	19,32	15,20	96,60	2,19	0,95	1,35

13.2.4 Results Evaluation

Conversion of soluble phosphorus was higher than 94% after 7 days, achieving up to 96%. This demonstrates that the rock has good reactivity. Rock granulometry was always with 100% of the material below 200 meshes, providing a higher specific contact area.

About the fluorine contents present in the rocks, the results were from 0.7 to 1.6%, which makes possible to state that, in gas scavenging, the system may be simpler to retain this effluent, if new further tests confirm the contents now achieved.

The reaction temperatures were recorded at each test. The results, between 110 and 112 °C, are in the usual range for the SSP reaction.

13.2.5 Conversion of P_2O_5

Conversion of the total P_2O_5 contained in the rock in soluble P_2O_5 in NCA or water present in the SSP produced is consequence of the sulfuric acid chemical attack. The objective is to transform the maximum three-calcium P_2O_5 (insoluble) into mono and dicalcium (soluble). We achieve good conversions for Santana rock over 94% after 7 days. However we observed reduction in the conversion for water soluble P_2O_5.

R_2O_3 (F_2O_3+Al_2O_3) contents present in the rock were high, 5.46%. These compounds are associated with lost in conversion, along the cure time and with temperature effect they are responsible for the soluble phosphorus downgrading. The effects will be increased in industrial scale where the large piles kept the temperature high.

The conversion is also reduced due the acidity neutralization in the granulation process where occur secondary reactions with the soluble phosphate.

13.2.6 Variable Evaluated

The main process variables already mentioned are:

a. Mass balance of acid and rock - amounts in weight of each raw material that will be added for composing the SSP product. They change according to the A/R ratio and to the sulfuric acid concentration that is used.
b. Contents of the main rock constituents - basically, P_2O_5, CaO, Fe_2O_3, Al_2O_3, MgO, SiO_2 and F contained in the rock. The variations of these components defines higher or lower acid consumption, its concentration, final phosphorus solubility in NCA and water, acidity residual, and gases release and effluents to be neutralized.
c. Concentration of sulfuric acid - acid to be used in the reaction must be diluted with water. This dilution is made in order to provide a proper aqueous phase and to allow better acid reactivity while attacking the rock.
d. Granulometry of the rock - phosphate rock must have a grain size profile that guarantees the best acid attack. The lower the granulometry (that is, higher specific area) the more favorable is the reaction.
e. Temperature - the temperatures of raw materials involved in the reaction may also be considered as a reactivity agent. Normally, with temperatures higher than 70 degrees there is more process reactivity (speed increase).

13.2.7 Single Super Phosphate Production Test – Conclusions

Considering the tests performed it may be stated:

a. For rocks with P_2O_5 higher than 35% is possible to produce SSP with minimum 19.8% P_2O_5 soluble in Neutral Citrate of Ammonia (NCA) plus water.
b. With these SSP contents it is possible to add limestone in order to reduce the free acidity in the granulation phase to produce SSP with a minimum content of 19% P_2O_5 soluble in NCA. However the P_2O_5 soluble in water did not achieve the minimum request of 15% for registering the product in the Ministry of Agriculture. Hardness was also below the specification (2 kgf/cm2) adding limestone.
c. It must be considered that SSP production with ammonia addition gives flexibility to the Granulation plant, since it is possible to produce ammonium GSSP with 1% Nitrogen and 19% P_2O_5 soluble in NCA and the ammonium SSP do not have a minimum request for P_2O_5 water solubility for registering the product.
d. Rock with grain size of 95% lower than 200# is proper to process reactivity.
e. With low contents of fluorine in the phosphate rock, it is easier for the gas scavenging system to absorb these effluent emissions.
f. The most adequate concentration of sulfuric acid to guarantee a proper aqueous phase, optimizing the process reactivity is near 65%.
g. The most adequate A/R is 0.65 tonnes of sulfuric acid per tonne of phosphate concentrate.
h. After 7th cure day it is possible to achieve a minimum conversion of 94% in P_2O_5 soluble in NCA.
i. It will be necessary additional tests to adjust the hardness of GSSP over 2 kgf/cm^2. We will consider using ammonia or other kind of additives to improve the Hardness.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

AMSL have estimated the Mineral Resource for the Santana Phosphate Project utilizing recent drilling data completed by MBAC during the 2011 and 2012 field campaigns. The database is current to the 17th of January 2013. The final database used to produce the mineral resource estimate totals 591 drill holes which comprise 314 diamond drill holes and a further 277 reverse circulation drill holes.

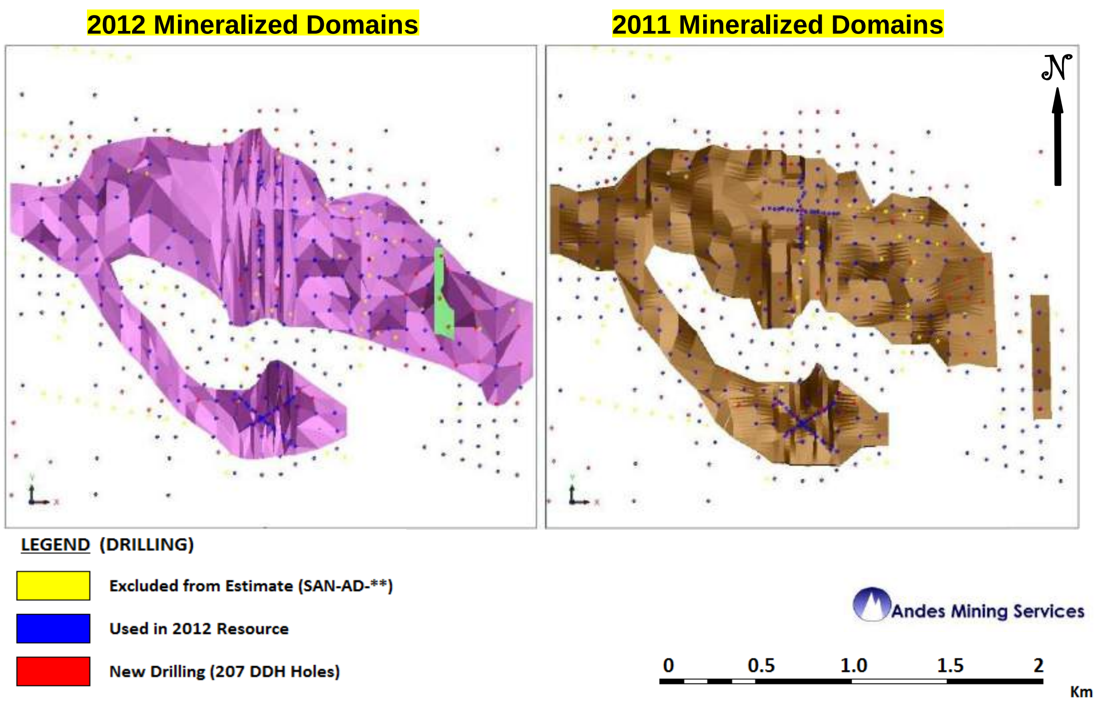
The mineral resource has been estimated by Mr. Bradley Ackroyd, BSc (Geo) (MAIG), Regional Manager and Principal Consultant for AMSL. Mr. Ackroyd is a professional geologist registered as a member of the Australian Institute of Geoscientists (MAIG) and has worked in exploration and development stage projects for metallic and non-metallic mineral deposits throughout the world. The author has been involved in mineral resource estimation work on a continuous basis over the past 10 years. Mr. Ackroyd is an independent Qualified Person as per section 5 of NI 43-101.

The mineral resource estimate is derived from a computerised resource block model. The construction of the block model started with the modeling of 3D wireframe envelopes of the mineralization using drill hole P_2O_5 analytical data and lithological information. Once the modelling had been completed, the analytical data contained within the wireframe solids was normalised to generate fixed length composites. The composite data was used to interpolate the grade of blocks regularly spaced on a defined grid that fills the 3D wireframe solids. The interpolated blocks located below the topographic surface and outside the default waste solid comprise the mineral resources. Individual blocks were then classified based on confidence levels using proximity to composites, composite grade variance and mineralised solids geometry. The 3D wireframe modeling was initially interpreted by MBAC, and then modified by the author based on final assay results and topographic survey data. The block model and mineral resource estimation were conducted by AMSL based on information provided by MBAC.

All grade estimations were completed using Ordinary Kriging (OK) for P_2O_5, Al_2O_3, CaO, Fe_2O_3, MnO_2, SiO_2, LOI and TiO_2. This estimation approach is considered appropriate based on a review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralization, as well as the style of mineralization under consideration.

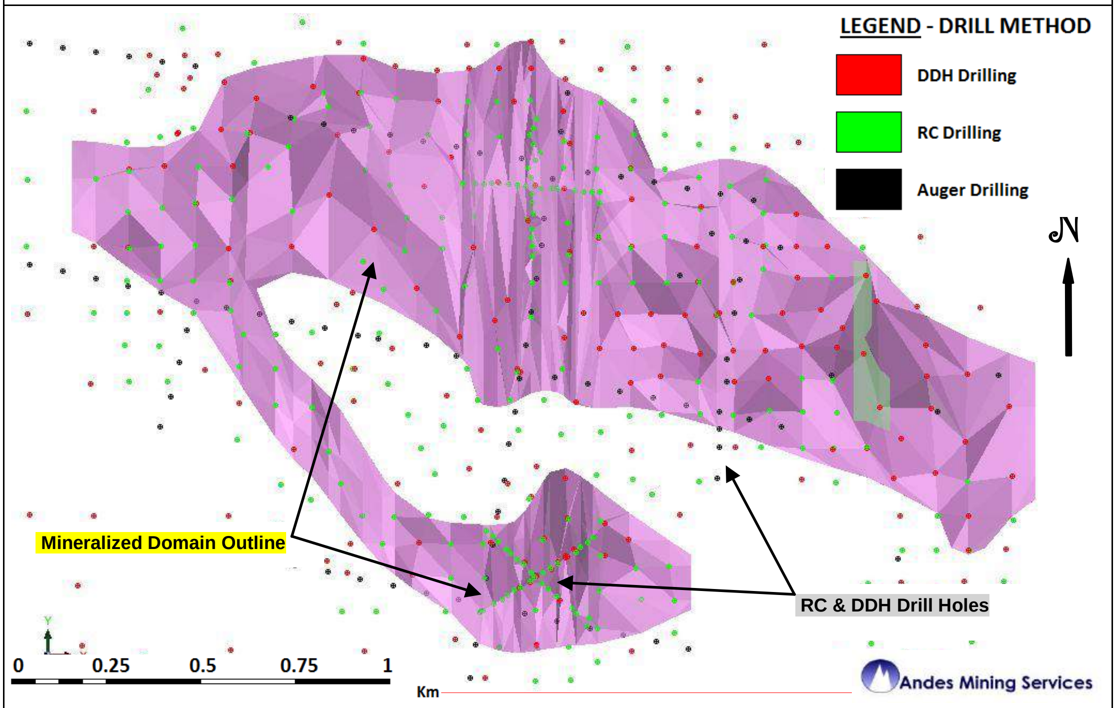
The mineral resource estimation was constrained entirely within soil, saprolite and fresh rock domains. Saprolite is generally well developed across the Santana Phosphate Project area, however is noted to have a highly variable / irregular boundary to the underlying fresh rock. General saprolite thickness varies from 20-60m in depth with significant quantities of P_2O_5 mineralization noted throughout both the soil and saprolite domains. Elevated phosphate grades are generally noted on the boundary between saprolite and fresh rock material. AMSL note a sharp decrease in P_2O_5 grades within the fresh rock domain.

Figure 14.1_1
2011 vs 2012 Wireframe Models - Santana Phosphate Project *(AMS, 28th October 2013)*



The Santana Phosphate Project mineral resource estimate is based on 591 drill holes (31,484.91m) drilled at a nominal spacing of approximately 100m by 100m. A total of 314 diamond drill holes (18,376.91m) and a further 277 reverse circulation drill holes (13,108m) have been completed across the resource area (Figure 14.1_2).

Figure 14.1_2
Plan View of Mineralized Domain Outline with Drillholes by Type *(AMSL, 28th October 2013)*



A number of diamond drill holes have been completed as twin holes to pre-existing reverse circulation drilling in an effort to provide suitable QA/QC comparison test work.

14.2 Database

A spreadsheet named Santana_Database_20121120.xlsx was received from MBAC. The following checks were performed:

- Holes that had no collar data;
- Overlaps in sample intervals;
- Gaps in sample intervals;
- Matching the geological logging length to the hole sample length.

There were no material errors noted by AMSL.

The drillholes were imported into Surpac software and were correlated with the topographic surface provided by MBAC. A poor correlation was noted between collar location points and the topographic surface provided. Given a recent detailed survey of collar locations across the Santana project area which has just been completed, AMSL decided to utilize the collar points to generate a topographic surface across the Santana project area. A detailed airborne topographic surface is due to due to be flown in late March 2013, however this has not been used as part of this mineral resource estimate.

Additional boundary surfaces were generated for the soil / saprolite boundary and also the saprolite / fresh rock boundary based upon detailed logging completed by MBAC geologists.

Statistics for drilling that intersects each weathering domain within the mineralized wireframe are presented below in Table 14.2_1.

Table 14.2_1
Summary Drilling Statistics within Santana Mineralized Domain

Domain	Diamond and Reverse Circulation Drilling Intercepts
Soil	7 DDH Intercepts (32.7m) / 49 RC Intercepts (222.03m)
Saprolite	104 DDH Intercepts (2096.18m) / 136 RC Intercepts (2856.76m)
Fresh	21 DDH Intercepts (199.5m) / 47 RC Intercepts (639.20m)

Auger drillholes have been used to assist with interpretation, but assay results have not been used for the final resource estimate.

Some of the upper portions of the RC and DC holes could not be sampled due to wash out of material by cutting fluids. Based upon a detailed individual review of core trays and available core photos, a decision was made to assign a default average grade to missing sample intervals within each specific domain (soil, saprolite and fresh rock).

Average grades were calculated for each domain and assigned to missing samples as displayed in Table 14.2_2 below.

Table 14.2_2
Default Grades Assigned to Missing Intervals within Mineralized Wireframe

Domain	Element	Total Interval (m)	Total Interval Assayed (m)	Intervals Unsampled (m)	Unsampled (%)	Default Grade (%)
Soil	Al_2O_3	254.73	252.63	2.1	0.83	16.75
	CaO	254.73	252.63	2.1	0.83	8.66
	Fe_2O_3	254.73	252.63	2.1	0.83	25.47
	LOI	254.73	252.63	2.1	0.83	9.88
	MnO_2	254.73	252.63	2.1	0.83	4.35
	P_2O_5	**254.73**	**252.63**	**2.1**	**0.83**	**10.97**
	SiO_2	254.73	252.63	2.1	0.83	19.29
	TiO_2	254.73	252.63	2.1	0.83	1.13
Saprolite	Al_2O_3	4952.94	4653.47	299.47	6.44	6.76
	CaO	4952.94	4653.47	299.47	6.44	20.32
	Fe_2O_3	4952.94	4653.47	299.47	6.44	15.77
	LOI	4952.94	4622.37	330.57	7.15	7.66
	MnO_2	4952.94	4653.47	299.47	6.44	2.55
	P_2O_5	**4952.94**	**4653.47**	**299.47**	**6.44**	**12.74**
	SiO_2	4952.94	4653.47	299.47	6.44	29.76
	TiO_2	4952.94	4653.47	299.47	6.44	0.64
Fresh Rock	Al_2O_3	838.70	744.08	94.62	12.72	0.978
	CaO	838.70	744.08	94.62	12.72	46.13
	Fe_2O_3	838.70	744.08	94.62	12.72	2.95
	LOI	838.70	742.08	96.62	13.02	34.58
	MnO_2	838.70	744.08	94.62	12.72	0.99
	P_2O_5	**838.70**	**744.08**	**94.62**	**12.72**	**3.93**
	SiO_2	838.70	744.08	94.62	12.72	5.24
	TiO_2	838.70	744.08	94.62	12.72	0.09

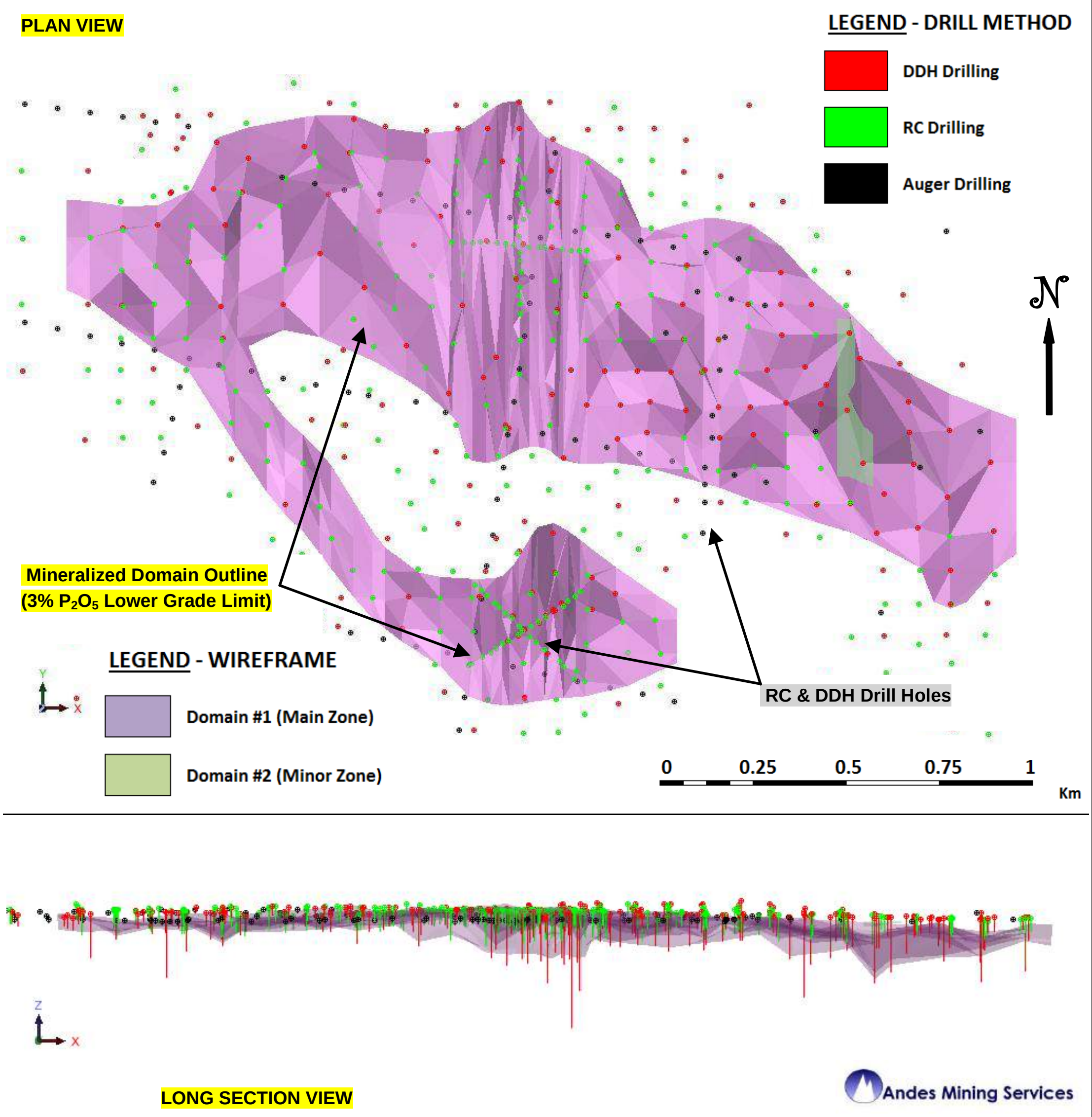
14.3 Geological Modelling

Given the extensive number of drill holes across the Santana Phosphate Project, a detailed geological model has been developed by AMSL as a basis for the mineral resource estimate.

AMSL note the majority of drilling completed by MBAC has been within the saprolite profile where P_2O_5 is enriched, and subsequent geological modelling has constrained mineralized intervals within the saprolite unit.

AMSL have generated a mineralized domain for the Santana Phosphate Project area based upon an interpretation of drillhole data as well as geological mapping data supplied by MBAC personnel. AMSL and MBAC have interpreted a mineralized P_2O_5 domain (which has been utilised for the estimation of P_2O_5, Al_2O_3, CaO, Fe_2O_3, MnO_2, SiO_2, LOI and TiO_2) utilizing a 3% P_2O_5 lower grade limit to guide the wireframing process (Figure 14.3_1).

Figure 14.3_1
Plan / Long Section View of Mineralized Domain Outline with Drillholes *(AMSL, 28th October 2013)*



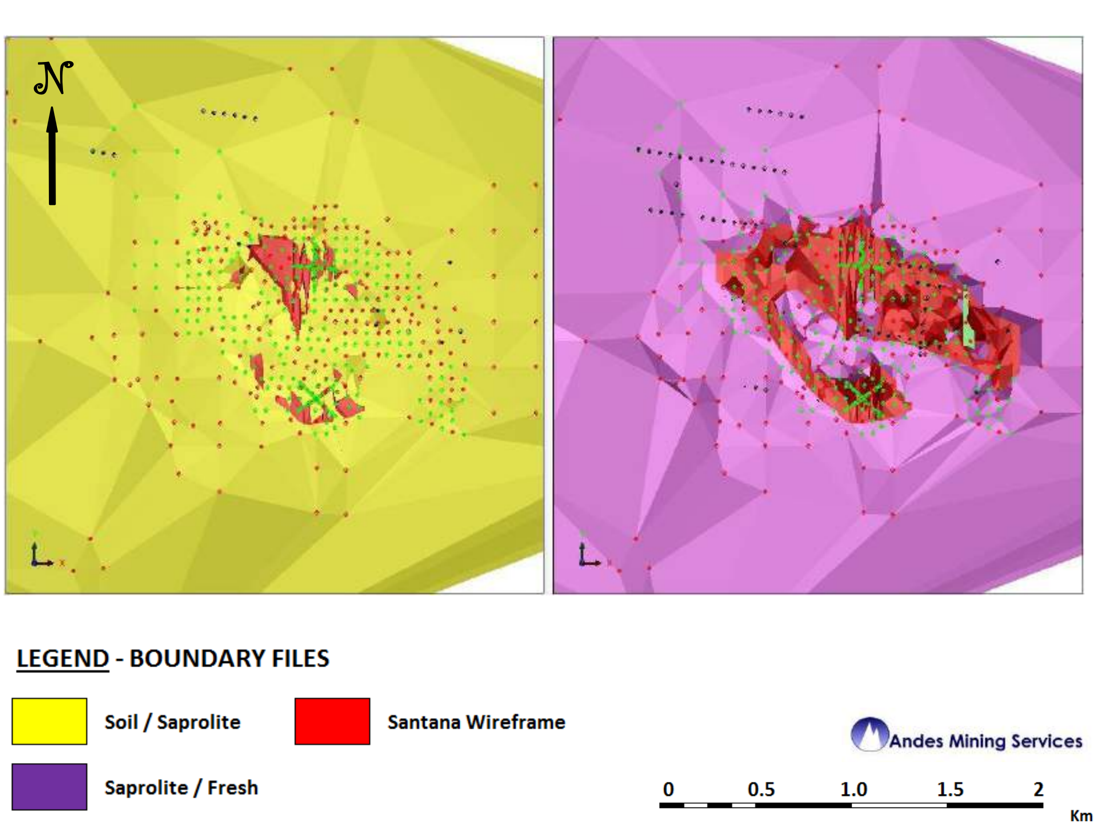
The soil to saprolite and saprolite to fresh rock boundaries were constructed utilizing the geological logging provided by MBAC geologists. The saprolite to fresh rock boundary is an important grade control boundary given that the highest grade P_2O_5 mineralization is tightly constrained within the supergene enriched saprolite. DC logging has been used as the final arbiter, where there may be a logging conflict between RC and DC logging lithologies.

The mineral resource estimate has focused on the main supergene enriched oxide mineralization with 2 flat lying mineralized domains defined using the saprolite and fresh geological boundary along with a 3% P_2O_5 grade cut-off to guide the wireframing process (Figures 14.3_1 and 14.3_3).

Cross sectional strings and wireframes have been created on a variety of 25m to 100m spaced sections by snapping to drillholes.

This 3% P_2O_5 interpretation transgresses into both the soil and fresh rock domains at times as displayed in Figure 14.3_2. A hard boundary has been utilised in these occasions for grade estimation to ensure minimal smearing of grade across defined geological boundaries.

Figure 14.3_2
Boundary DTM's for Constraining Santana Wireframe *(AMSL, 28th October 2013)*

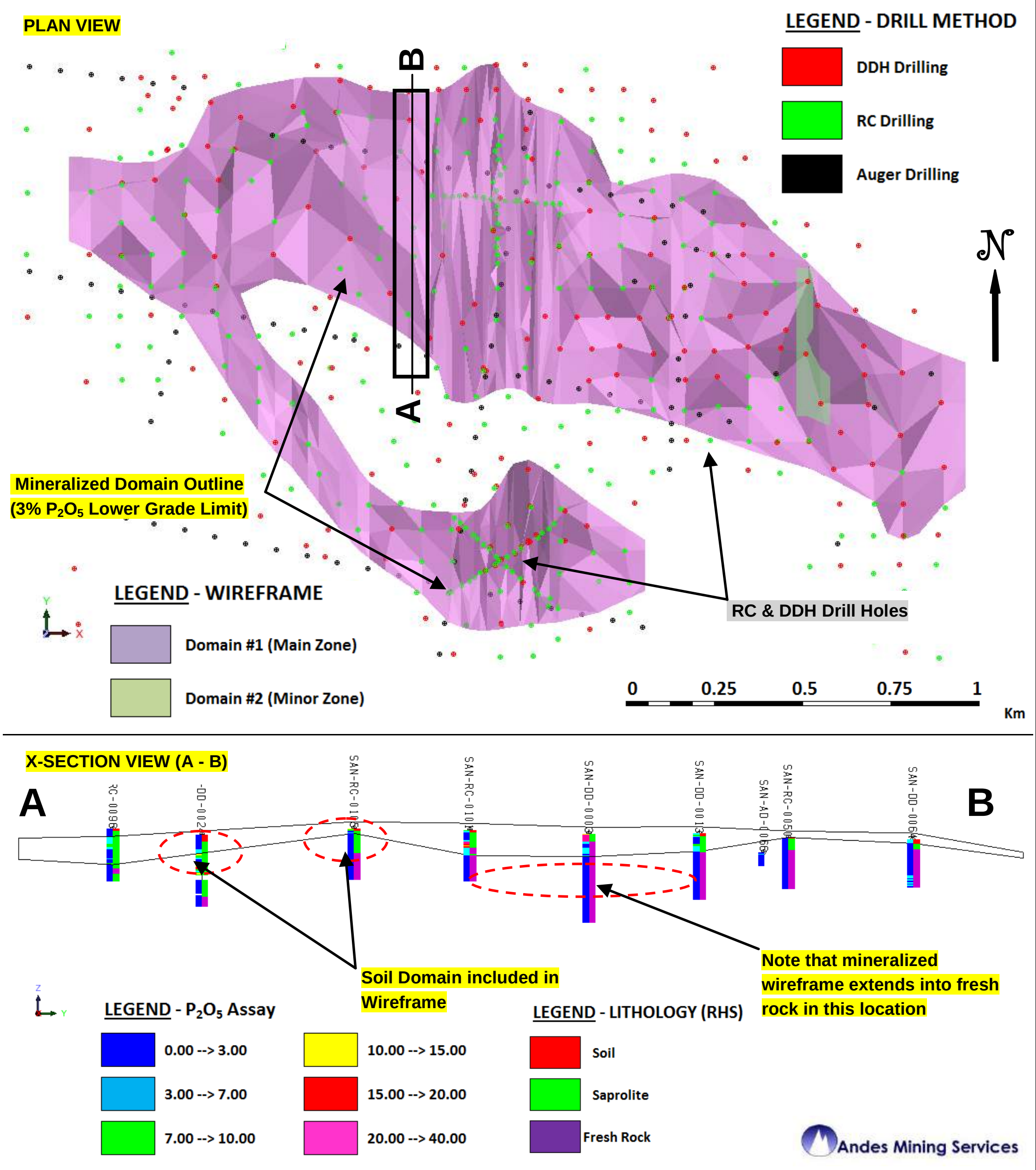


In addition, AMSL have utilized a recent detailed topographical survey across the Santana Phosphate Project area as an upper boundary surface for the wireframes. AMSL utilized collar coordinate data to generate a topographic surface across the Santana project area.

The mineral resource estimate is interpreted with a minimum thickness of 4m and is up to 60m thick in areas that are more deeply weathered. The interpretation shows good three dimensional consistency, and generally, a reasonably consistent thickness from section to section. There is, in general, good correlation between diamond core and reverse circulation drilling.

The interpretation and wireframe models have been developed using Surpac 3D resource modeling software package.

Figure 14.3_3
Plan / Long Section View of Mineralized Domain Outline with Drillholes *(AMSL, 28th October 2013)*



14.4 Sample Selection and Sample Compositing

Samples were selected for the mineral resource estimate from within the wireframes generated from geological and grade based domains. Samples were coded as either mineralized or non-mineralized, and then within soil, saprolite or fresh rock domains.

Selected samples were visually compared back to the interpretations to ensure that the flagging was appropriate.

The common raw sample length in the data is 1m as displayed in Figure 14.4_1.

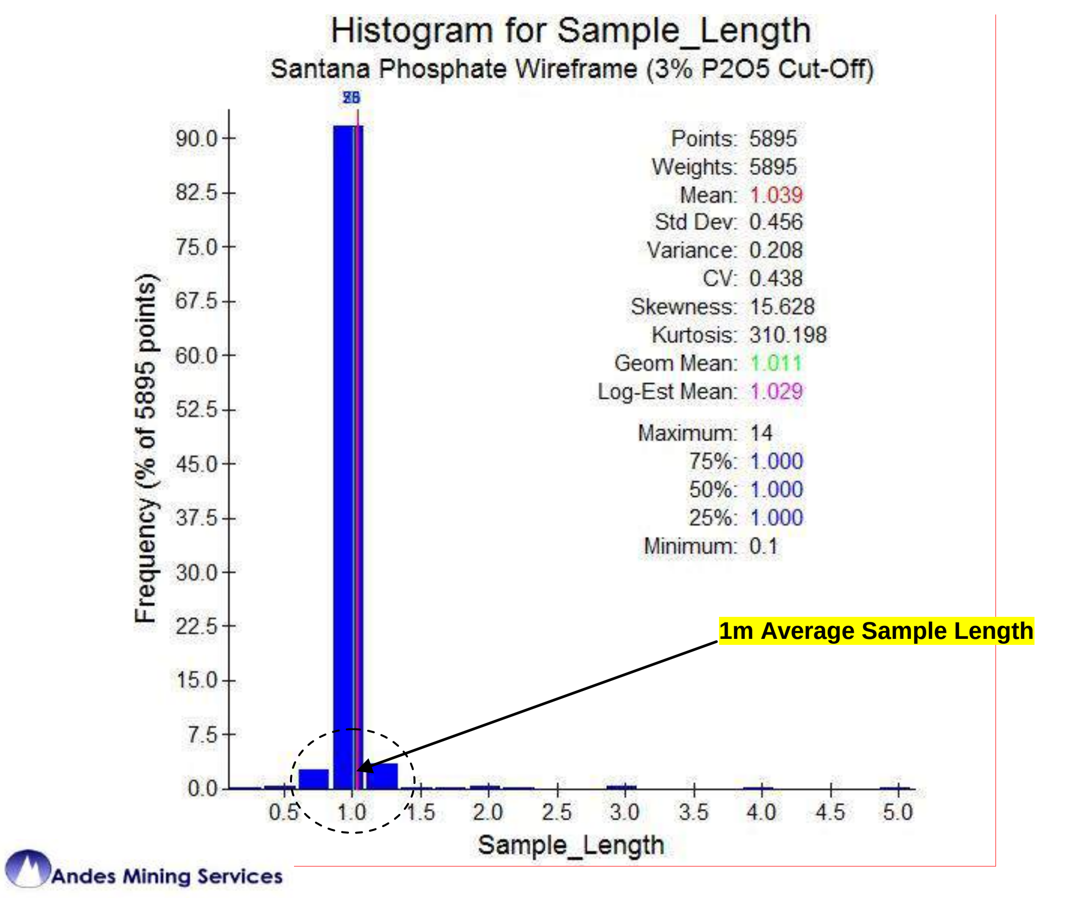
Block model grade interpolation is conducted on composited analytical data. Selected sample intervals were composited downhole to 4m intervals which AMSL considers is the likely mining bench height for a moderate scale open pit mining operation of this geometry and grade variability.

Composites were generated to 4m intervals based on a “best fit” approach and hence no residual samples were discarded. Given the bulk mining approach that will be adopted, this method of generating composites was considered appropriate.

No capping was applied to the assays before compositing.

The composite file was used as the basis for geostatistics and 3D modelling and estimation.

Figure 14.4_1
Average Sample Length Graph – Santana Phosphate Project



14.5 Statistical Analysis

The drill hole database was composited to a 4m down-hole composite interval, with the 4m composite used for all statistical, geostatistical and grade estimation studies.

The statistical analysis was undertaken based on the 4m composites separated into the various geological domains (soil, saprolite and fresh rock). Data was reviewed for all modelled elements. Statistical analysis of 4m composites separated into various geological domain is presented below in Table 14.5_1.

Table 14.5_1
Summary Statistics – 4m Composites within Santana Phosphate (Main Domain)

MAIN DOMAIN	Element	Count	Minimum Value	Maximum Value	Mean	Variance	Std Dev	CV
SOIL	Al_2O_3	70	0.6572	24.775	15.996	35.393	5.949	0.372
	CaO	70	0.4164	51.7778	10.209	163.295	12.779	1.252
	Fe_2O_3	70	1.0856	46.6762	23.912	105.783	10.285	0.430
	LOI	69	2.0922	26.1325	10.171	19.364	4.400	0.433
	MnO_2	70	0.01	10.0332	4.006	6.709	2.590	0.647
	P_2O_5	**70**	**2.7424**	**37.6833**	**11.158**	**68.474**	**8.275**	**0.742**
	SiO_2	70	1.1189	58.8103	19.913	162.020	12.729	0.639
	TiO_2	70	0.0356	1.6476	1.054	0.174	0.417	0.396
SAPROLITE	Al_2O_3	1348	0.019	26.775	6.822	23.216	4.818	0.706
	CaO	1348	0.1	54.14	20.172	202.578	14.233	0.706
	Fe_2O_3	1348	0.228	69.4858	15.823	103.570	10.177	0643
	LOI	1340	0.9033	41.7049	7.771	60.146	7.755	0.998
	MnO_2	1348	0.1284	28.96	2.510	6.854	2.618	1.043
	P_2O_5	**1348**	**0.219**	**37.897**	**12.546**	**78.371**	**8.853**	**0.706**
	SiO_2	1348	0.16	82.2	29.893	300.841	17.345	0.580
	TiO_2	1348	0.005	3.1404	0.641	0.180	0.424	0.662
FRESH ROCK	Al_2O_3	217	0.0069	15.075	1.129	3.960	1.990	1.762
	CaO	217	2.8925	54.74	45.659	60.469	7.776	0.170
	Fe_2O_3	217	0.18	20.8125	3.269	10.491	3.239	0.991
	LOI	217	5.5637	41.0359	34.003	34.872	5.905	0.174
	MnO_2	217	0.13	3.627	1.036	0.310	0.557	0.537
	P_2O_5	**217**	**0.3715**	**15.0701**	**4.198**	**4.760**	**2.182**	**0.520**
	SiO_2	217	0.088	45.825	5.474	56.295	7.503	1.371
	TiO_2	217	0.005	0.9325	0.101	0.019	0.138	1.376

Individual top cuts applied to each element are presented below in Table 14.5_2. No top-cuts were applied to the minor mineralized domain as highlighted previously in Figures 14.3_1 and 14.3_3.

No top cut has been applied to P_2O_5 within the soil and saprolite domains, however a top-cup of 13.5% was applied to P_2O_5 outliers within the fresh rock domain. Various top cuts have been applied to all other elements which include Al_2O_3, CaO, Fe_2O_3, LOI, MnO_2, P_2O_5, SiO_2 and TiO_2 based on a review of histogram and log probability curves. Main domain saprolite plots are shown below in Figures 14.5_1 to 14.5_8.

Table 14.5_2 Top Cuts Applied to 4m Composite Data for Santana Phosphate (Main Domain)						
MAIN DOMAIN	**Element**	**Count**	**Minimum Value**	**Maximum Value**	**Mean**	**Top Cut Applied (%)**
SOIL	**Al_2O_3**	70	0.6572	24.775	15.996	No Cut
	CaO	70	0.4164	51.7778	10.209	No Cut
	Fe_2O_3	70	1.0856	46.6762	23.912	No Cut
	LOI	69	2.0922	26.1325	10.171	**15.5 (4 comps edited)**
	MnO_2	70	0.01	10.0332	4.006	No Cut
	P_2O_5	**70**	**2.7424**	**37.6833**	**11.158**	No Cut
	SiO_2	70	1.1189	58.8103	19.913	No Cut
	TiO_2	70	0.0356	1.6476	1.054	No Cut
SAPROLITE	**Al_2O_3**	1348	0.019	26.775	6.822	No Cut
	CaO	1348	0.1	54.14	20.172	No Cut
	Fe_2O_3	1348	0.228	69.4858	15.823	**60.0 (4 comps edited)**
	LOI	1340	0.9033	41.7049	7.771	No Cut
	MnO_2	1348	0.1284	28.96	2.510	**23.5 (2 comps edited)**
	P_2O_5	**1348**	**0.219**	**37.897**	**12.546**	No Cut
	SiO_2	1348	0.16	82.2	29.893	No Cut
	TiO_2	1348	0.005	3.1404	0.641	**2.0 (8 comps edited)**
FRESH ROCK	**Al_2O_3**	217	0.0069	15.075	1.129	**9.0 (2 comps edited)**
	CaO	217	2.8925	54.74	45.659	No Cut
	Fe_2O_3	217	0.18	20.8125	3.269	**14.5 (5 comps edited)**
	LOI	217	5.5637	41.0359	34.003	No Cut
	MnO_2	217	0.13	3.627	1.036	**3.0 (3 comps edited)**
	P_2O_5	**217**	**0.3715**	**15.0701**	**4.198**	**13.5 (2 comps edited)**
	SiO_2	217	0.088	45.825	5.474	**29.0 (9 comps edited)**
	TiO_2	217	0.005	0.9325	0.101	**0.65 (1 comps edited)**

Figure 14.5_1
Histogram and Log Probability Plot – Al_2O_3 Composite Data (4m)

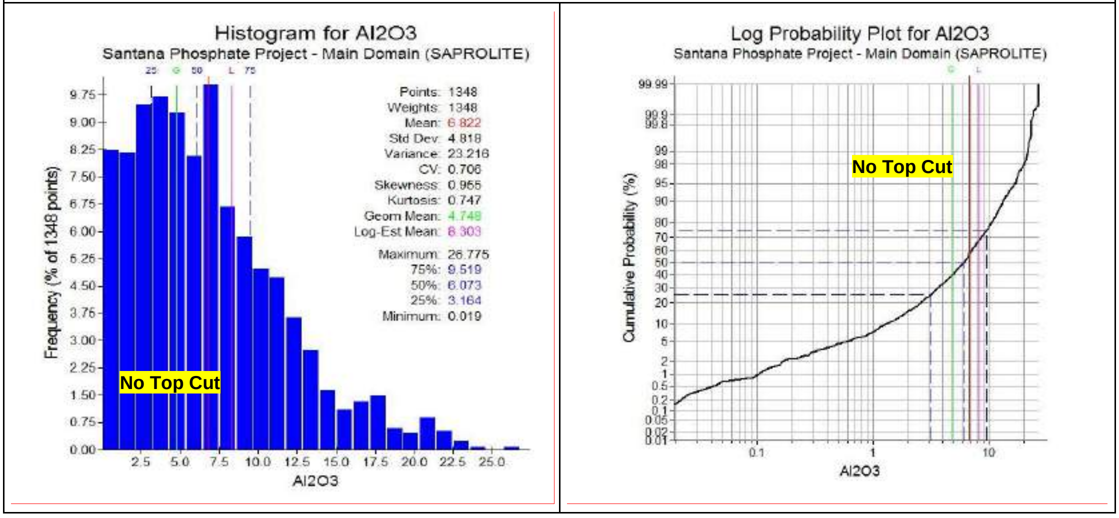


Figure 14.5_2
Histogram and Log Probability Plot – CaO Composite Data (4m)

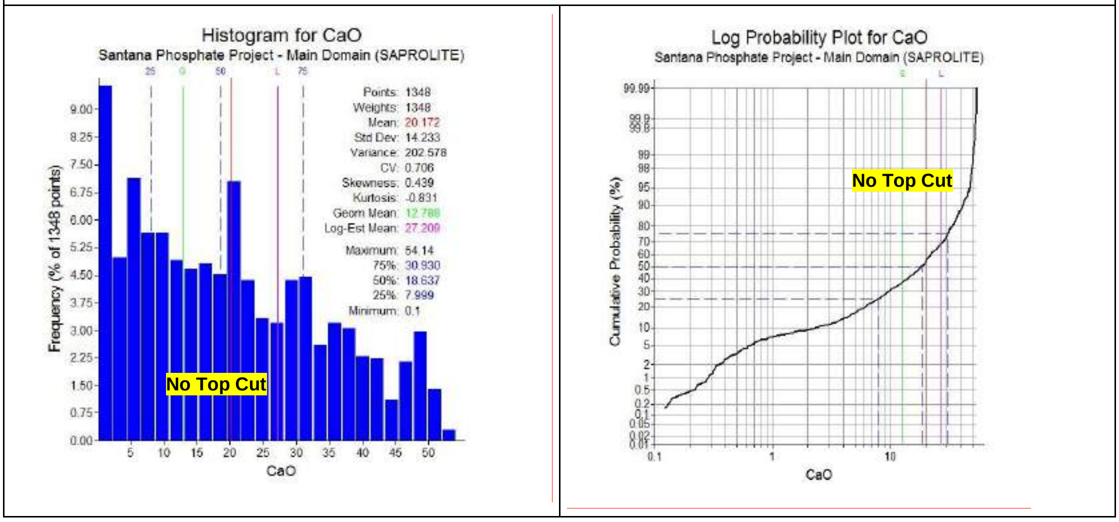


Figure 14.5_3
Histogram and Log Probability Plot – Fe_2O_3 Composite Data (4m)

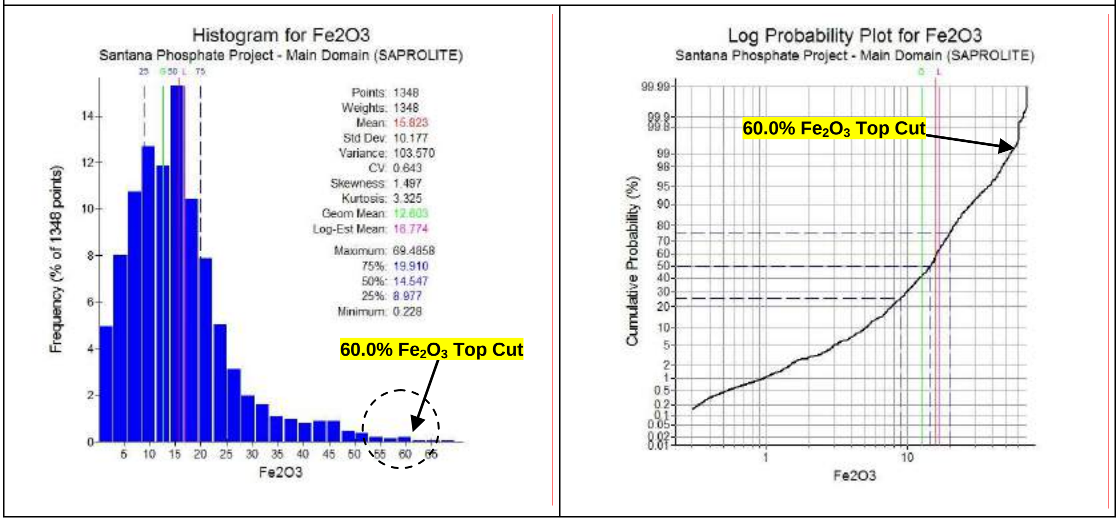
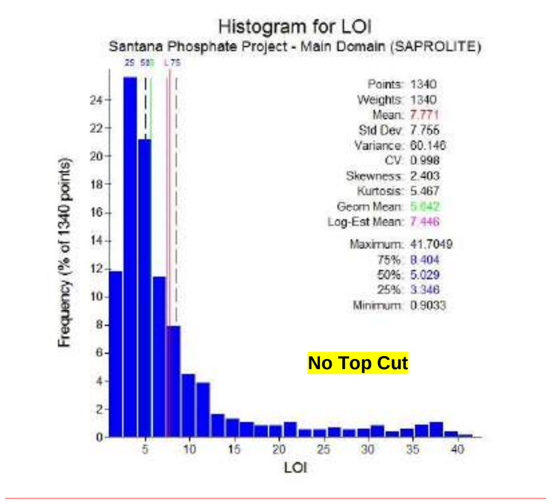


Figure 14.5_4
Histogram and Log Probability Plot – LOI Composite Data (4m)



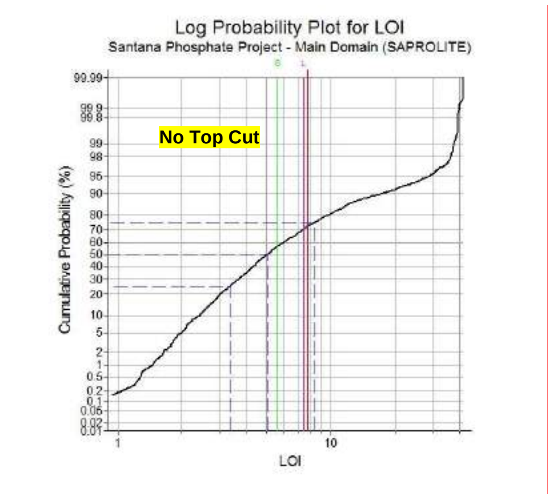
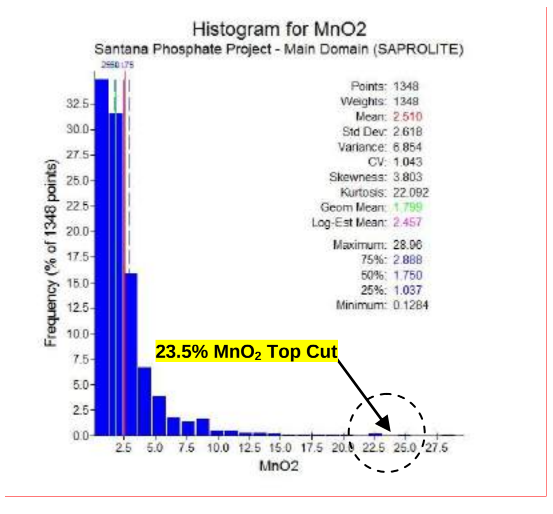


Figure 14.5_5
Histogram and Log Probability Plot – MnO_2 Composite Data (4m)



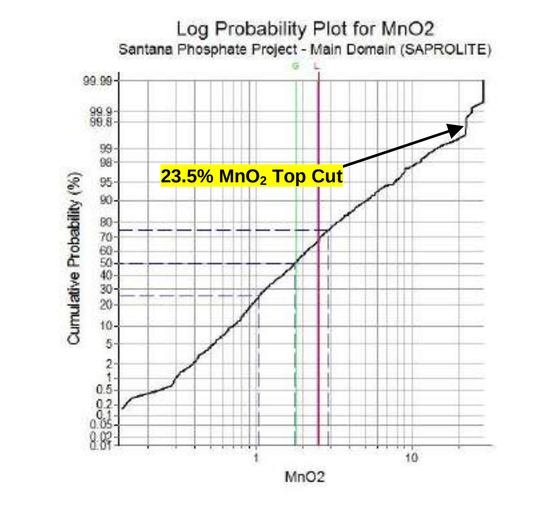


Figure 14.5_6
Histogram and Log Probability Plot – P_2O_5 Composite Data (4m)

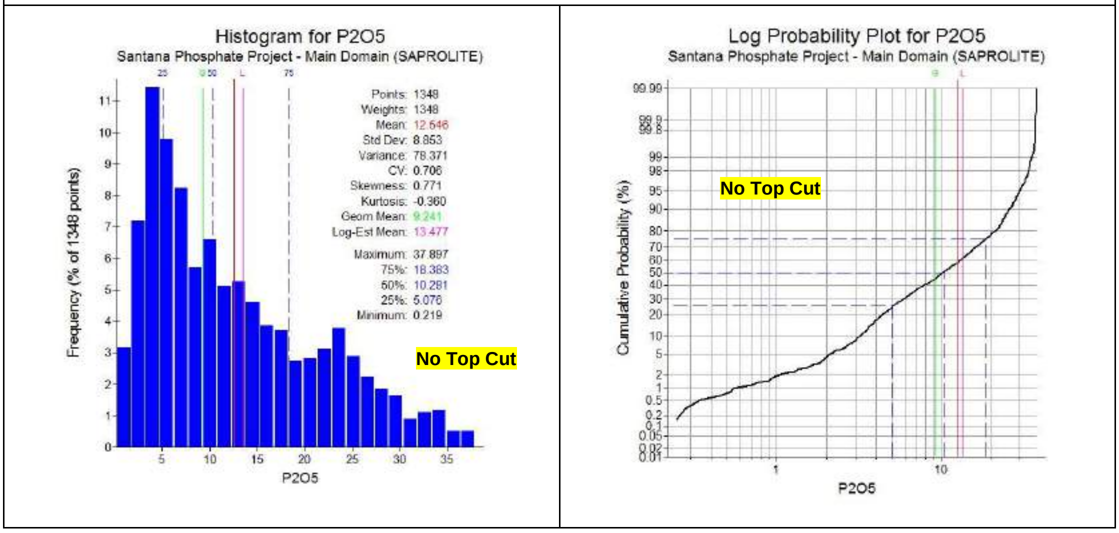


Figure 14.5_7
Histogram and Log Probability Plot – SiO_2 Composite Data (4m)

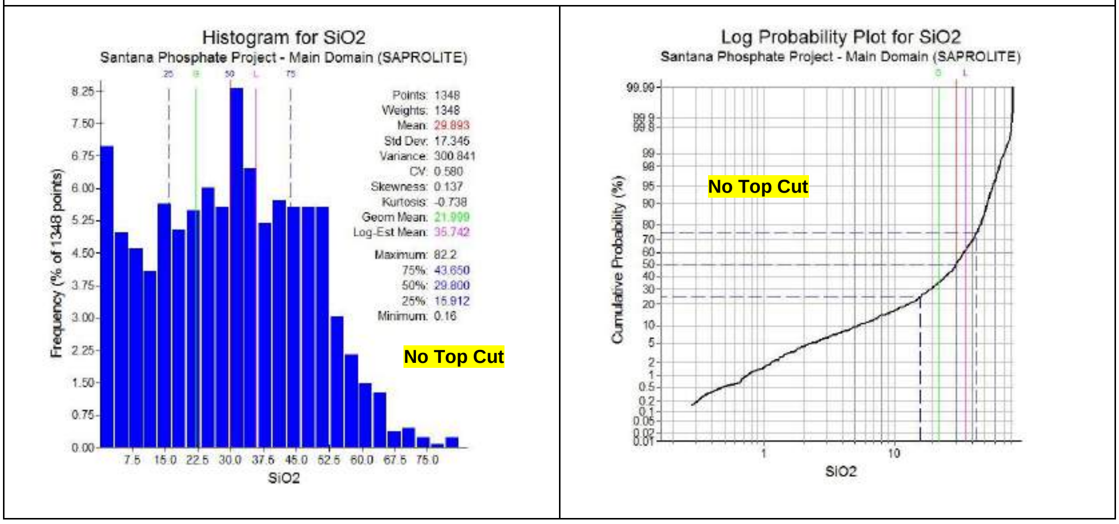
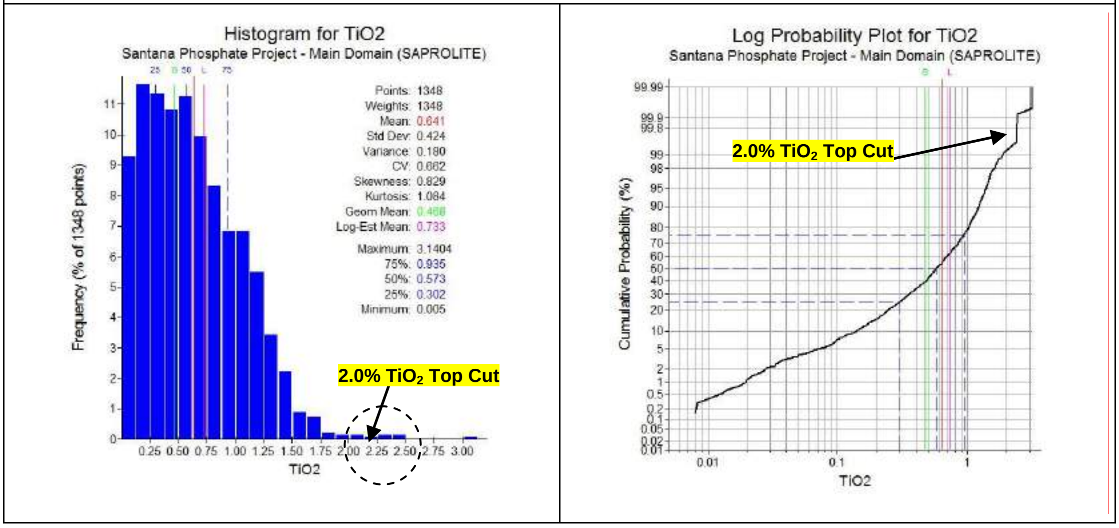


Figure 14.5_8
Histogram and Log Probability Plot – TiO_2 Composite Data (4m)



14.6 Variography

The variography was based on the 4m uncut composited data coded within the mineralization interpretation. The spatial continuity of composite grades for P_2O_5 as well as Al_2O_3, CaO, Fe_2O_3, LOI, MnO_2, SiO_2 and TiO_2 were assessed by means of a variety of types of variograms.

