Chariot Resources Limited

MARCONA COPPER PROPERTY
MINA JUSTA PROJECT

Definitive Feasibility Study
Technical Report NI 43-101

June 2009

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Title Page

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Qualified Persons:

- David D. (Dan) Greig, B.Sc, M.A.I.G. (1684-1722), employed by GRD Minproc Limited as Principal Geologist, was responsible for overall preparation of the report.
- Warwick Board, B.Sc (Hons.), M.Sc., Ph.D. Geology, P.Geo. (#31256), P.Geol. (#96031) Aus.I.M.M. (#212259), Pri.Sci.Nat. (#400009/03), employed by Snowden Mining Industry Consultants Inc. as Principal Consultant in the Resource Division, was responsible for the preparation of Sections 14 (Data Verification) and 17 (Mineral Resources), and for supervision of the preparation of Sections 7 through 13, and 15 of this report.
- Ross Oliver, B.Eng (Mining), Aus.I.M.M. 105137, employed by GRD Minproc Limited as Manager Mining & Geology, was responsible for the preparation of the whole or parts of sections 1 and 18.1, on Mineral Reserve and Mining.
- Adam Johnston, B.Eng (Extractive Metallurgy), employed as Chief Metallurgist by Transmin Metallurgical Consultants, was responsible for supervision of Metallurgical Testwork and interpretation of testwork results as reported in Section 16.
- Joe Schlitt, B.SC. (Met. Eng.), Ph.D., Qualified Professional member of the Mining and Metallurgical Society of America No. 01003QP and a Registered Professional Metallurgical Engineer (Texas No. 53603, President of Hydrometal, Inc., employed as a Metallurgical Consultant on behalf of GRD Minproc Limited, was responsible for Metallurgical Testwork and flowsheet design for the Mina Justa Project Oxide circuit (parts of Section 16 and Section 18).
- Dean David, M.Aus.I.M.M. (102351) Process Consultant for GRD Minproc Limited, was responsible for Mineral Processing and Metallurgical Testwork of Sulphide ores (Section 16.2), Sulphide plant design (Section 18.3) and for Plant Operating Costs (Section 18.13).
- Daniel Y. Yang, Senior Geotechnical Engineer, M.Eng., P.Eng. 28936, employed by Knight Piésold Ltd. was responsible for geotechnical studies, analysis and conclusions contained in Section 18 (specifically in 18.1.3, 18.2 and 18.7.1), relating to pit slope design criteria, waste dump design, geotechnical site investigations and sourcing of construction materials.

- Thomas F. Kerr, M.Sc., President of Knight Piésold and Co. USA, Registered Professional Engineer (Civil and Geotechnical), P.Eng., in British Columbia (#14906) and Ontario (#90407230) and P.E. in California (#C49260) and Alaska (#10969), was responsible for geotechnical matters related to the design and costing for the Tailings Storage Facility in Section 18.7.2.
- Branislav Grbovic, B.Sc, M.Sc, Mech Engineer, M.Aus.I.M.M. (302515), Senior Project & Design Manager, GRD Minproc Limited, was responsible for engineering aspects of the Project as contained in Section 18.
- Anthony Sanford, Manager of Environmental Services by Vector Peru, was responsible for environmental aspects described in Section 15

Other Expert contributors:

- John D. Kapusta, P.Geo, Marcobre's Vice-President Exploration and Geological Services is the Qualified Person responsible for Marcobre's exploration, drilling, sampling and data quality activities as described in Sections 7 to 13, and for Marcobre's geological modelling (Section 17).
- Klaus Meder, Dr. rer. nat., employed by Andes Resources Compañía Minera SAC, a 100% owned Peruvian subsidiary of Chariot Resources as Senior Geologist. Klaus Meder provided the electronic database information, assisted the lithological and grade shell model interpretation and was responsible for inputs for geological information as described in sections 6 to 15 and 17.
- David Brownrigg, B.Sc., B.Eng. (Mining), P.Eng. (#100051618) employed by Andes Resources Compañía Minera S.A.C, a 100% owned Peruvian subsidiary of Chariot Resources as General Manager and General Manager of Marcobre. In addition, a qualified person (QP) as defined by the guidelines within NI43-101. David Brownrigg was responsible for inputs for in-country information regarding tenement holdings, project logistics such as transport, shipping and manning, etc.
- Brent Cochrane, financial consultant to Marcobre, was responsible for the preparation of the financial model.
- Sean McCoy, employed by GRD Minproc as Senior Estimator, was responsible for preparation of the capital cost estimate reported in Section 18.
- Olimpio Angeles Girón employed by Knight Piésold Consultores S.A. as Senior Geotechnical Geologist was responsible for the geotechnical site investigation program reported in Section 18.2.
- Abel Ordóñez Huamán employed by Knight Piésold Consultores S.A. was responsible for the seismicity report in section 18.2.7.
- Kerry Lahti employed by Knight Piésold and Co. was responsible for the Climatological Study and overall site water balance contained in the section 18.5.1
- Victor Lishnevsky employed by Knight Piésold and Co., was responsible for the overall site water balance contained in the section 18.5.1
- Cynthia Parnow employed by Knight Piésold and Co. was responsible for the environmental waste characterization report.
- Robinson Ucanan, employed by GMI, was responsible for engineering aspects and estimates for Infrastructural parts of the project as contained in Section 18.

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1. SUMMARY

1.1 INTRODUCTION

The Marcona Copper Property is owned by Marcobre S.A.C. (Marcobre), a closed corporation with limited liability (i.e. sociedad anónima cerrada) existing under the laws of Peru. Marcobre was incorporated in Peru in 2004.

Chariot Resources Limited (Chariot) indirectly owns 70% of Marcobre, while Korea Resources Corporation (Korea Resources) and LS-Nikko Copper Inc. (LS-Nikko) indirectly own 30%. Chariot, Korea Resources, LS-Nikko, their respective shareholder entities, and Marcobre have entered into a shareholders agreement with respect to the corporate governance of Marcobre and the ownership, development and operation of the Marcona Copper Property.

The Marcona Copper Property is located approximately 400 km southeast of Lima in the province of Nazca, department of Ica, in the southern Peruvian coastal belt. The centre of the Marcona Copper Property lies approximately 25 km north of the coastal town of San Juan de Marcona (see Figure 4.1).

The Marcona Copper Property consists of a number of iron oxide-copper (silver-gold) deposits and prospects that are part of a very large iron oxide-rich hydrothermal system associated with the Marcona iron ore deposits located a few kilometres to the south and west.

The mineralisation to be exploited by the Mina Justa Project (Project) is contained in two deposits, the Main or Mina Justa deposit and the much smaller Magnetite Manto deposit, which are separated by 1 km of barren ground. The mineralisation is hosted by sedimentary and volcanic rocks of mid to late Jurassic age that have been hydrothermally altered and mineralised.

The Mina Justa and Magnetite Manto deposits are located on the TAI mining concession which covers approximately 3969 ha.

A definitive feasibility study (DFS), undertaken by GRD Minproc Limited (GRD Minproc), covers the proposed Project, which is designed initially to process 12 Mt/a of oxide ore by crushing, vat leaching, solvent extraction and electrowinning to produce up to 52 000 t/a of cathode copper. The facilities will be expanded during operating year 2 (quarter 7) to include a 5 Mt/a concentrator to treat copper sulphide ore underlying the oxide ore in certain portions of the Mina Justa deposit. The concentrator plant design, engineering and costing is developed to prefeasibility study (PFS) level.

The DFS covers all aspects of the Project, including the mine, process plant, metals production facility and associated infrastructure, as follows:

- Geology and resources
- Geotechnical studies
- Mining
- Metallurgy
- Process plant design

- General Infrastructure
- Water supply
- Power supply
- Process waste disposal
- Construction and accommodation camp
- Transport
- Mobile equipment
- Maintenance facilities
- Environmental and social impact assessment
- Capital costs
- Operating costs
- Project implementation
- Personnel, occupational health and safety, and training
- Marketing
- Financial analysis
- Risk assessment and risk management.

The currency used in this study is United States Dollars (US$). Currencies other than US$ are converted to US$ using the exchange rates set forth in Section 1.11.

Chariot is a Toronto Stock Exchange listed Canadian public company with its registered office at Suite 702, 55 University Ave., Toronto, Ontario, M5J 2H7 Canada.

1.2 GEOLOGY AND MINERAL RESOURCES

1.2.1 Geological setting

The Mina Justa Prospect forms part of Marcobre's Marcona Copper Project, located in the Coastal Belt of Peru. This northwest trending linear belt represents the westernmost part of the Central Andean Cordillera where the Nazca Plate subducts beneath the South American Plate, forming an active continental margin along the Peru-Chile Trench.

The geology of the Marcona-Mina Justa Iron-Copper District consists of a Precambrian high grade metamorphic basement (the Arequipa Massif), unconformably overlain by Neoproterozoic and Phanerozoic sedimentary rocks. Palaeozoic sediments (the Ordovician Marcona Formation) host the majority of the economic magnetite orebodies at the Marcona iron mine. Monzogranite, granodiorite and gabbro-diorite rocks of the post-kinematic San Nicolas batholith (dated at approximately 425 Ma) intrude the pre-Mesozoic rocks. The pre-Mesozoic rocks are unconformably overlain by a series of volcano-sedimentary and volcano-plutonic arc sequences that range in age from late Triassic to Holocene. The volcano-sedimentary rock sequences are intruded by porphyritic andesite dykes, sills

and plugs of the Tunga Andesite (also termed “ocoite”); and, in the eastern parts of the district, by granitoid plutons of the circa 109 Ma Coastal Batholith. Tertiary age shallow water marine sediments and Quaternary age marine terrace deposits unconformably overlie the volcano-plutonic arc succession.

The Mina Justa Prospect comprises two deposits, the Mina Justa and Magnetite Manto deposits, which are hosted by the Jurassic Upper Río Grande Formation, dominated by andesitic lavas and pyroclastics, intercalated with minor sandstone, siltstone and carbonate units. This volcano-sedimentary package displays a prolonged deformation history that includes a southeast verging overturned folding stage, followed by a shear faulting stage that generated curvilinear fault systems. The youngest deformation stage is normal block faulting along northwest trending structures that are closely associated with late stage emplacement of ocoite (porphyritic andesite) dykes.

Until recently it was believed the Mina Justa copper deposits and the adjacent Marcona iron mine were associated, as part of a large iron-rich hydrothermal system formed in an extensional environment along a subduction-related continental margin. Recent work suggests, however, that the Mina Justa Prospect is significantly younger (approximately 104-95 Ma) than, and geochemically distinct from, the Marcona Iron deposit (approximately 162-156 Ma). The Mina Justa Prospect is now interpreted as a hydrothermal deposit that was formed by the incursion of exotic and probably evaporite-sourced brines that were expelled from an adjacent sedimentary basin. The recent findings support the classification of the Mina Justa Prospect as an Iron Oxide Copper Gold (IOCG) deposit.

The Mina Justa deposit shares many mineralogical and textural characteristics with other major exocontact Andean Cu-rich IOCG deposits, e.g. Raúl-Condestable (Peru), Mantoverde and La Candelaria in the Punta del Cobre District (Chile).

1.2.2 Mineralisation and alteration

Massive, brecciated elongated magnetite (-pyrite) bodies host the highest-grade copper sulphide mineralisation at Mina Justa. The location of these bodies appears to be controlled primarily by a northeast striking and southeast dipping system of faults (the Mina Justa fault system). The mineralised bodies have, however, been dislocated by northwest striking and northeast dipping faults and by associated ocoite dykes, the latter ranging from less than a few metres to 70 m in thickness (typically 15 m to 30 m in thickness).

Seven stages of hydrothermal alteration and hypogene mineralisation have been recognised at the Mina Justa prospect, comprising a sequence of four distinct hydrothermal alteration stages (albite-actinolite alteration, K-Fe metasomatism, Ca metasomatism and an early haematite stage), followed by intense Fe-metasomatism that formed the magnetite bodies in the Mina Justa deposit. Subsequently the main Cu sulphide mineralisation stage of chalcopyrite, chalcocite and bornite replaced the precursor magnetite mineralisation in stratabound and structurally controlled ore bodies. The hydrothermal alteration sequence was concluded by a late-stage specular haematite deposition stage.

The mineralised bodies at the Mina Justa deposit extend over an area of approximately 2100 m north-south by approximately 1500 m west-east, and range in thickness from a few metres up to 150 m. The mineralisation is at or close to the surface in the northern and western parts of the deposit (the

"Northern Oxides", "Western Extensions" and "Cu40" zones, respectively), extending to depths approaching 550 m in the southeastern parts of the deposit (the "Sulphide Extensions" zone). The mineralised bodies are generally flat lying in the upper parts of the deposit (i.e. in the supergene oxidation zone). At depth the mineralisation follows the curvilinear faults, and resembles a flat bowl-like structure with an overall shallow plunge of approximately 15° to the southeast. Sulphide mineralisation at depth displays a central core of bornite and chalcocite (within the "Main Pit" in Figure 4.2) surrounded by predominantly chalcopyrite mineralisation. A narrow transition zone separates the sulphide mineralisation from the overlying oxide mineralisation. Sulphide mineralised bodies appear to increase in thickness from west to east, and with increasing depth.

The Magnetite Manto mineralised body strikes approximately northeast-southwest, with a moderate dip of approximately 60° to the northwest. The tabular body is some 700 m long by 350 m wide, ranging between 25 m and 35 m in thickness. The Magnetite Manto deposit is characterised by copper oxide mineralisation.

1.2.3 Exploration

Although surface mapping, geophysical and geochemical exploration have been undertaken, drilling has been the dominant tool used in the exploration of the Mina Justa Prospect. Rio Tinto carried out the original exploration drilling, completing a total of 31 025 m in 102 drill holes on the Mina Justa Prospect using a combination of diamond core and RC drilling techniques. From 2005 to 2008, Marcobre has drilled 227 844 m in 938 drill holes on the Mina Justa prospect, and 28 607 m in 137 drill holes on the Magnetite Manto deposit. Drill holes have been spaced between 25 m and 50 m apart on both deposits.

RC drilling has been the predominant method used by Marcobre. Drill hole inclinations and directions were selected and adjusted to intersect the mineralisation perpendicular to the structural trend and the interpreted trend of the copper mineralisation.

Drill hole collars have been surveyed, and downhole surveys undertaken using gyroscopic instruments.

Drilling in the Mina Justa deposit covers an area of approximately 7.5 km^2, with drill holes spaced between 25 m and 50 m apart (generally 35-40 m) and drilled to depths of up to approximately 630 m. Drilling in the Magnetite Manto deposit covers an area of approximately 0.23 km^2, with drill holes spaced between 25 m and 50 m apart (generally 25-30 m) and drilled to depths of up to approximately 410 m.

1.2.4 Logging, sampling, sample preparation and analysis

Marcobre and Snowden are unaware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. Samples are considered representative of the mineralisation in the Mina Justa Prospect. Summary details of all mineralisation intersections from the pre-2005 Rio Tinto, and the 2005 through 2008 Marcobre drilling programmes are included in Appendix 3.1 of the Feasibility Study.

A total of 127 868 samples were collected in and around the Mina Justa Prospect. All samples used for the October 2008 mineral resource are from drilling. A consistent sampling method and approach was maintained by Marcobre for each year's drilling programme.

Drill core was logged for geotechnical and geological features prior to being marked for sampling. Core sampling was conducted with respect to geological boundaries. Core sample intervals were generally 1 m for mineralised core and 2 m for non-mineralised core. Drill core sample recovery is generally better than 95%. Density measurements were conducted on selected core intervals after logging and before sampling. The standard weight-in-water-weight-in-air technique was used.

RC chips were collected at regular intervals and logged. RC samples were collected over 2 m intervals and riffle split to achieve 12.5% splits of approximately 10 kg. Marcobre attempted to quantify RC sample recovery through comparing sample mass, the results indicating an average recovery of better than 85%. The apparently low result for what should be a continuous sample is, however, considered to be a function of significant density variation in the deposit precluding an accurate assessment of recovery for the RC samples.

Reject and reference samples were stored in camp. Prepared coarse and fine blanks, oxide and sulphide standards as well as field, crush and pulp duplicates were inserted into the sample stream.

Sample preparation during Marcobre's 2005 through 2008 drilling programmes was carried out on-site. RC samples were dried and crushed to 95% passing 10 mesh. The crushed samples were then riffle split to produce 250 g samples, which were pulverised to 95% passing 200 mesh. Sample pulps were submitted to the SGS laboratory in Lima for analysis. Coarse sample preparation rejects were bagged and stored on site. Following analysis, the pulp sample was returned to Marcobre for storage on site. Diamond core was sawn and half-core samples prepared in the same manner.

The SGS laboratory in Lima was the primary laboratory to which all drilling samples collected from Marcobre's drilling programmes were submitted. The SGS Laboratory's Quality Assurance System has ISO 9002 accreditation and participates on a regular basis in round-robin testing with analytical laboratories in Canada, Sweden, and the USA, amongst others.

All sample pulps received were entered into the laboratory management system and uniquely bar-coded for Quality Assessment and Quality Control (QAQC) and tracking purposes. All preparation and analytical data recorded for the samples was done electronically. Marcobre submitted a total of 142750 samples (including QAQC samples) during the 2005 to 2008 drilling campaigns on the Mina Justa prospect.

Samples were analysed for total Cu (CuT) and sequential leaching (CuSeq-sulphuric acid extractable, cyanide extractable and residual Cu[1]) with an AAS finish. In addition, sulphide and transition zone samples were analysed for Ag using ICP-OES analysis with an aqua regia digest as part of a multi-element package (including Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, S, Sb, Sc, Se, Sn, Sr, Te, Ti, Tl, U, V, W, Y, Zn and Zr). Au analyses were carried out using a 30 g fire assay with an AAS finish.

[1] CuSS/Cu_SS, CuCN/Cu_CN and CuR/Cu_R, respectively

The assay data loaded into the database have been verified against the original laboratory certificates that are kept on file in the Marcobre Data Room. Marcobre's John D. Kapusta, P.Geo., Vice-President Exploration and Geological Services is the Qualified Person responsible for Marcobre's exploration, drilling, sampling and data quality.

Marcobre's site security includes a private road, entrance gate and around-the-clock site-based security guards. The SGS site-based laboratory is securely locked. SGS takes custody of all samples on site, once they have been appropriately bagged and labelled. Following sample preparation, sample pulps are transported by road (in the care of SGS) to SGS Lima for analysis. The SGS laboratory in Lima is completely surrounded by a security wall and all access is security controlled.

Marcobre and Snowden consider the sampling, sample preparation, security and analytical procedures to be of adequate quality to support the generation of mineral resource and mineral reserve estimates to DFS standard.

1.2.5 Data verification

Marcobre established a QAQC programme in 2005 to verify and monitor CuT, CuSeq, Au and Ag assay and analytical results provided by SGS. The QAQC programme established protocols for insertion of Quality Control (QC) samples, evaluation criteria, and secondary lab check analyses. The QC samples inserted into the sample batches submitted to SGS include: standard reference materials (SRMs), blank materials and duplicate samples (core or RC drill cuttings, coarse reject material and pulps). The rate of QC sample insertion is approximately 10% throughout the drill programme. In addition to insertion of QC samples, 5% of the drill hole samples were randomly selected, submitted to secondary (external) laboratories, and repeat assay results evaluated throughout the drill programme.

Analytical results were continually monitored by independent consultants, who evaluated accuracy, sample contamination, precision and bias on a routine basis.

Marcobre considers, based on the results of its QAQC programme, that the resource database (CuT, CuSeq, Au and Ag assays) is sufficiently accurate and precise for use in mineral resource and mineral reserve estimation.

Snowden's Dr. Warwick S. Board, P.Geo., has visited Marcobre's field and Lima based operations, the SGS on site preparation laboratory and the SGS laboratory on several occasions throughout the programme (in September 2005, December 2006, April 2007, April 2008 and June 2008), and observed drilling, collar and downhole surveying, drill core and RC chips, sampling, sample preparation, chain-of-custody, sample analysis, sample QAQC, density measurements, assay certificates and database maintenance, geological interpretation, as well as taking independent samples from selected drill holes. Based on observations made during the site visits, and the independent assessments of Marcobre's data, Snowden is of the opinion that Marcobre's drill hole sampling data is of adequate quality to support the generation of mineral resource and mineral reserve estimates to DFS standard.

1.2.6 Mineral resource estimation

1.2.6.1 Summary

Marcobre requested Snowden to prepare an updated resource estimate for the Mina Justa Prospect based on the latest validated database of sample information collected from exploration and infill drilling conducted on the deposit. The October 2008 mineral resource estimate presented in this report forms the basis for the Feasibility Study being conducted on the Mina Justa Prospect by GRD Minproc.

The October 2008 resource estimate was generated using a comprehensive database that included drilling data from the pre-2005 Rio Tinto exploration programme and the recent 2005, 2006, 2007 and 2008 (up to a cut-off date of 23 May 2008) Marcobre drilling programmes. The October 2008 mineral resource for the Mina Justa Prospect (i.e. including the Mina Justa and Magnetite Manto copper deposits) is summarised at three likely cut-off grades in Table 1.1 and Table 1.2.

Mineral resources presented in this section were prepared by Dr Warwick S. Board, P.Geo., Senior Consultant with Snowden. Dr Board is independent of Marcobre.

1.2.7 Block model generation

Iterative three dimensional geological modelling of lithology, mineralisation, structure and ocoite dyke intrusive units was conducted by Atticus and Associates in Lima, in conjunction with Marcobre's Lima-based geologists. Modelled solids and surfaces were verified and used, in conjunction with the validated drill hole database, in the generation of the October 2008 mineral resource estimate.

Assay (CuT, CuSeq, Ag and Au), lithological and mineralogical desurveyed drill hole files were created from assay, lithological, mineralogical and collar and survey information extracted from the validated database. The drill hole data were then domain-coded according to mineralisation domain, prior to being composited (to 2 m intervals to ensure consistency of sample support during estimation) within domains. Grade capping (top-cuts) was applied to the assay data, where required, to minimise the influence of extreme values in grade estimation. Three dimensional continuity analyses were conducted on the assay variables in the various domains to model mineralisation continuity for grade estimation.

A sub-celled, and appropriately coded (deposit, lithological unit, weathering zone and mineralisation domain) block model was generated using Datamine Studio 3 mining software. Average densities were determined for different lithological units in the oxide, transition and sulphide zones for each deposit, and applied to the relevant blocks as per the deposit, lithological and weathering codes. CuT, CuSeq (which includes Cu_SS, Cu_CN and Cu_R), Ag and Au grades were estimated into the block model using Ordinary Kriging with an expanding search. Alphanumeric mineralogical data, requested for mine planning purposes, were estimated into the block model using the Nearest Neighbour technique.

Block model grade estimates were reviewed in detail following completion of the estimation process, to ensure that the estimation process was valid. Sequential copper data were estimated into blocks using variogram and search volume parameters defined for total copper to honour ratios between the variables and the relationship CuT = Cu_SS + Cu_CN + Cu_R as far as possible. Following estimation, the sequential copper data were normalised to the total copper data on a block-by-block basis, given

that there is high confidence in the total copper data quality. The normalisation process was conducted to retain the ratios between the three sequential copper components. Detailed validation checks were conducted on the normalised sequential copper data to ensure that this process worked correctly. In general, the pre-normalisation sum of the sequential copper data matched fairly closely to the total copper grade. Consequently only minor adjustments were made to the individual sequential copper components during the normalisation process.

Resource model classification was conducted taking into account data quality (e.g. results of QAQC analysis, site visits etc.), geological continuity and confidence therein, confidence in the geological model and current level of domaining, grade continuity (from the variography), spatial representivity of density data, kriging efficiency and drill hole spacing. Based on the review of all of these factors, Snowden is of the opinion that Marcobre's total copper, sequential copper, silver and gold data are of sufficient quality and are sufficiently well spaced to support an Indicated and Inferred classification as per CIM (2005) for the Mina Justa and Magnetite Manto deposits.

The mineralogical data included in the October 2008 mineral resource update cannot be classified according to the CIM (2005) resource classification definitions. The mineralogical data incorporated into the model are, due to the nature of field logging by teams of field-based geologists, subjective. The data are also based on sample surface observations and cannot be considered representative of the entire solid mass of the sample.

The Indicated portion of the resource model was validated in detail as follows:

- Detailed review of block model compilation to ensure that all blocks were correctly coded in terms of deposit, lithology, weathering zone and mineralisation domain.
- Visual inspection of drill hole and block model grade data for each of the variables of interest, to assess that input data trends were honoured in the resource model.
- Global comparison of model and input drill hole grade data for the variables of interest by domain to assess for global bias.
- Comparison of grade trends of the variables of interest between declustered, domain-coded, composited and top cut input drill hole data and the block model, on easting, northing and elevation slices, to assess local bias.
- Grade-tonnage reporting checks.

Based on the results of the model validation steps outlined above, Snowden considers the October 2008 Mina Justa Prospect resource model to be valid, with the block estimates honouring the input drill hole data.

1.2.8 Comparison with previous estimates

Various generations of mineral resource estimates were generated for the Mina Justa Prospect between 2005 and 2008. Differences between these generations reflect the ongoing refinement of the current understanding of the geological and mineralisation continuity in the deposits as a function of additional exploration and closer-spaced infill drilling. Differences between the various resource model iterations (especially between the November 2006 (Snowden, 2007) and October 2008 iterations (this

report)) are expected and are considered to be a normal part of this iterative refinement process. Such differences are a direct reflection of changes to the geological interpretation, including changes to the interpreted mineralisation grade shell, location of the base-of-oxide and top-of-sulphide surfaces, lithological models and barren ocoite dyke models between iterations.

1.2.9 Mineral resources

It should be noted when considering the grade and tonnage estimates, that:

- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- A total copper cut-off grade of 0.3% exceeds operating costs as determined in the DFS.
- A Measured mineral resource (CIM, 2005) is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity. This classification requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.
- An Indicated mineral resource (CIM, 2005) is that part of a mineral resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed. An Indicated mineral resource estimate is of sufficient quality to support a preliminary feasibility study.
- An Inferred mineral resource (CIM, 2005) is that part of a mineral resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. Confidence in an Inferred mineral resource estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred mineral resources must be excluded from estimates forming the basis of feasibility or other economic studies.

The mineral resources at CuT cut-offs of 0.2%, 0.3% and 0.4% are summarised in Table 1.1. Detailed reporting by domain and over a range of cut-off grades is included in Section 17.

Table 1.1
Global Classified resources (October 2008)

Cut-off grade (CuT %)	Million Tonnes	CuT (%)	Cu_SS (%)	Cu_CN (%)	Cu_R (%)	Contained Cu (million lbs)
			Indicated			
0.2	411.3	0.67	0.26	0.19	0.22	6070
0.3	**336.8**	**0.76**	**0.29**	**0.23**	**0.25**	**5650**
0.4	246.9	0.91	0.31	0.29	0.30	4960
			Inferred			
0.2	77.5	0.72	0.08	0.12	0.53	1240
0.3	**64.6**	**0.82**	**0.08**	**0.14**	**0.60**	**1170**
0.4	50.9	0.94	0.08	0.15	0.72	1060

Silver and Gold grades are reported only for the Transition and Sulphide zones (Table 1.2)

Table 1.2
Global classified resources for silver and gold (October 2008)

Tonnage (Mt)	Ag (g/t)	Au (g/t)	Contained Ag (oz)	Contained Au (oz)
Indicated Resource				
161.8	8.75	55.95	45 530 000	291 000
Inferred Resource				
58.3	5.03	79.22	9 430000	148 500

1.3 MINING

1.3.1 Introduction

The Mina Justa and Magnetite Manto deposits are located at relatively low altitude, in an arid area with moderate topographic relief. Rock strengths are low to moderate. There is no groundwater and insignificant rainfall. These factors suggest that open pit mining should be routine and low cost. Some difficulties result from the relatively low grade of the Oxide ore, and the presence of barren dykes that are pervasive throughout the mineralisation. In order to minimise dilution and mining losses, selective mining on 10 m benches (5 m mining flitches) is specified, using excavators configured as backhoes.

The bulk of mining is focussed in the Mina Justa deposit, which includes Oxide (vat leach feed) and deeper Sulphide (concentrator feed) mineralisation. Higher grade Oxide ore is also mined from the Magnetite Manto deposit. Oxide ore is hauled to the vat feed crusher and long term Oxide stockpiles. Sulphide ore is hauled to a separate concentrator crusher. The majority of waste is hauled to the main waste dump, or to other destinations including the Magnetite Manto waste dump, the concentrator

tailings dam embankment, the ROM stockpile area (for the creation of a pad) and the ripios[2] disposal containment structure.

To meet throughput and selectivity requirements, mining uses 220 t class haul trucks and 20 m^3 backhoes and support equipment. All equipment is diesel powered. Total mining of approximately 60 Mt/a includes vat feed, concentrator feed and waste. The average waste to ore ratio is 2.46:1. Ripios is hauled using mine trucks. Mining operations continue over 12 years, inclusive of a 9 month pre-production period.

Mine planning activities performed during the course of the DFS included:

- Pit optimisation
- Pit and dump designs
- Mineral reserve estimation
- Mine and process scheduling
- Mine fleet assessment
- Mine operating and capital costs.

1.3.2 Pit optimisation

The resource model prepared by Snowden was based on 25x25x5 m parent block size with 5x5x1 m sub-cells. Several regularised mining models were prepared to simulate the impact on dilution and mining losses relative to the in-situ resource model, and a block size of 10 x 10 x 5 m was selected as the basis for mine planning.

Pit optimisation of the mining model (Indicated mineralisation only) was carried out using Whittle Four-X software. Optimisation input parameters were based on then-current information, including overall slope input (41° to 44°) from Knight Piésold and a copper price of $1.65/lb provided by Marcobre. Revenue was received from both Vat (Oxide) and Concentrator (Sulphide) ore processing streams. The optimisation was constrained to prevent mining on the adjacent Shougang property. A number of different scenarios and sensitivities were produced and shells were selected to form the basis for the ultimate and staged pit designs.

1.3.3 Pit and dump design

Ultimate and staged pit designs were created from the selected optimisation shells, incorporating access ramps.

Pit slope geotechnical investigations were carried out by Knight Piésold based on 15 geotechnical holes plus geotech data gathered from logging of another 17 exploration holes. Pit slope stability is most influenced by major structures and block faulting, with four major fault systems identified in the pit area. Stability analysis was carried out for overall slope rock mass failures and also for local failures controlled by rock structures. Recommended pit slopes incorporate a Factor of Safety of 1.3. Final

[2] Ripios is a term used for the residue following vat leaching

recommended bench heights are 20 m[3] with a range of 9 to 11 m. Bench face angles range from 65° ° to 70°, and average inter-ramp slopes range from 45° to 50° and average inter-ramp slope angles range from 45° to 50° depending on the design sector.

Access was generally by a single ramp of 30 m width at a maximum design grade of 10%. Ramps were narrowed to one way at depth in the pits to minimise associated waste. There are four discrete pits, of which two are developed in stages to defer waste stripping and improve ore presentation. Three of the four pits exploit the Mina Justa deposit, while the fourth exploits the Magnetite Manto deposit.

Pit inventories are summarised in Table 1.3.

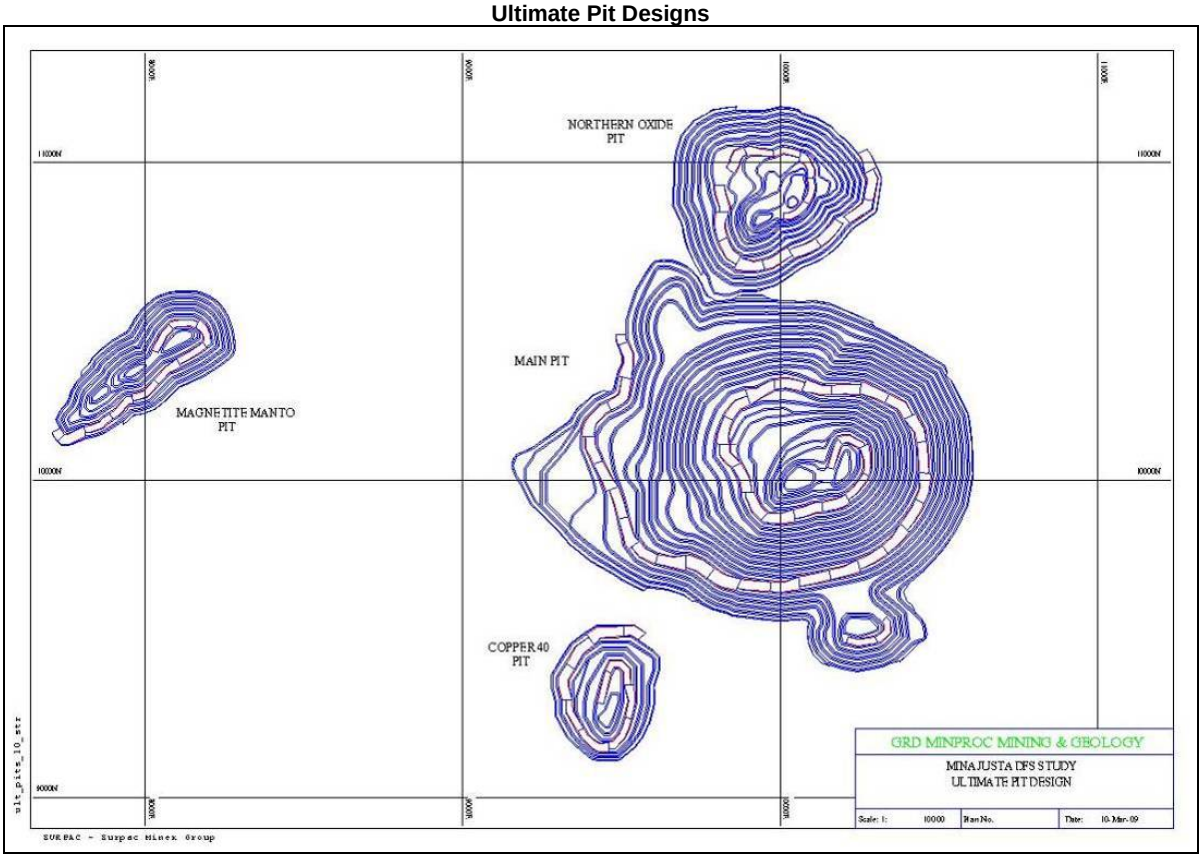
Table 1.3
Pit Inventory

	Unit	Main Pit	Northern Oxide	Copper 40	Magnetite Manto	Total
Vat Ore	Mt	90.4	17.0	2.6	4.6	**114.6**
Concentrator Ore	Mt	48.0	-	0.8	-	**48.8**
Waste Tonnes	Mt	329.5	43.5	10.9	18.5	**402.4**
Total Material	**Mt**	**467.9**	**60.5**	**14.3**	**23.1**	**565.8**
Strip Ratio SR	t:t	2.38	2.56	3.23	3.96	**2.46**

Ultimate pit designs are illustrated in Figure 1.1.

[3] Mining is conducted on 10 m benches

Figure 1.1
Ultimate Pit Designs



Mining will generate 402.4 Mt of waste rock. The main waste dump is located northeast of the main pit and also serves as the containment structure for ripios storage. The ripios will be surrounded by mine waste rock to maintain adequate long-term physical stability of the material. The ripios dump has a capacity of approximately 110 Mt, sufficient to contain the waste product from the vat leaching process.

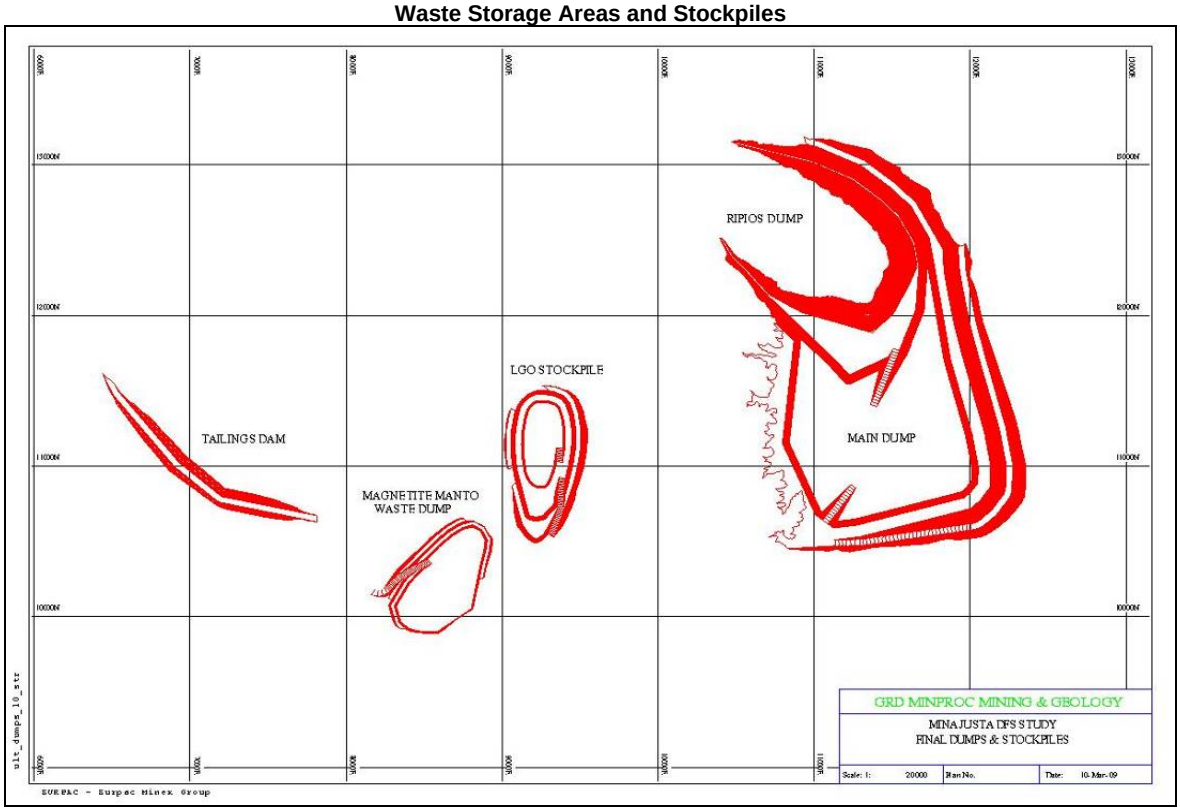
A separate dump is provided for Magnetite Manto pit waste, some of which will be used for tailing dam embankment construction. Some mine waste will be directed to enlarge the ROM pad to provide a suitable configuration for dumping, storing and rehandling of crusher feed. Opportunistic backfill of waste into the Northern Oxide pit may be feasible at the end of the mine life.

The disposal of small quantities of potentially acid generating (PAG) waste rock has been considered conceptually by ensuring that it is surrounded and covered with non-PAG material in the main waste dump.

A large, long-term stockpile is allowed for excess, lower grade vat leach feed (LGO) that accumulates during the early years of mining.

Waste storage areas and stockpiles are illustrated in Figure 1.2.

Figure 1.2
Waste Storage Areas and Stockpiles



1.3.4 Mineral reserve

The Mina Justa Mineral Reserve is that portion of the Indicated Resource that is contained within the ultimate pits and has recoverable metal values that allow economic treatment. The Mineral Reserve tabulated by classification is identified in Table 1.4.

Table 1.4
Mina Justa Probable Mineral Reserve (1), (2), (3)

Classification	Tonnes (Mt)	CuT (%)	CuSS (%)	Ag (ppm)
Vat Feed	114.6	0.56	0.46	-
Concentrator Feed	48.8	1.37	-	14.1
Total	**163.4**	**0.80**	**-**	**-**

Notes:

(1) Reported according to NI 43-101 reporting guidelines, QP is Ross Oliver, an employee of GRD Minproc

(2) No Measured resource so no Proved Mineral Reserve

(3) Mineral Reserve cut-off for both vat and concentrator feed is based on an NSR (Net Smelter Return) calculation and a copper price of $1.65/lb.

1.3.5 Mine and process schedules

Bench reporting of reserve information was performed for the pit stages and imported into a purpose-built mine scheduling spreadsheet. A variety of mining rates and vat leach cathode production profiles

were investigated. A final mining rate of 60 Mt/a was adopted as a sustainable rate that will bring forward the mining and treatment of higher grade concentrator feed, and also sustain a cathode production rate of 52 000 t/a of copper.

Mine and process scheduling was carried out on a monthly basis for the pre-strip (Yr-1) and first year of production, quarterly for years 2 through 5 and annually thereafter. The quarterly resolution was necessary to ensure ore availability for the deferred concentrator start-up.

Figure 1.3 illustrates the mining production rate by pit stage over the mine life. The majority of mining is associated with developing and sustaining the presentation of the deeper sulphides from the Mina Justa Main Pit stages. Figure 1.4 shows the ore types mined.

Figure 1.3
Mining by Pit Stage (Mt)

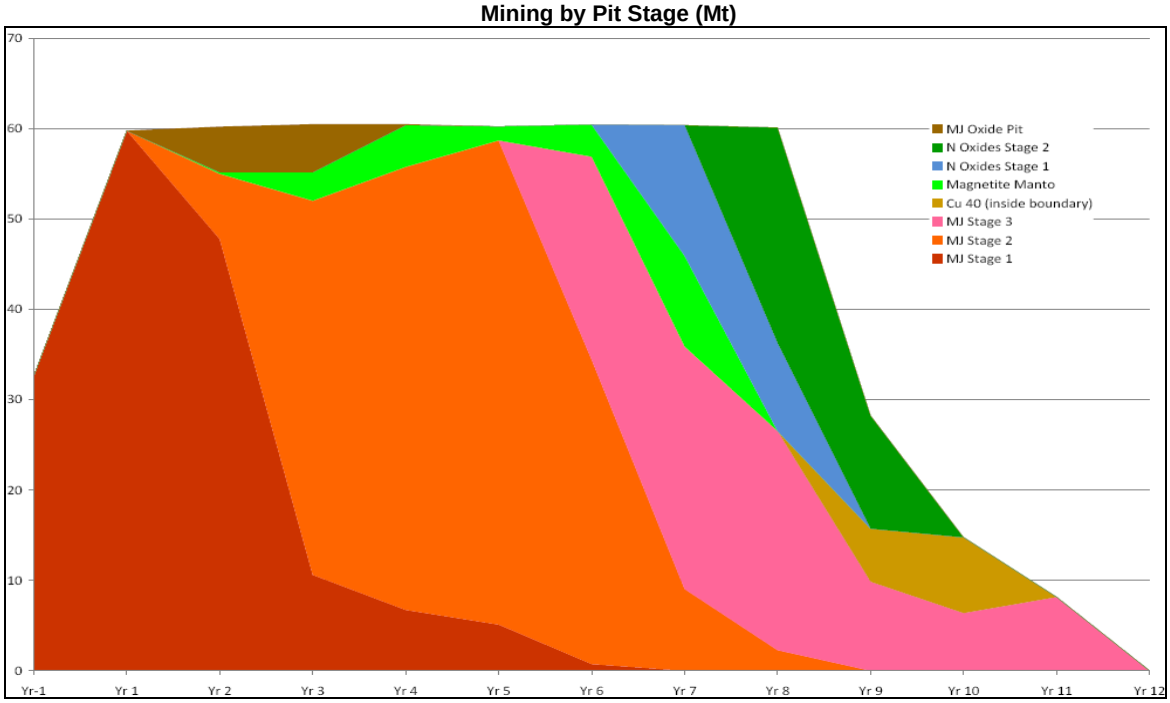


Figure 1.4
Ore Mining by Material Type (Mt)

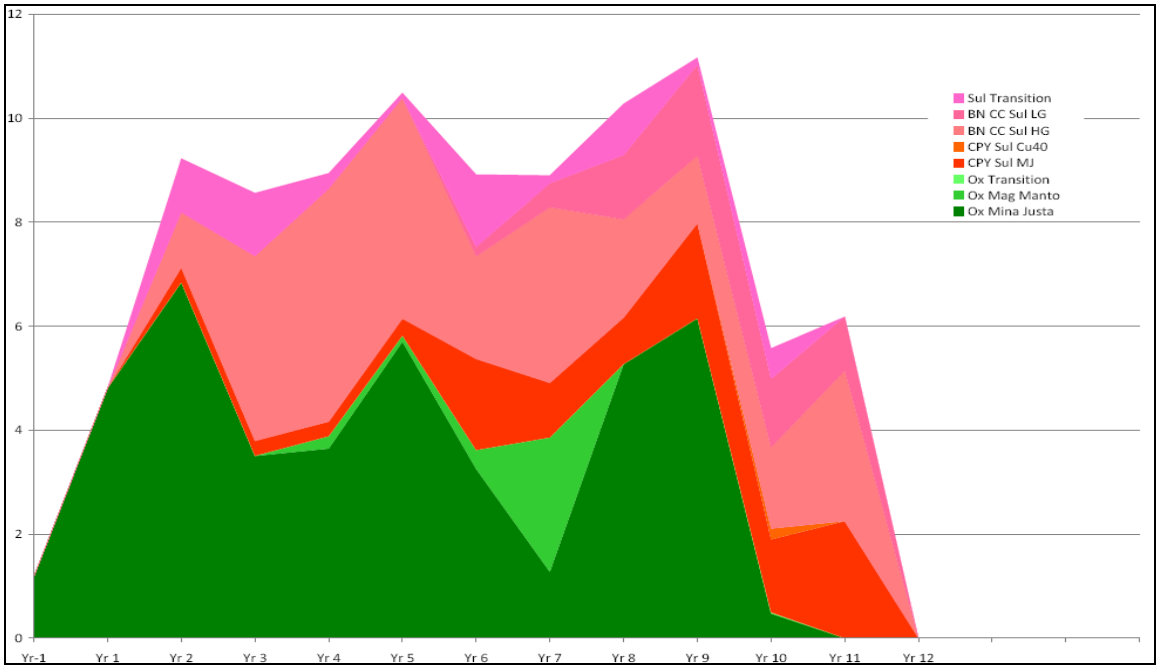


Table 1.5 shows the annual mining and processing production schedule

Table 1.5
Annual Production Schedule – Mining and Processing

Mining		**Total**	**Yr-1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**
Vat Ore	(kts)	**114 602**	4 640	14 791	20 568	8 977	8 866	14 836	12 811	8 158	10 059	8 762	2 134		
Cu	(%)	**0.56%**	0.44%	0.47%	0.54%	0.57%	0.57%	0.58%	0.54%	0.71%	0.60%	0.69%	0.47%	0.00%	
CuSS	(%)	**0.46%**	0.37%	0.40%	0.42%	0.44%	0.46%	0.47%	0.39%	0.58%	0.51%	0.62%	0.36%	0.00%	
Float Ore	(kts)	**48 794**			2 380	5 055	5 059	4 670	5 301	5 043	5 005	5 014	5 081	6 183	
Cu	(%)	**1.37%**			1.17%	1.43%	1.33%	2.79%	1.19%	1.04%	1.01%	1.07%	1.09%	1.53%	
Au	(g/t)	**0.029**			0.016	0.016	0.020	0.037	0.026	0.022	0.021	0.033	0.044	0.048	
Ag	(g/t)	**14.1**			8.8	12.2	11.8	32.0	12.9	10.7	11.7	10.8	9.6	18.2	
Total Ore	(kts)	**163 396**	4 640	14 791	22 948	14 032	13 925	19 506	18 112	13 201	15 064	13 776	7 216	6 183	
Waste	(kts)	**402 363**	27 942	44 972	37 251	46 446	46 550	40 724	42 330	47 157	45 038	14 442	7 544	1 968	
Total Mining	(kts)	**565 759**	32 582	59 763	60 199	60 478	60 475	60 231	60 442	60 358	60 102	28 219	14 760	8 151	
Strip Ratio		**2.46**	6.02	3.04	1.62	3.31	3.34	2.09	2.34	3.57	2.99	1.05	1.05	0.32	
Closing Stockpiles															
HG Vat Feed	(kts)		1 180	147	1 651			1 406		825	795	2 223			
MG Vat Feed	(kts)		1 940	4 101	4 950	2 383									
LG Vat Feed	(kts)		1 520	5 934	12 150	13 344	12 593	14 024	16 241	11 574	9 663	4 998			
Float Ore	(kts)				461	515	574	243	544	587	592	606	687	1 871	
Total	(kts)		4 640	10 183	19 212	16 241	13 167	15 673	16 785	12 986	11 050	7 827	687	1 871	
Processing															
Vat Ore	(kts)	**114 602**		9 248	12 000	12 001	12 000	12 000	12 000	12 000	12 000	12 000	9 355		
Cu	(%)	**0.56%**		0.56%	0.61%	0.59%	0.53%	0.56%	0.61%	0.58%	0.56%	0.54%	0.48%		
CuSS	(%)	**0.46%**		0.47%	0.47%	0.47%	0.43%	0.47%	0.45%	0.46%	0.47%	0.47%	0.39%		
CuRec	(%)	**0.42%**		0.43%	0.43%	0.43%	0.39%	0.43%	0.42%	0.43%	0.43%	0.43%	0.36%		
Acid	(kg/t)	**40.66**		40.15	40.32	40.39	41.10	40.49	40.67	40.91	40.39	40.48	41.89		
Cu Recovery	(%)	**74.5%**		78.1%	71.5%	72.2%	73.4%	76.2%	68.8%	75.2%	77.4%	79.7%	74.0%		
Cu in Cathode	(t)	**481 596**		40 100	51 999	51 513	46 867	51 601	50 157	52 010	51 997	52 056	33 295		
Float Ore Feed	(kts)	**48 794**			1 919	5 001	5 000	5 000	5 000	5 000	5 000	5 000	5 000	5 000	1 871
Cu	(%)	**1.37%**			1.23%	1.43%	1.33%	2.57%	1.31%	1.04%	1.01%	1.07%	1.10%	1.73%	0.73%
Au	(g/t)	**0.029**			0.016	0.016	0.020	0.035	0.028	0.022	0.021	0.033	0.044	0.055	0.018
Ag	(g/t)	**14.1**			9.3	12.3	11.9	28.9	14.1	10.7	11.7	10.8	9.7	20.8	8.9
Cu Rec to Con	(%)	**93.0%**			90.1%	91.9%	92.5%	95.3%	92.9%	91.8%	91.5%	92.6%	92.3%	95.0%	87.8%
Au Rec to Con	(%)	**80%**			80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Ag Rec to Con	(%)	**80%**			80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Concentrate	(dry t)	**1 643 741**			56 279	164 301	144 397	291 496	176 948	126 276	125 066	150 083	151 018	224 479	33 399
Cu Con Grade	(%)	**37.8%**			37.7%	40.1%	42.7%	42.1%	34.5%	38.0%	37.1%	33.0%	33.7%	36.7%	35.9%
Au Con Grade	(g/t)	**0.70**			0.44	0.39	0.56	0.48	0.63	0.70	0.67	0.88	1.17	0.98	0.81
Ag Con Grade	(g/t)	**335**			255	299	329	397	319	340	375	288	256	370	399
Cu in Con	(t)	**621 373**			21 192	65 867	61 618	122 596	61 053	47 945	46 431	49 528	50 834	82 319	11 989
Au in Con	(ozs)	**36 978**			792	2 082	2 608	4 516	3 565	2 850	2 679	4 261	5 700	7 057	867
Ag in Con	(kozs)	**17 725**			461	1 579	1 528	3 718	1 817	1 381	1 508	1 391	1 244	2 669	429
Total Copper	**(t)**	**1 102 969**		**40 100**	**73 191**	**117 379**	**108 485**	**174 198**	**111 210**	**99 955**	**98 428**	**101 584**	**84 129**	**82 319**	**11 989**

Note: Metal in concentrate is total contained metal. The financial analysis summary (Table 1.20) shows payable metal in concentrate after smelter deductions.

Vat and concentrator ore processing are shown in Figure 1.5, and long-term ore stockpile inventories are illustrated in Figure 1.6.

Figure 1.5
Vat and Float Ore Processing (Mt)

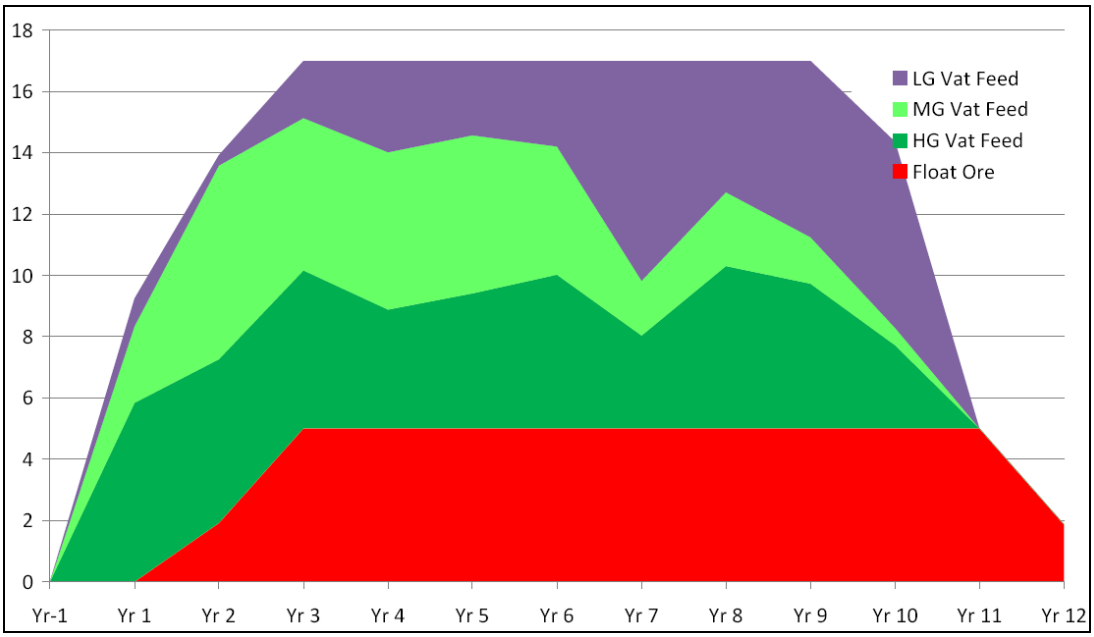
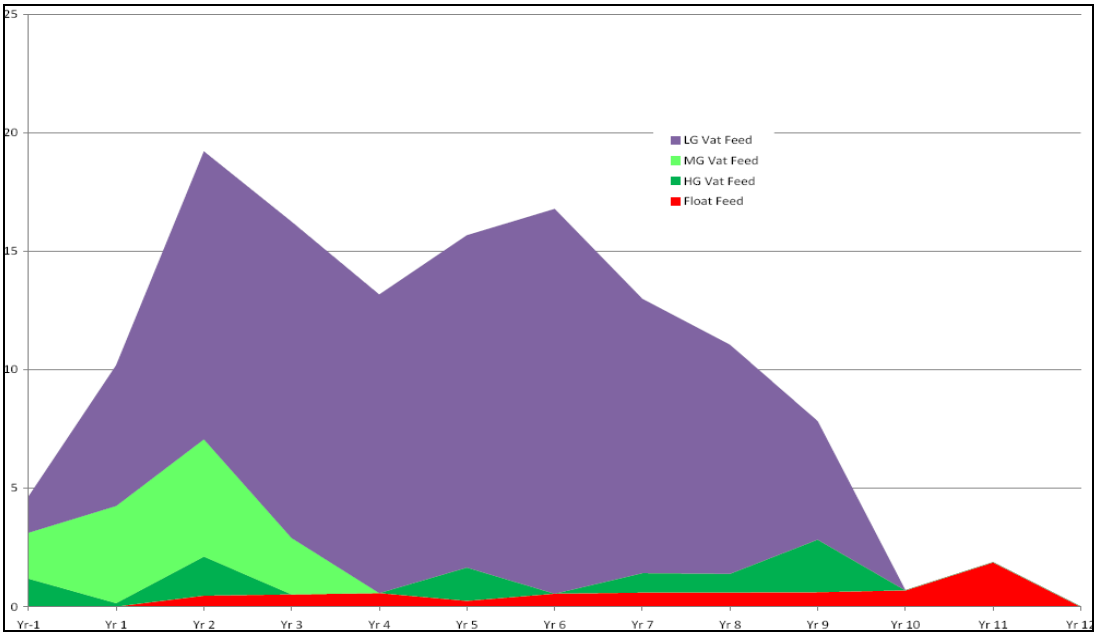


Figure 1.6
Long Term Stockpile Inventories (Mt)



1.3.6 Mine fleet assessment

A backhoe excavator was selected as the primary digging unit, in order to minimise dilution and mining losses, and maximise the mined ore grade.

The selected bench height is 10 m to allow 2 nominal 5 m flitches to be mined by the backhoe. After considering blast heave, the actual flitch height dug by the backhoe will average about 6 m. Backhoes in the 20 m^3 class can operate productively with this bench height. The selected excavator also has a production rate that is a good match for the peak throughput rates of the vat leach circuit primary crusher.