Three structure spherical models were used to model the variograms for P_2O_5. The other elements displayed poorly structured variograms and the variograms from P_2O_5 were utilised for Al_2O_3, CaO, Fe_2O_3, LOI, MnO_2, SiO_2 and TiO_2 during grade interpolation.

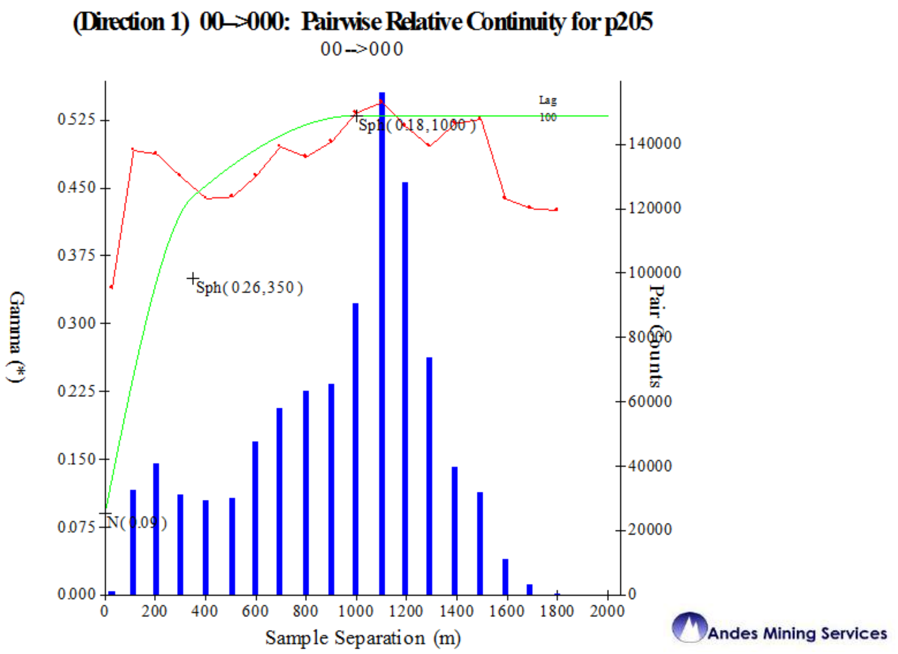
Normal variograms were not stable. Therefore pairwise relative variograms were computed and modelled for the 4m composites. Variogram fans were analysed for P_2O_5 in order to identify potential anisotropies in the grade continuity within the modelled mineralised envelope. The variogram parameters determined for P_2O_5 were applied to all other elements, with variogram orientations and anisotropies reflecting obvious geological and visible data trends.

Table 14.6_1 below presents the variogram model of P_2O_5 and Figure 14.6_1 shows the pairwise relative variogram graph for P_2O_5.

Table 14.6_1
Variogram Model of P_2O_5 Grade for 4m Composite

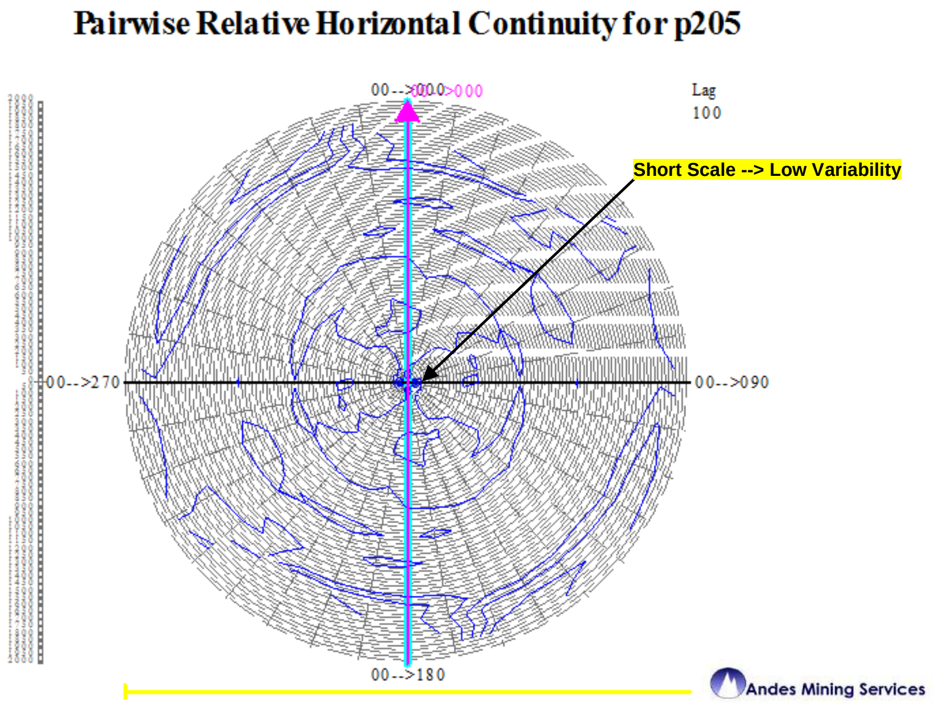
Nugget Effect	First Spherical Variogram Component							Second Spherical Variogram Component						
	Sill (C)	Ranges (in metres)			Orientation (in degrees)			Sill (C)	Ranges (in metres)			Orientation (in degrees)		
		Max	Interm	Min	Azi	Dip	Spin		Max	Interm	Min	Azi	Dip	Spin
0.09 17%	0.26 49%	350	350	87.5	0	0	0	0.18 34%	1000	1000	250	0	0	0

Figure 14.6_1
Pairwise Relative Variogram Graph for P_2O_5



The P_2O_5 grade variograms had a well defined nugget variance of 17%. Fifty percent of the variance is taken up within the first 350m, with a total range of 1,000m, as displayed in Figure 14.6_1. The best continuity in the analytical data, was observed on a horizontal plane (azimuth 0° and dip at 0°). The nugget effect is relatively low (17%) which is normal for deposits of this style and nature.

Figure 14.6_2
Horizontal Continuity Plot for P_2O_5



14.7 Block Model Development

A three-dimensional block model was defined for the Santana Phosphate Project, covering the interpreted P_2O_5 mineralized domains. A parent block size of 25 mE x 25 mN x 4 mRL has been used with standard sub-blocking to 3.125 mE x 3.125 mN x 0.5 mRL cell size to improve volume representation of the interpreted wireframe models. Estimation was only carried out into parent blocks, with sub-blocks assigned the parent cell grade estimates.

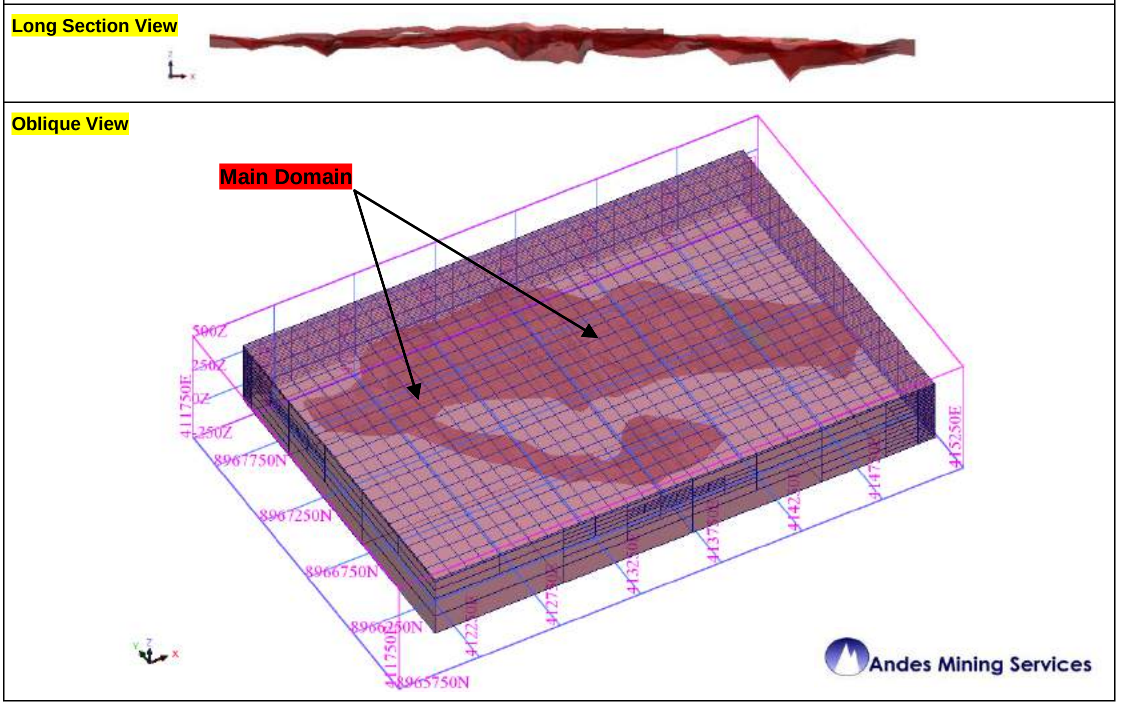
This parent cell size was chosen as it adequately reflects the drilling density and likely mining bench height. All wireframes were checked visually to ensure that there was adequate filling with blocks. The mineralization domain was projected above the topographic surface to ensure that there were no edge effects in volume filling and then it was cut with the surface topography.

Table 14.7_1 below shows the summary of the 3D block model created for the Santana Phosphate Project. A visual review of the wireframe solids and the block model indicates robust flagging of the block model (Figure 14.7_1).

Table 14.7_1
Block Model Summary – Santana Phosphate Project

	North (Y)	East (X)	Elevation (Z)
Minimum Coordinates	8965800	412000	0
Maximum Coordinates	8967900	415100	400
User Block Size	25	25	4
Sub-Block Size	3.125	3.125	0.5
Rotation	0	0	0
No. Blocks	84	124	100

Figure 14.7_1
Block Model for the Santana Phosphate Project



AMSL have also completed basic volume checks upon the mineralized wireframe domain and the block model reported volume, with results presented below in Table 14.7_2.

Table 14.7_2
Volume Check - Mineralized Wireframe vs Block Model Ore Domain

	Reported Volume (bcm)		
	Ore Wireframe	**Block Model Domain**	**% Difference**
Santana Project (Main Domain)	49,457,061	49,462,827	0.01
Santana Project (Minor Domain)	221,197	223,994	1.25
Total	49,678,258	49,686,821	0.02

The attributes coded into the block models include all elements (Al_2O_3, CaO, Fe_2O_3, LOI, MnO_2, P_2O_5, SiO_2 and TiO_2), density, topography, weathering, resource category, domain code, as well as a number of kriging attributes and sample variance data.

A full list of attributes coded to the model is listed below in Table 14.7_3.

Table 14.7_3
Attributes Assigned to 3D Model – Santana Phosphate Project ***(AMS, 28th October 2013)***

Attribute Name	Type	Decimal	Background	Description
al2o3	Real	6	0	Aluminium (Oxide)
avg_dist_p2o5	Real	2	-99	Average Distance to Find Samples
cao	Real	6	0	Calcium (Oxide)
density	Real	6	2.5	Density Value (Assigned)
domain	Character	-	Waste	Ore_Main or Ore_Minor
fe2o3	Real	6	0	Iron (Oxide)
kg_var_p2o5	Real	2	-99	Kriging Variance for Block Estimate
loi	Real	6	0	Loss On Ignition (LOI)
min_dist_p2o5	Real	2	-99	Minimum Distance to Find Samples
mno2	Real	6	0	Manganese (Oxide)
num_samp_p2o5	Integer	-	-99	Number of Samples for Estimate
p2o5	Real	6	0	Phosphorous (Oxide)
pass_no	Integer	-	0	Pass Number
pod	Character	-	999	Pod Number (Obj 8 or Obj 5)
rescat	Character	-	None	Measured, Indicated, or Inferred
sio2	Real	6	0	Silica (Oxide)
tenement_status	Character	-	Outside	Either Inside or Outside
tio2	Real	6	0	Titanium (Oxide)
topo	Integer	-	0	Assign 1 if Underneath Topography
weathering	Character	-	None	Either Fresh or Saprolite or Soil

14.8 Grade Estimation

The grade interpolation for all elements for the Santana phosphate mineral resource block model was estimated using Ordinary Kriging (OK). Anisotropic search ellipsoids were selected for the grade interpolation process based on the analysis of the spatial continuity of Al_2O_3, CaO, Fe_2O_3, LOI, MnO_2, P_2O_5, SiO_2 and TiO_2 grades using variography and on the general geometry of the modelled mineralized saprolite envelope. Limits are set for the minimum and maximum number of composites used per interpolation pass, and restriction are applied on the maximum number of composites used from each hole.

The interpolation process was conducted using 3 successive passes with relaxed search conditions from one pass to the next until all blocks were interpolated. The orientation of the search ellipsoids, which is identical for each interpolation pass, is 0° azimuth, 0° dip and 0° plunge consistent with a relatively uniform, sub-horizontal (flat lying) mineralized domain (Table 14.8_1).

In the first pass, the search ellipsoid distance was 200m (long axis) by 200m (intermediate axis) by 10 m (short axis). Search conditions were defined with a minimum of 10 composites and a maximum of 20 composites with a maximum of 2 composites selected from each hole. For the second pass, the search distance was increased to 400m (long axis) by 400m (intermediate axis) by 10m (short axis) and composites selection criteria were kept the same as the first pass, however with a lowering of the minimum number of samples required to make an estimate set at 4. Finally, the search distance of the third pass was increased to 800m (long axis) by 800m (intermediate axis) by 40m (short axis) and again the same composites selection criteria were applied with a lowering of the minimum number of samples to 2.

All blocks within the Santana phosphate mineralized domain were estimated as part of the three pass estimate.

Table 14.8_1 outlines the search direction and parameters used for the 3 pass interpolation, while Table 14.8_2 highlights the percentage of blocks estimated within each of the estimation passes completed.

Table 14.8_1
Summary of Search Direction and Parameters for 3 Pass Interpolation

Zone	Search Directions			
	Strike (degrees)			Dip / Dip Direction (degrees)
P_2O_5 Ore Domain (Main and Minor)	**000°**			**0° / 000°**
	1st Pass	**2nd Pass**	**3rd Pass**	**Discretization**
X	200m	400m	800m	4
Y	200m	400m	800m	4
Z	10m	10m	40m	2
MIN SAMPLE	10	4	2	-
MAX SAMPLE	20	20	20	-

Table 14.8_2 Percentage of Blocks Estimated for 3 Pass Interpolation	
Pass #	**Percentage of Blocks Estimated**
1st Pass	40.88%
2nd Pass	49.46%
3rd Pass	9.66%

The search ellipse was configured to match the main mineralization direction, which is sub-horizontal given the geological understanding at the time of this mineralization. Figure 14.8_1 shows the interpolation results visually for the three respective passes across the Santana phosphate project area.

Figure 14.8_1
Santana Block Model – Coded by Pass Number (ALL Domains) *(AMSL, 28th October 2013)*

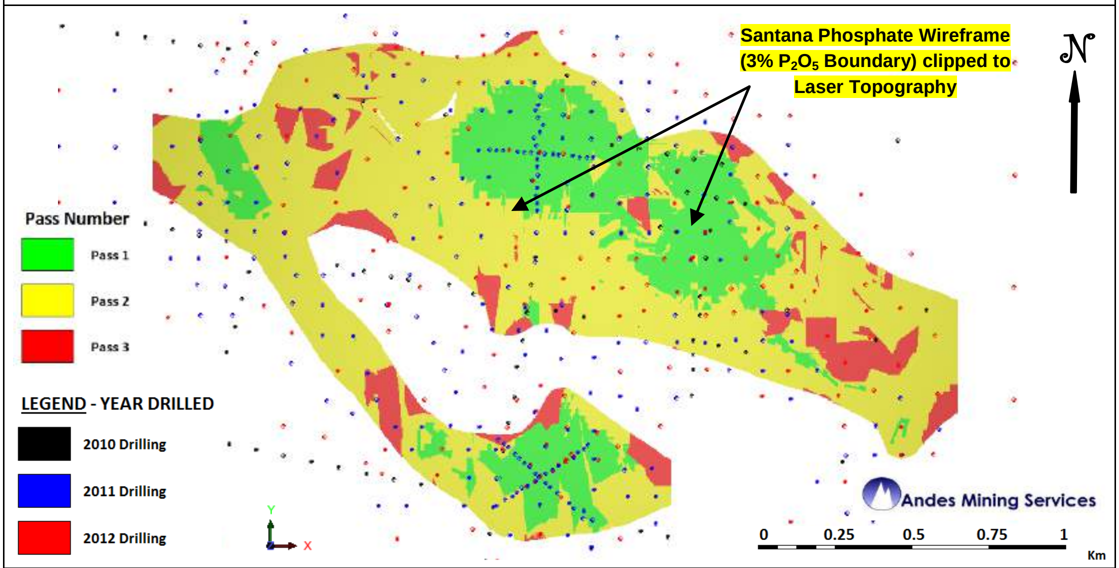
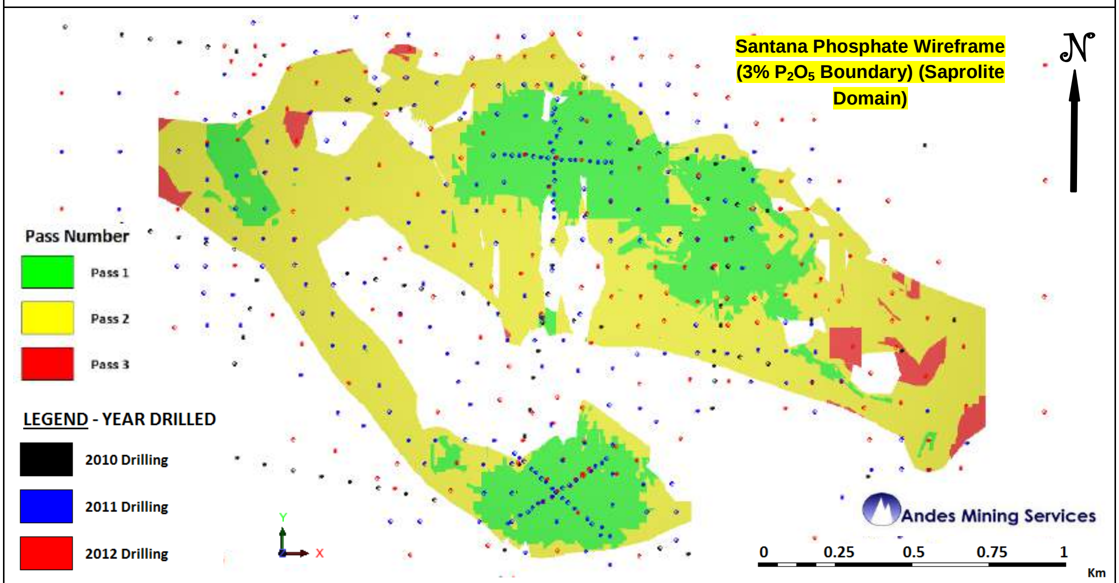


Figure 14.8_2
Santana Block Model – Coded by Pass Number (Saprolite Domain) *(AMS, 28th October 2013)*



Figures 14.8_3 and 14.8_4 below illustrates grade variations across the block model (mineralized domain) for P_2O_5.

Figure 14.8_3
Santana Block Model – Coded by P_2O_5 Grades (ALL Domains) *(AMSL, 28th October 2013)*

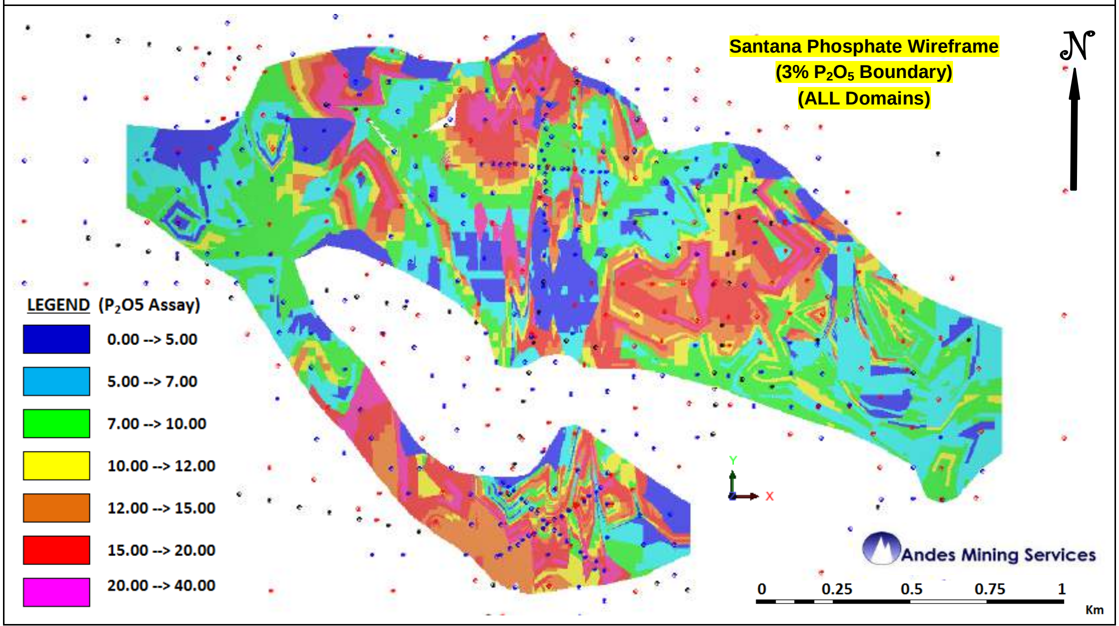
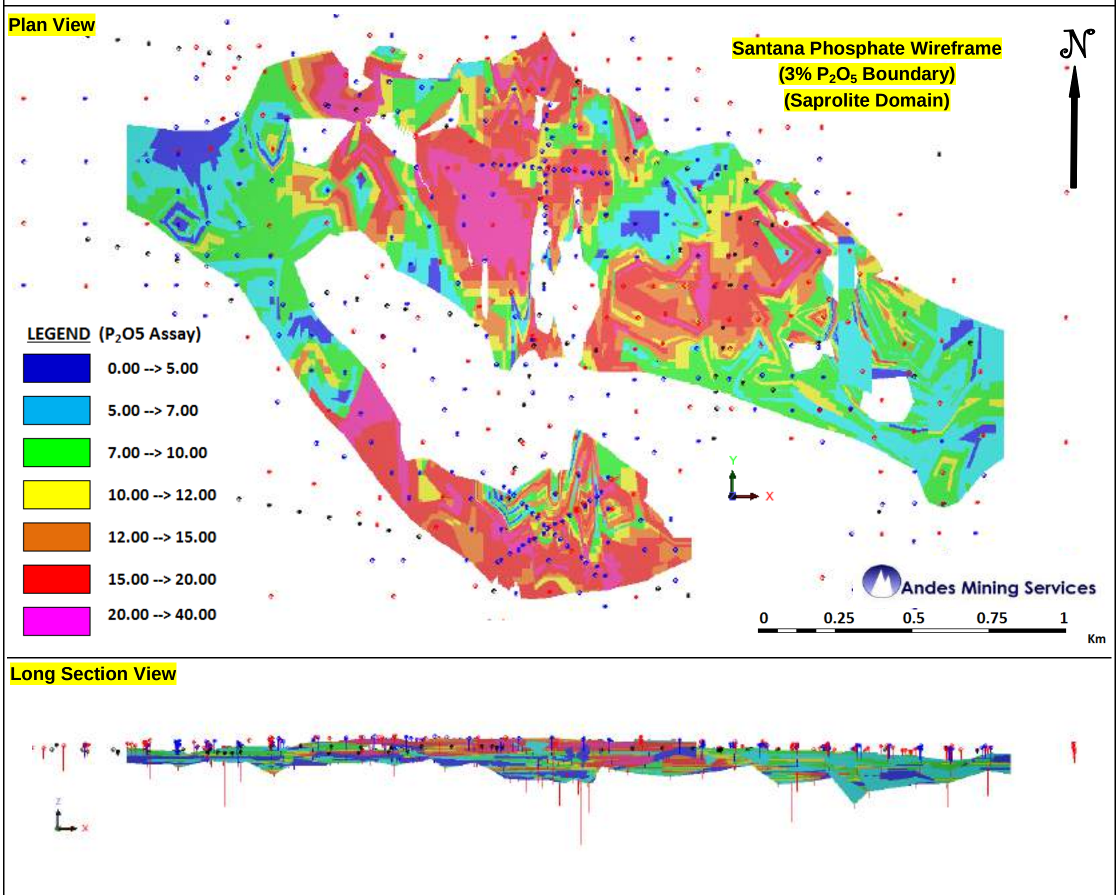


Figure 14.8_4
Santana Block Model – Coded by P_2O_5 Grades (Saprolite Domain) *(AMSL, 28th October 2013)*



Grade 'striping' observed within the model in Figures 14.8_3 and 14.8_4 is primarily due to the relatively thin vertical search parameters used to estimate blocks (10m vertical search on 1st and 2nd pass of estimate), as well as the natural variability in the data observed downhole. This problem is manifested in areas of highly variable weathering profile (fresh to saprolite boundary surface).

14.9 Model Validation

A validation of the mineral resource P_2O_5 grade as well as all other elements (Al_2O_3, CaO, Fe_2O_3, LOI, MnO_2, SiO_2 and TiO_2) was conducted as part of the verification process.

The validation includes:

- a visual comparison of the colour-coded block values versus the composites data in the vicinity of the interpolated blocks, and;
- a comparison of the grade average parameters for the composite data and the block model data.

Table 14.9_1 summarises the comparative statistics of the composite and block model datasets without any cut-off grades.

Table 14.9_1
Comparative Statistics of the Composite and Block Model Datasets

Dataset	Count	Average Grade (%) - Composites vs Block Model Comparison							
		Al_2O_3	CaO	Fe_2O_3	LOI	MnO_2	P_2O_5	SiO_2	TiO_2
Composites	1635*	6.48	23.07	14.50	11.34	2.37	**11.36**	26.23	0.59
Block Model	48457212	5.90	24.90	13.11	13.90	2.16	**9.98**	25.41	0.53

* Various number of composites available for each element. See Table 14.5_1 for details.

The variation in grade from the average composite value of 11.36% P_2O_5 to the block model grade of 9.98% P_2O_5, is reflected in the tendency for drilling to be more tightly spaced across the highest grade portions of the mineral resource. The effect of this is to give an overall average composite grade that is biased higher due to the uneven spread of assay data. A lower block model grade is to be expected, given that a tightly controlled search ellipse acts to limit the spread of higher grade samples concentrated in a number of small areas and the OK interpolation process in effect declusters the data.

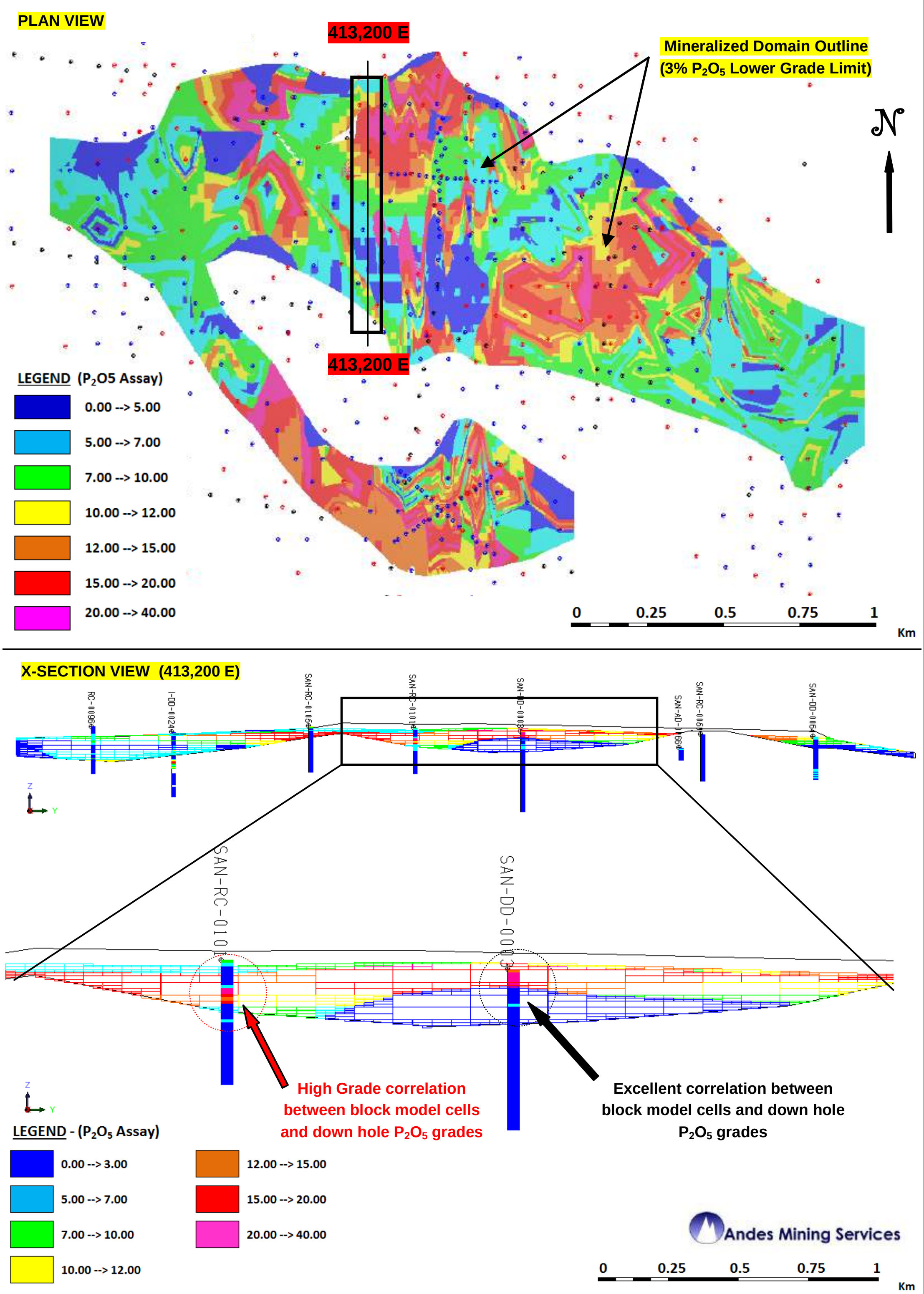
In order to check that the estimation is reliable, the model has been validated through a visual comparison of down hole drilling grades (assays) and estimated blocks in close proximity to those drill holes.

An example of the visual validation is shown below with a cross section of the block model (413,200 E) compared against the drill hole results (Figure 14.9_1).

There is an excellent correlation in block grades with down hole drilling assays for P_2O_5, with grade spreading both laterally and vertically found to be consistent with the input parameters for the block modelling.

In general, the model honours the data well as evidenced below in Figure 14.9_1.

Figure 14.9_1
Block Model Validation against Drilling – Section at 413,200E *(AMSL, 28th October 2013)*



14.10 Mineral Resource Classification

The mineral resource estimate for the Santana Phosphate Project has been classified as Indicated and Inferred Mineral Resources. The parameters used to determine the mineral resource classification include, but are not limited to; drilling density, estimation pass number, number of samples used to make a block estimate as well as the average distance to find samples to make a block estimate.

Table 14.10_1 below highlights some of the specific factors considered in the classification of the Santana Phosphate Project resource. Figures 14.10_1 and 14.10_2 show the resource classification across the Santana Phosphate Project area.

Table 14.10_1

Resource Classification Considerations - Santana Phosphate Project

	Considerations	% Estimated
Indicated	Block must estimate within either the 1st, 2nd or 3rd Pass to be considered "**INDICATED**". Blocks with a "FRESH ROCK" classification have been excluded. In addition, blocks must lie within a 'rescat' string file as highlighted by AMSL in Figure 14.10_1.	69.1
Inferred	Any block that made an estimate in the three passes, but has not already been assigned Indicated Status has been assigned "**INFERRED**".	30.9
Unclassified	Any block that **did not** make an estimate in the 3 Passes has been assigned "**UNCLASSIFED**"	0.0

Figure 14.10_1
Resource Classification String File used for Santana Phosphate Project

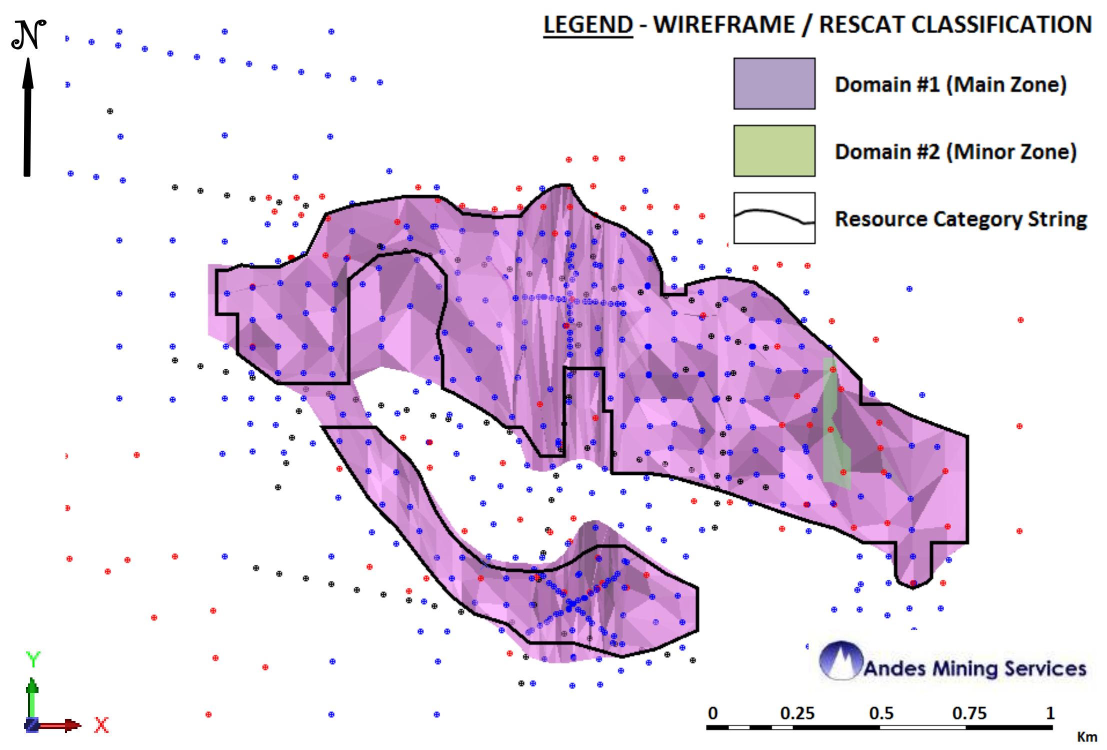
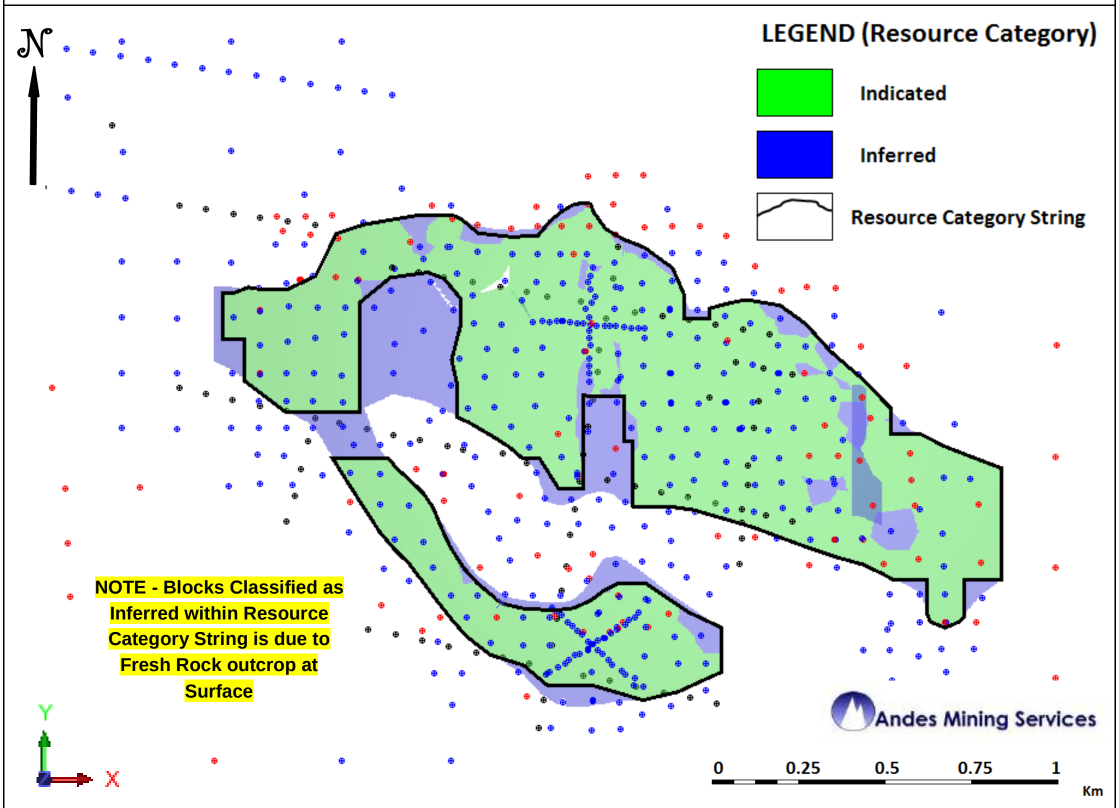


Figure 14.10_2
Resource Classification - Santana Phosphate Project



The lateral extent of both Indicated and Inferred mineral resources for the Santana Phosphate Project are highlighted below in Figures 14.10_3 and 14.10_4. With the Inferred mineral resource category removed, Figure 14.10_4 shows the true extent of Indicated mineral resource category and highlights the highest grade portions of this resource category.

Figure 14.10_3
Oblique View - Resource Classification (Santana Phosphate Project)

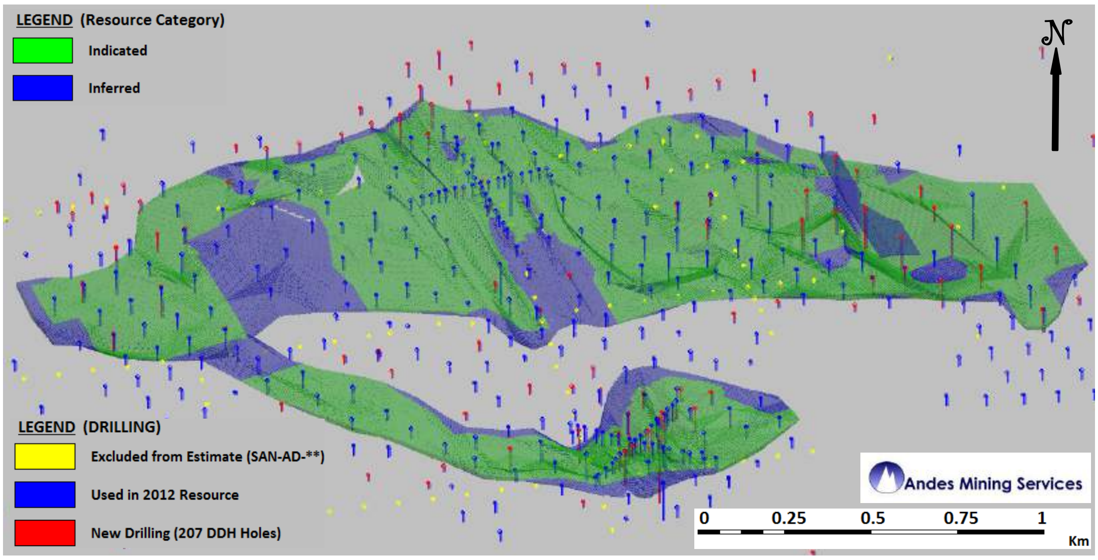
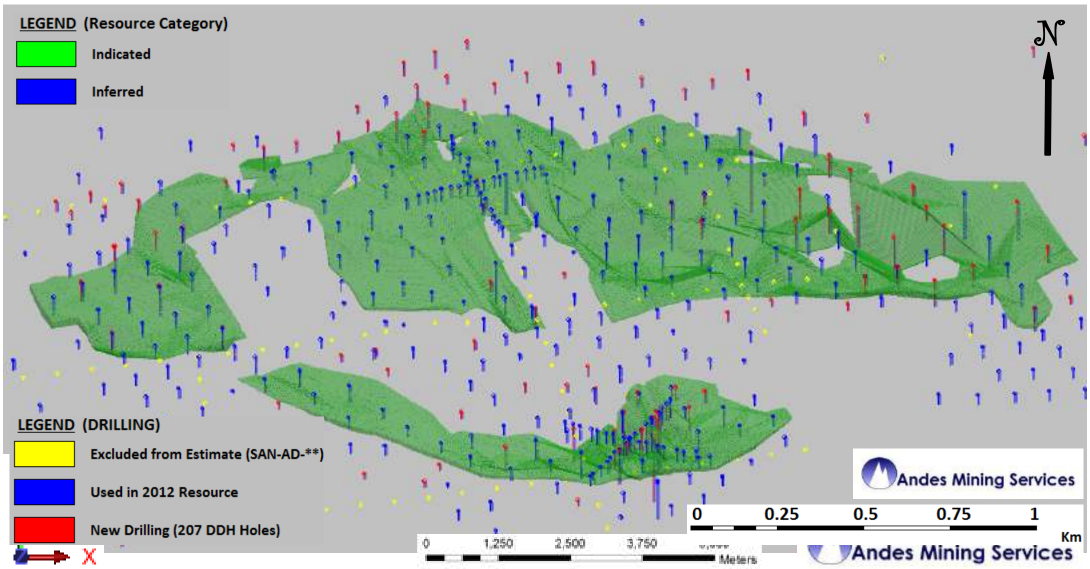
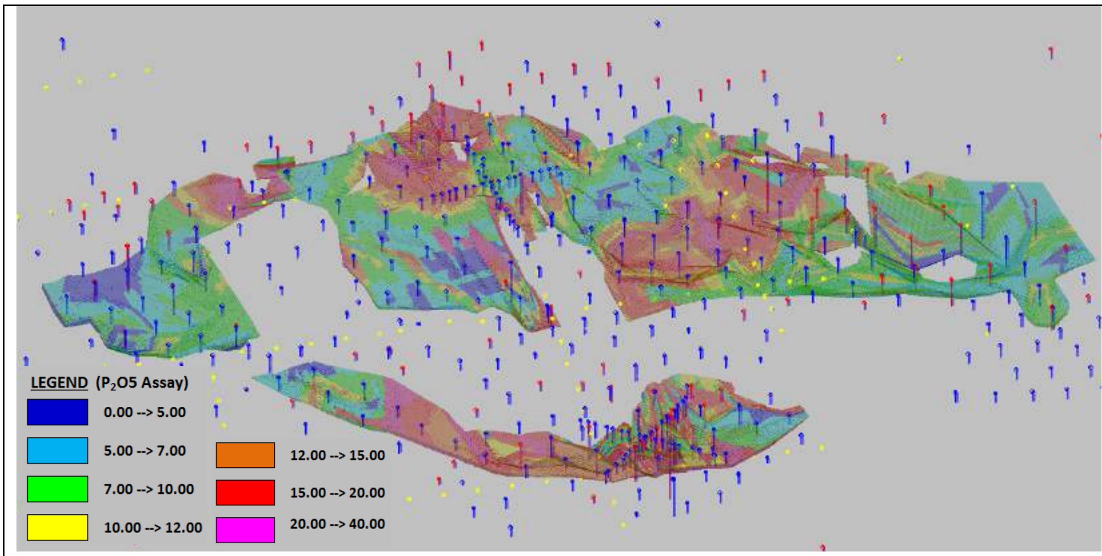


Figure 14.10_4
Indicated Resource Classification with P_2O_5 Grade Distribution (Santana Phosphate Project)





14.11 Mineral Resource Reporting

The grade estimates for the Santana Phosphate Project has been classified as an Indicated and Inferred mineral resource in accordance with NI 43-101 guidelines based on the confidence levels of the key criteria that were considered during the mineral resource estimation.

Key criteria are tabulated below in Table 14.11_1.

A detailed summary (various cut-off grades) of the estimated Indicated, and Inferred mineral resources for the Santana Phosphate Project is provided below in Table 14.11_2, with a more detailed summary table provided in Table 14.11_3.

Table 14.11_1

Santana Phosphate Project

Confidence Levels of Key Categorisation Criteria

Items	Discussion	Confidence
Drilling Techniques	Diamond drilling is industry standard with good recoveries exhibited throughout. RC Drilling was identified with wet sampling having been undertaken.	Moderate
Logging	Standard nomenclature used. Minor overlaps noted within database.	High
Drill Sample Recovery	Excellent recoveries recorded for new DC drilling. Moderate to low recovery levels noted for RC drilling programs completed in 2011 where approximately 25% of RC samples were noted to be moist / humid. Minor cavities noted from RC / DC Drilling.	Moderate
Sub-sampling Techniques & Sample Preparation	DC sampling completed on 1m intervals or to geological boundaries where they exist. All RC sampling was completed on 1m sample intervals. Sample preparation has been completed to industry standards. Twin hole comparisons for DC and RC display poor precision, but reasonable accuracy.	Moderate
Quality of Assay Data	Excellent for standards, blanks and duplicates (2011 / 2012 Programs for SGS and ALSO). Umpire assay test work is excellent with no bias observed between the SGS and ALS laboratories.	High
Verification of Sampling and Assaying	Duplicate sample data shows excellent correlation (coarse reject duplicates). DC twin hole drilling with pre-existing RC holes show poor correlation. Despite this there is no grade bias observed.	Moderate
Location of Sampling Points	Drill hole collar locations have been independently surveyed (DGPS pick-up) and a detailed topographic surface has been generated based on these collar coordinates.	Moderate / High
Data Density and Distribution	Drill spacing of approximately 100m by 100m. Infill drilling to a 50 x 50m grid spacing as recommended in previous 43-101 has not been completed. AMSL note highly variable saprolite to fresh rock boundary which requires detailed drill testing.	Moderate / High
Audits or Reviews	AMSL is unaware of external reviews.	N/A
Database Integrity	Only DDH and RC holes are considered for the resource. Auger drilling has been used to help guide wireframe interpretation, however auger drilling results have NOT been used for the purpose of resource estimation.	Moderate / High
Geological Interpretation	Entirely within soil, saprolite and fresh rock domains. Strong geological understanding with mineralization outcropping at surface in places. AMSL note a highly variable saprolite to fresh rock boundary surface, and further infill drilling is required to increase the confidence level in these areas of variability. A high quality geological team is continually improving the 3D geological model.	High
Estimation and Modelling Techniques	Reliable and conservative, due to large block size and composite length of 4m. Ordinary Kriging (OK) utilized which is appropriate given the distributions observed in the data.	High
Cut-Off Grades	A 3.0% P_2O_5 cut-off grade has been applied to the final reported resource numbers and defines the mineralization well. Recent metallurgical test work has proven this cut-off grade to be economic.	High
Mining Factors or Assumptions	25mE by 25mN by 4mRL SMU.	High

The statement has been classified by Qualified Person Bradley Ackroyd (MAIG) in accordance with the guidelines of NI 43-101. It has an effective date of 28th of October, 2013. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

AMSL and MBAC are not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that may materially affected the viability of the mineral resource estimate. The Santana Phosphate Project is a greenfield site and therefore is not affected by any mining, metallurgical or infrastructure factors.

Table 14.11_2
MBAC - Santana Phosphate Project
Indicated and Inferred Mineral Resource Grade Tonnage Report – 28th October 2013
Ordinary Kriging (OK) - 3.0% P_2O_5 Cut-Off Grade Utilized
(Block Model – 25mE X 25mN X 4mRL)

Resource Category	Cut-Off (% P_2O_5)	Tonnes (Mt)	P_2O_5	CaO	Fe_2O_3	LOI	MnO_2	Al_2O_3	SiO_2	TiO_2
SOIL HORIZON										
Inferred	0.0	0.21	9.21	9.58	19.68	7.30	1.60	14.15	34.25	1.00
	3.0	0.21	9.21	9.58	19.68	7.30	1.60	14.15	34.25	1.00
	5.0	0.21	9.21	9.58	19.68	7.30	1.60	14.15	34.25	1.00
	10.0	0.06	13.08	11.32	23.64	8.79	1.87	15.09	22.67	1.06
	15.0	0.00	15.21	14.39	21.86	8.25	2.24	13.61	20.86	0.93
Indicated	0.0	1.53	10.70	10.13	22.61	10.02	3.61	16.45	21.42	1.06
	3.0	1.53	10.70	10.13	22.61	10.02	3.61	16.45	21.42	1.06
	5.0	1.40	11.27	10.65	22.72	9.94	3.75	16.18	20.54	1.05
	10.0	0.66	15.95	16.65	19.92	8.85	3.67	13.64	16.98	0.88
	15.0	0.29	19.98	23.31	14.75	7.26	3.05	11.11	16.66	0.70
	20.0	0.12	24.34	31.70	9.98	5.35	2.39	7.30	15.68	0.48
	25.0	0.05	26.78	35.88	7.84	4.62	2.39	5.33	13.98	0.35
SAPROLITE HORIZON										
Inferred	0.0	4.93	10.04	14.86	14.92	5.88	1.84	8.00	39.85	0.70
	3.0	4.90	10.10	14.92	14.89	5.88	1.85	8.00	39.77	0.70
	5.0	4.43	10.72	15.49	14.44	5.69	1.91	7.95	39.45	0.67
	10.0	1.84	15.45	21.38	13.59	5.39	2.26	6.92	31.23	0.59
	15.0	0.90	18.86	25.83	12.08	4.98	2.62	5.70	26.47	0.48
	20.0	0.29	21.67	29.82	10.70	4.67	2.96	4.74	22.27	0.38
	25.0	0.00	26.35	35.59	7.55	2.60	1.45	3.24	20.71	0.31
Indicated	0.0	59.21	12.02	18.80	16.23	7.72	2.61	7.16	30.99	0.65
	3.0	58.83	12.08	18.86	16.22	7.72	2.63	7.15	30.89	0.65
	5.0	55.02	12.62	19.48	15.97	7.68	2.69	7.03	30.19	0.63
	10.0	33.62	15.92	23.73	14.47	7.57	2.89	6.41	24.96	0.55
	15.0	17.01	19.37	27.44	13.04	6.62	3.01	5.71	21.15	0.49
	20.0	6.47	22.86	31.49	11.73	5.67	3.02	4.79	17.04	0.43
	25.0	1.06	26.97	36.69	9.35	4.81	2.49	4.01	12.62	0.34
	30.0	0.10	31.41	43.54	7.04	3.15	1.60	1.54	8.59	0.20
FRESH HORIZON										
Inferred	0.0	22.01	4.45	44.76	3.59	32.67	0.90	1.22	7.33	0.11
	3.0	21.48	4.49	44.75	3.61	32.60	0.91	1.22	7.38	0.12
	5.0	5.63	5.90	42.37	5.06	29.10	1.02	1.44	10.72	0.16
	10.0	0.01	10.23	30.43	10.45	14.80	1.86	3.10	23.46	0.32

Mineral resources are not mineral reserves and do not have demonstrated economic viability.
Appropriate rounding has been applied to Table 14.11_2.

The mineral resource estimate has focused on a flat-lying, sub horizontal mineralized domain which has been defined at surface and drill tested to depth of mineralization using a nominal 3% P_2O_5 grade cut-off to guide the wireframing process.

An independent mineral resource has been estimated for the Santana Phosphate Project comprising an Indicated mineral resource of 60.36 Mt at 12.04% P_2O_5 (using a 3.0% P_2O_5 cut-off), and an Inferred mineral resource of 26.59 Mt at 5.56% P_2O_5 (using a 3.0% P_2O_5 cut-off grade).

Table 14.11_3
MBAC - Santana Phosphate Project (Summary Report)
Indicated and Inferred Mineral Resource Grade Tonnage Report – 28th October 2013
Ordinary Kriging (OK) - 3.0% P_2O_5 Cut-Off Grade Utilized

Domain	Cut-Off (% P_2O_5)	Tonnes (Mt)	P_2O_5	CaO	Fe_2O_3	LOI	MnO_2	Al_2O_3	SiO_2	TiO_2
Indicated Resource Category										
Soil	3.0	1.53	10.70	10.13	22.61	10.02	3.61	16.45	21.42	1.06
Saprolite	3.0	58.83	12.08	18.86	16.22	7.72	2.63	7.15	30.89	0.65
TOTAL INDICATED -->		**60.36**	**12.04**	**18.64**	**16.38**	**7.78**	**2.65**	**7.39**	**30.65**	**0.66**
Inferred Resource Category										
Soil	3.0	0.21	9.21	9.58	19.68	7.30	1.60	14.15	34.25	1.00
Saprolite	3.0	4.90	10.10	14.92	14.89	5.88	1.85	8.00	39.77	0.70
Fresh Rock	3.0	21.48	4.49	44.75	3.61	32.60	0.91	1.22	7.38	0.12
TOTAL INFERRED -->		**26.59**	**5.56**	**38.97**	**5.82**	**27.47**	**1.08**	**2.58**	**13.57**	**0.23**

** Mineral resources are not mineral reserves and do not have demonstrated economic viability. Appropriate rounding has been applied to Table 14.11_3.*

Grade tonnage curves for the Indicated and Inferred portions of the Santana Phosphate Project are shown below in Figures 14.11_1 to 14.11_2 respectively.