A large front end loader (FEL) has been specified to serve the following functions:

- Provide production loading back-up in the pit when a primary excavator is unavailable.
- Provide truck loading for rehandle of long-term vat feed stockpiles.
- Rehandle (by tramming) from short-term operational stockpiles located on the ROM pad.

In order to keep operating costs low, 220 t class haul trucks and support equipment have been selected. Crawler-mounted diesel drills capable of single pass drilling have been selected for productivity and operational flexibility.

A computerised dispatch system has been allowed, to monitor equipment, provide production statistics and provide the information to measure and improve fleet productivity.

Table 1.7 includes a summary of the major equipment selected.

1.3.7 Mine operating cost

The DFS mining concept is that the mine equipment will be owned, operated and maintained by the Owner with support by specialist contractors in the following areas:

- Down-the-hole explosive supply.
- Vendor-provided preventative maintenance services for major equipment, inclusive of labour, site support and consignment stock.
- Diesel supply, storage and dispensing services.

Explosive supply will be by a local vendor providing a down-the-hole service. Since the conditions are dry, ANFO has been specified as the sole explosive. After assessment of rock properties, powder factors of 0.20 kg/t in waste and 0.24 kg/t in ore have been adopted.

Key operating cost drivers are summarised in Table 1.6.

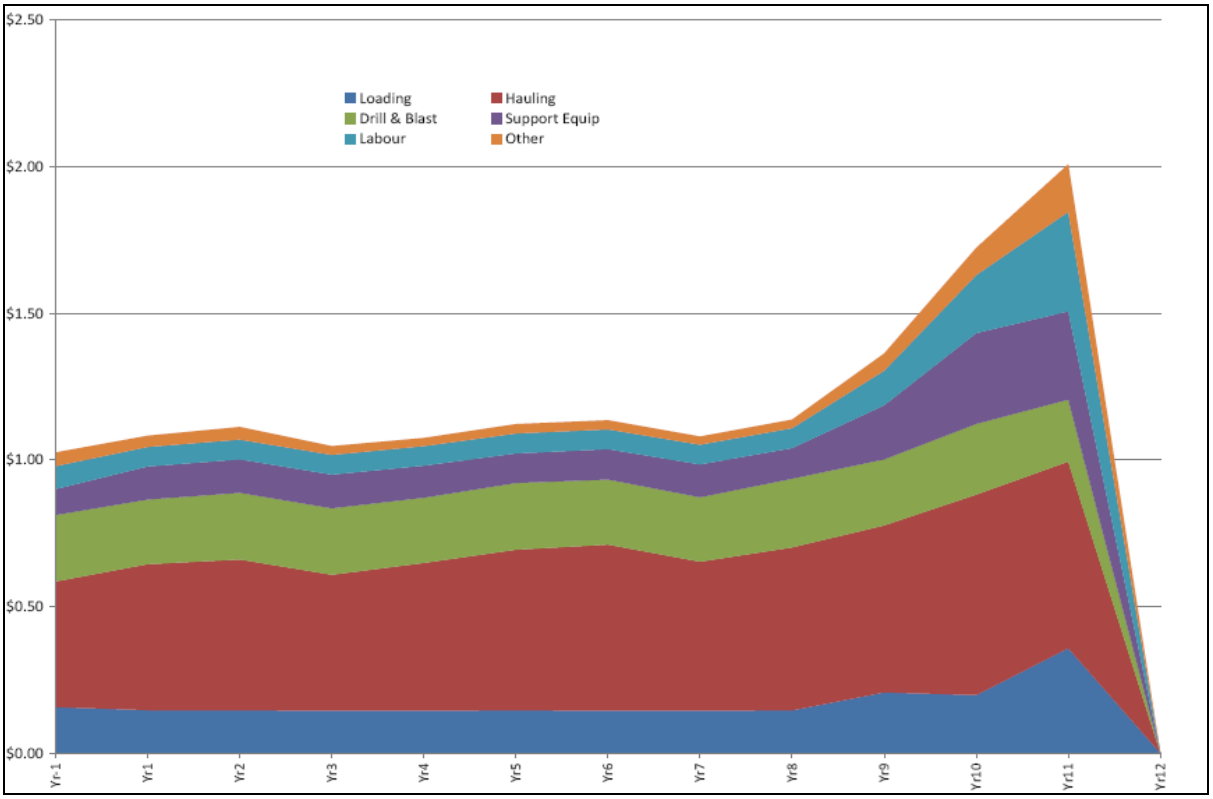
Table 1.6 Key Operating Cost Inputs			
Item	**Value**	**Unit**	**Comment**
Diesel	0.636	$/litre	includes storage & dispensing
AN Explosive	540	$/t	Dry, 100% ANFO used
Powder Factor Ore	0.24	kg/m^3	
Powder Factor Waste	0.20	kg/m^3	
Truck Tyre Life	5 000	hrs	

The proposed major equipment fleet make-up is summarised in Table 1.7, together with key equipment assumptions used to build up the operating cost estimate. While specific equipment models have been used to build up the estimate, actual fleet configuration would be subject to a further tendering and evaluation process to establish the most cost-effective mining solution.

Table 1.7 Equipment Fleet and Hourly Costs						
Type	**Equipment Class**	**Fleet Units**	**Operating Hours hr/yr**	**Operating Costs US$/hr**	**Purchase Price (US$ M)**	**Expected Life (hr)**
Excavator	20 m^3	3	6 701	$435	$5.55	60 000
Dump Truck	228 tonne	23	6 701	$219	$3.65	65 000
FEL	20 m^3	1	5 585	$280	$4.67	50 000
Track Dozer	433 kW	4	5 046	$105	$1.22	30 000
Wheel Dozer	372 kW	1	4 840	$89	$1.01	50 000
Grader	221 kW	3	5 606	$64	$0.82	40 000
Water Truck	45 kl	2	3 723	$91	$1.11	60 000
Production Drill	229 mm	4	4 906	$44	$1.61	50 000

Figure 1.7 illustrates the mining operating unit cost with time, showing the major operating cost components. The above mining costs are inclusive of all material handling from stockpiles, and the transport and placement of ripios on the ripios dump (although these ripios disposal costs would be more properly allocated to processing).

Figure 1.7
Operating Costs by Time (US$/t)



The average mine operating cost for the life of the mine is $1.14/t mined. Unit costs are seen to increase in later years as haul distances increase and the total tonnages mined decrease.

1.3.8 Mine capital cost

The total capital cost (including replacement, rebuilds and sustaining capital) of the mining component has been estimated at $139 M as detailed in Table 1.8. This cost does not include capitalisation of mining costs in the construction period.

Table 1.8
Mining - Capital, Sustaining and Replacement Costs (US$-M)

Mine Area	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Total
Loading	21.78													21.78
Hauling	65.70	14.59	0.45		0.45	3.65	0.45							85.28
Drill & Blast	6.45													6.45
Support	8.52	2.04				3.65		1.23						15.44
Other	7.42	0.60	0.03	0.63		0.09	0.81	0.20		0.21		0.04		10.02
Total:	109.87	17.24	0.48	0.63	0.45	7.39	1.25	1.43	0.00	0.21	0.00	0.04	0.00	138.97

1.4 METALLURGICAL TESTWORK

Metallurgical testwork has been completed on representative sub-samples of the Mina Justa and Magnetite Manto ore to determine the processing requirements. This sub-section summarises the metallurgical development process used to select the required processing flowsheets for the Oxide and Sulphide ores.

1.4.1 Oxide ore

1.4.1.1 Comminution testwork

Testwork was initially conducted by Metso Minerals on four samples of Mina Justa Oxide ore. These test results generated a Macon crushability result of 28%, indicating that the Oxide ore is competent and difficult to crush.

Following this preliminary testwork, a second series of comminution tests was conducted on diamond core samples representative of the ore types and geometry of the Oxide mineralisation. Two main lithologies were defined, namely sedimentary and andesitic rocks, with the latter being sub-divided into vesicular and non-vesicular sub-types.

This crushing testwork program was initially conducted at Phillips Enterprises LLC (Phillips) in Colorado, USA. The tests were conducted on 21 samples, using approximately 20 rock specimens per sample. Testwork, using a modified Bond test machine, generated results that were lower than anticipated, so a further series of tests was undertaken by Ammtec Limited (Perth, Western Australia). A total of twelve samples of varying numbers of rock specimens remaining from the Phillips testwork were used for the additional tests. It was confirmed that the Ammtec results were higher (by 45%) than the Phillips results. The Ammtec testwork and results were selected for detailed analysis and generation of the Bond crushing work index design values, while the Phillips testwork results were used to further investigate trends generated from the Ammtec results.

Assessment of crushing work index (CWi) results indicated no regular variation with down hole depth, nor with regards to spatial distribution, but the results displayed a large range of results between specimens for each sample tested.

The crushing testwork results showed a proportion of the rock specimens tested were sufficiently more competent than the main sample set. The increase in competency for these samples reflects rock with minimal inherent fractures. As the material is crushed finer (and the rock fractures are removed), the competency of the ore is expected to increase. This trend has been used in the selection of CWi values for the design criteria. A Bond crushing work index value of 10 kWh/t was selected for the primary to tertiary crushing stages, increasing to 16 kWh/t for the quaternary stage.

Unconfined compressive strength (UCS) testwork was undertaken by Advanced Terra Testing Inc. The overall dataset showed an average UCS value of 48.2 MPa, and a maximum value of 130.2 MPa.

Bond abrasion index tests were conducted by Phillips on two samples from the main programme. The results showed that the sedimentary material is the least abrasive, with an average abrasion index of 0.13. Andesitic material is more abrasive with an abrasion index of 0.22, increasing to 0.24 for the

amygdaloidal andesitic material. Limited results were also obtained for the Magnetite Manto deposit, which showed a large variation in results (0.08 to 0.32) due to the different lithologies tested. On average, the two deposits are expected to be moderately abrasive with an abrasion index value of 0.19.

Based on the parameters generated during the comminution testwork program, modelling of potential crushing circuits was undertaken. Several flowsheet options were modelled including three and four stages of crushing, open and closed circuit secondary crushing, crushed ore stockpile options and various equipment configurations. From this modelling, the optimal circuit design was determined to be a quaternary crushing circuit, with the secondary stage in open circuit, and the tertiary and quaternary stages in closed circuit.

1.4.1.2 Leach testwork

Testwork on the Mina Justa Oxide material started with bottle roll leaching, moved to individual column tests, and finally evolved into an integrated pilot program run continuously in locked cycle. Pilot testing was followed by an on-going program of variability testing on material from various areas of the proposed Mina Justa and Magnetite Manto open pits.

Bottle roll testing commenced prior to Marcobre's acquisition of the project. In the Marcobre tests, the principal variables were crush size (1 to 25 mm), acid level (pH 1.2 to 2.5) and lithology (andesitic or sedimentary). Both the Rio Tinto and Marcobre results demonstrated that the Mina Justa Oxide ore is inherently leachable, with recovery generally declining as the crush size or pH increased. Both andesite and sedimentary material yielded 100% extraction of the acid soluble copper (CuSS) at fine crush sizes (1 and 3 mm top size) and high acid levels (pH <1.5). Recovery from andesite was found to be more sensitive to both crush size and pH than for the sedimentary rock. Acid consumption by the andesite was also more sensitive to these variables than it was for the sedimentary material. Ancillary bottle roll tests suggested that a high chloride level would reduce acid consumption, but retard copper extraction.

A column testing program followed, based on the results of the bottle roll tests, with the focus being on identification of design parameters for a heap leach operation. However, the results were discouraging; high recoveries could be achieved (>90%), but long leach cycles (three to five months) and high acid levels were needed. The resulting gangue acid consumption (GAC) approached 100 kg/t, with specific consumptions of 11 to 20 kg acid/kg Cu. The runs with the lowest specific acid consumption (7 kg acid/kg Cu) achieved recoveries of only about 70%, with GAC levels just below 50 kg/t.

The problem proved to be one of relative kinetics. The initial copper leach rate was fast, with about half of the total recovery occurring in the first 10 days. After that, the leach rate slowed dramatically, typically taking another three or four months for the recovery to double. The rate of acid consumption behaved quite differently, rising linearly with time over the entire leach cycle. Thus after 10 days, recovery was reasonably good and acid consumption was still very low. However, after another three to four months of leaching, Cu extraction doubled, but GAC increased by a factor of 14 or 15 times.

One set of tests showed that increasing the irrigation rate improved copper recovery without increasing acid consumption. This observation and the kinetic factors led to a key series of experiments under which flows were increased in stages from 10 to 40 L/h.m^2. At the same time, the acid concentration in

the leach solution was decreased proportionately so that after 55 days of leaching, each test received the same total quantity of acid. The results were quite dramatic; recovery increased as the flow rate increased. At the same time, total acid consumption, GAC and specific acid consumption all declined progressively.

These results focussed attention on possible vat leaching with its high flow rate, short cycle time and good wash efficiency. A vat testwork program was developed, starting with a series of batch trials. These tests generally confirmed earlier studies, which showed that the pH needed to be below 1.5 to give effective leaching. Recovery generally increased as the crush size decreased. While the GAC values in kg/t tended to increase with finer crushing, the specific acid consumption (kg acid/kg Cu) declined in many cases. The results suggested that operating on a six day leach cycle with material crushed to between 6.0 and 9.5 mm would provide an optimum vat process.

In order to investigate the range of leaching outcomes, forty composites were tested under identical (but non-optimised) conditions. When samples containing less than 0.3% Cu (the expected cut-off grade) were deleted from the database, the average recovery was 62.5% of CuT (equivalent to 75% of the acid soluble copper) and the average GAC level was 44 kg/t.

Positive results in the batch vat tests led to testing in an integrated vat pilot plant operating continuously in locked cycle. The Phase 1 tests utilized four composites: low-grade and mid-grade samples from the Mina Justa deposit, and mid-grade and high-grade samples from Magnetite Manto. During this phase, some problems with acid control and copper stripping from the pregnant leach solution were experienced. In spite of this, the Phase 1 results showed that crushing to 8 mm gave the optimum recovery. The results also showed that acid cure dosages of 10 to 20 kg/t had minimal effects on leach performance. Over this range both extraction and GAC values were highest at the lowest cure dosage.

Overall, the higher grade Magnetite Manto material gave the best leach performance. It yielded the highest recovery of both total and acid soluble copper. It also had the lowest average GAC value (24.5 kg/t) and the lowest specific acid consumption of less than 4 kg acid/kg Cu.

At the end of Phase 1 testing, samples of the final leach solutions from each ore type were sent to solvent extraction reagent vendors for compatibility studies. Results were positive, with no copper transfer or phase disengagement problems noted.

After correcting operational problems (acid control and copper stripping), a second pilot plant campaign (Phase 2) was conducted, using the optimal process conditions established in the Phase 1 tests. The 37 Phase 2 samples were selected to provide a variability program that was intended to demonstrate the effects of various resource parameters such as head grade, mineralogy, lithology and depth in the deposit, along with composites (blends) based on the then-current mine plan.

While the extraction of copper is undoubtedly influenced to some extent by the various resource parameters, for the purposes of the DFS the key relationship is the one between recovery and head grade. Regression analysis showed that the grade-recovery relationship was positive , with the following form:

$$\text{Recovery of CuT (\%)} = (86.5 + 9.3 \times \text{CuSS}) \times (\text{CuSS/CuT})$$

Maximum copper recovery is capped at 95% of CuT to prevent the projection of 100% extraction from high grade ore.

Analysis of results showed that total acid consumption was primarily dependent on gangue characteristics. Regression analysis demonstrated a negative relationship between GAC and head grade, caused by the increased acid credit from higher grade material. An exponential expression was fitted to the data as follows:

$$\text{GAC (kg/t)} = 50.07e^{(-0.47 \times \text{CuSS})}$$

Theoretically, the GAC value could go negative if the electrowinning (EW) acid credit were greater than total acid consumption, but, in practice, the highest grade ore in the block model was projected to have GAC values only as low as 9.9 kg/t.

Several other design parameters were determined in conjunction with the Phase 2 tests. These included:

- Total suspended solids (TSS) in the vat overflow. Clarification tests on the PLS were conducted to provide the design basis for the PLS clarifier.
- The viscosity of the leach solution was also checked. It averaged 1.5 cP, but increased gradually during the tests and ended at 1.8 cP.
- The final moisture in the leach residues ranged from 9.0 to 16.4%, with an average of 11%.
- On average, the void space in the ore bed decreased from 42% to 37.5% during the six day leach cycle.

One resource parameter that did appear to affect performance was depth in the deposit. Surface and near-surface material typically yielded higher recoveries than deeper material. This may be related to enhanced near-surface weathering and better solution access to the mineralisation.

A follow-up leaching variability testwork program was initiated, using a large-scale bottle roll procedure. This involved testing of more than 200 samples from various locations in both Mina Justa and Magnetite Manto. One of the objectives was to characterize material so that the results could be used in the block modelling work, while another was to develop a fast, simple test that could be used on blast hole material to estimate leach behaviour during operations. Unfortunately, the results did not replicate those obtained in the pilot tests. Both the recoveries and acid consumptions in the variability tests were higher than expected. Additional testwork has been undertaken to resolve the differences and improve the variability test procedure. However, the results are still being assessed and it appears that further work may be necessary to develop a viable procedure to support grade control and ore material type classification.

1.4.1.3 Solvent extraction and electrowinning testwork

Solvent extraction (SX) testwork and modelling was conducted by Cognis Corporation in Chile. Additional SX modelling was conducted by Cytec Limited in Peru, based on analysis of the PLS solutions generated from phase 2 vat leach testwork.

Modelling of the SX process was carried out by Cognis to investigate a number of different circuit configurations, from which a circuit configuration of 2 extraction stages, 1 stripping stage, and 1 washing stage (i.e. 2E+1S+1W) was selected.

Further modelling of this circuit at a copper tenor of 8 g/L and revised pH of 1.9 showed that the LIX84-I reagent performed best.

Samples generated for SX testwork were also sent to Cytec in Peru, to be analysed at independent laboratories. Cytec investigated numerous circuit configurations, extractant systems and PLS characteristics to determine the optimum design with respect to configuration, capital cost, and operating cost. The various modelling exercises confirmed the selection of a 2E+1S+1W circuit configuration.

Extraction of 94% of the soluble copper in the PLS stream is expected to be achieved in the SX circuit, based on a PLS stream containing 8 g/L copper at a pH of 1.9. An extractant concentration of 25% v/v is required for effective extraction of the soluble copper. Stripping of copper from the extractant is achieved using an electrolyte solution containing 35 g/L copper and 180 g/L sulphuric acid. These electrolyte parameters are relatively standard for copper solvent extraction systems in plants world-wide.

Full ICP scans of the PLS solution from the Phase 2 vat leaching testwork program show several impurities that needed to be considered in the design of the SX circuit, including iron, manganese, silicon (colloidal silica) and chloride. Mitigation measures in the SX plant design include a wash mixer/settler and coalescer tank to remove aqueous entrainment carrying iron, manganese and chloride. Equipment for treating the organic stream with activated clay to mitigate the effects of colloidal silica has been included in the crud treatment area of the SX plant.

1.4.2 Sulphide ore

Several testwork campaigns were completed during the evaluation of the Mina Justa sulphide deposit at a PFS level of detail. The sample collection and testwork were directed by Transmin Metallurgical Consultants out of Lima (Transmin).

1.4.2.1 Comminution testwork

Comminution testwork was conducted on samples representing “Transitional”, “Primary” (Cpy), and “Secondary” (Bn-Cc) ore types. The initial testwork campaign was conducted at SGS Lakefield Research in Chile during 2006, and was limited to Bond abrasion, rod mill and ball mill work index tests. The testwork results obtained indicated that all three samples tested were of moderate grindability and abrasiveness.

A second comminution testwork campaign was conducted in 2008 at SGS Lakefield Research in Chile, and by JKTech in Australia, the results being generally similar to those of the earlier campaign.

The SMC test results showed a similar trend to that found with the Rod and Ball work indices. The highest competency ore type was Transitional ore, which was classified as very hard, while the Secondary Sulphide sample displayed moderate to hard competency. The Primary ore and mixed samples matched an average to soft classification.

To further investigate the comminution characteristics of the Sulphide ore, a third testwork campaign was conducted in late 2008. The testwork, performed by Laboratorio Plenge in Peru, included ball mill work index and SMC tests, with the results of the SMC tests evaluated by JKTech in Australia. The Primary and Secondary Sulphide ore samples were designated as very hard to hard, except for Primary ore from the Cu40 zone which was classified as having average competency. The Bond work Index (BWi) results for Primary and Secondary ores were higher than reported in Campaign 1, while results obtained for the Cu40 zone samples indicated significant variability, ranging from moderate to low grindability.

1.4.2.2 Flotation testwork

During the first testwork campaign, the impact of primary grind size on recovery, flotation kinetics and concentrate grades was assessed for the three main ore types. The trends showed that copper recovery increased with the extent of grind. All three sulphide types returned maximum bulk flotation recoveries of 92-98% at a grind size of P_{80} 75 µm. The copper distribution by particle size in the flotation tailings indicated that some copper remained locked with gangue in the coarser size fractions, with improved liberation observed at finer grind sizes.

The bulk flotation tests showed high mass pulls (18-30%) at relatively low bulk concentrate grades (3.8-11.5% Cu).

A series of regrind tests was conducted to determine the optimum regrind size. From this work concentrate regrinds of P_{80} 16 µm for the Transitional, P_{80} 18 µm for the Secondary Sulphide and P_{80} 16 µm for the Primary Sulphide material were selected by Transmin for the locked cycle tests.

Locked cycle tests were performed on the Transitional, Primary, and Secondary ore composites from Phase 1. The Transitional and Primary Sulphide ore samples required two stages of upgrading to produce marketable concentrate, whereas the Secondary Sulphide ore required one stage of cleaning. A circuit consisting of two cleaning stages followed by a cleaner scavenger stage was recommended for the cleaner circuit. Concentrates containing 25% copper or more were produced in all of the locked cycle tests after two stages of cleaning.

Composite samples were assembled for the second phase of metallurgical testwork from drill cores not previously used for testwork, in order to test the variability of the different ore types.

Flash flotation tests were conducted with 67% solids slurry after grinding to P_{80} 300 µm. The results showed that flash flotation is a viable process option for Secondary ore, but not for Primary ore which would require regrinding and cleaner flotation to produce a saleable concentrate.

Bulk flotation tests were performed to optimise the reagent scheme. A reagent scheme consisting of promoter A-3477 (isobutyl dithiophosphate) with collector Z-11 (sodium isopropyl xanthate) was evaluated at different dosage rates and also combined with sodium sulphide addition. A standard primary grind of P_{80} 150 µm was selected for all tests, with stage dosing of Aerofloat 3477 promoter prior to milling and 10 g/t xanthate collector before flotation. The addition of sulphidiser did not result in significant grade improvement, and final recovery was negatively affected. Promoter at an addition rate of 25 g/t produced a lower mass pull, higher copper grade and faster recovery rates than the other levels tested.

Bulk flotation tests were conducted on the Secondary Sulphide ore to optimise the slurry pH and to review the impact of sodium sulphide addition. The tests showed an increase in recovery as the pH was increased from 8.6 to 11. A slurry pH of 10 produced an acceptable recovery with the lowest mass pull and highest copper grade. Sulphidiser addition resulted in grade and recovery improvements.

Cleaner tests were conducted using rougher concentrates generated with promoter (A3477) and collector (Z-11), with regrinding to P_{80} 37 µm. The Primary ore type generated a 15% copper concentrate after four minutes of flotation, whereas the Secondary ore type produced a 44% copper concentrate.

A third testwork campaign, aimed at optimising regrind and cleaner flotation performance, was completed in early 2009. The composite samples evaluated in this phase of testwork were similar to the second phase samples, although Primary ore samples from the Cu40 zone were also included.

The Primary ore results indicate that 25% copper concentrate can be produced at P_{80} 37 µm and 43 µm, without cyanide addition. The Secondary ore regrind tests showed higher copper concentrate grades were produced at P_{80} 49 µm and 57 µm, again without cyanide addition. Transmin selected P_{80} 37 µm and P_{80} 49 µm for the Primary and Secondary ores, respectively, as the basis for the additional variability and locked cycle tests performed during the third testwork campaign.

Variability batch flotation tests with three cleaning stages and locked cycle tests were performed on similar composites. In general, lower bulk flotation mass pulls were observed in this campaign compared to the first two testwork campaigns. A revised reagent scheme was utilised, with reduced collector (Z11) and promoter (A3477) dosages. Concentrates containing 25% copper or more were produced from the Transitional and Secondary ore samples in the locked cycle tests after two stages of cleaning. Both recovery and concentrate grade improved for the Secondary ore. The Primary ore tests did not consistently produce 25% copper concentrate after two stages of cleaning, but sodium cyanide was not added during the cleaning stages as had been the case during the first two campaigns. The primary ore concentrate grade was therefore lower than previously reported.

Higher overall recoveries were evident for the Primary samples, while the Transitional ore tests returned lower recoveries.

Copper recovery and concentrate grade is influenced by different parameters, but predictive values were estimated for financial model inputs based on head grade for the PFS. Precious metals recoveries are based on the average metal recovery observed from the trials. The conclusions are

obtained from the variability batch flotation and locked cycle tests performed in the third flotation testwork campaign. The parameters adopted for the PFS are shown in Table 1.9.

Table 1.9
Predictive metallurgical summary

Mineralisation Type	Metal	Recovery (%)	Concentrate Grade (% Cu)
Transitional	Copper	85	32
Secondary			
(Bornite-chalcocite)	Copper	min (5.3892Ln(CuT)+90.956), 96	45
Primary (chalcopyrite)	Copper	94	23.5
Precious metals	Gold	80	-
	Silver	80	-

The bornite-chalcocite (Secondary) ore is a major component of the sulphide resource and proportionately contains the largest metal content. The head grade-recovery relationship derived from the data was positive and has the following form:

$$\text{Recovery of CuT (\%)} = (5.3892 \text{ Ln (CuT)} + 90.956)$$

Note that the recovery is capped at 96%. The Transitional ore parameters are derived from the grade-recovery relationship obtained from data generated during the third campaign. The remainder of the parameters for the Primary and Secondary mineralisation samples are based on the average values observed after two stages of cleaning.

1.4.2.3 Magnetite testwork

The Mina Justa rougher/scavenger flotation tailings contain recoverable quantities of magnetite. Magnetic separation laboratory test campaigns were performed during the PFS to evaluate the feasibility of producing a saleable magnetite concentrate.

The first testwork campaign was conducted at Laboratorio Plenge in Peru using the flotation tailings produced from the initial testwork campaign. The sample domains tested included Transitional, Primary Sulphide and Secondary Sulphide ore. Single pass and triple pass cleaner separation tests were performed after bulk magnetic recovery. The iron grades in the magnetite concentrates were generally below typical market specification, while the levels of copper, sulphur, silica and alumina suggested that further processing would be required to liberate locked magnetite and separate gangue.

The second testwork campaign was performed at CIMM Chile using flotation tailings from the second flotation testwork campaign. The testwork included both dry and wet magnetic separation on five samples. The wet magnetic separation tests produced higher magnetite grades in the final concentrates than the dry tests, but were still below typical market specifications. The rougher and first-pass cleaning steps produced substantial grade improvements. Minimal grade improvements were obtained with further magnetic concentration stages. Metal recovery to the rougher concentrate improved with increasing degree of regrind.

The third testwork campaign was aimed at confirming the conceptual magnetite recovery circuit derived from the preliminary testwork. This testwork was conducted by Transmin on five samples at the Pontificia Universidad Catolica del Peru. The testwork further investigated the effect of regrind, with the results showing an increase in magnetite concentrate iron grade as the extent of regrinding increased.

The magnetite testwork confirmed that a high grade (63% iron) magnetite concentrate can be generated from the Mina Justa flotation tailings. Intermediate processing, which includes regrinding, slimes removal and finishing magnetic separation stages, is required to improve the magnetite concentrate iron grade for all the sample domains. Production of a magnetite concentrate containing more than 63% Fe is not possible without intermediate processing.

1.4.2.4 Tailings characterisation testwork

Knight Piésold performed tailings characterisation to assess the acid generation and neutralisation potential of tailings from the proposed flotation circuit. The results were used to formulate tailings disposal strategies.

On the basis of testwork results, bulk flotation tailings are expected to be net acid neutralising. Cleaner scavenger tailings (CST) or combination tailings, particularly from the Primary Sulphide ore zones, are potentially acid generating. Separate disposal of cleaner scavenger tails is required to reduce the footprint of that part of the tailings storage facility (TSF) that needs to be lined and covered with non-potentially acid generating waste rock upon closure.

1.5 PLANT DESIGN

Two plants have been designed to treat Mina Justa ores, namely:

- Leach plant treating Oxide ore to produce copper cathodes
- Flotation plant to treat Sulphide ore to produce copper concentrates.

1.5.1 Oxide ore plant

The Mina Justa Project utilises sulphuric acid leaching to extract copper from the Oxide ore. The leached copper is purified and upgraded by SX to provide a rich electrolyte to the EW plant, producing copper cathodes. The feed to the leaching process is prepared by crushing and screening to achieve a -8 mm product size. Throughput is 12 Mt/a.

The Oxide ore process flowsheet is depicted in Figure 1.8, and the plant layout in Figure 1.9. The flowsheet comprises:

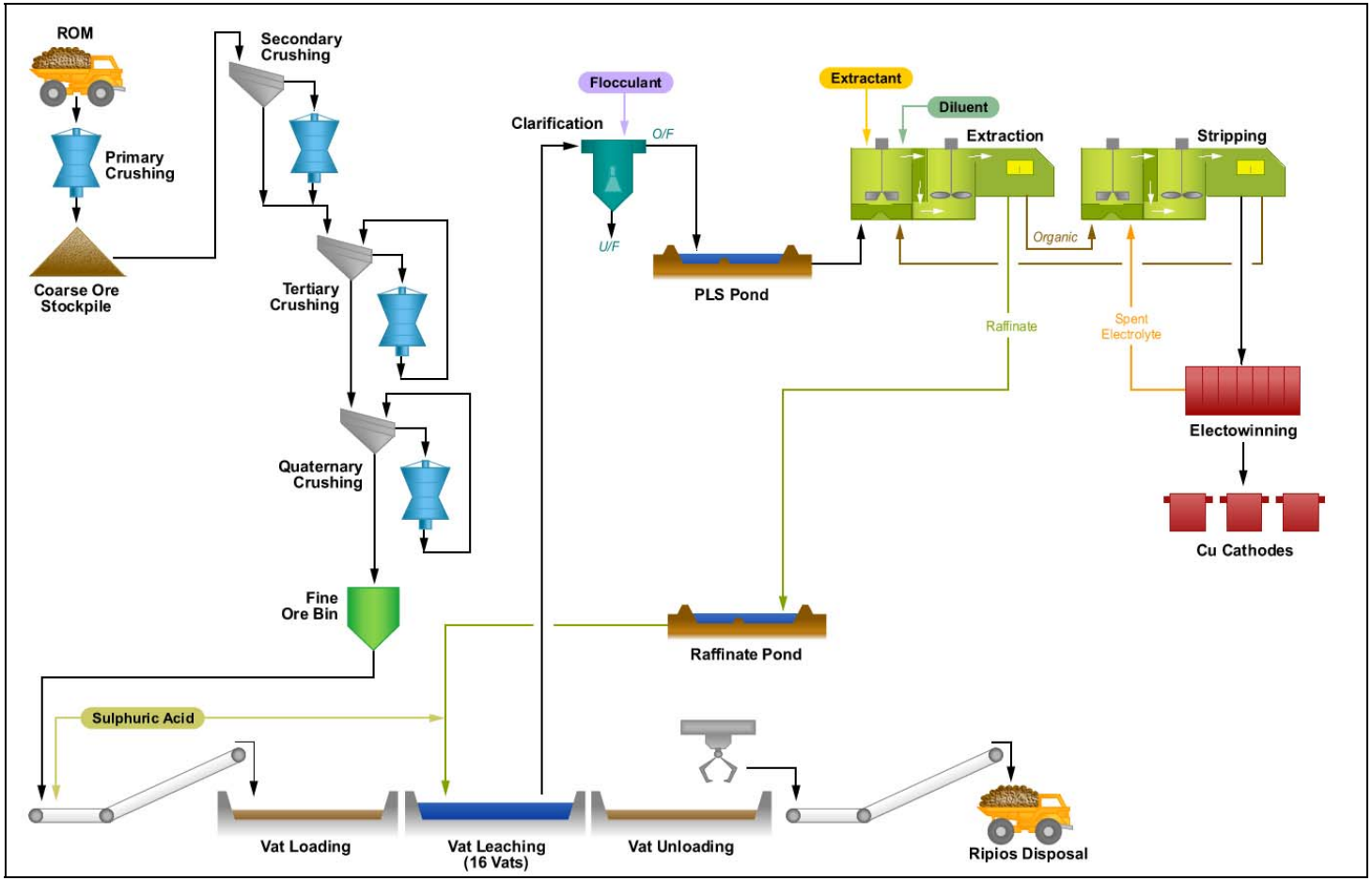
1.5.1.1 Four-stage crushing circuit

This is sized to handle 1712 t/h, assuming overall 80% availability. Ore is delivered by 220 t mine haul trucks tipping directly into the ROM bin, although provision is made for stockpiling and feeding by FEL. A coarse ore stockpile with a live capacity of 12 hours provides surge capacity between the primary and secondary crushing stages to account for mine trucking cycles and maintenance requirements.

To minimise capital cost, much of the crushing facility is exposed to the weather. An open type of building has been selected to minimise capital cost and permit easy access for mobile cranes for maintenance of equipment.

Dust generated throughout the crushing plant is controlled by a combination of dust suppression and dust collection systems. The truck tip-point is enclosed on three sides, and wetting sprays have been included, to suppress dust.

Figure 1.8
Mina Justa Oxide Circuit Flow Sheet



1.5.1.2 Vat Leaching

Crushed ore (-8 mm) is delivered to the fine ore bin, which provides a surge capacity of one hour. The crushed ore is drawn from the fine ore bin, and is sprayed with dilute sulphuric acid as it passes from one discharge conveyor to another to promote leaching. Nominal acid dosage is 15 kg/t ore. The acidified ore is transported by conveyor for loading into leaching vats.

The vats are essentially reinforced concrete shells, each measuring 30 m wide, 40.5 m in length and 7.6 m high, and capable of holding 12 800 t of ore (at the nominal density) for a six day leaching cycle. At any one time, 16 vats participate in the leaching process. However, 18 vats have been designed to allow for loading, unloading, filling, draining and maintenance. The vats are designed to be acid-resistant and are constructed to ensure that the leach solution is not lost due to leaks or seismic events.

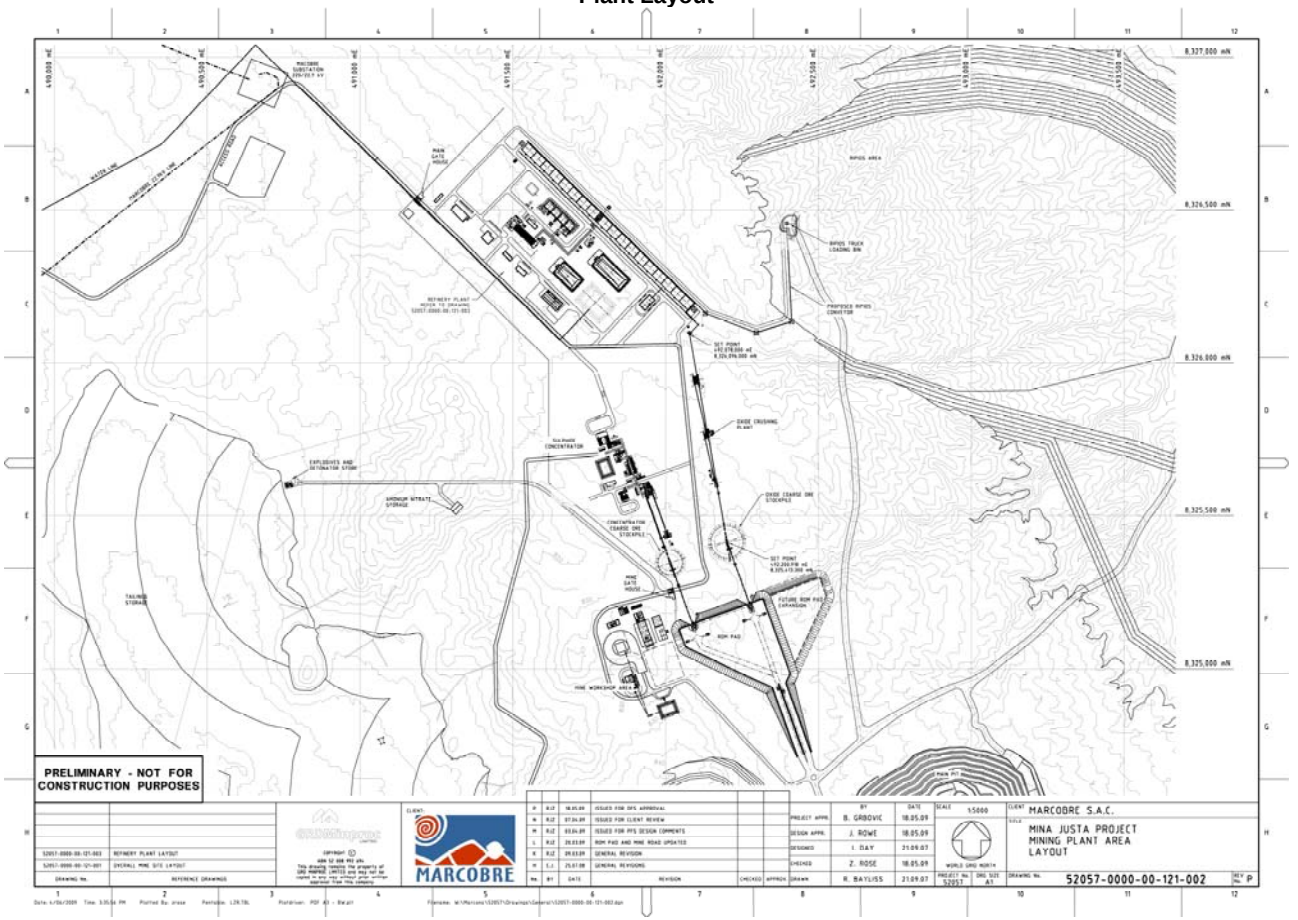
Acidified ore is loaded into a vat by means of a tripping conveyor until the vat is full, leaving 300 mm of freeboard in the vat. The vat is then flooded with a dilute sulphuric acid solution which is introduced through the base of the vat, under a filtration bed. The solution overflows from the top of the vat into a launder from which the solution is piped to the next vat or to a storage pond.

At the end of the leaching cycle, the remaining solution is drained from the vat and the moist waste solids (“ripios”) are removed by a clamshell grab, placed into a hopper and discharged onto a conveyor system for transfer to the ripios dump.

Solution management is designed as a counter-current system in order to maximise the copper content of the leach solution before treatment in the SX/EW plant. The highest tenor copper solution (pregnant leach solution or PLS) overflows from the vats containing the freshest ore. The PLS is clarified then stored in a covered holding pond before being pumped to the SX circuit.

The residual moisture in the ripios (approximately 11% by weight) is essentially raffinate and provides a bleed for impurities, so that no other bleed stream is required.

Figure 1.9
Plant Layout



1.5.1.3 Clarification

Pinned bed clarifiers have been specified because of their proven performance in removal of fines from copper leach solutions. Due to the acidic nature of the PLS solution, materials of construction include SAF2205 stainless steel for the clarifier feed tank and fibre-reinforced plastic (FRP) and SAF2205 internals for the clarifiers.

The clarified PLS solution gravitates to the PLS Pond, with the clarifier underflow solids being pumped to the vats.

1.5.1.4 Solution ponds

The PLS pond is 6 m deep and has been sized to contain 24 500 m^3 of solution. The pond provides over 24 hours of surge capacity, as the advance flow to SX is 980 m^3/h. This allows for some PLS blending and settlement of any remaining suspended solids. This pond is covered to reduce evaporation and prevent pick-up of wind-blown solids.

The raffinate pond is also 6 m deep and has been sized to contain 15 800 m^3 of solution. It is not covered. It also serves as an emergency reservoir in case one of the vats is drained by accident or intentionally in an emergency. Allowing for the liquid volume from one vat, the surge capacity ahead of the vats is over 10 hours.

Both ponds are lined with a double layer of HDPE membrane in order to avoid loss of valuable solution and prevent contamination of the environment. The dimensions also include an allowance to contain precipitation from a 100 year, 24 hour rainfall event.

1.5.1.5 Ripios

The ripios remaining after the leaching stage are removed from the vat by an unloading crane with a 22 m^3 clamshell grab.

The clamshell discharges the ripios into a hopper that feeds the ripios receiving conveyor. This material is then transported to the ripios area via three discharge conveyors. The last ripios conveyor feeds a bin, from which haul trucks are loaded for final disposal in the adjacent ripios dump.

1.5.1.6 Solvent extraction (SX)

The SX process involves the selective extraction of copper from the relatively dilute PLS to produce a high purity, high tenor copper sulphate solution suitable for the EW process. The SX system is composed of a single train that includes two extraction mixer-settlers treating the PLS. These are in series with a loaded organic wash mixer-settler and then an extraction mixer-settler, which produces the rich electrolyte feed to electrowinning. At the nominal PLS copper tenor of 7 to 8 g/L and pH at 1.9, copper recovery in SX is projected at 94%. Both extraction and strip units are expected to run in the organic continuous mode.

Solution is pumped from the PLS pond to the extraction circuit where it is contacted with the organic phase to extract copper from the aqueous phase. Loaded organic exiting the extraction circuit reports to the wash stage to remove entrained aqueous impurities such as iron, manganese and chloride.

Spent electrolyte from the EW process enters the strip circuit at the primary mix tank and is mixed with the loaded organic stream prior to passing through the strip settler for disengagement of the aqueous and organic phases. Copper-rich electrolyte flows by gravity to the "strong electrolyte" tank. Strong electrolyte contains minor amounts of particulate solids and entrained organic, which are removed prior to EW using CoMatrix dual media filters.

A reverse flow design is selected for the mixer/settler layout to minimise plant footprint and pipe run length. Primary and secondary single mix tanks are utilised for each stage. The settlers are constructed with concrete walls lined with FRP. The settler roofs are constructed of steel cladding with access ports for maintenance.

The SX area includes a series of floor drains that drain to a set of sumps/firetraps. This arrangement eliminates pooling of corrosive or combustible fluids in the bund.

Crud from various areas within the SX plant is pumped through the crud centrifuge that splits the crud into its three constituent phases (i.e. aqueous, organic and solid). The aqueous phase is returned to the SX circuit, whilst cleaned organic phase is either returned to the SX circuit or treated with activated clay. Contaminated solids are collected for separate disposal.

Fire protection is incorporated into the design of the SX plant with a philosophy of automatic detection and initiation of suppression measures. For this reason, the fire protection system for comprises:

- Foam suppression to the SX bunds, SX settlers and tanks containing organic.
- A fire detection system for the bund and inside of each of the vessels described above.

1.5.1.7 Electrowinning (EW)

The EW circuit utilises permanent cathode technology to produce LME Grade A cathode copper. EW is conducted using a total of 122 cells at a nominal current density of 320 A/m^2. Copper plating is continuous over a period of six days before the cathodes are removed and processed for dispatch.

The copper-rich electrolyte ("strong electrolyte") passes to the EW circuit where copper is recovered in the form of copper cathodes. Electrolyte that has been depleted of copper during the EW process ("spent electrolyte") is recycled to the strip stage in the SX circuit.

Polishing cells receive strong electrolyte and act as organic entrainment protection for the commercial cells. Electrolyte overflowing the polishing cells flows to the electrolyte circulation tank and mixes with the spent electrolyte from the commercial cells to result in a stream of circulating electrolyte.

The EW cells are of monolithic polymer concrete construction, comprising vinyl ester resin mixed with aggregate. Electrolyte is circulated throughout the cell via a PVC manifold mounted at the bottom of each cell. Holes drilled into the PVC manifold allow electrolyte to pass between the electrodes in the

cell. There are 26 polishing cells and 96 commercial cells. Each cell contains 69 cathodes (stainless steel blanks) and 70 anodes. Cathode quality is expected to be the same between the polishing and commercial cells. The polishing cells are generally viewed as insurance against contaminating the entire tank-house if organic breakthrough occurs in the filters.

Copper plating onto the stainless steel blanks is continuous over a period of approximately six days before the cathodes are removed for harvesting of the copper. Copper is removed from the cathodes by an automated cathode-stripping machine.

Approximately 3 t of copper sheets are accumulated before the bundle is sampled, strapped and transferred by forklift to a dedicated storage area prior to dispatch.

The EW cells are housed in a fully enclosed building to provide protection from climatic conditions (e.g. dust) and provide an acceptable working environment for the crane and stripping-machine operators.

Primary acid mist suppression is by a layer of polyolefin prills, which floats on the cell surface and disperses the bubbles of acid mist as they reach the solution line of the cell. A forced cross-flow ventilation system provides secondary mist suppression by removing acid mist from the building.

1.5.1.8 Reagents

Sulphuric Acid

Sulphuric acid (98%) is delivered by road tankers to the sulphuric acid unloading area. Four unloading stations have been provided to transfer the sulphuric acid into the two storage tanks. Each tank contains a live volume of 3187 m^3, sufficient to store a 7 day supply on site.

The tanks are sited within a HDPE-lined earth bund capable of containing 110% of the entire contents of sulphuric acid stored on site.

Flocculent

A non-ionic flocculent is dosed to the clarifier feed well. Flocculent is delivered to site by road on pallets containing 25 kg bags, and prepared with fresh water in a batching plant near the clarifier.

Extractant

Extractant (LIX984 or Acorga M5640) at a concentration of 25% by volume is used in the SX process to extract copper from the PLS.

Extractant is delivered to site in 1 m^3 intermediate bulk containers (IBCs), off-loaded by forklift, and stored in a covered shed. The containers are moved to the solvent extraction area as required. Extractant is added (by gravity) to the SX circuit on a demand basis.

Diluent

High flash-point diluent (Shelsol 2046 or equivalent) is delivered to site by road tanker and off-loaded into the diluent storage tank, which has a storage capacity equivalent to 45 days.

Diluent is transferred to the SX circuit and the crud treatment area on a demand basis, using a single positive displacement pump.

Guar

Guar is a high molecular weight organic polymer that acts as a smoothing agent for the deposition of copper during the EW process, thereby enhancing the appearance of the final copper product.

Guar is received as powder in 25 kg bags, with storage on-site equivalent to 28 days of usage. The guar is mixed in an automated system and the solution is pumped to the EW circuit.

Cobalt Sulphate

Cobalt sulphate is added to the EW circuit to maintain a cobalt concentration of 180 ppm, in order to enhance the stability of the lead anode coating.

The cobalt sulphate reagent is received in 25 kg bags, with storage on-site equivalent to 28 days of usage. The cobalt sulphate is mixed in a small mixing tank and dosed to the EW circuit as required.

1.5.1.9 Services

Raw Water

A 6 m deep raw water pond with 16 000 m^3 capacity is sited in the plant area to receive water from the Jahuay borefield for onward distribution around the site for process water, fire-water, dust control, camp, mine water and other purposes.

Fire-water

The raw water pond also serves as the source of fire-water, with the pond and pumps configured to ensure a minimum amount of fire-water is always available in the pond.

The fire-water pump set comprises an electrically powered main centrifugal pump, a diesel powered pump and an electrically powered jockey pump. The fire-water system pressure is maintained using the jockey pump, thereby preventing premature starting of the main fire-water pump.

Potable Water

Raw water is treated through the water treatment plant to produce potable quality water to be used for safety shower, drinking water and ablution facilities. The water treatment plant uses chlorination to destroy any harmful bacteria present. The resultant potable quality water is transferred to the 80 m^3 potable water storage tank.

Plant and Instrument Air

Plant air at 750 kPa is provided from the two main plant air compressors and stored in the plant air receiver, from where it is reticulated to the plant air utility stations. A separate portable air compressor is provided for use in the crusher plant areas.

A stream of plant air is diverted through a pair of air filters and fed to a duty/standby desiccant air drier to remove moisture from the plant air and generate instrument quality air. Instrument air is reticulated to points of demand.

1.5.2 Sulphide ore plant

The concentrator and related facilities have been designed and costed to a PFS standard.

The overall process flowsheet for the Sulphide ore is depicted in Figure 1.10 and the concentrator facilities are depicted in Figure 1.11.

1.5.2.1 Comminution

The comminution circuit is designed to treat 5 Mt/a of sulphide ore to produce a product size of P_{80} 150 µm. It comprises:

- Primary crushing circuit, allowing direct feeding by haul trucks. The primary gyratory crusher treats an average of 951 t/h, producing 5 Mt/a with an availability of 60%. A coarse ore stockpile provides 12 hours of surge capacity between the crushing and milling stages.

  To counter dust, the truck tip-point is enclosed on three sides, and wetting sprays and dust collection systems have been included.
- Primary Grinding and Pebble Crushing circuit, consisting of an open circuit semi-autogenous (SAG) mill with a pebble crushing circuit. The SAG mill feed conveyor transports crushed material reclaimed from the crushed ore stockpile to the SAG mill.

  Oversize pebbles from the SAG mill are transferred to the pebble crusher for size reduction, and the crushed pebbles are returned to the SAG mill via the SAG mill feed conveyor.
- Secondary Grinding and Classification: The secondary grinding circuit consists of a ball mill in closed circuit with a cyclone cluster. The circuit targets P_{80} 150 µm.

Figure 1.10
Mina Justa Sulphide Circuit Flowsheet

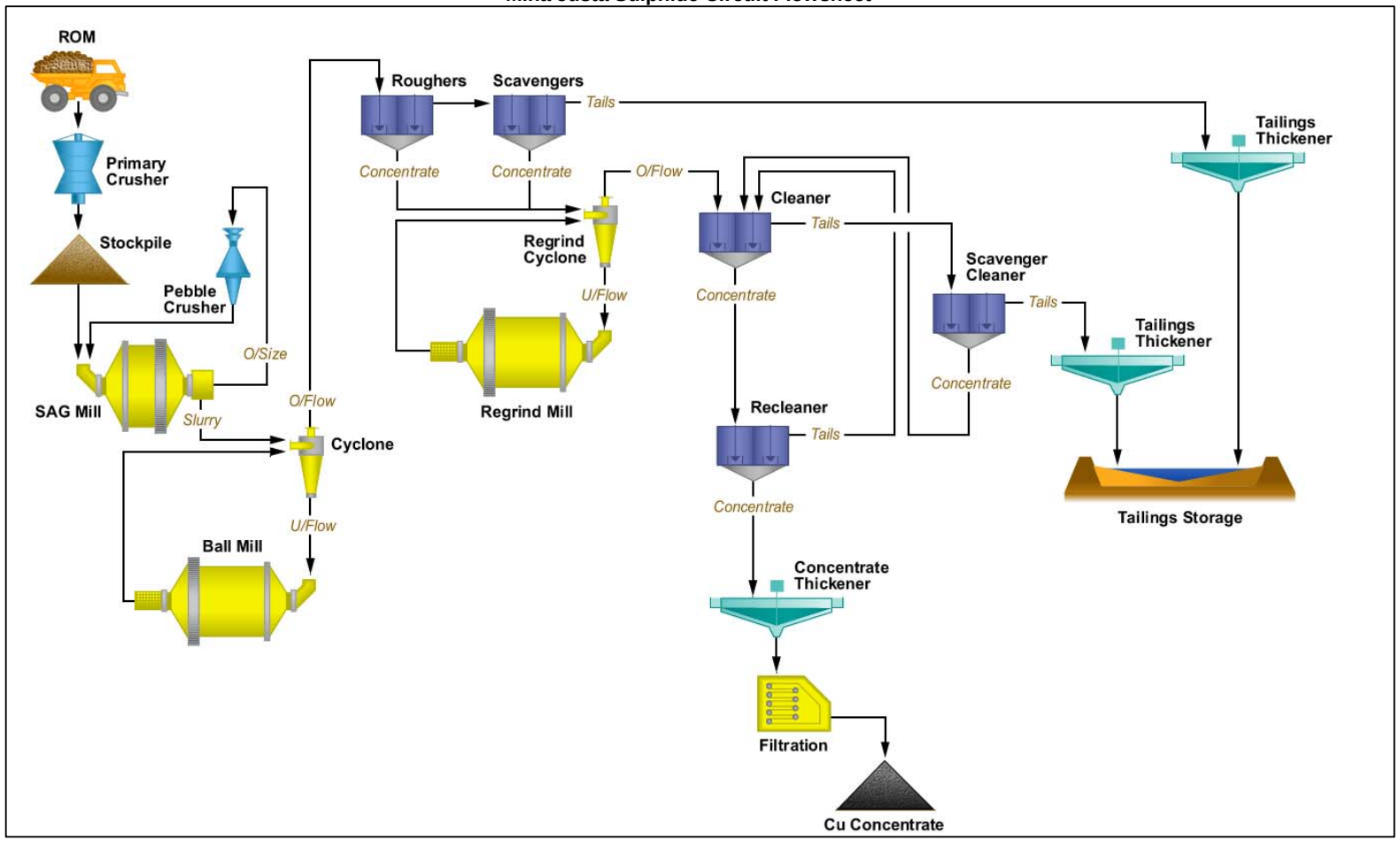
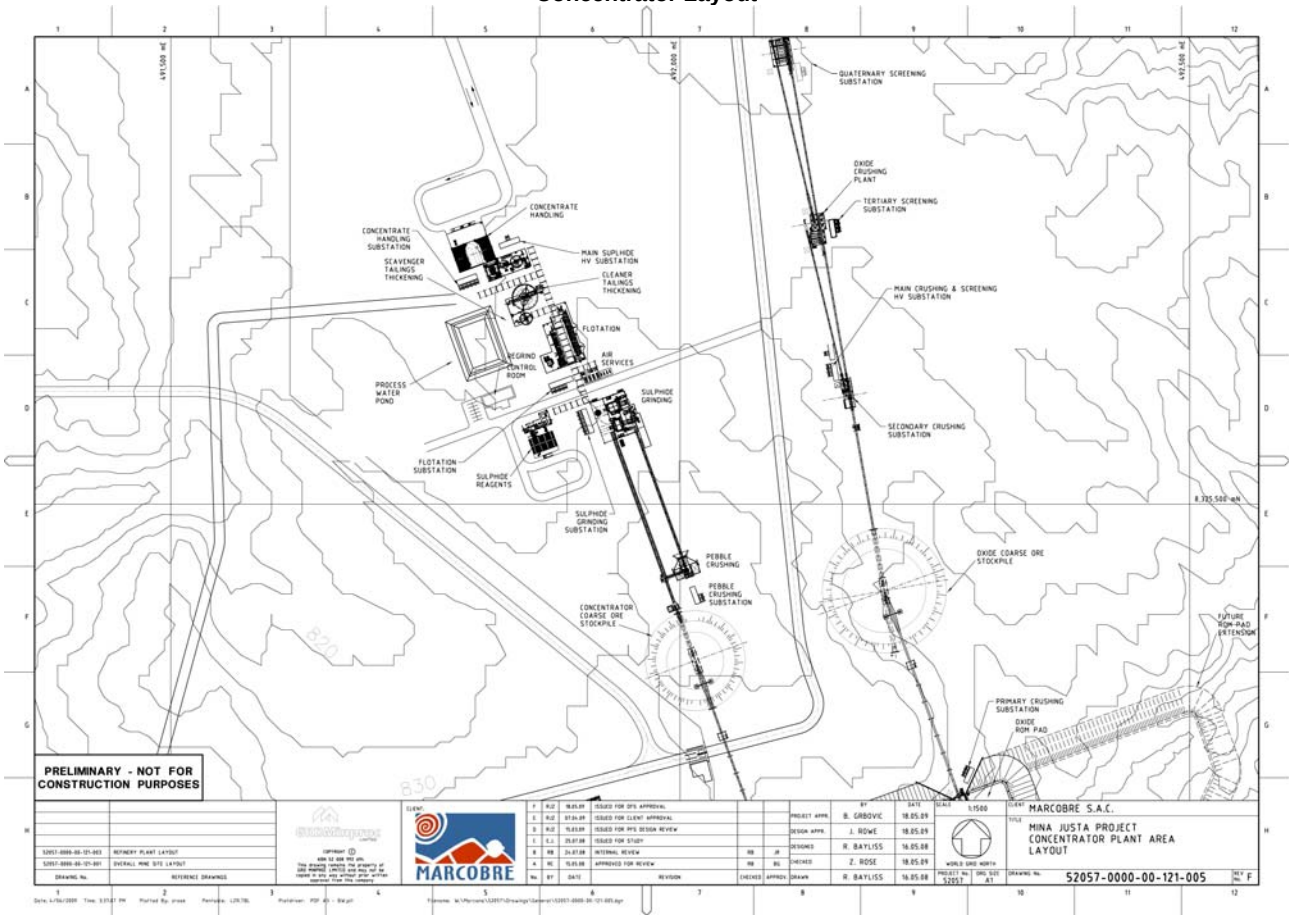


Figure 1.11
Concentrator Layout



1.5.2.2 Flotation

The flotation circuit comprises bulk flotation, concentrate regrind, cleaner flotation and on-stream analysis.

- Bulk flotation: cyclone overflow from the secondary grinding circuit, at a pulp density of 35% solids and pH of 9, reports to the rougher/scavenger circuit. The rougher flotation stage consists of two 70 m^3 tank cells, and the scavenger flotation stage consists of four 70 m^3 tank cells. The total installed residence time for the rougher-scavenger flotation circuit is 20 minutes.

  Rougher/scavenger flotation concentrates are pumped to the regrind circuit for further grinding.
- Concentrate regrind: the rougher and scavenger concentrates report to the regrind ball mill circuit for fine grinding. A P_{80} in regrind cyclone overflow of approximately 49 µm is achieved.
- Cleaner/recleaner flotation: selective flotation is achieved in the cleaner flotation circuit through the addition of collector and frother, and by increasing the pulp pH to 11. Cleaner flotation is carried out in four 38m^3 cells with a total nominal residence time of 10 minutes.

  Cleaner concentrate is transferred for further cleaning in the recleaner circuit. The recleaners consist of three 16 m^3 u-shaped flotation cells with a total nominal residence time of 10 minutes. The recleaner concentrate is pumped to the concentrate handling area.

  The cleaner flotation tailings flow to cleaner scavenger flotation. The cleaner scavengers consist of three 38 m^3 u-shaped flotation cells with a total nominal residence time of 10 minutes. The cleaner scavenger flotation tailings are transferred to the tailings disposal circuit.
- Sampling and analysis: eight sample streams are collected for on-line control of the flotation circuit. Various in-stream samplers collect samples and direct them to a multiple stream analyser. The analysis is undertaken using an XRF analyser.

  Rougher feed and regrind overflow samples pass through an analyser for particle size determination.

1.5.2.3 Concentrate handling

The recleaner concentrate is screened to remove debris from the slurry, in order to protect the thickener and downstream filter operation. Thickening of the concentrates is conducted using a 15 m diameter high-rate thickener to produce a product at 65% solids.

The thickened stream is transferred to the filter feed-tank, which provides a storage capacity equivalent to 12 hours. The concentrate solids are dewatered by a pressure filter. The filter discharges moist concentrate directly onto a storage slab below the filter. The filtrate returns to the concentrate thickener.

Concentrate is transferred from the stockpile into a storage shed by a FEL, which is also used to load road trucks for shipment.

1.5.2.4 Tailings thickening and disposal

Two tailings streams are produced by the concentrator, the cleaner scavenger tailings (CST) stream with potential for acid generation, and the rougher scavenger (RST) tailings stream with low acid generation potential. The two streams are disposed of separately, as follows:

- Rougher scavenger tailings

  The RST are pumped to a high-rate thickener. Thickener overflow discharges to the process water pond. The thickener underflow stream, at 60% solids, is pumped to the RST section of the TSF.

- Cleaner scavenger tailings

  The CST report to the CST thickener. Thickener overflow discharges to the process water pond, while the thickener underflow stream, at 60% solids, is pumped to the CST section of the TSF.

1.5.2.5 Reagents

Collector (Sodium Isopropyl Xanthate)
Sodium isopropyl xanthate (SIPX) is the collector used in the flotation process. SIPX is delivered to site in 1 t bulka bags, with storage being provided for 20 bags. A 2 t monorail hoist lifts the bags into a bag splitter chute above a 6 m^3 agitated collector mixing tank. A 20% solution is prepared with raw water and pumped to the collector header tank for distribution.

Promoter (Aerofloat 3477)
Aerofloat 3477 promoter is delivered as a liquid in 210 L drums, with storage being provided for 75 drums. The promoter is transferred to a 1 m^3 promoter storage tank, and is dosed into the flotation circuit by dedicated metering pumps.

Frother (Dow 250)
Dow 250 is received as liquid in 210 L drums and storage is provided for 75 drums. The frother is transferred from the drums to a 1 m^3 frother storage tank, from where it is dosed in the flotation circuit with dedicated metering pumps.

pH Modifier (Lime)
Hydrated lime (85% $Ca(OH)_2$) is delivered to site as a bulk solid and stored in a 60 t hopper. Lime solution is prepared with raw water and transferred to a 20 m^3 agitated lime storage tank prior to distribution to the plant through a ring main.

Flocculent
Flocculent is transported to site as a solid in 25 kg bags. Storage is provided on-site for 400 bags. Flocculent is mixed in an automated system and is made up to a concentration of 0.3% w/w. Flocculent solution is delivered to the respective thickeners using dedicated variable-speed metering pumps. Flocculent solution is diluted to 0.03% w/w prior to dosage.

Sodium Sulphide
Sodium sulphide is delivered to site in 1 t bulka bags, with storage allowance for 15 t in a secured area on-site. A 2 t monorail hoist lifts the bulka bags into a bag splitter chute above a 6 m^3 agitated mixing tank. Sodium sulphide is made up to a 15% solution concentration with raw water before being

transferred to a 10 m^3 storage tank sited in a concrete containment area capable of storing the entire contents of the tank in case of an emergency. The solution is metered to the rougher flotation feed box.

1.5.2.6 Services

Raw water
Sulphide plant raw water requirements are provided from the Oxide plant raw water pond. A set of raw water pumps is installed at the pond to supply the Sulphide plant with its raw water requirements.

Fire-water and potable water
Fire-water and potable water for the Sulphide circuit are supplied from the Oxide plant system as described in subsection 1.5.1.9.

Process water
The process water pond has a capacity of 4500 m^3. Provision exists to transfer raw water into the process water pond to maintain the level at a predetermined value. Process water is distributed to the process plant areas as required.

Plant and instrument air
As it would be impractical to pipe the compressed air over the distance between the two facilities, the concentrator has a dedicated plant and instrument air system, similar to that described for the Oxide plant.

1.6 INFRASTRUCTURE

1.6.1 Access roads

A 14.8 km access road has been designed, linking National Route 026 (connecting the Municipality of San Juan de Marcona to the Panamericana Sur highway) to the plant site and accommodation camp (Figure 1.12). This road has been designed for sustained, long-term use, including adequate foundations and a tarmac surface.

1.6.2 Internal roads

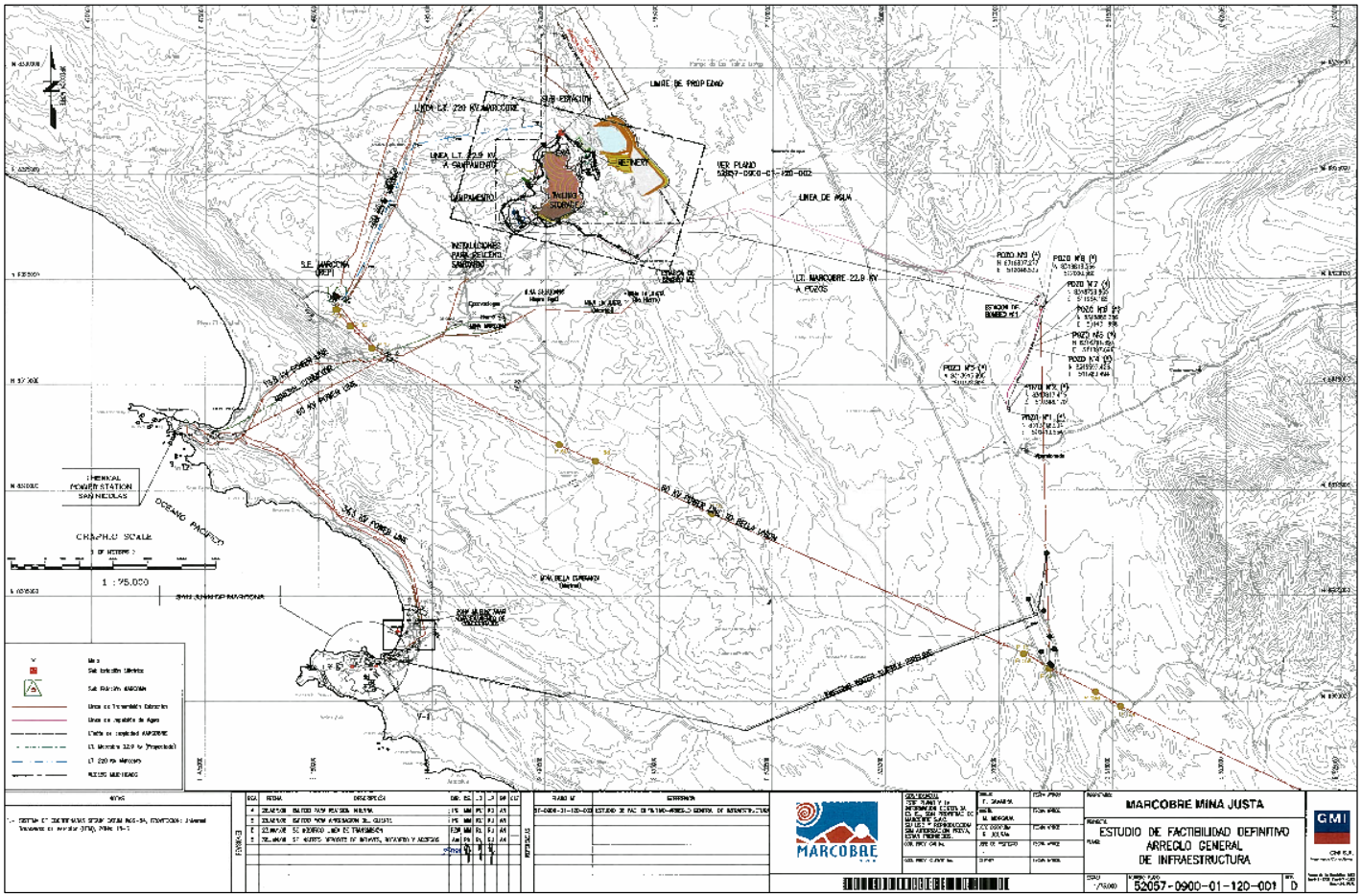
An internal road network is required to provide access from the main offices to the mine, waste rock and ripios dumps, stockpiles, crushing and other plant areas, and the TSF. These roads will be surfaced with crushed rock and maintained by watering (to control dust), grading and periodic resurfacing. A lighting system is provided for internal roads. Extensive road signage is considered very important for safety reasons, particularly where mine haul trucks are operating.

1.6.3 Buildings

The list of the buildings is comprehensive, including plant buildings, offices, reagent storage, warehousing, electrical buildings and sub-stations, workshops, changerooms, water supply pump stations and security buildings.

Figure 1.12

Mina Justa Project Plan



1.6.4 Construction and accommodation camp

A construction camp is situated on the site, some 4.4 km southwest of the plant. This is designed to hold 990 persons during construction, and 300 people during operations.

The camp will be equipped with accommodation, kitchen and eating areas, medical, security, communications and recreation facilities. Temporary construction accommodation will be in pre-fabricated transportable units, whereas permanent accommodation and other facilities to be used during operations will be of a more permanent nature, constructed of masonry and structural steel with roofing and siding, with concrete slabs and foundations.

1.6.5 Waste management

1.6.5.1 Sewage and waste water treatment

Sewage treatment plants meeting World Health Organisation standards will be installed at the construction camp, administration office, plant site and mine workshop areas.

Water treatment plants will be installed in each of the above areas to treat wash-down and other "grey" water, which will then be re-used for dust control, plant process water and vegetation programmes.

1.6.5.2 Other inert residual waste

A policy will be established to minimise usage and maximise recycling of domestic wastes such as paper, aluminium, glass, plastics, etc. Collection will be undertaken regularly, with separated materials transferred to reprocessors off site. A registered contractor will be used to undertake these transfers.

Organic wastes will be collected and composted in a suitable facility on site.

Used oil filters and vehicle and plant parts will be collected and taken to the central storage facility, recyclables separated, and the inert residual wastes landfilled in a specially designed, secure, registered, sanitary landfill site on the property.

Management of other inert residual wastes will be by burial in the sanitary landfill site on the property.

1.6.5.3 Dangerous waste management

Dangerous wastes will be collected and stored briefly at the point of generation, before being transferred to the central storage facility. Those materials that can be rendered inert will be treated at the site and then landfilled in the sanitary landfill, while others will be sent off-site either for treatment and re-use, or for permanent disposal in a compliant dangerous materials storage site. Registered transport and disposal companies will be used for this purpose.

Crud

Crud from the SX circuits will be processed to separate the material into its constituent phases. The separated aqueous and organic phases will be returned to the main circuit and re-used, while contaminated solids from the crud treatment circuit will be disposed of in a registered facility off-site.

Anode sludge

Anode sludge from EW cells is generally recognised as a toxic material, and stringent handling and disposal procedures are followed. Anode sludge is typically sent to a smelter for treatment, and will not be stored permanently on-site.

Other dangerous wastes

Products such as batteries, fluorescent products, welding rods, paint and other dangerous wastes will be transfer to and deposed of in a registered Dangerous Waste facility off-site.

1.6.6 Water supply system

1.6.6.1 Project water balance

A site-wide water balance has been developed by Knight Piésold to quantify the amount of make-up water required to sustain operations, as follows:

- **Leaching operations:** outside source make-up required to sustain operations for this facility is predicted to be roughly 140 m^3/hr for the life of the facility, taking account of moisture lost to the ripios dump.
- **Sulphide plant operations:** principal water losses from the Sulphide plant come from tailings disposal. A reclaim system is deemed to be impractical, and make-up water requirements are estimated to be roughly 375 m^3/hr.
- **Other areas:** approximately 45 m^3/hr of "other" make-up water is required for the operation of the camp and for water trucks used for dust suppression on the roads.
- **Total operational requirements:** the total make-up requirement is roughly 186 m^3/hr from year 1 through Q7 in year 2 (Oxides). From years 2 through 10 the requirement increases to 589 m^3/hr (Oxides + Sulphides), but it decreases for years 10 through 12 to 420 m^3/hr (Sulphides).
- **Closure and Reclamation Water Balance:** climatological data analysis indicates that there is no net accumulation of precipitation on the site, but the tailing storage facilities has capacity to store run-off associated with the PMP with 1 m of freeboard. Consequently, there is no need to regrade or construct a spillway in either of the facilities to divert run-off from large precipitation events.

  During closure, remaining water from the process ponds will be pumped to the tailings dam for evaporation.

1.6.6.2 Hydrological testwork and studies

Ground Water International SAC (GWI), now known as MWH Peru S.A., conducted a field investigation of the Jahuay and Lomas aquifers (Figure 1.12) to address the supply requirements. The program was designed to complement a previous study undertaken by Vector (2006) and included:

- Climatic water balance
- Test well drilling and installation in the Upper Jahuay and Lomas aquifers
- Numerical modelling and other analysis.

Lomas aquifer

Test pumping of the Lomas test bore indicated moderate supply potential, with a long-term safe yield estimated at about 5 L/s, with water quality suitable for the use intended. There is some potential to affect neighbouring wells in the area, should significant water withdrawal occur. Overall, however, this aquifer is considered to have a good water supply potential, and ranks as a potential back-up water source.

Jahuay aquifer

Test-production well MPA-2 was installed in the upper Jahuay aquifer, to a depth of 245 m. An 8" telescopic screen assembly was installed between depths of 220 and 226 m. The well has a static water level at approximately 83 m, and a long-term safe yield estimated at 34 L/s (not considering potential well interference from future neighbouring wells).

The water is fresh, pH-neutral with low TDS content and slightly elevated levels of iron (0.3 – 0.6 mg/l).

The average annual water surplus for the Jahuay basin is estimated in the range of 40 to 140 L/s. Numerical modelling indicates that withdrawals in the order of those required for the Mina Justa Project are sustainable in the medium-term (i.e. <50 year), but total combined annual withdrawals from the aquifer by Marcobre, Marcona and Shougang will exceed the average annual water surplus for the basin. Following closure of the Mina Justa operation, water levels will gradually recover to pre-operation levels.

Simulations indicate that pumping of the Marcobre wellfield should not adversely affect the Shougang/Marcona field, located 9.6 km south of the southernmost Mina Justa well.

The Upper Jahuay aquifer is therefore the preferred source, in view of its relative proximity, higher yield, better water quality and lower potential to interfere with other users.

1.6.6.3 Water supply system

The water supply system includes the following:

- Borefield and water collection system, which consists of nine water wells located in the Jahuay aquifer, four in Stage 1, and five more in Stage 2, installed to produce at an average rate of 25 L/s (90 m^3/h) each. Wells will be activated as the water requirement for the mine increases. To cover the maximum requirement during Stage 1, three wells will be operating and one on stand-by. For Stage 2, seven wells will be operating and two on stand-by. For stage 3, all wells will remain in commission with operating hours adjusted to suit the reduced demand.
- Water transfer system, which consists of the pump stations and transfer pipelines from the reception tank in the Jahuay aquifer to the pond at the mine site.
  - Two pump stations are required, one being located at the wellfield collection tank, 31 km from the plant site, and the second 13 km from the plant site.
  - During the initial phase of operation two pumps will be installed in the stations, one operating and other on stand-by. In the second phase, a third pump is installed, resulting in one pump used as a stand-by and the other two in operation.

  - The pipeline is constructed of carbon steel changing to HDPE in sections further away from the pumping stations where pressure decreases. Pipeline diameter is 14" between the first and second pump stations, reducing to 12" between Pump station No.2 and the mine site. The steel pipes have concrete sleepers installed every 6 m and anchor blocks whenever the direction changes. HDPE pipes will be restrained with earth anchors.
  - Fire extinguishers are installed at both pump stations, and fire detection signals are transferred to alarm control panels in the central control room.
- The water is discharged into the raw water pond located at the plant site, from where will be distributed to internal facilities.
- Electrical distribution system, consisting of 22.9 kV power distribution line from Mina Justa to the borefield and distribution to each pump station.
- Controls and instrumentation are placed at each pump station and at the borefield location, tied back via optical fibre. Each sector has a switch that connects the corresponding area with the control room. Instrumentation includes flowmeters, optical fibre communications and alarm systems.

1.6.7 Power supply system

1.6.7.1 Power supply to site

Marcobre will enter into a long term power supply agreement with a generator to deliver power to the Red de Energía del Perú (REP) Marcona sub-station on the main distribution grid.

A dedicated 15 km 220 kV overhead power line will be constructed to connect from the grid, terminating at the plant's HV switchyard on the 220 kV bus. A 220/22.9 kV transformer will be installed to supply the plant, and associated HV switchgear feeding a 22.9 kV switchboard located indoors at the main plant substation.

1.6.7.2 22.9 kV main switchboard

The 22.9 kV main switchboard is provided with a single incomer bay. The 22.9 kV main switchboard is provided with gas-insulated switchgear bays for distribution of 22.9 kV to plant load centres, power factor correction and HV motors.

1.6.7.3 Distribution

Power is distributed from the 22.9 kV main substation switchboard to major plant loads via an overhead line to the boundary of the process plants. Within the process plants, power cables are used. Plant load centres have varying secondary voltages supplied by step-down power transformers adjacent to each of the load centres.

Power supply
Power demand is as follows:

- Oxide plant electrical load: total connected load for the Oxide plant and mine infrastructure is estimated as 36 394 kW, and total running load is 30 124 kW. Predicted maximum demand is 33 411 kVA.
- Sulphide plant electrical load: total connected load is 29 061 kW, with total running load of 22 861 kW. Predicted maximum demand is 24 211 kVA.
- Other electrical loads: including camp, lighting and transfer station pumping, total power demand for other items is estimated to be 2551 kVA.

Power reticulation

Twelve substations service the Oxide crushing, screening and process plant facilities. The Sulphide plant requires an additional seven substations. Power factor correction (6 MVAr) is provided. This results in the overall power factors and maximum demands as follows:

- Oxide only – p.f. 0.9, maximum demand (with p.f. corrected) 33 411 kVA.
- Sulphide only – p.f. 0.95, maximum demand (with p.f. corrected) 24 211 kVA.
- Combined Oxide and Sulphide – p.f. 0.88, maximum demand (with p.f. corrected) 60 160 kVA.

HV switchboards

The Oxide plant HV switchboards are located within the Oxide crushing and screening area HV substation and the Oxide process area HV substation.

For the Sulphide plant, power is distributed from the 22.9 kV main substation switchboard to major plant loads via an overhead line to the boundary of the Sulphide plant. The Sulphide plant HV switchboard is located within the Sulphide plant HV substation. It handles the Sulphide primary crusher as well.

Emergency Generation

The following emergency power requirement has been identified:

- **Oxide Plant**: total 2321 kW, consisting of 2027 kW fixed loads (building loads, plant lighting and small power) and 294 kW process loads.
- **Sulphide Plant**: total 922 kW, consisting of 690 kW fixed loads (building loads, plant lighting and small power) and 232 kW process loads.

1.6.7.4 Control system

The plant is provided with a process control system (PCS) of moderate level of control complexity. The plant is designed to be operated primarily from the central control room (CCR) located adjacent to the EW building. The CCR contains four operating stations and an engineering workstation. Local field operator stations provide complete control room type information to the operators, but allow interaction from the field operators on a secured basis.

A CCR is provided for the Sulphide plant and is located near the grinding building.

1.6.8 Waste disposal

1.6.8.1 Mine and ripios waste dumps

Designs have been prepared for the two mine waste rock dumps (the Main Mina Justa and Magnetite Manto waste dumps), the ripios dump and the low grade stockpile, taking account of the physical and geochemical stability of the structures and the correct land use following closure.

The estimated amount of waste rock to be generated by the Project is approximately 402.5 Mt, of which 383 Mt will be placed in the Main waste dump, 14 Mt in the Magnetite Manto waste dump and the remaining 5.5 Mt (non-PAG material) will be used for construction of the tailings dam. The ripios dump has been designed with a capacity of approximately 114 Mt, and the low grade stockpile has a capacity of 20 Mt.

Geotechnical and geochemical investigations were conducted by Knight Piésold to characterization the foundations, rock waste and ripios materials, from which it was concluded:

- Geological units underlying the waste dumps and stockpile are adequate for foundations. However, aeolian material located at the toe of the final mine waste dump will be removed to improve stability.
- The water table lies some 400 m below the Main waste dump and 500 m below the Magnetite Manto waste dump and low grade stockpile.

Ripios and mine waste dump design

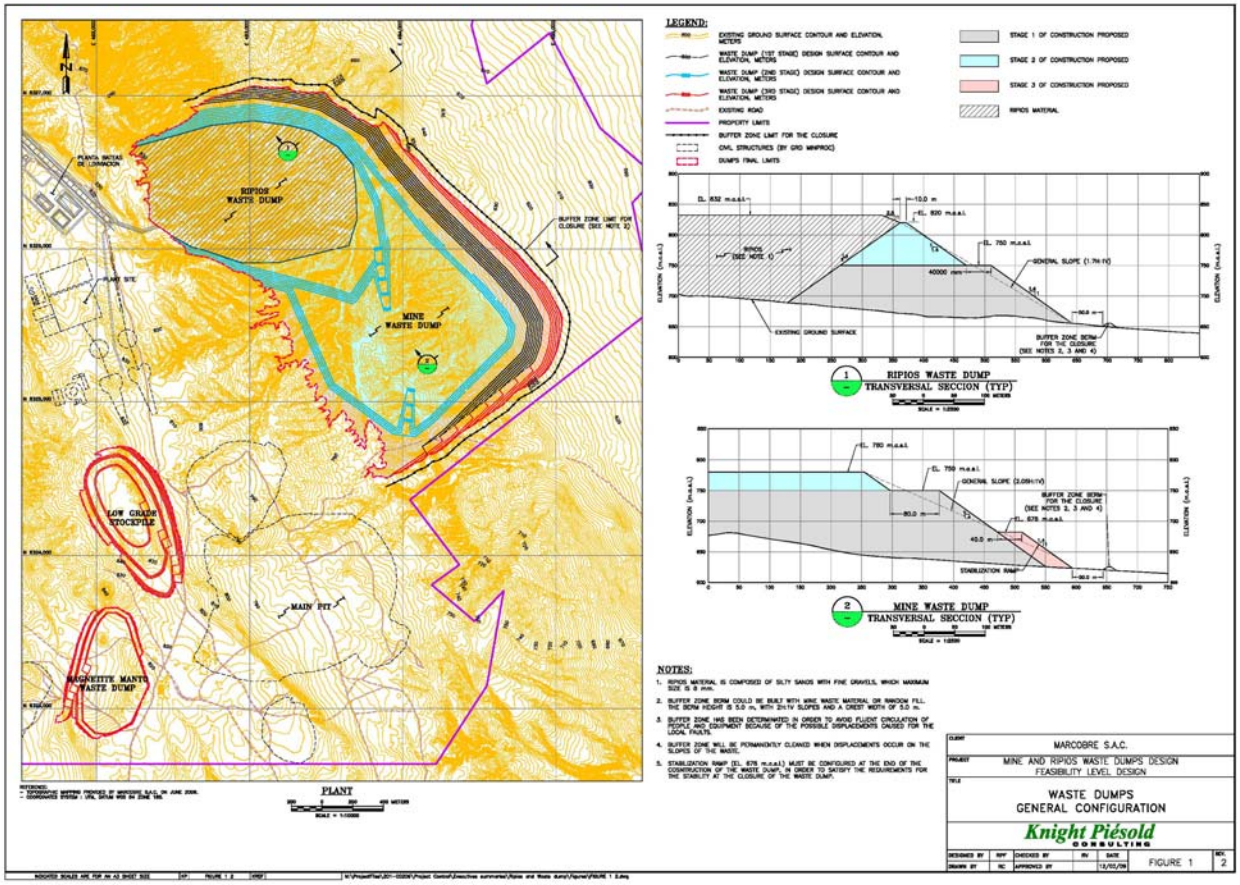
The design of the Main waste dump takes account of placement of the ripios material within its boundary, the mine waste acting as a dyke for the ripios material in the north area (Figure 1.13). Knight Piésold undertook stability analysis of proposed slopes, the results of which indicate acceptable static factors of safety (FoS) and post-seismic behaviour. Stability analyses for the ripios dump during operation indicated that a security distance of approximately 15 to 25 m is necessary as some minor (superficial) slope failures can be expected.

As part of the design of the Main waste dump, seepage analyses were undertaken. The results indicate that the saturation degree of the soil increases 5% in the first 5 to 10 m starting from the surface as consequence of a storm event (PMP). No variation on the saturation degrees is observed below those levels. Consequently, the potential to generate seepage through the dumps is limited, and any potential for flow to the groundwater table is low.

In order to monitor slope movements on the Main waste dump, marker points will be installed during and after operations. Underdrain systems will be installed at the base of the ripios dump to monitor potential seepage from the ripios area. Potential flow will be conducted to a water monitoring station.

PAG mine waste rock is estimated at approximately 15 Mt. This material will be encapsulated by non-PAG material in the Main waste dump, separated from the ripios material and from the final slopes of the waste dump. At closure, the PAG waste rock will be covered with a 1 m layer of non-PAG material to avoid potential acid dust generation and dermal contact. When the ultimate dump configuration has been reached, a security berm will be constructed at about 50 m from the final toe of the Main waste dump as a buffer zone.

Figure 1.13
Ripios and Mine Waste Dumps



Magnetite Manto waste dump and low grade stockpile design

The Magnetite Manto waste dump and the low grade stockpile will each be constructed in three layers.

For both structures the bench slopes and the overall slopes will be 1.4H:1V and 2.5H:1V, respectively. Stability analyses show that the FoS at the end of construction is greater than 1.5 and 1.0 for static and pseudo-static conditions, respectively.

1.6.8.2 Tailings storage facility

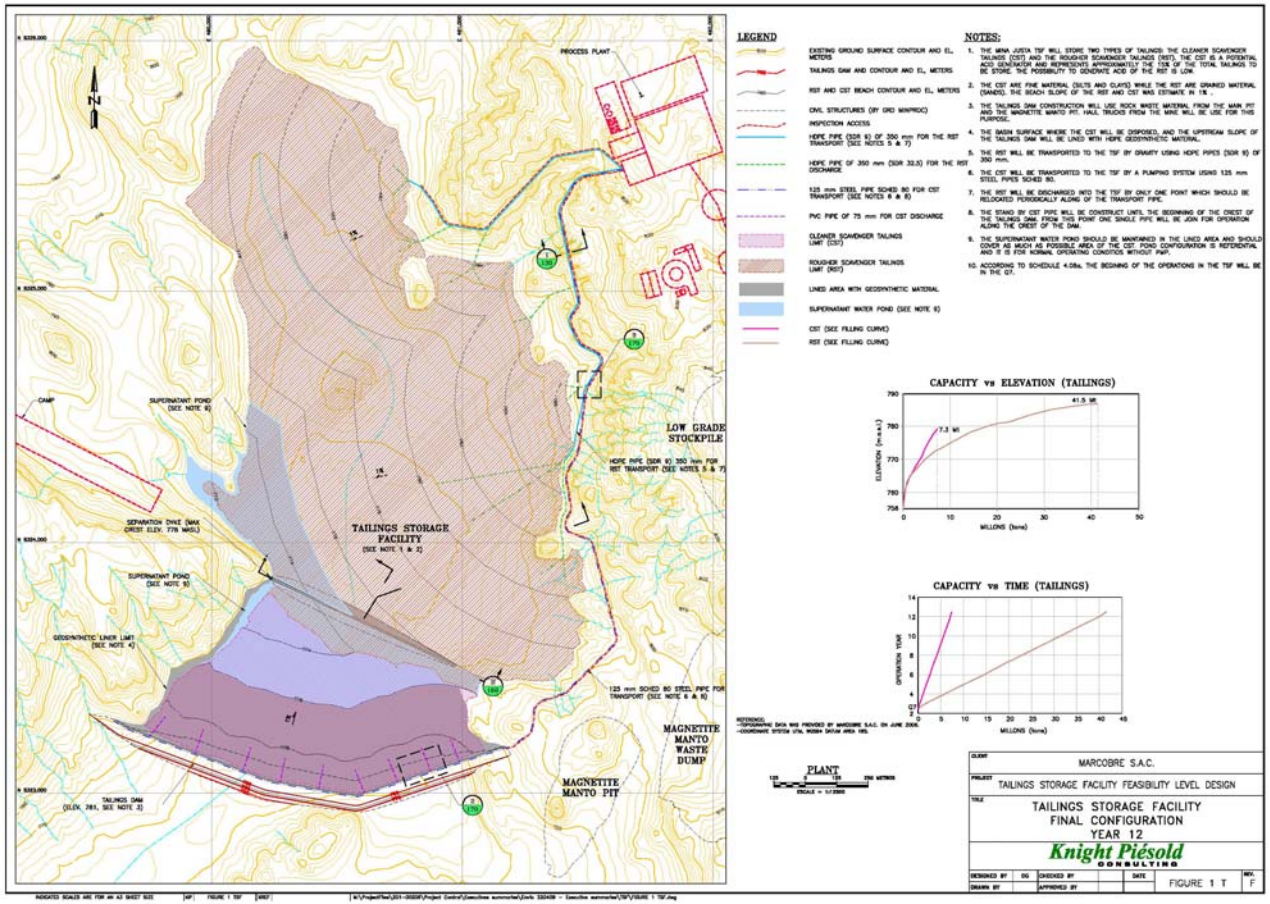
The TSF is designed to DFS level for an approximate capacity of 49 Mt of dry tailings over a period of 10 years; the tailings delivery systems have been designed to a PFS level.

The TSF is located in an area to the west of the plant site and to the northeast of Magnetite Manto open pit, and covers a surface area of about 372 ha (Figure 1.14).

The design was developed by Knight Piésold, based on site geotechnical investigation, including logging and sampling of drill holes and test pits, and on geochemical characterisation of tailings and waste material. Permeability tests were conducted in the drill holes and test pits, and piezometers were installed in selected drill holes. From the hydrogeological study developed by Vector, groundwater was encountered at an approximate depth of 450 m.

The location of the TSF was selected based on an “alternatives analysis” study of seven options for which environmental, economic and technical aspects were considered.

Figure 1.14
Tailings Storage Facility



The tailings dam is constructed of non-acid generating mine rock waste material from Magnetite Manto open pit, and has a length and height of 1.8 km and 27 m, respectively. Total volume is 2.75 Mm3. The dam is constructed in three stages using the downstream construction method. A geosynthetic liner is included on the upstream slope of the dam and beneath a portion of the tailings basin.

The Mina Justa tailings dam is assigned a "low" consequence classification according to the Canadian Dam Association (CDA) guidelines, due to the limited presence of population within 25 km around the site and the absence of surface or groundwater resources that could be affected.

The TSF is designed to store two types of tailings, the CST (potentially acid generating) and the RST (non-acid generating). It has been estimated that the CST correspond to 15% of total production, or 0.75 Mt/a of dry tailings.

A separation dike will be constructed between the CST and the RST. The CST tailings are discharged from the crest of the tailings dam, while the RST are discharged from the eastern edge of the TSF. Placement is such that the CST are kept between the tailings dam and the RST.

The RST and CST are thickened at the plant to 60% solids content (by weight). The upstream slope and portion of the tailings basin (limited by the separation dike) in which the CST are placed, is lined with a geosynthetic liner to reduce the likelihood for seepage into the foundation. In the last years of operation, the RST are discharged in a direction and sequence such that the separation dike and part of the RST containing the CST are kept within the lined portion of the basin.

The closure plan involves covering the CST using RST in order to limit the potential for oxidation and to limit inhalation or dermal contact with acidic tailings, if these conditions were to develop.

1.6.9 Port and transport

1.6.9.1 Port facilities

Sandwell (2009) completed a port evaluation study to identify costs and availability of port options, following which Marcobre determined a multi-port strategy as follows:

- San Martin, 250 km by road to the north of Mina Justa, is selected for cathode and acid shipments for the first five years.
- Matarani, 550 km by road to the south, is used for shipment of concentrates for one year.
- San Juan de Marcona, 30 km to the south, is selected for cathode, acid and concentrate shipments for the reminder of the Project.
- If San Juan de Marcona Port is not ready by 2015, then concentrates will continue to be shipped from Matarani until there is a facility in San Juan de Marcona. If the cathode handling and/or acid handling facilities are not ready in San Juan de Marcona by 2017, then these materials will continue to be handled from San Martin until facilities are available.

1.6.9.2 Road transport

The Mina Justa Project is linked to Lima by the Pan Americana Sur highway and a subsidiary highway that leads to the town of San Juan de Marcona. Total travel distance from Lima to the exploration site is 501 km, and takes approximately 6 to 7 hours. Road distances to other urban centres and port facilities are:

- Nazca - 50 km
- San Juan de Marcona – 30 km
- San Martin Port – 250 km
- Matarani port - 550 km

1.7 ENVIRONMENTAL CONSIDERATIONS

1.7.1 General

The Environmental and Social Impact Assessment (ESIA) for the Mina Justa Project forms the principal mechanism for identifying baseline conditions and evaluating the impact of the project. The ESIA is designed to satisfy the requirements of Peruvian legislation and to comply with internationally accepted guidelines for social and environmental protection followed by such organizations as the World Bank and International Finance Corporation, and followed by commercial banks through the Equator Principles.

Currently the ESIA-related work is in its final stage; more specifically, parameters found during the baseline study are being compared with the ones developed in the project description, so that mitigation measures can be elaborated.

1.7.2 Legal framework

Under current legislation, the Ministry of Energy and Mines (MEM) is the responsible environmental authority for approving the ESIA and authorising project development.

The Peruvian environmental legislation is being updated, primarily through the creation of the Ministry of Environment, which, in the future, will be the entity responsible for monitoring, controlling and promoting the care of the environment in the country. In the future all controls, permissions and authorisations will be centralised in this Ministry, but the process of developing and adapting the legislation and other organisms of the State for this purpose is still underway.

However, the ESIA now being finalised will fulfill all requirements that the new authority is likely to request.

1.7.3 Permitting

The ESIA will be submitted to regional and central offices of the MEM. The central office is in charge of conducting the evaluation process and issuing permits. The evaluation process also includes making the ESIA available to affected local communities for review and comment, publication of findings by the

agencies, a period for the applicant to respond, and then a period of final evaluation before approval and issuance of a concession to operate.

Given the presence to the north of the project of the San Fernando Reserve, MEM will seek the opinion of the National Institute of Natural Resources, prior to issung permits. Vector's evaluation of the impacts from construction, operation and closure of the Project concludes that there is no direct or indirect influence on the San Fernando Reserve, and no complications are expected in obtaining the permissions of that authority.

In addition, it is necessary to obtain agreement to the results of the archaeological evaluation from the National Institute of Culture, which is authorised to issue the required Certificate of Nonexistence of Archaeological Remains. No significant issues have been identified in this regard.

1.7.4 ESIA scope

The key objectives of the ESIA now nearing completion are discussed in the following sub-sections.

1.7.4.1 Baseline studies

A detailed description of environmental and social aspects of the project area was completed in 2008. The baseline studies were not restricted to the Mina Justa Project area, but cover the district of San Juan de Marcona.

The environmental and socio-economic impacts were identified by measuring the characteristics of the area, and comparing them with results anticipated following project implementation. In some cases (particularly in air and water studies), models were developed to evaluate the magnitude and extent of potential effects.

The results of the baseline studies indicate that Mina Justa Project site conditions are typical of a desert, with no surface water, and saline and poor soils, generally unsuitable for the development of any activity other than mining.

Surveys have recorded scant presence of flora and fauna typical of the desert environment, which are, in any case, represented throughout the San Juan de Marcona district.

No communities or population centres occur inside the zone of direct environmental influence of the Project.

Some archaeological vestiges were recorded, and these will be delimited or preserve as required.

1.7.4.2 Community relations and public consultation

An integrated community relations program has been developed by Marcobre with the following objectives:

- Establishment of ties with community leaders to enhance Marcobre's understanding of the social conditions of the neighbouring populations, their concerns and hopes for development.

- Disclosure and consultation regarding the technical and economic aspects of the project.
- Identification and establishment of mechanisms to support local development during and after operations.

Peruvian legislation recommends a minimum of three public consultation meetings during elaboration of the ESIA. Marcobre has conducted workshops during the last three years of studies, which has allowed it to communicate the development of the feasibility study, and to receive contributions and suggestions from the community.

In addition, the community relations office, established in San Juan de Marcona by Marcobre, has permanent contact with the community and has joined in the life of the population, continuously informing the community about the project and providing feed-back to Marcobre.

1.7.4.3 Identification and evaluation of effects

The main environmental and socio-economic impacts from the Project have been identified as:

- Generation of dust during construction and the operational phases
- Minor loss of vegetation
- Minor effect on some fauna species due to the presence of the operation
- An increase in immigrants to the district, particularly during the construction phase.

1.7.4.4 Environmental management

Marcobre has committed to instituting best practices for the environmental management of the project. The principal components of the management plan are:

- Monitoring program
- Program of management of domestic and industrial effluents
- Program of management of domestic and industrial residues
- Policy regarding the behaviour of Marcobre and contractor personnel
- Contingency plans.

Marcobre will establish the position of Environmental Manager, reporting directly to the General Manager, responsible for the control and of environmental programs relating to the operation.

1.7.5 Mine closure

Operations are required to have an approved closure plan and financial guarantees in place to cover the estimated closure costs. The closure plan must be developed within a year following the approval of the ESIA, and it must be approved by MEM prior to obtaining permission to operate.

The conceptual closure plan for the operation has the primary objectives of ensuring the physical and chemical stability of structures remaining after closure, and returning the environment to a condition similar to that prior to implementation of the Project.

The principal closure activities relate to:

- Review the long-term slope stability of waste dumps, in order to verify physical stability
- Covering potential acid generating material with inert material
- Demolition of infrastructure and levelling of the affected areas.

Finally, and depending on the requirements of Government regulators and the local communities, it is possible that ownership of some of the infrastructure, e.g. the water pipeline and/or the electrical transmission lines, might be transferred to the community for its use post-closure.

1.7.6 Socio-economic conditions

The Mina Justa Project is located in an agricultural region, where grapes, cotton, asparagus, olives and other produce are cultivated. The Ica region also hosts the major iron ore mine on the Pacific coast. Ica has experienced approximately 1.8% population growth between 1993 and 2005; however, a portion of the population still lacks access to basic services. Ica also has significant poverty rates, with about 29% of the population classified below the poverty line (INEI 2007). Over half the population earn their living through agriculture and fishing. Mining is also a significant contributor to the economy.

Most of the affected local people live in the town of San Juan de Marcona, which has a population of approximately 11 600 habitants and is located approximately 24 km from the Mina Justa Project.

The Mina Justa Project will contribute to the local community through jobs, local purchases of goods and services, and through taxes.

1.8 PROJECT IMPLEMENTATION PLAN

The Mina Justa Project will be implemented as an Engineering, Procurement and Construction Management (EPCM) contract in two stages. The first stage involves construction of the mine, Oxide process plant and supporting infrastructure, and the second stage, implemented directly after commissioning of the Oxide plant, involves construction of a Sulphide concentrator.

1.8.1 Implementation schedule

A 30-month timeframe is proposed from the commencement of detailed engineering to completion of Oxide plant construction, and an additional 3 months to complete commissioning and commence cathode production. It is assumed that all necessary permitting and environmental approvals are obtained within the timeframe indicated.

Similarly, a 30-month timeframe is proposed for the Sulphide plant implementation from commencement of engineering to completion of construction, and 3 months for commissioning.

The key drivers of the schedules are as follows:

- Very long delivery lead times for some critical equipment (e.g. crushers and mills)
- Large quantity of concrete works for the vat leaching area

- Construction of camp accommodation

The strategies employed in the schedule to achieve the project completion dates include the following essential element:

- Early award of EPCM contract
- Early award of critical path items such as supply of cone crushers and construction of the camp
- Maximum pre-assembly off-site to minimise construction time.

The following risks to the schedule have been identified:

- Preparation of the documents and tendering for the critical long lead equipment and camp construction may take longer than planned.
- Turn-around time for the review and approval of critical project drawings and documents could be prolonged.
- Delivery times and estimates supplied by vendors for the DFS are not firm. There is a risk that these could change when orders are placed.
- Delay in obtaining permits that affect the milestones for finance approval, project release, or construction start will adversely affect the schedule.

1.8.2 Implementation scope of work

The project scope of work includes the provision of facilities for mining, process plant, utilities and services, waste disposal and the associated infrastructure to support the construction work and on-going operations.

Marcobre's scope will include:

- Finance, insurance, governmental approvals, environmental approvals and licences.
- Land purchase, easements, rights-of-way, permits, approvals, licences, security, medical, taxes and duties.
- Mine planning and operations.
- Engagement of specialist consultants and contractors for blasting, geotechnical monitoring, hazardous and non-hazardous waste disposal, and other specialist scopes.

The EPCM Contractor's scope will include:

- Oxide and sulphide process plants.
- On-site infrastructure including camp and roads.
- Management of construction contractors.
- Management of off-site infrastructure works, including water and power supply.
- Management of specialist consultants and contractors as required.

1.8.3 Organisation

The EPCM Project Manager will be responsible for managing the EPCM works and other specialist consultants and subcontractors. Key personnel will be nominated to ensure that assigned areas of responsibility are delivered safely, on time, on budget and in accordance with specifications and project criteria.

The EPCM Construction Manager and management team will be based on-site to manage and oversee the construction contractors who will carry out the construction. A Project Sponsor's Committee, made up of senior management from Marcobre and the EPCM Contractor, will be appointed to provide resolutions to problems or issues that cannot be resolved by the project team.

1.8.4 Health, safety, environment and community

HSEC performance is critical to the success of the Project.

The Project HSEC Management Plan will be developed prior to project execution, and will identify the HSEC requirements, allocate duties and responsibilities, and detail the processes and procedures that are used to manage HSEC during the implementation of the project.

Systems, procedures and management plans will be used to align the key stakeholders, namely the Marcobre Team, the EPCM Contractor, contractors, vendors, the workforce and the community in order to achieve the HSEC objectives.

1.8.5 Permitting

Efforts to obtain all the applicable permits for the project will be grouped into two major phases. The first phase will include obtaining environmental approval for the project and all the construction and operation permits required for the mining and Oxide ore processing, including all supporting site and off-site infrastructure.

The second phase (concentrator) will begin during the execution of the first phase. The construction of the concentrator is scheduled to commence immediately after commissioning of the Oxide plant, and by that time all relevant construction and operation permits required for the Sulphide ore processing plant will be obtained.

In accordance with applicable law, the Mine Closure Plan will be completed within one year following the approval of the ESIA and will be updated accordingly.

1.8.5.1 Required consents

The competent environmental authorities for the mining sector are MEM and the Bureau for the Supervision of the Investments in Energy and Mines (OSINERGMIN). The former approves the environmental management instruments, in this specific case the ESIA and the Mine Closure Plan, whereas the latter is in charge of supervising compliance with the legal obligations in environmental matters.

Likewise, Marcobre will have to obtain other governmental consents in order to develop activities that are regulated by Peruvian legislation, i.e. mining and mineral processing, construction of hydraulic infrastructure and water use, electrical transmission, archeological evaluation projects, storage of fuel and the use of restricted chemicals, explosives, telecommunication equipment, radioactive substances, etc. These consents are issued by authorities within the Energy and Mines sector, as well as within other sectors.

Prior to commencement of construction, it is necessary to obtain the general Project consents, including the approval of the ESIA and the obtaining of the Certificate of Nonexistence of Archeological Remains (CIRA) for the area. Marcobre will have to apply for the approval of its Mine Closure Plan within one year after the approval of the ESIA.

In addition, Marcobre requires a number of other consents, including: a beneficiation concession; construction authorisation and title for the construction and operation of the Oxide ore processing facilities; authorisation to start mining activities; authorisation to construct the wellfield facilities and pipeline; a licence to use groundwater for domestic and mining purposes; electrical transmission concessions for the 220 kV and the 22.9 kV electrical lines; and an effluent treatment licence.

Prior to construction of the Sulphide plant, Marcobre will require a modification of the beneficiation concession title (includes construction authorisation), and an additional effluent treatment and reuse licence.

1.8.6 Labour requirements

Labour requirements have been estimated for construction and operation of the Project. The intention is to maximise local employment during both construction and operation, but it is recognised that certain skills may need to be obtained from outside the area.

Construction labour numbers will vary greatly over time, but it is estimated that the peak labour force will number approximately 1400 workers, not including Marcobre employees. Of these, up to 70% will be brought in from outside the area and will require accommodation on site, consequently the construction camp is designed to accommodate up to 980 people.

Additional accommodation may be required for the Owner's project team, operations manning and other works (mine developments, etc.).

Towards the end of the construction phase, a portion of the camp will be renovated and upgraded to provide offices and a permanent camp for Marcobre employees not living within an hour or so of the mine site.

Most of the construction camp will consist of portable modules which will be decommissioned and transported off site following construction of the Sulphide plant.

1.9 PROJECT OPERATIONAL PLAN

1.9.1 Production schedule

The mine production schedule is shown in Table 18.3.

1.9.2 Operational labour levels and sourcing

The workforce required to operate the Project is shown in Table 1.10. The expectation is that 60% of the workforce will be drawn from the towns of Marcona and Nazca, and will be bussed to site each day. The remainder will be based outside the region and will require accommodation in the camp. The operations camp is designed to hold 300 people in single person quarters.

Initially, it is expected that local skills will be limited to security, clerks, general labouring, drivers, technical assistants and plant operators. However, Marcobre intends to employ full-time personnel officers to conduct training courses aimed specifically at building up the skills base of local labour so that over time the proportion of local labour will rise to 90% of the total work force and will be strongly represented in management and technical areas.

Table 1.10
Summary Operations Manning Levels

	Phase 1 Oxide	Phase 2 Oxide + Sulphide	Phase 3 Sulphide
Mine			
All manning excl. maintenance	227	236	97
Maintenance manning	65	65	65
Mine - Total:	**292**	**301**	**162**
Process Plants			
Oxide	124	124	0
Sulphide	0	104	104
Subtotal (excl. Maintenance):	**124**	**228**	**104**
Cleaners	8	8	8
Maintenance manning	71	107	70
Plants - Total:	**203**	**343**	**182**
Site Organisation (excl. Maintenance)	75	84	75
Total Site Manning	**570**	**728**	**419**

1.9.3 Closure/post-closure plan

A preliminary closure and post-closure plan has been prepared as part of the DFS. The intention is that the Project will have no material, long term, negative impacts on the environment of the project area. The focus is on addressing potential impacts from waste rock, ripios and tailings disposal, and the closure plan ensures that any harmful components of these three waste streams are shielded permanently from the environment. All surface buildings and equipment will be removed from site unless otherwise agreed with the Peruvian authorities in accordance with applicable regulations.

Monitoring of the effectiveness of the closure plan will continue for a period of time to be determined in the corresponding Mine Closure Plan as approved by the regulatory authorities.

1.10 CAPITAL COST ESTIMATE

1.10.1 Project capital

GRD Minproc developed or supervised the capital cost estimates for the mining equipment, mine development, process plants, and associated infrastructure. Marcobre developed the Owner's Cost estimates. These capital costs are presented as an Oxide Plant DFS estimate (which includes the Mine DFS estimate) and as a Sulphide Plant PFS estimate.

The capital cost estimates are structured to encompass the following major categories:

- **Direct capital costs** include expenditures incurred for the construction of the process plants and infrastructure, mining and associated capital costs as defined in the Oxide Plant (DFS) and Sulphide Plant (PFS) scope of work. The costs include permanent materials and equipment, freight to site, construction labour and equipment (including contractors' supervision, overheads and profit), temporary construction facilities, construction mobile equipment, and commissioning assistance. (Note: GMI has considered the vendor representatives, first fill consumables and start-up spares within its Indirect costs for the infrastructure components.)
- **Indirect capital costs** are the expenditures related to the engineering design, procurement, project management, site construction management and commissioning supervision by the EPCM contractor. Indirect costs also provide for consultants required to supplement design engineering and construction activities.
- **The accuracy provisions** reflect the level of definition available relating to the scope of work, process design, conceptual engineering design and cost data at the time of the capital estimate development. These make appropriate allowances for uncertain elements of cost, for estimating anomalies and for omission in quantification, thereby reducing the risk of cost variation within the required accuracy level.
- **Owner's costs** include customs duties, insurance, Owner's project team, Owner's operating team prior to production, property costs (surface rights and mineral rights) and other Owner's intangibles, excluding sunk costs.

The Oxide Plant capital cost estimate has a level of accuracy of ±10%, whereas the Sulphide Plant estimate accuracy is ±20%. Both estimates are expressed in first quarter 2009 US dollars (1Q09).

The estimated total costs are summarised in Table 1.11 (Oxide Plant) and Table 1.12 (Sulphide Plant).

Table 1.11
Oxide Plant DFS Capital Cost Estimate, Summarised by Area

Area No.	Area Description	Bare Cost ($)	Accuracy Provision (%)	Accuracy Provision ($)	Total Cost ($)
001	General Plant	11,055,834	11.4%	1,261,807	**12,317,641**
010	Crushing and Screening	1,097,296	10.0%	109,731	**1,207,027**
011	Primary Crushing	12,011,627	8.9%	1,063,609	**13,075,236**
012	Primary Stockpile and Reclaim	5,399,360	9.3%	501,547	**5,900,907**
013	Secondary Screening/Crushing and Tertiary Crushing	20,287,527	6.1%	1,228,564	**21,516,091**
014	Tertiary Screening and Quaternary Crushing	22,968,439	6.0%	1,387,639	**24,356,078**
015	Quaternary Screening	10,418,274	7.3%	758,199	**11,176,473**
020	Vat Leaching	67,288,240	9.3%	6,248,419	**73,536,659**
030	Solvent Extraction	18,772,003	11.2%	2,094,912	**20,866,915**
040	Electrowinning	29,898,230	7.6%	2,262,348	**32,160,578**
050	Reagents – Oxide	2,391,151	10.3%	246,951	**2,638,102**
060	Services – Oxide	3,469,873	11.4%	395,997	**3,865,870**
070	Infrastructure – Oxide	21,533,336	9.7%	2,097,806	**23,631,142**
079	Mobilisation and Demobilisation	2,949,703	10.7%	315,812	**3,265,515**
080	Temporary Facilities	4,354,608	10.0%	435,461	**4,790,069**
081	Commissioning – Oxide	2,020,961	10.0%	202,096	**2,223,057**
082	Vendor Representatives	1,058,439	10.0%	105,844	**1,164,283**
083	First Fills and Spares	11,512,650	10.0%	1,151,265	**12,663,915**
084	Loose Tools and Equipment	1,221,938	10.0%	122,193	**1,344,131**
095	Power Supply	11,545,443	10.0%	1,154,545	**12,699,988**
096	Plant Access Road	7,134,713	13.6%	968,895	**8,103,608**
097	Construction Camp and Village	17,008,679	10.0%	1,700,868	**18,709,547**
098	Water Supply	16,576,532	13.7%	2,270,740	**18,847,272**
200	Mining	123,150,502	0.2%	224,776	**123,375,278**
	Direct Costs – Subtotals	**425,125,358**	**6.7%**	**28,310,024**	**453,435,382**
	EPCM	51,080,140	10.0%	5,108,014	56,188,154
	Indirect Costs – Subtotals	**476,205,498**	**7.0%**	**33,418,038**	**509,623,536**
	Owner's Costs	37,242,013	0.0%	0	37,242,013
	Totals	**513,447,511**	**6.5%**	**33,418,038**	**546,865,549**

Table 1.12
Sulphide Concentrator PFS Capital Cost Estimate, Summarised by Area

Area No.	Area Description	Bare Cost ($)	Accuracy Provision (%)	Accuracy Provision ($)	Total Cost ($)
001	General Plant	4,651,645	18.5%	860,088	**5,511,733**
098	Water Supply	3,598,297	12.8%	460,312	**4,058,609**
110	Sulphide Primary Crushing	15,066,043	14.3%	2,157,212	**17,223,255**
120	Sulphide Grinding	36,804,910	9.6%	3,545,430	**40,350,340**
130	Sulphide Flotation	17,230,211	15.2%	2,615,714	**19,845,925**
140	Sulphide Concentrate Thickening and Filtration	7,579,799	13.8%	1,045,210	**8,625,009**
160	Sulphide Tailings Thickening and Disposal	15,102,265	18.2%	2,753,420	**17,855,685**
170	Sulphide Reagents	2,444,480	16.9%	414,113	**2,858,593**
180	Sulphide Services	7,592,634	16.7%	1,265,847	**8,858,481**
188	Mobilisation and Demobilisation	2,136,838	13.8%	295,244	**2,432,082**
190	Temporary Facilities	2,125,302	15.0%	318,795	**2,444,097**
191	Commissioning	485,763	14.3%	69,596	**555,359**
192	Vendor Representatives	535,528	15.0%	80,329	**615,857**
193	First Fills and Spares	3,681,004	19.9%	732,099	**4,413,103**
	Direct Costs – Subtotals	**119,034,719**	**14.0%**	**16,613,409**	**135,648,128**
	EPCM	22,129,233	0.0%	0	22,129,233
	Indirect Costs – Subtotals	**141,163,952**	**11.8%**	**16,613,409**	**157,777,361**
	Owner's Costs	10,529,709	0.0%	0	10,529,709
	Totals	**151,693,661**	**11.0%**	**16,613,409**	**168,307,070**
	Owner's Costs	10 529 709		0	10 529 709
	Totals	**139 477 971**	**11.7**	**17 510 697**	**167 546 481**

1.10.1.1 Estimation methodology

Generally, for earthworks, concrete, structural steelwork and platework fabrication and installation supply rates and unit man-hours are based on information provided by GMI. GRD Minproc has independently checked GMI-provided rates against other Peruvian contractors. For these disciplines quantities were determined from material take-offs based on preliminary designs and layout drawings.

Equipment specifications were prepared and issued with tender packages for all major equipment items and packages. Budget equipment prices were obtained for all major items of equipment. Where budget quotes were not received, the balance of costing was derived from GRD Minproc's database and from allowances based on the database.

In-plant piping is derived from the actual costs of similar plants completed by, or currently in progress, designed by GRD Minproc and adjusted to Peruvian costs and productivity. The basis of the piping estimate is installed piping to number of pumps per area, where the area is the equivalent type of area.

Electrical equipment prices were obtained from multiple Peruvian suppliers for all major items of electrical equipment, the balance of pricing was based on quotes from suppliers for recent

GRD Minproc projects, some minor items were based on GRD Minproc electrical estimating database. The majority of unit rate items were built up from unit rates supplied by GMI. Site installation hours have been calculated from the GRD Minproc in-house database, using Peruvian norms as the basis.

For the Buildings a budget quotation was received from a local contractor and used within the estimate. Building area costs were compared and verified against similar buildings on current projects.

Transport of concrete-related bulk materials is included in the all-in concrete rate (reinforcing, cast-in steelwork, culverts, etc). Transport rates for steelwork and platework bulk materials were derived from rates received from installation contractors.