Figure 14.11_1
Grade Tonnage Curve – Indicated Resource Category *(AMSL, 28th October 2013)*

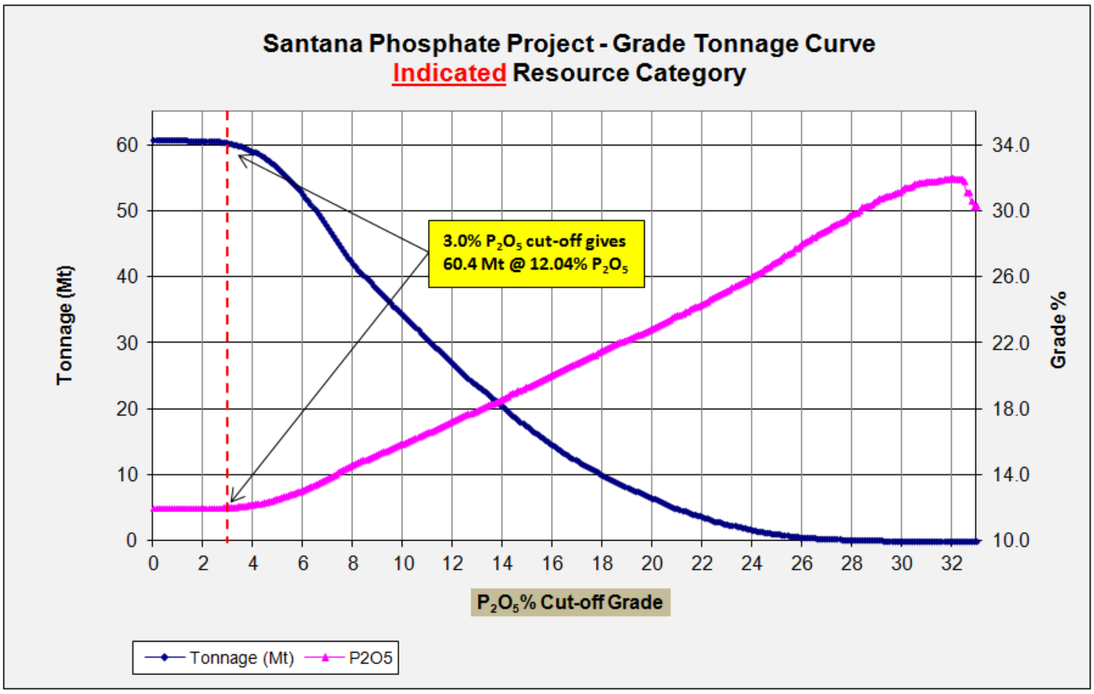
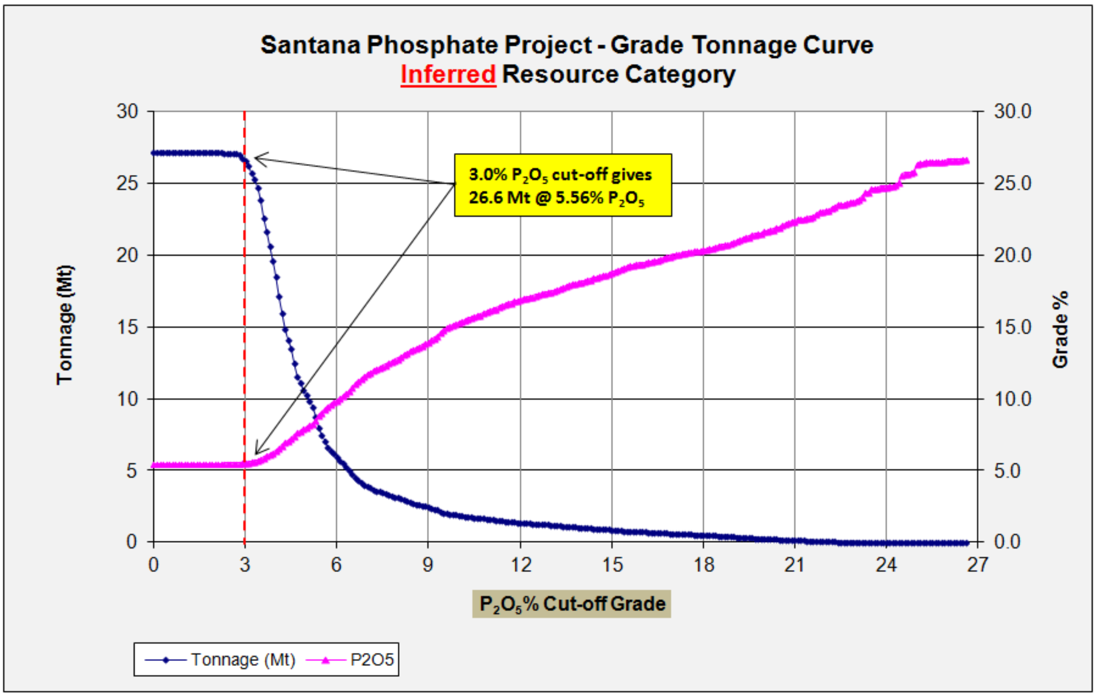
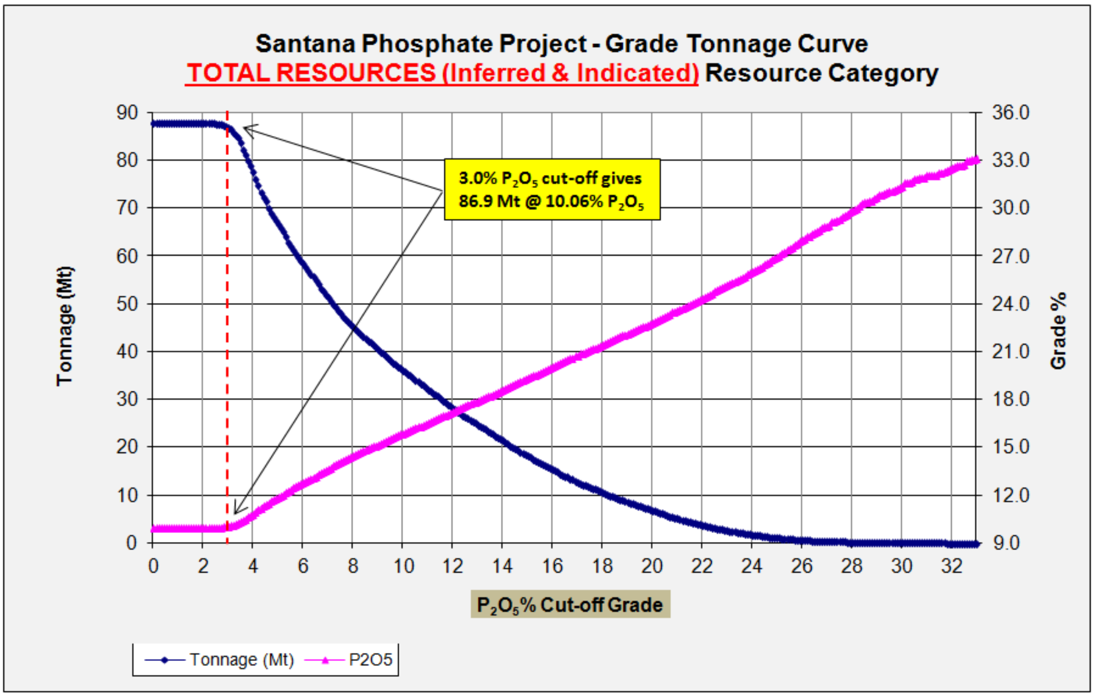


Figure 14.11_2
Grade Tonnage Curve – Inferred Resource Category *(AMSL, 28th October 2013)*



An additional grade tonnage curve for combined Indicated and Inferred resource is presented below in Figure 14.11_3.

Figure 14.11_3
Grade Tonnage Curve – Total Santana Resource (Indicated and Inferred) *(AMSL, 28th October 2013)*



15 MINERAL RESERVE ESTIMATES

NCL Brasil Ltda. (NCL) prepared the Mineral Reserve Estimates of the Santana Project in accordance with the NI 43-101 CIM Guidelines.

NCL prepared the Mineral Reserves Estimate under the supervision of Mr. Carlos Guzmán BSc Mining Engineering, RM Chilean Mining Commission 119, FAusIMM 229036, Director of NCL and an independant Qualified Person as defined in the CIM Guidelines.

15.1 Key Assumptions/ Basis of Estimate

The Life of Mine (LOM) mining schedule that supports the Mineral Reserves Estimate was prepared using the following parameters:

15.1.1 General Mining Concept

NCL studied the Santana Project as a conventional open pit operation, producing 300 kt per year of phosphate concentrate at a grade of 34% P_2O_5.

The mine is scheduled to work seven days per week or 356 days per year. Each day will consist of three 8-hour shifts. Four mining crews will cover the operation.

A diesel mining fleet will be used on the open pit. 97% of the material within the open pit can be removed without the use of explosives by a fleet of 5 m^3 hydraulic excavator that will load 32 t trucks. Ore will be hauled to the primary crusher for processing while waste will stored in the waste dumps surrounding the open pit or will be dumped in areas of the pit already exhausted.

15.1.2 Resource Block Model

NCL used the October 28th, 2013 block model. Andes Mining Services (Andes) prepared the resource block model; Section 14 of this report documents this process.

15.1.3 Geotechnical Recommendations

Pit geometry is based on work prepared by VOGBR (VG13-081-1-GL-RTE-0001, March, 2013).

15.1.4 Mining Dilution and Ore Losses

It is becoming common in the industry to develop resource models, which essentially take into account potential dilution within the blocks, or adopt selective mining unit (SMU) as part of the resource modeling process. The Santana mineral resource model has been assessed to achieve this outcome and hence considered a 'diluted' model. Furthermore, the relationship between the recommended mining fleet and the block model's block size allows for selective mining when proper mining practice is followed, and thus full mining recovery is expected.

15.1.5 Mine Production Schedule

Whittle Four-X pit optimization software was used to generate an optimal pit shell using the Lerch-Grossman algorithm. Table 15.1.5_1 shows the parameters used on the pit optimization.

Table 15.1.5_1

Pit Optimization Parameters

Super Simple Phosphate (SSP) Price	350	USD/tonne SSP
Recovery	55%	
Concentrate Grade	34%	P_2O_5
SSP Grade	19%	P_2O_5
Processing		
May be selected for processing Sand-Ore, Saprolite-Ore and Fresh-Ore where :		
- $CaO/P_2O_5 >= 1$		
- Al2O3 + Fe2O3 < 30%		
- Only Measured and Indicated resources were used.		
Costs		
Mining	1.63	USD/tonne mined
Processing	10.91	USD/tonne processed
G&A	1.86	USD/tonne processed
Others	3.58	USD/tonne processed
Sulphuric Acid	55.30	USD/tonne SSP
Granulation	43.30	USD/tonne SSP
Slope Angle	34	degrees

The economic shell generated at the reference SSP price (350 USD/t SSP) is the basis of the final pit design. Twenty intermediate phases were designed to prepare a LOM schedule.

The Santana Project mine schedule produces 300 kt per year of phosphate concentrate at a grade of 34% P_2O_5. Total material movement rate varies from 3.2 Mtpy at the start of the project up to a maximum of 9 Mt in later years. This scenario results in the processing of an average of 1.5 Mtpy of ore with an average P_2O_5 grade of 12.86%. The expected mine life for the project is 32 years besides the pre stripping period.

15.1.6 Conversion Factors from Mineral Resources to Mineral Reserves

Mineral Reserves have been determined from Mineral Resources by taking into account geologic, mining, processing, legal and environmental considerations and are therefore classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

Indicated Resources considered for extraction by the life of mine schedule have been converted to Probable Reserves.

All Indicated Resources within the final pit design have been scheduled for extraction in the LOM plan and have been converted to reserves.

Indicated Resources outside the final pit design have not been converted to Reserves.

Inferred Resources have not been converted to Reserves and instead treated as waste for mine planning purposes.

15.2 Mineral Reserves Statement

Based on the whole body of work prepared for the Santana Project FS, it is the opinion of NCL that the mine production schedule defines the mineral reserve for a mining project.

Table 15.2_1 reports the mineral reserve of the Santana Project based on the production schedule used for this study.

The Mineral Resources stated in Section 14 are inclusive of the Mineral Reserves.

Table 15.2_1

Mineral Reserve Summary

Ore Reserves	Mass (tonnes ´000)	P_2O_5 (%)	Phosphate Concentrate (tonnes ´000)
Proven Mineral Reserve	-	-	-
Probable Mineral Reserve	45,481	12.86	9,459
Total Reserve	45,481	12.86	9,459

Notes for the Mineral Reserves Statement:

(1) Based on a price for SSP (Super Simple Phosphate) of US$350.00 per tonne;

(2) At a 3% P_2O_5 cut off grade;

(3) Open pit reserves assume complete mining recovery;

(4) Open pit reserves consider the inherent dilution of the resource block model;

(5) Waste tonnes within the open pit amount to 143.3 Mt at a strip ratio of 3.15:1 (waste to ore);

(6) Numbers may not add due to rounding;

(7) Plant Recovery: 55%;

(8) The mineral reserves for the Project were estimated by Carlos Guzmán, RM Chilean Mining Commission 119, FAusIMM 229036 and Director of NCL Brasil Ltda. in accordance with the Canadian Securities Administrators National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and generally accepted Canadian Institute of Mining, Metallurgical and Petroleum "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines ("CIM Guidelines");

(9) Reserves Effective Date: October 28, 2013.

(10) All mining modifiers, including aspects relating to metallurgy, processing, infrastructure and/or mining have been included in the Mineral Reserve determination. Environmental, permitting, legal, title, taxation, socio-economic, marketing, and or political factors have also been considered, where relevant, and are discussed in various sections of this report.

16 MINING METHODS

16.1 Background Information

16.1.1 Mineral Resource Block Model

A 3-D block model was generated to enable grade estimation. The selected block size was based on the geometry of the domain interpretation and the data configuration. A block size of 25 mE x 25 mN x 4 mRL was selected. The sub cell block modeling technique was used to represent the volume of the interpreted wireframe models. Sufficient variables were included in the block model construction to enable grade estimation and reporting.

The block model was later converted to a "percent" type block model by NCL to better interface with its Software. Cross-Validation checks were prepared to assure an adequate conversion.

The block model attributes used by NCL are shown in the following table:

Table 16.1_1
Block Model Attributes

Attribute	Type	Units
Lithology	Code	1 = Soil , 2 = Saprolite, 3 = Fresh Rock
Category	Code	1 = Measured, 2 = Indicated, 3 = Inferred
Bulk density	Numeric	t/m3
P_2O_5	Numeric	%
Al_2O_3	Numeric	%
CaO	Numeric	%
Fe_2O_3	Numeric	%
LOI	Numeric	%
MnO_2	Numeric	%
SiO_2	Numeric	%
TiO_2	Numeric	%

NCL did not audit the sampling data or the block model used for this project.

The October 2013 block model contains Inferred and Indicated mineral resources. All optimization and mine planning work considers only Indicated mineral resources as potential ore, depending on its rock type and economic attributes. Inferred mineral resources have not been included because they are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves.

Table 16.1_2 shows the material densities and swelling factors used on the Santana project.

Table 16.1_2 Material Densities				
Material	Unit	Soil	Saprolite	Fresh Rock
in situ density	t/m³	1.50	1.64	2.69
Swell Factor	%	30%	30%	30%
Swelled Density	t/m³	1.15	1.26	2.07

16.1.2 Mining Dilution and Ore Losses

It is becoming common in the industry to develop resource models (particularly where nonlinear estimation techniques are applied), which essentially take into account potential dilution within the blocks, or adopt selective mining unit (SMU) as part of the resource modeling process. NCL did not add additional dilution to the provided block model as it was considered to be already diluted. Furthermore, the relationship between the recommended mining fleet and the block model's block size allows for selective mining when proper mining practice is followed.

16.1.3 Initial Topography

MBAC personnel provided the initial topography.

16.1.4 Geotechnical Recommendations

Pit geometry is based on work prepared by VOGBR (VG13-081-1-GL-RTE-0001, March, 2013). Figure 16.1.4_1 shows the geotechnical definitions used by VOGBR and Table 16.1.4__1 summarizes the slope angle recommendations.

Table 16.1.4_1 Open Pit Slope Angle Recommendations								
Rockmass Classification		**Lithology**	**Bench Height (m)**	**Berm Width (m)**	**Bench Face Angle (º)**	**Inter Ramp Angle (º)**		**Safety Berm or ramp**
						Max height (m)	**Max Angle (º)**	
V	Soil/ Completely altered saprolite (Very poor)	Soil/ Saprolite	8	5	50	40	34	20 m every 40 m
II - III	Slightly altered to fresh rock, fractured to slightly fractured (Fair to Good)	Fresh Rock	8	5	70	80	45	20 m every 80 m

Figure 16.1.4_1 contains the waste dump recommendations. This were used on the ex-pit dumps, in the case of the in pit dumps the bench height was reduced to 8 m to be compatible with existing pit configuration, reducing berm width accordingly to maintain the 22º overall angle.

Figure 16.1.4_1
Slope Geometry Definitions

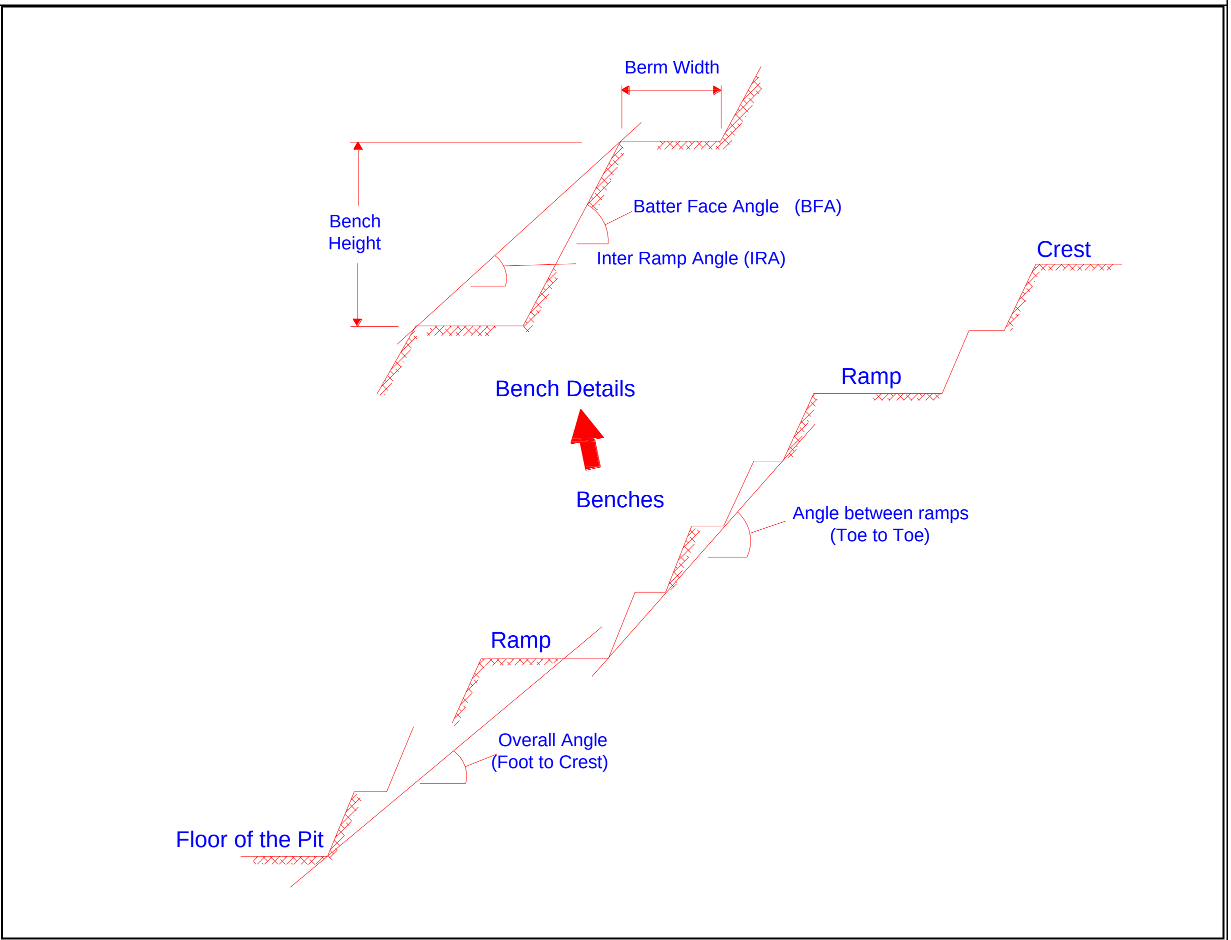
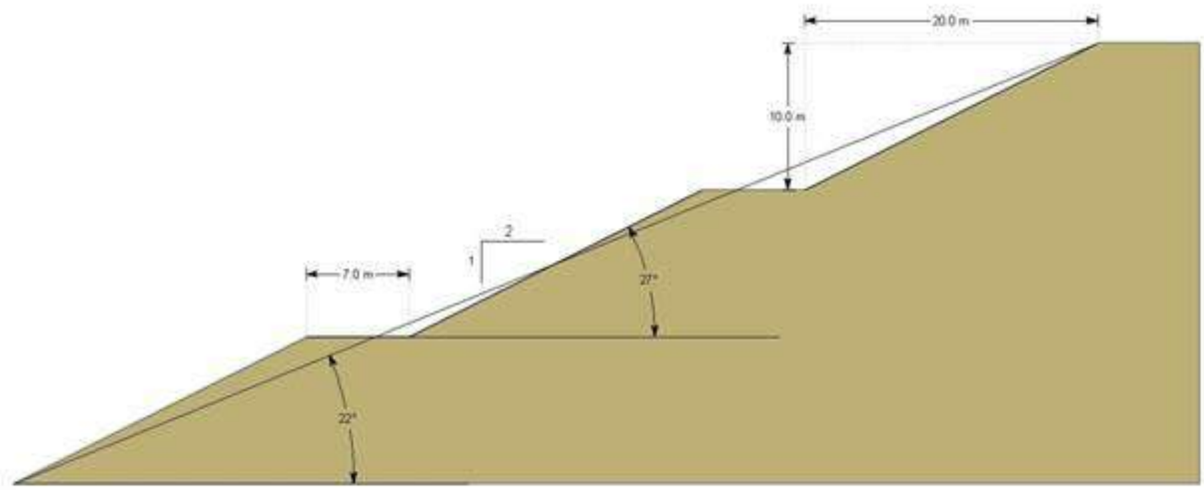


Figure 16.1.4_2
Waste Dump Recommendations



16.1.5 Processing Recoveries

Processing recoveries were provided to NCL by MBAC personnel, based on their own estimates and test work.

Potential ore was classified according to their recovery potential:

- Type I: $CaO/P_2O_5 \geq 1$ and $Al_2O_3 + Fe_2O_3 < 30\%$ simultaneously.
- Type II: $CaO/P_2O_5 < 1$;
- Type III: $Al_2O_3 + Fe_2O_3 \geq 30\%$;
- Type IV: $CaO/P_2O_5 < 1$ and $Al_2O_3 + Fe_2O_3 \geq 30\%$ simultaneously.

Only Type I ore is deemed suitable for processing. All Type I material was assigned a 55% metallurgical recovery.

In this study, only indicated mineral resources were considered as potential ore.

16.2 Open Pit Optimization

16.2.1 Base Case

Table 16.2.1_1 below summarizes the technical - economic parameters used for the base case optimization.

The mining cost estimate for the pit optimization process is based on the results of the 7th June 2012 PFS and posted on www.sedar.com. Product prices, processing costs and processing recoveries were provided to NCL by MBAC personnel, based on their own estimates and test work. Slope angles are based on VOGBR recommendations. It is important to note that the parameters are initial estimates, done at the beginning of the study, for starting the design process.

Table 16.2.1_1

Pit Optimization Parameters

Super Simple Phosphate (SSP) Price	350	USD/tonne SSP
Recovery	55%	
Concentrate Grade	34%	P_2O_5
SSP Grade	19%	P_2O_5
Processing		
May be selected for processing Sand-Ore, Saprolite-Ore and Fresh-Ore where :		
- CaO/P_2O_5 >= 1		
- Al2O3 + Fe2O3 < 30%		
- Only Measured and Indicated resources were used.		
Costs		
Mining	1.63	USD/tonne mined
Processing	10.91	USD/tonne processed
G&A	1.86	USD/tonne processed
Others	3.58	USD/tonne processed
Sulphuric Acid	55.30	USD/tonne SSP
Granulation	43.30	USD/tonne SSP
Slope Angle	34	degrees

.

Whittle Four-X pit optimization software was used to generate a series of nested pits of increasing economic value, using the Lerch-Grossman algorithm.

Based on the economic parameters of Table 16.2.1_1 NCL calculated a marginal cutoff grade of 2.15 % P_2O_5. Given that this value is lower than the cutoff grade used during wireframing process, which is compatible with the mineral processing plant configuration, NCL used a hard cut of grade of 3% during the optimization process

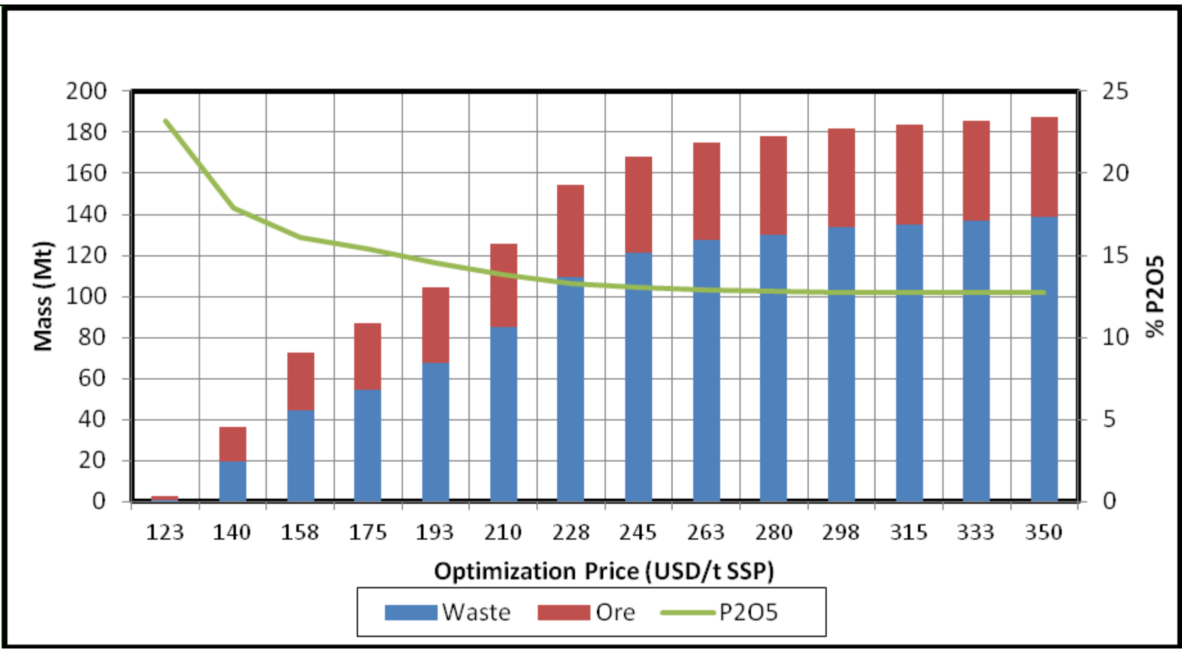
Table 16.2.1_2
Base Case Optimization Results

Rev factor	Price	Ore (cut off 3% P_2O_5)					Waste	Total Rock	Strip Ratio	Operational Cost (1) (US$/t SSP)	
	USD/t SSP	Mass (Kt)	P_2O_5 (%)	Recovery (%)	Concentrate (Kt)	SSP (Kt)	Mass (Kt)	Mass (Kt)		Average	Incremental
0.35	123	1,999	23.17	55.0%	749	1,341	870	2,869	0.44	120.02	120.02
0.40	140	17,024	17.90	55.0%	4,928	8,818	19,689	36,712	1.16	130.50	132.38
0.45	158	28,312	16.08	55.0%	7,363	13,177	44,312	72,625	1.57	136.26	147.92
0.50	175	32,276	15.36	55.0%	8,019	14,349	54,633	86,908	1.69	138.79	167.25
0.55	193	36,724	14.51	55.0%	8,619	15,424	67,439	104,163	1.84	142.08	186.01
0.60	210	40,628	13.87	55.0%	9,117	16,315	85,257	125,885	2.10	145.44	203.44
0.65	228	44,898	13.27	55.0%	9,635	17,242	109,500	154,398	2.44	149.32	217.63
0.70	245	46,677	13.02	55.0%	9,830	17,590	121,128	167,805	2.60	151.08	238.61
0.75	263	47,592	12.89	55.0%	9,922	17,755	127,413	175,005	2.68	152.04	253.77
0.80	280	48,013	12.82	55.0%	9,960	17,824	130,303	178,316	2.71	152.50	270.54
0.85	298	48,362	12.77	55.0%	9,993	17,883	133,623	181,985	2.76	152.95	290.89
0.90	315	48,545	12.75	55.0%	10,009	17,911	135,244	183,789	2.79	153.19	305.14
0.95	333	48,658	12.73	55.0%	10,021	17,933	137,141	185,799	2.82	153.40	324.45
1.00	350	48,786	12.71	55.0%	10,032	17,952	138,648	187,434	2.84	153.60	341.60

(1) Capital not included.

Figure 16.2.1_1 shows that there is not a considerable impact in the in pit resources with increases of the optimization price above 245 USD/t SSP. There is a change in ore mass of only 2.1 Mt when varying the optimization price from 245 to 350 USD/t SSP.

Figure 16.2.1_1
Tonnes and Grades versus Optimization Price



Analyzing the obtained shells, three sectors can be identified. Sector A located in the northeastern zone of the deposit is roughly tabular in shape with its main axis oriented northwest - southeast. This sector is 2,000 m long by 800 m wide. Sector B is located approximately 250 m to the south of the first; with a tabular shape as well, but dipping 30 degrees to the north. This sector is 1,500 m long by 400 m wide. The third sector (C) is located in the Northwestern side of the deposit, is shaped is also tabular with its main axis oriented southwest – northeast. This sector is 900 m long by 600 m wide.

Figure 16.2.1_2 shows the results of the base case optimization, Figure 16.2.1_3 shows the pit limits for selected revenue factors.

Figure 16.2.1_2
Final Pit Shell

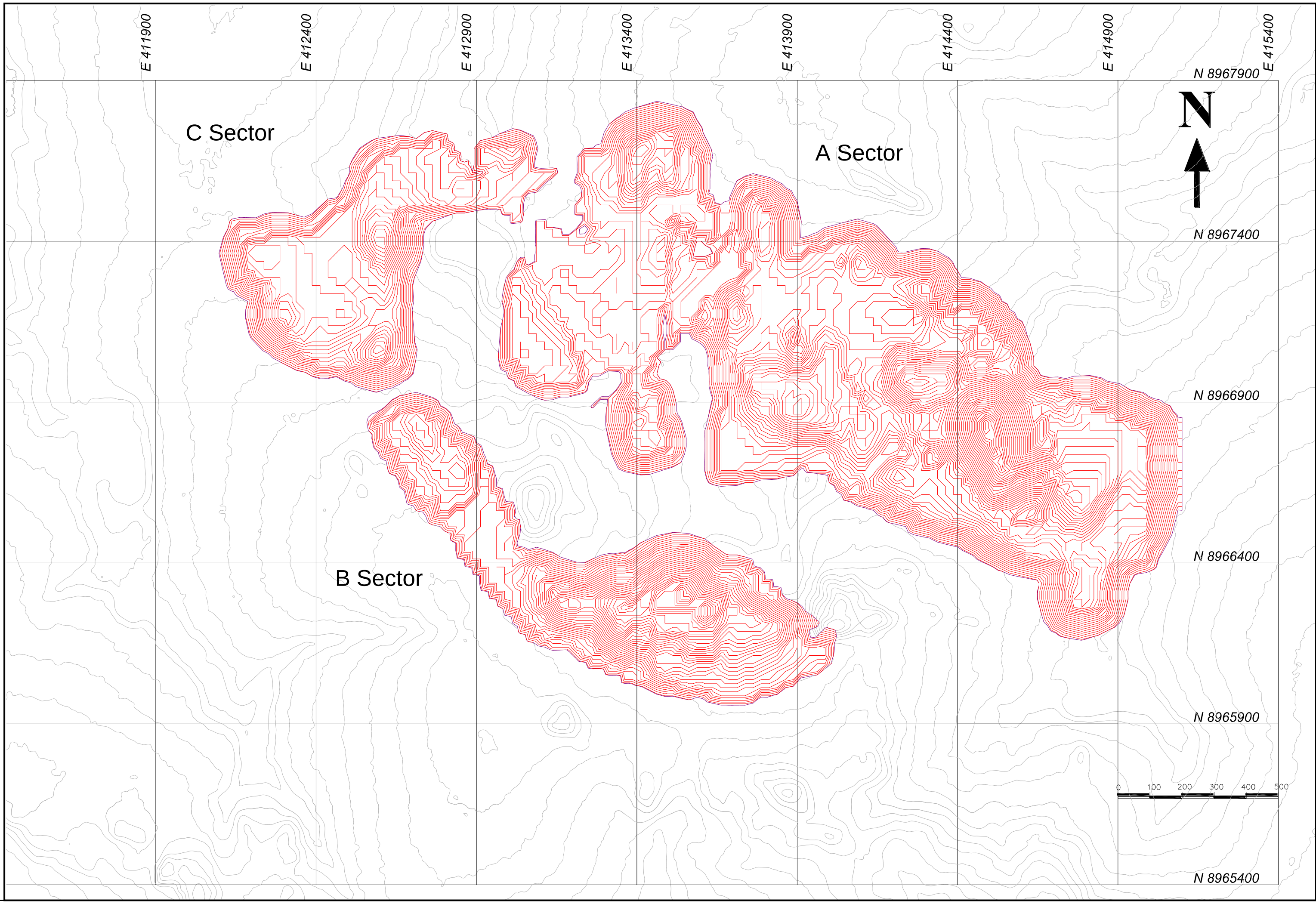
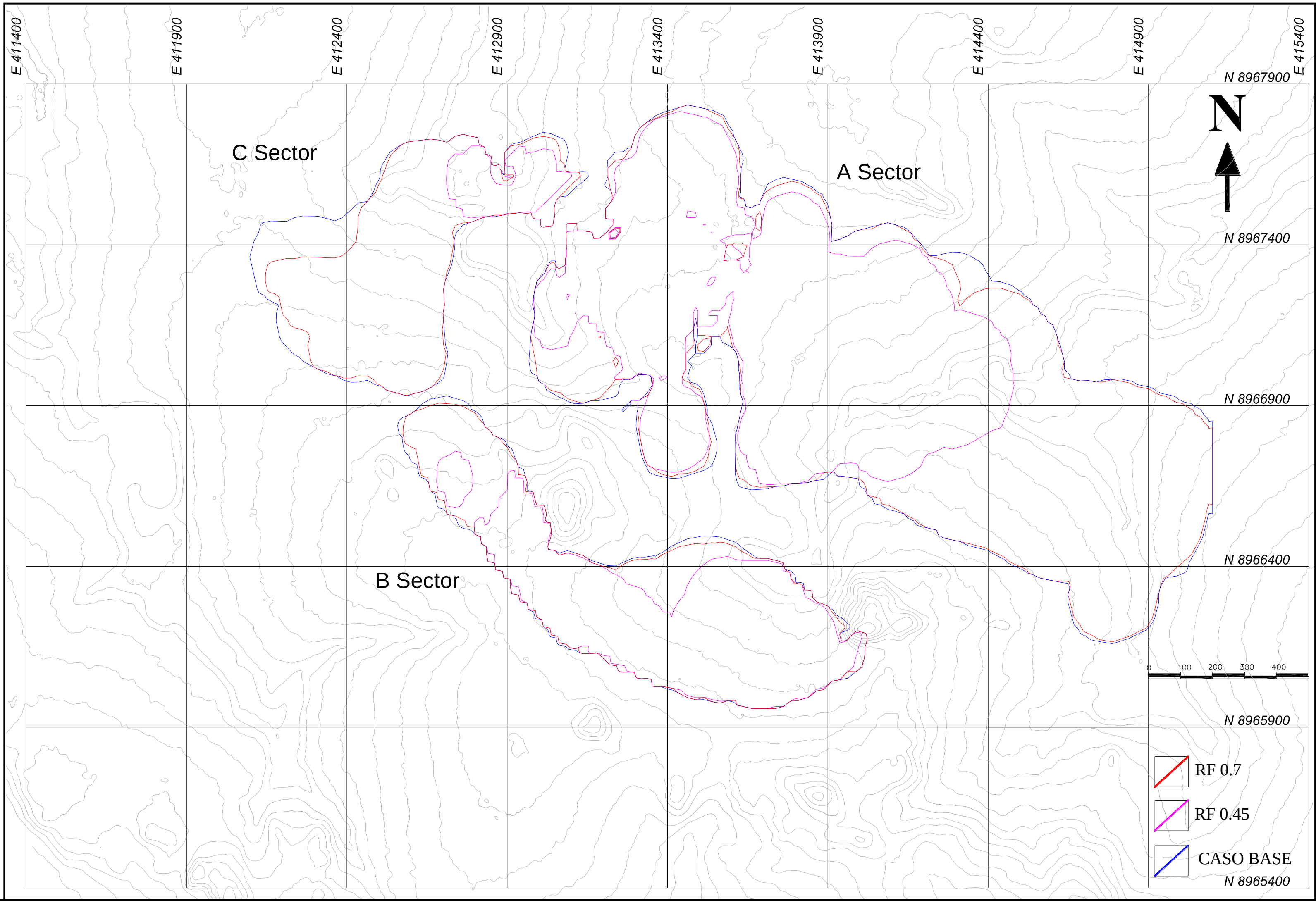


Figure 16.2.1_3
Base Case Optimization



16.2.2 Sensitivity Analysis

Several scenarios were analyzed to evaluate the effect of modifying technical - economical parameters utilized in the optimization process.

The parameters analyzed were:

Mining Cost: - 15% and +15%;

Process Cost: -15% and +15%;

Metallurgical recovery: 52% and 58%;

Slope angle: 30º and 40º;

Cutoff grade: 4% and 5% P_2O_5.

With the exception of the cutoff grade, modifications on the scenario did not produce a notable variation of the optimization results. Within the studied range, the mineral resource contained within the optimum pit varied less than 0.5%.

Increasing the cutoff grade to 5% P_2O_5 causes a reduction of 5% in the mineral resource contained within the optimal pit, but does not change pit limits in a material way.

Given the results above, NCL concludes that the mathematical optimization results are robust within the studied range of technical and economic parameters. This does not imply that the project´s economic results as a whole is not affected by variation of this parameters, simply that the amount of the contained mineral resource and the geometry of the final pit is largely unaffected.

Table 16.2.2_1 and Figure 16.2.2_1 to Figure 16.2.2_5 show the result of the sensitivity analysis.

Table 16.2.2_1
Sensitivity Analysis

Case	Price USD/t SSP	Ore					Waste (Kt)	Total Material (Kt)	Strip Ratio	Cash Cost (US$/t prod)	
		Mass (Kt)	P_2O_5 (%)	Recovery (%)	Concentrate (Kt)	SSP (Kt)				Average	Incremental
Mining Cost + 15%	350	48,669	12.72	55%	10,018	17,926	136,157	184,827	2.80	156	343
Base Case	350	48,786	12.71	55%	10,032	17,952	138,648	187,434	2.84	154	342
Mining Cost - 15%	350	48,893	12.70	55%	10,043	17,972	140,757	189,650	2.88	151	338
Process Cost 15%	350	48,492	12.76	55%	10,010	17,912	136,069	184,562	2.81	174	342
Base	350	48,786	12.71	55%	10,032	17,952	138,648	187,434	2.84	154	342
Process Cost -15%	350	48,921	12.69	55%	10,043	17,972	140,491	189,411	2.87	133	339
Recovery 58%	350	48,869	12.70	58%	10,587	18,945	139,966	188,835	2.86	151	342
Base	350	48,786	12.71	55%	10,032	17,952	138,648	187,434	2.84	154	342
Recovery 52%	350	48,657	12.73	52%	9,476	16,958	137,677	186,334	2.83	157	340
Overall Slope Angle 40°	350	48,858	12.70	55%	10,039	17,965	130,185	179,043	2.66	153	339
Base	350	48,786	12.71	55%	10,032	17,952	138,648	187,434	2.84	154	342
Overall Slope Angle 30°	350	48,676	12.72	55%	10,019	17,929	145,225	193,900	2.98	154	339
Base Case	350	48,786	12.71	55%	10,032	17,952	138,648	187,434	2.84	154	342
4% Cutoff grade	350	47,800	12.90	55%	9,977	17,854	138,756	186,556	2.90	153	339
5% Cutoff grade	350	46,164	13.20	55%	9,856	17,638	138,046	184,210	2.99	152	348

Figure 16.2.2_1
Mining Cost Sensitivity

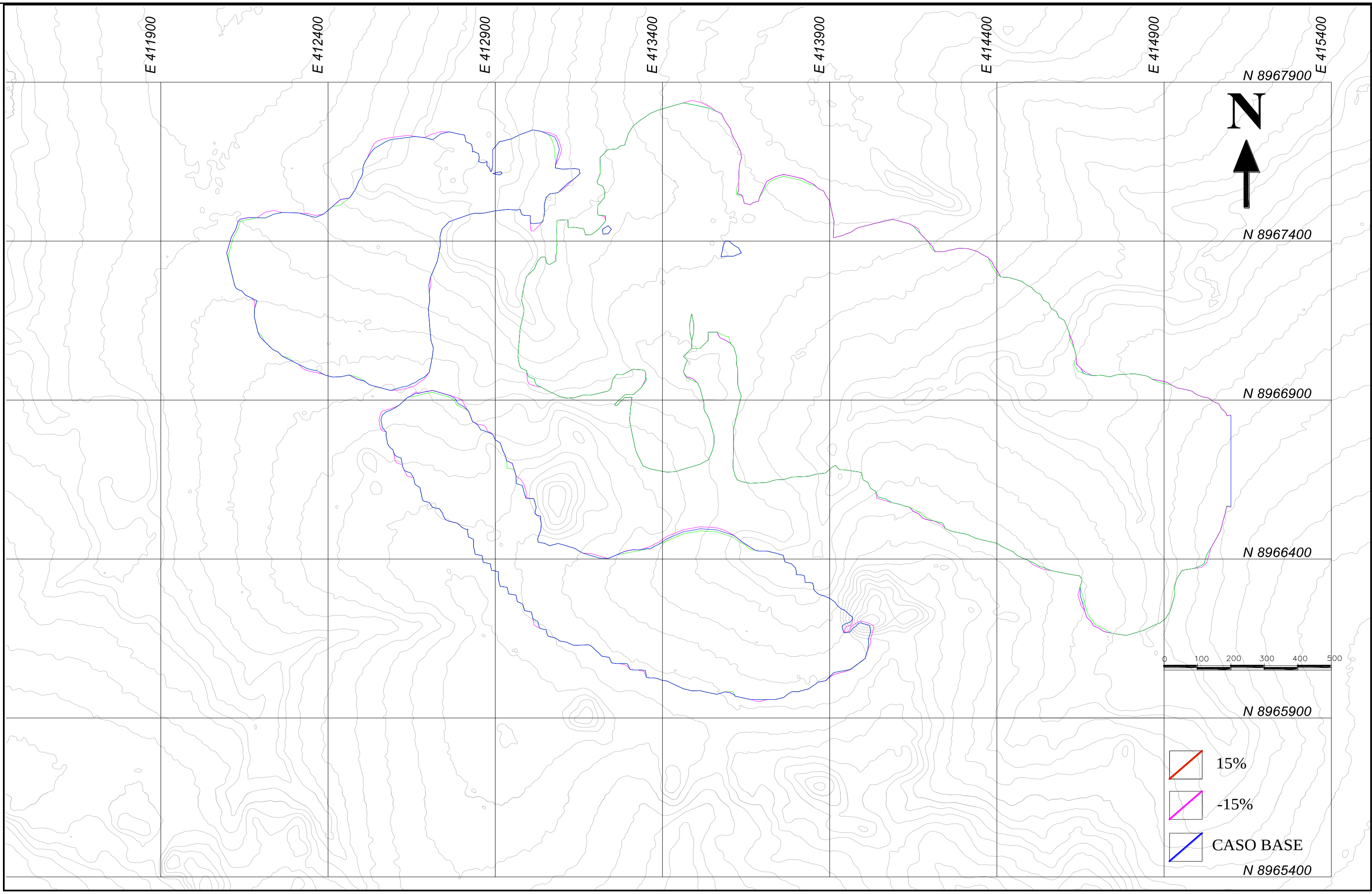


Figure 16.2.2_2
Process Cost Sensitivity

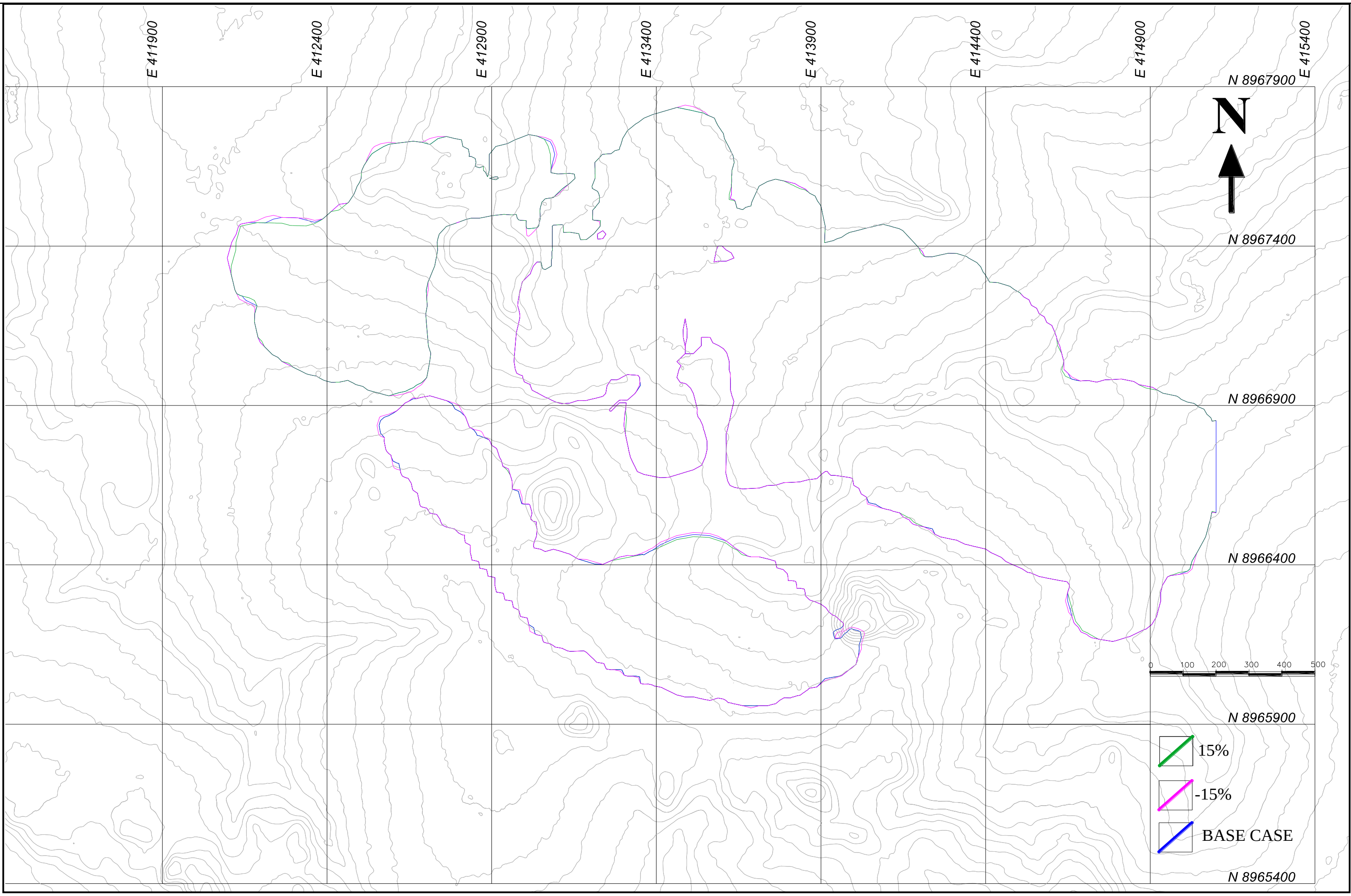


Figure 16.2.2_3
Recovery Sensitivity

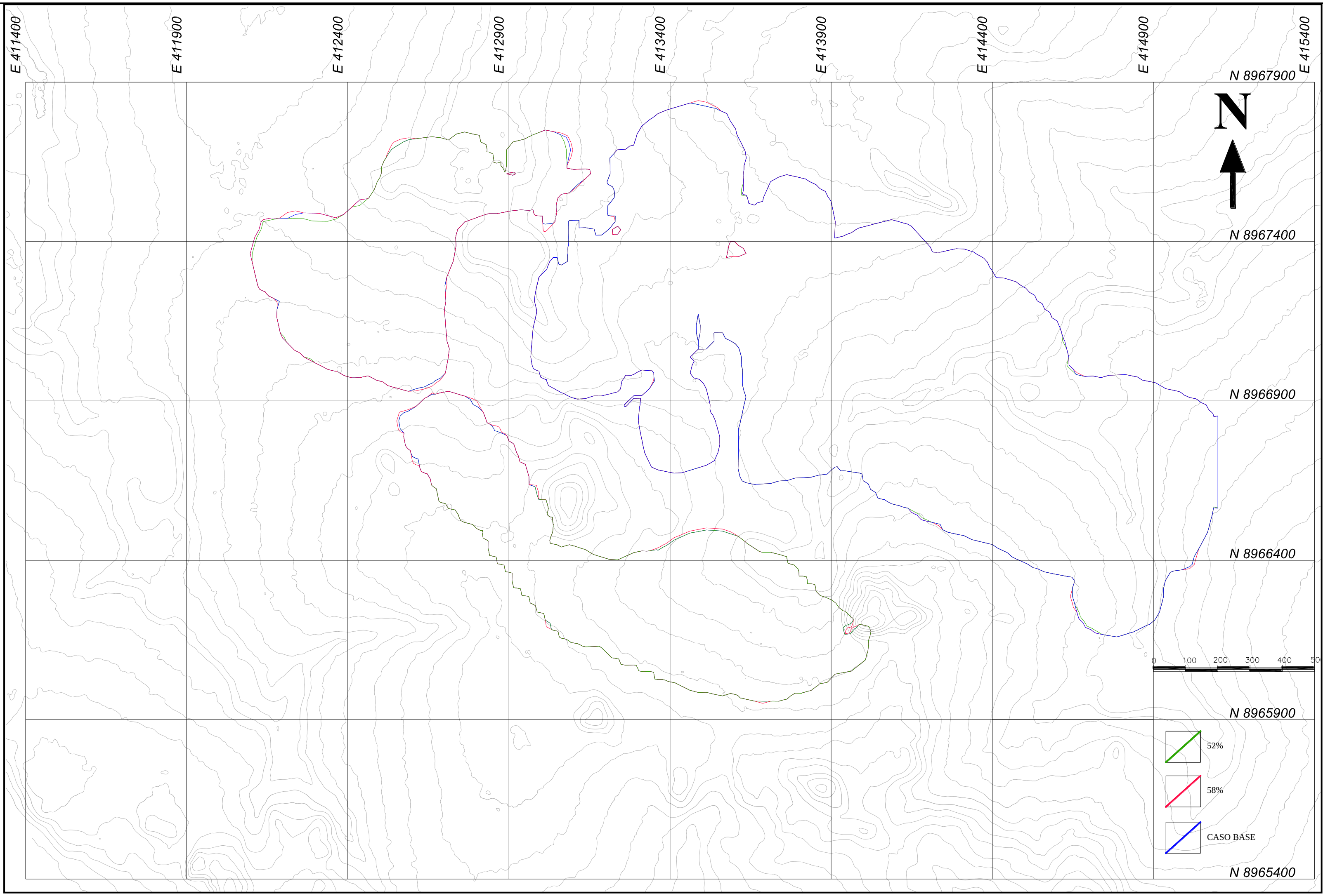


Figure 16.2.2_4
Slope Angle Sensitivity

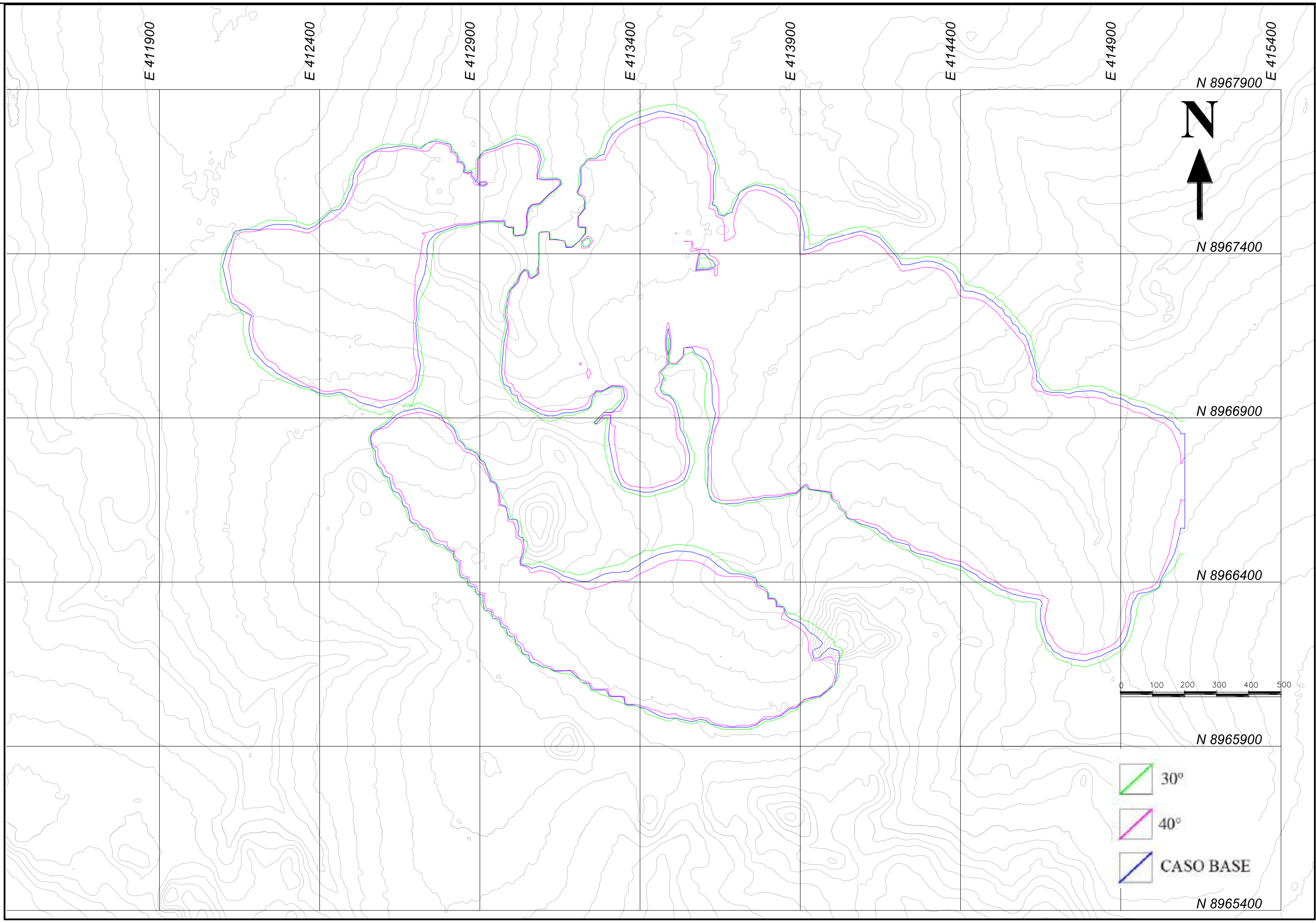
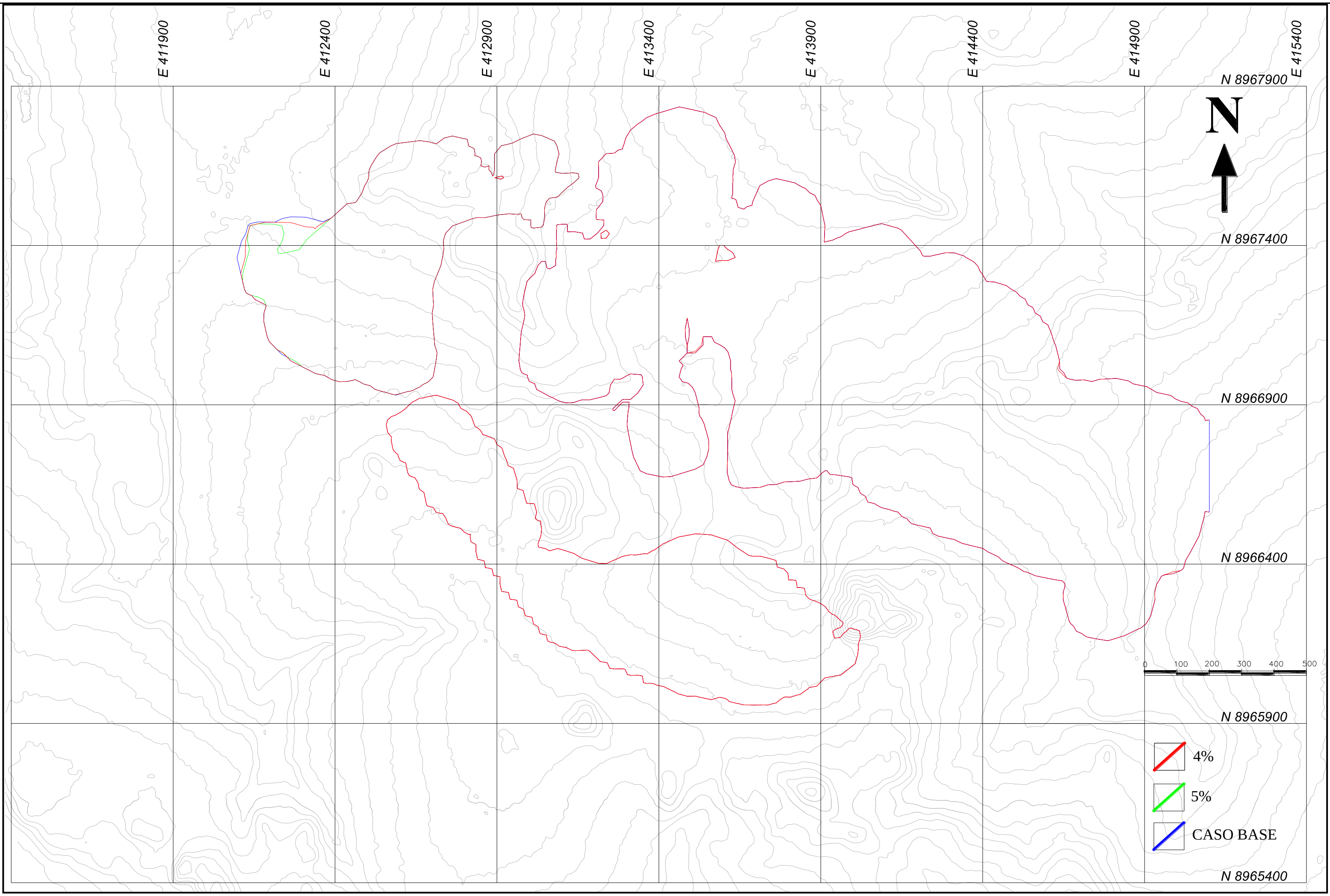


Figure 16.2.2_5
Cutoff Grade Sensitivity



16.3 Mine Design

16.3.1 Design Criteria

NCL established the following mine design criteria:

Reference optimization shell: The final pit design is based on the 350 USD/t SSP optimization price, to maximise economic value.