Transport for all equipment items is based on information received for steelwork and platework transport. Where this method of calculating freight costs was inappropriate, an allowance varying from 5% to 12% was applied. This is based on historical information, depending on original source of equipment, volume, weight, etc.

No equipment suppliers provided freight costs with their quotes other than to their closest port of departure in some instances only.

Subsequent to original pricing requests some transport information has been received from suppliers and incorporated into the estimate.

GRD Minproc did not prepare estimates for the following key components that apply to both plants:

- Owner's Costs (including all taxes, import duties, statutory charges, etc), which were prepared by Marcobre.
- Project Contingency: GRD Minproc recommends that Marcobre makes adequate allowances for items not included in the estimates, such as change of scope, abnormal or inclement weather, acts of God, industrial disturbances, foreign currency rate of exchange variations, or variances to the current market situation.
- Escalation: at Marcobre's request, escalation has been excluded from the project estimates.

1.10.2 Sustaining capital

Sustaining capital costs have been estimated for replacement of mining equipment and support equipment, computers and other office equipment, light vehicles, etc., and are summarised in Table 1.13. Deferred capital costs for expansions to the TSF, etc. are included.

Table 1.13
Sustaining/Deferred Capital Summary

PLANT	DESCRIPTION	TOTAL COST ($)
	Deferred Capital	
Sulphide	Recleaner concentrate pump 2	28,600
Sulphide	Tailings Storage Facility – Phase 2	3,220,000.00
Sulphide	Tailings Storage Facility – Phase 3	4,640,000
Sulphide	Pressure Filter Upgrade	510,000
	Sub-Total Deferred Capital	**8,398,600**
	Sustaining Capital	
Oxide/Sulphide	Replacement of computers	1,411,000
Oxide/Sulphide	Mining Sustaining Capital	15,185,638
Oxide/Sulphide	Vehicles Sustaining Capital	7,859,997
	Sub-Total Sustaining Capital	**24,456,635**
	TOTAL	**32,855,235**

1.11 OPERATING COST ESTIMATES

The operating costs for the project are summarised in Table 1.15 (Project operating costs) and Table 1.16 (Project closure costs). Costs have been determined for the following categories:

- Mining
- Oxide plant
- Sulphide plant
- General and administration (site and Lima office)
- Land transport, port, ocean freight, marketing, treatment and refining charges.

The operating cost estimate has an accuracy of ±10% (except the Sulphide Plant which was developed to a PFS level of ±20%). Operating costs are in United States Dollars and reflect an estimate base date of 1Q09 unless otherwise stated. Costs originally in currencies other than US Dollars have been converted to US Dollars at the exchange rates shown in Table 1.14.

Table 1.14
Exchange Rates

Currency Unit		Units per US$
AUD	Australia Dollars	1.54
CLP	Chile Pesos	595
EUR	Euro	0.785
JPY	Japan Yen	91.7
PEN	Peru Nuevos Soles	3.26
USD	United States Dollars	1
ZAR	South Africa Rand	10.1
CAD	Canadian Dollar	1.24
Note: Base date 16 February 2009		

IGV tax is not included in the operating cost estimates, as is expected to be fully recovered by Marcobre with a three month lag.

Except where specifically noted, no allowance has been made in the operating cost estimate for financing charges, contingency, escalation or exchange rate variations, depreciation, sustaining capital (which is included in capital expenditures), or on-going exploration.

Table 1.15
Summary of Project Operating Costs (US$/t ROM processed) Model 090821

Area	Period												
	2011	2012	2013	2013	2015	2016	2017	2018	2019	2020	2021	2022	2023
Mining	0	10.01	5.57	3.78	3.65	3.65	3.95	3.78	3.76	2.99	1.83	2.17	2.13
Oxide Plant	0	5.65	5.57	5.57	5.57	5.57	4.47	4.46	4.49	4.49	4.43	4.56	-
Sulphide Plant	0	-	-	4.81	4.73	5.11	5.02	4.97	4.72	5.15	4.71	5.07	4.83
General and Administration	0	2.05	1.25	1.08	1.04	1.04	1.04	1.04	1.04	1.04	1.03	1.86	2.92
Corporate Office (Lima)	0	0.25	0.15	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.22	0.38
Transport/ Marketing	0	0.21	0.20	3.40	3.32	4.99	5.18	2.84	2.75	2.82	2.79	7.65	11.64
Total	**0**	**18.18**	**12.73**	**18.74**	**18.33**	**20.56**	**19.77**	**17.20**	**16.88**	**16.59**	**14.90**	**21.53**	**22.90**

Table 1.16
Summary of Project Closure Costs

	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15
Progressive Closure [1)]	688 468	323 212	155 209	1 656 698	851 302	-	-	-
Final Closure [2)]	-	-	-	-	3 008 260	5 556 186	3 217 554	144 125
Total	**688 468**	**323 212**	**155 209**	**1 656 698**	**3 859 563**	**5 556 186**	**3 217 554**	**144 125**

[1)] Progressive closure costs include costs related to pits mined out before cessation of production and closure of the Oxide treatment facilities (including the ripios dump) which cease operation prior to the concentrator.

[2)] Final closure costs include Sulphide processing facilities, waste dumps, tailings pond, concentrator facilities, camp and infrastructure.

Key unit operating costs provided by Marcobre are summarised in Table 1.17.

Table 1.17
Key Unit Costs Provided by Marcobre

Item	Unit	Cost
Diesel (delivered to site, including site storage and dispensing)	$/litre	0.636
Electricity (including transmission)	$/MWh	55.00
Sulphuric acid (delivered to site)		
Via San Martin	$/tonne	90.40
Via San Juan	$/tonne	62.50

1.11.1 Mining cost

Mining costs are determined on the basis of Owner mining except for:

- Mining equipment maintenance and repair costs, which are assumed to be carried out under a maintenance and repair contract between Marcobre and the vendor of the mining equipment.
- Blasting, which is assumed to be carried out by the explosives supplier under contract with Marcobre.

The mining costs are dominated by costs associated with operating and maintaining the necessary mining fleet, but also include appropriate allowances for technical labour and materials to control the mining operation.

The accuracy of the Mining operating cost estimate is ±10%.

1.11.2 Plant and infrastructure costs

Operating costs have been developed under the following categories:

- Labour
- Power
- Maintenance Materials
- Reagents
- Consumables
- Miscellaneous Items.

Both estimates reflect the plants operating at design capacity.

1.11.3 General and Administration

The General and Administration (G&A) costs cover Labour and Miscellaneous items required to support site operations. The Labour cost includes administration, community relations, environmental, safety, security, accounting, logistics, laboratory and centralised maintenance personnel. Miscellaneous items include administration costs, insurance, personnel transport and accommodation, site services,

administration vehicles, road maintenance, consultants, and health, safety and environmental management related costs.

G&A costs encompass both site administration and Lima corporate office costs, but exclude costs attributable to exploration on other prospects.

1.11.4 Transport

Initially San Martin port is selected for cathode and acid shipments, and Matarani port for copper concentrates, switching to the Port of San Juan once available.

The reagents and consumables have been priced by suppliers on the basis of delivery to site where possible, or shipped via the port of Callao. Additional road transport costs from Callao to Mina Justa site have been estimated on the basis of the transportation cost assessment performed by Marcobre and Sandwell.

1.11.5 Environmental

The costs considered are related to the phases of construction, operation, closure and post-closure of the project.

The cost of environmental monitoring is based on Vector's work, which includes costs associated with:

- Atmospheric conditions monitoring
- Air and noise quality
- Biological monitoring
- Water quality (TSF seepage)
- Inspection and audit costs by governmental authorities, related to health, safety and environment
- Other costs including: dust control program, management of efluents and management of domestic and industrial solid wastes.

1.12 MARKETING AND PRODUCT PRICING

1.12.1 Copper cathode sales

Under the Shareholders Agreement, Marcobre's current plans commit 70% of the first 10 years of production of copper cathodes to LS-Nikko Copper. Marcobre has signed a letter of intent with Norddeutsche Affinerie AG (NA) for the remaining 30%.

In general, the agreements confirm that terms and conditions will be in line with those standard and normal in the industry for the long-term sale of copper cathodes. Marketing fees and cathode premiums are to be negotiated annually based on applicable benchmarks.

1.12.2 Copper concentrates

Details of the Shareholders Agreement provisions regarding sales of concentrates are set out in Subsection 18.14, and cover such areas as payable metal, pricing (based on LME and London Bullion Market prices), treatment charges, payment terms, insurance, etc. 90% of production is contracted with LS-Nikko; the remainder will be sold on a spot basis, most likely to smelters in the Far East. For the purposes of the DFS, it has been assumed that the balance of production is sold on the same terms as the market-related portion of the LS-Nikko contract.

1.12.3 Market review (copper and sulphuric acid)

Marcobre contracted Brook Hunt and Associates Limited (Brook Hunt) to provide a market review of:

- Supply and demand for copper cathodes and copper concentrates
- Copper price forecast
- Copper concentrate treatment and refining charge forecast
- Copper cathode premium forecast
- Penalty elements and standard penalty rates
- Freight rate forecast for copper concentrates to South Korea and copper cathodes to South Korea and Northern Europe
- Sulphuric acid supply, demand and price forecast in the Chile-Peru market
- Elemental sulphur price forecast FOB Vancouver
- Sulphur freight rate forecast from Vancouver to major Peruvian ports.

The initial report was completed in August 2008, with an up-date in February, 2009. The Brook Hunt projections are in Q1 2008 U.S. Dollar terms. US CPI inflation from Q1 2008 to Q1 2009 was -0.04%; accordingly, Marcobre determined that no inflation adjustment was required to convert the Brook Hunt forecasts to a 1Q09 basis.

Table 1.18 summarises the key forecasts from the updated Brook Hunt report, and compares the forecasts with the assumptions used in the DFS financial analysis.

Table 1.18 Summary of Forecast Prices and Terms Q1 2009 US$ terms, 2012 to 2023 Average				
	Brook Hunt Cases [1)]			DFS
	Low	Base	High	Case
Price				
LME Grade "A" Copper, $/lb	1.82	1.93	2.12	2.00
Cathode premium, $/tonne				
South Korea	63.82	69.17	77.50	69.55
Germany	82.08	89.17	100.42	89.55
Reagents				
Sulphuric Acid, $/t, CIF Main Peruvian Ports	33.33	44.17	57.50	45.00
Ocean Freight Rates				
Cathodes: South Korea, $/t	53.83	57.08	61.50	56.91
Cathodes: Germany, $/t	59.58	64.00	67.58	63.64
Concentrates: South Korea, $/t	41.83	45.17	61.00	45.50
Market Treatment and Refining Charges, Copper Concentrates				
Treatment Charge US$/t	79.17	86.25	87.50	87.00
Refining Charge c/lb	0.07917	0.08625	0.0875	0.087
Price Participation Base commencing 2015	1.50	1.60	1.75	1.60
Precious Metal Refining Charges				
Au (US$/payable oz)	5.00	5.00	5.00	5.00
Ag (US$/payable oz)	0.35	0.35	0.35	0.35

[1)] *Brook Hunt cases copyright Brook Hunt and Associates Limited*

The Brook Hunt forecasts are cyclical, but, for the DFS base case cash flow projections, Marcobre has elected to use the simple average of the Brook Hunt base case projections over the relevant period.

1.13 TRANSPORT, MARKETING AND REALISATION COSTS

The operating costs covered under this category include:

- Product trucking, port storage and handling
- Ship loading and ocean freight
- Marine cargo insurance for copper cathodes and copper concentrates
- Marketing fee (copper cathodes)
- Copper concentrate treatment and refining charges.

For the first five years of operations, from 2012 to 2016, transport of cathodes and acid are assumed to be via the Port of San Martin, switching to the Port of San Juan de Marcona in 2017. Concentrates are

assumed to be shipped via the Port of Matarani in the first year of production (2014), switching to San Juan in 2015.

Cost estimates prepared by Macobre are shown in Table 1.19.

Table 1.19
Transportation, Marketing and Realisation Costs

Category	Units	$
Copper Cathodes		
Truck to port, San Martin/San Juan	US$/t	19.59/5.48
Port handling and storage charges, Martin/San Juan	US$/t	18.20/14.50
Ocean freight: South Korea	US$/t	56.91
Ocean freight: Northern Europe	US$/t	63.64
Marine cargo insurance	% of CIF value	0.15125%
Marketing fee	US$/t	13.00
Copper Concentrates		
Truck to port, Matarani/San Juan	US$/wmt	36.77/5.48
Port handling and storage charges, Matarani/ San Juan	US$/wmt	10.50/22.00
Ocean freight	US$/wmt	45.50
Marine cargo insurance	% of CIF value	0.15125%
Market Portion (60% of LS-Nikko and 100% of other)		
Copper concentrate treatment charge	US$/dmt	87.00
Copper refining charge	US¢/payable lb Cu	8.7
Price-Sharing Portion (40% of LS-Nikko)		
Combined treatment and refining charge	% of Copper Price	24.5% Note 1
Gold refining charge	US$/payable oz Au	5.00
Silver refining charge	US$/payable oz Ag	0.35
Note 1: 24% for first 5 years, 25% for second five years, floor of 19 cents per payable pound.		

1.14 PROJECT FINANCIAL ANALYSIS

1.14.1 Background

GRD Minproc provided the capital cost, operating cost and production plan inputs for the financial analysis but expresses no opinion on the financial analysis prepared by Marcobre.

1.14.2 Key project assumptions

The production schedule is provided in Table 1.20.

The following key assumptions and parameters were used in preparing the project cash flow projections:

- Capital costs as described in this study.
- Operating costs as described in this study including the following key unit operating costs:
  - Diesel, US$0.636/L delivered to site, including site storage and dispensing

  - Electricity, US$55/MWh at the busbar at the San Juan de Marcona Substation of Red de Energia del Perú S.A. (REP)
  - Sulphuric Acid, US$45/t CIF main Peruvian ports.
- Project ramp-up and production as defined in Section 18.16.
- Corporate income tax rate of 32%, the rate applicable under a mining stability agreement, which Marcobre expects to qualify for and obtain.
- Tax depreciation at rates applicable under a mining stability agreement, being 5% for buildings, 20% for machinery, equipment and other fixed assets, and life of mine for intangibles.
- Temporary Net Assets Tax (ITAN) of 0.4% of net assets.
- Statutory employee profit sharing, 8% of pre-tax profits.
- Dividend withholding tax of 4.1%.
- Mining royalty (payable to the Government of Peru) of 1% on the first $60 M of annual sales, 2% on the next $60 M in annual sales, and 3% on annual sales in excess of $120 M, in accordance with applicable Peruvian law.
- Financial transactions tax (ITF) of 0.05% on all receipts and payments.
- Metal prices:
  - Cu price of US$2.00 /lb.
  - Silver price of US$11.00/oz.
  - Gold price of US$700.00/oz.
- Road transport, port and ship loading costs for copper cathodes and copper concentrates as outlined in Table 1.20.

Table 1.20
Annual Production Schedule

Years	Units	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	Total (*Average in (italics)*
Mine Production															
Oxide ore mined	000 t	215	10 845	20 157	13 846	7 768	11 329	14 923	10 484	9 109	9 411	5 750	765	-	114 602
Sulphide ore mined	000 t	-	-	11	4 901	5 038	5 062	4 803	5 172	5 024	5 010	5 048	5 632	3 092	48 794
Total ore mined	000 t	215	10 846	20 169	18 747	12 806	16 391	19 726	15 657	14 133	14 420	10 798	6 398	3 092	163 396
Waste mined	000 t	4 171	47 190	39 764	41 726	47 807	43 723	40 669	44 743	46 097	29 740	11 308	4 440	984	402 363
Total mined	000 t	4 386	58 036	59 932	60 474	60 612	60 114	60 395	60 400	60 230	44 160	22 106	10 838	4 075	565 769
Closing ore stockpile	000 t	215	7 511	15 980	18 307	14 112	13 502	16 229	14 886	12 018	9 438	3 236	1 279	-	-
Oxide Plant															
Oxide ore processed	000 t	-	3 550	11 699	12 001	11 999	12 001	11 999	12 000	12 000	12000	12000	3 335	-	114 602
Feed grade	% Cu	-	0.549	0.575	0.613	0.560	0.540	0.593	0.592	0.568	0.522	0.513	0.481	-	*0.564*
Cu recovery	%	-	79.0	75.4	70.7	79.2	75.7	71.8	71.9	76.3	78.5	77.0	74.0	-	*74.5*
Cathode production	t	-	15 390	50 708	51 995	48 989	49 001	51 075	51 079	52 000	52 000	47 418	11 940	-	481 596
Concentrator															
Sulphide ore milled	000 t	-	-	-	4 420	5001	5 001	5 000	5 000	5 000	5 000	5 000	5 000	4 371	48 794
Cu feed grade	% Cu	-	-	-	1.366	1.275	1.986	1.996	1.179	1.030	1.042	1.086	1.418	1.304	1.369
Cu Recovery	%	-	-	-	91.0	91.9	94.7	94.4	92.4	91.6	92.1	92.4	93.9	93.3	93.0
Recovered copper	t	-	-	-	54 947	58 595	94 069	94 189	54 499	47 188	47 979	50 181	66 577	53 148	621 373
Copper concentrate grade	% Cu	-	-	-	38.5	41.4	43.8	38.3	35.9	37.5	34.9	33.3	35.5	36.5	37.8
Copper concentrate production	t	-	-	-	142 598	141 683	214 802	245 863	15 1612	125 671	137 575	150 551	187 748	145 638	1 643 741
Au feed grade	g/t	-	-	-	0.02	0.02	0.03	0.03	0.02	0.02	0.03	0.04	0.05	0.04	*0.03*
Au recovery	%	-	-	-	80	80	80	80	80	80	80	80	80	80	*80*
Recovered gold	000 oz	-	-	-	1.8	2.3	3.4	4.3	3.2	2.8	3.5	5.0	6.4	4.4	37.0
Ag feed grade	g/t	-	-	-	10.95	11.18	20.0	22.86	12.43	11.23	11.27	10.24	15.21	15.69	*14.12*
Ag recovery	%	-	-	-	80	80	80	80	80	80	80	80	80	80	*80*
Recovered silver	000 oz	-	-	-	1 245	1 438	2 572	2 940	1 599	1 445	1 450	1 317	1 957	1 763	17 725
Payable Metal in Concentrate															
Copper	t	-	-	-	53 190	56 644	90 961	90 957	52 583	45 558	46 275	48 359	64 212	51 294	599 953
Gold	000 oz	-	-	-	-	-	-	-	-	-	1.2	3.8	4.5	2.2	11.6
Silver	oz	-	-	-	1 120	1 294	2 315	2 646	1 439	1 300	1 305	1 186	1 761	1 587	15 953
Total Payable Copper Production	000 t	-	15 390	50 708	105 104	105 633	139 963	142 033	103 661	97 558	98 275	95 777	76 153	51 294	1 081 549

- Working capital:
  - Receivables based on expected sailing times (30 days) and contract payment terms (90% 2 business days after arrival at port of discharge and 10% 60 days later)
  - Supplies equal to 30 days of operating costs other than labour and power
  - Finished product inventories averaging 3100 t of cathode and 11 250 t of concentrates
  - Early recovery of value added tax (IGV) during the construction period as permitted under an Investment Agreement which Marcobre would qualify for and plans to obtain
  - Payables equal to 30 days operating costs, except labour in which case it is 15 days.

The financial analysis is on a 100% equity basis, and has been conducted on both a before-tax and after-tax basis. The effects of inflation have not been assessed.

Costs from October 1, 2009 are considered project costs for the purpose of the evaluation, and cash flows are discounted back to that date.

Corporate income tax, statutory employee profit sharing, mining royalties, financial transaction tax, temporary net assets tax and dividend withholding tax have been included in the calculation of after-tax cash flow. The effects of debt financing have not been assessed.

1.14.3 Summary of results

Based on the assumptions set out above, the 100% equity basis after-tax internal rate of return (IRR) is 15.6% and the net present value (NPV) at 8% is $333.0 M. The project is expected to pay back initial capital, including the concentrator project capital, 4.6 years after the commencement of cathode production, (3.1 years after the commencement of concentrate production), on an after-tax basis.

The cash break-even copper price (the price at which revenues equal the sum of operating costs, sustaining and deferred capital, and closure costs) is $1.059 per pound. The economic break-even copper price (the price at which the 100% equity basis NPV at 8% is equal to zero) is $1.52 per pound.

Over the life of the Project, $1 615 M is expected to be paid in on-site operating costs and $748 M in realisation costs and other costs including the mining royalty.

Total C1 cash cost is estimated at $0.902 per payable pound of copper (including the mining royalty, transportation, marketing fees, treatment and refining charges, and silver and gold by-product credits).

1.14.4 Sensitivity analysis

1.14.4.1 Prices, operating costs and capital costs

The financial indicators are all very sensitive to copper price. A 1% change in copper price from the base case $2.00 per pound results in approximately a 4.1% change in after-tax NPV at 8%. With silver and gold credits amounting to only 3.7% of gross revenue, there is little sensitivity to by-product prices (a 0.2% change in NPV for a 1% change in by-product prices).

The project is less sensitive to capital costs (around a 1.4% change in NPV for a 1% change in capital costs) and total operating costs (a 1.3% change in NPV for a 1% change in operating costs).

Regarding cost components, acid, diesel and electricity are the largest, representing 21%, 13% and 12% of site costs respectively. A 1% change in acid price results in a 0.4% change in NPV, while a 1% change in diesel prices results in a 0.2% change in NPV. The result for electricity is similar to diesel.

1.14.4.2 Brook Hunt scenarios

As noted in Section 1.12.3, Brook Hunt provided forecasts of copper prices, copper realisation costs, ocean freight and sulphuric acid prices. With the exception of copper price, the DFS assumptions for these factors are set at the average over the relevant period of the Brook Hunt forecasts. The Brook Hunt forecasts are cyclical, however, and the use of averages can potentially obscure some important effects. Furthermore, Brook Hunt provided base, high and low case forecasts, which are useful in assessing the aggregate effect of a set of internally consistent changes in key prices and costs, rather than relying solely on the sensitivity analysis as set forth in the previous section.

Table 1.21 sets out the key results from employing the Brook Hunt forecasts in the financial model.

Table 1.21
Brook Hunt Scenarios

Brook Hunt Case	After-Tax IRR	After-Tax NPV at 8%: $ M	Payback Period: Years	C1 Cash Cost, ¢/lb
Low	15.6%	317.9	3.02	86.3
Base	16.7%	368.4	2.92	90.3
High	18.9%	476.3	2.82	94.9

There is a cyclical copper price peak in the Brook Hunt scenarios in 2016 and 2017. This just happens to coincide with the highest sulphide copper grades in the Mina Justa mine plan. The result is that the weighted average copper prices realised on Mina Justa production is considerably higher than the simple average price over the period. For example, in the Brook Hunt Base Case, the average copper price over the period 2012 to 2023 is $1.93/lb, while the weighted average realised price is $2.01/lb, an effective 4% increase in price. As shown in Table 1.21, the Brook Hunt Base cyclical copper forecast increases the after-tax NPV at 8% by $35.4M.

1.14.5 Opportunities

The following opportunities are identified for the Project and may result in an improved economic outcome or lower risk profile.

- **Additional ore reserve potential:** Whittle runs were performed using the current DFS costs and process recoveries as compared to earlier assumptions. Table 1.22 shows the results of two runs using $1.65/lb (Run 6) and $2.00/lb copper (Run 7), compared with the DFS case (Run 5).

Table 1.22
Mina Justa optimisation using final DFS parameters

Scenario	Oxide Ore				Sulphide Ore			Rec.Metal		Increase	Total
	Feed (kt)	Cu (%)	Cu_SS (%)	Cu Rec t	Feed (kt)	Cu (%)	Cu Rec t	Cu kt	Cu Mlb	Rec. Metal %	Mining kt
Run5	120,136	0.57	0.46	507,938	48,404	1.39	603,013	1,111	2,449		544,213
Run6	141,648	0.53	0.43	563,245	71,424	1.16	762,703	1,326	2,923		703,715
RUN7	171,970	0.49	0.40	630,451	82,965	1.064	810,433	1,441	3,177		795,906
Diff-Run6 & 5	21,512	0.31	0.29	55,307	23,019	0.69	159,690	215	474	19%	159,501
Diff-Run7 & 5	51,834	0.32	0.26	122,513	34,560	0.61	207,420	330	727	30%	251,693

If DFS recoveries and costs are used at the $1.65/lb Cu price then the potential ore reserves could realise a 19% increase in recovered metal. However, the benefit associated with additional oxide ore would not be as great, due to its lower average grade. The mining (total rock handling) increases by 29%.

For the $2.00/lb case, further marginal grade material could be added with a potential of 30% more recoverable copper for a 40% increase in total mining.

Additional reserves have not been determined, but the potential to increase the life of the operation constitutes an obvious opportunity to improve project economics.

- **Sulphide-Only option:** the Sulphide ore is of higher grade and higher margin than Oxide ore. If future acid prices rise significantly, or if there were a major shortage of acid, then the Sulphide Only option is a potentially attractive alternative to start development of the Mina Justa project.

  A Sulphide-only cost option would have high initial capital costs as a result of the stripping required to develop the ore at depth. Oxide mineralisation can be stockpiled for future treatment and initial capital costs (Processing) will be reduced significantly.

  GRD Minproc has revised the schedule from the main Mina Justa pit for a 5 Mt/a Sulphide-Only option, and developed mine capital and operating costs to support this option. The preliminary analysis of the Sulphide-Only case indicates that this alternative is feasible. Optimisation studies will be initiated after completion of the DFS.

- **Acid port change:** the DFS considers that sulphuric acid will initially be shipped via through San Martin port, 250 km by road from site, for the first five years of operations, switching to the port to be developed at San Juan de Marcona. There is an opportunity to ship acid via an acid terminal that Petral S.A.C. (Petral) is considering for a site it owns on San Nicholas Bay, 30 km from site. Petral estimates possible savings of $7-8 M per year for the first five years of operation, compared to San Martin. Realization of savings after five years arising from the use of the Petral site would, in part, be dependent on the specific volumes of acid that Marcobre contracts with the operators of this site. The impact of possible savings for the first five years of operation would be that the life-of-mine C1 cash cost would decrease from 90.2 cents per payable pound to 88.9 cents, and the after-tax NPV at 8% would increase from $333.0 M to $346.2 M.

- **Modified terms and conditions for copper concentrate sales contract with LS-Nikko:** price-sharing arrangements are included in the Shareholders Agreement for 40% of concentrate sold to

LS-Nikko. If it were possible to convert this volume to market terms, and to switch price participation for the first two years of concentrate production over to market terms as well, then the overall impact would be to reduce the overall C1 cost for copper from $0.902/lb to $0.848/lb. The effect on the after-tax NPV at 8% is an increase of $40.9 M.

- **Definitive Feasibility Study for Sulphide Plant:** limited and variable test data on ore competency and grindability was available for the sulphide plant PFS. Expanding the testwork database may indicate a decrease in ore competency, which would lead to a reduction in size and cost of the comminution circuit.

  Additional tailings characterisation studies are required to investigate if combined tailings deposition will result in non-acid generating tailings. If this is the case, a simplified tailings deposition system will result in capital, operating and closure cost savings.

- **Second-hand plant and cancelled orders:** it may be possible, in the current economic climate to reduce capital by as much as 20% through the inclusion of second-hand or cancelled-order equipment.

1.15 CONCLUSIONS AND RECOMMENDATIONS

1.15.1 Project overview

Based on the mining plan and capital and operating costs developed for the Oxide plant and Infrastructure at a DFS level, and for the Sulphide plant at a PFS level, the Mina Justa Project shows an after-tax IRR of 15.6% and an NPV at 8% of $333.0 M, using a copper price of $2.00/lb. Sensitivity analysis shows that the Project is highly sensitive to copper price, a 1% change in price leading to a 4.1% change in NPV. The Project is moderately sensitive to capital and operating costs, with the price of sulphuric acid, diesel and electricity being the major influences on operating costs.

While studies into mining, the Oxide plant and infrastructure have been undertaken to DFS standards, the economic potential of the Mina Justa Project relies heavily on the Sulphide plant which is supported by PFS level metallurgical testwork, plant design and engineering, although the tailings storage facility has been designed to DFS level. It is recommended that further metallurgical testwork be undertaken to support completion of a DFS for the Sulphide Plant.

1.15.2 Recommendations

The mining study is based on Indicated resources, generating Probable mineral reserves. GRD Minproc anticipates that financing of the Project could be contingent on developing a proportion of Measured resources and Proved reserves. Marcobre and potential financial backers need to determine what proportion, if any, of the current Indicated resource may have to be upgraded to Measured. The following additional work recommended by Snowden may be required:

- Further infill drilling to increase confidence in geological and grade continuity to support Measured resources. Drill hole spacing required for Measured resources varies from 25 m in the Oxide and Magnetite Manto domains, to 30-35 m in Transition and Sulphide domains.
- The density database should be expanded significantly in order to support the Measured classification, with density measurements supported by QAQC data.

- Further geological interpretation to improve domain definition. This should lead to improved domain statistics and variography, and better grade estimates.
- A conditional simulation study, to quantify the spatial uncertainty in the resource model. The potential benefits include:
  - Quantification of resource classification risk
  - Quantification of mineral reserve risk
  - Stockpile planning for high risk areas of the deposit
  - Quantifying mill feed variability
  - Optimisation of SMU and/or bench height
  - Optimisation of grade control configuration.

1.15.2.1 Mining inventory/mineral reserves

- The mining inventory has been shown to increase when the final metallurgical parameters, metal prices and operating costs results of the DFS are applied to pit optimisation. Part of these additional resources could be converted to Mineral Reserves through detailed pit design.

1.15.2.2 Process testwork and plant design

Additional testwork is required to determine a satisfactory ore variability test procedure and a suitable test to estimate recovery and acid consumption for blast hole material.

Additional metallurgical testwork is required to support the design and costing of the sulphide concentrator at a DFS level. This work would be wide-ranging, including:

- Definition of comminution characteristics
- Optimisation of sulphide flotation parameters (grind size, kinetics, reagents) for the three ore types
- Further work on a magnetite circuit to produce saleable concentrates
- Final definition of tailings characteristics, specifically PAG characteristics.

Once the concentrator flowsheet has been defined, DFS-level plant engineering and capital and operating cost estimation is required.

1.15.2.3 Environmental and permitting

The ESIA is in the process of being completed and will be submitted in the coming months. Assuming that Marcobre decides to develop the Mina Justa Project, the process of obtaining the necessary construction and operating permits and other consents should continue, following the path established by Vector.

Priority areas include acquisition of water rights for the Jahuay aquifer.

1.15.2.4 Project implementation

Strategies to achieve the proposed implementation schedule include early award of the EPCM contract, allowing detailed engineering and early ordering of long-lead items such as cone crushers and the construction camp.

1.15.2.5 Port options

There are material cost (and safety) benefits to the early establishment of port facilities at San Juan de Marcona. Marcobre should continue to investigate all avenues to promote and support this development.

The potential to import acid through a purpose-built facility at San Nicholas has been proposed by Petral. This offers material cost savings over utilising the San Martin port, and should be investigated with Petral as a priority.

2. INTRODUCTION

2.1 GENERAL

GRD Minproc previously undertook a Scoping Study for the Mina Justa Project for Chariot Resources, the results of which were presented as a NI 43-101 Preliminary Assessment Report issued in June 2006.

Thereafter, in August 2006, GRD Minproc was requested by Marcobre to undertake a DFS, the results of which form the basis of this Technical Report.

The DFS has been based on a recently updated resource model (the October 2008 model) prepared by Snowden utilising all drilling results available up to 23 May 2008, together with extensive metallurgical testwork to support a revised process flowsheet, plant design and engineering studies. Infrastructure requirements have been addressed through services provided by locally-based engineering consultants Grana y Montero Inc. (GMI). An environmental study is being undertaken by Vector, an independent consulting group with extensive experience in Peru; the ESIA is due for completion and submission by July 2009.

GRD Minproc and its sub-consultants have undertaken numerous visits to site, and maintained constant dialogue with Marcobre's representatives in Lima for general orientation.

This report has been prepared in accordance with form 43-101F (the "Technical Report") of the Canadian Securities Administrators National Instrument 43-101 (NI 43-101).

2.2 SOURCES OF INFORMATION

The preparation of the Technical Report was undertaken by GRD Minproc with significant input from Marcobre regarding exploration and other background information, and from other independent experts as identified in Subsection 2.5.

2.3 PERSONAL SITE INSPECTIONS

Dan Greig, Principal Geologist, GRD Minproc, visited the property in August 2005 and again in August 2006.

Warwick Board, Principal Consultant (Resource Division), Snowden, visited the site on several occasions during Marcobre's Mina Justa Prospect exploration programme, in September 2005, December 2006, April 2007, April 2008 and June 2008. Time on the Marcona Copper Property site varied between two and four days each visit.

Branislav Grbovic, Principal Mining Engineer and DFS Study Manager, GRD Minproc, visited the property in October 2007.

Adam Johnston, Metallurgical Consultant, visited the property on numerous occasions between 2005 and 2008.

Joe Schlitt, Metallurgical Consultant to GRD Minproc, has not visited the property. However, in 2006, 2007 and 2008, he visited all laboratories which have conducted testwork on the oxide portion of the ore.

Jodi Wright, then Senior Mining Engineer, GRD Minproc, visited the property in July 2007.

Daniel Yang, Senior Geotechnical Engineer, Knight Piésold Ltd., visited site between November 6 and 7, 2007.

Thomas F. Kerr, President, Knight Piésold and Co. (USA), Senior Geotechnical Engineer, visited site in September 2006.

Olimpo Angeles, Senior Geologist, Knight Piésold Consultores S.A., visited the site on several occasions between 2006 and 2008.

David Brownrigg, Marcobre General Manager, visited the property numerous times between August 2007 and present

John Kapusta, Marcobre's vice-President, Exploration and Geological Services, visited the property on numerous occasions between 2006 and 2008.

Klaus Meder, Andes Resources Senior Geologist, visited the property on numerous occasions between July 2004 and 2008.

Anthony Sanford, Manager of Environmental Services for Vector, visited the property on several occasions between 2006 and 2008.

2.4 TERMS OF REFERENCE

GRD Minproc is not an associate or affiliate of Chariot Resources or Marcobre, or of any associated company. GRD Minproc's fee for this Technical Report is not dependent in whole or part on any prior or future engagement or understanding resulting from the conclusions of this report. The fee is in accordance with standard industry fees for work of this nature.

In preparing this report, GRD Minproc has relied on input from Marcobre and a number of well-qualified independent consulting groups as recorded in Section 2.5, particularly regarding the resource model, geotechnical investigations and recommendations, infrastructure engineering, environmental and legal matters. Further, GRD Minproc has relied on results of metallurgical testwork undertaken by several qualified laboratories (as identified in this report) as a basis for its process flowsheet design.

2.5 CONTRIBUTORS TO REPORT

A large number of contributors provided data and other information to the Technical Report, as summarized in Table 2.1.

Table 2.1 Discipline Contributors and Responsibilities for the GRD Minproc Technical Report			
Discipline	**Section**	**Responsible Party**	**Qualified Person or other Expert**
Overall compilation	All	GRD Minproc	Dan Greig
Background, geology, history of exploration and sampling	4 to 15, 17	Marcobre	John Kapusta/Klaus Meder
Mineral Concessions and permits	4	Marcobre	David Brownrigg
Resources	7 to 15, 17.1, 17.2	Snowden	Warwick Board
Mineral Reserve, Mining	17.3, 18.1	GRD Minproc	Ross Oliver
Geotechnical	18.2	Knight Piésold	Daniel Yang/Olympio Angeles
Hydrogeological	18.2	Knight Piésold	Daniel Yang
Metallurgy and - Oxide leach	16.1	Transmin/GRD Minproc	Adam Johnston/Joe Schlitt
Metallurgy – Sulphide concentrator	16.2	GRD Minproc/Transmin	Dean David/Adam Johnston
Plant Engineering - Oxides	18.3.1, 18.3.2	GRD Minproc	Joe Schlitt
Plant Engineering - Sulphides	18.3.3	GRD Minproc	Dean David
Cost Estimation (Plant and plant infrastructure)	18.14 and 18.15	GRD Minproc	Sean McCoy/Dean David
Tailings storage facility	18.7.2	Knight Piésold	Tom Kerr
Power supply, access roads, water supply, camp, waste management*	18.4 to 18.7.1	GMI	Robinson Ucanan
Environmental	18.13	Vector - Peru	Anthony Sanford
Road transport, port operations	18.8	Marcobre	David Brownrigg
Labour management	18.10.2	Marcobre	David Brownrigg
Financial analysis	18.18	Marcobre	Brent Cochrane

* includes capital and operating cost estimation

3. RELIANCE ON OTHER EXPERTS

The author of this Report have relied on input from a number of parties who technically would not be considered Qualified Persons under 43-101, but who have the necessary qualifications and experience to provide input and opinions incorporated into the Report. These include, primarily, information regarding:

- Status of Mining Concessions
- Land ownership and permitting requirements to obtain surface rights, construction and exploitation permits and licences for plant and infrastructure
- Water rights
- Port development
- Labour regulations

Specific information provided by Marcobre included:

- Metal prices
- Acid costs, delivered to site
- Transport costs
- Power cost, including transmission
- Diesel cost, delivered, including storage and dispensing
- Information on product pricing, marketing and sales, including treatment and refining charges for copper concentrates,
- Exchange rates.
- Information concerning operations such as organisational structure, labour conditions (sourcing, salaries and on-costs, transport), camp services and maintenance.

Financial inputs were supplied by, and financial analysis undertaken by Mr Brent Cochrane of Marcobre, taking account of expert knowledge of Peruvian taxation and financing options available to Marcobre.

GRD Minproc does not claim to be expert in financial matters, and has relied on Marcobre's financial analysis in this Technical Report.

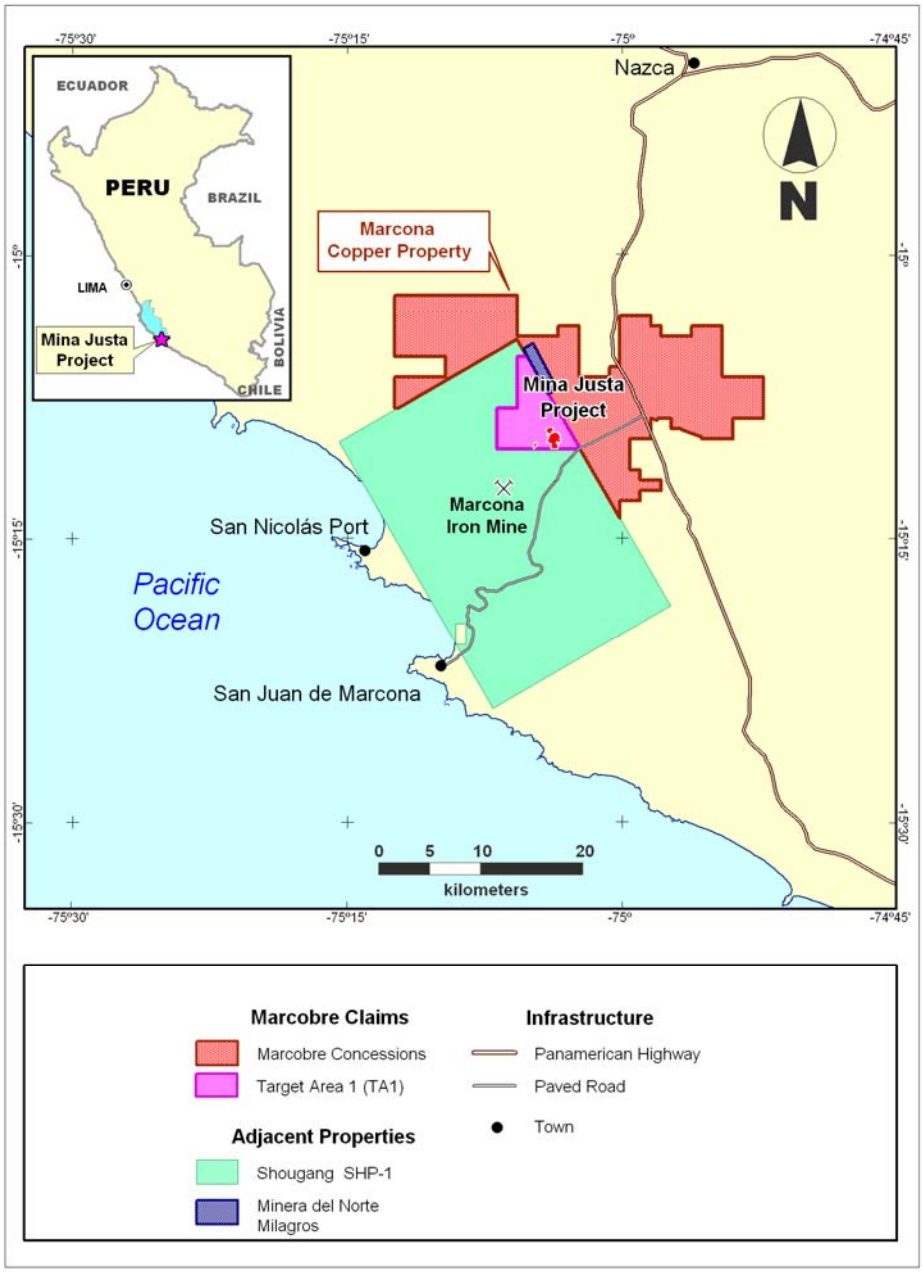
4. PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Mina Justa project is located approximately 400 km southeast of Lima within the Nazca Province, Ica Department of the southern Peruvian coastal belt. The Project lies approximately 25 km north of the coastal town of San Juan de Marcona, and the town of Nazca, on the Pan Americana Sur highway, is located approximately 35 km to the north-northeast (Figure 4.1). The geographic co-ordinates are approximately 15°08'S and 75°04'W. The Mina Justa Project occurs at elevations ranging from 785 masl to 810 masl[4].

[4] masl – metres above sea level

Figure 4.1
Marcona Copper Project - General Location Plan



4.2 LAND TENURE

The Marcona Copper Project covers approximately 32 889 ha. The Mina Justa deposit is located on the Target Area 1 (TA1) mining concession, which covers approximately 3969 ha. Marcobre also owns a group of 44 other mining concessions (jointly referred to herein as the "Marcobre Concessions") neighbouring the Target Area 1 mining concession, covering approximately 28 920 ha.

The location of the Mina Justa deposit, the TA1 mining concession and the Marcobre Concessions are shown in Figure 4.1.

4.3 PROJECT MINING CONCESSIONS

Pursuant to Peruvian law, title to a mining concession granted by the Peruvian state is required to carry out exploration and exploitation activities within the area covered by the mining concession.

Marcobre has acquired all material mining concessions related to the Mina Justa Project and the wider Marcona Copper Property. Marcobre's title to such mining concessions has been registered with the Mining Public Registry and is fully enforceable before the Peruvian State and third parties.

4.3.1 TA1 mining concession

Shougang Hierro Peru S.A.A. (Shougang) and Rio Tinto Mining and Exploration Limited Sucursal del Perú (Rio Tinto) were parties to an option agreement relating to portions of Shougang's CPS No. 1 mining concession (the "Option Agreement") dated December 14, 2000. By resolution No. 1731-2004-INACC/J, dated May 7, 2004, the competent Peruvian governmental agency approved the legal division of the CPS No. 1 mining concession, and granted title to the newly created TA1 mining concession to Shougang. The area granted for TA1 was 3 969.31 ha.

Pursuant to an amendment to the Option Agreement dated August 5, 2004 and formalized by public deed dated January 3, 2005, Rio Tinto and Shougang each consented to the other selling its respective interest in the Marcona Copper Property to Marcobre, including Shougang consenting to Rio Tinto assigning to Marcobre its rights and obligations under the Option Agreement.

Considering the above, by means of a transfer agreement dated August 6, 2004 and formalized by public deed dated January 3, 2005, Marcobre acquired (i) Rio Tinto's interest under the Option Agreement together with the relevant studies and information with respect to TA1, (ii) Shougang's title to TA1.

As a consequence of the agreements mentioned above, Marcobre is currently the sole and registered titleholder of the TA1 mining concession. This concession is in good standing and free of any liens and mortgages except for a first and preferential mortgage amounting to US$ 27 600 000.00 granted by Marcobre in favour of Shougang and Rio Tinto to secure Marcobre's payment obligations in connection with the transfer of the TA1 concession.

4.3.2 Marcobre concessions

At the same time as the transaction described in the previous section, Marcobre and Rio Tinto entered into a Claims Transfer Agreement, under which Marcobre acquired 44 contiguous claims covering approximately 28 930 ha bordering the TA1 mining concession on its northern and eastern margins. These claims were subsequently converted to mining concessions, as a result of which Marcobre is the sole and registered titleholder of the 44 concessions jointly referred to herein as the Marcobre Concessions. All of the Marcobre Concessions are in good standing and free of any liens and mortgages as of the date hereof.

4.3.3 Marcobre payment obligations

Based on the resources at Mina Justa and Magnetite Manto covered by this DFS, a contingent payment totalling $3 M will be payable following a production decision. Upon making such payment, the mortgage on the TA1 mining concession and the pledge of Marcobre's shares would be discharged.

4.4 SURFACE RIGHTS

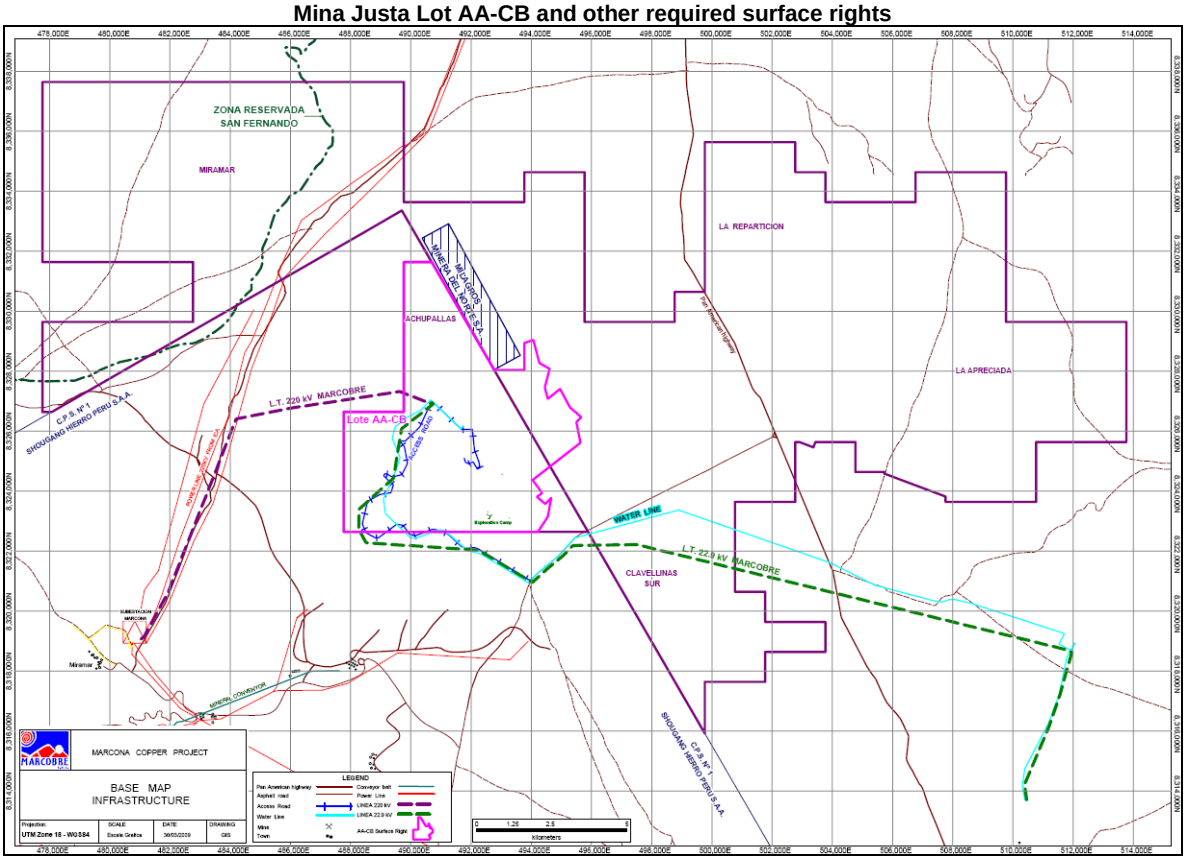
Pursuant to Peruvian law, a mining concession does not grant its holder ownership of the overlying surface area. In order for a mining concessionaire to develop a mine, it must either acquire ownership of the required surface rights or obtain authorisation from the owners. For the Mina Justa project surface rights are required for the processing facilities, the camp and offices and other site infrastructure, the open pits, waste dumps, and tailings and ripios storage. Additional surface rights, such as rights-of-way and easements for the access road, power lines and water pipeline, are also required.

4.4.1 Mine site surface rights

In order to secure the surface lands required for the development of the Mina Justa Project, Marcobre has initiated a direct acquisition procedure before the National Superintendence of Goods (SBN), which administers and manages the lands and certain other asserts belonging to the Peruvian State. On October 7, 2008, the SBN informed Marcobre about the viability of the acquisition procedure regarding lot AA-CB (which covers most of the TA1 mining concession), and ordered an on-site inspection and the valuation of this lot. After the valuation and determination of its commercial value, Marcobre will be entitled to acquire the lot, subject to the applicable procedures which include public notice, an ability of third parties to acquire the lot at the valuation price, and to Marcobre's ultimate right to acquire the property by exceeding the third party offer. On the basis of the DFS design, Marcobre considers that approximately 70% of lot AA-CB will ultimately be required for the Mina Justa Project and has designed the project in such way that all surface facilities, including waste and ripios dumps, remain within its border.

Figure 4.2

Mina Justa Lot AA-CB and other required surface rights



4.4.1.1 220 kV power line right-of-way

Marcobre requires an 15 km 220 kV power line to connect the mine site substation to the main regional substation located outside San Juan de Marcona. An electricity transmission concession granted by the Ministry of Energy and Mines (MEM) is required in order to construct a transmission line if the line requires the imposition of easements or affects the property of the State. Electricity transmission concessions allow the use of State property and grant the right to obtain the imposition of easements on property owned by third parties in order to build and operate generating stations and ancillary works, substations and transmission lines. Easements granted over State property (which is the case) are totally free, but, where the easement is granted over private property, the owner must be compensated.

From the boundary of the TA1 concession to the REP substation, the route of the 220 kV transmission line crosses Shougang's mining concession. In order to obtain the transmission concession and the related easements, procedures must be followed before the National Institute of Culture (INC) and the MEM evidencing fulfilment of the obligations related to the protection of the Nation's cultural heritage and environmental protection.

It should be noted that Shougang mining rights cover the land where the transmission line passes by, and in the past, it has successfully opposed the imposition of easements and construction of infrastructure in the area. Therefore, Marcobre is planning to have the easement granted by the MEM through an administrative procedure, having first sought to negotiate with Shougang for their consent for the easement.

Although requesting the imposition of the easement with the previous consent of Shougang is the fastest and easiest way to acquire the surface rights needed for the installation of the power transmission lines, if Marcobre cannot reach an agreement with Shougang, Marcobre has other legal alternatives to obtain the easement.

4.4.1.2 22.9 kV power line right-of-way

A 27.4 km 22.9 kV power line is required to connect the mine site substation to the substation that will supply the water wellfield. Matters concerning this procedure are the same as those referred to in the above subsection, except that most of the route of this power line does not run through Shougang's C.P.S. No. 1 mining concession. However, the 22.9 kV power line through lands owned by the Peruvian State that are theoretically reserved for the development of the Pampas Verdes Project. Should the Pampas Verdes Project oppose the imposition of the easement, Marcobre has to obtain its permission or challenge the alleged rights of the Pampas Verdes Project over such lands.

4.4.1.3 Water pipeline right-of-way

The building of pipelines and the right to impose the corresponding easements that may be required must be authorised by the Autoridad Nacional del Agua.

As for the 220 kV Power Line right-of-way, Marcobre intends to first negotiate with Shougang to obtain consent to the imposition of the easements access into mining concession, but other legal alternatives are available if agreement is not forthcoming.

Regarding the lands reserved to the Pampas Verdes Project, in the event that it opposes the imposition of the easements, Marcobre has a strong argument to challenge the Pampas Verdes Project alleged rights.

4.4.1.4 Wellfield surface rights

According to article 21 of Resolution No. 470-2008-INRENA/IRH, in order to obtain the licence to drill, dig or perform any work intended to find underground water, Marcobre must give proof to the ANA of its right over the surface land. However, to initiate the corresponding administrative procedure, it is not necessary, in principle, to evidence its superficial right over the area.

Surface rights at the wellfield are owned by the State. Marcobre plans to request the imposition of a compulsory easement which must be authorized by the ANA.

In the past, Shougang has successfully opposed the imposition of easements and construction of infrastructure in the area. Therefore, Marcobre plans to have the easement granted by the MEM through an administrative procedure, but will first negotiate with Shougang over this issue.

4.4.1.5 Site access road right-of-way

Marcobre requires a 10 km access road from National Route 30 to the mine site. A portion of this road will lie on property the surface rights to which are owned by the State and the underlying mining concession is owned by Shougang.

The access road will be private, and therefore no permits or licences are required for its construction. However, standards set by the Ministry of Transportation and Communications related to the construction and connection with public roads should be complied with. Marcobre will obtain an easement from the state for that portion of the road that lies on land owned by the State, and also plans to obtain the consent of Shougang. In the event that such consent is not forthcoming, Marcobre has other legal alternatives. Marcobre will be responsible for maintaining the private access road.

4.4.1.6 Total costs

Marcobre considers that the total cost of acquiring the surface rights and easements referred to will be US$3.1 million.

4.4.1.7 Construction materials

While Marcobre is the titleholder of the TA1 concession and is in the process of acquiring most of the surface rights overlying such mining concession, these rights will not give Marcobre the ability to freely use materials on site for construction purposes. Marcobre can, however, freely use waste rock extracted from Mina Justa and Magnetite Manto pits for construction purposes. In the event that such waste rock is not suitable for construction, Marcobre has identified quarry sites with materials that have been tested and found suitable for construction elsewhere on the TA1 concession. Marcobre can freely use such materials provide that it first carves a separate industrial minerals mining concession out of the TA1 concession. The procedure required for this is simple and inexpensive.

4.5 OVERVIEW OF PERUVIAN MINING LAW

The General Mining Law of Peru defines and regulates different categories of mining activities, from sampling and prospecting to commercialisation, exploitation, and processing. Mining concessions are granted using UTM coordinates to define areas generally ranging from 100 ha to 1000 ha in size. Mining titles are irrevocable and perpetual, as long as the titleholder makes annual maintenance fee payments of $3/ha (for metallic mineral concessions) for each concession (or for a pending application), at the time of acquisition and then by 30 June of each subsequent year to maintain the concession. The concession holder must sustain a minimum level of annual commercial production of greater than $100/ha in gross sales before the end of the sixth year of the grant of the concession; or, if the concession has not been put into production by the first semester of the seventh year) the annual rental increases through the imposition of a $6/ha penalty until the minimum production level is met. If by the start of the twelfth year the minimum production level has still not been achieved the annual penalty increases to $20/ha thereafter. The concession holder can be exonerated from paying the penalty if he can demonstrate that during the previous year he has “invested” an equivalent of no less than ten times the penalty for the total concession. This investment must be documented along with the copy of the annual tax statement and the payment of the annual fee. The concession will terminate if the annual

rental is not paid for three years in total or for two consecutive years. The term of a concession is indefinite provided it is properly maintained by payment of rental fees.

The holder of a mining concession is entitled to all the protection available to all holders of private property rights under the Peruvian Constitution, the Civil Code, and other applicable laws. A Peruvian mining concession is a property-related right, distinct and independent from the ownership of land on which it is located, even when both belong to the same person. The rights granted by a mining concession are defensible against third parties, are transferable and chargeable, and, in general, may be the subject of any transaction or contract.

To be enforceable, any and all transactions and contracts pertaining to a mining concession must be entered into as a public deed and be registered with the Public Mining Registry. Conversely, the holder of a mining concession must develop and operate his/her concession in a progressive manner, in compliance with applicable safety and environmental regulations and with all necessary steps to avoid third-party damages. The concession holder must permit access to those mining authorities responsible for assessing that the concession holder is meeting all obligations.

4.6 ENVIRONMENTAL AND SOCIO-ECONOMIC ISSUES

The Environmental and Social Impact Assessment (ESIA) for the Mina Justa Project forms the principal mechanism for identifying baseline conditions, evaluating the impact of the project, and, as appropriate, identifying alternatives and mitigants. The ESIA has been designed to satisfy the requirements of Peruvian Legislation and to comply with internationally accepted guidelines for social and environmental protection followed by such organizations as the World Bank and International Finance Corporation, and followed by commercial banks and other financial institutions through the Equator Principles.