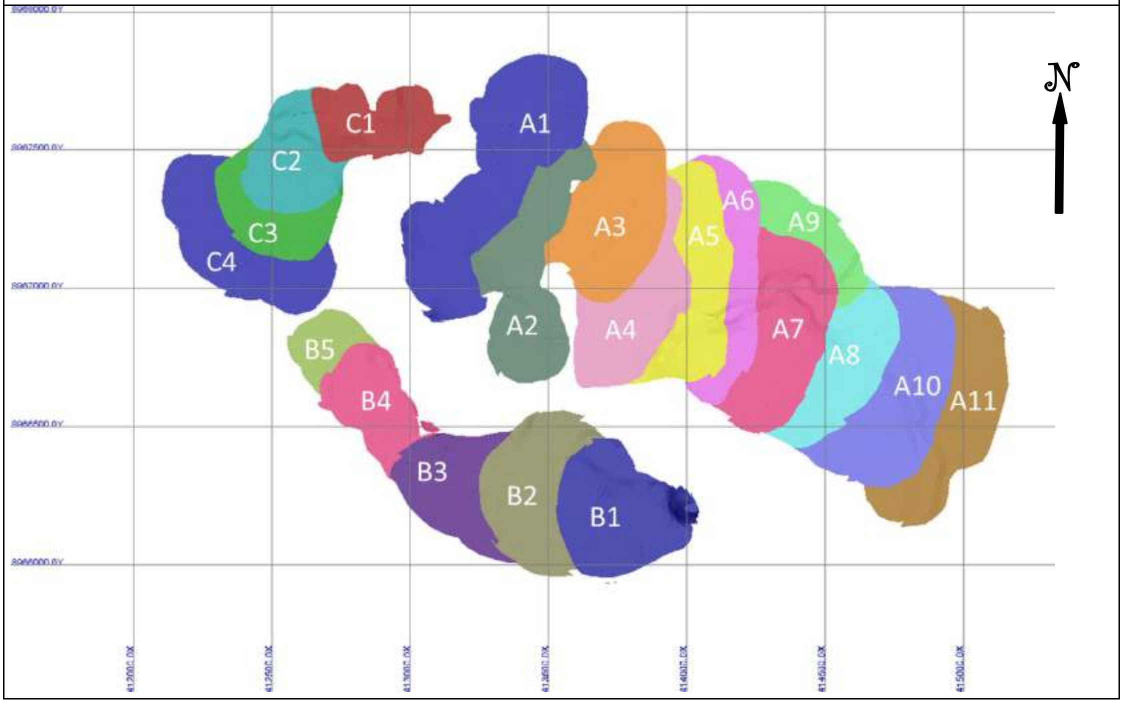
Minimum expansion width: 45m, which is adequate for the safe and efficient operation of the 5 m^3 hydraulic excavator and 5 m^3 front end loaders that have been chosen.

Haul Roads: The operational design uses 20m wide ramps with 10% gradient. These are adequate for the selected hauling fleet (32 t). Ramp widths comply with Brazilian regulations (NRM 13.6-c).

Geotechnical Parameters: The criteria and angles recommended by VOGBR in March, 2013 have been used for the design of each phase (VG13-081-1-GL-RTE-0001).

Operating phases are based on the incremental optimization shells sequence, targeting higher grades/ lower strip ratio areas first in order to reduce costs during the first years and increase the project´s NPV. The operational phases aims to provide with in pit dumping opportunities to reduce mining costs. To achieve this, each phase reaches to the bottom of the final pit and large incremental optimization shells have been partitioned to liberate dumping areas. Figure 16.3.1_1 shows the mining phases sequence.

Figure 16.3.1_1
Operational Phases Sequence



The major access routes for each phase are directed toward the primary crusher and waste dumps to minimize transport distances. In particular, given the high stripping ratio of the deposit; whenever possible waste haulage distance reduction was given preference as to reduce total mining cost.

16.3.2 Pit Design

Table 16.3.2_1 shows the final pit. The lower bench is at 192 m RL. The total area disturbed by the pits is approximately 2.8 km^2. Figure 16.3.1_1 shows the mineral resources contained inside the final pit.

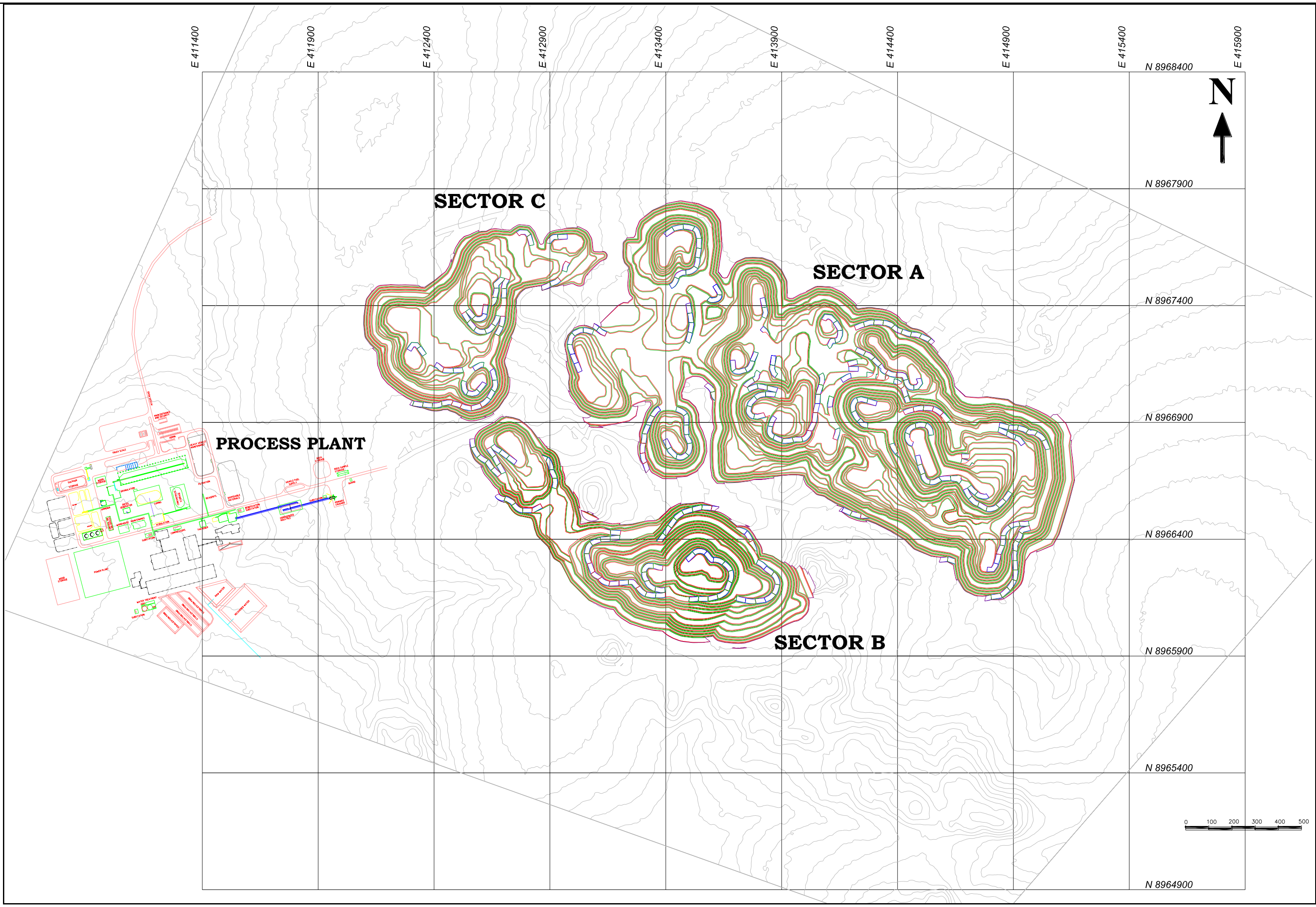
Table 16.3.2_1
In Pit Resources

	Ore									Waste	Total	
Cut-Off ($\%P_2O_5$)	Mass (kt)	P_2O_5 (%)	Al_2O_3 (%)	CaO (%)	Fe_2O_3 (%)	LOI (%)	MnO_2 (%)	SiO_2 (%)	TiO_2 (%)	Mass (kt)	Mass (kt)	Strip ratio
Total	45,481	12.86	6.53	20.97	13.73	7.58	2.20	31.96	0.58	143,225	188,706	3.15
Waste	-	-	-	-	-	-	-	-	-	143,225		
3%	45,481	12.86	6.53	20.97	13.73	7.58	2.20	31.96	0.58			
4%	44,835	12.99	6.49	21.12	13.70	7.57	2.22	31.76	0.58			
5%	43,407	13.27	6.43	21.40	13.65	7.52	2.26	31.39	0.57			
>6%	41,373	13.65	6.37	21.81	13.56	7.47	2.32	30.79	0.56			

Table 16.3.2_2 shows volumetrics by phase.

Table 16.3.2_2 Phase Volumetrics											
	Ore									Waste	Total
Phase	**Mass (kt)**	**P_2O_5 (%)**	**Al_2O_3 (%)**	**CaO (%)**	**Fe_2O_3 (%)**	**LOI (%)**	**MnO_2 (%)**	**SiO_2 (%)**	**TiO_2 (%)**	**Mass (kt)**	**Mass (kt)**
A Sector											
A1	2,870	15.56	6.64	24.90	12.88	9.68	3.89	22.12	0.53	8,051	10,922
A2	2,222	19.27	7.23	29.07	11.81	10.06	3.42	15.12	0.59	4,631	6,853
A3	1,318	18.57	5.19	27.11	14.96	8.04	3.41	17.43	0.47	8,074	9,392
A4	2,265	15.11	5.80	22.85	14.02	7.00	2.80	28.24	0.45	7,440	9,706
A5	3,600	14.05	6.83	21.50	14.36	7.63	3.03	28.41	0.53	7,298	10,898
A6	2,498	14.44	4.99	25.16	12.64	9.51	3.09	26.10	0.41	4,640	7,139
A7	4,806	11.61	7.32	22.08	10.96	9.22	2.29	31.59	0.51	11,292	16,098
A8	1,227	8.85	7.79	14.06	14.33	5.76	1.54	43.18	0.63	13,057	14,284
A9	265	12.93	6.57	23.15	9.71	8.14	1.62	33.31	0.42	4,969	5,235
A10	2,776	7.83	6.33	13.41	15.44	6.01	1.54	45.16	0.58	17,103	19,879
A11	2,735	7.37	5.82	12.69	17.46	5.81	1.71	44.52	0.66	12,124	14,858
Total	**26,582**	**12.97**	**6.48**	**21.26**	**13.60**	**8.05**	**2.65**	**30.56**	**0.53**	**98,680**	**125,262**
B Sector											
B1	4,755	15.80	6.56	26.24	12.28	8.97	1.69	25.55	0.52	4,200	8,955
B2	4,884	13.67	5.89	21.77	14.49	6.06	1.57	32.71	0.64	11,226	16,110
B3	3,176	13.17	6.95	18.28	16.52	4.65	1.27	34.84	0.85	5,320	8,496
B4	1,064	14.16	7.81	18.60	15.19	5.20	1.63	33.32	0.64	2,407	3,471
B5	636	8.83	8.62	11.55	14.75	4.21	1.39	46.53	0.72	1,610	2,247
Total	**14,515**	**14.08**	**6.60**	**21.79**	**14.27**	**6.56**	**1.54**	**31.48**	**0.65**	**24,763**	**39,278**
C Sector											
C1	519	18.39	6.43	26.05	9.96	5.64	4.28	24.98	0.43	1,664	2,183
C2	556	7.92	7.51	13.90	15.03	6.55	1.30	43.53	0.78	5,450	6,006
C3	980	7.65	6.69	15.83	14.15	8.00	1.38	42.16	0.71	4,264	5,244
C4	2,328	6.04	6.47	15.24	12.21	9.04	1.30	45.41	0.62	8,405	10,733
Total	**4,383**	**8.10**	**6.65**	**16.48**	**12.73**	**8.09**	**1.67**	**42.03**	**0.64**	**19,782**	**24,165**
Total											
Total	**45,481**	**12.86**	**6.53**	**20.97**	**13.73**	**7.58**	**2.20**	**31.96**	**0.58**	**143,225**	**188,706**

Figure 16.3.2_1
Final Pit Design



16.4 Mine Production Schedule

A mine production schedule was developed to show the ore mass, grades, total material and waste material by year throughout of the life of the mine. The distribution of ore and waste contained inside the final pit was used to develop the schedule, assuring that criteria such as continuous ore exposure, mining accessibility, and consistent material movements were met.

NCL used an in-house developed system to evaluate several potential production mine schedules. Required annual ore mass and user specified annual total material movements are provided to the algorithm, which then calculates the mine schedule. Several runs at various proposed total material movement schedules were done to determine a good production schedule strategy. It is important to note that this program is not a simulation package, but a tool for calculation of the mine schedule and haulage profiles for a given set of phases and constraints that must be set by the user.

The mine plan developed by NCL does not include any special provisions for dilution because the resource block model is considered as already diluted.

Nevertheless, careful grade control must be carried out during mining to minimize misplaced ore due to the important effect of head grade on phosphate concentrate production. These efforts should include, among others, the following standard procedures:

- Implement an intense and systematic program of sampling, mapping, laboratory analyses, and reporting.
- Utilize specialized in-pit bench sampling drills for sampling well ahead of production.
- Use of excavators and benches no higher than 4 m (as presently planned) to selectively mine ore zones.
- Maintain top laboratory staff, equipment, and procedures to provide accurate and timely assay reporting.
- Utilize trained geologists and technicians to work with excavator operators in identifying, marking, and selectively mining and dispatching ore and waste.
- Maintain an adequate communication and control system between excavator and truck operators and the mine dispatcher to ensure that the materials are sent to the appropriate destination.

A minimum cutoff grade of 3% P_2O_5 was selected for strategic reasons; and is well above the minimum economic Cut-Off grade.

The Santana Project mine schedule produces 300 kt per year of phosphate concentrate at a grade of 34% P_2O_5. Total material movement rate varies from 3.2 Mtpy at the start of the project up to a maximum of 9 Mt in later years. This scenario results in the processing of an average of 1.5 Mtpy of ore with an average P_2O_5 grade of 12.86%. The expected mine life for the project is 32 years besides the pre stripping period. Given the favorable ore presentation, there is no need for long-term stockpiles.

Table 16.4_1 shows the new surface area opened each period.

Table 16.4_2 shows the plant feed per period.

Table 16.4_3 shows the run of mine movements.

Table 16.4_4 shows the advance rate in benches per period (4 m benches).

Table 16.4_5 shows the ore extracted by phase by period.

Table 16.4_6 shows total mined by phase by period.

Table 16.4_1
Surface area

	Area
Period	**(´000 m2)**
PP	223
P01	135
P02	110
P03	11
P04	162
P05	21
P06	295
P07	0
P08	54
P09	52
P10	38
P11	109
P12	0
P13	210
P14	166
P15	4
P20	584
P25	255
P30	378
P32	0
Total	**2,809**

Table 16.4_2 Plant Feed											
Period	**Mass**	**P_2O_5**	**CaO**	**Al_2O_3**	**Fe_2O_3**	**SiO_2**	**TiO_2**	**LOI**	**Al_2O_3+Fe_2O_3**	**CaO/P_2O_5**	**Conc**
	(kt)	**(%)**	**(%)**	**(%)**	**(%)**	**(%)**	**(%)**	**(%)**	**(%)**		**(kt)**
PP	-	-	-	-	-	-	-	-	-	-	-
Y01	880	21.08	28.25	9.55	13.22	11.12	0.66	9.67	22.77	1.36	300
Y02	971	19.09	28.48	7.70	12.31	14.46	0.59	10.10	20.01	1.50	300
Y03	1,065	17.41	25.04	8.18	12.46	21.93	0.62	8.19	20.64	1.44	300
Y04	1,204	15.40	23.00	7.58	13.65	26.10	0.58	7.88	21.23	1.50	300
Y05	1,222	15.17	22.27	6.90	13.11	27.99	0.65	7.27	20.00	1.48	300
Y06	1,200	15.46	26.29	4.63	10.80	25.34	0.37	9.20	15.43	1.70	300
Y07	1,127	16.45	26.88	6.69	12.45	23.02	0.51	10.10	19.14	1.64	300
Y08	1,196	15.51	24.69	6.08	13.85	22.52	0.50	9.43	19.93	1.61	300
Y09	1,196	15.50	30.15	5.21	11.63	19.79	0.42	12.28	16.85	2.24	300
Y10	1,171	15.83	26.95	5.78	14.02	22.03	0.48	8.81	19.80	1.80	300
Y11	1,160	15.98	24.05	5.37	14.94	24.98	0.48	7.09	20.30	1.52	300
Y12	1,206	15.38	23.47	5.50	12.66	31.66	0.52	5.78	18.15	1.60	300
Y13	1,196	15.51	22.50	5.83	12.61	32.98	0.50	5.54	18.44	1.56	300
Y14	1,214	15.27	22.93	5.42	14.82	25.43	0.47	7.96	20.24	1.50	300
Y15	1,171	15.84	22.52	6.85	14.29	27.36	0.56	5.87	21.15	1.42	300
Y16	1,257	14.76	21.40	7.04	13.68	30.88	0.56	6.50	20.72	1.46	300
Y17	1,407	13.18	20.88	6.28	13.41	33.54	0.55	6.94	19.69	1.65	300
Y18	1,536	12.08	20.84	6.11	13.34	32.10	0.50	8.61	19.45	1.78	300
Y19	1,367	13.57	21.65	5.18	12.84	32.71	0.52	7.26	18.02	1.69	300
Y20	1,414	13.12	25.67	5.06	11.94	27.97	0.49	10.15	17.00	2.10	300
Y21	1,573	11.79	23.29	6.72	11.61	30.75	0.53	9.84	18.32	1.99	300
Y22	1,660	11.17	21.42	7.48	11.21	33.12	0.53	9.04	18.69	1.94	300
Y23	1,935	9.58	15.56	8.86	13.17	39.28	0.64	6.09	22.04	1.61	300
Y24	1,746	10.62	20.53	6.22	12.01	36.48	0.53	8.14	18.23	1.96	300
Y25	1,565	11.85	20.91	5.64	15.48	32.99	0.65	7.18	21.12	1.83	300
Y26	1,640	11.31	16.97	7.10	14.90	38.06	0.69	5.81	22.00	1.73	300
Y27	1,694	10.95	15.53	7.42	15.31	40.95	0.70	5.19	22.73	1.51	300
Y28	1,948	9.52	16.41	6.02	15.51	40.96	0.65	6.38	21.53	1.84	300
Y29	1,978	9.38	15.41	6.82	16.50	39.22	0.83	5.86	23.32	1.80	300
Y30	1,936	9.58	17.50	5.90	14.76	38.98	0.55	7.45	20.67	2.04	300
Y31	1,983	7.16	13.56	6.66	15.76	43.65	0.74	7.08	22.42	2.03	230
Y32	1,662	8.53	12.63	7.17	16.37	43.90	0.68	4.94	23.54	1.49	229
Total	**45,481**	**12.86**	**20.97**	**6.53**	**13.73**	**31.96**	**0.58**	**7.58**	**20.26**	**1.72**	**9,459**

Table 16.4_3
Long Term Mine Schedule – ROM Movements

	Plant Feed						Production Ore Stock		Pre Stripping Ore Stock		Waste	Total ROM	Total Moved
	From Pit		From Stock		Total								
Per.	Mass (kt)	P_2O_5 (%)	Mass (kt)	P_2O_5 (%)	Mass (kt)	P_2O_5 (%)	Mass (kt)	P_2O_5 (%)	Mass (kt)	P_2O_5 (%)	Kt	Kt	Kt
PP	-	-	-	-	-	-	-	-	183	25.43	1,442	1,625	1,625
Y01	697	19.94	183	25.43	880	21.08	107	19.94	-	-	2,396	3,200	3,383
Y02	971	19.09	-	-	971	19.09	55	19.09	-	-	2,274	3,300	3,300
Y03	1,066	17.41	-	-	1,065	17.41	49	17.41	-	-	2,986	4,100	4,100
Y04	1,204	15.40	-	-	1,204	15.40	57	15.40	-	-	2,839	4,100	4,100
Y05	1,188	15.08	35	18.33	1,222	15.17	-	-	-	-	3,412	4,600	4,635
Y06	1,200	15.46	-	-	1,200	15.46	39	15.46	-	-	3,361	4,600	4,600
Y07	1,127	16.45	-	-	1,127	16.45	14	16.45	-	-	3,459	4,600	4,600
Y08	1,147	15.41	49	17.85	1,196	15.51	-	-	-	-	3,453	4,599	4,648
Y09	1,174	15.46	23	17.85	1,196	15.50	-	-	-	-	3,426	4,600	4,623
Y10	1,171	15.83	-	-	1,171	15.83	25	15.83	-	-	3,404	4,600	4,600
Y11	1,151	15.97	9	17.64	1,160	15.98	-	-	-	-	3,449	4,600	4,609
Y12	1,206	15.38	-	-	1,206	15.38	42	15.38	-	-	4,753	6,000	6,000
Y13	1,186	15.50	10	17.29	1,196	15.51	-	-	-	-	4,814	6,000	6,010
Y14	1,214	15.27	-	-	1,214	15.27	5	15.27	-	-	4,781	6,000	6,000
Y15	1,171	15.84	-	-	1,171	15.84	19	15.84	-	-	4,810	6,000	6,000
Y16	1,230	14.71	27	17.16	1,257	14.76	-	-	-	-	4,770	6,000	6,027
Y17	1,395	13.15	12	17.16	1,407	13.18	-	-	-	-	4,605	6,000	6,012
Y18	1,536	12.08	-	-	1,536	12.08	46	12.08	-	-	4,419	6,000	6,000
Y19	1,347	13.53	19	16.37	1,367	13.57	-	-	-	-	4,653	6,000	6,019
Y20	1,414	13.12	-	-	1,414	13.12	10	13.12	-	-	4,576	6,000	6,000
Y21	1,559	11.75	14	16.26	1,573	11.79	-	-	-	-	4,441	6,000	6,014
Y22	1,631	11.08	29	16.26	1,660	11.17	-	-	-	-	4,369	6,000	6,029
Y23	1,930	9.57	5	16.26	1,935	9.58	-	-	-	-	4,070	6,000	6,005
Y24	1,746	10.62	-	-	1,746	10.62	44	10.62	-	-	7,210	9,000	9,000
Y25	1,508	11.72	57	15.37	1,565	11.85	-	-	-	-	7,492	9,000	9,057
Y26	1,640	11.31	-	-	1,640	11.31	77	11.31	-	-	7,283	9,000	9,000
Y27	1,655	10.87	39	14.33	1,694	10.95	-	-	-	-	7,345	9,000	9,039
Y28	1,948	9.52	-	-	1,948	9.52	17	9.52	-	-	7,035	9,000	9,000
Y29	1,956	9.32	23	14.03	1,978	9.38	-	-	-	-	7,044	9,000	9,023
Y30	1,936	9.58	-	-	1,936	9.58	63	9.58	-	-	5,276	7,275	7,275
Y31	1,913	6.95	70	13.16	1,983	7.16	-	-	-	-	2,489	4,401	4,471
Y32	1,413	7.71	249	13.16	1,662	8.53	-	-	-	-	1,092	2,505	2,754
Total	44,630	12.77	851	17.15	45,481	12.86	669	14.88	183	25.43	143,225	188,706	189,557

Table 16.4_4

Advance Rate (1)

Phase	A1	A2	A3	A4	A5	A6	A7	A8	A9	A10	A11	B1	B2	B3	B4	B5	C1	C2	C3	C4
Period	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr	#/yr
PP	3	3																		
Y01	2	1																		
Y02	1	2																		
Y03	1	1										4								
Y04	1	1										2								
Y05	2	6	3									0					5			
Y06	8		2									0					0			
Y07			3	1								2					0			
Y08			5	1								1					3			
Y09			3	4								2								
Y10			3	3								3						4		
Y11				4								1						3		
Y12				3	5							4						8	2	
Y13				7	3								5					2	1	
Y14					4								1						6	
Y15					2	8							2						2	
Y16					4	2	6						2						0	
Y17					2	1	2						3						1	
Y18					6	2	1	7					0						0	
Y19						1	3	2					0						1	3
Y20						5	0	1					0						8	3
Y21							3	1	5				1							1
Y22							4	0	6				1							0
Y23							6	3	1				2							0
Y24							5	7	8				3							2
Y25								8		7			8							1
Y26								4		3			3	6						4
Y27										5	4			4						1
Y28										5	4			4						1
Y29										8	3			7	4					1
Y30										7	6			5	4					1
Y31											5				4	6				3
Y32											8					7				

(1) 4 m high benches

Table 16.4_5
Ore Extraction By Phase

Phase	A1	A2	A3	A4	A5	A6	A7	A8	A9	A10	A11	B1	B2	B3	B4	B5	C1	C2	C3	C4
Period	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt
PP	9	174																		
Y01	401	403																		
Y02	406	620																		
Y03	239	311										565								
Y04	256	291										715								
Y05	518	423										53					195			
Y06	1,042		24									157					16			
Y07			153									968					20			
Y08			554									304					289			
Y09			492									682								
Y10			95	369								732								
Y11				961								145						46		
Y12				343								435						469		
Y13				592	155								398					42		
Y14					1,092								108						19	
Y15					715	45							245						185	
Y16					717	216	6						248						43	
Y17					355	226	125						589						99	
Y18					565	751	105	41					65						54	
Y19						326	628	93					33						268	
Y20						934	57	64					55						312	1
Y21							1,146	99					309							6
Y22							1,358	31	20				213							9
Y23							1,042	343	18				519							8
Y24							339	384	227				723							118
Y25								141		1			1,235							131
Y26								31		52			145	635						855
Y27										621				918						116
Y28										1,225	0			586						155
Y29										646	48			814	190					257
Y30										231	829			223	473					242
Y31											1,038				401	43				432
Y32											819					593				

Table 16.4_6
Movement By Phase

Phase	A1	A2	A3	A4	A5	A6	A7	A8	A9	A10	A11	B1	B2	B3	B4	B5	C1	C2	C3	C4
Period	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt	kt
PP	300	1,325																		
Y01	1,800	1,400																		
Y02	1,800	1,500																		
Y03	1,500	900										1,700								
Y04	1,450	800										1,850								
Y05	1,500	928	1,100									100					972			
Y06	2,572		1,630									300					98			
Y07			2,700	100								1,700					100			
Y08			2,850	237								500					1,012			
Y09			900	2,700								1,000								
Y10			212	2,300								1,000						1,088		
Y11				2,500								200						1,900		
Y12				900	1,500							605						2,850	145	
Y13				969	2,600								1,800					168	463	
Y14					3,000								600						2,400	
Y15					1,200	2,700							1,400						700	
Y16					1,300	1,000	2,300						1,300						100	
Y17					500	500	2,300						2,500						200	
Y18					798	1,100	800	3,000					200						102	
Y19						400	2,800	1,800					100						400	500
Y20						1,439	200	900					170						734	2,557
Y21							2,700	800	800				900							800
Y22							2,550	200	2,400				600							250
Y23							1,900	2,000	400				1,500							200
Y24							548	3,500	1,635				2,000							1,318
Y25								1,800		3,900			2,700							600
Y26								284		3,000			340	2,800						2,577
Y27										5,000	1,500			2,200						300
Y28										4,000	3,200			1,500						300
Y29										3,000	2,700			1,600	1,300					400
Y30										979	4,000			396	1,600					300
Y31											2,100				571	1,100				631
Y32											1,358					1,147				

16.5 Waste Management Facilities

Three ex pits waste dumps will be used to store the mine´s waste (Figure 16.5_1); these will be constructed according to the design parameters presented in Section 16.1.4. Additionally, waste will be stored in the exhausted areas of the mine to reduce mining costs. Waste will be dumped close to the dump´s edge and will be pushed over by bulldozer (Figure 16.5_2).

Periodical inspections should assess the stability of the in pit dumps, looking for cracks in the surface and other signs of potential instability.

Table 16.5_1 presents the new area that must be prepared each year in the ex-pit waste dumps. This excludes the benches over previously prepared area.

Table 16.5_2 shows the waste dumps filling sequence.

Table 16.5_1
New Dump Area

Period	Waste dump 1 (´000 m^2)	Waste dump 2 (´000 m^2)	Waste dump 3 (´000 m^2)
PP	97	0	126
P01	51	0	147
P02	135	0	19
P03	106	0	46
P04	120	0	85
P05	0	84	162
P06	0	0	0
P07	85	0	10
P08	0	3	0
P09	0	0	0
P10	0	15	0
P11	0	0	40
P12	0	0	0
P13	0	24	0
P14	63	0	0
P15	49	0	0
P20	731	120	0
P25	82	47	0
P30	90	0	0
P32	0	0	0
Total	**1,610**	**294**	**635**

Note: ´000 m2 equals thousands of square meters.

Table 16.5_2 Waste Dump Fill Sequence					
	Waste dump 1	Waste dump 2	Waste dump 3	In pit dumps	Total
Period	**kt**	**kt**	**kt**	**kt**	**kt**
PP	702	11	730	0	1,442
Y01	904	0	1,492	0	2,396
Y02	1,884	0	390	0	2,274
Y03	2,264	0	722	0	2,986
Y04	1,872	0	967	0	2,839
Y05	1,187	548	1,677	0	3,412
Y06	1,432	88	762	1,079	3,361
Y07	0	25	0	3,434	3,459
Y08	0	31	0	3,422	3,453
Y09	0	0	0	3,426	3,426
Y10	0	0	1,088	2,316	3,404
Y11	0	0	1,727	1,722	3,449
Y12	0	48	1,714	2,990	4,753
Y13	1,092	0	590	3,132	4,814
Y14	808	0	0	3,973	4,781
Y15	3,749	33	255	773	4,810
Y16	4,713	0	57	0	4,770
Y17	4,504	0	101	0	4,605
Y18	2,283	2,087	48	0	4,419
Y19	69	0	630	3,953	4,653
Y20	115	0	2,978	1,483	4,576
Y21	591	800	794	2,256	4,441
Y22	387	2,320	241	1,421	4,369
Y23	1,587	0	192	2,291	4,070
Y24	2,752	0	392	4,066	7,210
Y25	4,197	237	0	3,058	7,492
Y26	2,369	0	1,085	3,829	7,283
Y27	2,418	0	173	4,754	7,345
Y28	675	0	105	6,255	7,035
Y29	186	0	106	6,752	7,044
Y30	1,714	0	46	3,517	5,276
Y31	0	0	0	2,489	2,489
Y32	0	0	0	1,092	1,092
Total	**44,453**	**6,226**	**19,061**	**73,485**	**143,225**

Figure 16.5_1
Waste Dumps Layout

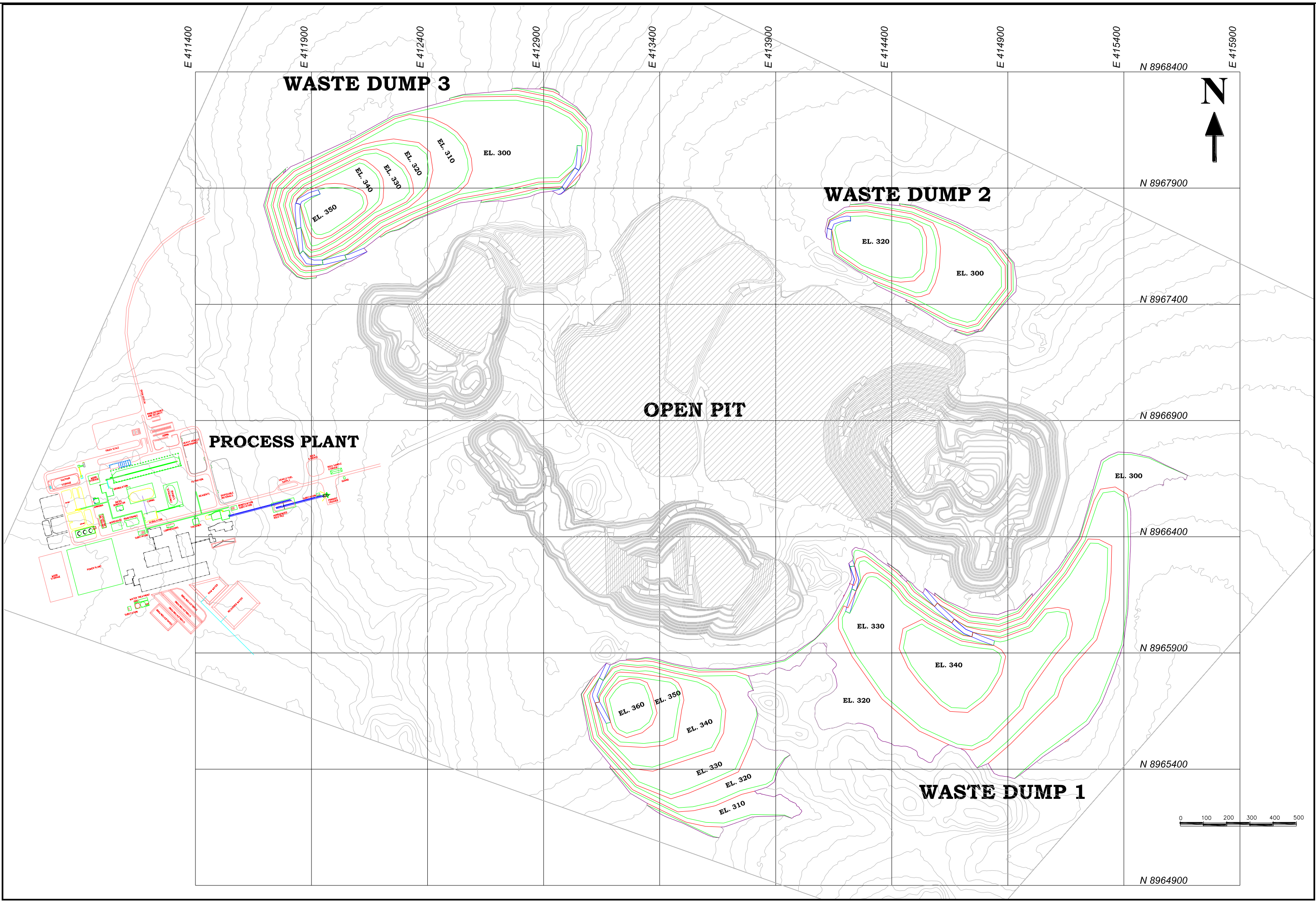
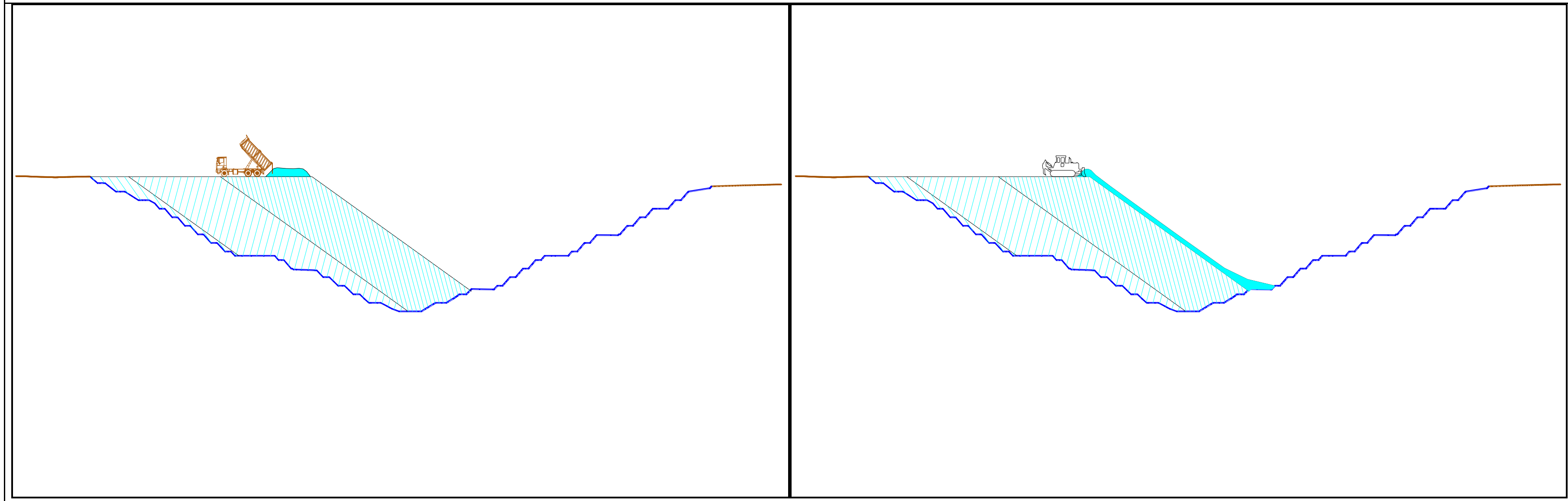


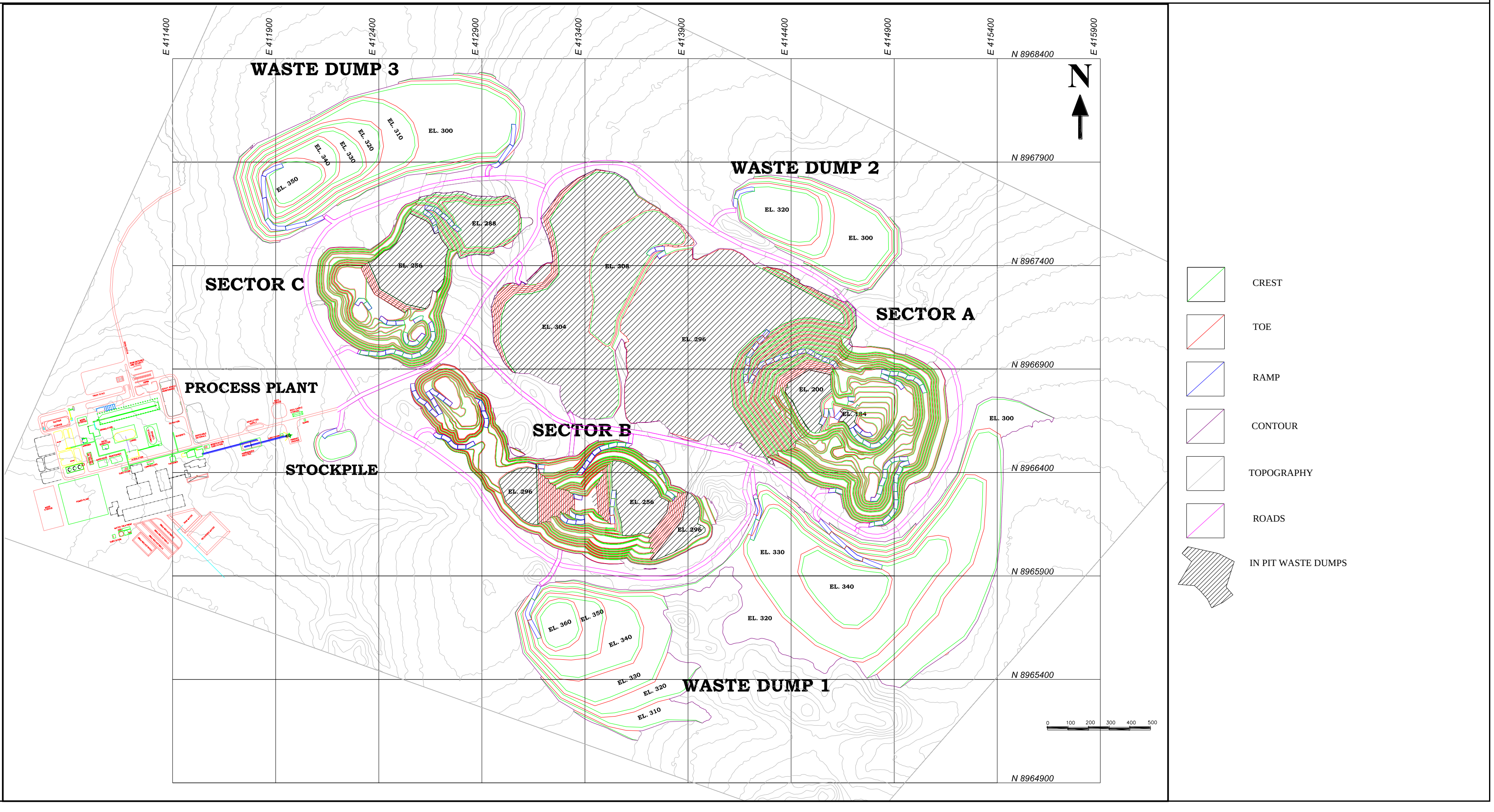
Figure 16.5_2
In Pit Waste Dump Construction



16.6 Mine General Layout

Figure 16.6_1 shows the mine´s general layout.

Figure 16.6_1
Mine General Layout



16.7 Equipment Fleet

The study is based on operating the Santana mine with 5 m3 backhoe excavators and front-end loaders and 32 t conventional haul trucks. The current situation at mines in Brazil is that contractors are using this type of equipment at low costs for material movement up several times the scheduled mining rate. Auxiliary equipment includes bulldozers, motor graders, and water trucks. Table 16.7_1 shows the selected equipment fleet characteristics.

Table 16.7_1
Equipment Fleet

Equipment	Specification	Reference Model
Drill Rig	4 1/2" Diameter	Sandvik DR540
Hydraulic Excavator	5 m³ bucket	Liebherr R964C
Front End Loader	5 m³ bucket	Liebherr L580 2+2
Haul Truck	32 tonne/ 20 m³	Scania G440 8X4 + 20m³ Rossetti Dump
Bulldozer	264 kW	Komatsu D155AX-7
Motor grader	136 kW	Caterpillar 140M
Water Truck	20m³	Scania P360 6X4 + 20m³ JAB Water Tank
Fuel Truck	20m³	Scania P360 6X4 + 20m³ JAB Fuel Tank
Lube Truck	5.000 L	Scania P360 6X4 + JAB 5.000lt Capacity Lube implement
Rock Breaker	330 kg Impact Hammer	330 Kg Impact Hammer + Volvo BL70B Backhoe
Lowboy Truck	150 t	Scania G440 CA 6X4 + 100 tonne Capacity Low Bed
Pickup	4X4 Diesel	Mitsubishi L200 Triton GL
Exploration Rig	4 1/2" Diameter	Atlascopco Explorac R50
Lightning Tower	4x1kW	Atlas Copco QLT M20 Lightining Tower

Main equipment requirements were estimated using the First Principles methodology, which consist in estimating the amount of work required based on the mining schedule and each equipment productivity, thus calculating the required work hours needed. Applying expected mechanical availability and utilization, these work hours are converted into required equipment quantities. Ancillary and support fleets' requirements were estimating using NCL experience in similar projects, based on main equipment requirements and characteristics of the Santana Project.

The mine will operate on a 365 days schedule, with three 8 hours shifts. Five days of unscheduled downtime have been considered, mainly to account for climate-associated delays, of which the more common is reduced visibility due to heavy rain.

16.7.1 Drilling and Blasting

Most of the material in Santana can be mechanically excavated (95%), only a small fraction requiring explosives. Given the small quantities, packaged explosives will be used. Table 16.7.1_1 shows the drilling parameters and Table 16.7.1_2 the blasting parameters to be used in the Santana project.

Table 16.7.1_1
Drilling Parameters

Material		Fresh Rock
Drill Diameter	mm	114
Bench Height	m	8.00
Subdrill	m	0.91
Stemming	m	2.97
Burden	m	4.00
Spacing	m	4.60
Specific drilling	m3/m	16.52
Redrill	%	0.05
Penetration rate	m/h	49.30
BCM per hole	m3/hole	147.20
Bit Life	m	3,500

Table 16.7.1_2
Blasting Parameters

Material		Fresh Rock
Drill Diameter	mm	114
Average Rock Density	t/m^3	2.69
Explosive Density	t/m^3	1.20
Bench Height	m	8.00
Burden	m	4.00
Spacing	m	4.60
Subdrill	m	0.91
Hole Length	m	8.91
Stemming Length	m	2.97
Explosive Column Length	m	5.94
Column Charge	kg/m	12.31
Total Charge	kg	73.14
Powder Factor	kg/m^3	0.50

16.7.2 Haul Profiles

Average transport distances have been estimated for each kind of material, source and destination combination. This estimate has been carried out using the MineHaul software, an in house developed package that determines distances by choosing the best route for each material from its source in the mine up to any of the possible destinations, in this way minimizing transport cycle times. The software complies with material type, capacity and connectivity constrains.

The software normalizes the distance to horizontal, up 10% and down 10% segments. A segment with a grade less than 10% (up or down) is decomposed into a flat and 10% segments. When grades are greater than 10% the segment length is increased to its 10% equivalent.

Table 16.7.2_1 shows the average truck speed for each segment type. These are based on NCL experience with mines in similar operating conditions.

Table 16.7.2_1
Average Haul Truck Speeds

Item	Units	Value
Loaded Up	km/h	15
Loaded Down	km/h	18
Loaded Flat	km/h	35
Empty Up	km/h	30
Empty Down	km/h	30
Empty Flat	km/h	35

Table 16.7.2_2 and Figure 16.7.2_1 show the average haul profile throughout the mine life.

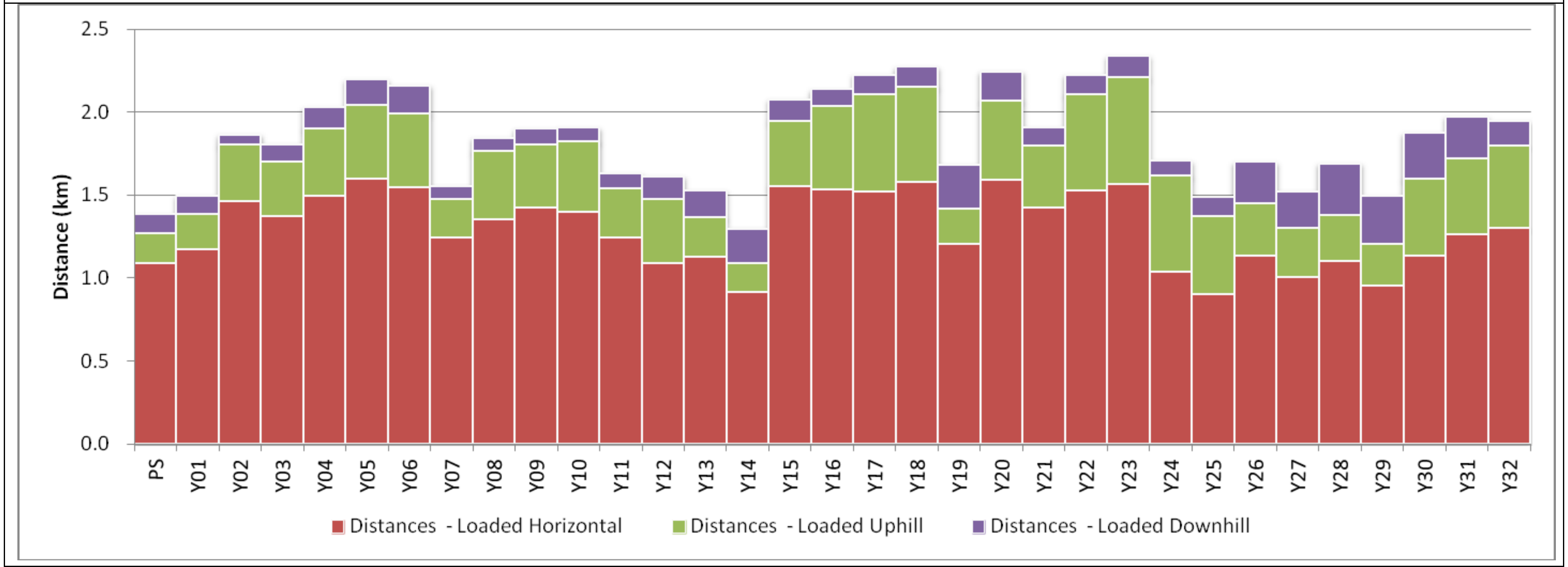
Table 16.7.2_2

Haul Distances

Leg	un	PS	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Loaded Flat	m	1,090	1,171	1,460	1,370	1,493	1,601	1,548	1,242	1,356	1,424	1,401	1,241	1,087	1,126	915	1,551	1,530
Loaded Up	m	177	214	343	329	408	441	445	231	406	383	422	297	390	238	174	394	509
Loaded Down	m	119	110	62	108	127	155	167	81	83	92	84	93	136	164	208	132	97
Empty Down	m	177	214	343	329	408	441	445	231	406	383	422	297	390	238	174	394	509
Empty Up	m	119	110	62	108	127	155	167	81	83	92	84	93	136	164	208	132	97
Empty Flat	m	1,090	1,171	1,460	1,370	1,493	1,601	1,548	1,242	1,356	1,424	1,401	1,241	1,087	1,126	915	1,551	1,530

Leg	un	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27	Y28	Y29	Y30	Y31	Y32
Loaded Flat	m	2,226	2,273	1,684	2,245	1,907	2,226	2,339	1,705	1,489	1,699	1,523	1,690	1,492	1,874	1,973	1,947
Loaded Up	m	1,522	1,579	1,206	1,588	1,424	1,528	1,566	1,034	903	1,132	1,003	1,098	951	1,131	1,265	1,301
Loaded Down	m	588	572	212	481	373	579	645	581	471	316	296	279	250	466	455	494
Empty Down	m	116	121	267	176	110	118	129	89	115	251	224	313	291	276	253	152
Empty Up	m	588	572	212	481	373	579	645	581	471	316	296	279	250	466	455	494
Empty Flat	m	116	121	267	176	110	118	129	89	115	251	224	313	291	276	253	152

Figure 16.7.2_1
Average Haul Distances – Loaded Leg



16.7.3 Equipment Fleet Estimation

Figure 16.6_1 shows the equipment fleet required for the mine operation. This represents the equipment necessary to perform the following duties:

- Construct haul and access roads to the initial mining areas as well as to the crusher, waste storage areas, and leach pads. Construct additional roads as needed to support mining activity.
- Perform the preproduction development required to expose ore for initial production.
- Develop new mining areas for ore extraction.
- Mine and transport ore to the crusher.
- Mine and transport waste material from the pit to the appropriate storage areas.
- Maintain all the mine work areas, in-pit haul roads, external haul roads, and maintain the waste storage areas.
- Load and transport topsoil to topsoil storage areas.