Based on ESIA work completed to date, there are no material impacts requiring mitigantion that have not been considered in the project design, and hence in the project operating and capital costs.

4.6.1 Legal framework

The legal and institutional framework in Peru is represented by a number of authorities that have the jurisdiction to permit and regulate implementation of mining projects. Primary among these is the MEM. The legal framework applicable to the Mina Justa Project is outlined by a number of environmental protection laws and documents. Key among these is Peruvian General Environmental Law (Law 28611).

The Peruvian environmental legislation is in the process of being updated, primarily through the creation of the Ministery of Environment, which, in the future, will be the entity responsible for monitoring, controlling and promoting the care of the environment in the country.

The intention is that in the future all the controls, permissions and authorisations are centralized in this Ministry, but the process of developing and adapting the legislation and other organisms of the State for this purpose is still underway.

It is clear that under current legislation, the MEM is the responsible environmental authority for approving the ESIA and authorising project development. However, the ESIA now being finalised will fulfill all requirements that the new authority is likely to request.

4.6.2 Permitting

The ESIA is submitted to regional and central offices of the MEM. The central office in Lima is in charge of conducting the evaluation process and, eventually, issuing permits. Depending on the location of the project and its characteristics, other agencies may be involved in the evaluation of the ESIA. The evaluation process also includes making the ESIA availabe to affected local communities for review and comment, publication of findings by the agencies, a period for the applicant to respond, and then a period of final evaluation before approval and issuance of a concession to operate.

Given the presence to the north of the project of the San Fernando Reserve, MEM will seek the opinion of the National Institute of Natural Resources (INRENA), prior to issung permits. The ESIA evaluation of the impacts from construction, operation and closure of the Project concludes that there is no direct or indirect influence on the San Fernando Reserve, and no complications are expected in obtaining the permissions of that authority.

In addition, it is necessary to obtain agreement to the results of the archaeological evaluation from the National Institute of Culture, which is authorised to issue the required Certificate of Nonexistence of Archaeological Remains.

4.6.3 ESIA scope

The key objectives of the ESIA are:

- Determine baseline environmental conditions in the Project area, i.e. establish the physical, biological and socio-cultural conditions before the establishment of the project.
- Identify environmental and socio-economic resources that could potentially be affected by the project.
- Predict positive and negative effects resulting from the Project, and determine to what degree the negative effects can be mitigated.
- Quantify and evaluate the significance of the effects wherever possible.
- Outline requirements for monitoring of the resources that could be affected by the project.
- Provide a conceptual closure plan for the mine site and associated facilities.
- Complete a cost-benefit analysis of the project.

4.6.4 Baseline studies

A detailed description of environmental and social aspects of the project area was developed; the studies began in 2006 and were completed in 2008. All of the baseline studies were developed by local professionals.

The baseline study has not been restricted to the Mina Justa Project area, but extends to cover communities within approximately 70 km of the site.

The environmental and socio-economic impacts were identified by measuring the characteristics of the area, and comparing them with results anticipated following project implementation. In some cases (particularly air and water studies), models were developed to evaluate the magnitude and extent of potential effects.

The results of the studies indicate that Mina Justa Project site conditions are typical of a desert, with no surface water, saline and poor soils, generally unsuitable for the development of any activity other than mining.

Surveys have recorded scant presence of flora and fauna typical of the desert environment, which are represented throughout the San Juan de Marcona district.

No agricultural activities, communities or population centres occur inside the zone of direct environmental influence of the Project.

Some archaeological vestiges have been recorded, and these will have to receive the treatment that the legislation dictates in order to delimit or preserve them as required.

4.6.5 Community relations and public consultation

In order to optimize relations between the community and the project, an integrated community relations program has been developed with the following objectives:

- Establishment of ties with community leaders to enhance understanding of the social conditions of the neighbouring populations, their concerns and hopes for development.
- Disclosure and consultation regarding the technical and economic aspects of the project.
- Identification and establishment of mechanisms to support local development processes throughout and after operations.
- Strengthening of the institutions through development of consensual programs based on mutual respect and transparency.

From the beginning of the environmental studies, Marcobre has implemented a policy of involvement with representative sectors of the community as part of the ESIA process.

Peruvian legislation recommends a minimum of three public consultation meetings during the ESIA. The initial meeting is designed to introduce the communities to the ESIA process, help them understand their rights and responsibilities, and to describe the baseline studies that form a part of the permitting process. Information about the general characteristics of the project (scale, lifecycle, etc), the complexity of the mining activity, and the relations that will be established with the local community are shared in each of these meetings. The local population is involved in the baseline studies, with community members participating in the field teams specialising in fauna, flora, water and soil surveys.

Marcobre has successfully conducted workshops during the last three years of studies, which have allowed it to communicate the development of the feasibility study, and to receive contributions and suggestions from the community.

The Office of Community Relations located in San Juan de Marcona has permanent contact with the community and has joined in the life of the population, continuously informing the community about the project and providing feed-back to Marcobre.

4.6.6 Identification and evaluation of effects

The environmental and socio-economic impacts were identified and compared with the anticipated impacts of the implementation of the project. The main effects and corresponding mitigation measures for the construction and operating stages are related mainly to water and land usage for the mine site.

Some of the impacts identified are:

- Changes in the current use of the soils.
- Changes in the topography and soil due to the presence of tailings and ripios, open pit and waste rock facilities.
- Generation of dust during the construction and operation phases.
- Generation of noise in the construction and operation areas.
- Minor loss of vegetation coverage.
- Migration to adjacent areas of some fauna species due to the presence of the operation.
- An influx of people from outside the area during construction; a reduction in numbers is expected during mine operation and again at closure.

Marcobre has committed to instituting best practices for the environmental management of the project. The implementation process will begin once the authorities have granted permission to proceed with the project. This will provide a global mechanism to ensure that appropriate environmental management is maintained during the life of the mine.

Environmental management is conceived for three principal stages of the project: construction, operation and closure.

The principal components of the environmental management plan are:

- Monitoring
- Management plan for domestic and industrial residues
- Management plan for domestic and industrial effluents
- Code of conduct for Marcobre and contractor personnel
- Contingency plans.

Marcobre will establish the position of Environmental Manager, responsible for the control and environmental management of operations. This area will report directly to the General Manager, and will be in charge of supervising and controlling all the environmental programs related to the Project.

4.6.7 Mine closure

Closure legislation (Peruvian Law 28090) requires that every operation must have an approved closure plan and financial guarantees of ability to cover the estimated closure costs. The closure plan must be developed and submitted to the MEM within a year following the approval of the ESIA. It must also be approved by the MEM prior to receipt of permission to operate.

The ESIA has developed the conceptual closure plan for the operation, with the objective of ensuring the physical and chemical stability of the diverse components of the project after closure and returning the environment to a condition similar to that found before implementation of the Project.

The principal closure activities relate to the reduction in slopes of waste dumps, in order to assure physical stability, and covering potential acid generating material with inert material.

In addition, the closure plan provides for demolition of facilities and most infrastructure, and leveling of the involved areas.

Depending on the requirements of Government regulators and the local communities, it is possible that ownership of some of the infrastructure, e.g. the water pipeline and/or the electrical transmission line, might be transferred to the community for its use post-closure.

4.6.8 Socio-economic conditions

The Mina Justa Project is located in the Department of Ica, an important agricultural region, where grapes, cotton, asparagus, olives and other produce is cultivated, and where the biggest deposits of iron on the Pacific coast occur. Ica has experienced approximately 1.8% population growth between 1993 and 2005; however, a portion of the population still lacks access to basic services. Ica also has significant poverty rates, with about 29% of the population classified below the poverty line (INEI 2007). More than half the population earn their living through agriculture and fishing. Mining is also a significant contributor to the economy.

Most of the affected local people live in the town of San Juan de Marcona, which has a population of approximately 11 600 habitants and is located approximately 24 km from the Mina Justa Project.

The Mina Justa Project will contribute to local economy through jobs, local purchases of goods and services, and through taxes, as well as through community development programmes such as those implemented by Marcobre during resource drilling and preparation of the DFS and ESIA.

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

5.1 ACCESSIBILITY

The Project lies close to the Pan Americana Sur highway, and can be reached by driving south from Lima approximately 6.5 to 7.5 hours (447 km). Driving distances from San Juan de Marcona and Nazca are 30 km and 48 km respectively.

The nearest airfields are at Nazca and at San Juan de Marcona. Neither are serviced by scheduled flights, but both are suitable for chartered aircraft. Flying time from Lima to San Juan de Marcona is approximately 1.0 hour.

5.2 PHYSIOGRAPHY, FLORA AND FAUNA

The Project is located within the coastal plain area of Peru approximately 20-30 km from the Pacific Ocean. The ground in the immediate vicinity of the Mina Justa project is relatively flat to low-lying hills, ranging in elevation from 630 to 880 masl. The eastern flank of the Mina Justa deposit is marked by a north-northwest trending, steep fault-bounded scarp.

Due to the desert climate, vegetation on the property is almost non-existent with less than 1% cover and limited to a few scattered clumps of Clavelinas that depend on moisture from the thick fogs. None of the property is used for agricultural purposes.

5.3 CLIMATE

The Marcona Project area is located in a desert area within the Peruvian coastal belt. The area has an arid climate with strong prevailing southerly winds during the day, switching to northerly winds at night. Annual rainfall ranges between 0 mm and 80 mm, averaging approximately 27 mm. The annual mean temperature is approximately 19°C average. Monthly maximum temperatures range between 22°C and 28°C and minimum monthly average temperatures range between 15ºC and 26°C. Relative humidity generally ranges between 65% and 85%. During the winter months of June to August, thick fogs, or “neblinas” are common.

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

Some infrastructure exists at the small town of San Juan de Marcona, which was developed to support on-going large-scale mining of the Marcona Iron ore deposits over the past 50 years. The town has a population estimated at 11 500 with nearly 1800 employed by the mine. The towns of Nazca and Vista Alegre have a combined population of approximately 20 000.

The region can provide the basic goods, services, medical care and some accommodation to assist in project development, as well as meet some labour requirements for various stages of exploration and development projects.

San Juan de Marcona and the Marcona mine operations are connected to the National Power Grid. A high tension line passes within 10 km of the Mina Justa Prospect.

There is no surface water on the Project site; sub-surface water has been intersected at a depth of 450 m at the Mina Justa deposit. Water for the San Juan de Marcona community is obtained from the Jahuay aquifer located 30 km southeast of the Mina Justa Prospect and along the Pan Americana Sur highway. The waterline passes within 30 km south of the Mina Justa Prospect. SHP is currently using one of the three aquifers at Jahuay, and a second underground water source is also located approximately 20 km further to the southeast at Lomas aquifer. Water rights will have to be acquired and developed by Marcobre.

The nearest suitable ports are at San Martin, 250 km by road to the north, and Matarani, 550 km to the south. A deep-water port facility for shipping the SHP iron concentrates was constructed at Puerto San Nicholas, located 20 km southwest of the Mina Justa Prospect, but it is appears unlikely that suitable access to this facility can be negotiated with SHP.

Cellular phone coverage is available to a limited extent in the Project area, but the grid is expanding. Communications are via satellite telephone.

5.5 OVERVIEW OF PERU

The Republic of Peru is a democratic constitutional republic governed by an elected government headed by a president who is both chief of state and the head of government. The president also appoints the members of the Council of Ministers.

Following presidential and congressional elections held in June 2006 anew president was elected. The elected president, Mr. Alan Garcia, together with the new congress assumed office for a five-year term, in july 2006. The legislative branch is a unicameral congress composed of 120 members elected to serve five-year terms. The judicial branch of government consists of a Supreme court of Justice whose judges are appointed by the National Council of the Judiciary.

The country is organised into 24 regions and one constitutional province, formerly known as departments. Peru is implementing a decentralization program whereby those 25 regional administrations will begin to exercise greater governmental authority over their territories. In November 2006, voters also elected new regional presidents and other municipal and local authorities. The authority that these regional governments currently exercise will increase over the years although it will take time to decentralize most government functions.

Peru's economy reflects its varied geography and local climates. The country consists of an arid coastal region; the Andes Mountains further inland and tropical lands bordering Columbia and Brazil. Abundant mineral resources are found in the mountainous areas and the coastal waters provide excellent fishing grounds. An overdependence on minerals and metals however, subjects the economy to fluctuations in world metal prices. After several years of inconsistent performance, the Peruvian economy now boasts one of the fastest growing economies in South America.

Peru has developed mining infrastructure, a large pool of skilled technical and professional personnel and an established legal system. Due to an abundance of mineral resources Peru has become a leading target for the attraction of foreign investment in South America's mining sector.

6. HISTORY

A summary of the exploration history on the Marcona Copper Property, with emphasis on the Mina Justa Prospect, is presented in Table 10.1.

7. GEOLOGICAL SETTING

The Mina Justa Prospect is located in the Marcona-Mina Justa Iron-Copper District in the Coastal Belt of Peru. This northwest trending linear belt represents the westernmost part of the Central Andean Cordillera, where the Nazca Plate subducts beneath the South American Plate, forming an active continental margin along the Peru-Chile Trench.

7.1 REGIONAL GEOLOGY

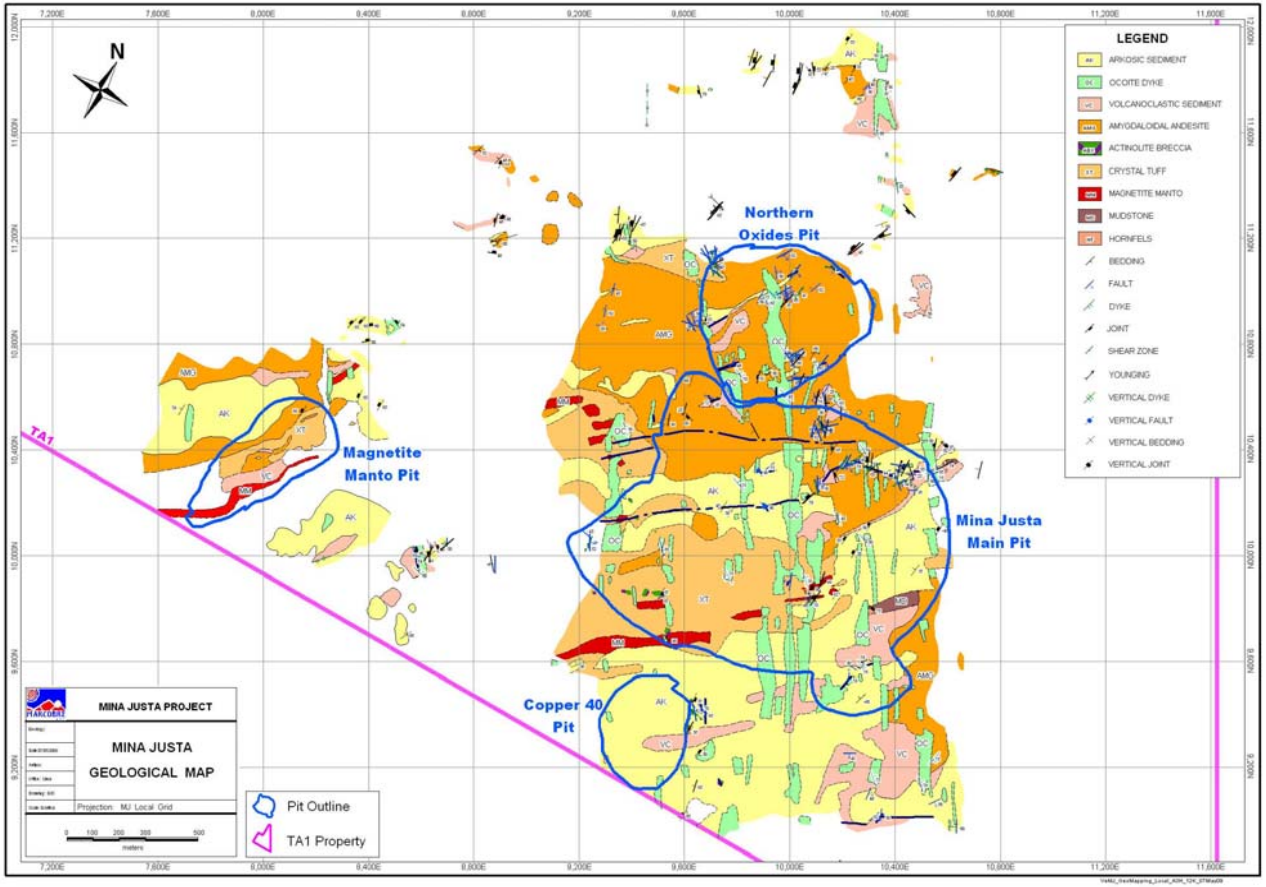
The geology of the Marcona-Mina Justa Iron-Copper District consists of a Precambrian high-grade metamorphic basement (the Arequipa Massif), unconformably overlain by Neoproterozoic and Phanerozoic sedimentary rocks. Palaeozoic sediments (the Ordovician Marcona Formation) host the majority of the economic magnetite orebodies at the Marcona iron mine. Monzogranite, granodiorite and gabbro-diorite rocks of the post-kinematic San Nicolas batholith (dated at approximately 425 Ma) intrude the pre-Mesozoic rocks. The pre-Mesozoic rocks are unconformably overlain by a series of volcano-sedimentary and volcano-plutonic arc sequences that range in age from late Triassic to Holocene. The volcano-sedimentary rock sequences are intruded by porphyritic andesite dykes, sills and plugs of the Tunga Andesite (also termed “Ocoite”); and, in the eastern parts of the district, by granitoid plutons of the circa 109 Ma Coastal Batholith. Tertiary age shallow water marine sediments and Quaternary marine terraces unconformably overlie the volcano-plutonic arc succession.

7.2 LOCAL GEOLOGY

The Mina Justa Prospect comprises two deposits, the Mina Justa and Magnetite Manto deposits (Figure 7.1), which are hosted by the Jurassic Upper Río Grande Formation, dominated by andesitic lavas and pyroclastics, intercalated with minor sandstone, siltstone and carbonate units. This volcano-sedimentary package displays a prolonged deformation history that includes a southeast verging overturned folding stage, followed by shear faulting that generated curvilinear fault systems. The youngest deformation stage is normal block faulting along northwest trending structures that are closely associated with late stage ocoite dykes.

Figure 7.1

Mina Justa Prospect geology showing location of Mina Justa and Magnetite Manto copper deposits



Conceptual pit outlines included for reference.

8. DEPOSIT TYPES

Until recently it was believed that the Marcona iron mine Figure 4.1 and the Mina Justa copper deposits were related, both being part of a large iron-rich hydrothermal system formed in an extensional environment along a subduction-related continental margin. Recent work (Chen, 2008) suggests that the Mina Justa Prospect is significantly younger (approximately 104-95 Ma) than, and geochemically distinct from the Marcona Iron deposit (approximately 162-156 Ma). The Mina Justa Prospect is now interpreted as a hydrothermal deposit that was formed by the incursion of exotic and probably evaporite-sourced brines that were expelled from an adjacent sedimentary basin. The recent findings support the classification of the Mina Justa Prospect as an Iron Oxide Copper Gold (IOCG) deposit, whilst arguing for the removal of the Marcona Iron deposits from the IOCG clan.

The Mina Justa Cu (-Fe, Ag and Au) deposit shares many mineralogical and textural characteristics with other major exocontact Andean Cu-rich IOCG deposits, e.g. Raúl-Condestable (Peru), Mantoverde and La Candelaria in the Punta del Cobre District (Chile).

9. MINERALIZATION AND ALTERATION

Massive, brecciated, elongated magnetite (-pyrite) bodies host the highest-grade copper sulphide mineralisation at Mina Justa. The location of these bodies appears to be controlled primarily by a northeast striking and southeast dipping system of faults (the Mina Justa fault system). The mineralised bodies have, however, been dislocated by northwest striking and northeast dipping faults (Huaca faults) and associated ocoite dykes, the latter ranging from less a few metres to 70 m in thickness (typically 15-30 m in thickness).

Seven stages of hydrothermal alteration and hypogene mineralisation were recognised in the Mina Justa prospect, providing evidence of protracted hydrothermal evolution linked to the deformational history of the Marcona-Mina Justa District. A sequence of four distinct hydrothermal alteration stages including albite-actinolite alteration, K-Fe metasomatism, Ca metasomatism and an early haematite stage was followed by intense Fe-metasomatism that formed the magnetite bodies in the Mina Justa deposit at circa 104-101 Ma. The subsequent main Cu sulphide mineralisation stage of chalcopyrite, chalcocite and bornite (at circa 99-95 Ma) frequently replaces the precursor magnetite mineralisation in stratabound and structurally controlled ore body geometries. The Mina Justa hydrothermal alteration sequence is concluded by late-stage specular haematite deposition.

Modified by post-mineralisation faulting, the supergene Cu oxide mineralisation of predominantly atacamite and chrysocolla typically extends down to 180 m depth. The supergene mineralisation is hosted mainly by rock fractures.

The mineralised bodies of the Mina Justa deposit extend over an area of approximately 2100 m north-south by approximately 1500 m west-east, and range in thickness from a few metres up to 150 m. The mineralisation is near-surface in the northern and western parts of the deposit (the “Northern Oxides”, “Western Extensions” and “Cu40” zones, respectively), extending to depths approaching 550 m in the southeastern parts of the deposit (the “Sulphide Extensions” zone). The mineralised bodies are generally flat lying in the upper parts of the deposit (i.e. in the supergene oxidation zone). At depth the mineralisation follows the curvilinear faults, and resembles a flat bowl-like structure with an overall shallow plunge of approximately 15° to the southeast. Sulphide mineralisation at depth displays a central core of bornite and chalcocite surrounded by predominantly chalcopyrite mineralisation. A narrow transition zone separates the sulphide mineralisation from the overlying oxides. Sulphide mineralised bodies appear to increase in thickness from west to east, and with increasing depth.

The Magnetite Manto mineralised body strikes approximately northeast-southwest, with a moderate dip of approximately 60° to the northwest. The tabular body is approximately 700 m long by 350 m wide, ranging between 25 m and 35 m in thickness. The Magnetite Manto deposit is characterised by copper oxide mineralisation.

10. EXPLORATION

A summary of the exploration history of the Marcona Copper Property, with emphasis on the Mina Justa Prospect, is presented in Table 10.1.

Table 10.1
History of the Mina Justa Prospect

Period	Event
Late 1800s to mid-1980s	Episodic small-scale artisanal mining for copper oxides in the Marcona district.
1950-1972	Regional airborne magnetic survey and follow-up geological mapping, geochemical sampling and drilling by Marcona Copper Company (subsidiary of Utah Mining Corporation), operators of Marcona iron mine.
1975-1980s	Small-scale mining of Marcona Iron Mine and (possibly) parts of the Mina Justa Prospect by Propiedad Minas Justa S.A., following nationalisation of the Marcona Iron Mine.
1992-2000	Surface mapping, rock chip sampling, trenching, electromagnetic surveys and preliminary resource estimation work conducted by Jindi (a subsidiary of Shougang Hierro Perú S.A.; SHP) throughout the Marcona Iron Mine mining concession, including the area of the Mina Justa Prospect.
1993-2003	Exploration by Rio Tinto: regional airborne magnetic and radiometric surveys, property acquisition, limited initial exploration drilling of Clavelinas Prospect (1995), joint venture agreement with SHP (2000), geological mapping, geochemistry, geophysics, exploration drilling, limited metallurgical testing, resource estimation and economic studies on Mina Justa Prospect. Rio Tinto conducts limited drill testing of other targets, including Achupallas, Miramar, Clavelinas and La Apreciada Prospects. The Mina Justa Prospect fails to meet Rio Tinto's minimum size requirement and its interest is placed for sale.
Aug. to Dec., 2004	Chariot Resources Ltd. (Chariot) acquires Marcona Copper Project, in a joint venture agreement with Korea Resources Corporation (KORES) and LG-Nikko Copper Inc., from Rio Tinto and SHP. Peruvian-registered Marcobre S.A.C. (Marcobre) incorporated, owned 70% by Chariot and 30% by KORES and LG-Nikko Copper Inc. (now known as LS-Nikko Copper Inc.).
March 2005	Marcobre commences first phase drilling programme on the Marcona Copper Project. Initial results reported in April 2005.
2005-present	Exploration by Marcobre: the primary focus of Marcobre's exploration programme was (and is) exploration and resource definition drilling on the Mina Justa Prospect. A regional exploration programme of geological mapping, surface sampling, geophysical surveys (ground magnetic and IP surveys) and trenching was initiated on the Achupallas, Miramar, Clavelinas and La Apreciada deposits, which occur on lands surrounding the Mina Justa Prospect, within the Marcona Copper Project. There is limited drill testing of targets in Achupallas (2005) and Clavelinas (2008).

11. DRILLING

Drilling has been the dominant tool used by Marcobre in the exploration of the Mina Justa Prospect. Marcobre drilled a total of 256 451.18 m in 1 075 drill holes on the Mina Justa Prospect from 2005 to 2008. The majority of the drilling was conducted on the Mina Justa deposit with 227 843.98 m in 938 drill holes. A total of 28 607.20 m in 137 drill holes was drilled on the Magnetite Manto deposit.

A summary of the drilling conducted on the Marcona Copper Property, with emphasis on the Mina Justa Prospect, is provided in Table 11.1. Drill hole locations are displayed in Figure 11.1.

Table 11.1
Drilling conducted on the Marcona Copper Project

Period	Company	Details
Pre-2005	Rio Tinto	Rio Tinto drilled a total of 41 142.65 m in 298 drill holes on the Marcona Copper Property. A total of 31 024.85 m in 102 drill holes was drilled on the Mina Justa Prospect using a combination of diamond core (total of 7 448.85 m) and RC (total of 23 576.00 m) drilling techniques. In addition, a further 159 short RC reconnaissance drill holes totalling 1 982 m were drilled in the prospect.
2005	Marcobre	Marcobre drilled a total of 50 401.60 m in 141 drill holes on the Mina Justa Prospect in 2005. Drilling was predominantly RC (122 RC drill holes out of 141 drill holes), with just two diamond core drill holes. Diamond core tails were completed in 17 RC drill holes.
2006	Marcobre	Marcobre drilled a total of 66 473.50 m in 300 drill holes in 2006. Drilling was predominantly RC (274 RC only out of 300 drill holes) with 13 diamond core drill holes (including nine metallurgical drill holes). Diamond core tails were completed in 13 RC drill holes (including four metallurgical drill holes).
2007	Marcobre	A total of 75 233.98 m in 357 drill holes was drilled on the Mina Justa Prospect in 2007. Drilling was predominantly RC (261 out of 357 drill holes totalling 57 354.00 m), with 96 diamond core drill holes totalling 17 879.98 m.
2008	Marcobre	A total of 37 935.80 m in 145 drill holes had been drilled and compiled into the mineral resource database for the Mina Justa Prospect at the 8 August 2008 cut-off date. Drilling was predominantly RC (127 out of 145 drill holes totalling 33 460.00 m), with 18 diamond core drill holes totalling 4 475.80 m. A total of 63 600.80 m in 277 drill holes was drilled on the Mina Justa Prospect by the end of 2008. This included 225 RC only drill holes (total of 52 386.00 m), 51 diamond core only drill holes (total of 10 764.80 m) and one combined RC and diamond core drill hole (total of 450.00 m). In addition, five RC holes drilled in 2007 were extended by four RC extensions totalling 516.00 m and a single diamond core tail of 225.30 m.

Notes: RC = reverse circulation. Numerous drilling contractors were used, including: Bradley (Lima), Geotec (Lima), HYS Drilling (Lima), MCA (Lima), Sonda Sur (Lima). Numerous drilling rigs were used, including: Longyear LF 70, Marca CBC CS-1000, Marca CBC CS-3000, Schramm T-660 Rotadrill, Foremost W750 Prospector, and CSR-3000.

A combination of reverse circulation (drilling 5.25 inch diameter) and diamond core drilling (HQ diameter core reducing to NQ diameter with depth, where necessary) was used. Drill hole inclinations and directions were selected and adjusted to intersect the mineralisation perpendicular to the structural trend and the interpreted trend of the copper mineralisation. Appropriately labelled concrete drill hole collar monuments with capped PVC piping oriented along the drilling direction permanently mark the location of each drill hole collar.

Drill hole collar surveys (surveyed using a Leica Total Station Model PCR 305 digital theodolite with mm scale accuracy and a range of up to 3.5 km) and gyroscopic downhole surveys (using an SRG borehole gyroscope with a ±0.1° accuracy manufactured by Goodrich Corporation of the USA) were conducted on the drill holes by independent surveying contractors (Proyectistas Tecnicos y Servicios S.A. of Lima, and Comprobe S.R.L. of Chile, respectively). Continuous downhole survey readings were collected, with time average readings made every 10 m downhole. Additional check readings were made every 50 m on the probe's return back up the drill hole.

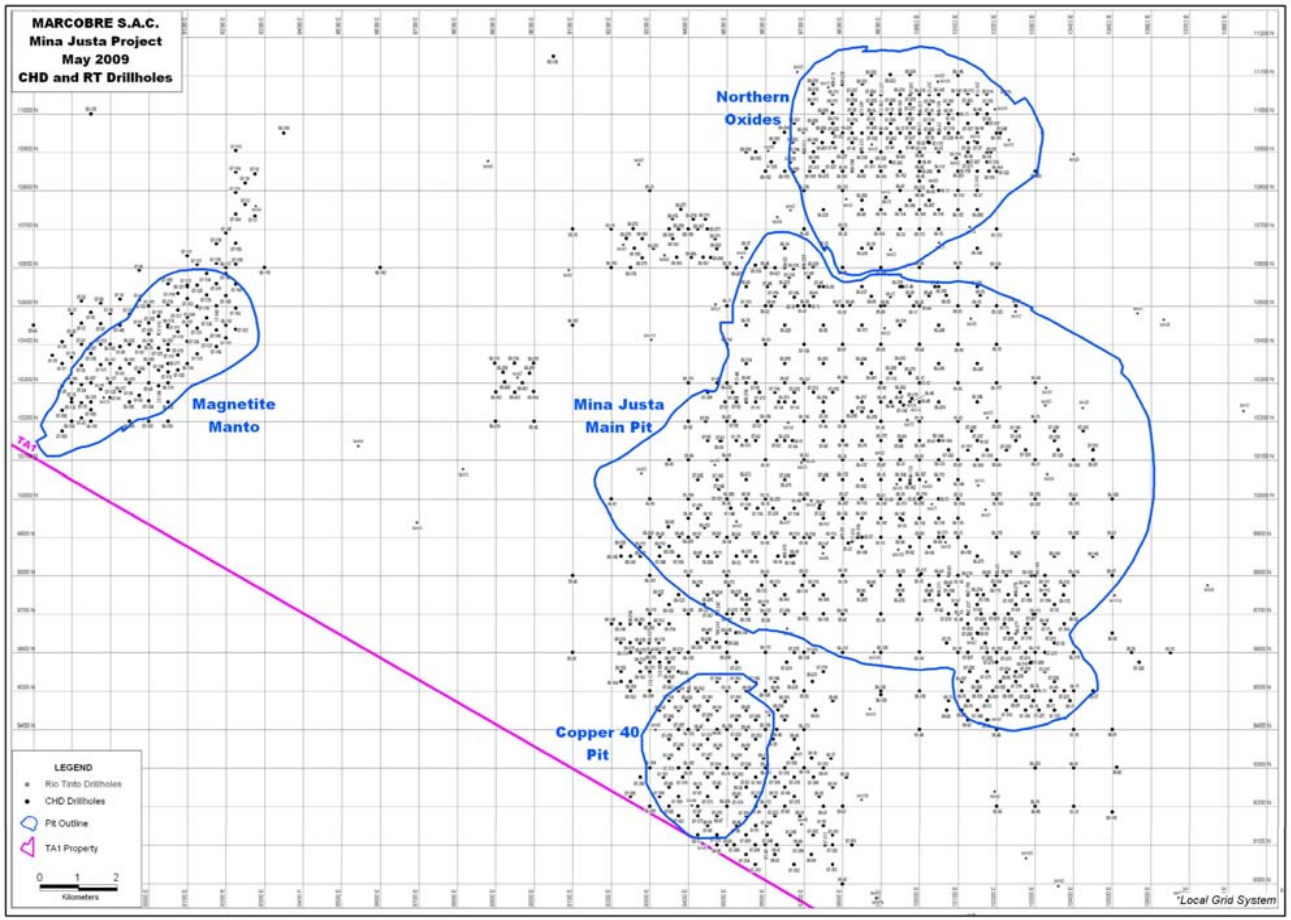
Drilling in the Mina Justa copper deposit covers an area of approximately 7.5 km^2, with drill holes spaced between 25 m and 50 m apart (generally 35-40 m) and drilled to depths of up to approximately 630 m. Drilling in the Magnetite Manto copper deposit covers an area of approximately 0.23 km^2, with drill holes spaced between 25 m and 50 m apart (generally 25-30 m) and drilled to depths of up to approximately 410 m.

Drill hole core recovery is generally better than 95% (with more than 90% of the recovery data displaying a recovery of better than 90%).

Marcobre attempted to quantify RC sample recovery, the results indicating a recovery generally better than 85%. The apparently low result for what should be a continuous sample is, however, considered to be a function of significant density variation in the deposit precluding an accurate recovery assessment for the RC samples.

Figure 11.1

Mina Justa Prospect drill hole location plan (as at August 2008)



12. SAMPLING METHODS AND APPROACH

Marcobre and Snowden are unaware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. Samples are considered representative of the mineralisation in the Mina Justa Prospect. Summary details of all mineralisation intersections from the pre-2005 Rio Tinto, and the 2005 through 2008 Marcobre drilling programmes are included in Appendix 3.1 to the DFS.

A total of 127,868 samples were collected in and around the Mina Justa Prospect. All samples used for the October 2008 mineral resource are from drilling. A consistent sampling method and approach was maintained by Marcobre for each year's drilling programme.

Drill hole core was logged for geotechnical and geological features prior to being marked for sampling core cutting. Core sampling was conducted with respect to geological boundaries. Core sample intervals were generally 1 m for mineralised core and 2 m for non-mineralised core. Drill hole core sample recovery is generally better than 95% (with more than 90% of the recovery data displaying a recovery of better than 90%). Density measurements were conducted on selected core intervals after logging and before sampling. The standard weight-in-water-weight-in-air technique was used.

RC chips were collected at regular intervals and logged. RC samples were collected over 2 m intervals and riffle split to achieve 12.5% splits of approximately 10 kilograms. Marcobre attempted to quantify RC sample recovery through comparing sample mass, the results indicating a recovery of better than 85%. The apparently low result for what should be a continuous sample is, however, considered to be a function of significant density variation in the deposit precluding an accurate recovery assessment for the RC samples.

Reject and reference samples were stored in camp. Prepared coarse and fine blanks, oxide and sulphide standards as well as field, crush and pulp duplicates were inserted into the sample stream.

13. SAMPLE PREPARATION, ANALYSES AND SECURITY

Sample preparation conducted on samples collected during Marcobre's 2005 through 2008 drilling programmes was carried out by the on-site preparation facility operated by SGS del Peru SAC laboratory personnel. The RC samples were dried and crushed to 95% passing #10 mesh. The crushed sample was then riffle split to produce a 250 g sample, which was then pulverised to 95% passing 200 mesh. The sample pulp was then submitted to the SGS laboratory in Lima for analysis in complete drill hole batches. Coarse sample preparation rejects were bagged and stored on site. Following analysis, the pulp sample was returned to Marcobre for storage on site. Half-core samples were also prepared following the same sample preparation as protocol as the RC samples through to the pulp sample stage on site.

The SGS laboratory in Lima was the primary laboratory to which all drilling samples collected from Marcobre's 2005 through 2008 drilling programmes were submitted. The SGS Laboratories Quality Assurance System has ISO 9002 accreditation and participates on a regular basis in round robin testing with analytical laboratories in Canada, Sweden, and the USA, amongst others.

All sample pulps received were entered into the laboratory management system and uniquely bar-coded for Quality Assessment and Quality Control (QAQC) and tracking purposes. All preparation and analytical data recorded for the samples was doe electronically. Marcobre submitted a total of 142 750 samples (including QAQC samples) during the 2005 to 2008 drilling campaigns in the Mina Justa prospect.

All of the samples were analysed for total Cu (CuT) and sequential leaching (sulphuric acid extractable, cyanide extractable and residual Cu[5]) with an AAS finish resulting in four Cu assay values per sample. In addition, sulphide and transition zone samples were analysed for Ag using ICP-OES analysis with an aqua regia digest as part of a multi-element package (including Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, S, Sb, Sc, Se, Sn, Sr, Te, Ti, Tl, U, V, W, Y, Zn and Zr). Au analyses were carried out using a 30 g fire assay with an AAS finish.

The assay data loaded into the database has been verified against the original laboratory certificates that are kept on file in the Marcobre Data Room. Marcobre's John D. Kapusta, P.Geo., Vice-President Exploration and Geological Services is the Qualified Person responsible for Marcobre's exploration, drilling, sampling and data quality.

Marcobre's site security includes a private road, entrance gate and around-the-clock site-based security guards. The SGS site-based laboratory is securely locked. SGS take custody of all samples on site, once they have been appropriately bagged and labelled. Following sample preparation, sample pulps are transported by road (in the care of SGS) to SGS Lima for analysis. The SGS laboratory in Lima is completely surrounded by a security wall and all access is security controlled.

Marcobre and Snowden consider the sampling, sample preparation, security and analytical procedures to be of adequate quality to support the generation of mineral resource and mineral reserve estimates to Feasibility Study standard.

[5] CuSS, CuCN & CuR, respectively

14. DATA VERIFICATION

Details of data verification conducted on the pre-2005 Rio Tinto data are presented in the AMEC report (2004).

A quality assurance programme (QA) was established by Marcobre in 2005 to verify and monitor the analytical results provided by the primary laboratory, SGS Perú. The QA established protocols for insertion of quality control (QC) samples, evaluation criteria, and secondary laboratory check analyses.

The QC samples inserted by Chariot into the sample batches submitted to the SGS laboratory included the following:

- 11 oxide standard reference materials (SRMs)
- Eight sulphide standard reference materials
- Coarse blank materials (quartz 1" or quartz ½")
- Fine blank materials (silica -10 mesh and -140 mesh)
- Field or drill duplicate samples (¼ core or 1/8 RC drill cuttings split by Chariot)
- Coarse duplicates (split of -10 mesh reject material by SGS)
- Pulp duplicate samples (split of -140 mesh sample by SGS).

Assay results are delivered to Marcobre in both digital (*.txt) and hard copy certificate format. Digital text files are formatted to mimic that of the original sample submission. SGS laboratory standards, blanks and duplicates are appended to the text-file for a given batch. The batch files are imported into an acQuire database and the batch QAQC assessed using pre-defined QAQC parameters prior to being appended to the sample database. Marcobre used the recommended acceptance values for the various materials to flag potentially problematic batches, which were then investigated in greater detail to ascertain whether the batch, or portions thereof, were to be rejected and re-analysed by the primary laboratory.

The oxide and sulphide SRMs have been prepared from material from the Mina Justa Prospect and certified by the SGS laboratory in Lima for CuT, and sequential Cu (CuSS, CuCN, CuR). Some of the sulphide SRMs were also certified for Au and/or Ag. Certified blank materials were acquired from the SGS laboratory. The rate of QC sample insertion was approximately 10% throughout the drill programme.

Marcobre submitted approximately 10% of the samples collected from the pre-2005 Rio Tinto exploration programme to the SGS laboratory for cross-check purposes. Marcobre also submitted all available Rio Tinto field standard pulps for round robin recertification (the original certificates for these standards have never been sourced by Marcobre), so that additional QAQC checks could be made on the Rio Tinto sample data. In addition, Marcobre submitted a total of 13 133 sample pulps or approximately 9% of the assay samples from the 2005 to 2008 drilling campaigns for secondary analysis at two third-party laboratories, namely ALS Peru S.A and Actlabs Peru SA. All samples were re-analysed for CuT and sequential Cu, and samples within the transition and sulphide zones were analysed for Au and Ag.

Marcobre's QAQC data have been reviewed by independent consultants throughout the drilling campaign (e.g. Snowden 2008d). Results of these reviews indicate that the available total copper, sequential copper, silver and gold data from Marcobre's 2005 through 2008 drilling programmes are of acceptable quality (i.e. sufficiently accurate, precise and without cross-contamination) for use in mineral resource and mineral reserve modelling to feasibility study standard.

Snowden's Dr. Warwick S. Board, P.Geo., visited Marcobre's field- and Lima-based operations, the SGS on site preparation laboratory and the SGS laboratory on several occasions throughout the programme (in September 2005, December 2006, April 2007, April 2008 and June 2008), and observed drilling, collar and down-hole surveying, drillcore and RC chips, sampling, sample preparation, chain-of-custody, sample analysis, sample QAQC, density measurements, assay certificates and database maintenance, and geological interpretation, as well as taken independent samples from selected drill holes.

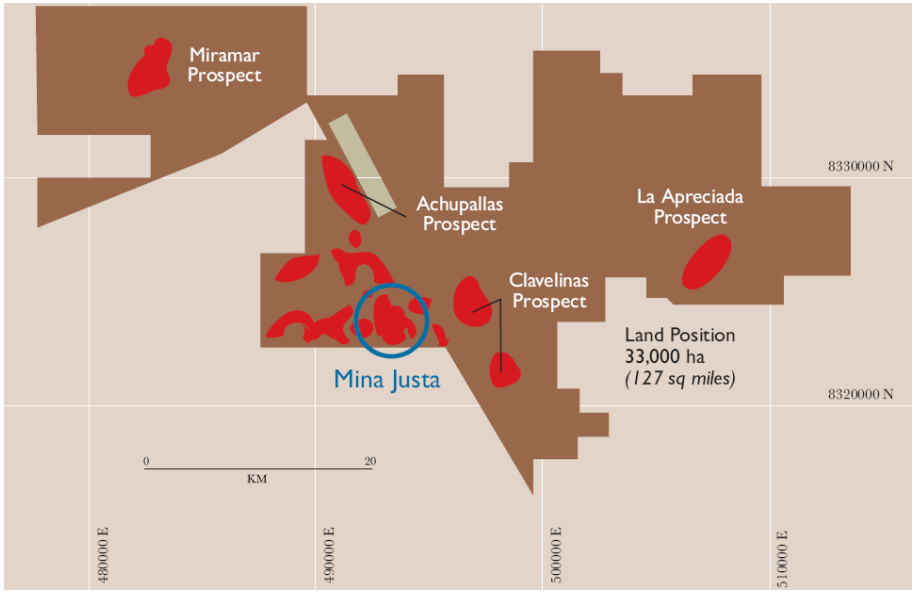
Based on observations made during the numerous site visits, as well as numerous independent assessments of Marcobre's data, Snowden is of the opinion that Marcobre's drill hole sampling data is of adequate quality to support the generation of mineral resource and mineral reserve estimates to feasibility study standard.

Snowden has not reviewed the results of the check analyses conducted by the independent third party analytical laboratories. Snowden has not reviewed the quality of Marcobre's post-23 May drill hole data, as this information was not available at the cut-off date for the October 2008 Mina Justa Prospect mineral resource estimation process.

15. ADJACENT PROPERTIES

The western portion of Marcobre's TA1 property boundary is coincident with the eastern boundary of the Marcona iron mine, owned by Shougang Hierro-Perú S.A (see Figure 4.1 and Figure 15.1). It is the largest known magnetite deposit in South America, estimated at approximately 1400 Mt iron ore grading 54.1% Fe and 0.11% Cu (AMEC, 2004) dispersed in at least eight major and 47 minor orebodies. As of 2002, 358 Mt of ore had been produced. Current production is stated as between 4.5 Mt and 5 Mt per annum according to Shougang's website. Recent research indicates that the Marcona iron deposit and Mina Justa Prospect are not coeval, despite their proximity; instead their parageneses represent separate mineralising events occurring approximately 60 Ma apart under widely different conditions (Chen, 2008).

Figure 15.1
Copper prospects identified within the Marcona Copper Property area



After Chariot (2007).

The Marcona Copper Property completely surrounds the Milagros property owned by Minera del Norte S.A. This property is characterized by a magnetic anomaly in the vicinity of historic workings with weak copper and molybdenum anomalies hosted in andesite. Unoxidised primary magnetite hosted in the matrix of the volcanic rocks is thought to be the cause of the magnetic anomaly in this area. Marcobre and Snowden are unaware of any additional work conducted on the Milagros property.

Four copper prospects have, to date, been identified within the Marcona Copper Property in areas adjacent to the Mina Justa Prospect, comprising:

- Achupallas Prospect, located about 6 km north of Mina Justa
- Miramar Prospect, located about 18 km northwest of Mina Justa
- Clavelinas Prospect, located about 5 km east from Mina Justa

- La Apreciada Prospect, located about 15 km east of Mina Justa.

Limited exploration has been conducted on these prospects to date, and no economically significant mineralisation has been recorded.

Although not situated adjacent to the Mina Justa deposit, the district is host to another giant magnetite deposit, the Pampa de Pongo, which is located approximately 30 km to the southeast of the Mina Justa Prospect. Pampa de Pongo is hosted by Jurassic andesites and intercalated sedimentary strata, containing an Inferred resource of 863 Mt grading 41.3% Fe, 0.1% Cu and 0.07 g/t Au at a 15% Fe cut-off grade, as of 30 September 2008 (Cardero Resource Corporation News Release NR08-27, 29 October, 2008).

16. MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testwork has been completed on representative sub-samples of the Mina Justa and Magnetite Manto ore to determine the processing requirements. This section outlines the metallurgical development process used to select the processing flow sheets for the Mina Justa Project Oxide and Sulphide ores.

16.1 OXIDE ORE

16.1.1 Comminution testwork

Testwork was initially conducted by Metso Minerals on four samples of Mina Justa Oxide ore. These test results generated a Macon crushability result of 28%, indicating that the Oxide ore is competent and difficult to crush. Following this preliminary testwork, a comprehensive testwork program was recommended and completed.

Samples for the Mina Justa Vat Leaching Pilot Plant campaign (Phase 2) were drilled and collected by Marcobre during 2007. As part of this drilling program, samples for the DFS crushing testwork program were also selected. The aim of the selection process was to provide samples that represented the main ore types of the Oxide deposit, provide samples that varied with depth, and provide samples that varied with location. The samples show two main lithologies, namely sedimentary and andesitic. The sedimentary lithology is classified as arkosic and medium grained. The alteration associated with this lithology is generally chlorite/orthoclase. The andesitic lithology often contains amygdaloidal vesicles. The alteration associated with this lithology is generally chlorite/actinolite.

The crushing testwork program was initially conducted at Phillips Enterprises LLC (Phillips) in Colorado, USA. The tests were conducted on 21 samples using approximately 20 rock specimens per sample. Testwork, using a modified Bond test machine, generated results that were lower than anticipated. GRD Minproc subsequently requested that further testwork be conducted to investigate the lower than expected values generated by the Phillips results. It was recommended that Bond crushing work index tests be completed at an alternative facility to cross-check the initial results. Ammtec Limited (Perth, Western Australia) was selected due to the level of previous validation undertaken by Ammtec to confirm the performance of its test unit. A total of twelve samples of varying numbers of rock specimens remaining from the Phillips testwork were used for the additional tests. It was confirmed that the Ammtec results were higher (by 45%) than the Phillips results. The Ammtec testwork and results were selected for detailed analysis and generation of the Bond crushing work index design values, while the Phillips testwork results were used to further investigate trends generated from the Ammtec results.

The crushing work index results were assessed in terms of lithology, depth, and spatial distribution to determine if there were any significant trends. Assessment of the effect of down hole depth did not show any significant differences in the Bond crushing work index results, and therefore the crushing characteristics are not expected to vary with depth. Similarly, no significant trends were identified with regards to spatial distribution. Assessment of the ore lithologies did highlight some differences in the Bond crushing work index results, although the results were not consistent between the Phillips and Ammtec data sets. The Ammtec results tended to show similar results between lithologies with the

amygdaloidal andesitic material being possibly marginally more competent. The Phillips testwork results, on the other hand, indicated that the sedimentary material was slightly more difficult to crush.

The crushing work index results obtained did show a large range of results between specimens for each sample tested. For example, sample SC04 from hole MJV-07-140D produced CWi values which ranged between 1.7 kWh/t and 17.6 kWh/t. Other samples also exhibited a large degree of variability from individual rock specimens tested.

The crushing testwork results showed a proportion of the rock specimens tested were more competent than the main sample set. The increase in competency for these samples reflects rock with minimal inherent fractures. As the material is crushed finer (and the rock fractures are removed), the competency of the ore is expected to increase. This trend has been used in the selection of CWi values for the design criteria. A Bond crushing work index value of 10 kWh/t was selected for the primary to tertiary crushing stages, increasing to 16 kWh/t for the quaternary stage.

Unconfined compressive strength (UCS) testwork was undertaken by Advanced Terra Testing Inc (under the direction of Phillips). The overall dataset showed an average UCS value of 48.2 MPa, and a maximum value of 130.2 MPa.

Bond abrasion index tests were conducted by Phillips on the two sample types from the main programme. The results showed that the sedimentary material was less abrasive than the other lithologies, with an average abrasion index of 0.13. The andesitic material was more abrasive with an abrasion index of 0.22, increasing to 0.24 for the amygdaloidal andesitic material. Limited results were obtained for the Magnetite Manto deposit, which showed a large variation in results (between 0.08 and 0.32) for the different lithologies tested. On average, the two deposits are expected to be moderately abrasive with an abrasion index value of 0.19.

Based on the parameters generated during the comminution testwork program, modelling of potential crushing circuits was undertaken. Several flowsheet options were modelled, including three and four stages of crushing, open and closed circuit secondary crushing, crushed ore stockpile options and various equipment configurations. From this modelling, the optimal circuit design was determined to be a quaternary crushing circuit, with the secondary stage in open circuit, and the tertiary and quaternary stages in closed circuit.

16.1.2 Leach testwork

Testwork on the Mina Justa Oxide material followed a typical development pattern. Testing started with bottle roll leaching, moved to individual column tests, and finally evolved into an integrated pilot program run continuously in locked cycle. Pilot testing was followed by an on-going program of variability testing on material from various areas of the proposed Mina Justa and Magnetite Manto open pits.

Bottle roll testing commenced prior to Marcobre's acquisition of the project. In the Marcobre tests, the principal variables were crush size (1 to 25 mm), acid level (pH 1.2 to 2.5) and lithology (andesite or sedimentary). Both the Rio Tinto and Marcobre results demonstrated that the Mina Justa Oxide ore is inherently leachable. As might be expected, recovery generally declined as the crush size or pH

increased. However, both andesite and sedimentary material achieved 100% extraction of the acid soluble copper (CuSS) at fine crush sizes (1 and 3 mm top size) and high acid levels (pH <1.5). Recovery from andesite was found to be more sensitive to both crush size and pH than the sedimentary rock. Acid consumption by the andesite was also more sensitive to these variables than it was for the sedimentary material. Ancillary bottle roll tests suggested that a high chloride level would reduce acid consumption, but retard copper extraction.

A column testing program followed, based on the results of the bottle roll tests. Initially, the main focus was towards identifying the design parameters for some type of heap leach operation. Testing started with 12 trials on the andesite composite used in bottle roll tests. Variables included crush size, acid cure dosage, irrigation rate, acid concentration in the leach solution and test duration. The results were somewhat discouraging. While high recoveries could be achieved (>90%), long leach cycles (three to five months) and high acid levels were needed. The resulting gangue acid consumption (GAC) approached 100 kg/t, with specific consumptions of 11 to 20 kg acid/kg Cu. The runs with the lowest specific acid consumption (7 kg acid/kg Cu) achieved recoveries of only about 70%, with GAC levels just below 50 kg/t.

Results were no better for columns charged with sedimentary material. In general, recoveries were lower, while the GAC levels were higher. The problem proved to be one of relative kinetics. The initial copper leach rate was fast, with about half the total recovery occurring in the first 10 days. After that, the leach rate slowed dramatically, typically taking another three or four months for the recovery to double. The rate of acid consumption behaved quite differently, rising linearly with time over the entire leach cycle. Thus after 10 days, recovery was reasonably good and acid consumption was still very low. However, after another three to four months of leaching, Cu extraction doubled, but GAC increased by a factor of 14 or 15 times.

One set of tests showed that increasing the irrigation rate improved copper recovery without increasing acid consumption. This observation and the kinetic factors led to a key series of experiments using composite FSOX-14. In these tests, flows were increased in stages from 10 to 40 L/h.m^2. At the same time, the acid concentration in the leach solution was decreased proportionately, so that after 55 days of leaching, each test received the same total quantity of acid. The results were quite dramatic. Recovery increased as the flow rate increased. At the same time the total acid consumption, GAC and specific acid consumption all declined progressively as the flow increased and the acid concentration decreased.

These results led to the conclusion that vat leaching with its high flow rate, short cycle time and good wash efficiency might be the most effective process route for the Mina Justa oxide material. Therefore, a vat testwork program was developed, starting with a series of batch trials. These tests generally confirmed earlier studies, which showed that the pH needed to be below 1.5 to give effective leaching. Recovery generally increased as the crush size decreased. Although most head samples showed significant upgrading in the finer fractions, the residue assays showed no such effect. Thus, the fines leached better than the coarser material. Virtually no acid soluble copper remained in the minus 10 mesh residues, confirming the inherent leachability of the ore. While the GAC values in kg/t tended to increase with finer crushing, the specific acid consumption (kg acid/kg Cu) actually declined in many cases. These results suggested that operating on a six day leach cycle with material crushed to between 6.0 and 9.5 mm would provide an optimum vat process.

An effort was also made to develop some perspective on the range of leach behaviour that might occur in operations. This involved testing forty FSOX composites from various locations under identical (but non-optimised) conditions. Recovery based on total copper (CuT) analyses averaged 58%, with a range of 13 to 84%. When samples containing less than 0.3% Cu (the expected cut-off grade) were deleted from the database, the average recovery increased to 62.5% of CuT (equivalent to 75% of the acid soluble copper). The average GAC level was 44 kg/t, but ranged from 19 to 139 kg/t.

Positive results in the batch vat tests prompted a decision to proceed with testing in an integrated vat pilot plant that would operate continuously in locked cycle. The Phase 1 tests utilized four composites: low-grade and mid-grade samples from the Mina Justa deposit, and mid-grade and high-grade samples from Magnetite Manto. During this phase, some problems with acid control and copper stripping from the pregnant leach solution were experienced. In spite of these, the Phase 1 results showed that crushing to 6 or 8 mm gave the same recovery and the same residue assays. However, the 9.5 mm crush size was too coarse and led to a drop in recovery. The results also showed that acid cure dosages of 10 to 20 kg/t had minimal effects on the leach performance. Over this range both extraction and GAC values were actually highest at the lowest cure dosage.

Overall, the higher grade Magnetite Manto material gave the best leach performance. It had the highest recovery of both total and acid soluble copper. It also had lowest average GAC value (24.5 kg/t) and the lowest specific acid consumption, less than 4 kg acid/kg Cu.

At the end of Phase 1 testing, samples of the final leach solutions from each ore type were sent to the solvent extraction reagent vendors for compatibility studies. The results were positive, with no copper transfer or phase disengagement problems noted.

After correcting operational problems (acid control and copper stripping), a second pilot plant campaign (Phase 2) was conducted. All Phase 2 testing was done using the optimal process conditions established in the Phase 1 tests. The 37 Phase 2 samples were selected to provide a variability program that was intended to demonstrate the effects of various resource parameters such as head grade, mineralogy, lithology, depth in the deposit and blends based on the then-current mine plan. When the test results were evaluated, the back-calculated head assays were found to be biased low. As a result, head assays provided more accurate determinations of recovery and GAC.

While the extraction of copper is undoubtedly influenced by the various resource parameters, for the purposes of the DFS the key relationship is the one between recovery and head grade. The latter can be expressed either in terms of CuT or CuSS, with the correlation being better for CuSS. Regression analysis showed that the grade-recovery relationship was positive and has the following form:

$$\text{Recovery of CuT (\%)} = (86.5 + 9.3 \times \text{CuSS}) \times (\text{CuSS}/\text{CuT})$$

Maximum copper recovery is capped at 95% of CuT. This prevents the projection of 100% extraction from high grade ore.

Over the life of the mine, applying the recovery equation to the minable reserve on a block-by-block basis, the average recovery of CuT is 74.5%, representing 92.0% of acid-soluble copper.