Table 16.7.3_1
Required Equipment Fleet

Period	PS	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Main Equipment																	
Production Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Front End Loader	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2
Haul truck	7	7	7	7	7	8	8	8	8	8	8	8	9	9	9	11	11
Ancillary Equipment																	
Bulldozer	2	3	3	3	3	3	3	4	4	4	4	3	3	3	3	3	3
Motor grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2
Support Equipment																	
Fuel Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Rock Breaker	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lube Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Lowboy Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pickup	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Exploration Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Tower	8	8	8	8	8	9	11	11	11	9	8	10	12	12	12	12	9

	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27	Y28	Y29	Y30	Y31	Y32
Main Equipment																
Production Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Excavator	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Front End Loader	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Haul truck	11	11	11	11	11	11	11	12	12	14	14	14	13	13	8	6
Ancillary Equipment																
Bulldozer	3	3	3	3	3	3	3	4	4	3	3	3	3	3	3	3
Motor grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Support Equipment																
Fuel Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Rock Breaker	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lube Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Lowboy Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pickup	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Exploration Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Tower	9	10	10	10	11	11	11	11	12	12	11	11	11	11	9	9

16.8 Mine Personnel

16.8.1 Administrative and Technical Staff

Administrative and technical staff consists of 27 persons through the life of mine. The salaried staff requirements are commensurate with the complexity of a mine the size of Santana.

Table 16.8.1_1 Administrative and Technical Staff	
Position	**#**
Management	**3**
Mine Manager	1
Administrative Assistant	1
Trainee	1
Operations	**10**
Mine Coordinator	1
Mining Engineer	1
Mine Technician	1
Mine Supervisor	4
Production Assistant	2
Production Foreman	1
Maintenance	**7**
Vehicle Machine Shop Coordinator	1
Vehicle Maintenance Coordinator	4
Mine Equipment Maintenance Inspector	2
Technical Services	**7**
Jr Topography Leader	1
Topographer	1
Geology and Mine Planning Coordinator	1
Geologist	1
Geology Technician	1
Topography Assistant	2
Total	**27**

16.8.2 Hourly Labor

Mine total hourly personnel requirements are shown in Table 16.8.2_1. Year 26 has the maximum personnel requirements at 192 people. The majority of persons in mine operations are equipment operators. The number of operators for major equipment was calculated based on equipment operating requirements. Also shown is the number of maintenance personnel requiems. An additional 10% VS&A allowance is included based on 30 vacation days plus 6 sick days out of 365 scheduled days per person per year.

Table 16.8.2_1 – Hourly Labour																	
	PS	**Y01**	**Y02**	**Y03**	**Y04**	**Y05**	**Y06**	**Y07**	**Y08**	**Y09**	**Y10**	**Y11**	**Y12**	**Y13**	**Y14**	**Y15**	**Y16**
	#	#	#	#	#	#	#	#	#	#	#	#	#	#	#	#	#
Equipment Operators	**64**	**72**	**74**	**76**	**76**	**85**	**85**	**88**	**90**	**90**	**90**	**83**	**96**	**96**	**95**	**104**	**104**
Front End Loader	5	5	5	5	5	5	5	5	5	5	5	5	10	10	10	10	10
Excavator	5	5	5	5	5	10	10	10	10	10	10	10	10	10	10	10	10
Haul Truck	22	25	27	29	29	33	33	31	33	33	33	31	37	37	36	45	45
Bulldozer	10	15	15	15	15	15	15	20	20	20	20	15	15	15	15	15	15
Motor Grader	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Water truck	3	3	3	3	3	3	3	3	3	3	3	3	5	5	5	5	5
Production Drill	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Rockbreaker	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel Truck	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Maintenance Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Lowboy Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Exploration Drill	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Blasting	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**
Blaster	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Blaster Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Other Operators	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**
Operations Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Dispatch Operator	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Labor	**37**	**42**	**43**	**44**	**44**	**49**	**50**	**51**	**53**	**53**	**52**	**49**	**55**	**55**	**55**	**60**	**59**
Vehicle Lubricator I	7	8	9	9	9	10	10	10	11	11	10	10	11	11	11	12	12
Vehicle Lubricator II	6	6	6	7	7	7	8	8	8	8	8	7	8	8	8	9	9
Vehicle Lubricator III	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	5	5
Vehicle Maintenance Mechanic I	7	8	9	9	9	10	10	10	11	11	10	10	11	11	11	12	12
Vehicle Maintenance Mechanic II	6	6	6	7	7	7	8	8	8	8	8	7	8	8	8	9	9
Vehicle Maintenance Mechanic III	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	5	5
Vehicle Maintenance Welder	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	5	5
Vehicle Electrician II	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	4	4

	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27	Y28	Y29	Y30	Y31	Y32
	#	#	#	#	#	#	#	#	#	#	#	#	#	#	#	#
Equipment Operators	**104**	**104**	**104**	**104**	**104**	**104**	**104**	**116**	**116**	**117**	**115**	**117**	**113**	**113**	**87**	**75**
Front End Loader	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Excavator	10	10	10	10	10	10	10	10	10	10	10	10	10	10	5	5
Haul Truck	45	45	45	45	45	45	45	52	52	58	56	58	54	54	33	21
Bulldozer	15	15	15	15	15	15	15	20	20	15	15	15	15	15	15	15
Motor Grader	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Water truck	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Production Drill	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Rockbreaker	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel Truck	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Maintenance Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Lowboy Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Exploration Drill	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Blasting	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**
Blaster	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Blaster Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Other Operators	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**
Operations Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Dispatch Operator	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Labor	**59**	**61**	**61**	**61**	**62**	**62**	**62**	**66**	**68**	**69**	**67**	**67**	**65**	**65**	**51**	**45**
Vehicle Lubricator I	12	12	12	12	12	12	12	13	14	14	13	13	13	13	10	9
Vehicle Lubricator II	9	9	9	9	9	9	9	10	10	10	10	10	10	10	8	7
Vehicle Lubricator III	5	5	5	5	5	5	5	5	5	6	5	5	5	5	4	4
Vehicle Maintenance Mechanic I	12	12	12	12	12	12	12	13	14	14	13	13	13	13	10	9
Vehicle Maintenance Mechanic II	9	9	9	9	9	9	9	10	10	10	10	10	10	10	8	7
Vehicle Maintenance Mechanic III	5	5	5	5	5	5	5	5	5	6	5	5	5	5	4	4
Vehicle Maintenance Welder	5	5	5	5	5	5	5	5	5	6	5	5	5	5	4	4
Vehicle Electrician II	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	3

17 RECOVERY METHODS

17.1 Beneficiation Plant

Process Summary

The mineral processing plant is based on the metallurgical test work and considers the following design parameters:

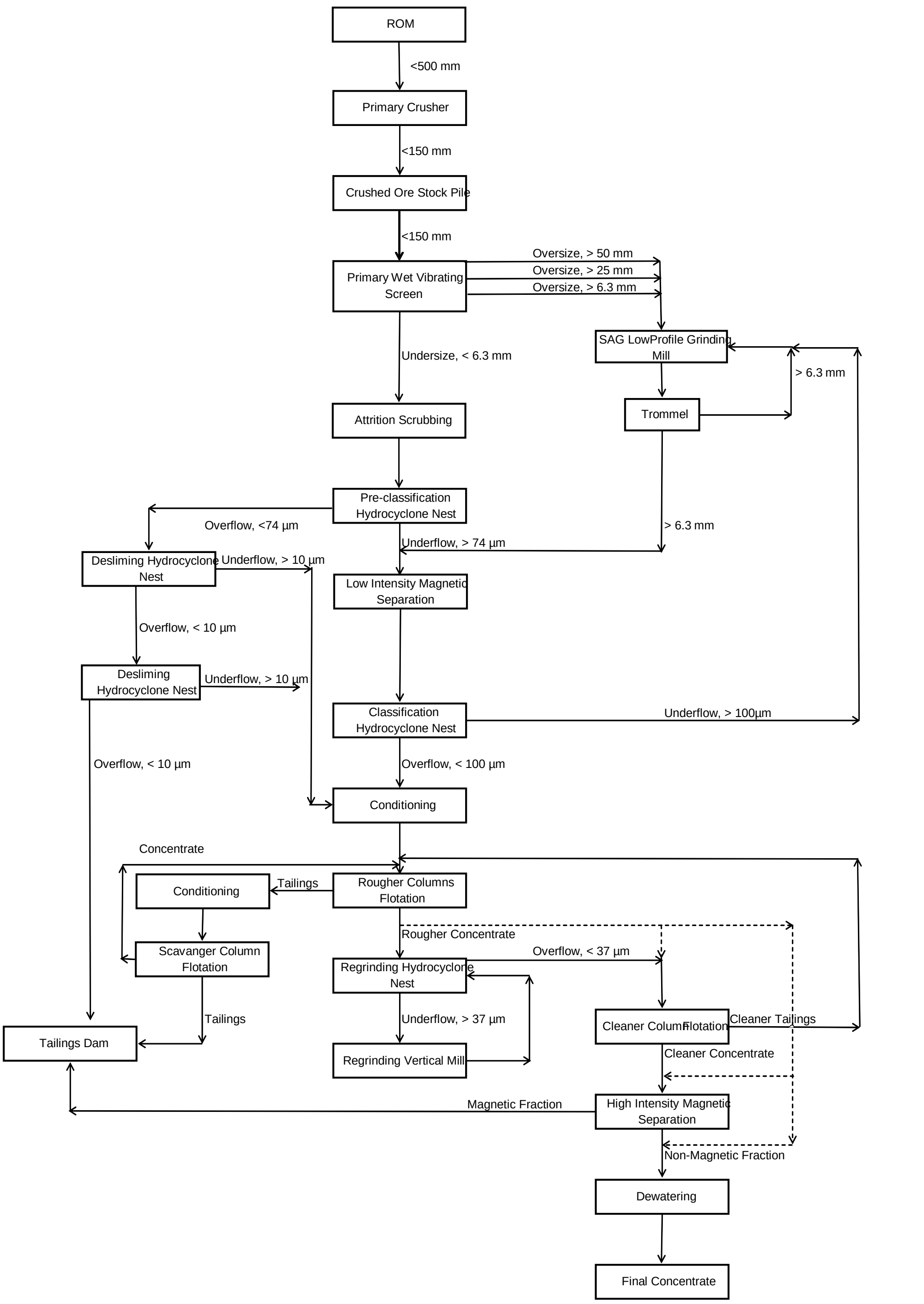
- Total ROM feed to the plant: 1.5 million tpy
- Phosphate Ore Feed grade: @ 12.5% P_2O_5
- Production of concentrate: 300,000 tpy @ 34% P_2O_5 (dry basis)
- Recovery from flotation: 55%
- Total plant availability: 89%

The process consists of the following steps, shown in Figure 17.1._1:

- Communition and Classification
  - Primary crushing of the ROM material
  - Primary screening into four size fractions
  - Primary scrubbing and desliming
  - Grinding of the coarse products
  - Wet low intensity magnetic separation of ground material
  - Classification at D_{80} 57 µm
  - Desliming at 10 µm
- Flotation section
  - Rougher flotation of ground and classified product
  - Scavenging the rougher column tailings
  - Classifying and regrinding the rougher column concentrate
  - Wet high intensity magnetic separation of cleaner concentrate
  - Recycling of cleaner tailings
  - Final Concentrate dewatering
  - Final Concentrate delivery

The Scavenger Tailings, slimes, and both magnetic separation waste streams combine and are pumped to the Tailings Dam.

Figure 17.1._1
Beneficiation Process Block Diagram



Grinding and Classification (Figure 17.1._2)

Grinding

Due to the friable characteristics of the ROM material, as determined by characterization studies and metallurgical test work, the maximum lump size is limited to approximately 600 mm.

The ROM phosphate ore discharges from a truck directly into a Feed Bin with a <600 mm grizzly. The overs are stockpiled for reclamation at a later time. The <600 mm material feeds a MMD Primary Crusher by conveyor. The resulting <150 mm material transports by two Belt Conveyors and Shuttle Reversible Belt Conveyor to two homogenizing stock piles of approximately 8000 m^3 each. The stock piles provide about 2 days of inventory.

Classification

A stock pile Reclaimer is used to feed the ore from the homogenizing stockpiles through a Hopper to a Belt Conveyor equipped with a Weigh Scale which records the throughput to the Beneficiation Plant. An auto cleaning electromagnet, a metal detector, and a sampler are also planned for this conveyor. A front-end loader (FEL) can be used to fill the Hopper and Conveyor if the Reclaimer is out of service. The material is conveyed to a triple-deck wet Vibrating Screen with 50.8 mm, 25.4 mm, and 6.3 mm inclined separation screens. The undersize particles are fed to an Attrition Cell with two double-propeller agitators. The pulp overflows by gravity into a Pump Feed Tank and is pumped to a Pre-Classification Hydrocyclone Nest. The combined material from the screen (>6.3 mm) is fed by gravity to the SAG Low Aspect Grinding Mill.

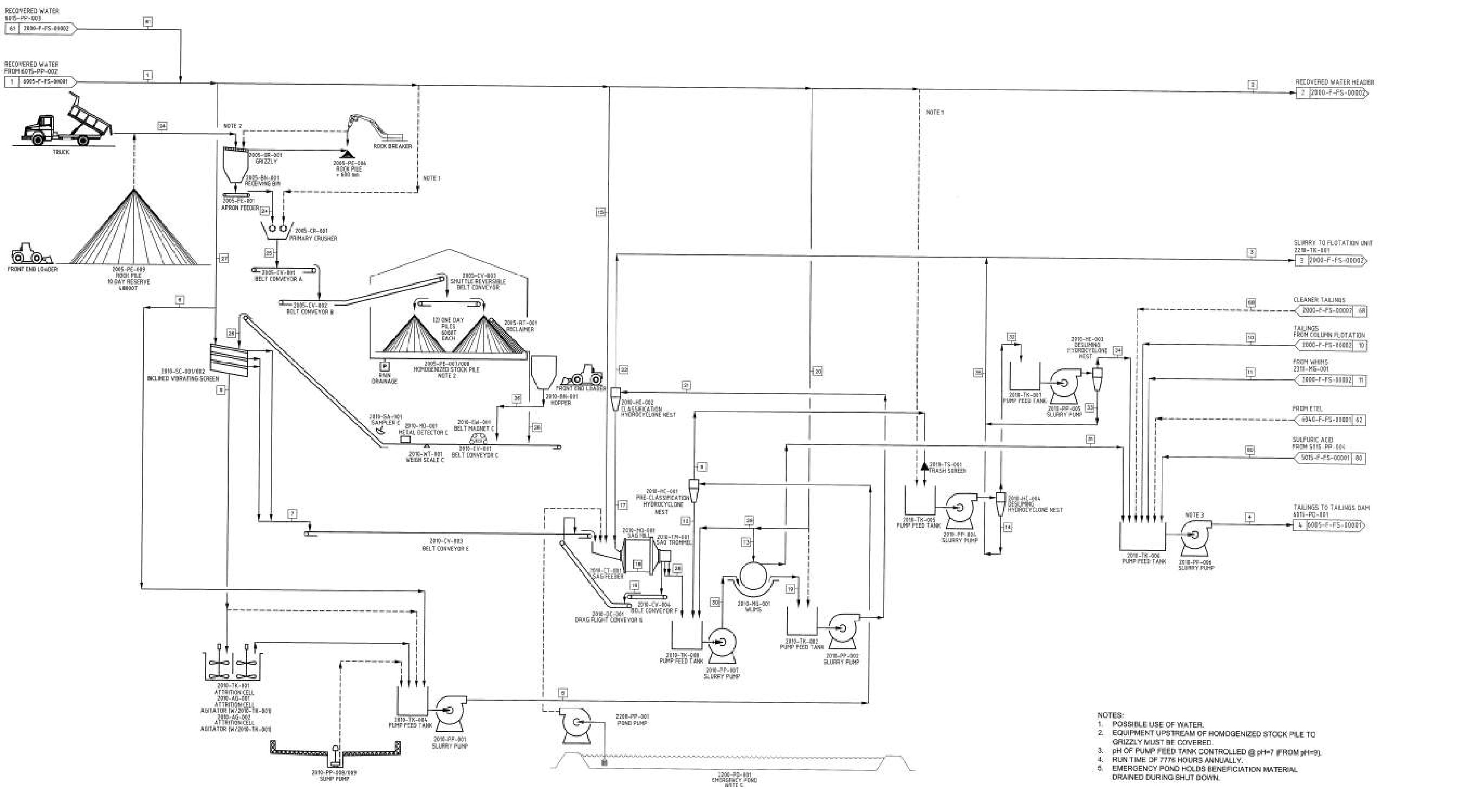
The Pre-Classification Hydrocyclone Nest consists of one battery of 4 - 381 mm hydrocyclones with a d_{50} of 32 µm for a mesh of separation of 75 µm and d_{80} of 64 µm. The Pre-Classification Hydrocyclone Nest overflow feeds a Pump Feed Tank by gravity through a Trash Screen and is pumped to the first Desliming Hydrocyclone Nest, which consists of twenty 24 - 101.6 mm diameter cyclones with a d_{50} of 6.5 µm for a mesh of separation of 12 µm and d_{80} of 10 µm. The overflow of the first Desliming Hydrocyclone Nest feeds a Pump Feed Tank by gravity and is pumped to a second Desliming Hydrocyclone Nest, which consists of 80 - 51 mm diameter cyclones with a d_{50} of 4 µm for a mesh of separation of 10 µm and d_{80} of 6 µm. The overflow of the second Desliming Hydrocyclone Nest feeds into the Tailings Pump Feed Tank by gravity and is pumped to the Tailings Dam. The underflows of the first and second Desliming Hydrocyclone Nests are fed by gravity to the Flotation Feed Conditioning Tanks.

The Pre-Classification Hydrocyclone Nest underflow recycles into a pump feed tank by gravity, and is pumped through a Wet Low-Intensity Magnetic Separation (WLIMS) Unit to remove magnetic material. The non-magnetic slurry feeds into another Pump Feed Tank by gravity, and is pumped into the Classification Hydrocyclone Nest. The magnetic fraction is sent to the Tailings Dam.

The Low Aspect SAG Mill is fed with the oversize material from the Vibrating Screens and the Classification Hydrocyclone Nest underflow. The SAG Mill discharge material passes through a screening Trommel, recycling the oversize material (>6.3 mm) by one Belt Conveyor and one Drag Flight Conveyor back into the SAG Mill Feeder. The <6.3 mm rock slurry discharges into the Pump Feed Tank, where it combines with the underflow material from the Pre-Classification Hydrocyclones. This material is pumped through a Wet Low Intensity Magnetic Separation Unit. The non-magnetic slurry feeds into a second Pump Feed Tank by gravity, and is pumped into the Classification Hydrocyclone Nest.

The Classification Hydrocyclone Nest consists of 4 - 508 mm with a d_{50} of 57 μm for a mesh of separation of 106 μm and D_{80} of 350 μm. The overflow of the Classification Hydrocyclone Nest feeds the Flotation Feed Conditioning Tanks by gravity. The underflow feeds back to the Low Aspect SAG Mill.

Figure 17.1._2
Flotation Feed Preparation - Process Flow Sheet



Flotation (Figure 17.1._3)

Flotation

The overflow from the Classification Hydrocyclone Nest and both underflows from the Desliming Hydrocyclone Nests feed two Flotation Feed Conditioning Tanks in series. In the first Conditioner, Sodium Hydroxide and Sodium Silicate are added. In the second Conditioner the Soybean Oil is added. The overflow from the second Conditioner feeds by gravity to a Pump Feed Tank and is pumped to a diverter to distribute the feed between two Rougher Column Flotation Cells. The Rougher Columns tailings are pumped to a third Conditioner. The third conditioner additional Soybean Oil is added to recover P_2O_5. The rougher tailings are pumped to a Scavenger Column Flotation Cell in an attempt to recover additional P_2O_5. The overflow of the Scavenger discharges to a Pump Feed Tank and is pumped to the Pump Feed Tank which returns to the Rougher Column Flotation Cell. The tailings of the Scavenger is sampled, pumped to the Tailings Pump Feed Tank, and is sent to the Tailings Dam.

The primary path of the rougher concentrate is sampled and feeds by gravity through a Trash Screen into to a Pump Feed Tank. The slurry is pumped to the Regrinding Hydrocyclone Nest. The Regrinding Hydrocyclone Nest consists of 20 - 152 mm Hydrocyclones with a d_{50} of 17 µm, a mesh of separation of 37 µm, and d_{80} of 44 µm. The underflow of the Regrinding Hydrocyclone Nest feeds by gravity to the Regrinding Vertical Mill. This mill discharges into the Pump Feed Tank of the pump feeding the Regrinding Hydrocyclone Nest in closed circuit. The overflow of the Regrinding Hydrocyclone Nest is sampled, feeds by gravity to a Pump Feed Tank, and is pumped to the Cleaner Column Flotation Cell. Depending on the particle size of the rougher concentrate, the regrinding circuit may be bypassed and the slurry will feed directly into the Cleaner Column Flotation Cell.

The Cleaner Column Flotation Cell underflow discharges to a Pump Feed Tank and recycles to the Rougher Column Flotation Cells. If the rock is poor quality, and recovery of P_2O_5 is not optional, the underflow may be pumped directly to the Tailings Pump Feed Tank, and sent to the Tailings Dam. The cleaner concentrate feeds through a sampler to a Pump Feed Tank to be pumped to the Wet High-Intensity Magnetic Separation (WHIMS) unit. If the Cleaner Concentrate contains low magnetic material, the WHIMS unit may be bypassed. When high-grade phosphate ore is fed to the Beneficiation plant, and particle size is of product size, the rougher concentrate may be fed directly to magnetic separation.

If high-grade phosphate ore is fed to the Beneficiation plant, with correct particle size, and very low concentration of magnetic material, the ore may bypass the entire system and may be fed directly to the Thickener.

Magnetic Separation

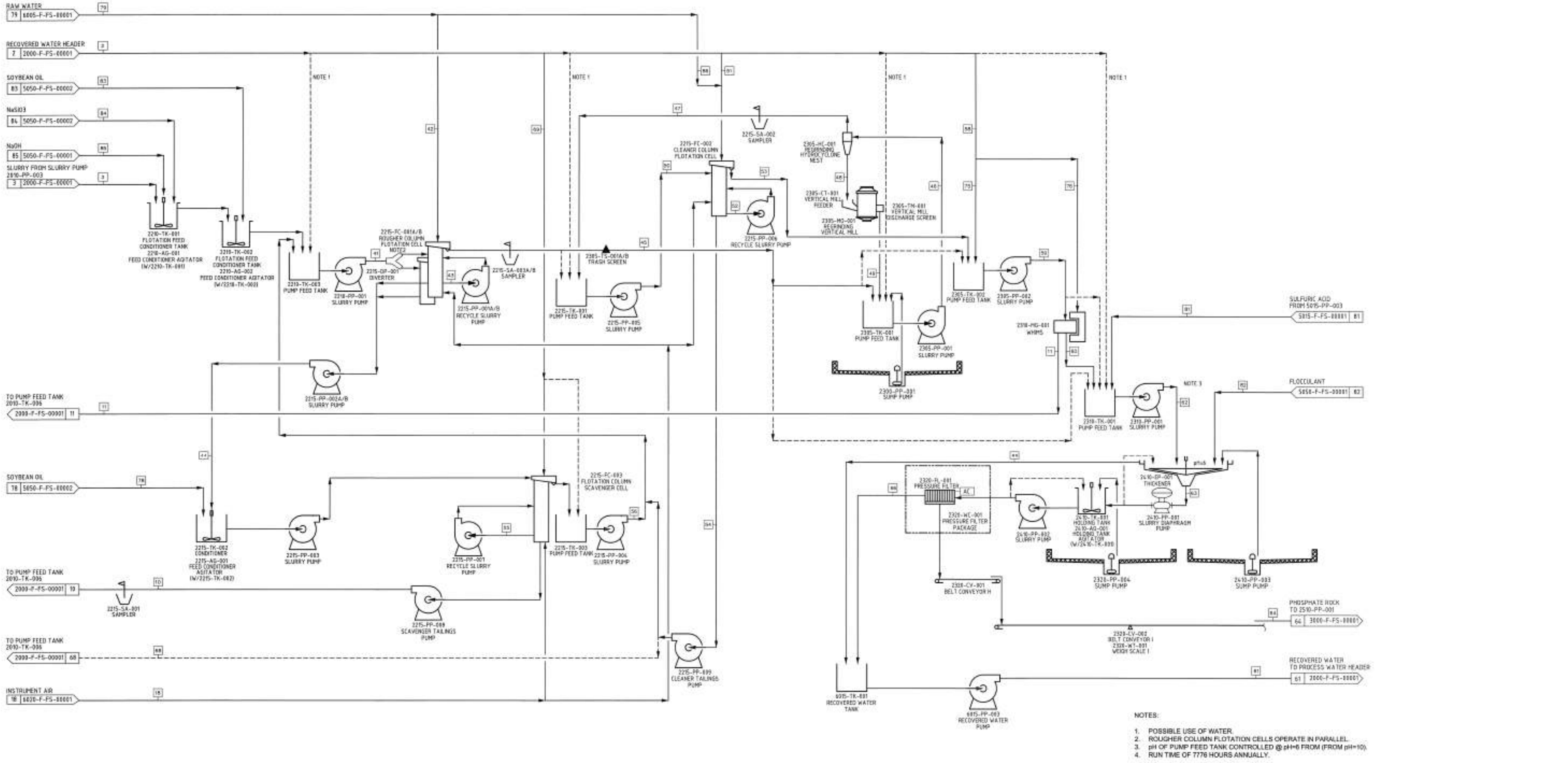
The Cleaner, or Rougher, concentrate flows by gravity to a Pump Feed Tank and is pumped to the Wet High-Intensity Magnetic Separator (WHIMS). The magnetic fraction of the Cleaner, or Rougher, concentrate combines with the slimes and Scavenger Column Flotation Cell tailings, and is pumped to the Tailings Dam. The non-magnetic material flows to a Pump Feed Tank via gravity and is pumped to the Thickener.

Dewatering of the Final Concentrate and Storage

The dewatering system consists of a high capacity Thickener, a Holding Tank with agitator, and a pressure Filter. The overflow from the Thickener flows by gravity to the Recovered Water Tank and is combined with filtrate from the Pressure Filter. This stream is pumped back to recovered water header in Beneficiation. The Thickener underflow is pumped, by a diaphragm pump, to a Holding Tank, and pumped to a horizontal Pressure Filter. The final concentrate is expected to be discharged at 15% moisture.

The cake from the Pressure Filter is delivered to a rock storage pile building in Acidulation by two Belt Conveyors and a Rotating Belt Conveyor. The storage pile has a capacity of 30000 tonnes, equivalent to one month of storage.

Figure 17.1._3
Flotation - Process Flow Sheet



Water Accumulation and Tailings disposal

The Tailings Dam area is positioned at a distance around 5 km from the Beneficiation Plant. The purpose of this reservoir is to receive the waste from the concentrate beneficiation plant, while accumulating water available during the rainy season, thus compensating for the eventual lack of water during the dry season. The dam project allows a convenient solids flow settling distribution, thus promoting adequate water recirculation.

A specialized Brazilian consulting company, Geoconsultoria was engaged to provide a consolidation to the conceptual level study for the solution of the tailings disposal for the Santana project presented by Pimenta de Ávila in PFS.

The Santana tailings dam was projected to receive the flow of materials from the flotation tailings, the magnetic separation and the slimes for the 30 years of operation. The project consisted of the sizing of the dam itself and accessory structures as the river deviation, the flow restitution system, the emergency drain and the surface draining system.

For the Feasibility Study it was considered the previously selected option from the Pre-Feasibility Study for the disposal system and for the location.

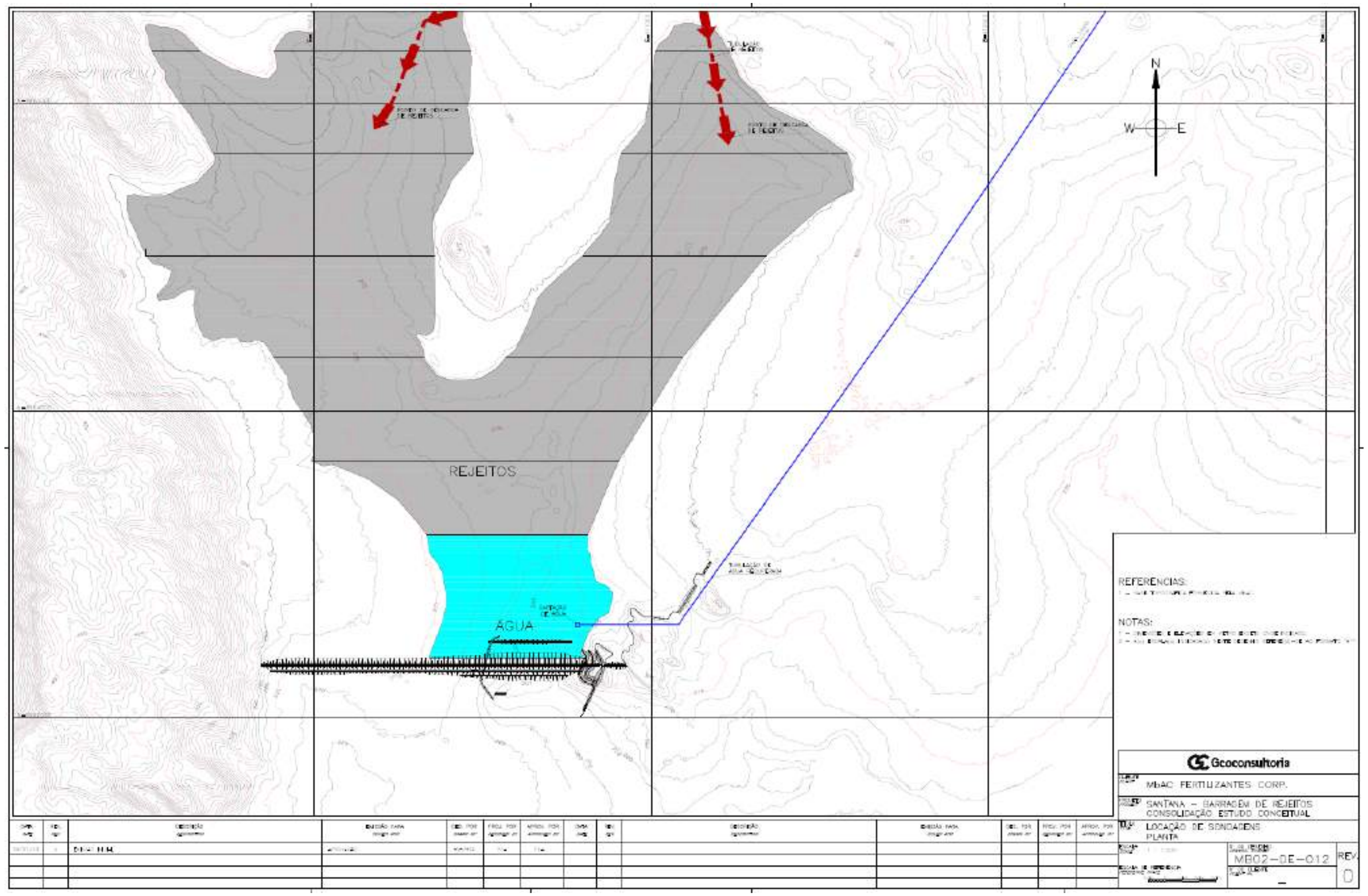
It was reassessed the alternative of pumping the material after a High Density thickener, and to dispose the material in a paste form (high solid content); we confirm that this alternative is not adequate, due to the higher costs involved, not only operating costs giving the high energy demand for pumping (higher operating pressure) and flocculants/reagents consumption, but also higher investment required for the overall system.

The selected option for the disposal system was the Hydraulic Deposition – the material is pumped to the tailings dam without being thickened. In this option the tailings are pumped as they are produced, and disposed at the dam with lower solid content on the pulp. The solids are separated naturally by sedimentation, and the cleaned water drained or recovered to be reused in the process. This system is most widely used in Brazil and in other regions, where water supply is not a major constraint.

Tailing dam location selected in Pre-Feasibility Study can be seen in Figure 17.1_4 and 17.1.5.

Figure 17.1._4
Tailing Dam Location

Figure 17.1._5
Tailing Dam Location



Dam Design

The massif of the Santana Project Dam was defined as a homogenous massif. The final elevation of the dam crest was defined taking in consideration the required tailing volume to be disposed, as well as the total volume of sediments to be accumulated along the life of the Project. The final reservoir volume calculated is 60 M m3.

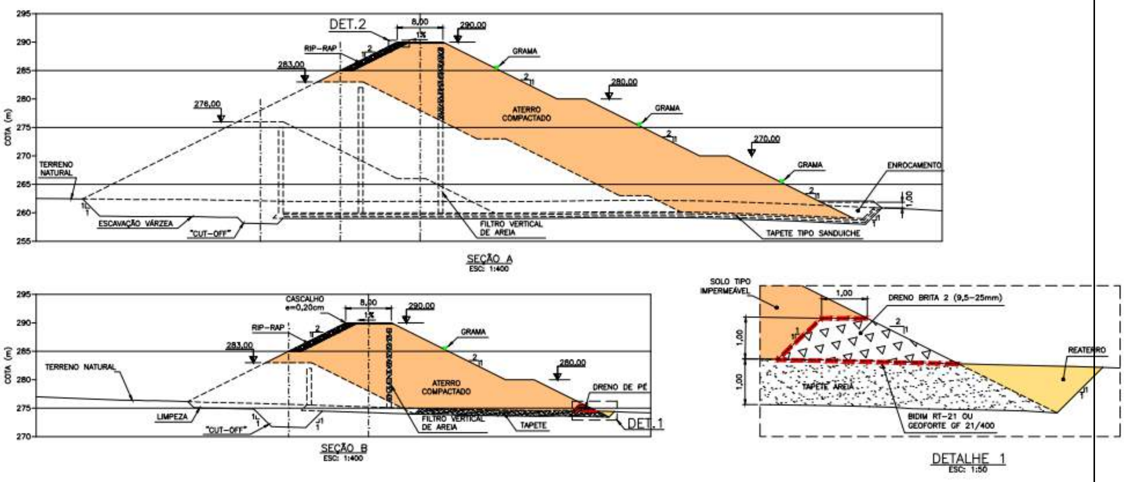
The hydrologic –hydraulic studies pointed the need of a free berm of at least 2.0 meters at the dam embankment for the design of an appropriate draining system. The dam characteristics were then defined as follow:

- Embankment elevation: 290m;
- Freeboard: 2.0m;
- Berms height: 10.0m;
- Slope inclination between berms: 1v:2h;
- Upstream slope inclination: 1v:2h;
- Embankment width at maximum elevation: 8.0m;
- Berm width: 5.0m;
- Embankment length: 1.5km

In view of the overall Project safety and following international standards for dam design the adopted structure considers a vertical sand filter linking the draining surface to its basis. Its dimensions were calculated based on the flow results obtained after the percolation analysis. The draining surface is linked to the bottom drain at the rock base. The rock base contributes to the safety of the dam basis and also defines the final off-set of the structure. As the dam is positioned at an open valley no underneath drain was projected.

The typical dam section is shown on Figure 17.1._6.

Figure 17.1._6
Typical Dam Section



Dam Monitoring and Instrumentation

The dam monitoring and control of the factors that may contribute to affect the safety and stability of the dam is a priority in the dam operation and routine. In the next stage of the project an Operating Manual, a Monitoring and Control Plan and an Emergency Action Plan will be prepared.

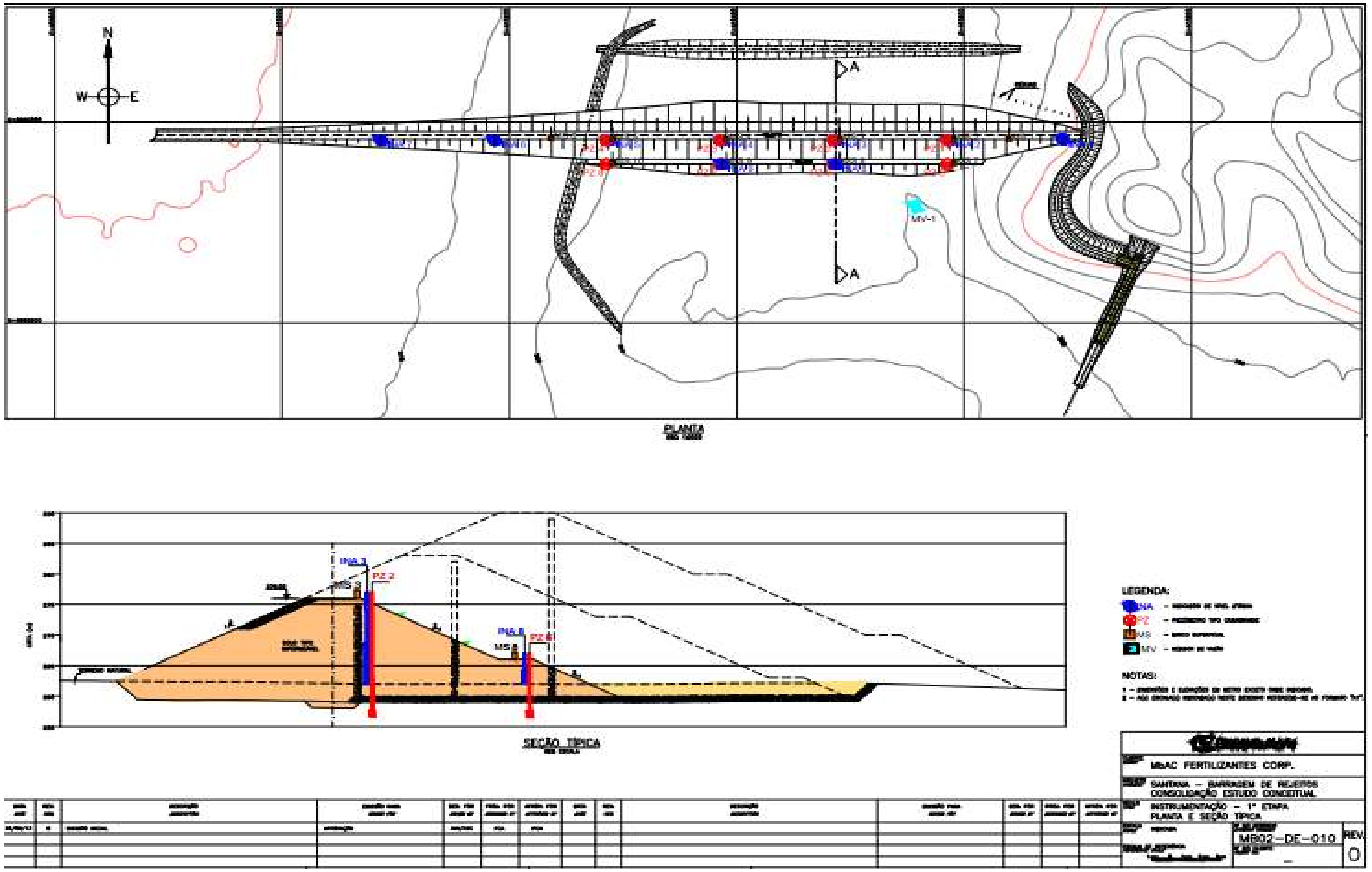
An Instrumentation Project was prepared to be implemented during the dam construction. The following controls were included for the Santana Dam:

- Piezometers
- Inclinometers
- Surface landmarks
- Water level meters at the abutments

In addition to these instruments the Instrumentation Project suggests the use of a flow meter at the bottom drain of the dam.

The dam instrumentation is shown on Figure 17.1._7.

Figure 17.1._7
Dam Instrumentation



17.2 Granular Single Superphosphate Plant

Process Summary

Phosphate concentrate from the Beneficiation site is converted to Granular Single Superphosphate (GSSP) at the Industrial site using the following design parameters:

- Total Concentrate feed to the plant is 300 ktpy (dry basis)
- Feed concentration: 34% P_2O_5
- Shipped product: 500 ktpy (dry basis)
- Co-Generation Power production: 7.5 MWe at 13.8 Kv

The chosen process consists of the following:

- Sulfuric Acid plant to produce 700 tpd of 98.5% H_2SO_4
- Acidulation of the Phosphate Concentrate to produce SSP
- Granulation of the SSP

Sulfuric Acid Plant

The sulfuric acid plant has a capacity to produce 700 tonnes per day with an on-stream factor of 89%. The technology adopted for the study is the MECS double absorption route.

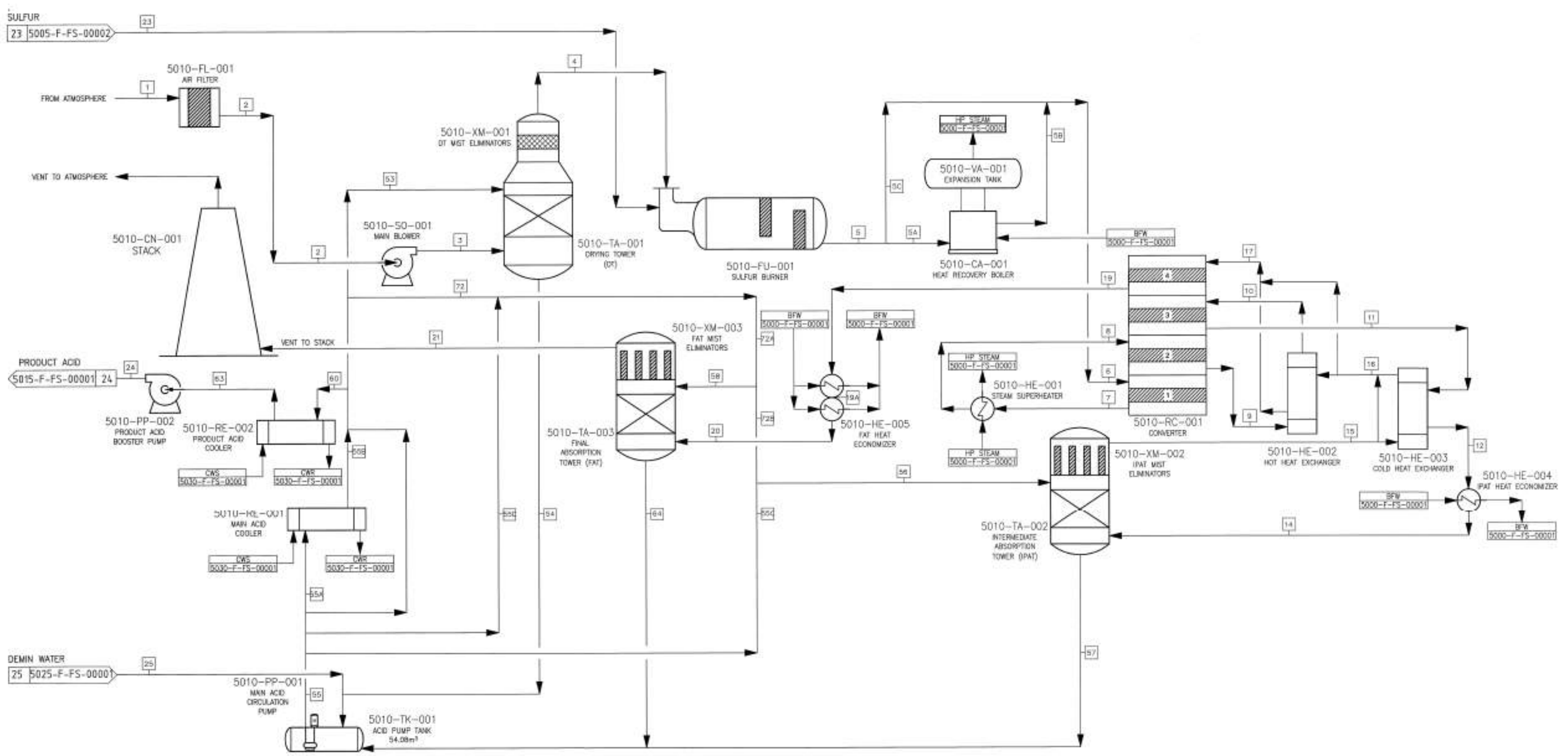
Process phases are as follows:

- Sulfur melting and filtration
- Sulfur burning in reaction with air
- SO_2 conversion into SO_3 in catalytic beds using Vanadium Pentoxide catalyst
- SO_3 gas absorption in packed towers
- Cooling and storage of acid product

The plant is continuously monitored to assure both the emissions of SO_x and sulfur mist meet the requirements of the applicable law.

In the sulfuric acid production process several chemical reactions occur: the sulfur combustion; SO_2 conversion into SO_3, and SO_3 absorption. See Figure 17.2_1.

Figure 17.2_1
Sulphuric Acid Plant Flow Sheet



The oxidation and absorption steps in the manufacturing of sulfuric acid from sulfur are all highly exothermic. The excess heat generated at each step of the process is recovered in the waste heat boiler, superheater, and economizers. The recovered heat is in the form of high pressure superheated steam which will be used to generate electric power in a turbo-generator. The process is designed to give a conversion of sulfur dioxide to sulfuric acid of 99.7% in the acid plant as well as a high conversion of process heat to steam. The only normal plant effluent streams are the blowdown streams from the boiler and the stack.

Atmospheric air is drawn through an air filter and air silencer and is compressed in the Main Blower which provides the motive force to move the gases through the downstream equipment. The gas leaving the Main Blower goes to the Drying Tower where water vapor in the air is removed by contacting a stream of 98-99% H_2SO_4 which flows down through the tower. From the Drying Tower the gas enters a horizontal spray-type Sulfur Furnace where the temperature of the sulfur dioxide gas from the Sulfur Furnace is higher than is required for inlet to the conversion system. Therefore, the gas is cooled in a Heat Recovery Boiler which recovers the surplus heat as high pressure saturated steam. The gas temperature from the boiler is controlled by a gas-side bypass. The boiler steam temperature is a function of the boiler steam pressure.

From the Heat Recovery Boiler, the gas flows to the first pass of the Converter system where it is partially converted to sulfur trioxide gas in the presence of vanadium catalyst. The conversion reaction produces heat. The gases must be cooled to improve the yield of the sulfur dioxide oxidation in the next catalyst pass. Gases leaving the first converter flow to a Steam Superheater where they are cooled by heating the export steam. The temperature of the export steam and pass 2 inlet gas temperature are controlled in the proper range by bypassing a portion of the steam flow around the superheater.

The cooled gas stream flows from the superheater to the second converter pass where additional conversion of the sulfur dioxide to sulfur trioxide takes place accompanied by the generation of additional heat. Hot gases leaving the second converter pass are cooled to improve the yield in the next catalyst pass by sending them through the Hot Heat Exchanger. Exit gas temperature is controlled by a bypass.

Cooled gases leaving the Hot Heat Exchanger flow to the third converter pass where additional conversion of sulfur dioxide to sulfur trioxide takes place. Hot gases leaving the third converter pass are cooled by sending them through the Cold Heat Exchanger and the IPAT Heat Economizer.

From the Economizer, the cooled gas enters the Intermediate Absorption Tower (IPAT) where SO_3 is removed from the gas stream by contacting the gas with circulating 98-99% sulfuric acid. The gas stream leaves the IPAT and is heated in the second side of the Cold Heat Exchanger by hot gases leaving the third converter pass.

From the Cold Heat Exchanger, the gas stream flows to second side of the Hot Heat Exchanger where it is further heated by hot gases leaving the second converter pass.

From the Hot Heat Exchanger, the gas stream flows to the fourth converter pass where final conversion of SO_2 to SO_3 is accomplished. The temperature to the fourth converter pass is controlled by bypassing a portion of gas around the cold and hot heat exchangers.

The gas stream leaving the fourth pass enters the Final Absorption Tower (FAT) Heat Economizer where it is cooled by boiler feed water. Water side bypasses are used to control the exit temperature from the economizers. The exit gas temperatures can be regulated to prevent economizer drip acid formation normally associated with variable hydrocarbon content in the sulfur feed.

Gas leaving the FAT Heat Economizer enters the FAT prior to exhausting to the atmosphere through a stack.

In the FAT, SO_3 in the gas stream reacts with water in the 98-99% circulating acid. The temperature of the strong acid circulated over the final absorbing tower increases due to the heat of formation and the sensible heat of the gas stream entering the tower. The 98% sulfuric acid from the bottom of the FAT and the IPAT flows into the Acid Pump Tank.

The 98% acid is cooled by cooling water in the Main Acid Cooler. Most of the cooled acid is further cooled in a Product Acid Cooler and pumped to a storage tank. The remainder of this stream is used as recirculation liquid in the Drying Tower, the IPAT, and the FAT.

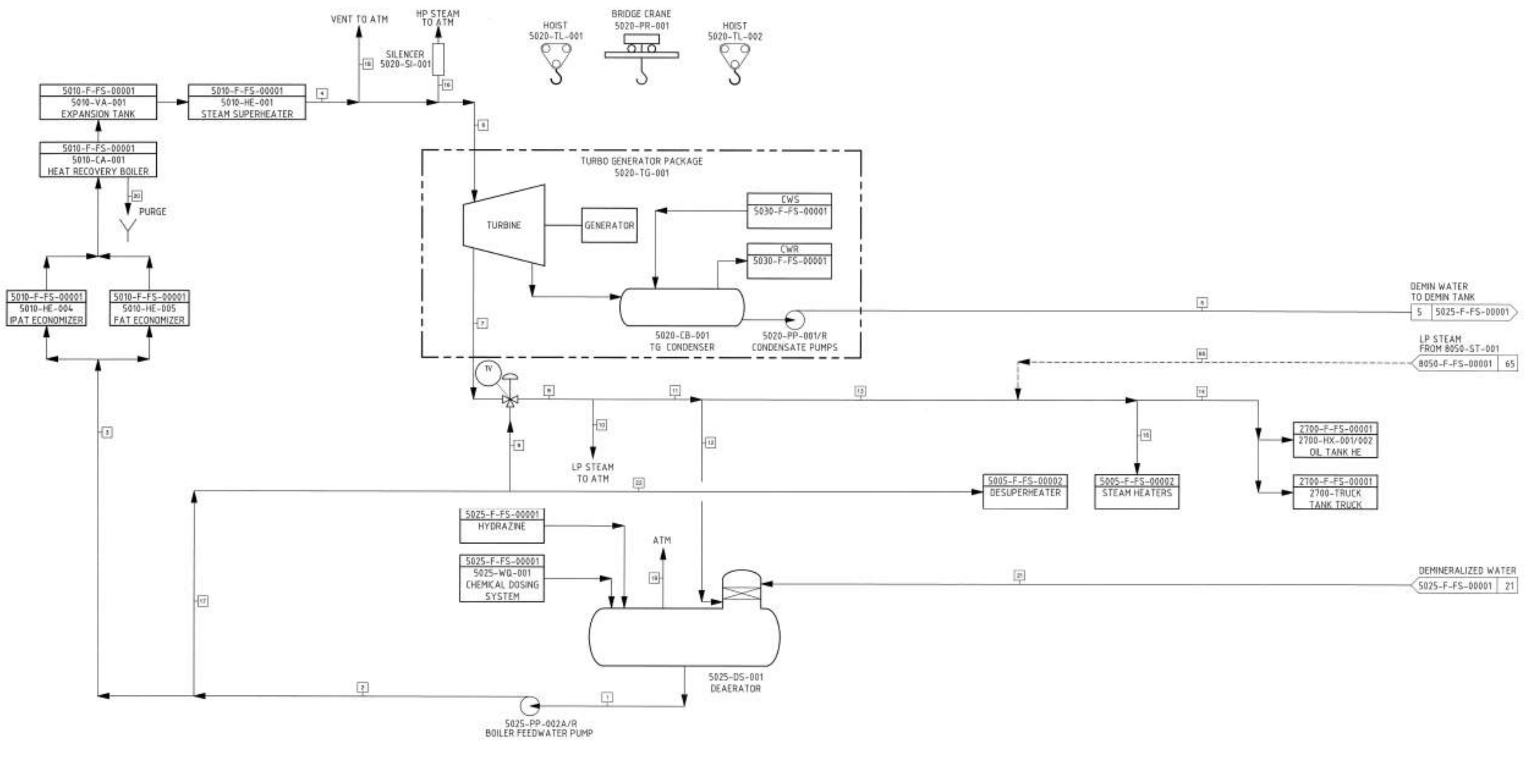
The Sulfuric Acid plant generates superheated steam which is fed to the Turbo Generator.

Co-generation

The Sulfuric Acid plant exports superheated steam which feeds the Turbo Generator, which is designed to produce 7.5 MW of power. (Figure 17.2_2)

The Turbo Generator is a condensing turbine. A small portion of low pressure (LP) steam is extracted from the turbine casing. The LP steam is de-superheated with boiler feed water and used throughout the Industrial Facility. The vacuum steam is condensed by cooling water in the condenser, and the condensate is returned to the Deaerator.

Figure 17.2_2
Co-generation Flow Sheet



Acidulation Plant

The Acidulation plant is designed to produce 500 ktpy of SSP. The on-stream factor is 76% (Figure 17.2_3).

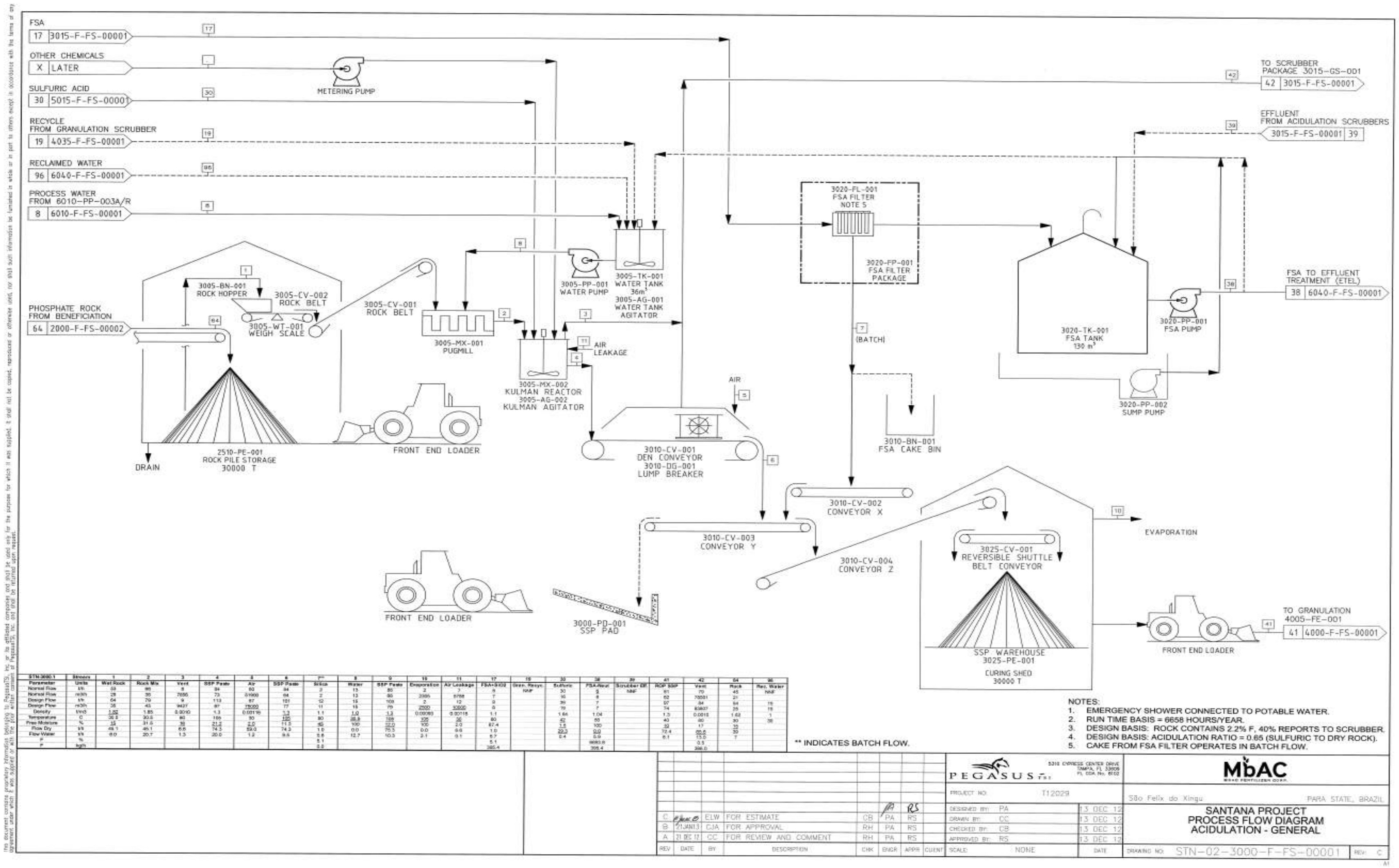
The phosphate rock from the Rock Pile Storage is reclaimed by a front-end loader (FEL) to a Rock Hopper feeding two Rock Belts. A Weigh Scale on the conveyor measures the feed rate.

The Rock Belt feeds the Pugmill, Process Water is introduced. Process Water will be the primary type of water fed to the Pugmill, but the options of using granulation scrubber recycle, acidulation recycle, and diluted FSA solution is available. The slurry product from the Pugmill and sulfuric acid feed the Kulman Reactor, an intensive vertical mixer. The Kulman Reactor quickly mixes the slurry and acid, which reacts to form SSP. The reacting SSP feeds onto the Den Conveyor. The reaction slurry solidifies on the Den Conveyor, and passes through a Lump Breaker before being transferred to the covered SSP Pile by Belt Conveyors.

During the reaction, fluorine evolves from the Kulman Reactor and Den Conveyor. The gases with are transferred to the scrubber system, where the fluorosilicic acid (FSA) and hydrated silica are removed by contact with water. The gas is treated according to environmental standards, and releases to the atmosphere through the Vent Stack. The blowdown from the scrubber system is pumped to the FSA Pressure Filter. The Scrubbers include Scrubber Tanks and circulation Scrubber Pumps. The two types of makeup water available are raw water from the river and reclaimed water from ETEL.

The FSA Pressure Filter receives the suspended FSA solution in batches, and separates the silica (solid) from the FSA (liquid). The silica combines with the drying SSP material from the Den Conveyor. The filtered FSA solution is collected in the FSA Tank. The solution may be pumped to the Pugmill to be reused or pumped to ETEL for disposal.

Figure 17.2_3
Acidulation Plant Flow Sheet



Granulation Plant

The granulation plant is designed to produce up to 526,000 tpy of GSSP. (Figure 17.2_4)

The single super phosphate (SSP) from the Curing Shed is transferred by FEL to a Belt Conveyor with the recycle fines from Shipping. The Belt Conveyor discharges to a second Belt Conveyor and enters the Drum Granulator. Scrubber Recycle, Water, LP Steam, and air enter the Drum Granulator to create ideal conditions for granulation. The gases vent from the Granulator to the Granulator Scrubber to be cleaned before being released to the atmosphere via Vent Stack. The Granulated SSP (GSSP) discharges directly to the Drum Dryer.

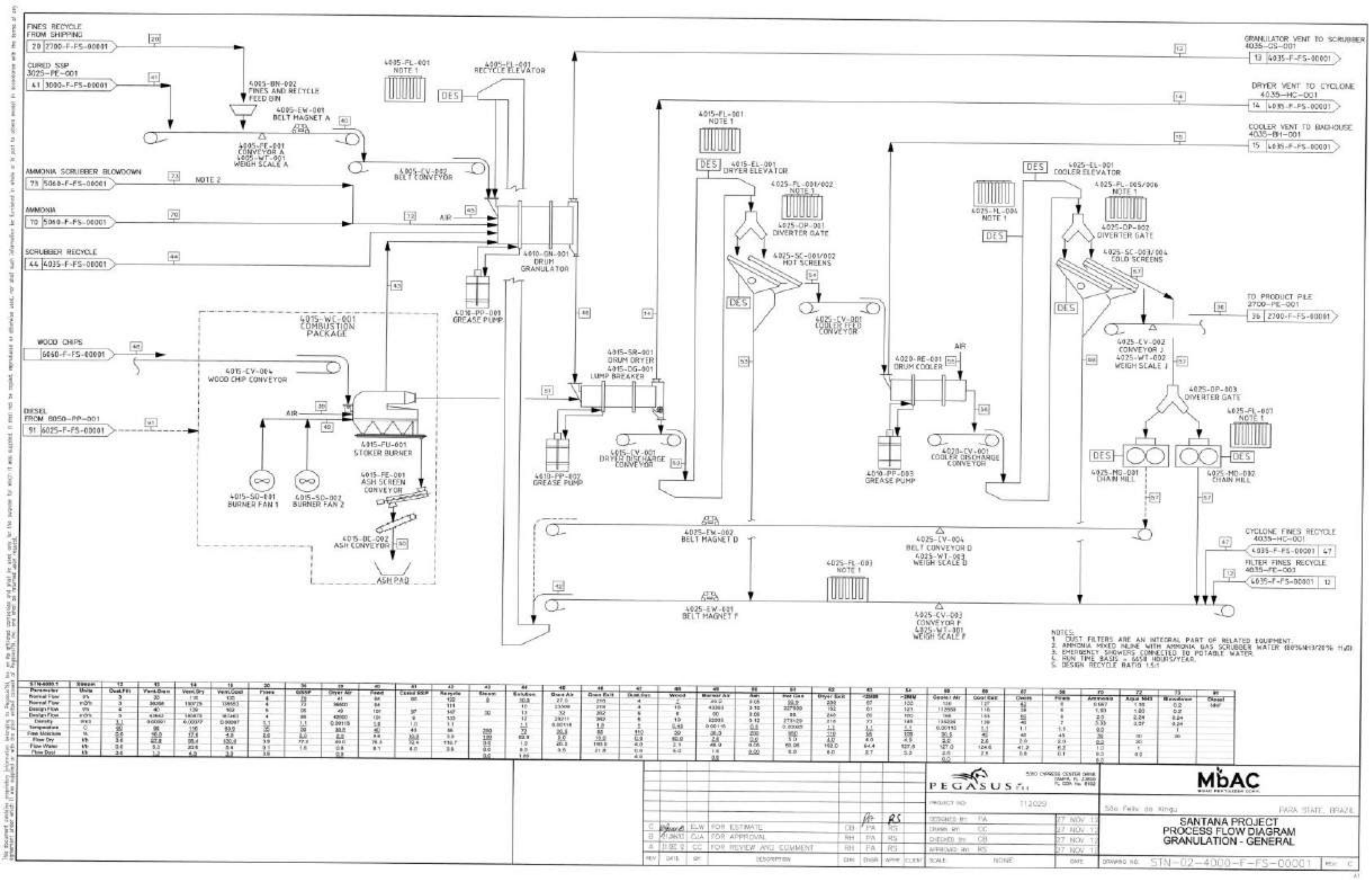
Hot gases from the Stoker Burner flow into the Dryer to reduce water concentration from 10% to 4%. The combustion chamber burns wood chips, fed by a series of three conveyors from the Wood Chipping area. Additionally, a small amount of LP steam is produced and feeds the Drum Granulator. The gas from the Dryer is cleaned by the Dryer Cyclone and the Dryer Scrubber before being released to the atmosphere via Vent Stack. The dried GSSP exits the Drum Dryer after passing through a Lump Breaker and transfers, by Dryer Discharge Conveyor, to the Dryer Elevator. The Dryer Elevator lifts the dried GSSP to a diverter which feeds both Hot Screens.

The oversize material (overs) from the Hot Screens is cooled by air in a Drum Cooler. Gases from the Drum Cooler vent to a Cooler Baghouse to recover the particulate material and recycles to the Granulator by Belt Conveyor and Recycle Elevator. The undersized material from the Hot Screens discharges to a Belt Conveyor, and recycles to the Granulator by Belt Conveyor and Recycle Elevator. The overs from the Hot Screens exits then Drum Cooler as cooled GSSP and transfers, by Cooler Discharge Conveyor, to the Cooler Elevator. The Cooler Elevator lifts the cooled GSSP to a diverter which feeds both Cold Screens.

The Cold Screens have two decks. The overs from the upper decks feed into a distributor to feed one of two Chain Mills. The overs are ground and recycle to the Drum Granulator by Belt Conveyor and Recycle Elevator. The fines from the lower decks combine discharges to a Belt Conveyor and recycles to the Granulator by Belt Conveyor and Recycle Elevator. The overs from the lower decks are of correct particle size for shipping, and are stockpiled in the Product Pile by Belt Conveyor. A Weigh Scale measures product flow rate from the Cold Screens to the Product Pile.

The hot screen fines, cold screen fines, chain mill product, and recovered fines from gas treatment recycle by a single common conveyor and elevator.

Figure 17.2_4
Granulation Plant Flow Sheet



Shipping

Multiple front-end loaders (FEL) transfer granular single super phosphate (GSSP) from the Product Pile into two Feed Bins discharging to two Belt Conveyors. Each conveyor includes a Lump Breaker and Belt Magnet. Both conveyors discharge into Elevator A, which feeds into two double deck Polishing Screens.

The overflows from the first deck of the Polishing Screens discharges to a Belt Conveyor and feed the Roller Mill. The overflow material is reground, and recycles back into Elevator A. The underflow feeds a Fines Bin for reprocessing in Granulation. The overflow material from the second deck discharges to a Belt Conveyor which feeds a Dedust Drum, where a coating additive is introduced.

After coating GSSP, the Dedust Drum discharges into Elevator B. The GSSP diverts to one of two conveyors, which feed the East and West Truck Bins. The East Truck Bin feeds GSSP into a Weigh Scale, which loads bulk bags located in the trucks. The West Truck Bin feeds GSSP directly into the trucks, and weighs the GSSP by Truck Scale.

Local air filters collect dust along various points of the shipping process.

Coating Oil

Coating oil is transported to the plant by truck. Oil Pump 1 transfers the coating oil from truck to a heated Oil Tank. Steam is available to heat up the truck to assist in unloading the oil. An Oil Tank Heater is used to maintain the oil temperature and decreased viscosity. Oil Pump 2 transfers the oil from the heated Oil Tank to the Dedust Drum. The flow from Oil Pump 2 may be fixed, or controlled by recycle based on the flow rate of GSSP entering the Dedust Drum.

18 PROJECT INFRASTRUCTURE

18.1 Background

Santana Phosphate Project is located in the South-eastern region of Pará State, in the Northern Brazil. The DNPM concession area for mineral evaluation is of about 233,070 hectares located in a farmland deforested area, showing a savannah aspect.

The mineral resource estimate is based on 314 diamond holes (18,376.91m) and 277 RC holes (13,108m) drilled at a spacing of approximately 100m by 100m.

An independent mineral resource has been estimated comprising an indicated mineral resource of 60.36 Mt at 12.04% P_2O_5 and an inferred mineral resource of 26.59 Mt at 5.56% P_2O_5 (using a 3% P_2O_5 cut-off)

Bench scale metallurgical test work on rock samples collected from outcrops in the Project area, Indicated that a high grade (34% P_2O_5) phosphate concentrate with a good reactivity, can be generated utilizing a simple beneficiation process.

The mineral deposit is strategically located, considering the growing fertilizer market in the project region of influence (Mato Grosso and Pará State).

Those facts encouraged the Company to develop a Project to produce low concentration fertilizers, mainly oriented to soya bean crops.

18.2 Project Scope

18.2.1 Production Units

The Santana Phosphate Project takes into account the favorable geographic position of the mine allied to the good rock characteristics, resulting in a lower production and logistics costs.

The Project aims the implementation of the following production Units:

- Ore Beneficiation Plant - 300 kty
- Sulphuric Acid Plant - 227 kty
- Acidulation Plant (SSP) - 500 kty
- Granulation Plant (GSSP) - 500 kty
- Cogeneration Unit - 7.5 MWh

The Santana Phosphate Project consists in one project site with two operational facilities: a Mine Facility and Industrial Facility. The Mine Facility is anticipated to include an open pit phosphate mine to produce an average of 1,500,000 tonnes per year of phosphate ore with an average P2O5 grade of 12.86%. The expected mine life for the project is 32 years besides the pre stripping period.

The Industrial Facility consider a beneficiation plant to produce 300,000 tonnes per year of a 34% P2O5 concentrate, and a Single Super Phosphate (SSP) plant expected to produce 500,000 tonnes per year and a Sulfuric Acid Plant with a production capacity of 230,000 tonnes per year (700 tpd). The phosphate concentrate will be transported by trucks between the mine and industrial facilities.

18.3 Project Location Characteristics

- The Project Site is located in the Southern of Pará State in the Brazil northern region at an elevation of 309m and geodesical coordinates: S 9° 40'.23" W 51° 47'.23".
- The area is located at 229 km west from Santana do Araguaia (near Araguaia river) and at about 25 km from the Xingu river, on the east side.
- The topography is relatively flat with gentle slopes.
- The climate is tropical with a well defined dry season (May – October), followed by a wet summer.
- Typical rainfall is of 1800 mm during the summer period and temperature may vary from 20 to 36°C.
- The original forest vegetation was drastically reduced and substituted by grassland for cattle pasture; as a consequence, the hydrography of the region was seriously affected by the reduction of water streams.
- Water for the Project will come from Capivara River located 9 km south from the project site.

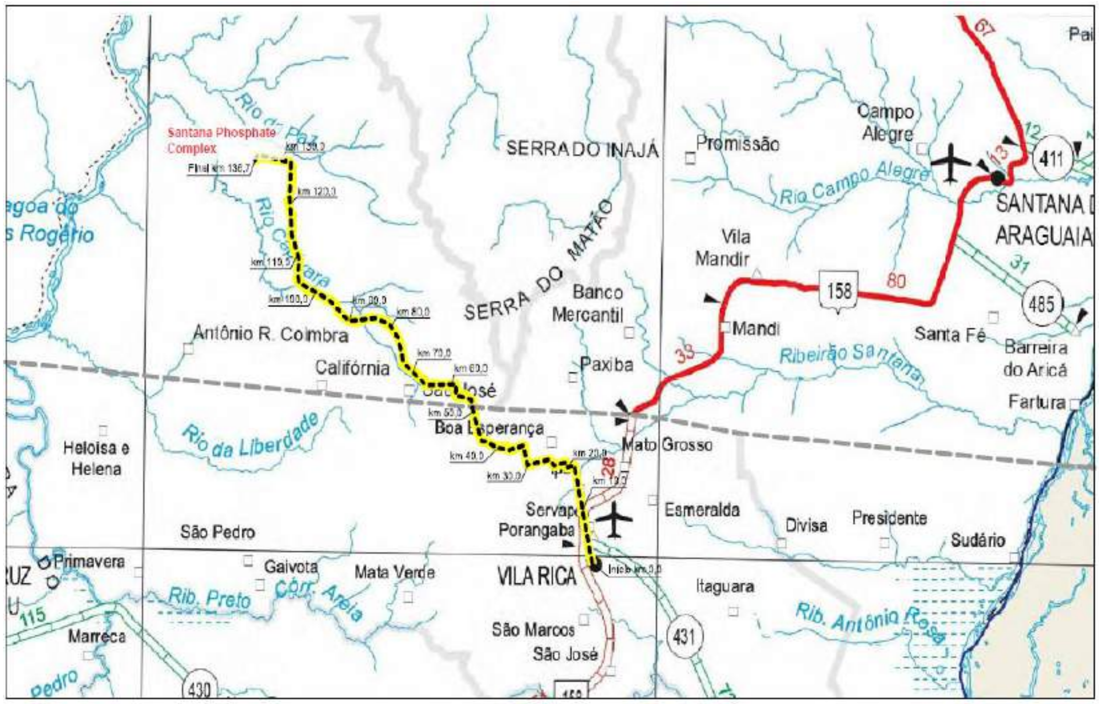
Figure 18.3_1
Overall Site Plan – Santana Project

18.4 Road Access

Pará State road structure is very poor currently; this creates some logistic difficulties for this Project; however, this situation improved in the last years, with the implementation of PAC (Government Infrastructure Program) which contemplates the paving of BR 158.

This important highway will cross the south east of Mato Grosso up to the north, reaching the city of Redenção in Pará state. See Figure 18.4_1.

Figure 18.4_1
Location Plan



Currently, access to the Project site is very difficult even utilizing the shorter route. Starting from Vila Rica (MT), you must travel 140 km on unpaved road in order to access to the Santana Phosphate Complex project site, mainly used by farmers to transport cattle.

This is a country route, not adequate for heavy trucks due to its limited width. In the dry season, the distance from the Mine up to Vila Rica, can be covered by a small truck in about 5 hours. However is difficult to estimate travelling time during the rainy season.

Considering the importance of this route for the Project, it was recommended that MBAC considers an investment of about US$ 16.6 million to update this road from the Project Site up to Vila Rica (140 km), to support the heavy traffic of trucks with raw materials, reagents, sulfuric acid and GSSP product.

In addition, a negotiation with farmers will be needed for their permission to improve and upgrade this road.

It must be emphasized however, that this situation can be radically improved in the future if the Government implements the road BR 235 scheduled for the five years plan. this new road BR 235 is planned to cross the Pará State from east (Caximbo) to West (Santa Maria das Barreiras) connecting the BR 153 (Xingu basin) to BR 158 (Araguaia basin).

18.5 Infrastructure Aspects

Livestock is the economic driver for this region (Pará State), having the highest cattle concentration in Brazil. The Project region is completely occupied by cattle farms with no social infrastructure.

The closest village (80 km) is Garimpinho which has only 200 inhabitants and absolutely no social infrastructure.

Vila Rica (140 km) and Vila Mandi (141 km) have a very poor infrastructure. Vila Rica and Vila Mandi are likely too far from the project site and cannot be considered as a support for human resources and services.

Human Resources

Based on the above considerations, there is very little availability of human resource with basic technical knowledge. As well no professional schools were identified in this first survey. As a recommendation, a plan to create manpower for the Project should be considered, utilizing professionals to set up a “training program” for the operation of each specific plant.

MBAC intends to implement training facilities in agreement with the local authorities to develop a Training Program for the project.

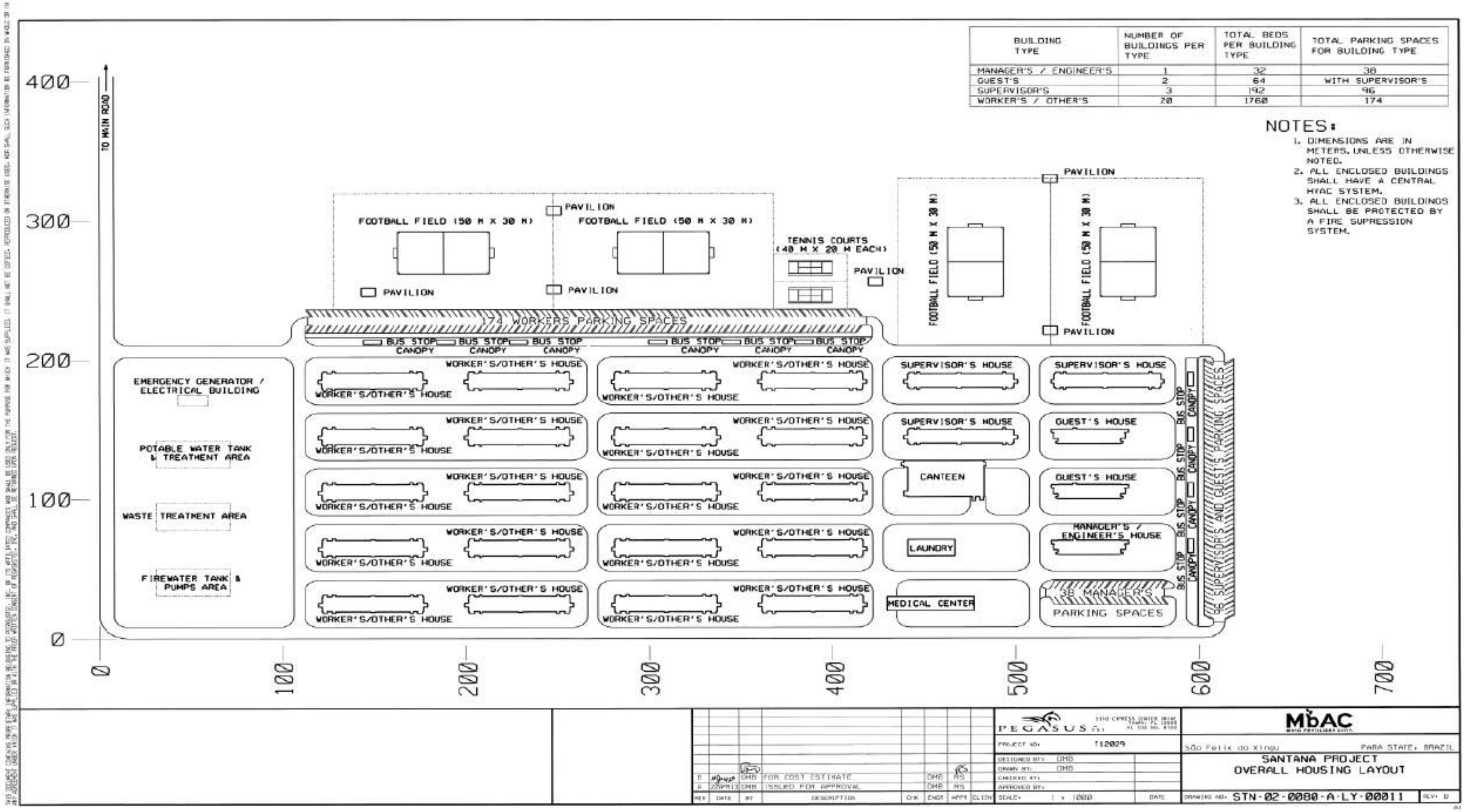
Housing

The Santana Project site will operate with approximatedly 420 workers including direct and indirect people. During construction phase it is forecasted a requirement of more than 2,000 people during pick of construction schedule.

This will require the building of accommodations, including a basic infrastructure to enable a normal living (see Figure 18.5_1).

This site will be built in the outer limits of the mine and industrial facilities (see Figure 18.3_1)

Figure 18.5_1
Housing Area Plot Plan



18.6 Industrial Utilities

18.6.1 Electrical Power Networks

Due to the enormous distances required to be covered, the integration of high tension power transmission in Brazil is not complete specifically in the State of Amazonas and Pará. In addition, the high tension power distribution in Pará is not well balanced with a concentration in the north and eastern portions of the state.

As a consequence, 40 thermoelectric power stations were installed to provide power to small cities (under 15,000 inhabitants).

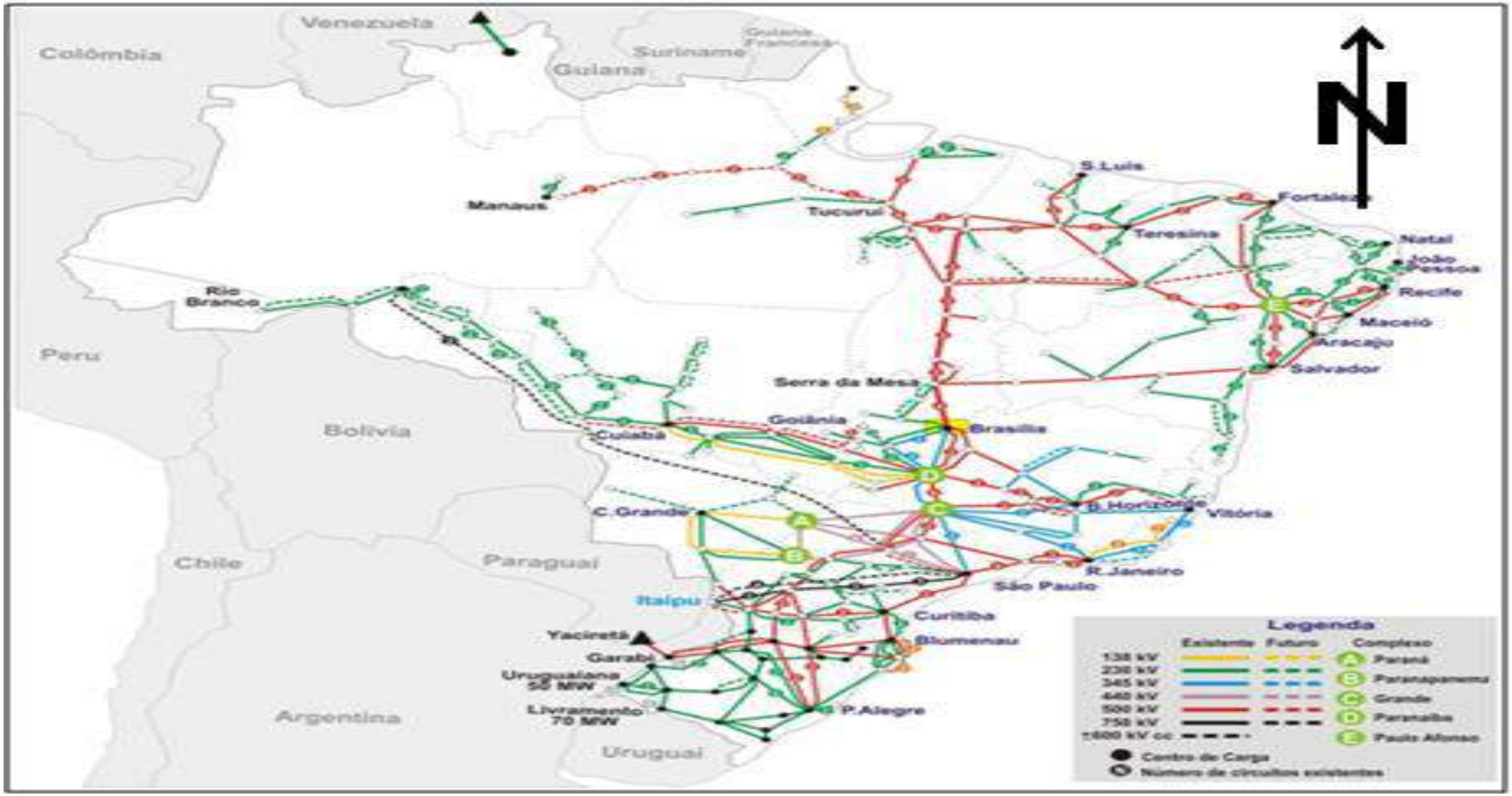
Within the Project Area, only a low tension distribution line for farmer's utilization exists.

The nearest available high tension point is located in Miracema, (Tocantins state) approximately 370 km from the Project site. CELPA (Centrais Elétricas do Pará) is the power distribution company, which receives energy from Tucurui, a hydroelectric power Company which generates 8,370 MWh and is located in the north of Pará State.

However, the distribution network (1,840 km) is mostly oriented to the northern part of Pará and Maranhão State

As seen in Figure 18.6.1_1 the center and southeast of Pará state, as well as the north of Mato Grosso, are not presently included in the high tension distribution grid.

Figure 18.6.1_1
High Tension Electricity



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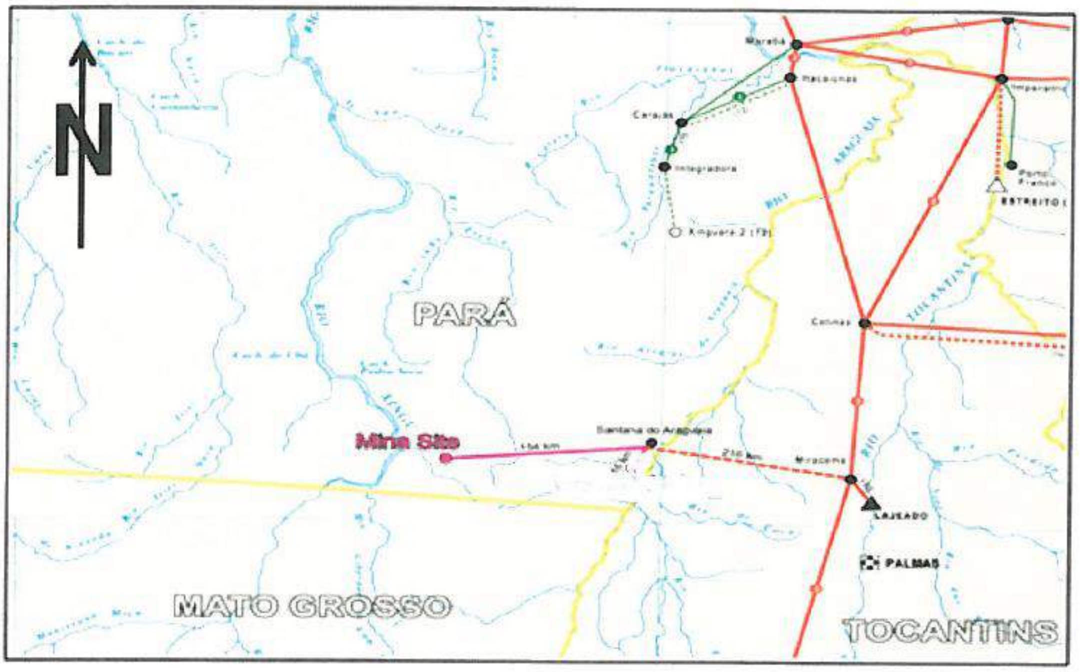
Due to the geographical location, the project operation will be directly affected by the situation above mentioned.

The Project site is about 370 km from the closest high tension facility located in the city of Miracema.

The electric power consumption in the mine and industrial facilities is estimated to be 10 -12 MWh. The Sulfuric Acid Plant has a cogeneration capacity of 7.5 MWh

CELPA sent a reference term to ANEEL with the request of a Power line (238 kV) to Santana de Araguaia. EPE (Energy Research Company from Brazilian govern) is the responsible to verify the demand and approve the Power line. It will allow MBAC the launching of a high tension power line (138 kV) from Santana do Araguaia up to the project site. CELPA belongs to GRUPO REDE which also controls CELTINS in Tocantins State, which is working with MBAC on the Itafos-Arraias Project.

Figure 18.6.1_2
Electrical Power



18.6.2 Water Supply

The total estimated water needed for this site will be 1,100 m3/h. A major part of this will be re-circulated from the water and tailings reservoir. However due to the high evaporation index, it will be necessary a make-up of 550 m3/h of fresh water.

The Project will take this new water from Capivara river, which is at approximately 9 km south from the industrial facility (see figure 18.3_1). This investment related to fresh water impoundment was included in CAPEX estimates.

In addition, a drilling survey to evaluate the potential for ground water will be undertaken in the near future.

18.6.3 Water Accumulation and Tailings disposal

The Tailings Dam area is positioned at a distance around 5 km from the Beneficiation Plant. The purpose of this reservoir is to receive the waste from the concentrate beneficiation plant, while accumulating water available during the rainy season, thus compensating for the eventual lack of water during the dry season. The dam project allows a convenient solids flow settling distribution, thus promoting adequate water recirculation. The Santana tailings dam was projected to receive the flow of materials from the flotation tailings, the magnetic separation and the slimes for the 30 years of operation.

18.7 Logistic Aspects

18.7.1 Raw Materials and Products

Final Products

The final product will be the Single Superphosphate with a minimum of 18% P_2O_5 soluble in Neuter Ammonium Citrate (NAC) plus water. The production from Santana will be mainly delivered to customers located in southern Pará and the northeastern region of Mato Grosso

But currently the access to the project site is very difficult even utilizing the shorter route. Starting from Vila Rica (MT), you must travel 140 km on unpaved road in order to access to the Santana Phosphate Complex project site, mainly used by farmers to transport cattle.This is a country route, not adequate for heavy trucks due to its limited width.

Therefore, MBAC considered an investment of US$ 16.6 million to improve the unpaved road to support the trucks traffic between the Santana Phosphate Complex and Vila Rica (140 km).This will enable MBAC to provide products up to 500 km radius, reaching the central region of Mato Grosso in a highly competitive basis.

Sulphur

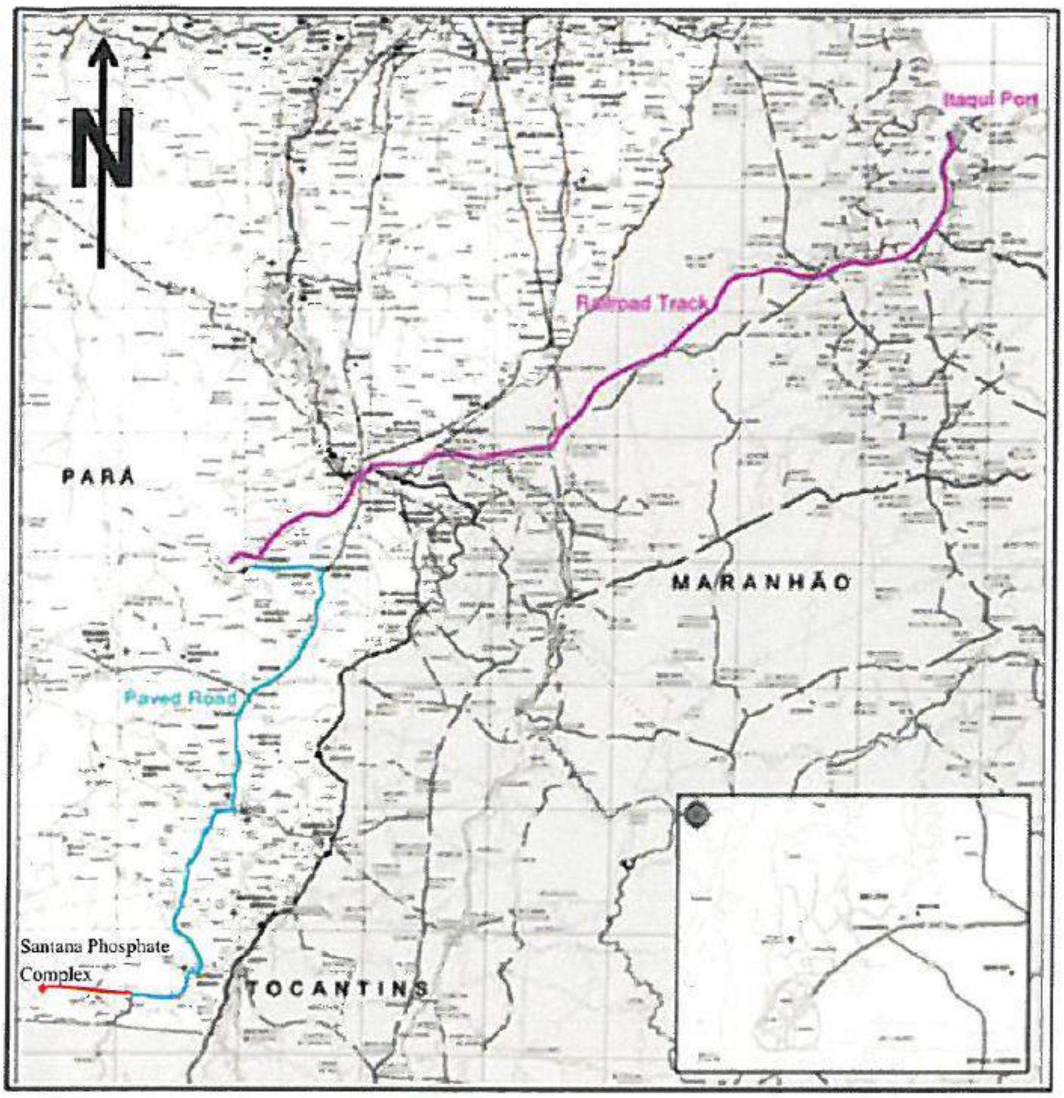
This material is imported and utilized to produce Sulphuric Acid. The annual need for this Project will be of 74 ktpy.

The present logistic plan is to receive this input via Itaqui port in Maranhão State, taking advantage of the synergy between the two plants (Itafos and Santana), thus reducing the sea freight cost.

Sulphur will travel 850 km from Itaqui to the Santana Industrial Site by Carajás railway up to Parauapebas (Pará State), followed by truck, through roads PA 275 and PA 150 and finally road BR 158 to the Santana Phosphate Complex facility, covering a total of 1,502 km at an estimated inland freight of US$ 115.00/t (see Figure 18.7.1_1)

Another alternative would be to use the port of Vila do Conde (Belem-PA) and the Araguaia Tocantins system waterway up to Santa Maria das Barreiras 980 km then by road BR 158 (84 km) up to the Chemical Plants. The total inland freight cost, utilizing this alternative should be certainly lower.

Figure 18.7.1_1
Freight Route



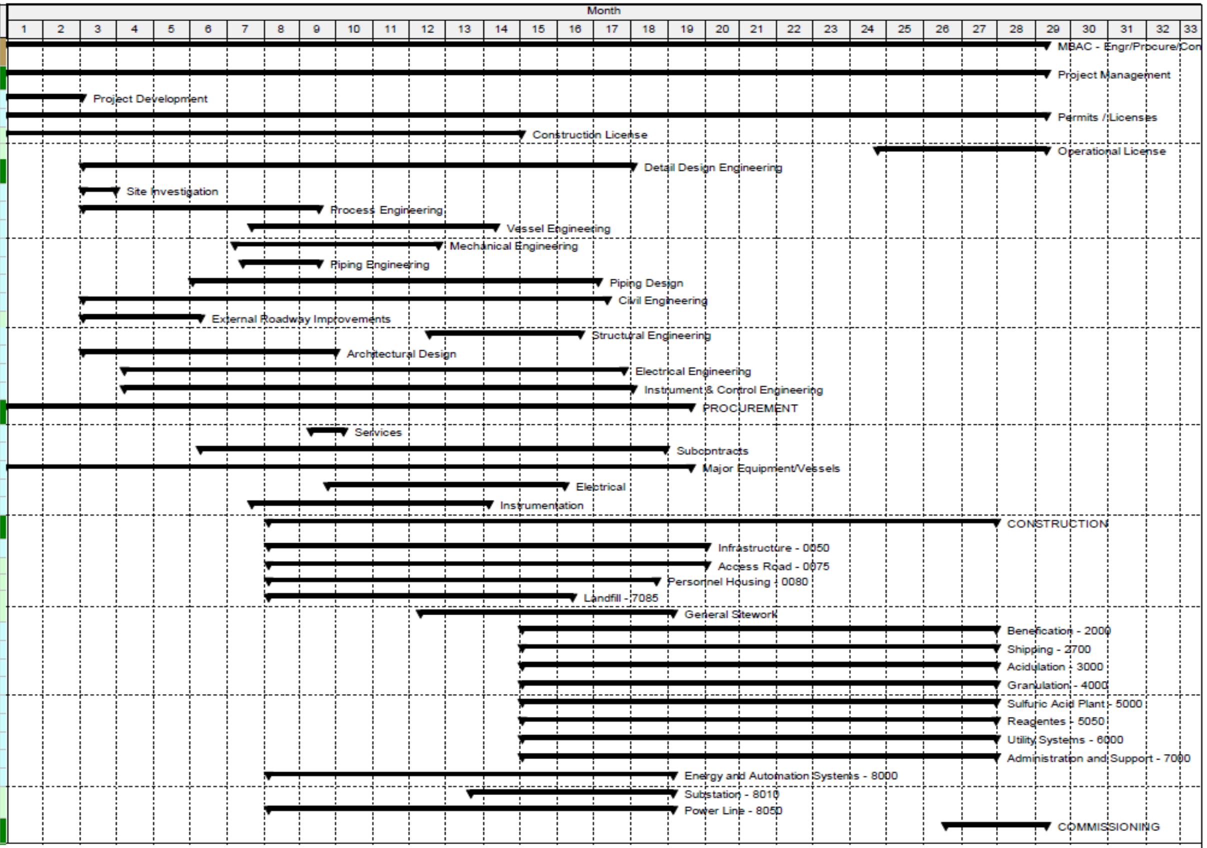
Fuel & Biomass

Wood chips or "brickets" are available in the Pará State in order to be utilized in the drying process of the Granulation Plant.

18.8 Project Implementation Schedule

Figure 18.8_1
Project Implementation Schedule

Activity ID	Predecess	Activity Name	Remaining Duration
		MBAC - Engr/Procure/Construct/Commission	617d
		Project Management	617d
		Project Development	46d
		Permits / Licenses	617d
		Construction License	306d
		Operational License	100d
		Detail Design Engineering	326d
		Site Investigation	20d
		Process Engineering	140d
		Vessel Engineering	145d
		Mechanical Engineering	121d
		Piping Engineering	45d
		Piping Design	241d
		Civil Engineering	311d
		External Roadway Improvements	70d
		Structural Engineering	90d
		Architectural Design	150d
		Electrical Engineering	296d
		Instrument & Control Engineering	301d
		PROCUREMENT	407d
		Services	20d
		Subcontracts	275d
		Major Equipment/Vessels	407d
		Electrical	141d
		Instrumentation	141d
		CONSTRUCTION	431d
		Infrastructure - 0050	260d
		Access Road - 0075	260d
		Personnel Housing - 0080	230d
		Landfill - 7085	180d
		General Sitework	150d
		Benefication - 2000	281d
		Shipping - 2700	281d
		Acidulation - 3000	281d
		Granulation - 4000	281d
		Sulfuric Acid Plant - 5000	281d
		Reagentes - 5050	281d
		Utility Systems - 6000	281d
		Administration and Support - 7000	281d
		Energy and Automation Systems - 8000	240d
		Substation - 8010	120d
		Power Line - 8050	240d
		COMMISSIONING	60d



◆ Milestone
Summary

MBAC EPC PHASE
Preliminary Summary Schedule
Data Date: 07/01/13 Print Date: 08/21/13
Page 1 of 1

Date	Revision	Checked	Approved
08/14/13	Preliminary Summary Schedule	CTT	RS

PEGASUS TSI

19 MARKET STUDIES AND CONTRACTS

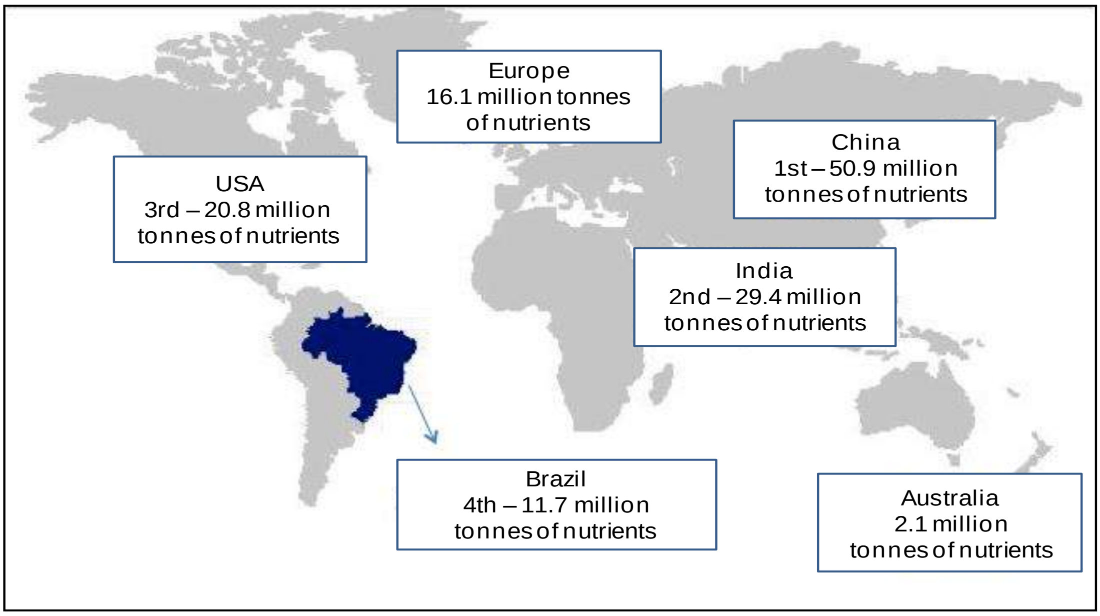
19.1 Brazilian Market Overview

19.1.1 Agribusiness

Brazil is a major global food producer. One of the reasons for this is the vast amount of land available for farming. However, the soil is poor in nutrients and to realize its full potential, a significant amount of fertilizer needs to be used. As a result, despite Brazil already being the fourth largest market for fertilizers, it still has the most potential for growth.

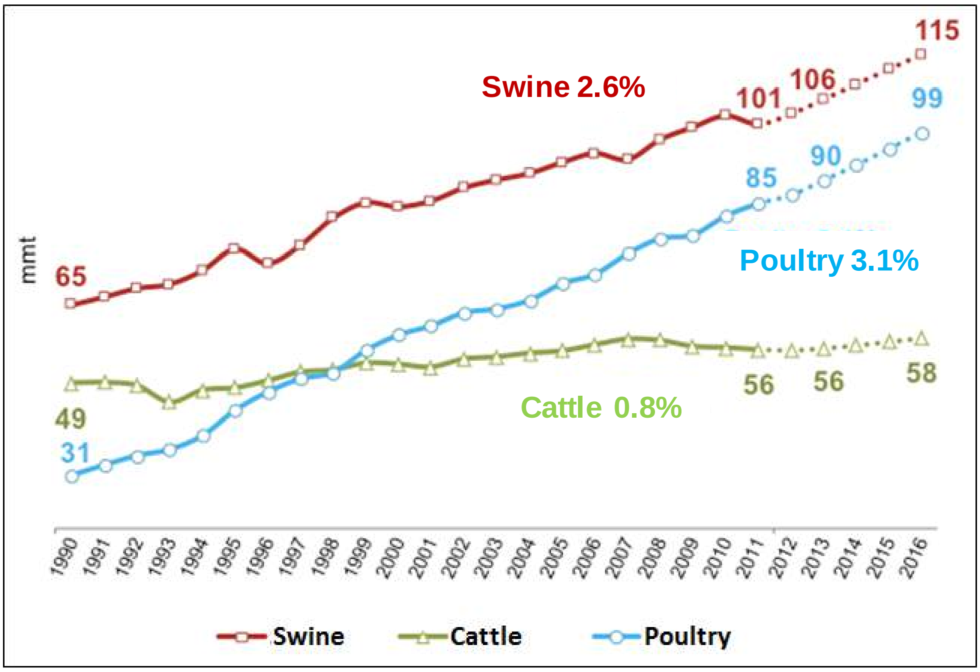
Figure 19.1.1_1

Brazil World Ranking - 2011



One of main reasons underlying this growth outlook is the food demand is expected to continue growing in the coming years as the global population grows and as average incomes improve, particularly in countries with large populations. It should be remembered that there is increasing urbanization in these countries. They are seeing diets change to include more elaborate foodstuffs which are also richer in protein. For example, a city-dweller in China may have three times the purchasing power of someone living in a rural area.

Figure 19.1.1_2
World Meat Demand



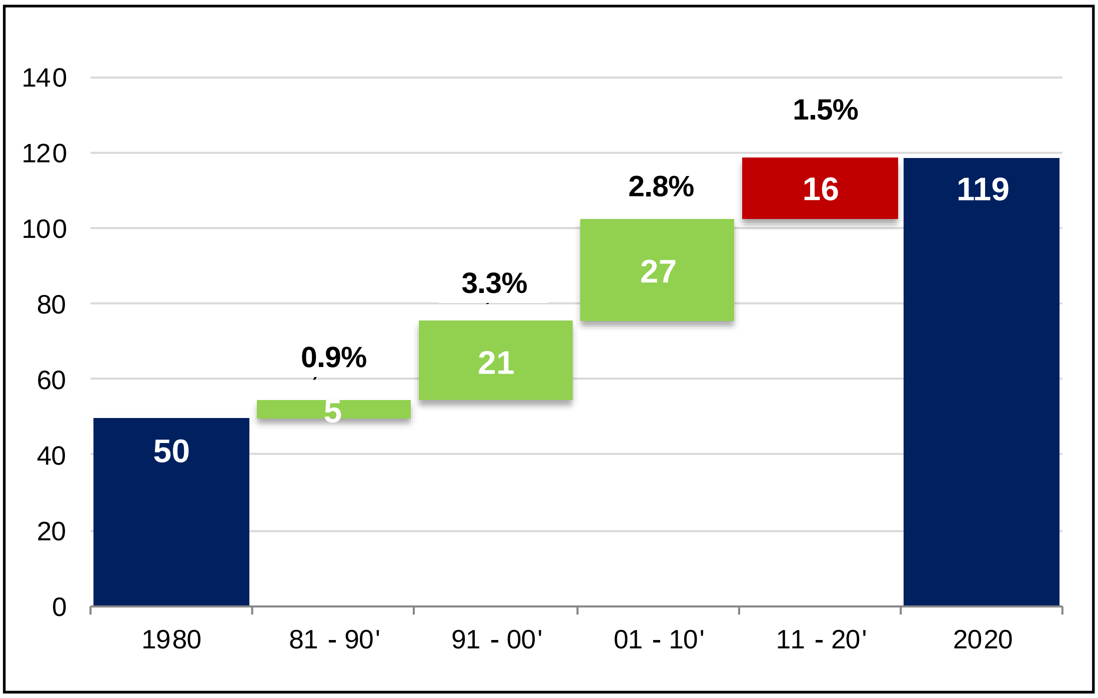
A greater use of feed is required to meet demand for animal protein. But there is another important structural change taking place in this market. In China, for example, most poultry and swine production is moving from very low technology systems where family-based, almost subsistence farming dominates, towards more technical production systems that involve using balanced animal feeds. This change is picking up speed and is having a significant impact on demand for soy meal and other feed components. As a result, from 2013 the animal protein industry is likely to post 2.3% growth for poultry and 1.5% for swine annually.

The changes in the sector producing oil for human use need to be examined. Per capita oil demand is set to rise 2.5% per annum, especially palm oil, which is set to increase 3.6% per annum, and soy oil, 2.7% per annum. China is once again one of the main drivers here, with total oil demand rising 8% annually. Growth in soy oil demand for dietary purposes is set to grow fastest over the next 5 years, from 1.4% to 3.0% annually.

This means that increased production of animal protein and oil over the next 5 years will increase soybean demand by 54 million tonnes, equivalent to 3.6% annually. This is faster than the growth of the feed market, which is rising at 2.6% per annum, and is based on greater use of soy meal in animal feed to improve feed quality, as mentioned previously.

Figure 19.1.1_3

Soybean area growth in the world. Millions hectares.



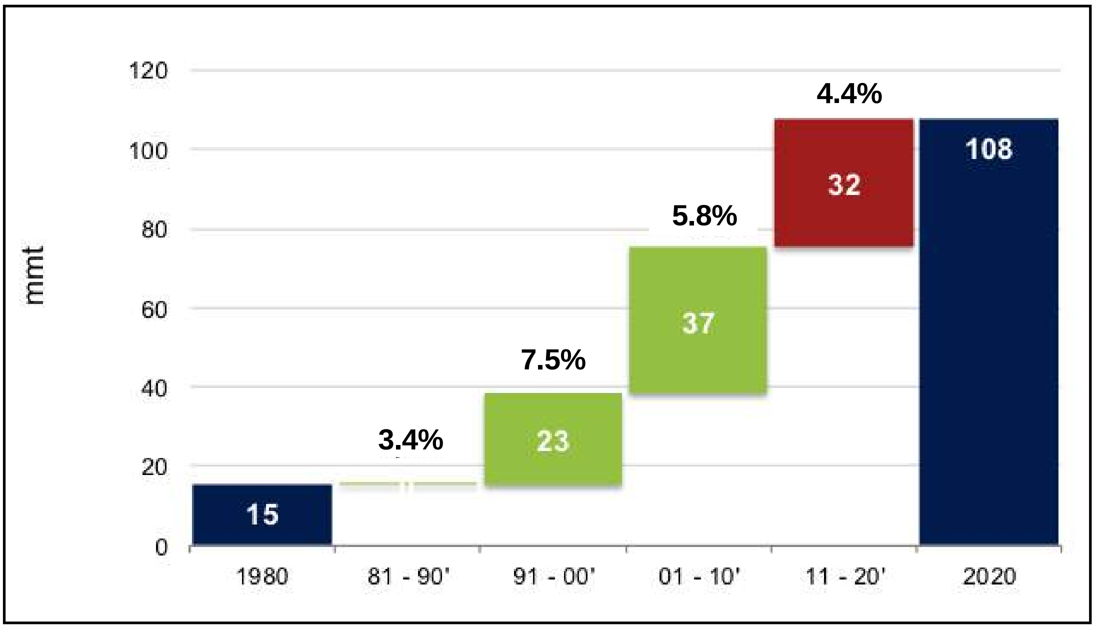
Demand growth for corn is also linked to rising feed output and ethanol production. Corn demand is set to grow to 110 million tonnes at an average annual rate of 2.4%. Brazil is likely to see increasing demand for grain. Domestic demand for soybeans and corn is set to rise 3.6% and 4.1% p.a., respectively.