The other important numerical expression required for such activities as mine planning and financial analysis is the relationship between head grade and acid consumption. Here, regression analysis showed that total acid consumption was independent of the head grade, indicating that total acid consumption is primarily dependent on gangue characteristics. Regression analysis demonstrated that GAC was related to head grade, but that the relationship was negative (GAC declined as the grade increased). This was caused by the increased acid credit from higher grade material. Initially, a simple linear correlation was established for the relationship. However, further evaluation showed that an exponential expression fits the data better and has a higher correlation coefficient. Both are strictly empirical fits of the data. Numerically, the exponential relationship is expressed as follows:

$$\text{GAC (kg/t)} = 50.07e^{(-0.47 \times \text{CuSS})}$$

Theoretically, the GAC value could go negative if the electrowinning (EW) acid credit were greater than total acid consumption. However, there is no data below 15 kg/t, so the extrapolation of GAC to lower values cannot be validated. Thus, there was discussion that the minimum GAC value should be capped at 15 kg/t. However, this cap was not utilised when the expression was applied to the block model. As a result, the highest grade ore (<1% of the total Oxide resource) was projected to have GAC values as low as 9.9 kg/t.

Applying this relationship to minable reserves on a block-by-block basis results in an average GAC of 40.7 kg/t, or nearly 0.5 Mt/a of acid required to be imported.

Several other design parameters were determined in conjunction with the Phase 2 tests. These included:

- Total suspended solids (TSS) in the vat overflow. Clarification tests on the PLS were conducted to provide the design basis for the PLS pin bed clarifier.
- The viscosity of the leach solution was also checked. It averaged 1.5 cP, but increased gradually during the tests and ended at 1.8 cP.
- The final moisture in the leach residues ranged from 9.0 to 16.4%, with an average of 11%.
- On average, the void space in the ore bed decreased from 42% to 37.5% during the six day leach cycle.

Efforts were made to evaluate the effect that the various resource parameters have on recovery and acid consumption. Due to interactions between mineralogy, lithology, alteration and depth, isolating the impact of any one parameter proved difficult and the results are probably only semi-quantitative at best. With regard to mineralogy, one group of samples with low recovery and a high average GAC value contained significant sulphide mineralisation, which was responsible for the poor results. Two other groups with high recovery and low GAC values included high-grade material, which drove the high extraction and low acid consumption. The remaining six mineralogical groupings produced results that were within about ±5% of the average for all groupings. This indicates that the mineralogy alone generally does not have a big impact on extraction.

The effects of lithology, too, were difficult to isolate. One group of samples with poor performance contained both a low grade sample (0.20% CuT) and the sulphidic material. By contrast, two groups that contained the high grade material performed well. The remaining seven groupings did not differ greatly, suggesting that lithology does not have a major effect.

One resource parameter that did appear to affect performance was depth in the deposit. Surface and near-surface material typically yielded higher recoveries than deeper material. This may be related to near-surface weathering and better solution access to the mineralisation.

The number of samples that could be run in the pilot plant was limited by time and the amount of material required. Therefore, once pilot testing was complete and the results evaluated, a follow-up variability testwork program was initiated. This involved testing of more than 200 samples from various locations in both Mina Justa and Magnetite Manto. One of the objectives was to characterize material so that the results could be used in the block modelling work. Another was to develop a procedure that would provide a fast, simple test to be used on blast hole material to estimate leach behaviour during operations. A large-scale bottle roll procedure was adopted for these tests. Unfortunately, when applied to the 200 samples, the results did not replicate those obtained in the pilot tests. Both the recoveries and acid consumption in the variability tests were higher than expected. In fact, recoveries for the Transition ore (<80% acid soluble) actually exceeded the acid soluble content of the samples. Additional testwork has been undertaken to resolve the differences and improve the variability test procedure. However, the results are still being assessed and it appears that further work may be necessary to develop a viable procedure to support grade control and classification of ore type.

16.1.3 Solvent extraction and electrowinning testwork

Metallurgical process development for the Mina Justa SX operation has been drawn primarily from vendor testwork/modelling and verification of the PLS characteristics from data generated in the Phase 2 vat leaching testwork campaigns. A number of circuit configurations have been considered during the testwork and modelling phases to determine the optimum circuit for the Mina Justa project.

Samples generated from the vat leaching pilot plant were sent to the Cognis laboratory in Chile for batch scale SX testing. These samples were used to generate McCabe Thiele isotherms for reagents LIX973N and LIX612N-LV at concentrations of between 20% v/v and 28% v/v.

Modelling of the SX process was subsequently carried out by Cognis to investigate a number of different circuit configurations. This demonstrated that the required mass transfer of copper could be achieved from a number of circuit configurations, including series parallel, optimum series parallel, series and dual SX trains in a series configuration. From a review of the possible configurations and assessment of the capital and operating cost implications, the 2E+1W+1S (i.e. 2 extraction stages, 1 wash stage, and 1 strip stage) was selected as the preferred circuit configuration.

Cognis conducted further modelling using the selected 2E+1W+1S circuit at a copper tenor of 8 g/L and revised pH of 1.9. These simulations were run using both LIX973N and LIX84-I as they both offer performance benefits at this pH. The modelling was conducted at several extractant concentrations and organic to aqueous (O:A) ratios, and showed that the LIX84-I reagent was superior.

Samples generated for SX testwork were also sent to Cytec in Peru, to be analysed at independent laboratories. The analyses showed high chloride levels of up to 6 g/L which is included in the design. Cytec conducted a comparison of two potential circuit configurations, namely, 2E+2S (2 extraction stages, 2 stripping stages) and 2E+1S (2 extraction stages, 1 stripping stage). The simulations also included the impact of varying pH levels and extractant concentrations. The results obtained confirmed that acceptable copper recoveries are achievable for both circuit configurations. The 2E+2S circuit would require a lower extractant concentration to achieve the same recovery as the 2E+1S circuit, however, an additional stage of mixer/settlers would be required. Further modelling exercises confirmed the selection of the 2E+1W+1S as the preferred circuit configuration.

An extraction of 94% of the soluble copper in the PLS stream is expected to be recovered through the SX circuit, based on a PLS stream containing 8 g/L copper at a pH of 1.9. An extractant concentration of 25% v/v is required for effective extraction of the soluble copper. Stripping of copper from the extractant is carried out using an electrolyte solution containing 35 g/L copper and 180 g/L sulphuric acid. These electrolyte parameters are relatively standard for copper solvent extraction systems in plants world-wide.

Full ICP scans of the PLS solution from the Phase 2 vat leaching testwork program were conducted. The ICP data from the vat leaching solutions shows several impurities that need to be considered in the design of the SX and EW circuits. These include iron, manganese, silicon (colloidal silica) and chlorides. Mitigation measures in the SX plant design include a wash mixer/settler and coalescer tank to remove aqueous entrainment carrying iron, manganese and chloride. All three impurities can impact the EW operation and will be further controlled via the electrolyte bleed. Equipment for treating the organic stream with activated clay to mitigate the effects of colloidal silica has been included in the crud treatment area of the SX plant.

16.2 SULPHIDE ORE

Several testwork campaigns were completed during the evaluation of the Mina Justa sulphide deposit at a PFS level of detail. The sample collection and testwork were directed by Transmin Metallurgical Consultants (Transmin).

16.2.1 Comminution testwork

Comminution testwork was conducted on selected intervals to represent Transitional, Primary, and Secondary[6] ore types. The initial testwork campaign was conducted at SGS Lakefield Research in Chile during 2006, and was limited to Bond abrasion, rod mill and ball mill work index tests. The testwork results obtained indicated that all three samples tested were of moderate grindability and abrasiveness.

The second comminution testwork campaign was conducted in 2008 at SGS Lakefield Research in Chile, and by JKTech in Australia. With the exception of a more elevated Primary Sulphide abrasion index, the results were similar to those tested in Campaign 1. A Mixed Sulphide ore sample returned

[6] Primary ore = Cpy dorminant; Secondary = Bn-Ce dorminant

relatively low rod mill and ball mill indices. This indicates that the ball mill grinding energy for this material type will be low (approximately 50-60% of that required for the other ore types).

The SMC test results showed a similar trend to that found with the Rod and Ball work indices. The highest competency ore type was Transitional ore, which was classified as very hard, while the Secondary Sulphide sample displayed moderate to hard competency. The Primary ore and Mixed samples displayed an average to soft classification. The large range in ore competency indicated by this preliminary testwork was a concern for the selection of comminution design criteria, particularly for a SAB or SABC type circuit.

To further investigate the comminution characteristics of the Sulphide ore, a third testwork campaign was conducted in late 2008. The testwork performed by Laboratorio Plenge in Peru included ball mill work index and SMC tests, with the results of the SMC tests evaluated by JKTech in Australia. The Primary and Secondary Sulphide ore samples tested were designated as very hard to hard. The Primary ore from the Cu40 zone was classified as having average competency. The BWi results returned by the Primary and Secondary ores were higher than reported in Campaign 1, which impacts negatively on equipment selection. The testwork results obtained for the Cu40 zone samples indicated significant variability, ranging from moderate to low grindability.

16.2.2 Flotation testwork

A diamond drill program was undertaken during 2005 to provide core for metallurgical testwork. Samples were selected for the first testwork phase based on mineralisation, which was identified as Transitional, Secondary Sulphide and Primary Sulphide.

During the first testwork campaign, the impact of primary grind on recovery, flotation kinetics, and concentrate grades was assessed for the three main ore types. The trends showed that copper recovery increased with the extent of grind. For the Transitional material, the highest bulk flotation recovery of 92% was achieved at a grind size of P_{80} 75 µm. The Secondary Sulphide ore produced a bulk flotation recovery of 98% at a grind of P_{80} 75 µm, with recovery decreasing for the coarser grinds. The P_{80} 106 µm grind grade-recovery relationship was the most favourable, but was not significantly better than 150 µm. For the Primary Sulphide material, the highest bulk flotation recovery of 98% was achieved at a grind size of P_{80} 75 µm, with recovery decreasing for grinds coarser than P_{80} 106 µm. The copper distribution by particle size in the flotation tailings indicated that some copper remained locked with gangue in the coarser size fractions, with improved liberation observed at the finer grinds.

The bulk flotation tests showed that the copper minerals were readily recovered by flotation, but high mass pulls were evident. A rougher concentrate grade of 4.5% at a mass pull of 18% was achieved for the Transitional material. For Secondary Sulphide ore, a rougher concentrate grade of 11.5% at a mass pull of 30% was obtained. A rougher concentrate grade of 3.8% at a mass pull of 26% was achieved for the Primary material.

A series of regrind tests was conducted on the Primary Sulphide and Secondary Sulphide rougher flotation concentrates to determine the optimum regrind size. These tests showed an increase in copper concentrate grade with increased degree of regrind. For the Secondary ore, regrinding rougher concentrate to P_{80} 52 µm resulted in the fastest recovery rates and highest terminal recovery rates, but

with the lowest copper grades. The P_{80} 43 µm regrind and P_{80} 37 µm regrind tests performed at pH11 produced similar results. For the Primary ore, the most favourable concentrate grade, mass pull and recovery resulted from a P_{80} 43 µm concentrate regrind combined with a slurry pH of 12. A concentrate regrind of P_{80} 16 µm for the Transitional, P_{80} 18 µm for the Secondary Sulphide material, and P_{80} 16 µm for the Primary Sulphide material was selected by Transmin for the locked cycle tests.

Cleaner flotation response to sodium cyanide addition was evaluated during this phase. The tests showed a positive response, with improved concentrate grades and recoveries noted. However, the tests were completed at high additions of sodium cyanide; tests at lower levels were not evaluated. Other depressants were also evaluated during this testwork, however, no significant benefits were noted.

Locked cycle tests were performed on the Transitional, Primary, and Secondary ore composites from Campaign 1, to determine the expected flotation performance and concentrate parameters. These tests were conducted at a significantly finer regrind size (between P_{80} 16 µm to 18 µm) than the initial regrind tests. The Transitional ore sample required two stages of upgrading to produce marketable concentrate. The Primary Sulphide ore required two stages of cleaning to produce final copper concentrate grades, whereas, the Secondary Sulphide ore required one stage of cleaning to produce marketable concentrate. A circuit consisting of two cleaning stages, followed by a cleaner scavenger stage was recommended for the cleaner circuit to produce marketable flotation concentrate. Concentrates containing 25% copper or more were produced in all the locked cycle tests after two stages of cleaning.

Composite samples were assembled for the second phase of metallurgical testwork based on the composites of the first phase. The samples were assembled from drill cores not previously used for testwork, in order to test the variability of the different ore types. Testwork conducted during this second campaign was aimed at further exploring flash flotation, rougher flotation and cleaner flotation performance.

Flash flotation tests were conducted with 67% solids slurry after grinding to a particle size P_{80} of 300 µm. The Primary ore produced a copper recovery of 88% at a copper grade of 11%, indicating that, whilst flash flotation may be feasible, achievement of required concentrate grades is unlikely without regrinding and cleaner flotation. The results from flash flotation tests on the Secondary ore produced a recovery of 79% at a copper grade of 30%, indicating that flash flotation is a viable process option for this ore type, the concentrate grades obtained being suitable as final concentrate.

Bulk flotation tests were performed to optimise the reagent scheme. A reagent scheme consisting of promoter A-3477 (isobutyl dithiophosphate) with collector Z-11 (sodium isopropyl xanthate) was evaluated at different dosage rates, and also combined with sodium sulphide addition. A standard primary grind of P_{80} 150 µm was selected for all tests, with stage dosing of Aerofloat 3477 promoter prior to milling and 10 g/t xanthate collector before flotation. The addition of sulphidiser did not result in significant grade improvement, but final recovery was negatively affected. Promoter at an addition rate of 25 g/t produced a lower mass pull, higher copper grade and faster recovery rates than the other levels tested.

Bulk flotation tests were conducted on the Secondary Sulphide ore to optimise the slurry pH and to review the impact of sodium sulphide addition. The tests showed an increase in recovery as the pH was increased from 8.6 to 11. A slurry pH of 10 produced an acceptable recovery with the lowest mass pull and highest copper grade. Sulphidiser addition resulted in some grade and recovery improvements.

Cleaner tests were conducted using rougher concentrates generated with promoter (A3477) and collector (Z-11), with regrinding to P_{80} 37 µm. The Primary ore type generated a 15% copper concentrate after four minutes of flotation, whereas the Secondary ore type produced a 44% copper concentrate.

A third testwork campaign aimed at optimising regrind and cleaner flotation performance was completed in early 2009. A series of tests was conducted on the Primary Sulphide and Secondary Sulphide rougher flotation concentrates to determine the optimum circuit parameters. The composite samples evaluated in this phase of testwork were similar to the second phase samples, although Primary ore samples from the Cu40 zone were also included.

Regrind and cleaner tests excluding sodium cyanide dosing were conducted on Primary Sulphide and Secondary Sulphide rougher flotation concentrates. The Primary ore results indicate that 25% copper concentrate can be produced at PSD P_{80} of 37 µm and 43 µm, without cyanide addition. The results suggest that overall copper recovery decreases as the extent of regrind increases. Improved copper recoveries to concentrate were achieved in the third round of testwork, when compared with the results from the preceding test campaigns. The Secondary ore regrind tests showed an increase in copper concentrate grade and recoveries compared with the results obtained during campaign 1. Higher copper concentrate grades were produced at PSD P_{80} of 49 µm and 57 µm, again without cyanide addition. Transmin selected P_{80} 37 µm and P_{80} 49 µm for the Primary and Secondary ores, respectively, as the basis for the additional variability and locked cycle tests performed during the third testwork campaign.

Variability batch flotation tests with three cleaning stages and locked cycle tests were performed on similar composites. In general, lower bulk flotation mass pulls were observed in this campaign, when compared to the first two testwork campaigns. A revised reagent scheme was utilised, with reduced collector (Z11) and promoter (A3477) dosages. Concentrates containing 25% copper or more were produced from the transitional and secondary ore samples in the locked cycle tests, after two stages of cleaning. Both recovery and concentrate grade improved for the secondary ore. The primary ore tests did not consistently produce 25% copper concentrate after two stages of cleaning, but sodium cyanide was not added during the cleaning stages as had been the case during the first two campaigns. The primary ore concentrate grade was therefore lower than previously reported.

Higher overall recoveries were evident for the primary samples, while the transitional ore tests returned lower recoveries.

Higher overall recoveries were evident for the Primary and Secondary samples, while the Transitional ore tests returned lower recoveries. The Transitional and Secondary ore types showed improvements in final concentrate grades, but the Primary ore concentrate grade was lower than previously reported.

16.2.3 Magnetite testwork

The Mina Justa rougher/scavenger flotation tailings contain recoverable quantities of magnetite. Magnetic separation laboratory test campaigns were performed during the PFS to evaluate the feasibility of producing a saleable magnetite concentrate. The concentrates produced generally contained in excess of 63% iron.

The first testwork campaign was conducted at Laboratorio Plenge in Peru using the flotation tailings produced from the initial testwork campaign. The sample domains tested included Transitional, Primary Sulphide (flotation) and Secondary Sulphide ore. Single pass and triple pass cleaner separation tests were performed after bulk magnetic recovery. The iron grades in the magnetite concentrates were generally below market specification. The levels of copper, sulphur, silica and alumina obtained in the concentrate suggested that intermediate processing of the first-pass magnetic concentrate would be required to liberate locked magnetite and separate gangue.

The second testwork campaign was performed at CIMM Chile using flotation tailings from the second flotation testwork campaign. The testwork included both dry and wet magnetic separation on five samples. The wet magnetic separation tests produced higher magnetite grades in the final concentrates than the dry tests, but were still below typical market specifications. The rougher and first-pass cleaning steps produced substantial grade improvements. Minimal grade improvements were obtained with further magnetic concentration stages. Metal recovery to the rougher concentrate improved with increasing degree of regrind.

The third testwork campaign was aimed at confirming the conceptual magnetite recovery circuit derived from the preliminary testwork. This testwork was conducted by Transmin on five samples at the Pontificia Universidad Catolica del Peru. The testwork further investigated the effect of regrind, with the results showing an increase in the iron content of the magnetite concentrate as the extent of regrinding increased.

The magnetite testwork completed during the campaigns confirmed that a high grade (63% iron) magnetite concentrate can be generated from the Mina Justa flotation tailings. Intermediate processing, which includes regrinding, slimes removal and finishing magnetic separation stages, is required to improve the magnetite concentrate iron grade for all the sample domains. Production of a magnetite concentrate containing more than 63% Fe is not possible without intermediate processing.

16.2.4 Test work design criteria

Copper recovery and concentrate grade is influenced by different parameters, but predictive values were estimated for financial model inputs based on head grade for the PFS. Precious metals recoveries are based on the average metal recovery observed from the trials. The conclusions are obtained from the variability batch flotation and locked cycle tests performed in the third flotation testwork campaign. The parameters adopted for the PFS are shown in Table 16.1.

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Table 16.1
Predictive Concentrator Metallurgy Summary

Mineralisation Type	Metal	Recovery (%)	Concentrate Grade (%Cu)
Transitional	Copper	85	32
Bornite-chalcocite (Secondary)	Copper	min (5.3892Ln(CuT)+90.956), 96	45
Primary (chalcopyrite)	Copper	94	23.5
Precious metals	Gold	80	-
	Silver	80	-

- The bornite-chalcocite ore is a major component for the sulphide resource and proportionately contains the largest metal content. The head grade-recovery relationship derived from the data was positive and has the following form:

  Recovery of CuT (%) = (5.3892 Ln (CuT) + 90.956)

  Note that the recovery is capped at 96%.

- Transitional ore parameters are derived from the grade-recovery relationship obtained from data generated during the third campaign.

- The remainder of the parameters for the Primary and bornite-chalcocite mineralisation samples are based on the average values observed after two stages of cleaning.

The flotation concentrates produced in the locked cycle tests performed in the first two campaigns were submitted for chemical analyses. The results are shown in Table 16.2 and Table 16.3.

Table 16.2
Campaign 1: Final Flotation Concentrate Chemical Analyses

Element	Units	Transitional	Bornite - chalcocite	Chalcopyrite	Market Concentrate		
					Example 1	Example 2	Example 3
Ag	g/t	579	671	62.7	-	-	100
Au	g/t	0.79	0.61	2.97	-	-	4
Cu	%	50.91	39.6	29.7	>27	-	28
Mo	%	0.01	0.01	0.01	-	-	0.35
Fe	%	11.4	20	32.8	-	-	26
Pb	%	0.01	0.01	0.8	<0.5	<1	0.05
Zn	%	0.03	0.01	0.23	<0.5	<2	0.9
K	%	0.04	0.05	0.04	-	-	-
Mg	%	0.57	0.89	0.11	-	-	-
Mn	%	0.09	0.08	0.01	-	-	-
Na	%	0.05	0.06	0.04	-	-	-
Al	%	0.5	0.6	0.13	-	<2	-
Ca	%	1.2	0.8	0.24	-	-	-
Ti	%	0.06	0.04	<0.01	-	-	-
Sb	%	<0.02	<0.02	<0.02	<0.1	<0.05	0.03
S	%	16.2	18.2	32.8	-	-	37
Insoluble	%	10.9	13.3	1.6	-	-	-
As	ppm	103	145	334	<1000	<2000	1800
Ba	ppm	36	43	8	-	-	-
Be	ppm	41	56	79	-	-	-
Bi	ppm	55	45	50	<300	<500	30
Cd	ppm	<2	<2	9	<300	-	-
Co	ppm	145	261	1000	-	-	-
Cr	ppm	195	120	170	-	-	-
Hg	ppm	6.7	0.8	5.4	<20	<10	1.5
Ni	ppm	<2	<2	<2	-	-	100
P	ppm	537	700	369	-	-	-
Sc	ppm	17	13	7	-	-	-
Sn	ppm	<5	<5	<5	-	-	-
Sr	ppm	32	14	11	-	-	-
V	ppm	180	140	12	-	-	-
W	ppm	258	360	199	-	-	-
Y	ppm	8	7	19	-	-	-
Zr	ppm	10	22	21	-	-	-
SiO_2	%	-	-	-	-	-	9
Al_2O_3	%	-	-	-	-	<5	3.5
Cl	%	-	-	-	<0.05	<0.05	0.05
F	%	-	-	-	<0.03	<0.02	0.04
Se	%	-	-	-	<0.05	-	0.02
Te	%	-	-	-	<0.02	-	0.01

Table 16.3
Campaign 2: Final Flotation Concentrate Chemical Analyses

Description	Chemical Analysis (%)					Chemical Analysis (ppm)	
	Cu	Fe	S	Insolubles	F	Cl	U
Specification	-	-	-	-	0.02	500	-
Chalcopyrite 1	22.03	31.3	40.3	1.0	<0.001	<10	<10
Chalcopyrite 2	29.20	28.6	37.7	1.1	<0.001	<10	<10
Bornite-chalcocite	52.08	11.6	18.2	8.2	<0.001	15	<10

The results indicate that the penalty elements analysed for did not present a concern.

16.2.5 Tailings characterisation testwork

Knight Piésold performed tailings characterisation to assess the acid generation and neutralisation potential of flotation tailings from the proposed flotation circuit. The results were used to formulate tailings disposal strategies.

The dry climate reduces the environmental impact of the tailings. Percolation of regulated contaminants to the surrounding environment will be minimised, as the absence of surface and near-surface water removes the potential to degrade surface and groundwater resources[7]. Tailings disposal therefore focuses on preventing inhalation or dermal contact with airborne dust from acidic waste and evaporite crusts.

Treatment of the dominant sulphide mineralisation types identified potentially acid generating (PAG) and net acid neutralising tailings. On the basis of testwork results, the tailings produced during bulk flotation (roughers/scavengers) are expected to be net acid neutralising. Cleaner scavenger tailings or combination tailings, specifically from the Primary Sulphide ore zones, are potentially acid generating. Separate disposal of cleaner scavenger tails is required to reduce the footprint of the tailings storage facility (TSF) that needs to be lined and covered upon closure. Therefore, the cleaner scavenger tails will be separately disposed, (but within the same TSF). This part of the TSF will be lined to prevent the migration of potentially acidic solutions, downward or to neighbouring properties (outside of TA1). The cleaner scavenger part of the TSF will be covered with non-PAG waste rock upon closure.

[7] Groundwater has been recorded 450 m below surface.

17. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 INTRODUCTION

Marcobre commissioned Snowden to prepare an updated resource estimate for the Mina Justa Prospect based on the latest validated database of information collected from exploration and infill drilling conducted on the deposits. The October 2008 mineral resource estimate presented in this report forms the basis for the DFS conducted on the Mina Justa Prospect by GRD Minproc.

The October 2008 resource estimate was generated using a comprehensive database that includes drilling data from the pre-2005 Rio Tinto exploration programme and the 2005, 2006, 2007 and 2008 Marcobre drilling programmes (up to a cut-off date of 23 May 2008). Previous Mina Justa resource estimates were reported in Technical Reports by Snowden in October 2005, February 2007 and June 2008, and by AMEC in November 2004.

Mineral resources presented in this section were prepared by Dr. Warwick S. Board, P.Geo., Senior Consultant with Snowden. Dr. Board is independent of Marcobre.

17.2 DATABASE AND BLOCK MODEL

17.2.1 Database

Marcobre supplied a database comprising survey, assay, density, geological, mineralogical and structural data, together with QAQC data (standard, blank and duplicate assays). Snowden conducted a series of validation tests and made recommendations to Marcobre to fix several minor errors and inconsistencies, following which the database was adopted for the October 2008 resource model.

The database included:

- 1070 drill hole collar records
- 133 170 assays (excluding QAQC results). Results include CuT, Cu_CN, Cu_SS, Cu_R, Ag, Au, and H_2SO4_Kg/t, although there was insufficient information available for H_2SO4 at the 23 May 2008 database cut-off date to generate a meaningful estimate for this variable
- 3768 density measurements

The database was uploaded into Datamine Studio 3 mining software for resource modelling. Basic validation checks were undertaken, less than detection values converted to half the detection limit, and zero values set as missing data.

The data was desurveyed, and a final resource database produced, excluding 17 holes without surveys and those drilled for metallurgical purposes. The final database contained 132 165 records.

17.2.2 Wireframes and domain coding

Marcobre supplied the following wireframes to constrain the resource model

- 0.2% Cu grade shells for Mina Justa and Magnetite Manto
- wireframes for the ocoite dykes at Mina Justa and Magnetite Manto

- wireframes for the various lithological units at Mina Justa and Magnetite Manto
- topography, base-of-oxide and top-of-sulphide surfaces
- a bornite-chalcocite solid for Mina Justa

Drill hole data was coded by these wireframes as listed in Table 17.1, and the coding verified by visual comparison with the wireframe solids.

Table 17.1
Domain Code Details

Domain	Code	Details
Waste		Drill hole intersections outside of the 0.2% grade shell and ocoite dyke model
Waste/dyke	0	Drill hole intersections within the ocoite dyke model
Oxide	10	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit above the base-of-oxide surface (>60% Cu_SS/Cu)
Transition	151	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit between the oxide and sulphide wireframes, not including the Cu40 area
Transition	152	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit between the oxide and sulphide wireframes in the deeper, steeply dipping parts of the Cu40 area
Transition	153	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit between the oxide and sulphide wireframes in the upper, flat-lying Cu40 area
Chalcopyrite sulphide	201	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit, within the sulphide wireframe (<30% Cu_SS/Cu), not including the Cu40 area
Chalcopyrite sulphide	202	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit, within the sulphide wireframe and the Cu40 area (represents the deeper sulphide extension of the 152 transition domain in the Cu40 area)
Bornite-chalcocite sulphide	211	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit within the bornite-chalcocite wireframe solid, high grade western and central zones
Bornite-chalcocite sulphide	212	Drill hole intersections within the 0.2% Cu grade shell for the Mina Justa copper deposit within the bornite-chalcocite wireframe solid, low grade eastern edge
Magnetite Manto	100	Drill hole intersections within the 0.2% Cu grade shell for the Magnetite Manto copper deposit

17.2.3 Mineralogy data

Mineralogical data (57 545 records) were imported as alphanumeric codes, validated and desurveyed.

17.2.4 Lithology and density data

Lithological data were imported for wireframe validation. Some Rio Tinto lithological logging was included, but, for the most part, relogging of Rio Tinto drill holes was undertaken by Marcobre, and given priority. This data, too, was desurveyed, and compared to the lithological wireframes provided by Marcobre, which were found to be reasonable. However, Snowden noted the presence of a number of ocoite intersections which were not captured by the dyke wireframes, particularly near the Cu40 area. The lithology codes are included as Table 17.2.

Table 17.2
Lithology Codes

Block Model code field	Code	Details
LROCK	0	Air (above topography)
	10	Volcanic sediment/default lithology
	20	Arkose
	30	Crystal tuff
	40	Andesite
	50	Magnetite manto lithology (not deposit)
	90	Ocoite dykes
WROCK	0	Air
	1	Oxide
	2	Transition
	3	Fresh

Density data were imported and desurveyed. This file was coded according to the base-of-oxide and top-of-sulphide surfaces, the portion between the two being classified as Transition mineralisation. The sulphide portion was further sub-divided into Bn-Cc or Cpy types. Semi-massive to massive magnetite zones were separately identified by the lithology code.

Snowden considered that the density data was insufficient for interpolation into the resource model. Consequently, average values were calculated for each combination of lithology and weathering (Table 17.3). The density data was particularly inadequate for the Bn-Cc domain; this data was combined with that from the Cpy domain and the overall average applied to the entire sulphide domain.

Table 17.3
Details of block model density assignment by deposit

Details	Deposit	Density (t/m^3)
Air	Both	0.00
Default oxide	Mina Justa	2.81
	Magnetite Manto	2.82
Default transition	Mina Justa	3.08
Default sulphide	Mina Justa	3.12
Oxidised dyke	Both	2.81
Transition dyke	Both	2.88
Fresh dyke	Both	2.90
Oxidised magnetite manto lithology	Magnetite Manto	3.84
	Mina Justa	3.41
Magnetite manto lithology in transition zone	Mina Justa	3.70
Magnetite manto lithology in sulphide zone	Mina Justa	3.53
Oxidised andesite	Magnetite Manto	2.82
	Mina Justa	2.73
Andesite in transition zone	Mina Justa	3.04
Andesite in sulphide zone	Mina Justa	3.12
Oxidised crystal tuff	Magnetite Manto	2.82
	Mina Justa	2.81
Crystal tuff in transition zone	Mina Justa	3.04
Crystal tuff in sulphide zone	Mina Justa	3.12
Oxidised arkose	Magnetite Manto	2.82
	Mina Justa	2.80
Arkose in transition zone	Mina Justa	3.04
Arkose in sulphide zone	Mina Justa	3.12
Oxidised volcanosedimentary rocks	Mina Justa	2.81
Volcanosedimentary rocks in transition zone	Mina Justa	3.08
Volcanosedimentary rocks in sulphide zone	Mina Justa	3.12

17.2.5 Block model generation

17.2.5.1 Prototype

The block size was set at 25x25x5 m, with sub-celling to 5x5x1 m to honour wireframe boundaries. The block size was selected to take account of drill hole spacing and the scale of mining being considered.

17.2.5.2 Model compilation

The block model was generated by sequential addition involving three steps, namely:

- Preparation of the mineralisation model with preliminary coding.
- Creation of lithology/weathering model with density coding.
- Combination of mineralisation and lithology models.

Mineralisation - Grade shell model

The Mina Justa copper grade shell model was interpreted based on a 0.2% Cu cut-off using 25 m east-west spaced and 50 m north-south spaced cross-sections. Additional interpretation, when necessary, was done within a three-dimensional environment. Individual drill hole intercepts were snapped to the wireframe during generation, to honour the sample data. The interpretation was completed iteratively using the assistance of Marcobre's Dr. Klaus Meder and John Kapusta.

It was noted that during the construction of the 0.2% Cu grade shell, drill hole intersections containing barren, post-mineralisation (ocoite) dykes were ignored. Snowden later superimposed the ocoite dykes onto the resource model in Datamine, to account for their effects. Small drill hole intersections (1 cm to 5 cm) were omitted from their interpretation of the grade shell to limit complexity.

The Magnetite Manto copper grade shell model was interpreted based on a 0.2% Cu cut-off using 25 m spaced north-south cross-sections. The same methodologies and parameters used in creating the Mina Justa 0.2% Cu grade shell were applied.

Lithological Model

A total of 1051 drill holes (107 Rio Tinto drill holes and 944 Marcobre drill holes) were available for use in the lithological modelling process, as at 8 August 2008. Lithological data recorded during core logging were electronically stored as LithoType and LithoSubType fields. Atticus and Associates used these fields in the generation of lithological models for the Mina Justa Prospect.

The principal mappable lithological units in the vicinity of the Mina Justa Prospect are andesite, crystal tuff, arkose, volcanogenic sediment, magnetite manto, and ocoite dyke. Separate lithological models were generated for the volcanic, sedimentary, intrusive (dyke) and metasomatic (magnetite manto) lithological units. Spatial continuity of lithological units was interpreted and modelled using north-south sections spaced at 50 m intervals. The presence of intrusive and metasomatic lithological units was ignored during the initial modelling of the volcanic and sedimentary units. Modelling of the ocoite dyke and magnetite manto units was conducted directly in three dimensions rather than on a sectional basis, due to their complex geometries. All surfaces and solids were generated using the Leapfrog modelling software and were then exported in Datamine Studio wireframe format for validation and subsequent use in resource modelling.

Volcanogenic sediments form the base of the sequence in the vicinity of the Mina Justa deposit and are overlain sequentially by crystal tuff, a lower andesite unit, an arkose unit, a further andesite unit, and finally an upper arkose unit. The stratigraphic younging direction has not yet been determined for this sequence.

Lithological modelling in the vicinity of the Magnetite Manto deposit is slightly different to that observed in the vicinity of the Mina Justa deposit in that the lower arkose unit, which is intercalated with a thin crystal tuff unit, is overlain sequentially by an andesite unit, followed by an upper arkose unit. Again, the stratigraphic younging direction has not yet been determined.

Atticus and Associates noted the absence of any correlation between volcanic and sedimentary rock units and intensity of copper mineralisation. Although the magnetite manto units display elevated levels of copper mineralisation there is no significant control on copper mineralisation as a function of pre-ocoite dyke lithology (elevated grades are also hosted by volcanic and sedimentary units). This may be a reflection of the timing of copper mineralisation relative to magnetite mineralisation. Ocoite dyke units are barren and post-date mineralisation.

Mineralogical model

Copper mineralisation at the Mina Justa Prospect was analysed by sequential copper techniques. Total copper (CuT) for a sample is the sum of copper determined in the sequential steps, i.e. CuT = CuSS + CuCN + CuR. Samples were also analysed for total copper and compared to the sum of the sequential copper analyses to ensure this relationship was maintained.

The mineralogical model, created by Atticus and Associates, was divided into Oxide, Transition, Sulphide, and Bornite-Chalcocite domains. The Oxide domain was delineated by a minimum CuSS/CuT ratio of 60%. Mineralogical information from drill logs was used to supplement the interpretation when sparse sequential copper data was encountered. A maximum CuSS/CuT ratio of 30% defined the Sulphide domain. The Transition zone was defined as the zone between the Oxide and Sulphide domains. Definition of the Bornite-Chalcocite domain used a CuCN/CuT ratio of 50%. Atticus and Associates also experimented with a 90% CuCN/CuT ratio to test the sensitivity of the ratio on the volume of the resulting domain. Minimal effects on the volume were found, thus the 50% CuCN/CuT ratio was used.

For the purposes of resource estimation, the Transition zone three-dimensional solid was not used by Snowden. Instead, the Transition domain was defined in Datamine Studio software by using overlapping relationships, where the block model was given a default “transition zone” code (CODE=152), and other domains were superimposed on top of it.

In Snowden’s opinion, the lithological, mineralogical, and grade shell models generated for the purpose of resource estimation take into account the lithological, mineralogical, and structural characteristics observed at the Mina Justa and Magnetite Manto copper deposits.

17.2.6 Grade modelling

17.2.6.1 Sample composites

Snowden selected a composite length of 2 m after analysing the LENGTH field data. The domain-coded raw drill hole data were composited to 2 m, retaining all residual composites regardless of length, with compositing controlled by domain. The resulting drill hole file was validated by comparison against

the input domain-coded drill hole file, as well as by 'metal content' (i.e. comparing the sum of length x grade pre- and post-compositing).

17.2.6.2 Top cuts

Statistical analysis conducted by Snowden indicated the presence of extreme outlier values in the raw and composited data for the variables of interest. Extreme values will influence the variogram and can result in local grade smearing during kriging. Snowden conducted a top cut (grade capping) analysis on the composited, domain-coded drill hole data for each of the variables of interest.

Top cuts were applied to the domain-coded and composite data. The resultant top cut data file is the input data file for grade estimation. Top cuts for the sequential copper data were set at the same percentile as those determined from the total copper data, to preserve relationships between these variables.

Details of the top cuts applied to CuT in each domain are presented in Table 17.4. Validation of the top cut application was focussed on total copper as this is the economically most important constituent of the deposit. Top cuts applied to the total copper data were validated globally against the input uncut domain-coded composited drill hole data, as well as in terms of 'metal' loss. Top cuts were also determined for Ag and Au data.

Table 17.4
Top-cuts applied – cut by domain

Domain	Variable	Top-cut	No data (uncut)	No. top cut	Top cut Percentile	Mean (uncut)	Mean (top-cut)
10	Cu	7.0%	18,701	39	99.79%	0.485%	0.478%
151	Cu	No TC	1,258	0	-	0.742%	-
152	Cu	7.00%	771	2	99.74%	0.755%	0.747%
153	Cu	2.25%	556	9	98.38%	0.338%	0.300%
201	Cu	9.4%	2,988	8	99.73%	0.755%	0.751%
202	Cu	8.4%	3,333	7	99.79%	0.745%	0.742%
211	Cu	22.3%	1,913	4	99.79%	2.073%	2.064%
212	Cu	2.29%	405	4	99.01%	0.523%	0.516%
100	Cu	6.4%	2,018	10	99.48%	0.620%	0.616%

Note: Limited Ag and Au data in Oxide and Magnetite Manto domains due to relative absence of silver and gold in these domains, and so top cut analysis not undertaken for this data.

The top cuts applied do not significantly reduce the mean grade, either globally or on a domain basis except for domain 153. Snowden attributes the approximate 12% loss of metal in the latter domain to the low number of samples within this domain, and the fact that this domain has a mixed distribution (i.e. the application of any top cut will have a significant effect in a sparsely populated mixed domain where there are one or more extreme values). Otherwise the expected decrease in 'metal' content is within acceptable limits for all other domains. Snowden considers the top cuts applied to be sufficient to limit local smearing of high grades during kriging, whilst not significantly affecting metal content.

17.2.6.3 Variography

Three-dimensional continuity analyses were conducted on the total copper and sequential copper data for the Oxide and Magnetite Manto domains; and on the copper, silver and gold data for the Transition, Cpy sulphide and Bn-Cc sulphide domains, using Snowden's Supervisor software.

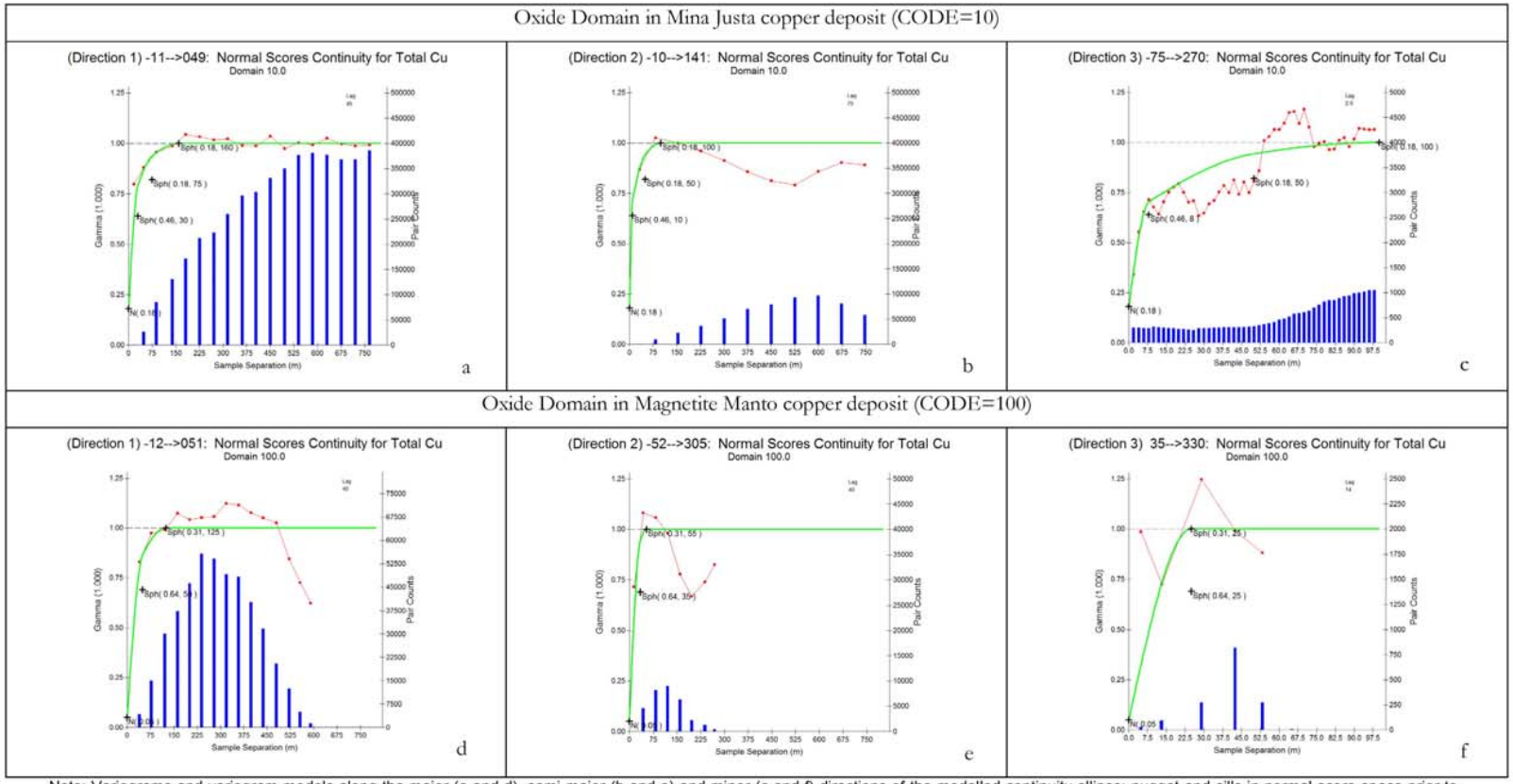
Horizontal, across-strike and dip-plane continuity directions were defined based on variogram fans, geological constraints and discussions with Marcobre's Dr. Klaus Meder. Experimental variograms were generated for the three principle directions of the modelled continuity ellipsoid. Variogram modelling was conducted on normal scores transformed data, with all modelled variances back transformed prior to estimation. Variogram models generated in normal scores space were checked against untransformed and log-transformed experimental variograms, and results indicated that the selected model parameters were robust. All modelled continuity ellipsoids were discussed with Marcobre's Dr. Klaus Meder prior to finalisation to ensure that they were geologically realistic within the context of the Mina Justa Prospect geology.

Variogram directions and model parameters for total copper were superimposed on variograms of sequential copper data for each domain and found to be a close approximation of the continuity of this data. This supports the use of the total copper data variogram parameters for all the sequential copper data in the estimation process so that the relationship CuT = CuSS + CuCN + CuR, as well as the ratios between the different copper components, are retained during grade interpolation.

Results of the continuity analyses and variogram model parameters for total copper are presented in Table 17.5. Example experimental variograms and variogram models are presented for total copper data along the major, semi-major and minor directions for selected domains in Figure 17.1 and Figure 17.2.

Figure 17.1

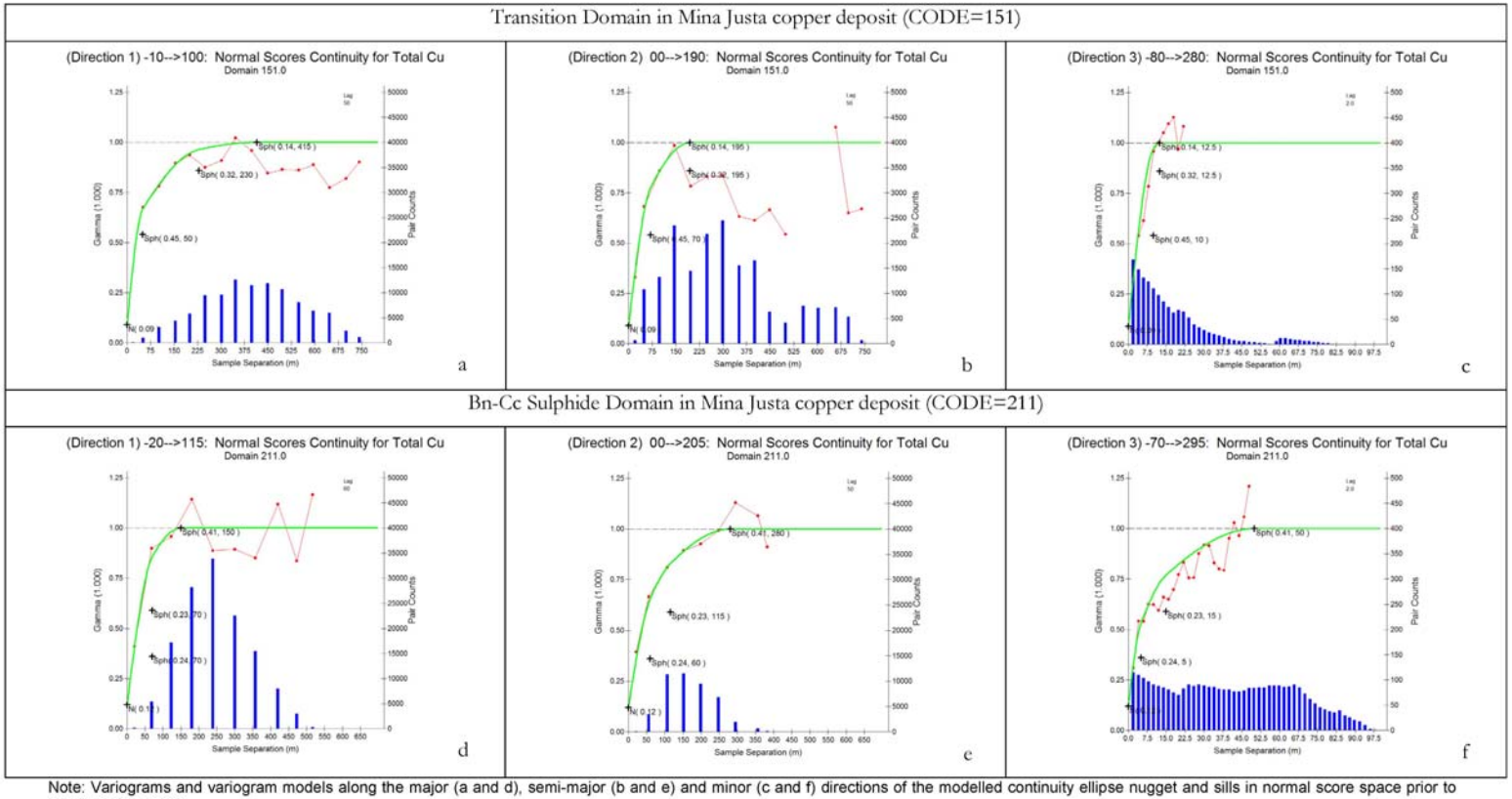
Example variograms and variogram models for Oxide domains in the Mina Justa Prospect



Note: Variograms and variogram models along the major (a and d), semi-major (b and e) and minor (c and f) directions of the modelled continuity ellipse; nugget and sills in normal score space prior to backtransformation.

Figure 17.2

Example variograms and variogram models for selected Transition and Sulphide domains in the Mina Justa Prospect



Note: Variograms and variogram models along the major (a and d), semi-major (b and e) and minor (c and f) directions of the modelled continuity ellipse nugget and sills in normal score space prior to backtransformation.

17.2.6.4 Search volume parameters

An expanding search strategy was used, based on the results of the variography. Ranges for the first search pass were defined on the basis of the variogram and were generally set within the variogram range. Details of the search volume parameters are presented in Table 17.6.

Table 17.5

Variogram parameters – October 2008 resource update

Domain	Direction			Datamine	Nugget	Sill1	Range1 (m)			Sill2	Range2 (m)			Sill3	Range3 (m)		
	Major (D1)	Semi Major (D2)	Minor (D3)	ZXZ Angles			D1	D2	D3		D1	D2	D3		D1	D2	D3
Total copper (Cu)*																	
10	-11°→049°	-10°→141°	-75°→270°	{-90, 165, 40}	0.28	0.49	30	10	8	0.13	75	50	50	0.10	160	100	100
151	-10°→100°	00°→190°	-80°→280°	{-80, 170, 0}	0.09	0.45	50	70	10	0.32	230	195	12.5	0.14	415	195	12.5
152	-30°→105°	00°→195°	-60°→285°	{-75, 150, 0}	0.08	0.64	130	75	9	0.28	175	150	17	-	-	-	-
153	-09°→090°	-04°→180°	-80°→295°	{-65, 170, 25}	0.06	0.62	60	60	5	0.32	135	100	10	-	-	-	-
201	-20°→135°	00°→225°	-70°→315°	{-45, 160, 0}	0.10	0.50	30	95	7	0.40	220	155	32	-	-	-	-
202	-45°→140°	00°→230°	-45°→320°	{-40, 135, 0}	0.07	0.34	65	55	35	0.59	115	175	35	-	-	-	-
211	-20°→115°	00°→205°	-70°→295°	{-65, 160, 0}	0.12	0.24	70	60	5	0.23	70	115	15	0.41	150	280	50
212	-20°→115°	00°→205°	-70°→295°	{-65, 160, 0}	0.12	0.24	70	60	5	0.23	70	115	15	0.41	150	280	50
100	-12°→051°	-52°→305°	35°→330°	{-30, 55, 75}	0.05	0.64	50	35	25	0.31	125	55	25	-	-	-	-

Note: a direction of -11°Æ141° means a plunge of 11 degrees towards an azimuth of 141 degrees, Mina Justa Prospect local grid system; backtransformed nugget and sill values shown.

Table 17.6

Search volume parameters

Domain	Datamine ZXZ Angles	First search pass (SVOL1)		Second search pass (SVOL2)		Third search pass (SVOL3)	
		D1, D2, D3	Samples	Factor	Samples	Factor**	Samples
Total copper (Cu)*							
10	{-90, 165, 40}	160, 100, 50	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	5 x SVOL1	min. 6, max. 15, mpd. 5
151	{-80, 170, 0}	210, 130, 10	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	7 x SVOL1	min. 6, max. 15, mpd. 5
152	{-75, 150, 0}	125, 100, 10	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	6 x SVOL1	min. 6, max. 15, mpd. 5
153	{-65, 170, 25}	135, 100, 10	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	6 x SVOL1	min. 6, max. 15, mpd. 5
201	{-45, 160, 0}	175, 125, 25	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	6 x SVOL1	min. 6, max. 15, mpd. 5
202	{-40, 135, 0}	100, 150, 30	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	6 x SVOL1	min. 6, max. 15, mpd. 5
211	{-65, 160, 0}	105, 200, 30	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	6 x SVOL1	min. 6, max. 15, mpd. 5
212	{-65, 160, 0}	70, 100, 10	min. 12, max. 30, mpd. 5	1.5 x SVOL1	min. 10, max. 30, mpd. 5	5 x SVOL1	min. 6, max. 15, mpd. 5
100	{-30, 55, 75}	75, 35, 15	min. 12, max. 30, mpd. 5	2.0 x SVOL1	min. 10, max. 30, mpd. 5	6 x SVOL1	min. 6, max. 15, mpd. 5

Notes: *Search volume parameters for total copper used for sequential copper components Cu_SS, Cu_CN and Cu_R for the relevant domains; mpd=maximum number of samples per drill hole;
**SVOL3 set large to ensure all blocks in 0.2% Cu grade shells informed with grade; orientation data reference Mina Justa Prospect local coordinate system.

17.2.6.5 Grade estimation techniques

Total copper, sequential copper, silver and gold grades were estimated into the relevant domain-coded blocks in the input block model using ordinary kriging. The domain-coded, composited and top cut drill hole file was used as the input drill hole data file for grade estimation. Mineralogical data were estimated into the input model using the nearest neighbour technique.

The total copper, sequential copper, silver and gold assay data are of suitable quality, quantity and spatial density to support both Indicated and Inferred classifications in the October 2008 Mina Justa Prospect mineral resource.

The subjective nature of the mineralogical logging and the challenges of representatively logging chip samples collected from reverse circulation drilling preclude the classification of the mineralogical estimates in accordance with the CIM (2005) resource confidence categories. These data should be used for indicative purposes only during mine planning.

Marcobre elected not to estimate gangue acid consumption (GAC) data due to the limited quantity of available data (as at 23 May 2008) and poor spatial representivity of the available data for this variable. Snowden recommends that Marcobre obtains a significant quantity of additional gangue acid consumption data before using them for resource estimation.

Block model grade estimates were reviewed in detail to ensure that the estimation process had worked. Sequential copper data were estimated into blocks using the variogram and search volume parameters defined for total copper, to honour ratios between the variables and the relationship CuT=CuSS+CuCN+CuR as far as possible. Following block grade estimation, the sequential copper data were normalised to the total copper data on a block-by-block basis, given that there is confidence in the quality of the total copper data. This normalisation process was conducted to ensure that the ratios between the three sequential copper components were retained. Detailed validation checks were conducted on the normalised sequential copper data to check that this process worked correctly. In general, the pre-normalisation sum of the sequential copper data matched the total copper grade fairly closely and so only minor adjustments were made to the individual sequential copper components during the normalisation process.

17.2.6.6 Model confidence classification

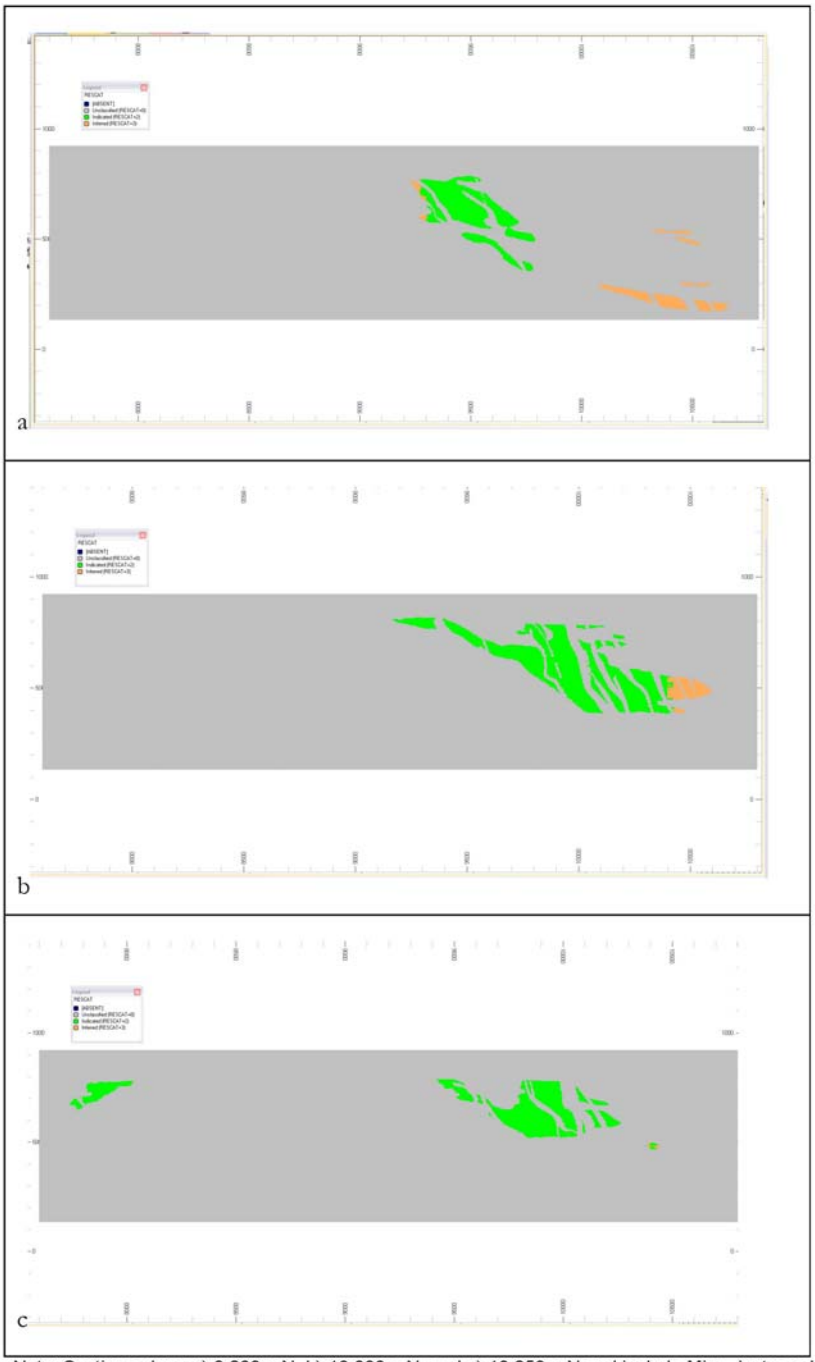
Resource model classification was conducted taking data quality, geological continuity, confidence in the geological model and current level of domaining, grade continuity (from the variography), kriging efficiency and drill hole spacing into account. Based on review of all of these factors, Snowden is of the opinion that Marcobre's total copper, sequential copper, silver and gold data are of sufficient quality and spacing to support Indicated and Inferred classifications as per CIM (2005) definitions for the Mina Justa and Magnetite Manto deposits.

Example views of the October 2008 mineral resource model colour-coded according to resource category are presented in Figure 17.3. The relatively low proportion of Indicated material in the deep sulphide domains and in the Magnetite Manto domain are due to excessive extrapolations of the grade

shell interpretation into areas that are not well supported by the current drilling. Snowden recommends that Marcobre undertakes additional drilling to improve confidence within these areas.

Figure 17.3

W-E section through Mina Justa Prospect, coded by resource category



Note: Sections along a) 9,200 mN, b) 10,000 mN, and c) 10,250 mN and include Mina Justa and Magnetite Manto copper deposits. Green: Indicated; Orange: Inferred.

Although Marcobre's data is, in Snowden's opinion, of sufficient quality and at appropriate drill hole spacing to support a Measured classification in parts of the deposit (e.g. the Cu40 area, the Northern Oxides and part of the Cpy Sulphide domain), there are insufficient geologically and spatially representative density data to raise confidence in the applied density model to the required level at this stage. This is particularly important in a prospect such as Mina Justa in which density varies significantly in plan and with depth as a function of lithology, weathering, alteration, and mineralisation.

Considering that density data directly determines tonnage estimates, Snowden recommends that Marcobre obtains additional density data that is geologically and spatially representative of all of the domains.

The mineralogical data cannot be classified according to the CIM (2005) resource classification definitions. The mineralogical data incorporated into the model are subjective, due to the nature of field logging by teams of field-based geologists. The data are also based on sample surface observations and cannot be considered representative of the entire solid mass of the sample.

17.2.6.7 Model validation

The following block model validation steps were conducted on the Indicated material within the October 2008 resource model:

- Detailed review of the compilation of the input block model to ensure that all blocks were correctly coded.
- Visual inspection of drill hole and block model grade data for each of the variables of interest, colour coded according to grade, to ensure that input data trends are honoured in the resource model.
- Global comparison of model and input drill hole grades for total copper, acid-soluble copper, silver and gold (in relevant domains) to assess global unbiasedness (Table 17.7).
- Comparison of grade trends between the declustered, domain-coded, composited and top cut input drill hole data and the block model on easting, northing and elevation slices to assess areal bias. Snowden conducted this validation step on the total copper, acid-soluble copper, silver and gold variables on a by-domain basis, and noted that the block model data honoured trends in the input drill hole data for each variable of interest. A selection of these plots is presented in Figure 17.4.
- Grade-tonnage reporting checks. Grade-tonnage reports were generated in independent software packages to ensure no errors were made during reporting.

Table 17.7
Model validation – global mean grade comparisons by domain

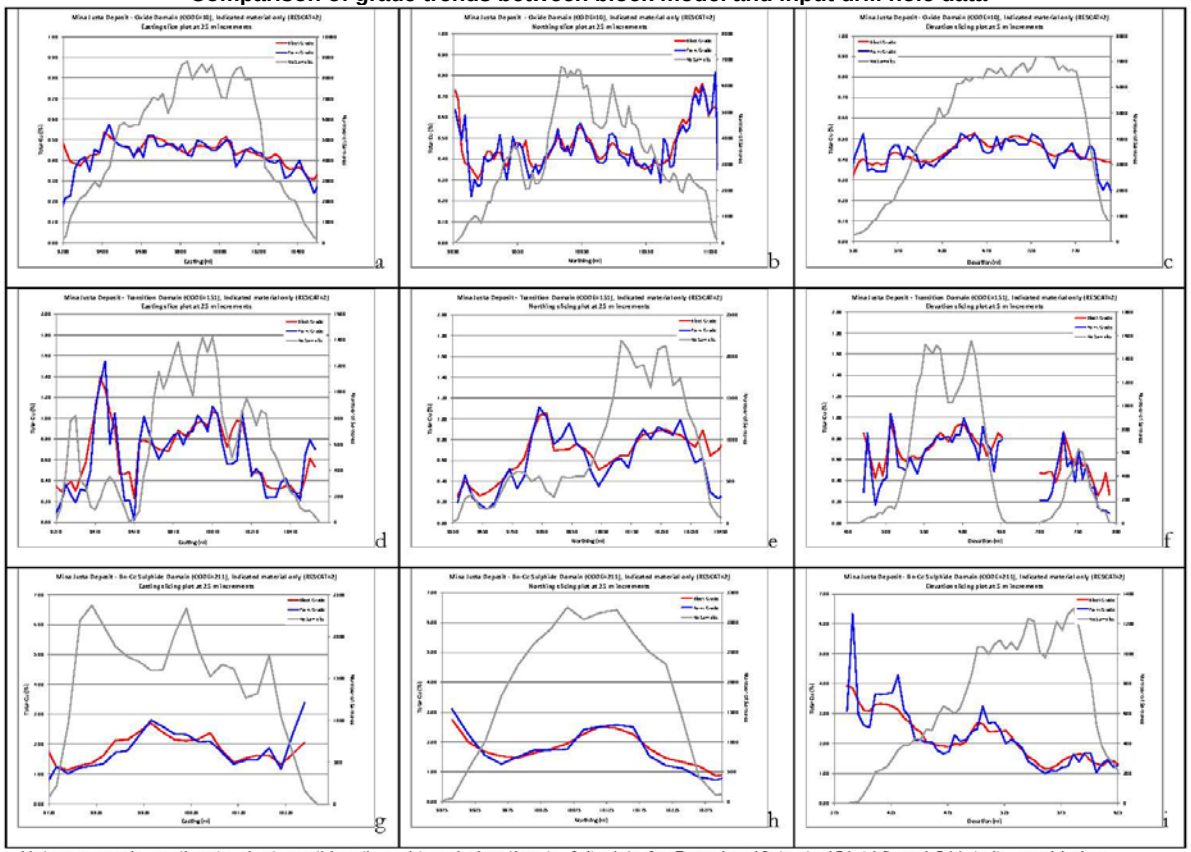
Variable	Domain	Mean Grade	
		Input data*	Block Model
Cu	10	0.45%	0.46%
	151	0.70%	0.73%
	152	0.87%	0.83%
	153	0.33%	0.32%
	201	0.69%	0.73%
	202	0.70%	0.68%
	211	1.90%	1.93%
	212	0.52%	0.53%
	100	0.58%	0.58%
Cu_SS	10	0.36%	0.37%

*Note: Input drill hole data are declustered, domain-coded, composited and top cut. All data from that part of the resource classified as Indicated. Declustering conducted using nearest neighbour interpolation into block model for relevant variables and domains.

Based on the results of the model validation steps outlined above, Snowden considers the October 2008 Mina Justa Prospect resource model to be valid, with the block estimates honouring the input drill hole data. Example local grid west-east cross-sections through the validated block model, filtered such that CODE>0.1, colour coded according to total copper are presented in Figure 17.5, Figure 17.6 and Figure 17.7.

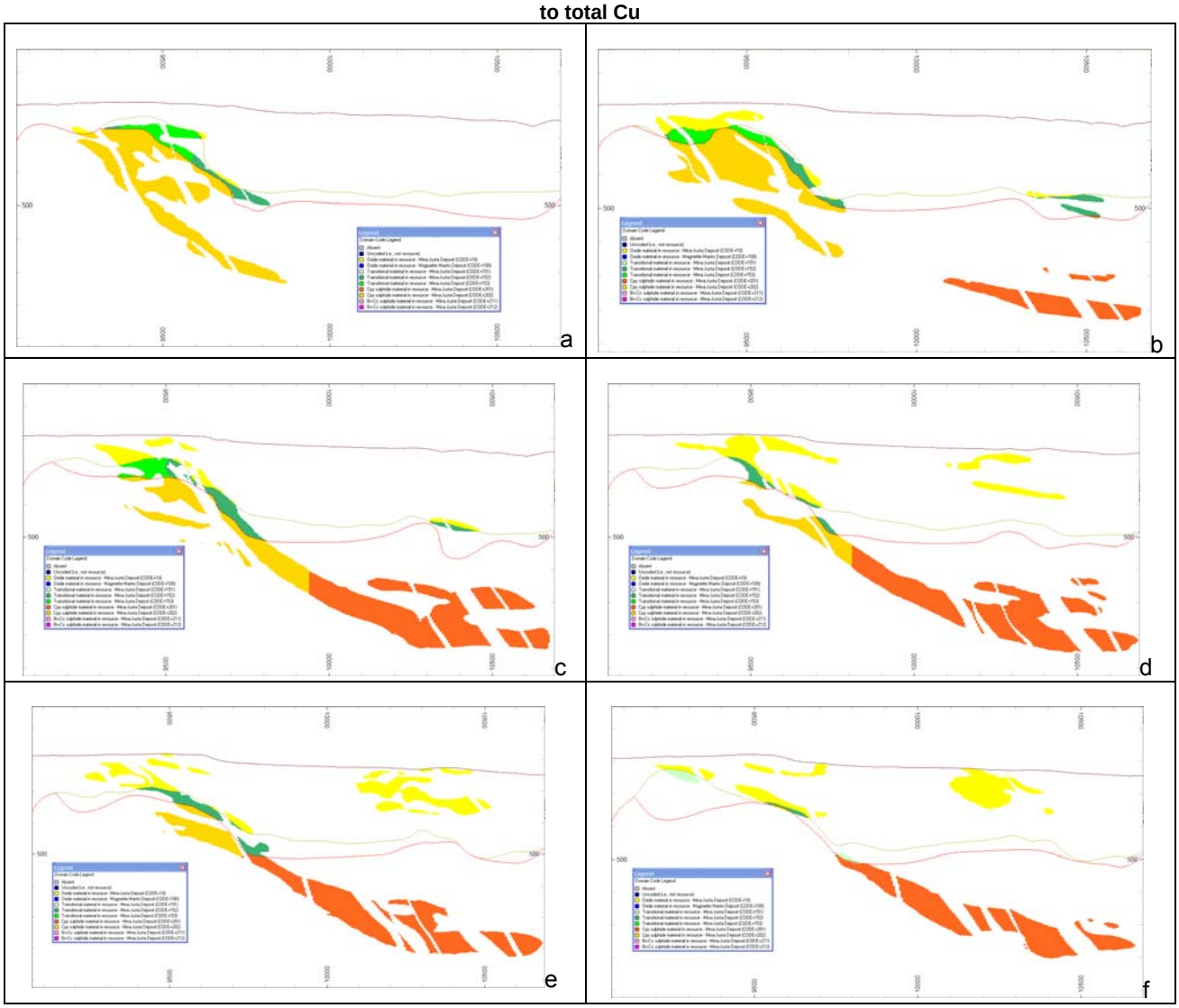
Figure 17.4

Comparison of grade trends between block model and input drill hole data



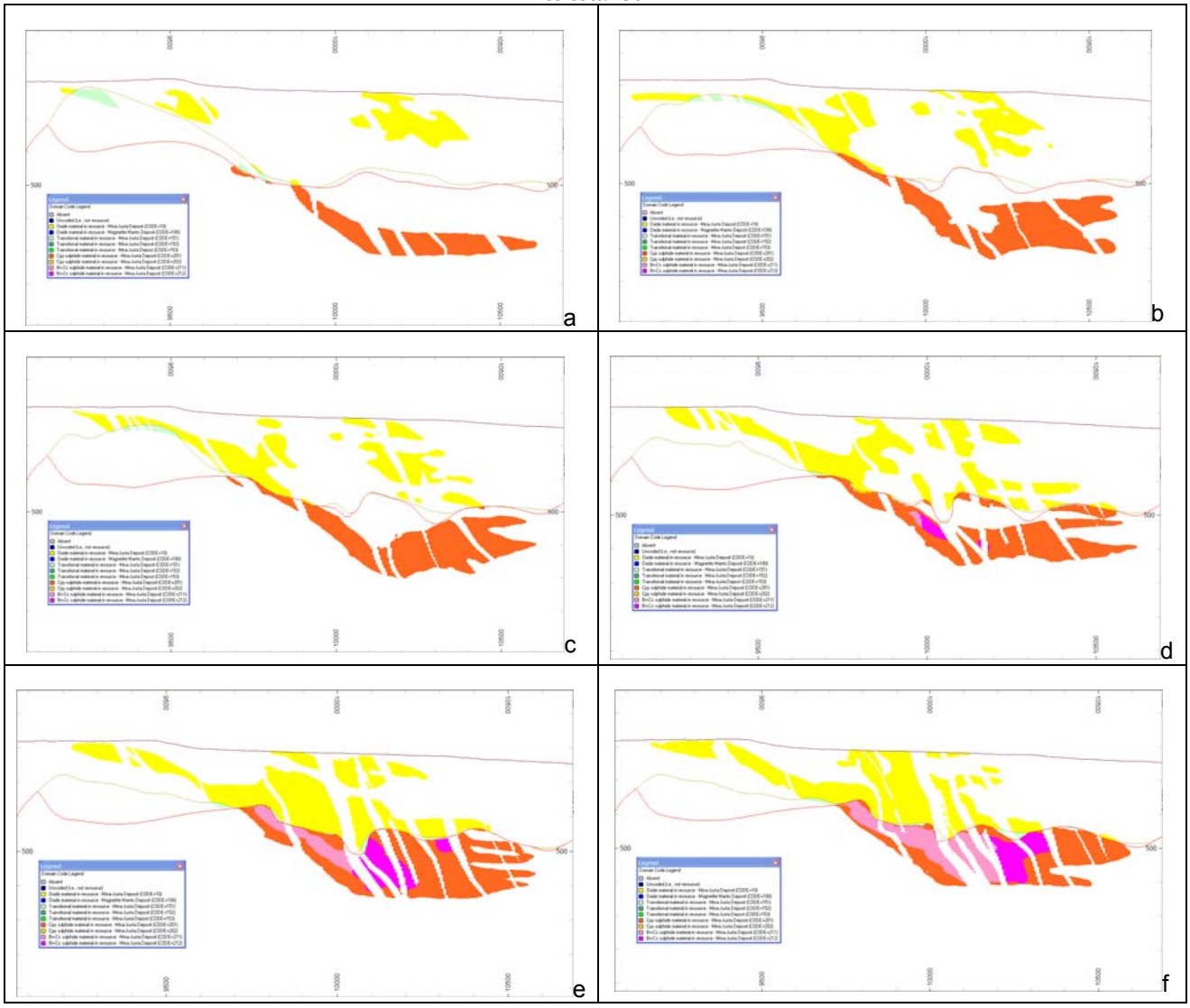
Notes: example easting (a, d, g), northing (b, e, h) and elevation (c, f, i) plots for Domains 10 (a-c), 151 (d-f) and 211 (g-i), considering material classified as Indicated only; local grid coordinate system.

Figure 17.5
Local grid west-east cross-sections through the Mina Justa Prospect resource model, colour coded according to total Cu



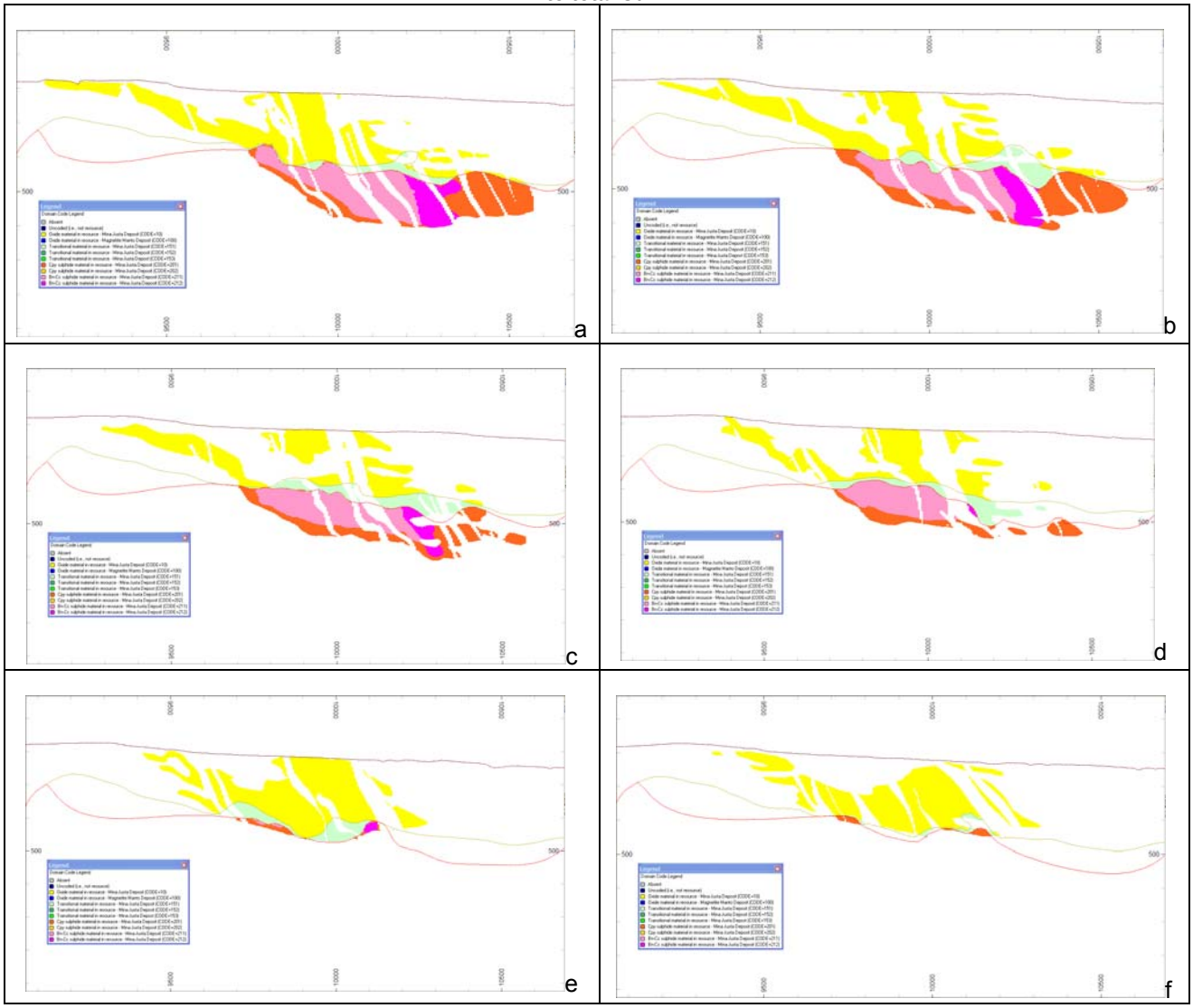
Notes: Cross-sections are filtered such that CODE>0.1. Sections along a) 9,100 mN, b) 9,200 mN, C) 9,300 nN, d) 9,400 mN, e) 9,500 mN; topography (brown), base of oxides (ochre), and top-of-sulphide (red) surfaces are shown for reference. Yellow: Oxide, Mina Justa Deposit (CODE=10); Dark blue: Oxide, Magnetite Manto Deposit (CODE=100); Light blue: Transition, Mina Justa (CODE=151); Dark green: Transition, Mina Justa (CODE=152); Light green: Transition, Mina Justa (CODE=153); Orange: CPY sulphide, Mina Justa (CODE=201); Light orange: CPY sulphide, Mina Justa (CODE=202); Pink: Bn-Cc sulphide, Mina Justa (CODE=211); Magenta: Bn-Cc sulphide, Mina Justa (CODE=212).

Figure 17.6
Local grid west-east cross-sections through the Mina Justa Prospect resource model, colour coded according to total Cu



Notes: Cross-sections are filtered such that CODE>0.1; sections along a) 9,700 mN, b) 9,800 mN, c) 9,850 mN, d) 9,900 mN, e) 9,950 mN, and f) 10,000 mN; topography (brown), base-of-oxide (ochre) and top-of-sulphide (red) surfaces are shown for reference. Yellow: Oxide, Mina Justa Deposit (CODE=10); Dark blue: Oxide, Magnetite Manto Deposit (CODE=100); Light blue: Transition, Mina Justa (CODE=151); Dark green: Transition, Mina Justa (CODE=152); Light green: Transition, Mina Justa (CODE=153); Orange: CPY sulphide, Mina Justa (CODE=201); Light orange: CPY sulphide, Mina Justa (CODE=202); Pink: Bn-Cc sulphide, Mina Justa (CODE=211); Magenta: Bn-Cc sulphide, Mina Justa (CODE=212).

Figure 17.7
Local grid west-east cross-sections through the Mina Justa Prospect resource model, colour coded according to total Cu



Notes: Sections are filtered such that CODE>0.1. Sections along a) 10,050 mN, b) 10,100 mN, c) 10,150 mN, d) 10,200 mN, e) 10,300 mN, and f) 10,400 mN; topography (brown), base-of-oxide (ochre) and top-of-sulphide (red) surfaces are shown for reference. Yellow: Oxide, Mina Justa Deposit (CODE=10); Dark blue: Oxide, Magnetite Manto Deposit (CODE=100); Light blue: Transition, Mina Justa (CODE=151); Dark green: Transition, Mina Justa (CODE=152); Light green: Transition, Mina Justa (CODE=153); Orange: CPY sulphide, Mina Justa (CODE=201); Light orange: CPY sulphide, Mina Justa (CODE=202); Pink: Bn-Cc sulphide, Mina Justa (CODE=211); Magenta: Bn-Cc sulphide, Mina Justa (CODE=212).

17.2.6.8 Differences between the October 2008 and June 2008 resource updates

Differences between the October 2008 and June 2008 resource updates for the Mina Justa Prospect are minor. The October 2008 Mina Justa Prospect mineral resource estimate is based on better domain definition, with an improved and tighter 0.2% total copper grade shell interpretation relative to

the June 2008 mineral resource. This has resulted in an increase in the overall proportion of the resource classified as Indicated. The proportion of Indicated material in the latest resource is artificially lower than the drill hole density would suggest, however, due to relatively extensive areas in the deeper sulphide part of the deposit, interpreted as being within the total copper grade shell, which have little or no drilling data and have been classified as Inferred.

There is no significant difference in the total estimated contained copper between the October 2008 and June 2008 mineral resource estimates for the Mina Justa Prospect. There is, however, slightly more contained copper classified as Indicated at the expense of that classified as Inferred in the latest mineral resource.

There is an increase in the overall contained silver and gold in the October 2008 mineral resource relative to the June 2008 mineral resource. This is due to improved domain definition, tighter grade shell interpretation and the resultant relaxation of stringent top cuts for these variables. There is also an increase in the proportion of contained silver and gold in the Indicated category relative to the Inferred category in the current model compared to the June 2008 model.

Comparisons between the June 2008 and March 2008 Mina Justa Prospect mineral resource estimates are discussed in further detail in Snowden (2008b). Comparisons between the March 2008 and November 2006 Mina Justa Prospect mineral resource estimates, and between the October 2007 and November 2006 Mina Justa Prospect mineral resource estimates, are discussed in further detail in Snowden (2008c) and Snowden (2008a), respectively. The main conclusion of these studies was that differences between the various resource model iterations are a direct reflection of changes to the geological interpretation (notably changes to the interpreted 0.2% CuT grade shell, location of the base-of-oxide and top-of-sulphide surfaces, lithological models and barren ocoite dyke models). Such changes are expected and considered as being a normal part of the refined resource modelling process.

17.2.7 Mineral resource reporting

17.2.7.1 Important information

It is important to note the following when considering the grade and tonnage estimates presented in this Technical Report:

- Mineral resources that are not mineral reserves do not have demonstrated economic viability. The mineral resources presented in this section are inclusive of any ore reserves that are defined in other parts of the Technical Report.
- A Measured mineral resource (CIM, 2005) is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity. This classification requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

- An Indicated mineral resource (CIM, 2005) is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed. An Indicated mineral resource estimate is of sufficient quality to support a preliminary feasibility study.
- An Inferred mineral resource (CIM, 2005) is that part of a mineral resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. Confidence in an Inferred mineral resource estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred mineral resources must be excluded from estimates forming the basis of feasibility or other economic studies.

17.2.7.2 Resource reporting

Global classified resource - October 2008

The October 2008 global classified resource for the Mina Justa Prospect is presented for total Cu in Table 17.8 and for Ag and Au in Table 17.9.

Table 17.8
Mina Justa Prospect global classified resource for total Cu (October 2008)

Cut-off grade (CuT %)	Million Tonnes	CuT (%)	Contained Cu (million lbs)
Indicated			
0.2	411.3	0.67	6,070
0.3	**336.8**	**0.76**	**5,650**
0.4	246.9	0.91	4,960
Inferred			
0.2	77.5	0.72	1,240
0.3	**64.6**	**0.82**	**1,170**
0.4	50.9	0.94	1,060

Notes: Mineral resources that are not reserves do not have demonstrated economic viability. Resource classification categories were in accordance with CIM (2005) definition standards. Data may not tally exactly due to rounding. CuT=total Cu%

Table 17.9 Global classified resource for Ag and Au – October 2008					
Cut-off grade (CuT %)	**Million Tonnes**	**Ag (g/t)**	**Au (ppb)**	**Contained Ag (kOz)**	**Contained Au (Oz)**
Indicated					
0.2	189.3	7.77	51.48	47,290	313,300
0.3	**161.8**	**8.75**	**55.95**	**45,530**	**291,000**
0.4	135.4	9.93	61.61	43,230	268,200
Inferred					
0.2	68.8	4.50	71.13	9,960	157,400
0.3	**58.3**	**5.03**	**79.22**	**9,430**	**148,500**
0.4	48.1	5.63	89.89	8,700	138,900

Notes: Mineral resources that are not reserves do not have demonstrated economic viability. Silver and gold data only reported for the transition and sulphide domains. Resource classification categories were in accordance with CIM (2005) definition standards. Data may not tally exactly due to rounding. CuT=total Cu%.

Global classified resource – sequential copper data (October 2008)

The global classified resource detailing total and sequential copper data as per the October 2008 Mina Justa Prospect resource update is presented in Table 17.10.

Table 17.10
Mina Justa Prospect global classified resource – sequential copper data (October 2008)

Cut-off grade (CuT %)	**Million Tonnes**	**CuT (%)**	**Cu_SS (%)**	**Cu_CN (%)**	**Cu_R (%)**	**Contained Cu (million lbs)**
Indicated						
0.2	411.3	0.67	0.26	0.19	0.22	6,070
0.3	**336.8**	**0.76**	**0.29**	**0.23**	**0.25**	**5,650**
0.4	246.9	0.91	0.31	0.29	0.30	4,960
Inferred						
0.2	77.5	0.72	0.08	0.12	0.53	1,240
0.3	**64.6**	**0.82**	**0.08**	**0.14**	**0.60**	**1,170**
0.4	50.9	0.94	0.08	0.15	0.72	1,060

Notes: Mineral resources that are not reserves do not have demonstrated economic viability. Resource classification categories were in accordance with CIM (2005) definition standards. Data may not tally exactly due to rounding

17.2.7.3 Disclosure

Known issues that materially affect mineral resources

Neither Marcobre nor Snowden know of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues that could materially affect the mineral resource estimates. The

size and quantity of potentially extractable and economically viable mineralised material from the southern parts of the Magnetite Manto and Cu40 mineral resources is, however, dependent upon the relationship between SHP and Marcobre: a favourable relationship would allow the open pits to push back onto SHP's property, thereby allowing additional material to be mined.

Factors that materially affect mineral resources

Neither Marcobre nor Snowden know of any mining, metallurgical, infrastructural, or other relevant factors that could materially affect the mineral resource estimates.

17.3 MINERAL RESERVES

The Mina Justa Mineral Reserve is that portion of the Indicated Resource that is contained within the ultimate pits and has recoverable metal values that allow economic treatment. The Mineral Reserves were determined by Ross Oliver of GRD Minproc, based on mining studies, and metallurgical parameters and costs determined as part of the DFS.

The Mineral Reserve tabulated by classification is identified in Table 17.11

Table 17.11
Mina Justa Probable Mineral Reserve (1), (2), (3)

Classification	Tonnes (Mt)	CuT (%)	CuSS (%)	Ag (ppm)
Vat Feed	114.6	0.56	0.46	-
Concentrator Feed	48.8	1.37	-	14.1
Total	**163.4**	**0.80**	**-**	**-**

Notes:

(1) Reported according to NI 43-101 reporting guidelines. QP is Ross Oliver, an employee of GRD Minproc Ltd.
(2) No Measured resource so no Proved Mineral Reserve
(3) Mineral Reserve cut-off is based on an NSR (Net Smelter Return) calculation and a copper price of $1.65/lb.

18. OTHER RELEVANT DATA AND INFORMATION

18.1 MINING STUDIES

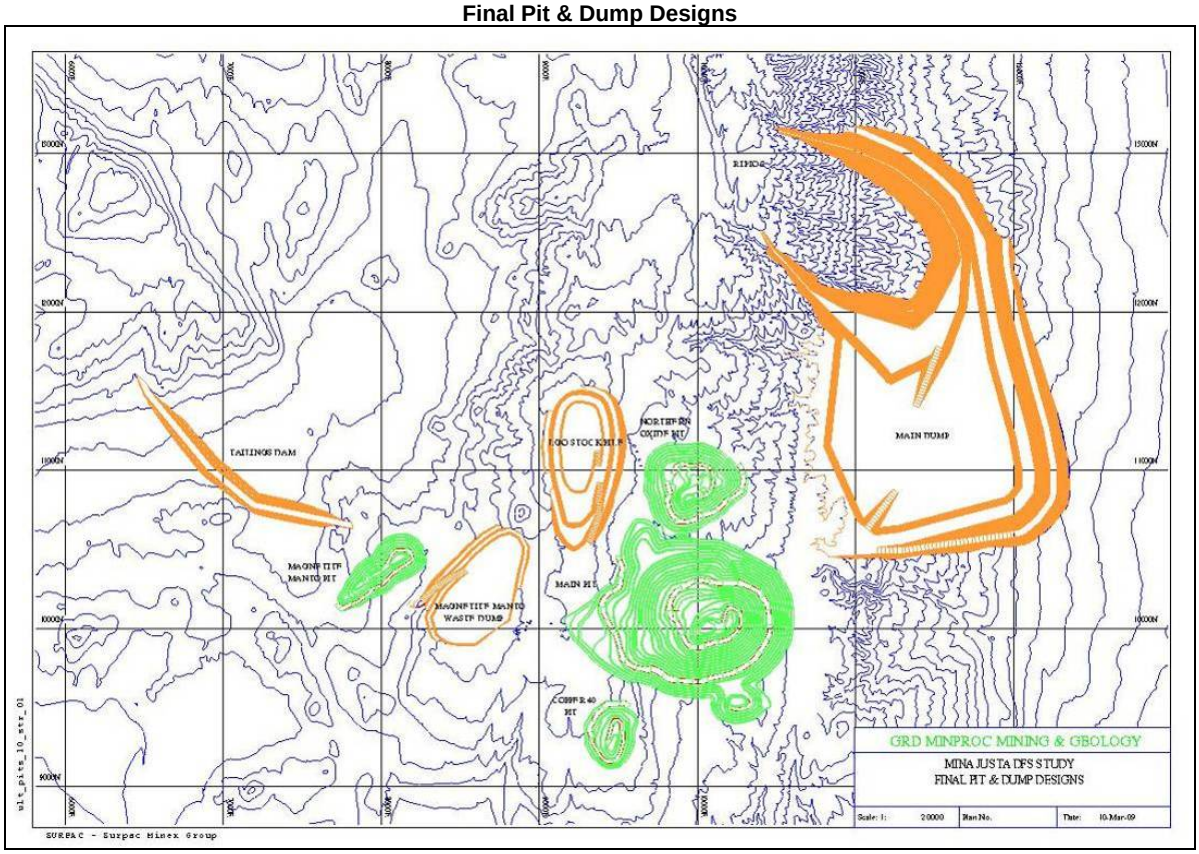
18.1.1 Introduction

The Mina Justa and Magnetite Manto deposits are located at low altitude in an arid area of moderate topography. Rock strengths are low to moderate. There is no groundwater at planned mining depths and insignificant rainfall. These factors suggest that open pit mining should be routine and low cost. The only negative aspects are the relatively low grade of the Oxide mineralisation, and the presence of barren dykes that are pervasive throughout the mineralisation. In order to minimise dilution and mining losses, selective mining on 10 m benches (5 m mining flitches) is specified using excavators configured as backhoes.

The bulk of mining is focussed on the Mina Justa deposit, which includes Oxide (vat leach feed) and deeper Sulphide (concentrator feed) mineralisation. Higher grade Oxide ore is also mined from proximal satellite pits. Oxide ore is hauled to the vat feed crusher and the long term Oxide stockpiles. Sulphide ore is hauled to a separate concentrator crusher. The majority of waste is hauled to the main waste dump with other destinations including the Magnetite Manto waste dump, the tailings dam embankment, the ROM stockpile area and the ripios disposal containment structure.

Figure 18.1 displays the general mining site plan including final pits, waste dumps, stockpiles, ripios dump and tailings dam.

Figure 18.1
Final Pit & Dump Designs



Mining uses 220 t class haul trucks and 20 m^3 backhoes and support equipment. All equipment is diesel powered. Total mining of approximately 60 Mt/a includes vat feed, concentrator feed and waste. The average waste to ore ratio is 2.46:1 (including pre-production stripping). Ripios is hauled using mine trucks. Mining operations continue over 12 years, inclusive of a 9 month pre-production period.

Mine planning activities performed during the course of the DFS included:

- Pit optimisation
- Pit and dump designs
- Mineral reserve estimation
- Mine and process scheduling
- Mine fleet assessment
- Mine operating and capital costs.

18.1.2 Pit optimisation

The resource model prepared by Snowden was based on 25 x 25 x 5 m parent block size with smaller sub-cells (Section 17). Several regularised mining models were prepared to simulate the impact on

dilution and mining losses relative to the in-situ resource model, and a block size of 10 x 10 x 5 m was selected as the basis for mine planning.

Pit optimisation of the mining model (Indicated mineralisation only) was carried out using Whittle Four-X software. Optimisation input parameters were based on the information at that time, including overall slope input (41^{O} to 44^{O}) from Knight Piésold and a copper price of $1.65/lb. Blocks were populated with NSR values prior to optimisation. Revenue was received from both Vat and Concentrator ore streams. The optimisation was constrained to prevent mining on the adjacent Shougang property. A number of different analytical scenarios and sensitivities were produced and shells were selected to form the basis for the ultimate and staged pit designs.

18.1.3 Pit and dump design

Ultimate and staged pit designs were created from the selected optimisation shells, incorporating access ramps. Access was generally by a single ramp of 30 m width at a maximum design grade of 10%. Ramps were narrowed to one way at depth in the pits to minimise associated waste removal. There are four discrete pits, of which two are developed in stages to defer waste stripping and improve ore presentation.

Bench and pit slope design parameters are described in Section 18.2. Bench heights are 20 m, and average inter-ramp slopes range from 45^{O} to 50^{O} depending on the design sector.

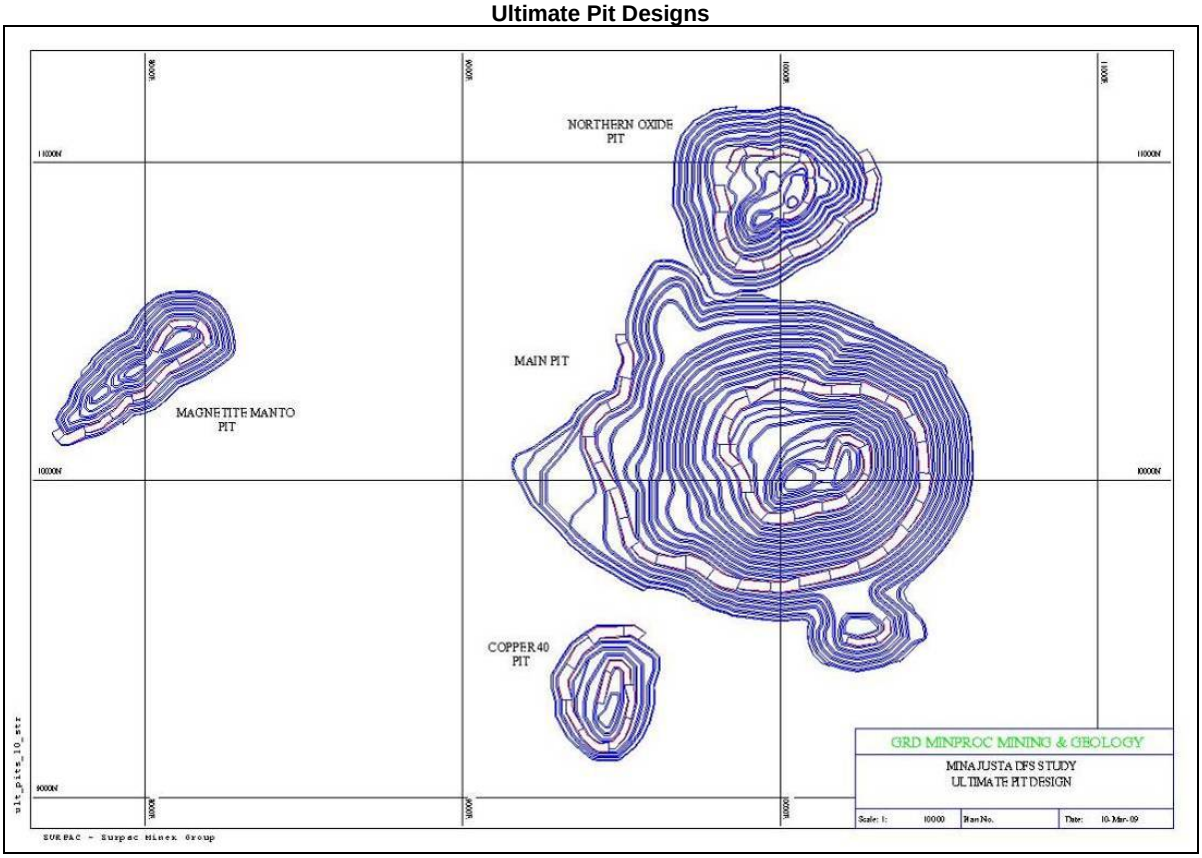
Pit inventories are summarised in Table 18.1

Table 18.1
Pit Inventory

	Unit	Main Pit	Northern Oxide	Copper 40	Magnetite Manto	Total
Vat Ore	Mt	90.4	17.0	2.6	4.6	**114.6**
Concentrator Ore	Mt	48.0	-	0.8	-	**48.8**
Waste Tonnes	Mt	329.5	43.5	10.9	18.5	**402.4**
Total Material	**Mt**	**467.9**	**60.5**	**14.3**	**23.1**	**565.8**
Strip Ratio SR	t:t	2.38	2.56	3.23	3.96	**2.46**

Ultimate pit designs are illustrated in Figure 18.2.

Figure 18.2
Ultimate Pit Designs



The ripios dump has a capacity of approximately 110 Mt to contain the waste product from the vat leaching process. Mining will generate 402.4 Mt of waste rock.

The main waste dump, located northeast of the main pit, also serves as the containment structure for ripios storage. The ripios dump will be surrounded by mine waste rock to maintain adequate long-term physical stability of the material.

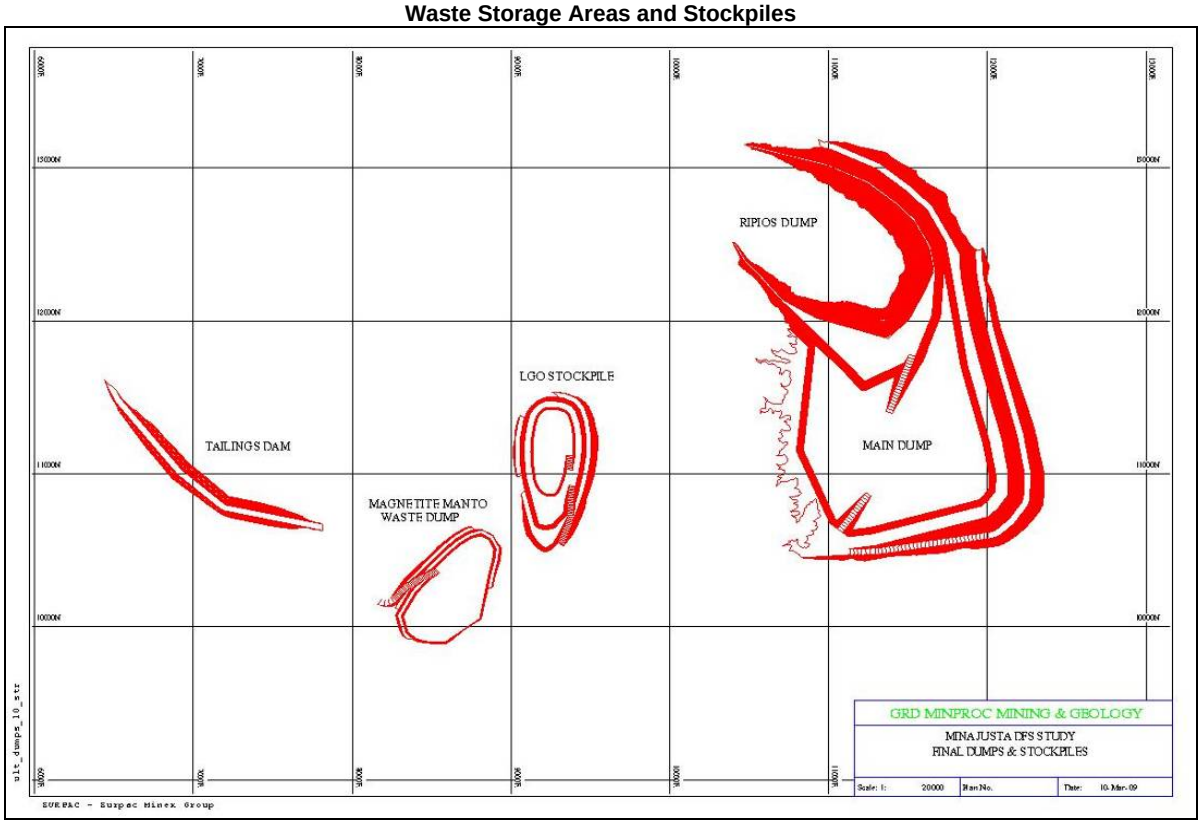
A separate dump is provided for Magnetite Manto pit waste, some of which will be used for tailing dam embankment construction. Some mine waste will be directed to enlarge the ROM pad to provide a suitable configuration for dumping, storage and rehandle of crusher feed. Some opportunistic backfill of waste into the Northern Oxide pit may be feasible at the end of the mine life, but has not been considered in determining mining costs.

The disposal of small quantities of potentially acid generating (PAG) waste rock has been considered conceptually by ensuring that it is surrounded and covered with non-PAG material in the main waste dump.

A large long-term stockpile is also allowed for excess, lower grade vat feed that accumulates during the early years of mining.

Waste storage areas and stockpiles are illustrated in Figure 18.3.

Figure 18.3
Waste Storage Areas and Stockpiles



18.1.4 Mineral reserve

The Mina Justa Project Mineral Reserve is that portion of the Indicated Resource that is contained within the ultimate pits and has recoverable metal values that allow economic treatment. The Mineral Reserve tabulated by classification is identified in Table 18.2.

Table 18.2
Mina Justa Probable Mineral Reserve (1), (2), (3)

Classification	Tonnes (Mt)	CuT (%)	CuSS (%)	Ag (ppm)	Tonnes (Mt)
Vat Feed	114.6	0.56	0.46	-	114.6
Concentrator Feed	48.8	1.37	-	14.1	48.8
Total	**163.4**	**0.80**	**-**	**-**	**163.4**

Notes:

(1) Reported according to NI 43-101 reporting guidelines. QP is Ross Oliver, an employee of GRD Minproc

(2) No Measured resource so no Proved Mineral Reserve

(3) Mineral Reserve cut-off for both vat and concentrator feed is based on an NSR (Net Smelter Return) calculation and a copper price of $1.65/lb.

18.1.5 Mine and process schedules

Bench reporting of reserve information was performed for the pit stages and imported into a purpose-built mine scheduling spreadsheet. A variety of mining rates and vat leach cathode production profiles were investigated. A final mining rate of 60 Mt/a was adopted as a sustainable rate that could bring forward the mining and treatment of higher grade concentrator feed, and also sustain a cathode production rate of around 52 000 t/a of copper.

Mine and process scheduling was carried out on a monthly basis for the pre-strip (Year -1) and first year of production, quarterly for Year 2 through Year 5 and annually thereafter. The quarterly resolution was necessary to ensure ore availability for the deferred concentrator start-up.

Figure 18.4 illustrates the mining production rate by pit stage over the mine life. The majority of mining is associated with developing and sustaining the presentation of the deeper sulphides from the Mina Justa Main Pit stages. Figure 18.5 shows the material types mined.

Figure 18.4
Mining by Pit Stage (Mt)

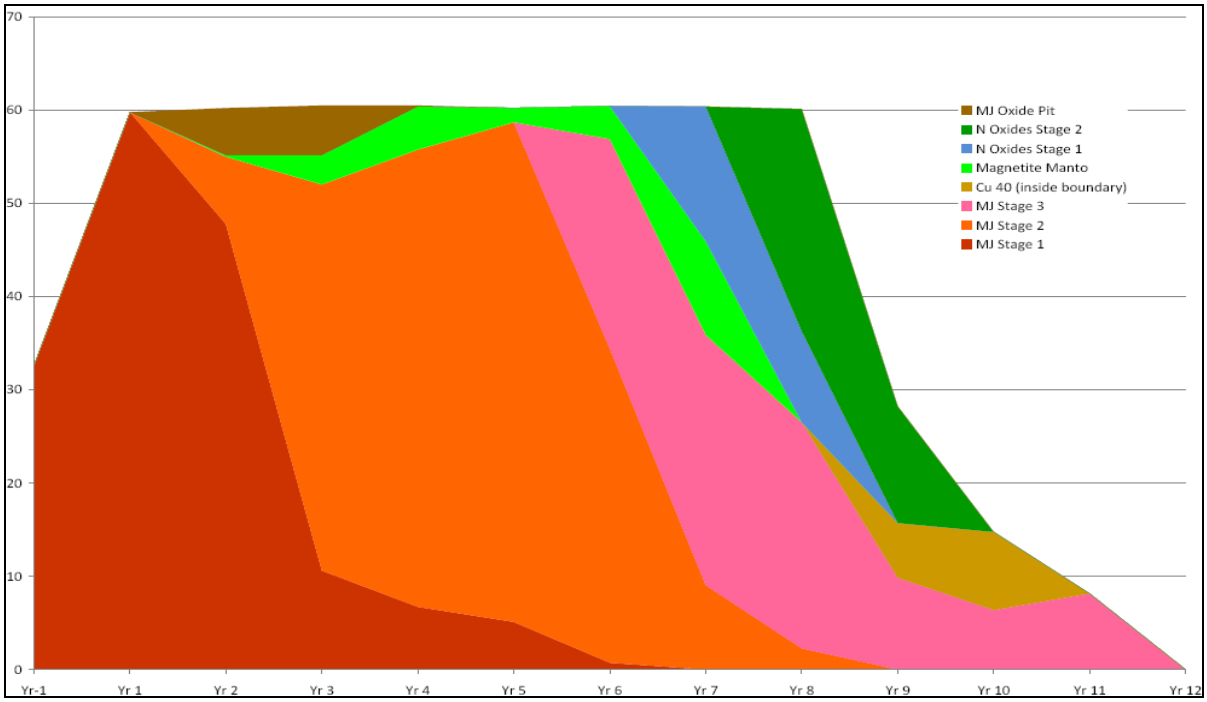


Figure 18.5
Mining by Material Type (Mt)

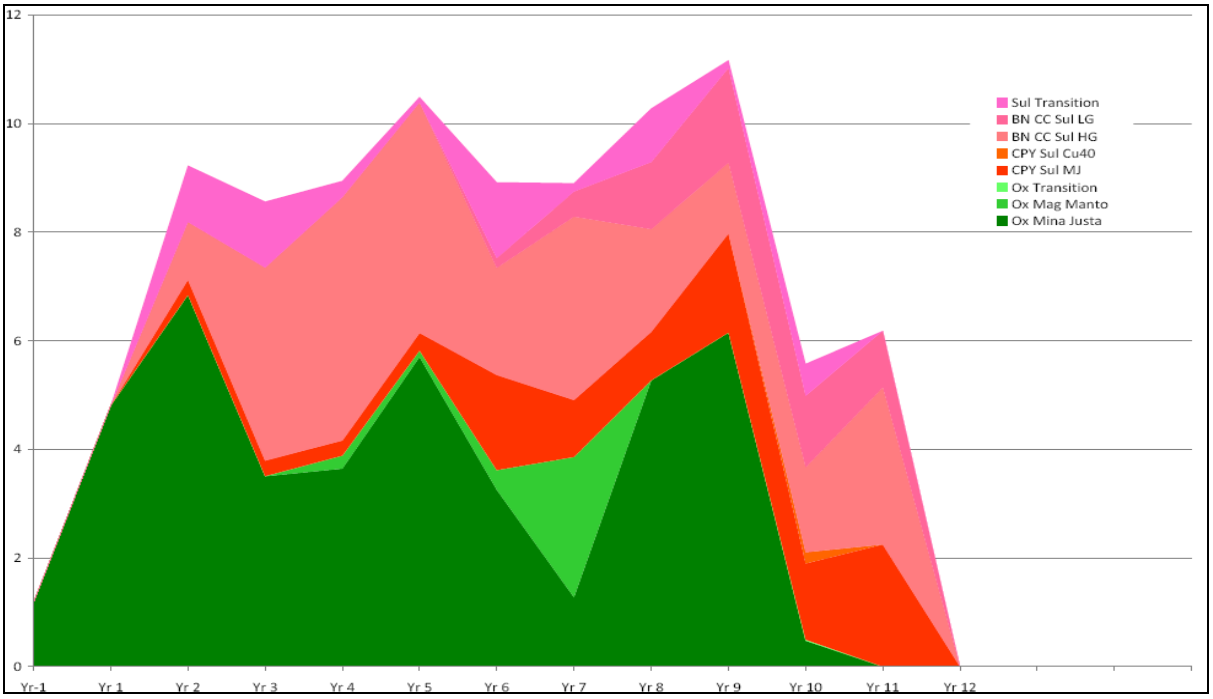


Table 18.3 shows the annual mining and processing production schedule.

Table 18.3
Annual Production Schedule – Mining and Processing

Mining		Total	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12
Vat Ore	('000 t)	**114 602**	4 640	14 791	20 568	8 977	8 866	14 836	12 811	8 158	10 059	8 762	2 134		
Cu	(%)	**0.56%**	0.44%	0.47%	0.54%	0.57%	0.57%	0.58%	0.54%	0.71%	0.60%	0.69%	0.47%	0.00%	
CuSS	(%)	**0.46%**	0.37%	0.40%	0.42%	0.44%	0.46%	0.47%	0.39%	0.58%	0.51%	0.62%	0.36%	0.00%	
Float Ore	('000 t)	**48 794**			2 380	5 055	5 059	4 670	5 301	5 043	5 005	5 014	5 081	6 183	
Cu	(%)	**1.37%**			1.17%	1.43%	1.33%	2.79%	1.19%	1.04%	1.01%	1.07%	1.09%	1.53%	
Au	(g/t)	**0.029**			0.016	0.016	0.020	0.037	0.026	0.022	0.021	0.033	0.044	0.048	
Ag	(g/t)	**14.1**			8.8	12.2	11.8	32.0	12.9	10.7	11.7	10.8	9.6	18.2	
Total Ore	('000 t)	**163 396**	4 640	14 791	22 948	14 032	13 925	19 506	18 112	13 201	15 064	13 776	7 216	6 183	
Waste	('000 t)	**402 363**	27 942	44 972	37 251	46 446	46 550	40 724	42 330	47 157	45 038	14 442	7 544	1 968	
Total Mining	('000 t)	**565 759**	32 582	59 763	60 199	60 478	60 475	60 231	60 442	60 358	60 102	28 219	14 760	8 151	
Strip Ratio		**2.46**	6.02	3.04	1.62	3.31	3.34	2.09	2.34	3.57	2.99	1.05	1.05	0.32	
Closing Stockpiles															
HG Vat Feed	('000 t)		1 180	147	1 651			1 406		825	795	2 223			
MG Vat Feed	('000 t)		1 940	4 101	4 950	2 383									
LG Vat Feed	('000 t)		1 520	5 934	12 150	13 344	12 593	14 024	16 241	11 574	9 663	4 998			
Float Ore	('000 t)				461	515	574	243	544	587	592	606	687	1 871	
Total	('000 t)		4 640	10 183	19 212	16 241	13 167	15 673	16 785	12 986	11 050	7 827	687	1 871	
Processing															
Vat Ore	('000 t)	**114 602**		9 248	12 000	12 001	12 000	12 000	12 000	12 000	12 000	12 000	9 355		
Cu	(%)	**0.56%**		0.56%	0.61%	0.59%	0.53%	0.56%	0.61%	0.58%	0.56%	0.54%	0.48%		
CuSS	(%)	**0.46%**		0.47%	0.47%	0.47%	0.43%	0.47%	0.45%	0.46%	0.47%	0.47%	0.39%		
CuRec	(%)	**0.42%**		0.43%	0.43%	0.43%	0.39%	0.43%	0.42%	0.43%	0.43%	0.43%	0.36%		
Acid	(kg/t)	**40.66**		40.15	40.32	40.39	41.10	40.49	40.67	40.91	40.39	40.48	41.89		
Cu Recovery	(%)	**74.5%**		78.1%	71.5%	72.2%	73.4%	76.2%	68.8%	75.2%	77.4%	79.7%	74.0%		
Cu in Cathode	(t)	**481 596**		40 100	51 999	51 513	46 867	51 601	50 157	52 010	51 997	52 056	33 295		
Float Ore Feed	('000 t)	**48 794**			1 919	5 001	5 000	5 000	5 000	5 000	5 000	5 000	5 000	5 000	1 871
Cu	(%)	**1.37%**			1.23%	1.43%	1.33%	2.57%	1.31%	1.04%	1.01%	1.07%	1.10%	1.73%	0.73%
Au	(g/t)	**0.029**			0.016	0.016	0.020	0.035	0.028	0.022	0.021	0.033	0.044	0.055	0.018
Ag	(g/t)	**14.1**			9.3	12.3	11.9	28.9	14.1	10.7	11.7	10.8	9.7	20.8	8.9
Cu Rec to Con	(%)	**93.0%**			90.1%	91.9%	92.5%	95.3%	92.9%	91.8%	91.5%	92.6%	92.3%	95.0%	87.8%
Au Rec to Con	(%)	**80%**			80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Ag Rec to Con	(%)	**80%**			80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%
Concentrate	(dry t)	**1 643 741**			56 279	164 301	144 397	291 496	176 948	126 276	125 066	150 083	151 018	224 479	33 399
Cu Con Grade	(%)	**37.8%**			37.7%	40.1%	42.7%	42.1%	34.5%	38.0%	37.1%	33.0%	33.7%	36.7%	35.9%
Au Con Grade	(g/t)	**0.70**			0.44	0.39	0.56	0.48	0.63	0.70	0.67	0.88	1.17	0.98	0.81
Ag Con Grade	(g/t)	**335**			255	299	329	397	319	340	375	288	256	370	399
Cu in Con	(t)	**621 373**			21 192	65 867	61 618	122 596	61 053	47 945	46 431	49 528	50 834	82 319	11 989
Au in Con	(oz)	**36 978**			792	2 082	2 608	4 516	3 565	2 850	2 679	4 261	5 700	7 057	867
Ag in Con	(koz)	**17 725**			461	1 579	1 528	3 718	1 817	1 381	1 508	1 391	1 244	2 669	429
Total Copper	**(t)**	**1 102 969**		**40 100**	**73 191**	**117 379**	**108 485**	**174 198**	**111 210**	**99 955**	**98 428**	**101 584**	**84 129**	**82 319**	**11 989**

Note: Metal in concentrate is total contained metal. The financial analysis summary (Table 1.20) shows payable metal in concentrate after smelter deductions.

Vat and concentrator ore processing are shown in Figure 18.6, and long-term ore stockpile inventories are illustrated in Figure 18.7.