Brazil is one of the countries with the most potential to expand agricultural output in order to meet rising global demand. Apart from the available land, there are good water supplies and high-tech grain production.

Approximately 22.1 million ha are available for soybean expansion in Brazil and 11.4 million ha for corn expansion. Bear in mind that certain areas in Brazil face environmental policy restrictions and logistics problems. Any increase in production will be based on the availability of new land and greater use of technology. More technology will boost productivity gains beyond traditional levels. Productivity growth for soybeans over the next decade is likely to be 2.1% p.a., above the historical average growth rate of 1.8%.

Figure 19.1.1_4

Brazil soybean production increase.

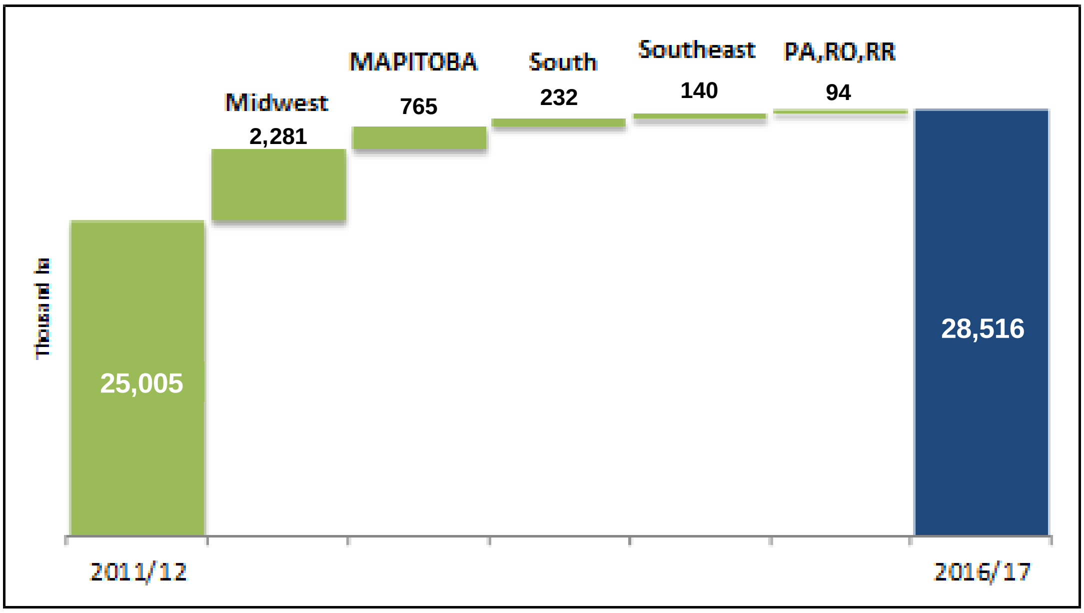


As a result, in order to meet global demand for soybean oil and meal over the next five years the cropping area will have to grow from 104.1 million ha to 115.1 million ha, an increase of 11 million ha. This 10% growth in acreage over the next five years will be concentrated mainly in South America, where Brazil has a leading role. By 2020, the cropping area is set to rise to 119 million ha.

The soybean area in Brazil will rise by 3.7 million ha by 2016 to 28.5 million ha, at a growth rate of around 2.7% per annum. From then until 2031, growth should slow slightly to around 1.4% p.a. until it reaches 35.7 million tonnes. Over the next five years, we expected a little over 2.2 million additional tonnes of soybean output in the Mid-West, and around 800 thousand ha in Maranhão, Piauí, Tocantins and Bahia, a region known as MAPITOBA.

Figure 19.1.1_5

Brazil soybean cropped area forecast.



Growth in corn acreage is likely to be concentrated mainly in Eastern Europe and Africa. In Brazil, summer corn acreage should remain stable, giving up space for soybeans in regions with less potential output.

In Brazil, the rise in the corn cropping area will focus mainly on the second season, supported by greater use of technology, larger and more efficient planting and harvesting machines and growth in soybean area. Of particular interest is the rise in second corn crop acreage in MT, which is likely to represent 56% of the expected 2 million ha increase in Brazil through 2016.

Looking at economic sustainability, Agroconsult has estimated the yield outlook for soybean and corn crops in Brazil. These estimates point towards high long-term yields, sustained by high productivity, controlled costs and high average prices over a five-year horizon.

For mid-level technology summer corn, yields are decreasing based on falling average prices. As a result, Agroconsult believes that mid-level technology corn will lose acreage to soybeans. The same is not expected for high-technology corn production that, despite falling yields, should remain competitive compared to soybeans.

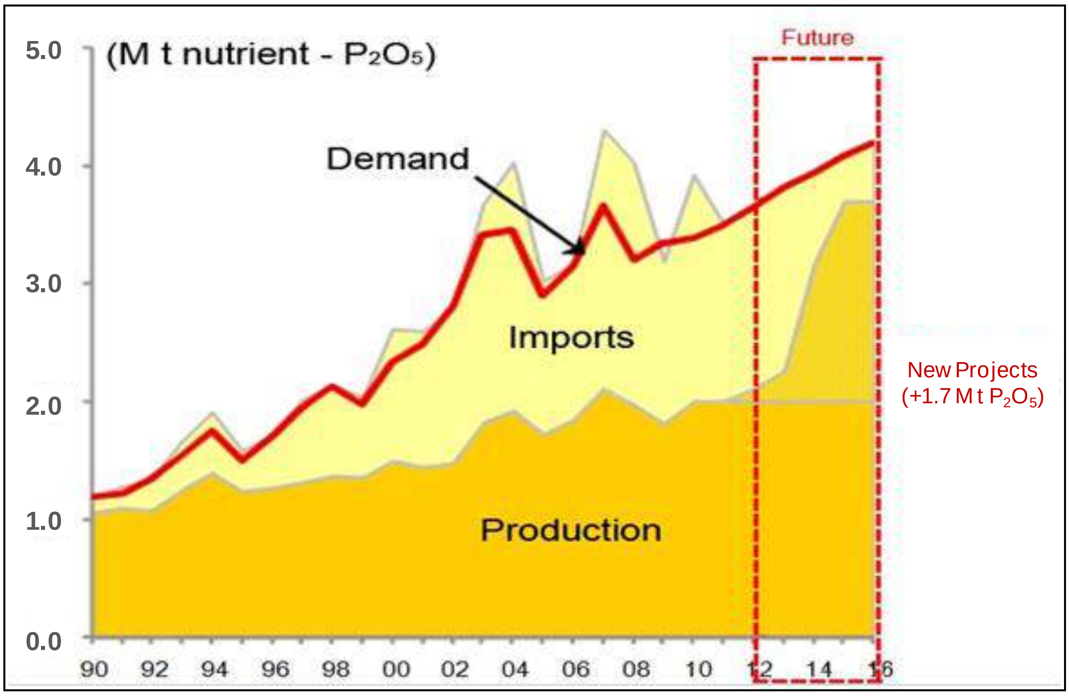
In both cases, long-term yields are positive and should support increasing acreage in Brazil for both soybeans and corn.

19.1.2 Fertilizer

Brazil is dependent on fertilizers imports; currently, almost 67% of nutrients are imported. The strong growth in agriculture output (~5.0% py) will tend to increase the need of fertilizer imports, even when taking into account the new fertilizer plants to be implemented in the coming years.

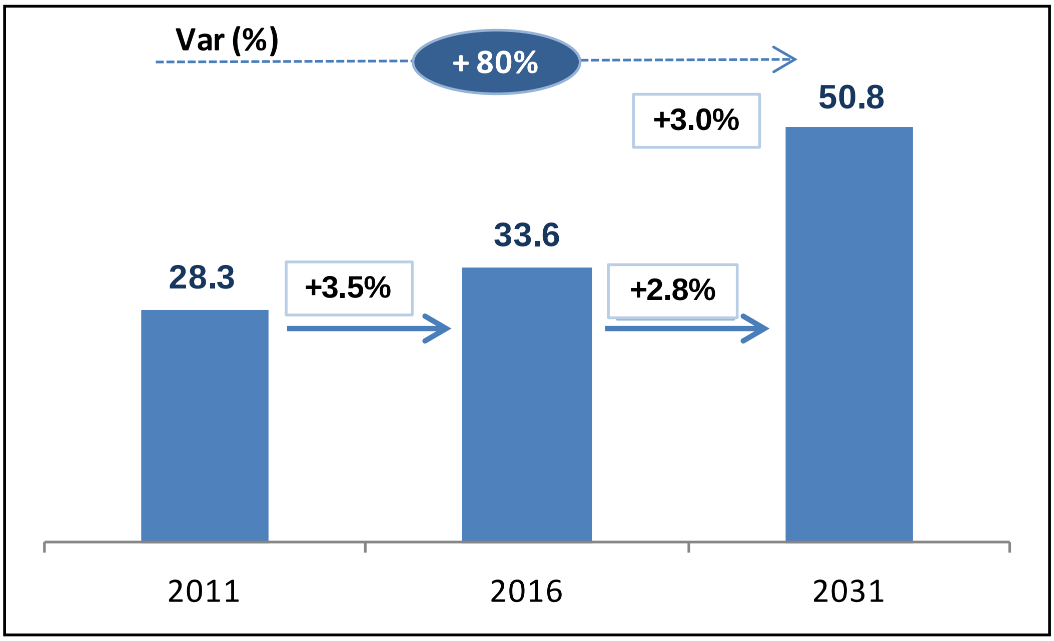
According to the National Association for the Promotion of Fertilizers ("ANDA"), in 2010, Brazil produced 9.3 million tonnes of fertilizers and has imported 15.0 million tonnes

Figure 19.1.2_1
Brazilian P_2O_5 Market Scenario



Based on the crop outlook, we are projecting an 80% increase in fertilizer demand through 2031, reaching 50.9 million tonnes. By 2016, average annual growth is estimated to be 3.4% and should be close to 2.8% in the following years. Both increases in area and more intensive usage, based on application of greater quantities of fertilizer per unit of area, are likely to drive this growth. We estimate that fertilized area will rise from 84.2 million ha to 93 million ha in 2016 and to 114.8 million ha in 2031 and average fertilizer use will increase from 337 kg/ha in 2011 to 361 in 2016 and 442 kg/ha in 2031.

Figure 19.1.2_2
Brazil Fertilizer Demand Forecast



The market by crop

Corn is the only one of the most important crops not likely to see acreage increase during the analysis period. Highlights among other crops are: reforestation, with average annual growth of 3.1% between 2011 and 2031, cotton, 2.9% p.a., sugarcane, 2.7% p.a. and second crop corn and soybeans, close to 1.8% p.a. We expect to see acreage for the groups of crops assessed to increase around 2% p.a. between 2011 and 2016 and 1.4% p.a. between 2016 and 2031. By 2031, average, annual acreage growth should be approximately 1.6%.

Table 19.1.2_1
Fertilizer Market by Crop.

Fertilizer Volume (tonne)				
	2011	2016	2031	CAGR 11-31
Soybeans	9,476,171	11,452,414	17,465,773	3.1%
Summer Corn	3,115,617	3,083,564	4,020,429	1.3%
Second Crop Corn	1,820,671	2,505,085	4,847,676	5.0%
Cotton	1,872,807	2,511,181	3,883,347	3.7%
Sugar Cane	4,212,284	5,388,783	8,666,785	3.7%
Coffee	1,815,007	1,841,446	2,303,435	1.2%
Pasture	404,613	471,658	762,046	3.2%
Reforestation	899,109	1,093,575	1,927,755	3.9%
Other	4,685,829	5,185,240	6,987,910	2.0%
Total	**28,302,109**	**33,532,945**	**50,865,156**	**3.0%**

The market by state

The Brazilian states where the biggest acreage variations are expected are Goiás, Bahia and Mato Grosso. These regions offer the most available acreage for agriculture. By 2016, average annual growth is expected to be 4.5% in MT, 3% in Goiás and 2.8% in Bahia, with overall average growth in Brazil standing at 2% p.a. Much of this growth will be driven by rising soybean planting. This means that the market will generate significant demand for phosphate fertilizers, especially low concentration fertilizes like simple superphosphate. In these areas farmers normally apply between 300 and 800 kg/hectare of phosphate and fertilize during planting, using NPK formulas containing mainly phosphorus and potassium.

Between 2016 and 2031 these same states should present growth rates of approximately 2% per annum, with average Brazilian growth at around 1.4% per annum. The southern region is likely to post slower growth, at around 0.5% per annum throughout the period.

Table 19.1.2_2
Fertilizer Market by Region.

Fertilizer Volume (t)					
	2011	2016	2031	CAGR 11-31	Var (%)
Southern Region	7,551,745	8,191,310	11,025,536	1.9%	46.0%
MT	4,672,867	6,357,981	10,793,201	4.3%	131.0%
GO	2,660,311	3,118,892	4,718,926	2.9%	77.4%
MG	3,631,192	3,950,939	5,640,939	2.2%	55.3%
SP	4,130,501	4,677,110	5,965,574	1.9%	44.4%
BA	1,865,035	2,402,470	3,969,269	3.8%	112.8%
Others	3,790,458	4,834,244	8,751,711	4.3%	130.9%
Brazil	**28,302,109**	**33,532,945**	**50,865,156**	**3.0%**	**79.7%**

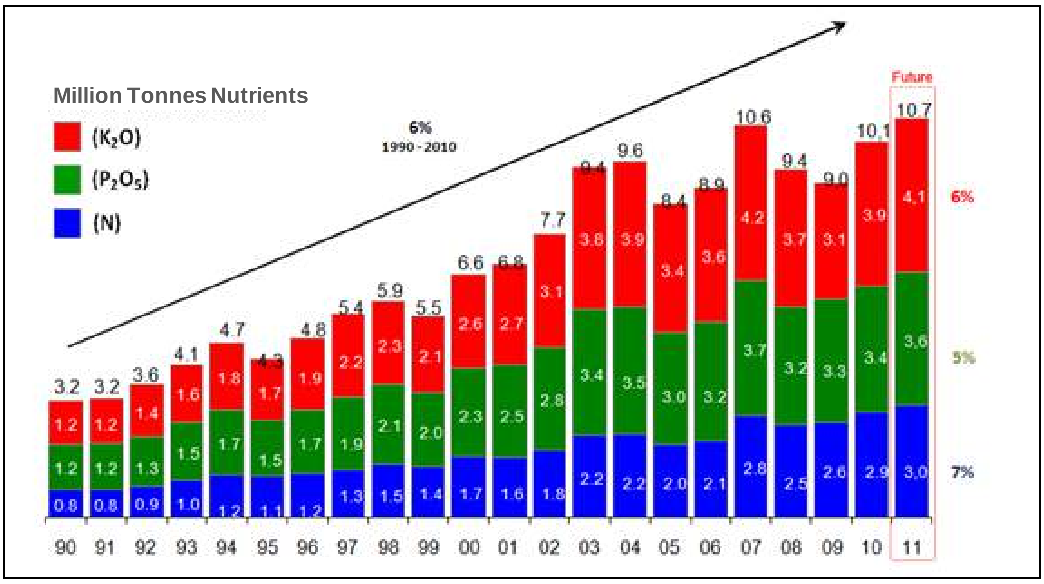
All regions and crops should see a rise in output. In Brazil, average fertilizing encompassing 335 kg/hectare in 2011 is likely to increase to 362 kg/hectare in 2016 and 442 kg/hectare in 2031. Among the biggest agricultural states, BA and MT present the highest increases in average demand of around 2% per annum. The crops recording the biggest increase in average fertilizing our second crop corn, likely to increase a little over 80% from 263 kg/hectare in 2011 to 480 kg/hectare in 2031 and;, rising from 365 kg/hectare to 482 kg/hectare. Fertilizer usage on pasture is also likely to be one of the significant risers, increasing from 197 kg/hectare in 2011 to 275 kg/hectare in 2031, a change linked to the expected increase in pasture usage based on the need to incorporate degraded areas for grain production.

The biggest regional increases in demand will come from BA and MT where the increase in fertilizer usage during this period is expected to exceed 100%. MT will exceed 10.7 million tonnes and BA will exceed 3.8 million tonnes in 2031.

The nutrient market

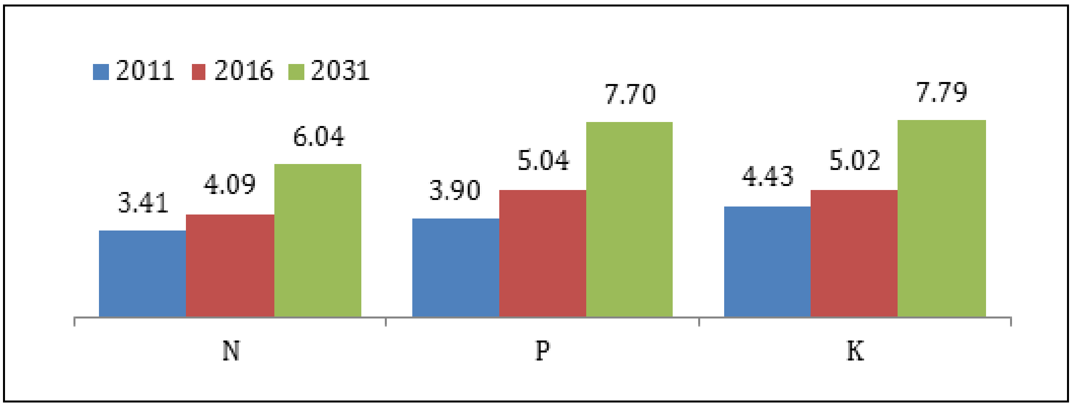
In the last 23 years, Brazil has recorded a yearly increase 5.2% in fertilizer production; the evaluation of domestic consumption of nutrients is demonstrated in the Figure 19.1.2_3.

Figure 19.1.2_3
Domestic Nutrients Consumption



Crops which require most nitrogen in Brazil are sugarcane, corn, coffee and cotton. Demand between 2011 and 2031 is expected to increase 75%, with second crop corn registering the biggest rise (+168%), followed by sugarcane (+168%) and cotton (+125%).

Figure 19.1.2_4
Brazil Fertilizer Demand Forecast by Nutrients – million tonne



Demand for phosphorus will increase 98% between 2011 and 2031, driven mainly by second crop corn and sugarcane. The crops with the biggest demand for phosphorus in 2011 were, in order of importance, soybeans, corn (summer and second crops), sugarcane and cotton.

The regions with the highest phosphorus demand are the southern states and Mato Grosso. The biggest growth driver is MT that, according to our forecasts for 2031, will exceed the southern region and consume 1.9 million tonnes of nutrients. This means that total demand will rise from the current level of 3.9 million tonnes (2011) to 7.7 million tonnes in 2031. In the southern region, growth is based on increasing levels of average fertilizing. In the Brazilian Cerrado region, particularly the states of MT, TO, GO, BA, MA and PI, growth will be driven by rising acreage, increasing average fertilizing and the need to apply phosphorus for soil correction. This technology involves using low concentration phosphates, such as simple superphosphate, which in addition to providing phosphorus also includes calcium sulfide, providing better phosphorus distribution throughout the soil and acting as a major source of sulfur, the nutrient limiting grain production in the Brazilian Cerrado region.

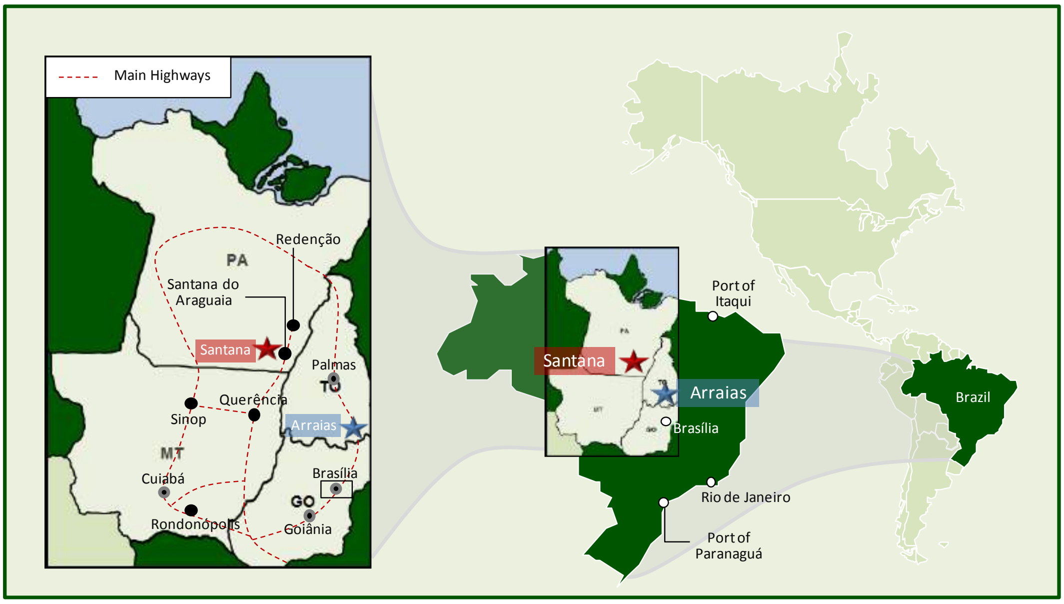
The market for raw materials

When the market is "translated" into raw materials, we find that unlike major global markets, low concentration phosphates are very important for the Brazilian market. However, volumes of other sources of phosphorus, nitrogen and potassium are also important to Brazil, as demonstrated in the Table 19.2_1.

19.2 Region of Project Influence

Santana Phosphate Project is strategically located in relation to the target area for the Project taking in account the development of the new agricultural frontier. Project influence region is: (i) Mato Grosso State (MT), and (ii) Pará State (PA). And Project target area is: (i) the northern east of Mato Grosso, (ii) and south Pará.

Figure 19.2_1
Project Area Immediate Market



Project influence region accounted for 26% of the total Brazilian Single Super Phosphate (SSP) demand in 2011 and it is expected to account for 30% of the total Brazilian demand in 2016. Total SSP demand in the target region in 2016 is expected to reach almost 2 million tonnes of product, being delivered mainly to the cultivation of soybean, corn, and cotton.

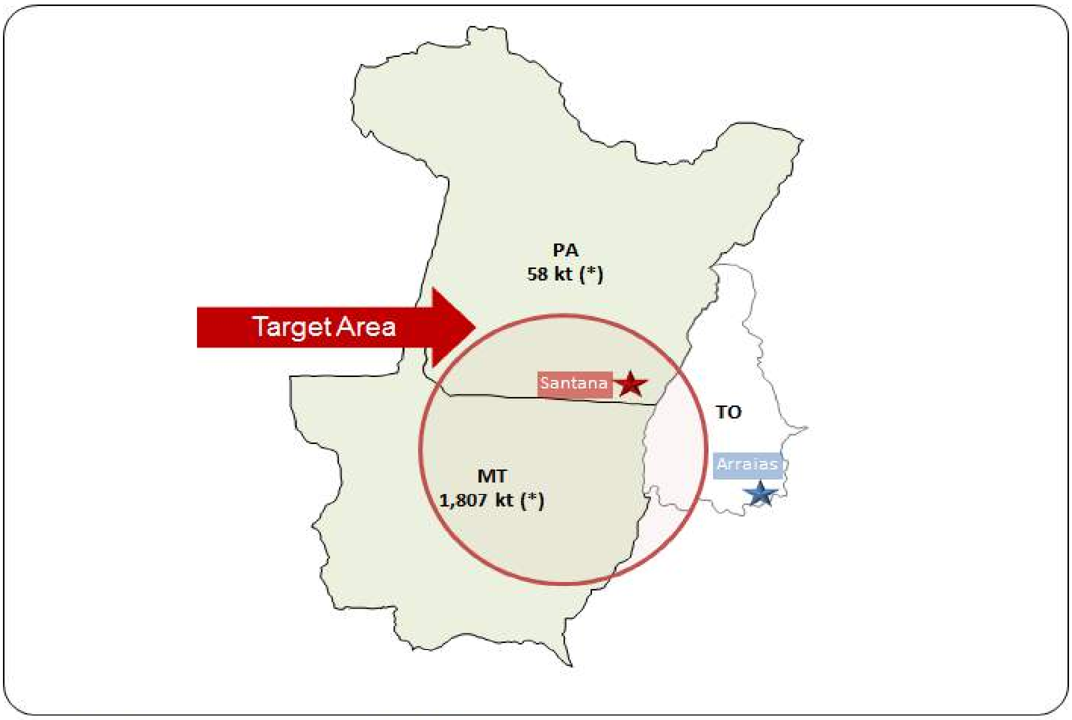
Table 19.2_1
Project Influence Region Raw Material Demand.

Demand - Influence Region	2011	2016	2021	2026	2031	CAGR			
Raw Materials	Vol. (t)	Vol. (t)	Vol. (t)	Vol. (t)	Vol. (t)	2011-2016	2011-2021	2011-2021	2011-2031
Ammonium Sulphate	304,110	459,234	568,355	678,823	805,329	8.6%	6.5%	5.5%	5.0%
Urea	390,709	600,173	728,892	852,892	986,745	9.0%	6.4%	5.3%	4.7%
Nitrates	183	243	310	396	508	5.9%	5.4%	5.3%	5.2%
Single Super Phosphate (SSP)	1,297,734	1,865,137	2,250,394	2,671,580	3,080,555	7.5%	5.7%	4.9%	4.4%
Triplo Super Phosphate (TSP)	325,722	544,801	664,181	798,110	930,553	10.8%	7.4%	6.2%	5.4%
Diammonium Phosphate (DAP)	123,049	187,458	243,997	304,964	377,738	8.8%	7.1%	6.2%	5.8%
Monoammonium Phosphate (MAP)	512,339	737,063	905,270	1,090,431	1,284,759	7.5%	5.9%	5.2%	4.7%
SSP w/ Ammonia (Binary)	55,541	50,871	61,455	71,521	81,671	-1.7%	1.0%	1.7%	1.9%
Termo-Phosphate	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Reactive Natural Phosphate	114,031	186,868	225,889	267,784	308,541	10.4%	7.1%	5.9%	5.1%
Potash Sulphate	17	17	17	17	17	0.8%	0.2%	0.1%	0.0%
Potash Chlorite (KCl)	1,488,811	1,852,298	2,260,327	2,700,457	3,149,641	4.5%	4.3%	4.0%	3.8%
Complexes	143,366	98,798	121,968	145,275	170,788	-7.2%	-1.6%	0.1%	0.9%
Micro-Nutrients	136,137	80,529	98,404	116,890	135,942	-10.0%	-3.2%	-1.0%	0.0%
Others	22,123	149	178	222	264	-63.2%	-38.3%	-26.4%	-19.9%
Total	**4,913,874**	**6,663,639**	**8,129,637**	**9,699,361**	**11,313,051**	**6.3%**	**5.2%**	**4.6%**	**4.3%**

Source: Agroconsult

During the next 5 years, demand will increase at high levels mainly because of the high number of new cultivate areas and the increase of the amount of fertilizer used per hectare in planted areas.

Figure 19.2_2
Santana Project influence region.



(*) Demand of SSP in 2016 (in tonnes of product)

Mato Grosso state accounts for the most representative SSP demand in the influence region, representing 97% of the total influence region's SSP demand in 2016, and reaching more than 1.8 million tonnes of SSP products.

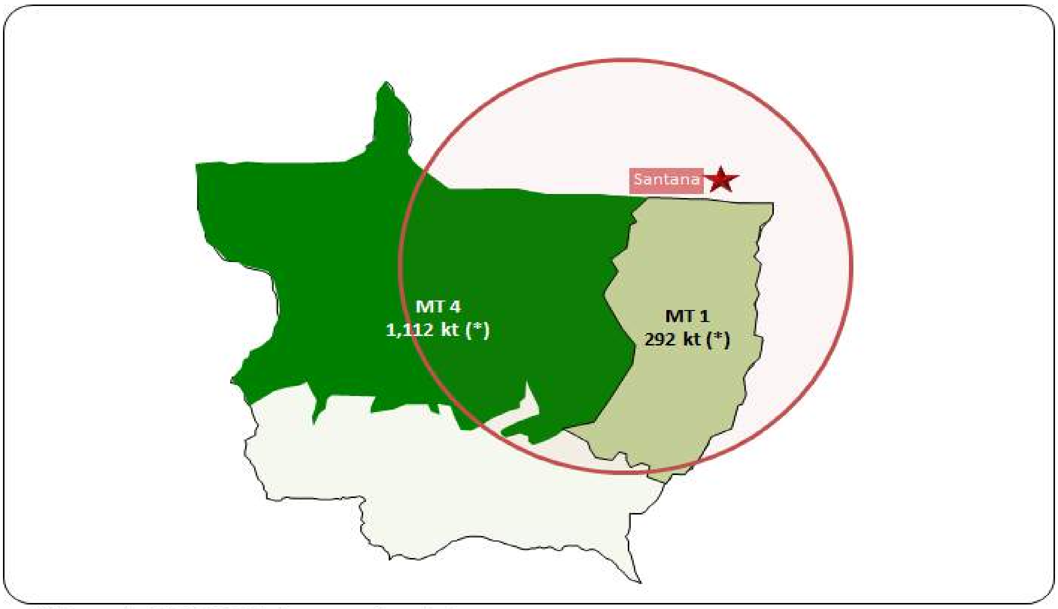
Table 19.2_2
Mato Grosso State Raw Material Demand.

Demand - MT State	2011	2016	2021	2026	2031	CAGR			
Raw Materials	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	2011-2016	2011-2021	2011-2021	2011-2031
Ammonium Sulphate	270,254	419,783	522,885	626,242	744,271	9.2%	6.8%	5.8%	5.2%
Urea	370,158	558,599	681,000	797,568	922,564	8.6%	6.3%	5.3%	4.7%
Nitrates	11	13	15	17	20	3.5%	3.1%	3.0%	3.0%
Single Super Phosphate (SSP)	1,253,385	1,807,159	2,181,964	2,591,088	2,985,937	7.6%	5.7%	5.0%	4.4%
Triplo Super Phosphate (TSP)	308,281	518,243	631,390	757,732	880,872	10.9%	7.4%	6.2%	5.4%
Diammonium Phosphate (DAP)	123,043	187,451	243,991	304,957	377,731	8.8%	7.1%	6.2%	5.8%
Monoammonium Phosphate (MAP)	487,318	695,021	852,492	1,024,127	1,201,115	7.4%	5.8%	5.1%	4.6%
SSP w/ Ammonia (Binary)	55,541	50,871	61,455	71,521	81,671	-1.7%	1.0%	1.7%	1.9%
Termo-Phosphate	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Reactive Natural Phosphate	111,267	176,521	213,132	252,219	289,687	9.7%	6.7%	5.6%	4.9%
Potash Sulphate	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Potash Chlorite (KCl)	1,426,221	1,777,459	2,171,027	2,593,970	3,022,093	4.5%	4.3%	4.1%	3.8%
Complexes	135,117	88,932	110,453	132,187	155,802	-8.0%	-2.0%	-0.1%	0.7%
Micro-Nutrients	132,260	77,846	95,157	112,987	131,271	-10.1%	-3.2%	-1.0%	0.0%
Others	12	85	105	138	166	48.0%	24.2%	17.7%	14.0%
Total	4,672,867	6,357,981	7,765,065	9,264,753	10,793,201	6.4%	5.2%	4.7%	4.3%

Source: Agroconsult

However, considering the location of the Santana Project, the main focus of the project influence is the Mato Grosso State sub-regions 1 and 4 that together will account for 75% of SSP demand in the influence region in 2016. Picture below shows the location of that both sub-regions.

Figure 19.2_3
Sub-regions 1 and 4 of Mato Grosso State.



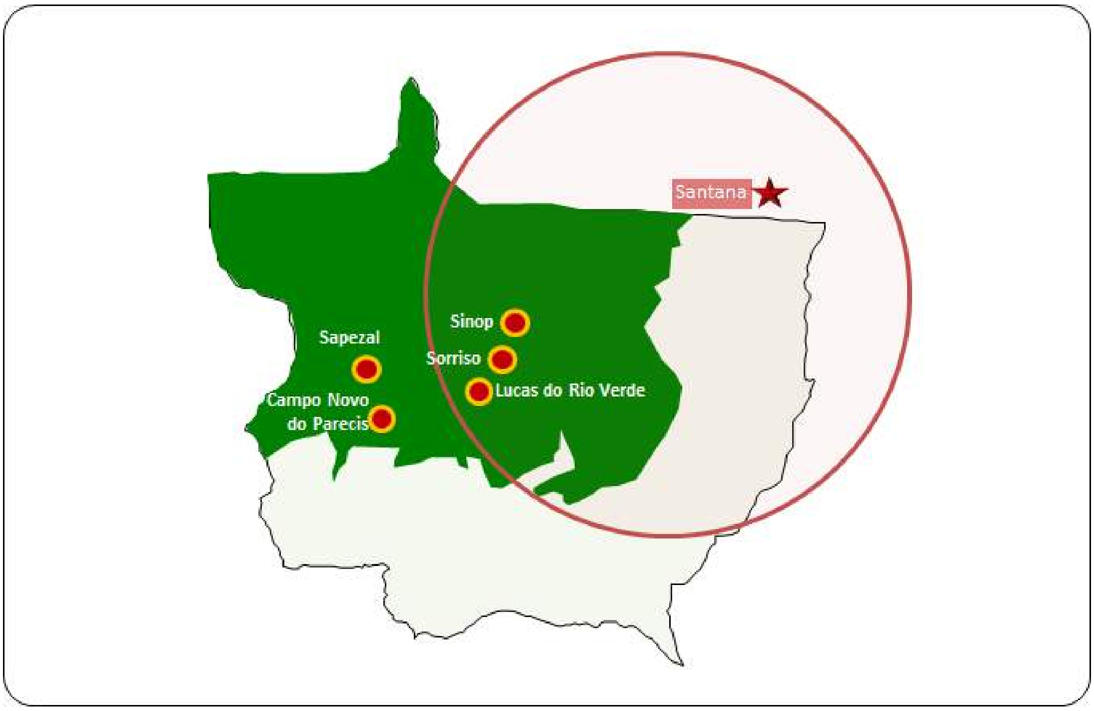
(*) Demand of SSP in 2016 (in tonnes of product)

MT Sub-Region 4

In 2016, Mato Grosso Sub-region 4 will account for 60% of the total SSP expected demand of the Santana Project influence region, reaching 1.1 million tonnes of SSP products. The main important fertilizer clusters in that sub-region are the cities of: Sepezal; Campo Novo do

Parecis; **Sinop; Sorriso; and Lucas do Rio Verde**, being the last 3 the main target of the Santana Project.

Figure 19.2_4
Mato Grosso Sub-region 4.



During the next 5 years, SSP demand in MT sub-region 4 will increase at 6.6% per year, which is lower than the Mato Grosso state growth in the period, mainly because the region's higher development when compared with the other Mato Grosso sub-regions.

Table 19.2_3
Mato Grosso Sub-Region 4 Raw Material Demand.

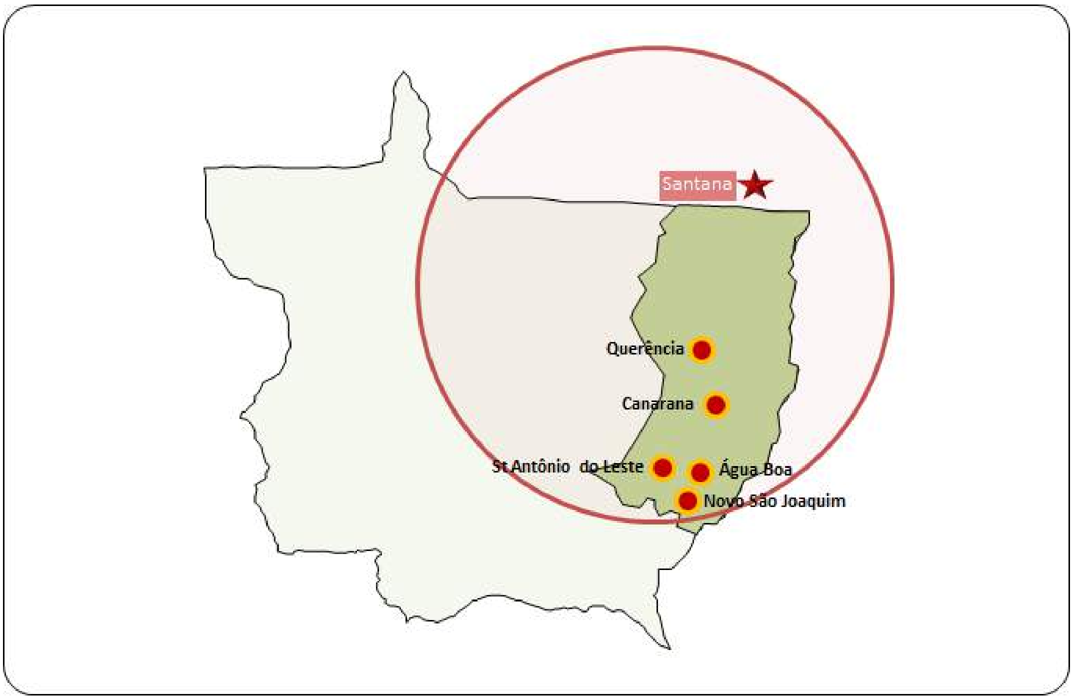
Demand - MT 4	2011	2016	2021	2026	2031	CAGR			
Raw Materials	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	2011-2016	2011-2021	2011-2021	2011-2031
Ammonium Sulphate	162,121	247,017	304,629	361,766	426,721	8.8%	6.5%	5.5%	5.0%
Urea	192,455	256,109	303,218	342,871	384,841	5.9%	4.7%	3.9%	3.5%
Nitrates	6	7	7	8	10	2.8%	2.7%	2.7%	2.7%
Single Super Phosphate (SSP)	807,069	1,112,449	1,282,270	1,449,723	1,588,328	6.6%	4.7%	4.0%	3.4%
Triplo Super Phosphate (TSP)	189,805	264,865	303,339	340,975	369,546	6.9%	4.8%	4.0%	3.4%
Diammonium Phosphate (DAP)	88,449	158,521	205,622	256,106	316,114	12.4%	8.8%	7.3%	6.6%
Monoammonium Phosphate (MAP)	294,769	343,466	409,656	476,154	543,469	3.1%	3.3%	3.2%	3.1%
SSP w/ Ammonia (Binary)	31,287	27,243	31,486	35,088	38,199	-2.7%	0.1%	0.8%	1.0%
Termo-Phosphate	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Reactive Natural Phosphate	72,473	109,074	125,191	140,340	152,269	8.5%	5.6%	4.5%	3.8%
Potash Sulphate	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Potash Chlorite (KCl)	871,013	1,113,421	1,309,864	1,502,854	1,684,063	5.0%	4.2%	3.7%	3.4%
Complexes	68,808	46,892	57,358	67,216	78,116	-7.4%	-1.8%	-0.2%	0.6%
Micro-Nutrients	85,594	77,384	90,098	102,313	113,058	-2.0%	0.5%	1.2%	1.4%
Others	0	1	2	2	2	0.0%	0.0%	0.0%	0.0%
Total	**2,863,850**	**3,756,448**	**4,422,739**	**5,075,416**	**5,694,736**	**5.6%**	**4.4%**	**3.9%**	**3.5%**

Source: Agroconsult

MT Sub-Region 1

In 2016, Mato Grosso Sub-region 1 will account for 16% of the total SSP expected demand of the Santana Project influence region, reaching 0.3 million tonnes of SSP products. The main important fertilizer clusters in that sub-region are the cities of: **Querência; Canarana**; St Antônio do Leste; Água Boa; and Novo São Joaquim, being the first 2 the main target of the Santana Project.

Figure 19.2_5
Mato Grosso Sub-region 1.



During the next 5 years, SSP demand in MT sub-region 1 will increase at 12.4% per year, representing the highest growth rate in Mato Grosso State, having Querência as the main drive to that high rate increase.

Mato Grosso Sub-Region 1 is the main target of Santana Project mainly because its high demand increase and because of the very short distance to the Project's site, resulting in a great logistic advantage. The distance covered by products coming from the South east region is about 2,000km and for those coming from Goias is 1,500km.

Table 19.2_4
Mato Grosso Sub-Region 4 Raw Material Demand.

Demand - MT 1	2011	2016	2021	2026	2031	CAGR			
Raw Materials	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	Vol. (tonnes)	2011-2016	2011-2021	2011-2021	2011-2031
Ammonium Sulphate	24,927	34,519	47,753	63,097	81,606	6.7%	6.7%	6.4%	6.1%
Urea	24,842	38,273	52,594	68,246	87,000	9.0%	7.8%	7.0%	6.5%
Nitrates	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Single Super Phosphate (SSP)	163,132	292,390	424,511	589,837	777,879	12.4%	10.0%	8.9%	8.1%
Triplo Super Phosphate (TSP)	37,895	70,936	103,618	144,564	190,987	13.4%	10.6%	9.3%	8.4%
Diammonium Phosphate (DAP)	7,668	14,892	20,822	27,823	36,690	14.2%	10.5%	9.0%	8.1%
Monoammonium Phosphate (MAP)	51,545	69,648	98,942	134,812	176,522	6.2%	6.7%	6.6%	6.3%
SSP w/ Ammonia (Binary)	5,080	5,769	8,443	11,661	15,350	2.6%	5.2%	5.7%	5.7%
Termo-Phosphate	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Reactive Natural Phosphate	14,965	28,088	41,109	57,297	75,692	13.4%	10.6%	9.4%	8.4%
Potash Sulphate	0	0	0	0	0	0.0%	0.0%	0.0%	0.0%
Potash Chlorite (KCl)	151,201	242,415	349,801	482,187	634,107	9.9%	8.7%	8.0%	7.4%
Complexes	7,030	5,631	7,926	10,471	13,564	-4.3%	1.2%	2.7%	3.3%
Micro-Nutrients	15,887	17,969	26,217	36,423	48,074	2.5%	5.1%	5.7%	5.7%
Others	0	5	6	7	9	0.0%	0.0%	0.0%	0.0%
Total	**504,179**	**820,534**	**1,181,743**	**1,626,426**	**2,137,481**	**10.2%**	**8.9%**	**8.1%**	**7.5%**

Source: Agroconsult

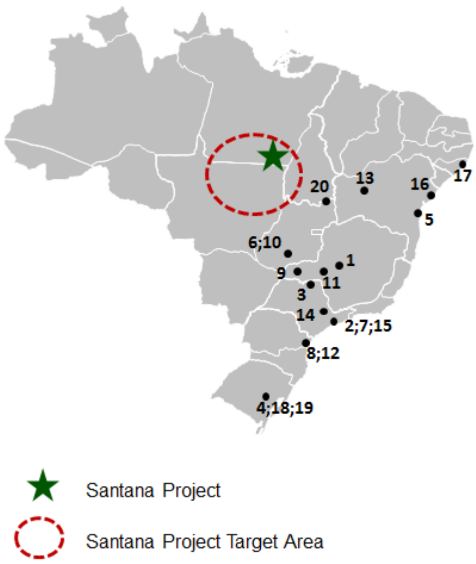
19.3 Competitors

Currently, there is no alternative to P_2O_5 required at the target area, resulting the requirement of import fertilizer from other domestic regions or abroad.

Figure 19.3_1
Competitors

Company	Mill Plant Location	Capacity	
Bunge	Araxá (MG)	1,300,000	1
Bunge	Cubatão (SP)	700,000	2
Bunge	Guará (SP)	350,000	3
Bunge	Rio Grande (RS)	225,000	4
Cibrafertil	Camaçari (BA)	230,000	5
Copebrás	Catalão (GO)	550,000	6
Copebrás	Cubatão (SP)	250,000	7
Heringer	Paranaguá (PR)	250,000	8
Fosfertil	Uberaba (MG)	280,000	9
Fosfertil	Catalão (GO)	350,000	10
Fosfertil	Patos de Minas (MG)	100,000	11
Fospar	Paranaguá (PR)	520,000	12
Galvani	LEM (BA)	400,000	13
Galvani	Paulínia (SP)	700,000	14
Mosaic	Cubatão (SP)	295,000	15
Timac Agro	Candeias (BA)	180,000	16
Timac Agro	Santa Luiza do Norte (AL)	120,000	17
Timac Agro	Rio Grande (RS)	250,000	18
Yara	Rio Grande (RS)	800,000	19
Itafós	Arraias (TO)	500,000	20
Total		**8,350,000**	

Mill Plant Location



Source: ANDA, 2011

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The mainly supplement of the P_2O_5 consumed at the target region came from (i) Port of Itaqui (imported); (ii) Port of Santos (imported); (iii) Port of Paranaguá (Imported); and (iv) southern Minas Gerais and Goias states (Domestic), where there are some competitors.

The principal product route is via Port of Itaqui, because it is the closest port of the region and domestic producers are more competitive in other domestic regions, for example, south eastern Brazilian region.

19.4 Project Santana Competitive Advantage

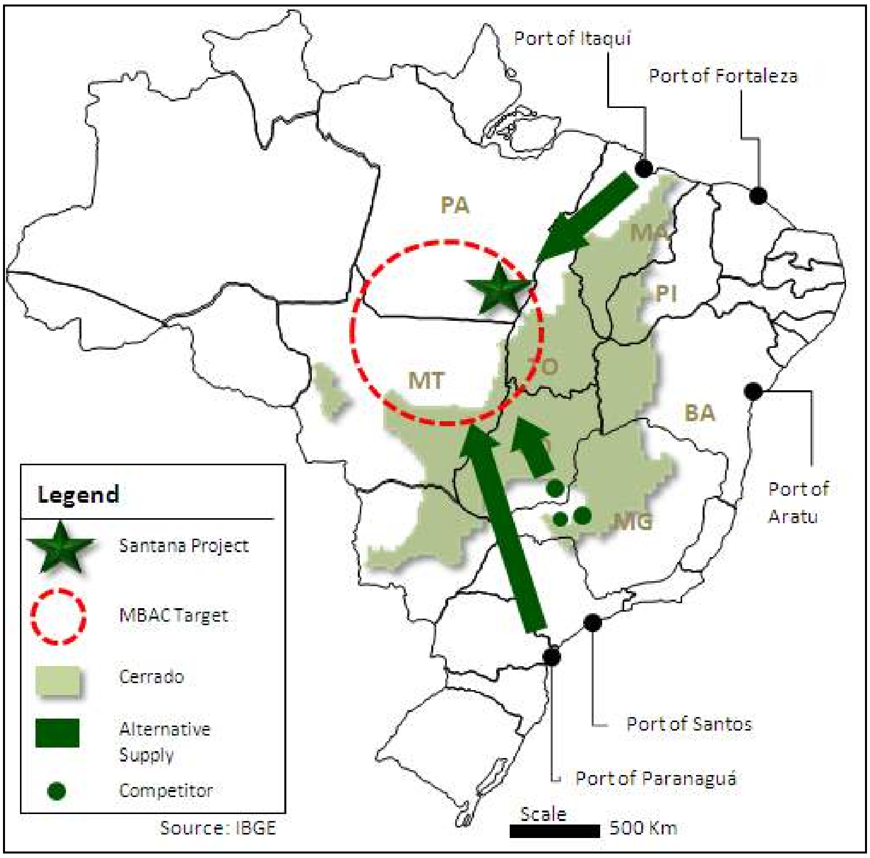
Santana Phosphate Project has a significant competitive advantage resulting from strategic location.

The distance covered by products coming from the South east region of Brazil (Port of Paranaguá and Port of Santos) is about 2,000km; for those coming from Goias and Minas Gerais states is 1,000km; and 1,500 km from Itaqui Port.

Those large distances results in a large freight costs comparing to Santana Phosphate Project product, giving the project a large competitive advantage.

Figure 19.4_1 shows Santana Phosphate Project location and the location of their main competitors.

Figure 19.4_1
Competitors Distances to Santana Phosphate Project Area



20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Issues

The Santana Phosphate Project is inserted into an area dominated by cattle breeding activities, with large areas where native vegetation has been converted to pasture planting.

The site where it will take place the ore extraction has no native vegetation, so the local impact with the implementation of the project will be minor on the region's typical vegetation.

From a social standpoint, the region is dominated by large properties and there are no cities or communities, even small groups of homes are unusual. There are house groups in particular properties for employees, which do not constitute communities. Due to these characteristics, four or five large farms surrounding the area affected by PFS may maintain their economic activities after the project installation.

Figure 20.1_1
Santana do Araguaia

Although the project is located in Sao Felix do Xingu County, the distance to the nearest city, Santana do Araguaia, is approximately 210 kilometers by road. There is no road from Sao Felix do Xingu downtown area to the project site. This distance between project site and downtown areas reduces the possibility of big impacts on the municipal seats, caused by operation of the project.

Figure 20.1_2 **Cleaning Areas of Pasture**	**Figure 20.1_3** **Farms in the Project Area**
	

Figure 20.1_4 **Palm Species Identified in the Study Area**	**Figure 20.1_5** **Pasture Area**
	

20.2 Brazilian Environmental Regulation

The current Brazilian Environmental Regulation was built from 1981, with the approval of the 6938 act, on 31 August 1981, which created the National Environment Policy, and regulates the 1st Article of the Federal Constitution of Brazil, sections VI and VII of 23rd Article. This Article determines the responsibility of the Federal Government, States, Federal District and municipalities to protect the environment and fight pollution in any form, and preserve the forests, fauna and flora.

In addition to the National Environmental Policy were established your objectives and mechanisms for formulating and implementing the National Environmental System (SISNAMA) and the Environmental Defense Register.

The SISNAMA consists of the organs and entities of Federal, State, Federal District, the Territories and the Municipalities, as well as foundations established by the Government, responsible for protecting and improving environmental quality.

The SISNAMA's consultative and deliberative agency is the National Environment Council (CONAMA), whose objective is to propose guidelines, norms and standards for an ecologically balanced environment and essential to a healthy quality of life.

The executing agency is the Brazilian Institute of Environment and Natural Renewable Resources (IBAMA), which objective is to implement and enforce, as a federal agency, government policy and guidelines established for the environment, just as the state and municipalities agencies responsible for implementing programs, projects and the control and supervision of activities capable of causing environmental degradation.

States and municipalities within its sphere of competence and areas of their jurisdiction may prepare supplementary and complementary rules and standards related to the environment, observing that are established by CONAMA.

The first CONAMA Resolution of January 23, 1986, establishes definitions, responsibilities, basic criteria and general guidelines of the Environmental Impact Study as an instrument of the National Environmental Policy.

This resolution considers the environmental impact of any change in physical properties, chemical and biological environment, caused by any form of matter or energy resulting from human activities that directly or indirectly affect the health, safety and welfare of the population; the social and economic activities; biota; aesthetic and sanitary conditions of the environment, and quality of environmental resources.

So, it will depend on development of Environmental Impact Study (EIA) and Environmental Impact Report (EIR), to be submitted for approval by the competent state agency or IBAMA, the licensing of activities that modify the environment, including the extraction of ore.

Also according to this resolution, the environmental impact study should include all technological alternatives and location of project, comparing them with the hypothesis of non-execution of the project; identify and systematically assess the environmental impacts generated during the deployment and operation of the activity; define the limits of the geographic area to be directly or indirectly affected by, called the project area of influence, considering in all cases in which the watershed is located; and consider the plans and government programs, proposed and under implementation over the area of influence of the project and its compatibility.

Under the regulation, whenever necessary, IBAMA or State agency may promote the public hearing for information about the project and its environmental impacts.

The 6.938/81 Act is considered the origin of the Brazilian Environmental Regulation because it determines, in its article 10 that "the construction, installation, expansion and operation of facilities and activities that use environmental resources considered effective and potentially polluting as well as capable in any form, to cause environmental degradation, depend on prior licensing of competent state agency, member of the National System of Environment - SISNAMA, and the Brazilian Institute of Environment and Natural Resources - IBAMA, in a supplementary fashion, without prejudice to other required licenses. " (Wording given by the 7.804 Act, in 1989).

In 1997, the CONAMA Resolution No. 237 established a review of the procedures and criteria used in the environmental permission process, including the need to establish criteria for exercising the power to license (state or federal).

The CONAMA 237 further stipulates that licenses will be issued for the activity. The main licenses are:

- The Preliminary License (LP), which is given in the preliminary planning stage approving its location and design, environmental sustainability certifying and establishing the basic requirements and conditions to be met in the next stages of its implementation;
- The Installation License (LI), authorizing the installation of the project or activity in accordance with the specifications of the plans, programs and projects approved, including the environmental control measures and other conditions, which constitute a determinant reason;
- The Operating License (LO), which authorizes the operation of the activity or project, after the verification of effective compliance with the permits listed above, with the environmental control measures and requirements for the operation.

To receive these permits, the activity must meet the environmental licensing procedures, which comprises the following steps:

1. Definition by the competent environmental agency, with the participation of the entrepreneur, about documents, projects and environmental studies needed to initiate the licensing process corresponding to the permit being requested;
2. Application of environmental permit by the entrepreneur, together with the documents, relevant projects and environmental studies, giving due publicity;
3. Analysis by the competent environmental agency, about documents, projects and environmental studies presented and the performance of technical inspections when necessary;
4. Request for clarification and additional information by environmental agency, once as a result of the analysis of documents, projects and environmental studies presented, when appropriate, and there may be a reiteration of the same request if the clarifications and additions have not been satisfactory;
5. Public hearing, when applicable, in accordance with relevant regulations;
6. Request for clarification and additional information by the competent environmental agency arising out of public hearings, when appropriate, and there may be restatement of the request when the clarifications and additions have not been satisfactory;
7. Issue of technical opinion and, where applicable, a legal opinion;
8. Approval or rejection of license application, giving due publicity.

Once completed these steps the activity may be performed, and it’s considered legal under Brazilian Environmental Laws.

20.3 Current Status of Santana Phosphate Project

The Santana Phosphate Project has finished its initial phase of environmental licensing. According to the analysis, the project and its impacts are restricted to the state of Pará, and it was concluded that the process for obtaining environmental permits could be pursued and approved at the state level, as permitted by applicable law.

So, on 1st November 2011, the MBAC Fertilizer requested the Secretariat of Environment of the State of Pará (SEMA/PA) the Guidelines (TR) for the environmental licensing of the Project Phosphate Santana, as enterprise large, therefore subject to the submission of the EIA /RIMA (EIS – Environmental Impact Study).

The company that performed the EIA/RIMA (EIS – Environmental Impact Study) was AMBIENGER ENGINEERING, who participated in the environmental licensing of another enterprise of MBAC, the Itafos Arraias Project, in Tocantins State.

The EIS was completed in early 2013 and in June 2013, SEMA/PA manifested the Santana Project EIS acceptation, which means the studies were made according to the best technical procedures, and it is currently been analyzed by the SEMA/PA technical team.

The next step is the scheduling of the public hearings and final review of the environmental studies presented, and then the issuance of the Preliminary Permit (LP) of the enterprise

After the Preliminary Permit MBAC should prepare and present the Basic Environmental Programs (PBAs), containing actions, programs and environmental plans to minimize or offset the negative effects and maximize positive effects related to the project. SEMA/PA will analyze the PBAs and if agreed it will issue the Installation Permit (LI), allowing the beginning of construction.

According to Brazilian law a mine closure plan will be elaborated few years after the mine operations start. This report already considers spending US$20 million in the last 2 years of the mine life for closure requirements.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

21.1.1 Mine Capital Costs

21.1.1.1 Unit Cost Assumptions

An owner-operator model will used on the Santana Project.

Table 21.1.1.1_1 shows unit costs for each equipment, with and without taxes. Equipment prices are taken from suppliers quotations. Prices include freight and erection.

Table 21.1.1.1_1
Equipment Prices

Equipment	Life (h)	Cost with Taxes	Cost without Taxes
	h	´000 US$	´000 US$
Production Drill Rig	30,000	1,316	908
Excavator	35,000	618	543
Front End Loader	30,000	318	280
Haul Truck	20,000	251	234
Bulldozer	40,000	730	642
Motor grader	35,000	270	238
Water Truck	20,000	144	134
Fuel Truck	20,000	159	148
Rock Breaker	25,000	104	96
Lube Truck	20,000	205	190
Lowboy Truck	20,000	206	192
Pickup	20,000	37	32
Exploration Drill Rig	30,000	568	528
Lightning Tower	15,000	13	12

The estimated mine capital cost includes the following items:

- Mine major equipment
- Mine support equipment
- Shop tools
- Initial spare parts
- Engineering and geology equipment
- Equipment residual value

This estimate does not include the following mine physical structures:

- Fuel and lubricant storage facilities
- Explosive storage facilities

- The mine shop, offices, and warehouse

It is anticipated that the vendors will provide storage for fuel, lubricants, and explosives as part of their contract of work and that the prices for these items are included in the delivered price. The mine shop and warehouse are included in the infrastructure capital cost.

Contingency is not included in the mine capital cost. It is possible that final negotiated sales prices, with fleet discounts, will be somewhat lower than the budget quotes used for this study.

21.1.1.2 Mine Capital Cost

Initial capital is US$12.41 million. This includes US$ 7.74 million in mine equipment, US$ 4.21 million in operational expenditures during the prestripping period and US$ 0.47 million in other expenses.

Sustaining capital amounts to US$37.79 million. These are all capital expenditures from Y01 onwards, and include equipment replacement and fleet increases needed to maintain a yearly production of 300 kt of P2O5 concentrate.

Total capital expenditures during the life of the project, including both initial capital and sustaining capital is US$ 50.20 million.

All the above-mentioned costs are inclusive of taxes.

Table 21.1.1.2_1 summarizes the mine capital costs.

Table 21.1.1.2_1
Capital Expenditures

Period	Opex	Equipment	Others	Total
	‘000 US$	‘000 US$	‘000 US$	‘000 US$
PS	4,205	7,744	465	12,413
Y01		730	44	774
Y02		0	0	0
Y03		0	0	0
Y04		0	0	0
Y05		2,641	158	2,799
Y06		170	10	180
Y07		730	44	774
Y08		1,206	72	1,278
Y09		2,010	121	2,130
Y10		284	17	301
Y11		1,630	98	1,728
Y12		739	44	784
Y13		2,010	121	2,130
Y14		1,205	72	1,277
Y15		1,403	84	1,487
Y16		395	24	419

Table 21.1.1.2_1 Capital Expenditures				
Period	Opex	Equipment	Others	Total
	'000 US$	'000 US$	'000 US$	'000 US$
Y17		3,501	210	3,711
Y18		13	1	14
Y19		555	33	589
Y20		2,303	138	2,441
Y21		2,193	132	2,325
Y22		588	35	623
Y23		1,120	67	1,187
Y24		1,246	75	1,320
Y25		2,641	158	2,799
Y26		646	39	685
Y27		1,282	77	1,359
Y28		1,442	87	1,529
Y29		2,673	160	2,833
Y30		291	17	308
Y31		0	0	0
Y32		0	0	0
Total	**4,205**	**43,391**	**2,603**	**50,200**

21.1.2 Mine & Industrial Sites Capital Costs

Santana Project Capital Costs have been estimated on October 2013. Total capital cost for the project is estimated to be US$ 426.7 million, including US$ 50 million in contingencies. Table 21.1.2_1 shows all Capital Cost breakdown.

The accuracy assumed for the CAPEX is ±10%.

BR$/US$ exchange rate considered was 2.84, which was based on the weighted average exchange rates forecast between 2013-2016 that were obtained from Bloomberg.

Table 21.1.2_1 Capital Cost Breakdown		
Item	BRL '000s	US$ '000s
Mine Fleet & Preparation		12,413
Beneficiation Plant		38,298
Infrastructure, Buildings & Utilities		71,514
Energy, Automation, Telecommunication		21,768
Power Line & Main Substation		14,311
Tailing Dam		8,088
Housing Infrastructure		12,814
Access Road		16,621
Sulphuric Acid Plant		38,237
Acidulation Plant		15,032
Granulation Plant		30,247
Indirect Construction Costs		21,220
EPCM, Commissioning & Start-up Costs		25,521

Table 21.1.2_1 Capital Cost Breakdown		
Item	**BRL '000s**	**US$ '000s**
EIS, Licences & Permits		4,046
Freight, Shipping & Insurance		7,572
Capital Spares Parts		3,259
Taxes, Import Duties		17,806
Land Acquisition		6,000
Owner Cost		11,971
Contingency		50,000
TOTAL		**426,738**

21.2 Operating Costs (OPEX)

21.2.1 Mine Facility Operating Costs

21.2.1.1 Unit Cost Assumptions

Mine operating cost was estimated using Activity Based Costing. This method is based on estimating the resources (Fuel, Labor, etc.) required for each activity (Drilling, Blasting, etc.) and applying unitary cost for each resource to calculate each activities cost. Indirect costs such Administrative, Supervisory and Technical overheads where also accounted for.

21.2.1.1.1 Equipment Operating Costs

Table 21.2.1.1.1_1 shows unit hourly costs for each equipment. Maintenance parts costs were taken from quotations, fuel consumption rate and other costs were provided by the equipment suppliers.

Table 21.2.1.1.1_1 Equipment Hourly Costs					
Equipment	Maintenance	Diesel	Lube	Others	Total(*)
	US$/h	US$/h	US$/h	US$/h	US$/h
Production Drill Rig	10.0	54.2	1.2	170.9	236.3
Excavator	11.2	58.2	0.7	13.2	83.3
Front End Loader	5.8	18.1	0.5	3.3	27.7
Haul Truck	2.6	14.1	1.1	2.4	20.2
Bulldozer	18.0	45.2	2.4	14.8	80.4
Motor grader	40.2	28.3	1.2	7.5	77.3
Water Truck	2.6	14.1	1.1	1.6	19.4
Fuel Truck	2.6	14.1	1.1	1.6	19.3
Rock Breaker	7.9	10.2	1.0	1.6	20.8
Lube Truck	2.6	14.1	1.1	1.6	19.3
Lowboy Truck	2.6	14.1	1.1	1.6	19.3
Pickup	0.2	5.0	0.3	0.2	5.7
Exploration Drill Rig	16.0	68.3	8.6	16.5	109.4
Lightning Tower	0.2	1.5			1.7

(*) Excludes labor

21.2.1.1.2 Consumables

Diesel fuel cost was provided by MBAC from suppliers quotations and amounts to 1.004 US$/l

Blasting costs are based on an explosives price of 1.69 USD/Kg and accessories that amount to approximately 10% of the explosives costs. These costs were taken from suppliers quotations.

Tires and wear elements costs are based on a database of similar projects.

21.2.1.1.3 Labor Costs

The personnel costs used for this project were provided by MBAC personnel and correspond its actual costs in other operations.

Table 21.2.1.1.3__1 and Table 21.2.1.1.3_2 show the company labour cost per job category. The company cost includes salaries as well as legal obligations. Benefits, perks and other related costs such as recruiting and training are not included as part of the labour costs.

Table 21.2.1.1.3_1
Yearly Salary – Hourly Personnel

Equipment Operators	**US$/y**
Front End Loader	19,000
Excavator	25,000
Haul Truck	19,000
Bulldozer	19,000
Motor Grader	19,000
Water truck	19,000
Production Drill	19,000
Rockbreaker	19,000
Fuel Truck	19,000
Maintenance Truck	19,000
Lowboy Truck	19,000
Exploration Drill	19,000
Blasting	
Blaster	53,000
Blaster Assistant	13,000
Other Operators	
Operations Assistant	13,000
Dispatch Operator	13,000
Maintenance Labor	
Vehicle Lubricator I	18,000
Vehicle Lubricator II	23,000
Vehicle Lubricator III	27,000
Vehicle Maintenance Mechanic I	14,000
Vehicle Maintenance Mechanic II	23,000
Vehicle Maintenance Mechanic III	27,000
Vehicle Maintenance Welder	25,000
Vehicle Electrician II	23,000

Table 21.2.1.1.3_2 Administrative and Technical Staff	
Position	**US$/y**
Management	
Mine Manager	206,000
Administrative Assistant	16,000
Trainee	6,000
Operations	
Mine Coordinator	127,000
Mining Engineer	94,000
Mine Technician	35,000
Mine Supervisor	61,000
Production Assistant	13,000
Production Foreman	16,000
Maintenance	
Vehicle Machine Shop Coordinator	143,000
Vehicle Maintenance Coordinator	71,000
Mine Equipment Maintenance Inspector	34,000
Technical Services	
Jr Topography Leader	19,000
Topographer	11,000
Geology and Mine Planning Coordinator	103,000
Geologist	79,000
Geology Technician	53,000
Topography Assistant	13,000

21.2.2 Mining Operating Cost

Based on the detailed estimates for the required operating hours by kind of equipment and estimated hourly costs, total costs have been calculated for each unit activity.

Table 21.2.2_1 and Table 21.2.2_2 show the estimated operating expenditures.

Given that the ore is close to the surface, only 1.6 Mt of pre-stripping is required. Operational expenses during the prestripping period amount to US$ 4.21 million or 2.59 USD/t moved, these are counted as part of the capital expenditures

Mine operating cost during commercial production is US$ 294.30 million or US$ 1.57/t moved.

Total operational expenditures amount to US$ 298.51 million or US$ 1.57/t moved.

Table 21.2.2_1
Operational Expenses

Item	Units	PS	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Total Movement	kt	1,625	3,383	3,300	4,100	4,100	4,635	4,600	4,600	4,648	4,623	4,600	4,609	6,000	6,010	6,000	6,000	6,027
Loading	‘000 US$	398	793	790	924	924	1,121	1,115	1,121	1,114	1,110	1,107	1,109	1,472	1,475	1,473	1,474	1,476
Hauling	‘000 US$	544	1,207	1,321	1,432	1,492	1,738	1,727	1,554	1,674	1,701	1,704	1,594	1,945	1,893	1,855	2,295	2,361
Drilling	‘000 US$	30	66	102	106	113	131	124	93	94	110	112	84	95	97	73	87	95
Blasting	‘000 US$	1	9	53	58	68	89	82	42	43	64	66	31	44	48	18	35	45
Ancillary	‘000 US$	660	1,825	1,876	2,014	2,007	1,988	2,132	2,477	2,643	2,683	2,540	1,753	2,255	2,342	2,218	2,214	2,227
Support	‘000 US$	697	1,081	1,081	1,081	1,081	1,150	1,177	1,171	1,213	1,186	1,161	1,146	1,194	1,198	1,195	1,243	1,192
Eng&Adm	‘000 US$	1,402	1,860	1,876	1,881	1,886	1,918	1,919	1,936	1,941	1,944	1,947	1,928	1,977	2,011	2,008	2,016	2,021
Pit Dewatering	‘000 US$	2	7	10	15	20	37	38	41	46	49	52	47	68	102	99	92	97
Mining Roads	‘000 US$	472	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	‘000 US$	4,205	6,848	7,107	7,511	7,590	8,172	8,314	8,434	8,767	8,847	8,690	7,692	9,051	9,166	8,940	9,456	9,514
Loading	US$/t mov.	0.24	0.23	0.24	0.23	0.23	0.24	0.24	0.24	0.24	0.24	0.24	0.24	0.25	0.25	0.25	0.25	0.24
Hauling	US$/t mov.	0.33	0.36	0.40	0.35	0.36	0.38	0.38	0.34	0.36	0.37	0.37	0.35	0.32	0.31	0.31	0.38	0.39
Drilling	US$/t mov.	0.02	0.02	0.03	0.03	0.03	0.03	0.03	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.01	0.01	0.02
Blasting	US$/t mov.	0.00	0.00	0.02	0.01	0.02	0.02	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.00	0.01	0.01
Ancillary	US$/t mov.	0.41	0.54	0.57	0.49	0.49	0.43	0.46	0.54	0.57	0.58	0.55	0.38	0.38	0.39	0.37	0.37	0.37
Support	US$/t mov.	0.43	0.32	0.33	0.26	0.26	0.25	0.26	0.25	0.26	0.26	0.25	0.25	0.20	0.20	0.20	0.21	0.20
Eng&Adm	US$/t mov.	0.86	0.55	0.57	0.46	0.46	0.41	0.42	0.42	0.42	0.42	0.42	0.42	0.33	0.33	0.33	0.34	0.34
Pit Dewatering	US$/t mov.	0.00	0.00	0.00	0.00	0.00	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.02	0.02	0.02	0.02
Mining Roads	US$/t mov.	0.29	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total	US$/t mov.	2.59	2.02	2.15	1.83	1.85	1.76	1.81	1.83	1.89	1.91	1.89	1.67	1.51	1.53	1.49	1.58	1.58

Table 21.2.2_2 Operational Expenses - Continued																	
Item	**Units**	**Y17**	**Y18**	**Y19**	**Y20**	**Y21**	**Y22**	**Y23**	**Y24**	**Y25**	**Y26**	**Y27**	**Y28**	**Y29**	**Y30**	**Y31**	**Y32**	**Total**
Total Movement	kt	6,011	6,000	6,019	6,000	6,014	6,029	6,005	9,000	9,057	9,000	9,039	9,000	9,023	7,275	4,471	2,754	189,557
Loading	'000 US$	1,480	1,462	1,438	1,467	1,481	1,461	1,449	1,907	1,915	1,909	1,936	1,939	1,921	1,601	1,076	751	44,188
Hauling	'000 US$	2,374	2,367	2,318	2,400	2,417	2,423	2,414	2,768	2,736	3,090	3,018	3,087	2,878	2,809	1,743	1,064	67,944
Drilling	'000 US$	85	104	84	148	87	130	154	211	145	72	104	85	108	118	62	71	3,379
Blasting	'000 US$	32	56	31	111	35	88	118	189	107	16	56	32	62	73	3	15	1,824
Ancillary	'000 US$	2,227	2,321	2,455	2,211	2,063	2,258	2,281	2,548	2,674	2,411	2,297	2,318	2,330	2,287	1,892	2,042	72,468
Support	'000 US$	1,192	1,237	1,237	1,237	1,261	1,261	1,258	1,254	1,299	1,299	1,258	1,261	1,261	1,261	1,105	1,110	39,030
Eng&Adm	'000 US$	2,027	2,059	2,046	2,078	2,054	2,068	2,066	2,101	2,131	2,135	2,121	2,121	2,120	2,138	2,071	2,076	65,884
Pit Dewatering	'000 US$	103	135	122	154	130	144	142	148	179	168	168	168	168	186	177	205	3,318
Mining Roads	'000 US$	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	472
Total	'000 US$	9,519	9,741	9,730	9,806	9,529	9,832	9,881	11,126	11,186	11,100	10,958	11,011	10,848	10,472	8,128	7,332	298,507

Loading	US$/t mov.	0.25	0.24	0.24	0.24	0.25	0.24	0.24	0.21	0.21	0.21	0.21	0.22	0.21	0.22	0.24	0.27	0.23
Hauling	US$/t mov.	0.39	0.39	0.39	0.40	0.40	0.40	0.40	0.31	0.30	0.34	0.33	0.34	0.32	0.39	0.39	0.39	0.36
Drilling	US$/t mov.	0.01	0.02	0.01	0.02	0.01	0.02	0.03	0.02	0.02	0.01	0.01	0.01	0.01	0.02	0.01	0.03	0.02
Blasting	US$/t mov.	0.01	0.01	0.01	0.02	0.01	0.01	0.02	0.02	0.01	0.00	0.01	0.00	0.01	0.01	0.00	0.01	0.01
Ancillary	US$/t mov.	0.37	0.39	0.41	0.37	0.34	0.37	0.38	0.28	0.30	0.27	0.25	0.26	0.26	0.31	0.42	0.74	0.38
Support	US$/t mov.	0.20	0.21	0.21	0.21	0.21	0.21	0.21	0.14	0.14	0.14	0.14	0.14	0.14	0.17	0.25	0.40	0.21
Eng&Adm	US$/t mov.	0.34	0.34	0.34	0.35	0.34	0.34	0.34	0.23	0.24	0.24	0.23	0.24	0.24	0.29	0.46	0.75	0.35
Pit Dewatering	US$/t mov.	0.02	0.02	0.02	0.03	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.03	0.04	0.07	0.02
Mining Roads	US$/t mov.	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total	US$/t mov.	1.58	1.62	1.62	1.63	1.58	1.63	1.65	1.24	1.24	1.23	1.21	1.22	1.20	1.44	1.82	2.66	1.57

21.2.3 Industrial Facility Operating Costs

OPEX for Santana Phosphate Project have been organized into five categories:

- Labor
- Electric power
- Reagents and consumables
- Maintenance
- Miscellaneous

Annual costs have been estimated as of the project operation commencement except for maintenance since spare parts and first fill for the first year were included in the CAPEX.

The accuracy assumed for the OPEX is ±10%. A contingency of 5% was also included in the estimates.

Estimating methodology

The following summarizes the methods used for the Industrial Facility OPEX.

Labor

Labor relates to personnel including position, manpower quantity, salaries and social charges for the plant as defined by MBAC. The direct labor, indirect labor and administrative personnel were considered as a single group.

Direct labor includes beneficiation, fertilizer, granulation, acidulation and sulphuric acid plants and utilities. Indirect labor includes laboratory, plant maintenance, asset security, supporting services, internal transportation and logistics personnel. Finally, the administrative group relates to all administrative area personnel, management position level and higher.

Electricity

Electrical power consumption was calculated in kWh/a across the mine and industrial sites by applying utilization and efficiency factors to the installed power for each motor and multiplying by the number of hours in a year. Power costs were calculated by multiplying the calculated power consumption of each motor by the electricity rate.

Initial discussions between MBAC and GRUPO REDE were held in order to develop a project which will contemplate to set up a new 138 KV line coming from Miracema (TO) up to Santana do Araguaia (215 km) and then, continuing up to the Project Site (184 km). See Figure 18.6.1_2.

At the mine site the project will install a new 138 KV line coming from Santana do Araguaia. The electrical power supply cost is estimated in US$ 130/kWh

The electric power consumption in the mine and industrial facilities is estimated to be 10 -12 MWh. The Sulfuric Acid Plant has a cogeneration capacity of 7.5 MWh

Reagents and Consumables

Consumption of reagents and consumables are based on consumption rates applied to the mass balance outflow. Prices for reagents have been based on in-house data information.

Table 21.2.3_1 shows the reagents consumption and supplier.

Table 21.2.3_1
Reagent Consumption and Vendor

Reagent	Unit	Consumption ku/year	Unit Price USD/unit	Vendor
Collector (Soybean oil)	kg	795	1.60	Miracema Nuodex Indústria Química
Sodium silicate	kg	696	0.98	Unaprosil / Diatom
Caustic soda100%	kg	951	0.84	Base Quimica / Brenntag / Rosario Quimica.
Flocculant	kg	18	5.48	Base Quimica / Rosario Quimica / SNF
Hydrated lime	kg	5,000	0.417	Cal Araguaia / Consplec / Base Quimica.
Limestone	kg	25.5	0.02	estimated
Additives (Dedust oil)	kg	1,325	1.75	Arr-Maz do Brasil Ltda.

Consumables for this project are listed in Table 21.2.3_2. Annual consumption of mill grinding media and linings were estimated from rates based on previous experience in similar projects. Firewood consumption includes firewood for the granulate superphosphate drying.

Table 21.2.3_2
Consumables Summary

Consumable	Unit	Consumption ku/year	Unit Price USD/unit
Grinding media/linings	kg	303.70	1.64
Firewood (Granulation drying)	tonne	55	83

Raw materials are summarized in Table 21.2.3_3.

Table 21.2.3_3 Raw Material Summary				
Raw Material	**Unit**	**Consumption ku/year**	**Unit Price USD/unit (1)**	**Rate source**
Sulphur	t	74.84	225.00	Based on an independent authority in fertilizer industry

Maintenance

Maintenance costs were budgeted from plants costs based on experience derived from other similar projects. Maintenance costs were defined by applying a CAPEX factor upon the total costs. Maintenance includes spare parts and lubricants

Table 21.2.3_4 Maintenance cost factor per area (% CAPEX).	
Description	**Factor (%)**
Infra-structure	1
Beneficiation	1
Shipment	1
Acidulation	1
Granulation	1
Sulphuric Acid Plant	1
Utility System	1

Miscellaneous

Miscellaneous costs are those related to specialized outsourced work and other activities such as community relationship, insurance, building maintenance, etc. Total of miscellaneous annual costs estimates is US$ 3.43 million.

21.2.4 Extra Operating Costs

It was considered the payment to CFEM (Brazilian Federal Contribution of Mineral Exploration) that represents 2% of the total cost of the Concentrate and a payment of royalties of 1% also of the total cost of the Concentrate.

21.2.5 Total SSP Annual Cost

Based on all operating costs assumptions, Project Santana SSP annual cost at site is US$113.00/tonne. Table below shows SSP cost breakdown. A 5% contingency was included in these estimates.

Table 21.2.5_1 SSP Unit Cost per Process ($/tonne)	
Process	**USD**
Mine	13.20
Beneficiation	28.30
Sulphuric Acid	34.80
Acidulation	11.40
Granulation	25.30
TOTAL	**113.00**

Note: operating cost per tonne SSP in 2017, first year of full production.

22 ECONOMIC ANALYSIS

An Economic Analysis was developed based on the information provided by AMSL, NCL, and PegasusTSI related to mineral resources, mining methods and engineering work, respectively. The Economic Analysis calculations were based on the Capital and Operating Cost information mentioned in Section 21. The following sections summarize the basis and results of the Economic Analysis.

22.1 SSP Long term price

The SSP price was based on the 2013 price provided by Agroconsult. This price forecast is FOB Rondonópolis, a fertilizer hub distribution center in the Mato Grosso state, plus adjustments for logistics from the Project to the target region. All prices were inflated at 2% per annum.

The Table 22.1_1 shows the prices from 2013 to 2047 that were contemplated to the project economic analysis.

Table 22.1_1

SSP Prices (US$/tonne)

2013	2014	2015	2016	2017	2018	2019
$325	$332	$338	$345	$352	$359	$366

2020	2021	2022	2023	2024	2025	2026
$373	$381	$388	$396	$404	$412	$420

2027	2028	2029	2030	2031	2032	2033
$429	$437	$446	$455	$464	$473	$483

2034	2035	2036	2037	2038	2039	2040
$493	$502	$512	$523	$533	$544	$555

2041	2042	2043	2044	2045	2046	2047
$566	$577	$589	$600	$612	$625	$637

22.2 Valuation model

22.2.1 Assumptions

The FS indicates that the Project is expected to generate robust returns. The assumptions for the economic analysis are as follows:

<u>Capital Cost</u>

The estimated capital cost is US$ 426.7 million including US$ 50 million of contingencies.

Capital Expenditure will be spent through the years 2013, 2014, 2015 and 2016 in a ratio of 2%, 28%, 45%, and 25% respectively.

<u>Sustaining Capital</u>

The total sustaining capital is US$229 million, distributed as follow:

- Environmental Compensation: US$2.5 million expended one time at the first year of operation (2016)
- Mine equipment: US$37.8 million, starting in 2015 with funds expended during the whole project life.
- Tailing dam: US$21 million, starting in 2019 and expended at a rate of US$2.1 million every 3 years.
- Industrial site: US$148 million, starting in 2017 and expended at a rate of US$ 2 million each year between 2017 and 2020; at a rate of US$ 4 million each year between 2021 and 2025; and at a rate of US$ 6 million each year between 2026 and 2045.
- Closure cost: US$20 million, expended in the last 2 years of the operational life, (US$10 million per year)

<u>Operating Costs</u>

Operating Costs include labor, electricity, consumables, fuel and lubricant, equipment maintenance, management and administrative expenditures, and all others costs contemplated in section 21.2.

<u>Exchange Rates</u>

The Company has used forward USD/BRL exchange rate curve in the Santana Definitive Feasibility Study to accurately estimate the economics of the project over the life of mine. A forward rate curve is calculated by applying an arbitrage-free mathematical formula to the spot rate curve. A forward rate represents the yield for a certain period, starting at a certain point in the future, and is used to forecast future variable cash flows. In other words, the forward curve is the best estimate of the forward rate of the USD/BRL based on the current market rates. The following forward rates were obtained from Bloomberg used in the study: – Bloomberg forward rates is a compilation (average) of the USD/BRL forward curve provided by all the major financial institutions in the world:

Table 22.2.1_1 Bloomberg Forward Rates

Year	1 USD = BRL
2013	2.39
2014	2.62
2015	2.86
2016	3.08
2017 – 2047	3.25

Depreciation

All Investments, except land, were depreciated with no residual value at a rate of 10% in ten years, (Straight Line Method).

Working Capital

Working Capital was estimated using average number of days of receivables, payables and stocks, as shown below:

- Receivables : 30 days
- Payables : 30 days
- Stocks : 50 days

Initial working Capital of US$9 million.

22.2.2 Economics

Project Life

Project life is 32 years after ramp-up, from 2016 to 2047.

Economic Evaluation

The Economic Evaluation of the Project was done using the Discounted Cash Flow Approach.

Table 22.2.2_1 shows the results of the discount cash flow.

Sales

Sales were calculated net of taxes.

Taxes

Taxes on Profits calculated according to Brazilian Tax Law.

- Income Tax: 15% plus 10% on the excess of R$ 240,000.00
- Social Contribution: 9%
- Not considered the losses (NOLs) carry-forward (Tax Benefit)

A reduction of 75% in the Income Tax was considered according to fiscal incentives (SUDAM area) during the period between 2017 and 2026.

Economic Indicators

Based on project assumptions preliminary financial model indicates robust project economics with a Net Present Value of US$ 396 million as at 2013 (the estimated start of construction).

The following economic indicators were used in the financial analysis:

- Weighted Average Cost of Capital (WACC): 10%
- Internal Rate of Return (IRR): 19.9%
- Net Present Value @ WACC: US$ 396 million
- Payback Period: 5 years

Table 22.2.2_1

Economic Evaluation - Discounted Cash Flow (MMUS$) & Economic Indicators

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
	2013	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**
REVENUE																		
(+) GSSP	0.0	0.0	0.0	82.8	175.9	179.4	183.0	186.7	190.4	194.2	198.1	202.0	206.1	210.2	214.4	218.7	223.1	227.5
(+) Sulphuric Acid	0.0	0.0	0.0	2.5	5.2	5.3	5.4	5.6	5.6	5.7	5.8	5.9	6.1	6.2	6.3	6.4	6.6	6.7
(+) Electricity	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
(-) Revenue discount (allowance)	0.0	0.0	0.0	(1.7)	(3.6)	(3.7)	(3.8)	(3.8)	(3.9)	(4.0)	(4.1)	(4.2)	(4.2)	(4.3)	(4.4)	(4.5)	(4.6)	(4.7)
(+) Net Capex ICMS recovery	0.0	0.0	0.0	1.0	2.1	2.1	2.1	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL REVENUE	**0.0**	**0.0**	**0.0**	**84.6**	**179.6**	**183.1**	**186.7**	**189.5**	**192.1**	**195.9**	**199.8**	**203.8**	**207.9**	**212.1**	**216.3**	**220.6**	**225.0**	**229.5**
OPERATING COST and SG&A																		
(-) Mine & Beneficiation	0.0	0.0	0.0	(16.4)	(20.7)	(21.6)	(22.1)	(23.1)	(23.7)	(24.3)	(25.2)	(25.8)	(26.1)	(25.5)	(27.6)	(28.3)	(28.6)	(29.9)
(-) Industrial GSSP	0.0	0.0	0.0	(21.6)	(38.3)	(38.9)	(39.7)	(40.8)	(41.3)	(42.1)	(42.9)	(43.8)	(44.7)	(45.6)	(46.5)	(47.4)	(48.4)	(49.3)
TOTAL COST	**0.0**	**0.0**	**0.0**	**(38.1)**	**(59.1)**	**(60.4)**	**(61.8)**	**(63.9)**	**(65.0)**	**(66.4)**	**(68.1)**	**(69.6)**	**(70.8)**	**(71.0)**	**(74.1)**	**(75.7)**	**(77.0)**	**(79.2)**
EBITDA	**0.0**	**0.0**	**0.0**	**46.5**	**120.5**	**122.6**	**124.9**	**125.5**	**127.1**	**129.5**	**131.7**	**134.2**	**137.1**	**141.0**	**142.2**	**144.9**	**148.1**	**150.4**
(-) Depreciation	(1.0)	(14.5)	(35.7)	(47.7)	(47.8)	(48.0)	(48.4)	(48.9)	(49.3)	(50.0)	(49.6)	(36.7)	(16.2)	(5.1)	(5.7)	(6.7)	(7.2)	(7.6)
EBIT	**(1.0)**	**(14.5)**	**(35.7)**	**(1.2)**	**72.7**	**74.6**	**76.5**	**76.6**	**77.7**	**79.4**	**82.1**	**97.5**	**120.9**	**135.9**	**136.5**	**138.2**	**140.9**	**142.8**
(-)Taxes	0.0	0.0	0.0	0.0	0.0	(5.6)	(10.6)	(10.6)	(10.7)	(10.9)	(11.3)	(13.7)	(17.2)	(19.5)	(45.1)	(45.7)	(46.6)	(47.2)
(-) Capital Expenditure	(9.9)	(135.0)	(212.1)	(119.6)	(1.9)	(1.9)	(4.0)	(4.7)	(4.2)	(7.0)	(5.5)	(6.5)	(7.0)	(8.6)	(7.7)	(11.8)	(8.6)	(9.0)
(+/-) Change Working Capital	0.0	0.0	0.0	(9.0)	(9.0)	(0.4)	(0.4)	(0.3)	(0.3)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.5)	(0.4)	(0.4)	(0.5)
(+) Depreciation	1.0	14.5	35.7	47.7	47.8	48.0	48.4	48.9	49.3	50.0	49.6	36.7	16.2	5.1	5.7	6.7	7.2	7.6
CASH FLOW	**(9.9)**	**(135.0)**	**(212.1)**	**(82.1)**	**109.7**	**114.8**	**110.0**	**109.9**	**111.9**	**111.1**	**114.4**	**113.6**	**112.5**	**112.6**	**88.9**	**87.0**	**92.5**	**93.7**

	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35
	2031	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**	**2039**	**2040**	**2041**	**2042**	**2043**	**2044**	**2045**	**2046**	**2047**
REVENUE																	
(+) GSSP	232.1	236.7	241.5	246.3	251.2	256.2	261.4	266.6	271.9	277.4	282.9	288.6	294.3	300.2	306.2	239.6	244.4
(+) Sulphuric Acid	6.8	7.0	7.1	7.2	7.4	7.5	7.7	7.8	8.0	8.2	8.3	8.5	8.7	8.8	9.0	7.0	7.2
(+) Electricity	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
(-) Revenue discount (allowance)	(4.8)	(4.9)	(5.0)	(5.1)	(5.2)	(5.3)	(5.4)	(5.5)	(5.6)	(5.7)	(5.8)	(5.9)	(6.1)	(6.2)	(6.3)	(4.9)	(5.0)
(+) Net Capex ICMS recovery	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL REVENUE	**234.1**	**238.8**	**243.6**	**248.5**	**253.4**	**258.5**	**263.7**	**268.9**	**274.3**	**279.8**	**285.4**	**291.1**	**296.9**	**302.9**	**308.9**	**241.7**	**246.5**
OPERATING COST and SG&A																	
(-) Mine & Beneficiation	(30.5)	(31.1)	(32.1)	(32.7)	(33.5)	(33.7)	(34.9)	(35.6)	(38.3)	(39.1)	(39.8)	(40.3)	(41.2)	(41.8)	(42.0)	(35.5)	(34.8)
(-) Industrial GSSP	(50.3)	(51.3)	(52.3)	(53.4)	(54.5)	(55.5)	(56.7)	(57.8)	(58.9)	(60.1)	(61.3)	(62.5)	(63.8)	(65.1)	(66.4)	(54.6)	(55.7)
TOTAL COST	**(80.8)**	**(82.5)**	**(84.4)**	**(86.1)**	**(87.9)**	**(89.3)**	**(91.5)**	**(93.4)**	**(97.2)**	**(99.2)**	**(101.1)**	**(102.9)**	**(105.0)**	**(106.8)**	**(108.3)**	**(90.1)**	**(90.4)**
EBITDA	**153.3**	**156.4**	**159.2**	**162.4**	**165.5**	**169.2**	**172.2**	**175.5**	**177.1**	**180.6**	**184.3**	**188.2**	**191.9**	**196.0**	**200.6**	**151.6**	**156.1**
(-) Depreciation	(8.2)	(8.7)	(8.9)	(9.4)	(9.8)	(10.1)	(10.5)	(10.4)	(10.6)	(11.3)	(12.3)	(11.1)	(11.9)	(12.1)	(12.0)	(12.9)	(11.7)
EBIT	**145.1**	**147.6**	**150.2**	**153.0**	**155.7**	**159.1**	**161.6**	**165.2**	**166.6**	**169.3**	**172.1**	**177.1**	**180.1**	**183.9**	**188.6**	**138.7**	**144.4**
(-)Taxes	(47.9)	(48.8)	(49.6)	(50.5)	(51.4)	(52.6)	(53.4)	(54.6)	(55.0)	(55.9)	(56.8)	(58.5)	(59.5)	(60.7)	(62.3)	(45.7)	(47.6)
(-) Capital Expenditure	(10.4)	(12.1)	(7.7)	(11.3)	(11.2)	(11.3)	(12.0)	(10.1)	(10.5)	(16.0)	(10.0)	(11.2)	(15.0)	(14.0)	(10.2)	(19.9)	(16.8)
(+/-) Change Working Capital	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.5)	(0.7)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	6.5	(0.4)
(+) Depreciation	8.2	8.7	8.9	9.4	9.8	10.1	10.5	10.4	10.6	11.3	12.3	11.1	11.9	12.1	12.0	12.9	11.7
CASH FLOW	**94.5**	**95.0**	**101.4**	**100.1**	**102.4**	**104.9**	**106.2**	**110.3**	**111.0**	**108.2**	**117.0**	**118.0**	**116.9**	**120.7**	**127.6**	**92.5**	**91.3**

WACC	10%	
NPV	**$396 MMUS$**	
IRR	**19.9%**	
Payback Period	**5.0 Years**	(from starting of operation in 2016)

22.2.3 Sensitivity Analysis

Risk Analysis was done using the sensitivity of relevant inputs creating scenarios.

The Sensitivity Analysis showed low risk. All scenarios in the sensitivity analysis were feasible as shown in the tables below (in US$ million):

Table 22.2.3_1

NPV at 2013 Sensitivity Analysis by SSP and Sulphur prices

NPV MMUSD	**SSP Price (% Var)**				
Sulphur Price CIF Santana (% Var)	-20%	-10%	0%	10%	20%
-20%	184	302	419	535	651
-10%	172	290	408	524	639
0%	160	279	396	512	628
10%	149	267	384	500	616
20%	137	255	372	489	605

Table 22.2.3_2

NPV at 2013 Sensitivity Analysis by SSP Price and Total CAPEX

NPV MMUSD	**SSP Price (% Var)**				
Total CAPEX (% Var)	-20%	-10%	0%	10%	20%
-20%	238	356	472	588	703
-10%	199	317	434	550	665
0%	160	279	396	512	628
10%	121	239	357	474	590
20%	82	200	318	435	552

Table 22.2.3_3

NPV at 2013 Sensitivity Analysis by SSP Price and Total OPEX

NPV MMUSD	**SSP Price (% Var)**				
Total OPEX (% Var)	-20%	-10%	0%	10%	20%
-20%	246	364	480	596	711
-10%	203	321	438	554	669
0%	160	279	396	512	628
10%	117	235	353	470	586
20%	74	193	311	428	544

Table 22.2.3_4

NPV at 2013 Sensitivity Analysis by SSP Price and WACC

NPV MMUSD	**SSP Price (% Var)**				
WACC (%)	-20%	-10%	0%	10%	20%
8%	286	435	583	730	877
9%	218	350	481	611	741
10%	160	279	396	512	628
11%	112	218	324	428	532
12%	72	168	263	357	450

23 ADJACENT PROPERTIES

There are no adjacent or nearby phosphate permits to those of MBAC.

The Santana Phosphate Project is related to the east extension of the regional 450km long northwest-southeast Cuiú-Cuiú - Tocantinzinho lineament which also hosts several important gold deposits including the Palito mine, Tocantinzinho deposit and Cuiú-Cuiú, Bom Jardim and Batalha gold prospects.

24 OTHER RELEVANT DATA AND INFORMATION

AMSL/NCL/PegasusTSI is not aware of other relevant data pertaining to the Santana Phosphate Project.

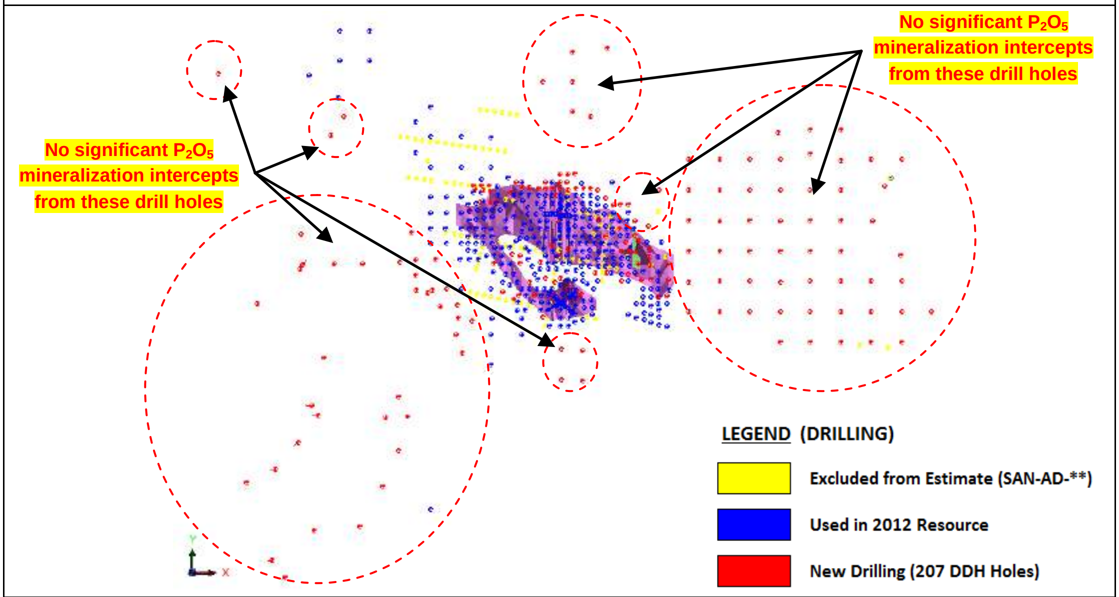
25 INTERPRETATION AND CONCLUSIONS

MBAC has undertaken a systematic exploration program in the last year that has been successful in defining significant resources of phosphate in close proximity to one of the largest agricultural centres in Brazil.

AMS is of the opinion that MBAC has successfully confirmed the mineral resource potential of the Santana Phosphate Project based on the 2011 and 2012 exploration programs. There remains significant further upside for addition of Measured and Indicated resource through well planned infill drilling programs. The drilling has defined an Indicated and Inferred mineral resource which is currently subject to a FS following on from the positive economic results in the PFS from the AMSL/NCL/PegasusTSI completed in June 7, 2012, as amended August 27, 2012.

Future exploration drilling programs across the Santana concessions should be carefully planned, given the quantity of drilling completed as part of the 2012 drilling campaign which failed to intercept any significant mineralization (Figure 25_1).

Figure 25_1
2012 Diamond Drilling with No Significant P_2O_5 Mineralization Intercepts



Overall, AMS concludes that there are no fatal flaws in the current mineral resource estimate.

AMS considers the project to be sufficiently robust to warrant the undertaking of additional infill drilling (50m x 50m spacing) in an effort to augment the confidence level of the current mineral resource and provide a high grade portion of measured category resource the start of open pit mining.

Based on the project developed information presented in this FS Report a financial model indicates a robust project economics.

The pertinent observations and interpretations which have been developed in producing this report are detailed in the sections above.

26 RECOMMENDATIONS

26.1 Exploration and Resources

Drilling and studies completed to date have defined Indicated and Inferred mineral resource at Santana. The data collected is considered to be of moderate to high quality and suitable for resource estimation.

Further scope exists to improve the geological and mineral resource estimation confidence. AMSL makes the following specific recommendations:

- Increase the drill density of a 400m by 400m area to 50m by 50m spacing to allow measured resources to be defined. DC is the preferred method of sample recovery for this infill drilling program, however RC drilling is suitable if dry samples can be procurred.
- To utilize airborne radiometrics surveys as a first pass exploration tool to help guide regional exploration drilling programs.

A recent airborne radiometrics survey has highlighted the location of the Santana Phosphate Project quite effectively. Based upon a recent radiometrics survey completed across the mineralized domain(s) at Santana, AMSL would suggest the opportunity for lateral extensions to mineralization in the vicinity of the currently defined resource is limited.

Further step-out exploration drilling is not warranted at this stage of the project, and further exploration drilling should be focused on infill drilling portions of the existing resource to a 50 x 50m drill spacing to allow Measured resource category to be defined.

Infill drilling a portion of the existing resource to a 50m x 50m drill spacing was a key recommendation presented in the previous resource estimate completed by AMSL in April 2012, and in the more recent PFS completed by AMSL/NCL/PegasusTSI in June 7, 2012, as amended August 27, 2012.

26.2 Mineral resource and Evaluation Budget

MBAC has also provided AMSL with an ongoing exploration and evaluation budget, summarised in Table 26.3_1 below.

Table 26.3_1
Santana Phosphate Project
Proposed Resource and Evaluation Expenditure

Activity	**Total (US$)**
DC and RC Drilling	$ 1,000,000
Assaying and Characterization	$ 200,000
Geology	$ 50,000
Travel and Accommodation	$ 30,000
Field Supervision and Support	$ 150,000
Administration	$ 70,000
Sub-Total	**$1,500,000**

The proposed expenditure of US$ 1,500,000 is considered to be consistent with the potential of the Santana Phosphate Project and is adequate to cover the costs of the proposed programs.

26.3 Detail Design and Implementation Phase

PegasusTSI has worked closely with MBAC during the Santana Project Feasibility Study Phase in order to implement industry best-practices in fertilizer plant design as well as capturing some benefits based on the very recent design and construction experience from the Itafos SSP Project. In this way a more cost competitive product can be produced on a consistent basis. For example, a much more compact footprint for the Santana SSP production has been achieved, which permits a large reduction in the cost associated with construction materials like structural steel, concrete, wiring and belt conveyors, as well as the associated construction cost while making the facility more operable.

Based on an intimate familiarity with the Santana Project, PegasusTSI recommends that further engineering and procurement “lessons learned” activities from the Itafos project should be included as part of the detailed engineering and procurement implementation plan. Throughout this next pivotal phase of work this valuable source of experience from the Itafos Project will be used, including specific experience from the (Pre-) Commissioning and Start-Up/Operations Team.

It is therefore strongly recommended that prior to the commencement of the detail design phase of the Santana Project, a formal and detailed "lessons learned" sessions be conducted. The formal capture of these learnings, with input from all stakeholders, will identify value-added design, procurement, construction, commissioning and start-up issues from the Itafos project. These sessions will provide feedback for the new plant design that will usefully inform all engineering and design disciplines, as well providing insight for improved equipment and/or processing systems selection. Constructability and maintainability reviews, using the latest 3-D plant design models can also be conducted as part of the lessons learned sessions. Improvements will be documented and formalized in an extended design package to be cast into the new plant design.

The outcomes from the formal lessons learned sessions will then better inform the design basis. This will greatly reduce potential design changes during the detailed design phase, which will reduce exposure to cost and schedule overruns.

The contracting approach for the Santana project should also be revisited. It is recommended that MBAC retain an experienced Engineering, Procurement and Construction Management (EPCM) contractor to insure that irrespective of where the detailed engineering work is performed, or whether this work is performed on a fixed price or reimbursable commercial basis, it is executed in strict accordance with the design basis documents and execution philosophy. Managing the entire project under this mandate will provide greater assurance to MBAC, its investors and its stakeholders that the Santana Project will be delivered in strict accordance with the design basis, in the most cost effective manner possible and in accordance with the project schedule.

This commitment to meeting project schedule is especially important to MBAC's future SSP customers from the Santana project. Customers must be provided with solid assurance in advance of the project start-up that a quality SSP product will be made available as projected to meet the demands of their growing season, by implementing the above mitigation steps MBAC will be able to provide this delivery commitment with confidence.

27 REFERENCES

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28 DATE AND SIGNATURE PAGE

The “qualified persons” (within the meaning of NI43-101) for the purposes of this report are Bradley Ackroyd, who is an employee of Andes Mining Services Ltd.; Carlos Guzman, who is an employee of NCL Brasil Ltda.; and Robert Alexander, who is an employee of PegasusTSI Inc. The effective date of this report is 28 October 2013.

(signed by)
Bradley Ackroyd B.Sc Geol. MAIG
Regional Manager & Principal Consulting Geologist
Andes Mining Services Ltd.

Signed on the 28 October 2013

(signed by)
Carlos Guzmán RM Chilean Mining Commission / FAusIMM
Principal /Project Director
NCL Brasil Ltda.

Signed on the 28 October 2013

(signed by)
Robert Alexander PE, Florida, USA
Manager of Engineering
PegasusTSI. Inc

Signed on the 28 October 2013

29 CERTIFICATES OF QUALIFIED PERSONS

Andes Mining Services Limited.

Certificate of Qualified Person

I, Bradley Ackroyd, do hereby certify that:

1. I have been working since 2012 as a Principal Consulting Geologist with the firm Andes Mining Services Ltd. of Avenue Diagonal 550, Departmento 203, Miraflores, Lima, Peru 18. My residential address is Jose Pardo 1040, Miraflores, Lima, Peru 27.
2. I am a practising geologist with 11 years of Mining and Exploration geological experience. I have worked in Australia, PNG, West Africa and the Americas. I am a member of the Australian Institute of Geoscientists - Member (MAIG).
3. I am a graduate of the University of Western Australia (UWA) and hold a Bachelor of Science Degree in Geology (Hons) (2000).
4. I have practiced my profession continuously since 2001. I have 12 years of geological experience ranging from open pit and underground mine production, resource definition to grass roots exploration.
5. I am a “qualified person” as that term is defined in National Instrument 43-101 Standards of Disclosure for Mineral Projects (the “Instrument”).
6. I have visited the Santana Phosphate Project between the 20th and 23rd November 2012.
7. I am responsible for sections 4 to 12 and 14; and jointly responsible for sections 1 to 3 and 23 to 27 of the technical report dated effective 28th October 2013 and titled “Feasibility Study - Santana Phosphate Project, Pará State, Brazil” (the “Report”).
8. I am independent of MBAC Fertilizer Corp pursuant to section 1.5 of the Instrument.
9. I have read the Instrument and Form 43-101F1 (the “Form”) and the Report has been prepared in compliance with the Instrument and the Form.
10. I do not have nor do I expect to receive a direct or indirect interest in the Santana Phosphate Project of MBAC Fertilizer Corp and I do not beneficially own, directly or indirectly, any securities of MBAC Fertilizer Corp or any associate or affiliate of such company.
11. I have not had any prior involvement with the Santana Phosphate Project for MBAC Fertilizer Corp.
12. As of the effective date of the Report, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated in Lima, Peru, on the 28 October 2013.

(signed by)

Bradley Ackroyd BSc(Geo) Member (MAIG)
Principal Consulting Geologist

NCL Brasil Ltda.

Certificate of Qualified Person

I, Carlos Guzmán, do hereby certify that:

1. I have been working since 2001 as the Principal Mining Engineer and Project Director with the firm NCL Brasil Ltda. of Alameda da Serra 500/315, Nova Lima,-MG, Brazil, CEP 34000-000.
2. I am a practicing mining engineer, a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM, No. 229036); and a Registered Member of the Chilean Mining Commission.
3. I am a graduate of the Universidad de Chile and hold a Mining Engineer title (1995).
4. I have practiced my profession continuously since 1995 and acted as "qualified person" for several Technical Reports since 2007 for base metals, precious metals and fertilizers mining projects.
5. I am a “qualified person” as that term is defined in National Instrument 43-101 Standards of Disclosure for Mineral Projects (the “Instrument”).
6. I have visited the Santana Phosphate Project between the 25th and 27th July 2011.
7. I am responsible for sections 15 and 16, and jointly responsible of sections1 to 3, 21 and 23 to 27 of the technical report dated effective 28th October 2013 and titled “Feasibility Study - Santana Phosphate Project, Pará State, Brazil” (the “Report”).
8. I am independent of MBAC Fertilizer Corp pursuant to section 1.5 of the Instrument.
9. I have read the Instrument and Form 43-101F1 (the “Form”) and the Report has been prepared in compliance with the Instrument and the Form.
10. I do not have nor do I expect to receive a direct or indirect interest in the Santana Phosphate Project of MBAC Fertilizer Corp and I do not beneficially own, directly or indirectly, any securities of MBAC Fertilizer Corp or any associate or affiliate of such company.
11. I have had prior involvement with the Property, namely, I was a “qualified person” responsible for the preparation of portions of the technical report titled “Pre-Feasibility Study (PFS), Santana Phosphate Project, Pará State, Brazil, as Amended and Restated" Technical Report NI 43-101”, effective date June 7, 2012, as amended and restated August 27, 2012..
12. As of the effective date of the Report, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated at Belo Horizonte, Brazil, on 28 October, 2013

(signed by)

Carlos Guzmán RM, Chilean Mining Commission / FAusIMM
Principal Mining Engineer / Project Director

PegasusTSI, Inc.

Certificate of Qualified Person

I, Robert B. Alexander, do hereby certify that:

1. I am the Manager of Engineering at PegasusTSI, Inc., which is a worldwide engineering, procurement and construction management company specializing in chemical processing facilities, with a business address at 5310 Cypress Center Drive, Suite 200, Tampa, Florida 33609.
2. I hold a Bachelor's Degree in Mechanical Engineering from California Polytechnic State University (1971), and hold a Professional Engineers License in the State of Florida, USA (PE No. 44621)
3. I have practiced my profession continuously since 1971. I have an accumulation of approximately 24 years experience in phosphate fertilizer processing including beneficiation, acid production and granulation.
4. I am a Registered Professional Engineer in the State of Florida, Certificate No. 44621. I am also a member of the American Society of Mechanical Engineers.
5. I am a "qualified person" as that term is defined in National Instrument 43-101 Standards of Disclosure for Mineral Projects (the "Instrument").
6. I have visited the Santana Phosphate Project between the 29th and 31st of October 2012..
7. I am responsible for sections 13, 17 to 20 and 22; and jointly responsible of sections 1 to 3, 21 and 23 to 27 of the technical report dated effective 28th October 2013 and titled "Feasibility Study - Santana Phosphate Project, Pará State, Brazil" (the "Report").
8. I am independent of MBAC Fertilizer Corp pursuant to section 1.5 of the Instrument.
9. I have read the Instrument and Form 43-101F1 (the "Form") and the Report has been prepared in compliance with the Instrument and the Form.
10. I do not have nor do I expect to receive a direct or indirect interest in the Santana Phosphate Project of MBAC Fertilizer Corp and I do not beneficially own, directly or indirectly, any securities of MBAC Fertilizer Corp or any associate or affiliate of such company.
11. I have had prior involvement with the Property, namely, I was a "qualified person" responsible for the preparation of portions of the technical report titled "Pre-Feasibility Study (PFS), Santana Phosphate Project, Pará State, Brazil, as Amended and Restated" Technical Report NI 43-101", effective date June 7, 2012, as amended and restated August 27, 2012.
12. As of the effective date of the Report, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated at Tampa, Florida USA, on 28 October, 2013

(signed by)

Robert B. Alexander, P.E. (FL)
Manager of Engineering