Figure 18.6
Vat and Float Ore Processing (Mt)

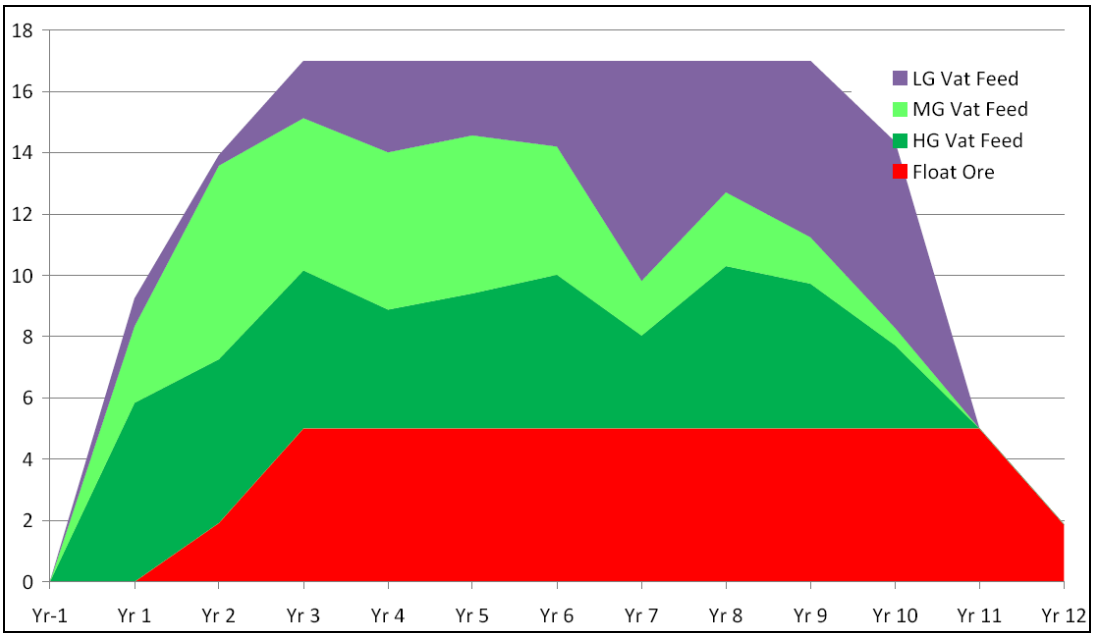
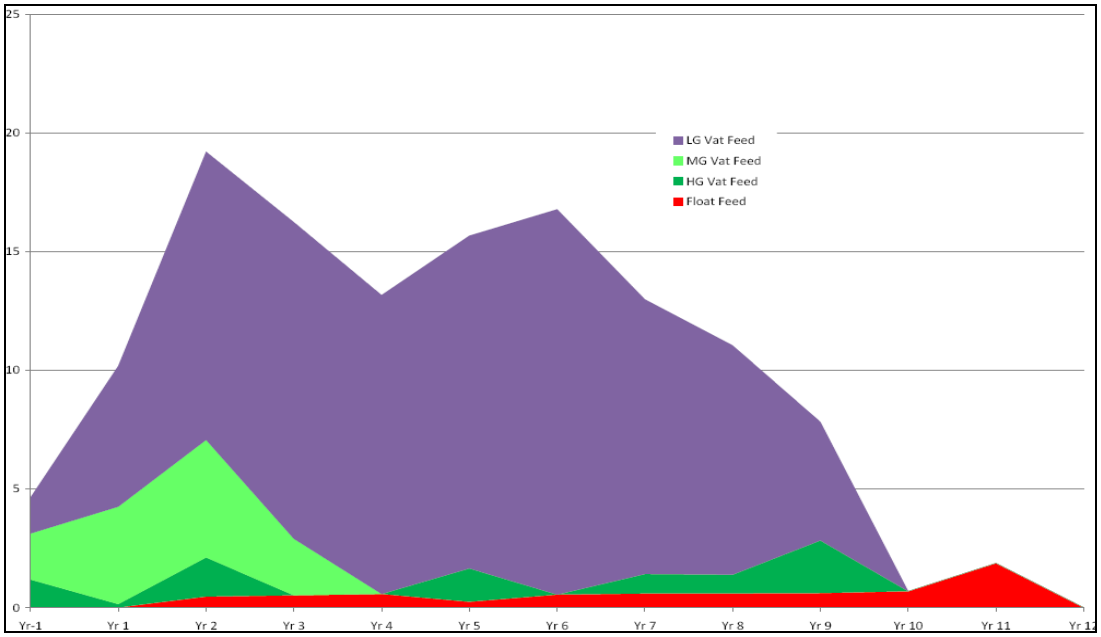


Figure 18.7
Long Term Stockpile Inventories (Mt)



18.1.6 Mine fleet assessment

In order to minimise dilution and mining losses and maximise the mined ore grade, a selective approach to mining mandated the use of the most selective, large scale, digging unit, the backhoe excavator. Relative to a front-end loader (FEL) or face shovel, the backhoe configuration allows tight control of digging boundaries both in plan and elevation. This configuration requires a maximum mining height (flitch) that allows the backhoe to operate productively.

The selected bench height is 10 m to allow two nominal 5 m flitches to be mined by the backhoe. After considering blast heave the actual flitch height dug by the backhoe will average about 6 m. Backhoes in the 20 m^3 class can operate productively with this bench height. A further primary loading unit selection consideration is the match to the primary crusher capacity. The selected excavator has a production rate that is a good match for the peak throughput rates of the vat leach circuit primary crusher. This will minimise production inefficiency that can occur when excavator capacity needs to be compromised to match crushing rates.

A large FEL has been specified to serve the following functions:

- Provide production loading back-up in the pit when a primary excavator is unavailable.
- Provide truck loading for rehandle of long-term vat feed stockpiles.
- Rehandle (by tramming) from short-term operational stockpiles located on the ROM pad.

In order to keep operating costs low, 220 t class haul trucks and support equipment have been selected. Crawler-mounted diesel drills capable of single pass drilling have been selected for productivity and operational flexibility.

A computerised dispatch system to monitor equipment, provide production statistics and provide the information to measure and improve fleet productivity has been specified.

18.1.7 Mine operating cost

The DFS mining concept is that the mine equipment will be owned, operated and maintained by the Marcobre team with support in some key areas by specialist contractors. Peru has many trained operations and maintenance personnel experienced in the class of mining equipment specified for the project.

Specialist support will include:

- Down-the-hole explosive supply
- Vendor provided preventative maintenance services for major equipment, inclusive of labour, site support and consignment stock
- Diesel supply, storage and dispensing services.

Explosive supply will be by a local vendor providing a down-the-hole service. Since the conditions are dry, ANFO has been specified as the sole explosive. After assessment of rock properties, powder factors of 0.20 kg/t in waste and 0.24 kg/t in ore have been adopted.

Key mine operating cost drivers are summarised in Table 18.4.

Table 18.4
Key Mine Operating Cost Assumptions

Item	Value	Unit	Comment
Diesel	0.636	$/litre	includes storage & dispensing
AN Explosive	540	$/t	Dry, 100% ANFO used
Powder Factor Ore	0.24	kg/m^3	
Powder Factor Waste	0.20	kg/m^3	
Truck Tyre Life	5 000	hrs	

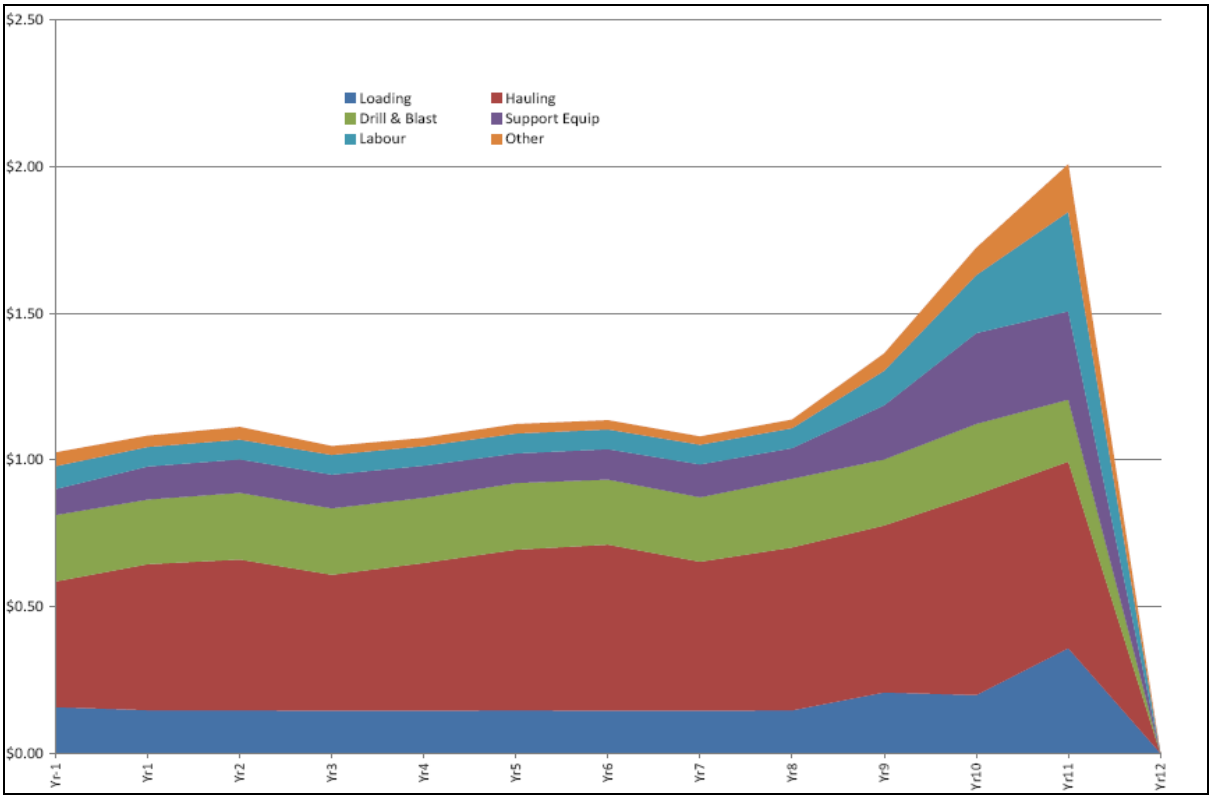
The proposed major equipment fleet make-up is summarised in Table 18.5, together with key equipment assumptions used to build up the operating cost estimate. While specific equipment models have been used to build up the estimate, actual fleet configuration would be subject to a further tendering and evaluation process to establish the most cost-effective mining solution.

Table 18.5
Equipment Fleet and Hourly Costs

Type	Equipment Class	Fleet Units	Operating Hours hr/yr	Operating Costs US$/hr	Purchase Price (US$ M)	Expected Life (hr)
Excavator	20 m^3	3	6 701	$435	$5.55	60 000
Dump Truck	220 t	23	6 701	$219	$3.65	65 000
FEL	20 m^3	1	5 585	$280	$4.67	50 000
Track Dozer	433 kW	4	5 046	$105	$1.22	30 000
Wheel Dozer	372 kW	1	4 840	$89	$1.01	50 000
Grader	221 kW	3	5 606	$64	$0.82	40 000
Water Truck	45 kL	2	3 723	$91	$1.11	60 000
Production Drill	229 mm	4	4 906	$44	$1.61	50 000

Figure 18.8 illustrates the mining operating unit cost with time, showing the major operating cost components. Mining costs are also inclusive of all material handling from stockpiles, and the transport and placement of ripios on the dump.

Figure 18.8
Mine Operating Costs by Time (US$/t)



The average mine operating cost over the life of the mine is $1.14/t mined. Included in the mine operating cost is $22.0 M for ripios disposal and $10.6 M for rehandling ore from the long term stockpile. Exclusive of these amounts, the average mine operating cost is $1.08/t mined.

Costs in the earlier years are lower, but increase in later years as haul distances increase and the total tonnages mined decrease.

18.1.8 Mine capital cost

The total capital cost (including replacement, rebuilds and sustaining capital) of the mining component has been estimated at $139 M as detailed in Table 18.6. This cost does not include capitalisation of mining costs in the construction period.

Table 18.6
Mining - Capital, Sustaining and Replacement Costs (US$)

Mine Area	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Total
Loading	21.78													21.78
Hauling	65.70	14.59	0.45		0.45	3.65	0.45							85.28
Drill & Blast	6.45													6.45
Support	8.52	2.04				3.65		1.23						15.44
Other	7.42	0.60	0.03	0.63		0.09	0.81	0.20		0.21		0.04		10.02
Total:	109.87	17.24	0.48	0.63	0.45	7.39	1.25	1.43	0.00	0.21	0.00	0.04	0.00	138.97

18.2 GEOTECHNICAL STUDIES

18.2.1 Introduction to geotechnical investigations

Knight Piésold Consultores S.A. (Knight Piésold) has undertaken geotechnical investigations at the DFS level for pit slope design, tailings storage, waste rock dumps, ripios dumps, stockpiles and process plant structures.

Geological/geotechnical conditions associated with the Mina Justa Project were defined by a geotechnical study, carried out between October 2006 and November 2007, during which representative samples were collected for laboratory tests. Sources of borrow material and materials to be used during the construction of the Mina Justa Project were also identified and classified.

The geotechnical program and geological and structural mapping results are presented in Figure 18.9 and Figure 18.10.

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Figure 18.9
Geotechnical Plan – West Sector Plant

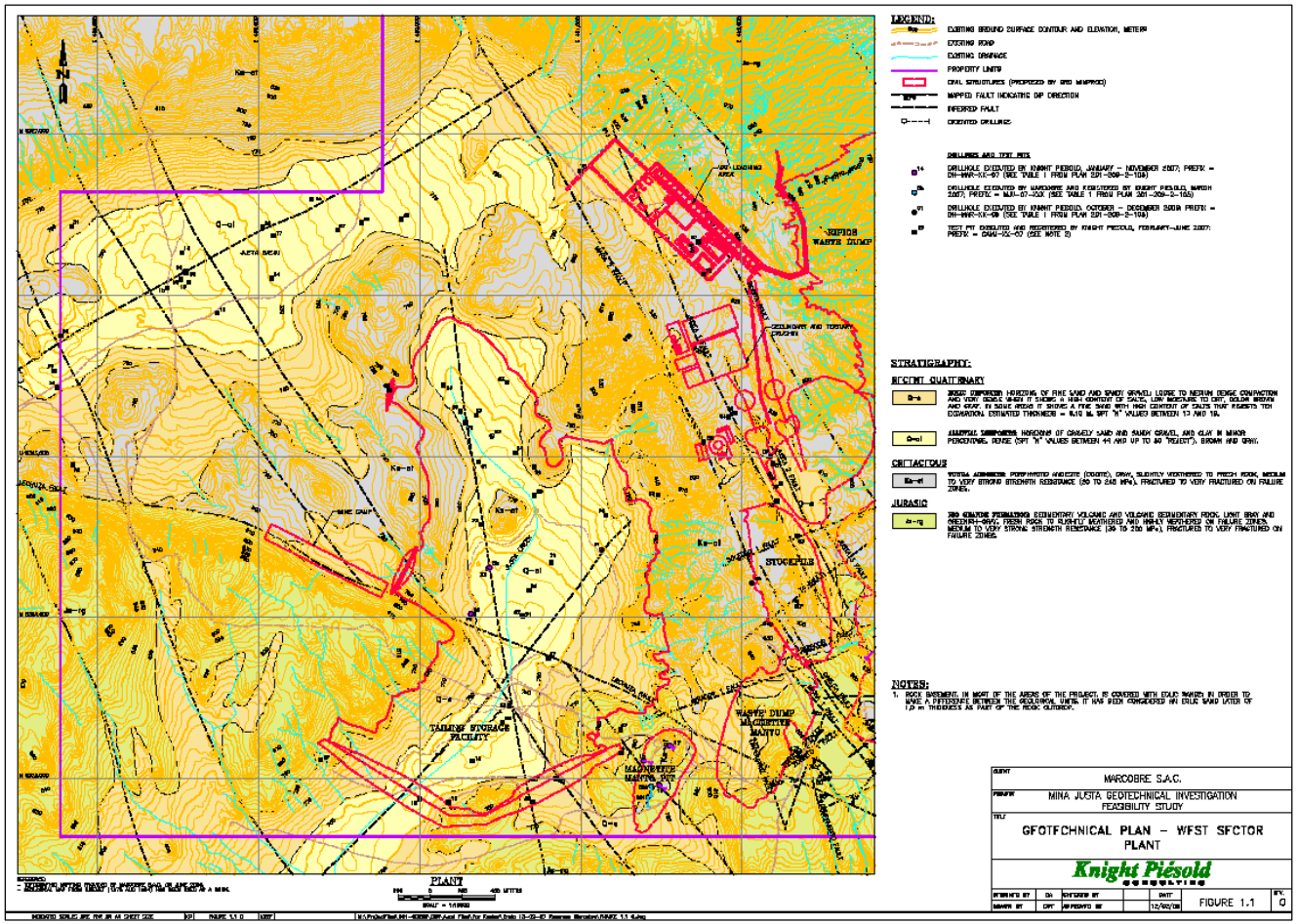
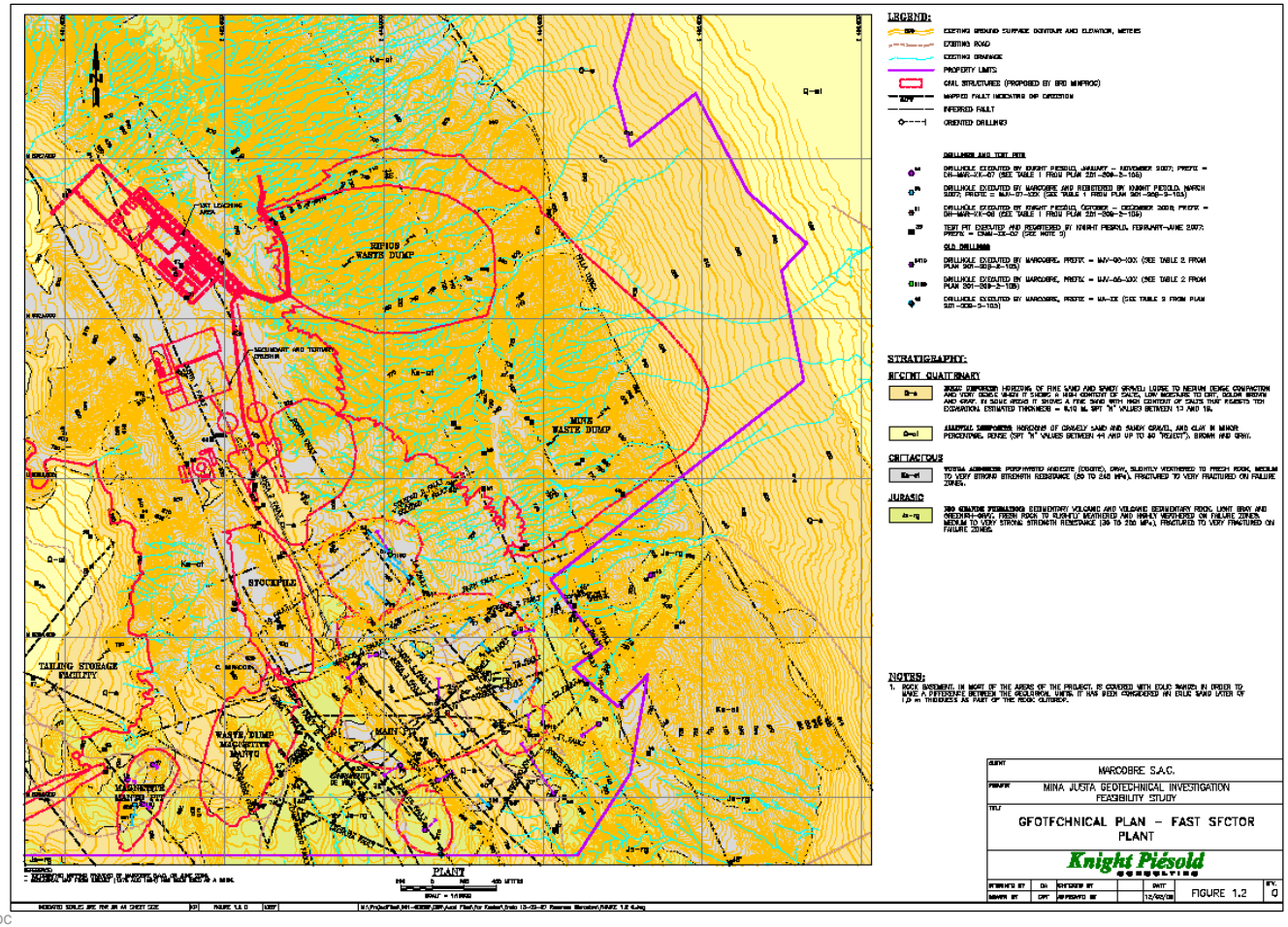


Figure 18.10
Geotechnical Plan – East Sector Plant



The geotechnical investigation program consisted of the following activities:

- Ten oriented drill holes in the main (Mina Justa) pit area, with depths ranging from 260 to 750 m.
- Five oriented drill holes in the Magnetite Manto pit area, with depths ranging between 101 and 173 m.
- Eight drill holes distributed over the areas where the main project facilities are located, with depths ranging between 30 and 120 m.
- Geomechanical mapping and lithological description of material from 23 drill holes completed by Knight Piésold.
- Geomechanical mapping and lithological description of material from 17 drill holes completed by Marcobre in 2005 and 2006.
- Geological/geomechanical surface outcrop mapping and detailed line mapping (7 traverses).
- Field testing: Standard Penetration Testing, permeability tests in soil and rock, Point Load Tests and in-situ density measurements of soils.
- Rock Laboratory Testing: index properties, triaxial tests, UCS and direct shear.
- Tests of representative rock samples to characterize the foundation materials and locate borrow materials.
- Installation of two open pipe piezometers and a vibrating wire piezometer.

18.2.2 Open pit geotechnical design parameters

The final Main (Mina Justa) open pit will measure 2.2 km by 1.3 km with a maximum depth of 430 m, while the final Magnetite Manto pit will be 0.55 km by 0.31 km by 155 m.

Ten oriented drill holes were completed at Mina Justa, and five at Magnetite Manto to assess geotechnical conditions and define discontinuities. In addition, geotechnical logging of rock quality was completed for a further 17 holes drilled by Marcobre, while surface mapping provided additional information.

The overall analysis of rock mass quality, using the RMR system (Bieniawski, 1989) shows that 21% correspond to “poor” rock, 61% to “fair” and 18% to “good”. Little difference was observed between ocoite and volcano-sedimentary host rocks.

The most important structural characteristics observed in the project area and surroundings are the block faulting and fracturing related to major structures. These structures are of critical importance as they directly influence the quality of the rock mass and, thus, affect the development and operation of the open pits.

In the main pit area the result of the analysis of major structures indicates that the structural arrangement is represented by four main fault systems:

- System 1, the most important in the area of the pits, has a NW-SE direction and dips varying between 35º and 70º NE. This system is related to regional faults such as the Tunga and Treinta Libras faults.
- System 2 has an ENE-WSW direction with dips varying between 44º (Zorrito fault) and 66º to the southeast.
- System 3 has a NE-SW direction with dips varying between 45º and 75º to the northwest.
- System 4 has a NW-SE direction, with dips varying between 60º and 80º to the southwest.

Analysis of small scale structures shows the presence of three main systems and two random systems, as follows:

- Main systems:
  - N22ºW trending, dipping 56ºNE
  - N23ºE trending, dipping 54ºSE
  - N83ºW trending, dipping 54ºSW
- Random systems:
  - N18ºW trending, dipping 40ºSW
  - N49ºE trending, dipping 58ºNW

In the Main pit area, three main lithological domains were defined, namely:

- Overburden: aeolian and alluvial deposits, 1.0-6.1 m thick, that have no significant influence on pit slope design.
- Tunga andesite (ocoite): ranging from fresh to slightly weathered, but locally highly weathered in fault zones. UCS is high on average (77 Mpa).
- Rio Grande volcano-sedimentary rocks: weathering as above and with similar UCS and RMR values to ocoite.

Based on analysis of the data and pit configuration, 13 design sectors were defined for Mina Justa and three for Magnetite Manto. However, a review of the data shows that the quality of the rock mass and the structural features show little variation between design sectors.

The minimum factor of safety (FoS) criteria adopted for stability analysis are in accordance with Peruvian regulations, and are 1.3 under static conditions and >1.0 (post-earthquake) taking account of maximum seismic acceleration for 100-year return events. Based on a seismic study for the Marcona Copper Project by Knight Piésold, a maximum acceleration of 0.20 g is predicted for this return period; a value of 0.12 is assumed as the horizontal seismic coefficient, based on standard practice.

Two types of stability analysis of pit slopes were carried out for each design sector, namely kinematic stability analysis controlled by rock mass structures, and overall slope stability analysis controlled by rock mass quality.

From stability analysis, individual bench angles range from 65-70° for all design sectors. The angle of the overall slope, assuming 20 m final benches ranges from 45-49°, and up to a maximum of 50° in one part of the Magnetite Manto pit (Table 18.7).

Table 18.7
Pit Slope Design Criteria

Pit	Design Sector	Bench Slope Angle	Bench Height (m)	Bench Width (m)	Inter-ramp Slope angle
Northern Oxide	I	65	20	11	45
	II	70	20	10	49
	III	70	20	10	49
	IV	65	20	11	45
Main	V	65	20	11	45
	VI	65	20	9	48
	VII	70	20	10	49
	VIII	70	20	10	49
	XII	65	20	11	45
Cu40	XVI	65	20	11	45
	XVII	65	20	9.5	47
	XVIII	65	20	9.5	47
	XIX	65	20	11	45
Magnetite Manto	XIII	70	20	9.5	50
	XIV	65	20	9	48
	XV	65	20	9	48

Pit wall designs were found to be sensitive to mass blasting disturbance (disturbance factor 1.0) under pseudo-static (earthquake) conditions in three sectors. Consequently, controlled basting is recommended for development of final pit walls.

Knight Piésold also determined that 35% of the sectors have potential for planar and/or wedge daylighting and minor bench ravelling. Regular bench maintenance is recommended to control these.

Finally, a geotechnical instrumentation system is recommended to monitor pit wall stability during operations.

18.2.3 Site geotechnical investigations of tailings storage and process facility areas

Bedrock in the Mina Justa area is of Mesozoic (Jurassic) age, and consists primarily of volcaniclastic material of the Rio Grande Formation, comprising crystal tuff, epiclastic sandstone and lithic volcaniclastic rocks of andesitic composition, together with fine-grained andesite. This sequence is intruded by sub-volcanic porphyritic andesite (Tunga Andesite) of pre- and post-mineralisation age.

The basement is covered by Quaternary deposits of aeolian origin over the majority of the Mina Justa area, with alluvial materials locally important, for example in the tailings storage facility (TSF) area.

Site investigations indicate the cover material to be loose to medium dense aeolian sand, with thickness between 0.3 and 3 m.

Beneath the aeolian sand there is a thin layer of poor quality and weathered rock (RMR value of 29 approximately). Below this layer there is fair to good quality rock (RMR value between 45 and 55) extending to greater than 700 m below surface.

In the TSF area, the aeolian sands are underlain by dense alluvial materials, with a thickness of approximately 60 m, below which there is slightly consolidated sedimentary rock of Quaternary age from the Chaquillo Formation (about 15 m thick, which overlies rock of good quality (RMR 55-65), with thickness greater than 700 m.

The plant site area will be founded on fair to good quality intrusive/sedimentary rock (average RMR of 51) with an average depth of foundation of 1.6 m. The overlying aeolian sands and the poor quality rock material will be removed to reach the foundation level.

18.2.4 Borrow materials

Source areas for suitable borrow materials to be used during construction have been located. An approximate volume of 1 295 000 m^3 of common fill material is estimated to be contained in the North Quarry and South Quarry adjacent to the plant site.

Approximately 933 000 m^3 of fine construction materials have been identified in the North, South and Chauchilla Fine Material Quarries. While it is more distant from the construction area, the Chauchilla quarry is required to provide a source of suitable clay-fraction material for blending to achieve the required plasticity, permeability and particle size.

Further, it was confirmed that waste rock from the Magnetite Manto Pit is suitable for construction of the tailings dam. GRD Minproc has confirmed that waste rock for construction of the tailings dam is available from Magnetite Manto pit as required.

18.2.5 Groundwater

Results obtained from a vibrating wire piezometer in the main Mina Justa pit area indicate that the phreatic level (depth to ground water) is 470 m deep (312 masl).

The water level is approximately 90 m below the base of the final pit. Consequently the pit will be developed under dry conditions.

18.2.6 Site stability

The slopes, hills and low mountains in the area display stable conditions and there is no risk in respect to mass displacement events such as huaycos (a flash flood caused by torrential rains occurring high in the mountains), landslides or other mass displacement phenomena.

18.2.7 Seismic risk analysis

Both deterministic and probabilistic analyses were carried out by Knight Piésold. The deterministic analysis defined the potential source zones, types and properties of earthquakes that could be experienced at the site based on its seismic setting and included defining the potential Maximum Credible Earthquakes (MCEs) that the site could experience. The probabilistic analysis produced a statistical relationship between peak ground acceleration (PGA) and return period for earthquakes that the site could experience based on the recorded seismic history.

The deterministic analysis identified potentially significant earthquake as follows:

- Magnitude Mw 8.0 upper intraplate subduction earthquake, producing a mean plus one standard deviation PGA at the site of 0.48 g.

The event is considered to represent the MCE for the site, since it has been calculated to produce a significantly higher PGA. The probabilistic analysis gave PGAs at the site, in the free field, for selected return period events, as follows:

Table 18.8
Probabilistic Analysis - Peak Ground Acceleration in Rock

Site	Average Return Period (Years)	PGA[1] (g)	Comments
Mina Justa	50	0.15	
	100	0.20	
	475	0.36	10% prob. of exceedance in 50 years.
	1000	0.44	
	2475	0.58	2% prob. of exceedance in 50 years.
	5000	0.70	
	10000	0.84	

[1] . PGAs associated with 63% probability of exceedance minus the representative of the case where the design life in years equals the return period.

18.3 PROCESS PLANT DESIGN

The processing plant for the Mina Justa Project is conceived as being built in two stages. The first stage consists of the plant to treat Oxide ore to produce copper cathodes, and the second stage consists of a plant to treat Sulphide ore to produce copper concentrates. Much of the infrastructure for the two processing plants is common, thereby reducing overall costs compared to comparable separate facilities.

18.3.1 Oxide ore plant

18.3.1.1 Design basis

The Mina Justa Project utilises sulphuric acid leaching to extract copper from the Oxide ore. The leached copper is purified and upgraded by SX to provide a rich electrolyte to the EW plant, which produces copper cathodes. The feed to the leaching process is prepared by crushing and screening to achieve an -8 mm product size.

The Oxide ore process flowsheet is depicted in Figure 18.11, and the plant layout in Figure 18.12.

18.3.1.2 Crushing and screening

The crushing circuit is designed to reduce 12 Mt/a (1712 t/h at an overall 80% availability) of Oxide ore to minus 8 mm prior to leaching. This is accomplished using a four stage crushing circuit.

Primary crushing and stockpiling

Run-of-mine (ROM) ore is delivered to the crushing area by mine haul trucks, each carrying a load of 220 t. The trucks tip directly into the ROM bin, which is configured to allow tipping from one side only. Ore that cannot be fed directly to the primary crusher (e.g. due to maintenance down-time) is dumped on the adjacent ROM oxide ore stockpile, and later fed to the crusher by FEL.

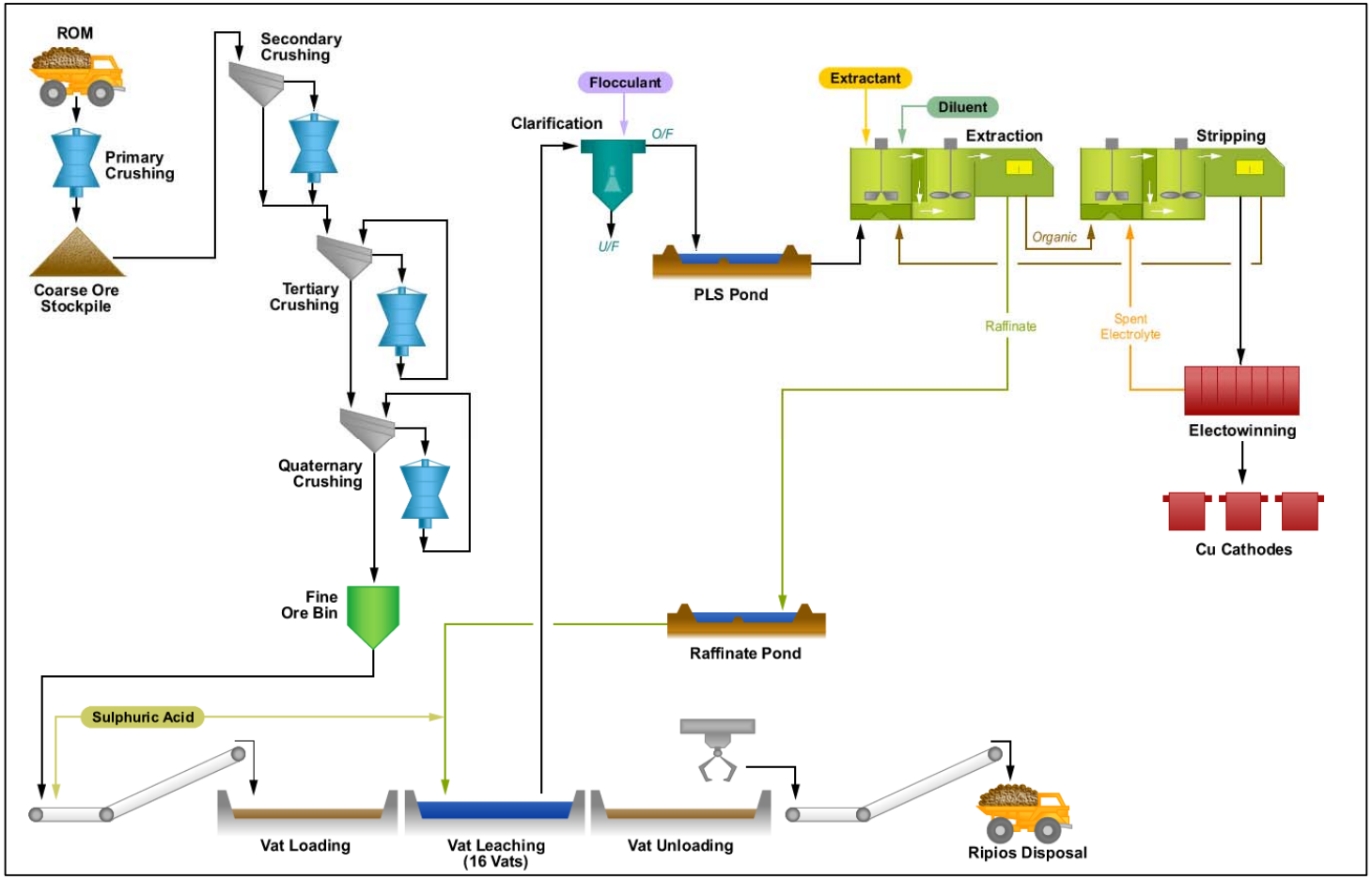
The primary crusher consists of a 54" x 75" gyratory crusher, which treats an average of 2283 t/h, sufficient to crush 12 Mt/a at 60% availability. A coarse ore stockpile provides surge capacity between the primary and secondary crushing stages to account for mine trucking cycles and maintenance requirements. The coarse ore stockpile has been sized to provide a live capacity of 12 hours.

Secondary, tertiary, and quaternary crushing

The coarse ore reports to a 12' x 27' scalping screen prior to secondary crushing. Secondary crushing is conducted via a 750 kW cone crusher operating in open circuit.

Tertiary screening utilises two 12' x 27' double deck screens. To reduce the height of the tertiary screening facility, the tertiary screens have been lowered and 9 m transfer conveyors have been installed under the screens to allow removal of the undersize material. Two 750 kW cone crushers operating in closed circuit are utilised for the tertiary crushing stage.

Figure 18.11
Mina Justa Oxide Circuit Flow Sheet



Quaternary screening is conducted by four 14' x 27' double deck screens. The quaternary screens have been lowered and 11 m transfer conveyors have been installed under the screens to allow removal of the undersize material, lowering the overall height of the crusher building. Quaternary screen oversize material reports to the quaternary crushing stage which utilises three 750 kW cone crushers operating in closed circuit.

Metal detectors and/or electro-magnets are included ahead of the secondary, tertiary, and quaternary crushers to detect and/or remove tramp metal to prevent damage to the crushers. When handling ores with large amounts of magnetite, such as at Mina Justa, the most effective means of capturing tramp metal is to have a detector tuned specifically for the ore mounted upstream of the magnet. The magnet normally runs at a low current (i.e. not enough to remove tramp metal or magnetite). Should tramp metal be detected, the magnet current is ramped up and timed such that as the tramp metal passes it is removed by the magnet. The magnet current is then ramped down to its normal operating current. A second detector is mounted downstream for additional safety.

To minimise capital cost wherever possible, much of the crushing facility is fully exposed to the weather. An open type of building is selected with easy access provided for mobile cranes for maintenance of equipment.

Dust control

Dust generated throughout the crushing plant is controlled by a combination of dust suppression and dust collection systems.

The primary crusher will be a major potential source of dust, particularly at the ROM bin during truck dumping. The truck tip-point is enclosed on three sides. Wetting sprays have been included to dampen the ore while on the truck, and a set of sprays has been included in the ROM bin to wet the ore stream as it is dumped, to suppress dust.

Ducted dust collection systems and high-efficiency wet scrubbers are included with extraction hoods at all major dust generating locations throughout the crushing circuit, including transfer points, crusher discharges, vibrating screens and feeders.

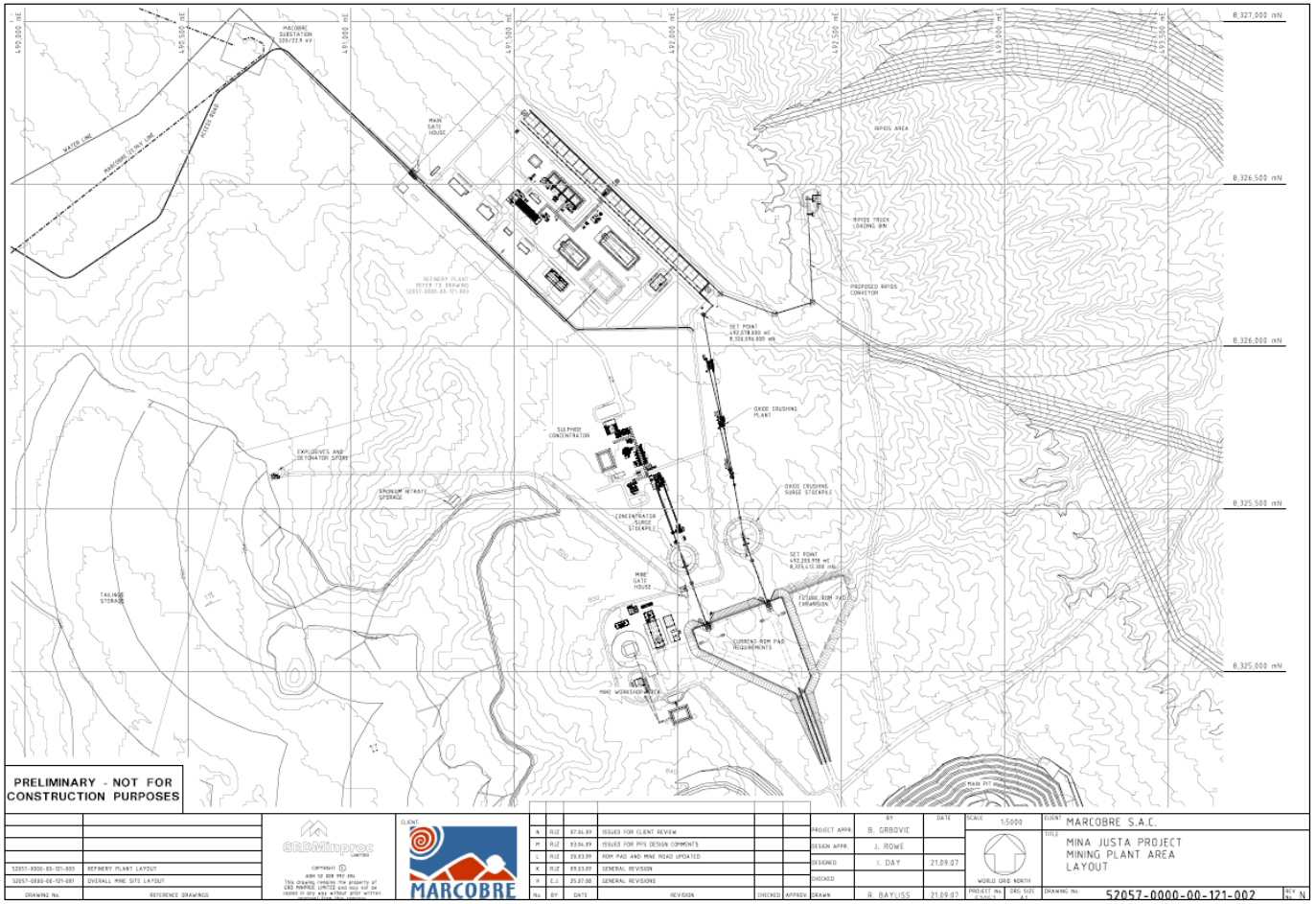
18.3.1.3 Vat Leaching

Although heap leaching is commonly adopted as a means of treating copper oxide ores, testwork has demonstrated that, for the Mina Justa ore, vat leaching is more effective economically. This method achieves higher copper recovery, and has been adopted for the Project.

Acid curing

Ore crushed to 100% passing 8 mm is delivered to the fine ore bin, which has a surge capacity of one hour. Due to the dry and windy site conditions, a bin has been provided as opposed to an open stockpile to minimise generation of dust.

Figure 18.12
Oxide Plant Layout



The crushed ore requires a short period of curing with acid to optimise leaching recovery. Therefore, after the crushed ore is drawn from the fine ore bin, it is sprayed with dilute sulphuric acid as it passes from one discharge conveyor to another. The nominal acid dosage is 15 kg/t.

Vat leaching

The acidified ore is transported by conveyor to the vat leaching area, where it is loaded into vats for leaching (Figure 18.13).

The vats are essentially reinforced concrete shells, each measuring 30 m wide, 40.5 m in length and 7.6 m high, and capable of holding a nominal 12 800 t of ore for a six day leaching cycle. At any one time, 16 vats are on-line. However, 18 vats have been designed to allow for loading, unloading, filling, draining and maintenance. The vats are designed to be acid resistant and are constructed to ensure that the leach solution is not lost due to leaks or seismic events.

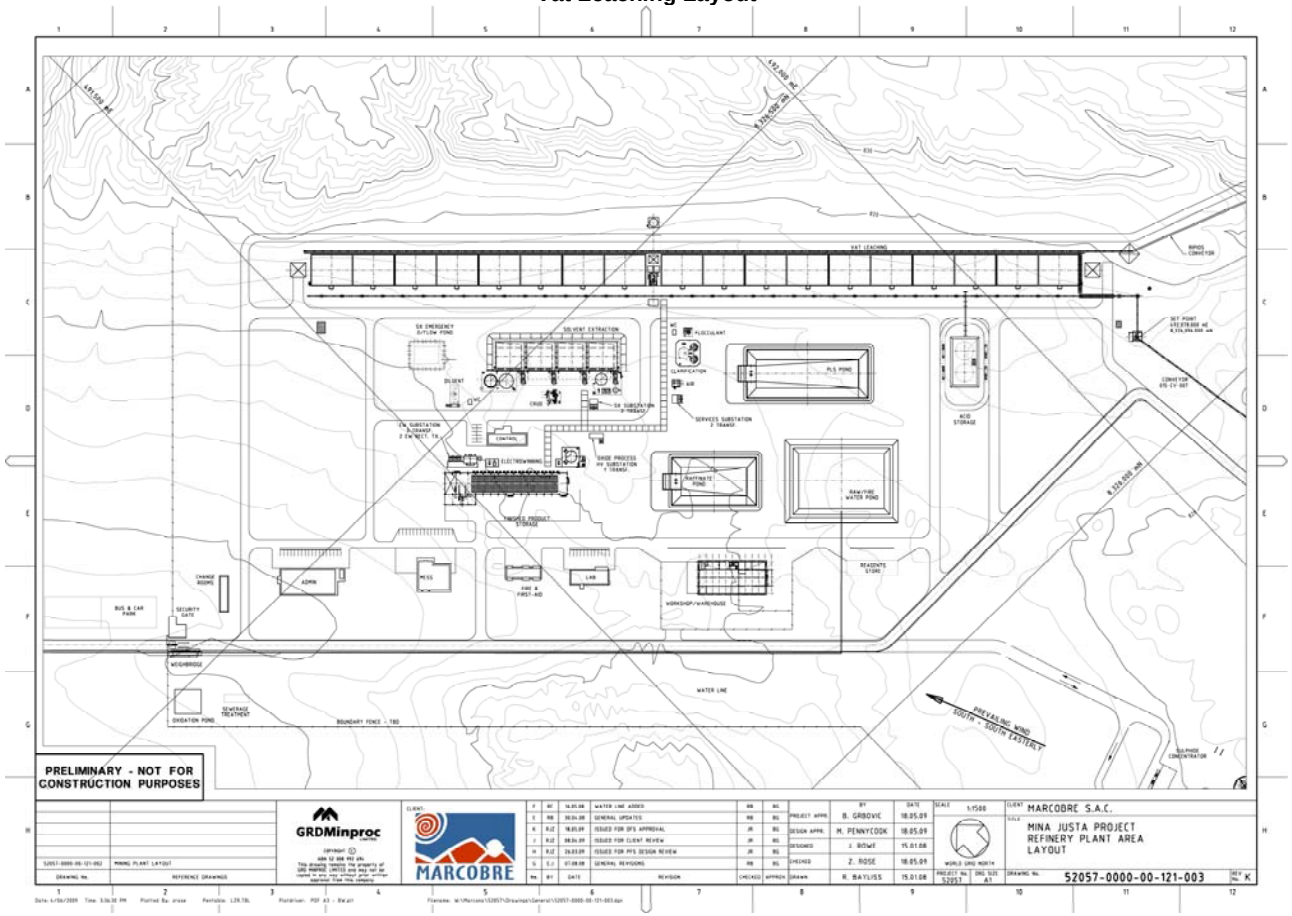
Acidified ore is loaded into a vat by means of a tripping conveyor until the vat is full, leaving 300 mm of freeboard in the vat. The vat is then flooded with a dilute sulphuric acid solution which is introduced through the base of the vat, under a filtration bed. Solution exits the vat by overflowing into a launder from which the solution is piped to the next vat or to a storage pond.

Leaching of acid-soluble (oxide) copper occurs over a period of six days, at the end of which most of the acid-soluble copper in the ore has been extracted. The exact percentage of acid-soluble copper recovered is dependant on various factors including ore grade and lithology. It is projected that over the life of the mine the average recovery of total contained copper will be 74.5%. At the end of the leaching cycle, the remaining solution is drained from the vat and the residual ore (ripios) is removed by a clamshell grab, placed into a hopper and discharged onto a conveyor belt for transfer to the ripios dump.

Solution management is designed as a counter-current system. Solution advances progressively to fresher ore in order to maximise the copper content of the leach solution before treatment in the SX/EW plant. The solution with the highest copper tenor overflows from the vats containing the freshest ore; this is known as pregnant leach solution (PLS). PLS is clarified then advanced to a covered holding pond before being pumped to the SX circuit.

The near barren solution returning from the SX plant, known as raffinate, is depleted of copper, containing only 0.48 g/L compared to 8 g/L for the PLS. It is the last solution in contact with ore before the ore is removed from a vat and sent to the ripios dump, thereby minimizing loss of copper in solution. The residual moisture in the ripios (approximately 11% by weight) is essentially raffinate and provides a bleed for impurities, so that no other bleed stream is required.

Figure 18.13
Vat Leaching Layout



Clarification

Pinned bed clarifiers have been specified because of their proven performance in removal of fines from copper leach solutions. A flocculant is added to enhance settling of the suspended solids. Due to the acidic nature of the PLS solution, materials of construction include SAF2205 stainless steel for the clarifier feed tank and fibre-reinforced plastic (FRP) and SAF2205 internals for the clarifiers.

The clarified PLS solution gravitates to the PLS Pond, with the clarifier underflow solids being pumped back to the vats.

Solution ponds

The PLS pond is 6 m deep and has been sized to contain 24 500 m^3 of solution. This pond is covered to reduce evaporation and prevent pick up of wind-blown solids. The raffinate pond is also 6 m deep and has been sized to contain 15 800 m^3 of solution. It is not covered, and it also serves as an emergency reservoir in case one of the vats is drained by accident or intentionally in an emergency.

Both ponds are lined with a double layer of HDPE membrane in order to avoid loss of valuable solution and prevent contamination of the environment. The dimensions also include an allowance to contain precipitation from a 100 year, 24 hour rainfall event.

Ripios

The ripios from the leaching stage are removed from the vat by an unloading crane with a 22 m^3 clamshell grab.

The clamshell discharges the ripios into a hopper that feeds a receiving conveyor. This material is then transported to the ripios area via three discharge conveyors. The last ripios conveyor discharges into the truck loading bin which, in turn, loads haul trucks for final disposal in the adjacent ripios dump. This dump is described in more detail in Section 18.7.1.

18.3.1.4 Solvent extraction

The SX process involves the selective extraction of copper from the relatively dilute PLS to produce a high purity, high tenor copper sulphate solution suitable for the EW process.

Configuration

The SX mixer/settler units are configured for two stages of extraction, one stage of wash and one stage of stripping. Solution is pumped from the PLS pond to the extraction circuit where it is contacted with the organic phase to extract copper from the aqueous phase. Loaded organic from the extraction circuit advances to the wash stage to remove entrained impurities such as iron, manganese, and chloride.

The washed organic and spent electrolyte from the EW process are combined in the primary mix tank. Due to the high acid content of the electrolyte, the copper transfers from the organic to the electrolyte during mixing. Then the mixture advances to the strip settler for disengagement of the aqueous and organic phases. Copper-rich electrolyte flows by gravity to the “strong electrolyte” tank. Strong electrolyte contains minor amounts of particulate solids and entrained organic, which are removed prior to EW using CoMatrix dual media filters.

A reverse flow design is selected for the mixer/settler layout to minimise the plant footprint and pipe run length. Primary and secondary mix tanks are utilised for each stage. The settlers are constructed with concrete walls lined with FRP. The settler roofs are constructed of steel cladding with access ports for maintenance. The roofs minimize the accumulation of dust in the SX circuit, as this would exacerbate crud formation.

The SX area includes a series of floor drains that connect to a set of sumps/firetraps. This arrangement eliminates pooling of corrosive or combustible fluids in the bund.

Crud handling

Crud from various areas within the SX plant is pumped through the crud centrifuge that splits the crud into its three constituent phases (i.e. aqueous, organic, and solid). The aqueous phase is returned to the SX circuit, whilst cleaned organic phase is either returned to the SX circuit or treated further with activated clay. Contaminated solids are collected for separate disposal.

Fire protection

SX fires in recent years have demonstrated that the fire must be suppressed in the very early stages to prevent destruction of the plant and surrounding infrastructure. Fire protection is incorporated into the design of the SX plant with a philosophy of automatic detection and initiation of suppression measures. For this reason, the fire protection system for Mina Justa comprises the following:

- Foam suppression to the SX bunds, SX settlers and tanks containing organic.
- A fire detection system for the bund and inside of each of the vessels described above.

18.3.1.5 Electrowinning

The copper EW circuit utilises permanent cathode technology to produce LME Grade A cathode copper. EW is conducted using a total of 122 cells at a nominal current density of 320 A/m^2. Copper plating is continuous over a period of six days before the cathodes are removed and processed for dispatch.

Electrowinning cells

The copper-rich electrolyte (“strong electrolyte”) passes to the EW circuit where copper is recovered by in the form of copper cathodes. Electrolyte that has been depleted of copper during the EW process, (“spent electrolyte”), is recycled to the strip stage in the SX circuit.

There are 26 polishing cells that receive strong electrolyte and act as organic entrainment protection for the commercial cells if there is a failure of the CoMatrix organic filters. Electrolyte overflowing the polishing cells flows to the electrolyte circulation tank and mixes with the spent electrolyte from the 96 commercial cells to result in a stream of circulating electrolyte. Each cell contains 69 stainless steel cathodes and 70 inert lead alloy anodes.

The EW cells are of monolithic polymer concrete construction, comprising vinyl ester resin mixed with aggregate. Electrolyte is circulated throughout the cell via a PVC manifold mounted at the bottom of each cell. Holes drilled into the PVC manifold allow electrolyte to pass between the electrodes in the

cell. Cathode quality is expected to be the same between the polishing and commercial cells. The polishing cells are generally viewed as insurance against contaminating the entire tank-house if organic breakthrough occurs in the filters.

Cathode stripping

Copper plating onto the stainless steel blanks is continuous over a period of approximately six days before the cathodes are removed for harvesting of the copper. Copper is removed from the cathodes by an automated cathode-stripping machine.

Approximately 3 t of copper sheets are accumulated before the bundles are sampled, strapped, and transferred by forklift to a dedicated storage area prior to dispatch.

Ventilation system

The EW cells are housed in a fully enclosed building to provide protection from climatic conditions (e.g. dust) and provide an acceptable working environment for the tank house operators.

Primary acid mist suppression is by a layer of polyolefin prills, which float on the cell surface and disperse the bubbles of acid mist as they reach the solution line of the cell. A forced cross-flow ventilation system provides secondary mist suppression by removing acid mist from the building.

18.3.1.6 Reagents

Sulphuric acid

Sulphuric acid (98%) is delivered by road tankers to the sulphuric acid unloading area. Four unloading stations have been provided to transfer the sulphuric acid into the two storage tanks. Each tank contains a live volume of 3187 m^3, sufficient to store a 7 day supply on site.

The tanks are sited within a HDPE-lined earth bund capable of containing 110% of the entire contents of sulphuric acid stored on site.

Flocculent

A non-ionic flocculent is dosed to the clarifier feed well. Flocculent is delivered to site by road on pallets containing 25 kg bags, and prepared with fresh water in a batching plant near the clarifier.

Extractant

Extractant (LIX984 or Acorga M5640) at a concentration of 25% by volume, is used in the SX process to extract copper from the PLS.

Extractant is delivered to site in 1 m^3 intermediate bulk containers (IBCs), off-loaded by forklift, and stored in a covered shed. The containers are moved to the solvent extraction area as required. Extractant is added (by gravity) to the SX circuit on a demand basis.

Diluent

High flash-point diluent (Shelsol 2046 or equivalent) is delivered to site by road tanker and off-loaded into the diluent storage tank, which has a storage capacity equivalent to 45 days.

Diluent is transferred to the SX circuit and the crud treatment area on a demand basis, using a single positive displacement pump.

Guar

Guar is a high molecular weight organic polymer that acts as a smoothing agent for the deposition of copper during the EW process, thereby enhancing the appearance of the final copper product.

Guar is received as powder in 25 kg bags, with storage on-site equivalent to 28 days of usage. The guar is mixed in an automated system and the solution is pumped to the EW circuit.

Cobalt sulphate

Cobalt sulphate is added to the EW circuit to maintain a cobalt concentration of 180 ppm, in order to enhance the stability of the lead anode coating.

The cobalt sulphate reagent is received in 25 kg bags, with storage on-site equivalent to 28 days of usage. The cobalt sulphate is mixed in a small mixing tank and dosed to the EW circuit as required.

18.3.1.7 Services

Raw water

Raw water is supplied from a borefield in the Jahuay aquifer, 31 km to the southeast. A network of bores fitted with screens and submersible pumps feeds into a holding tank at the borefield. From there water is pumped to site, an intermediate pumping station also being required to overcome the difference in elevation.

A 6 m deep raw water pond with 16 000 m^3 capacity is sited in the plant area to receive water from the borefield for redistribution around the site for process water, fire-water, dust control, camp, mine water trucks and other purposes.

Fire-water

The raw water pond also serves as the source of fire-water, with the pond and pumps configured to ensure a minimum amount of fire-water is always available in the pond. The raw water pump suctions are located above the fire-water pump suctions, so that the required four hours of water are available.

The fire-water pump set comprises an electrically powered main centrifugal pump, a diesel powered pump, and an electrically powered jockey pump. The fire-water system pressure is maintained using the jockey pump, thereby preventing premature starting of the main fire-water pump.

Potable water

Raw water is treated through the water treatment plant to produce potable quality water to be used for safety shower, drinking water, and ablution facilities. The water treatment plant uses chlorination to destroy any harmful bacteria present. The resultant potable quality water is transferred to the 80 m^3 potable water storage tank.

Plant and instrument air

Plant air at 750 kPag is provided from the two main plant air compressors and stored in the plant air receiver. From here it is reticulated to the plant air utility stations (excluding the crusher buildings). A separate portable air compressor is provided for use in the crusher plant areas.

A stream of plant air is diverted through a pair of air filters and fed to a duty/standby desiccant air drier to remove moisture from the plant air and generate instrument quality air. Instrument air is reticulated to points of demand.

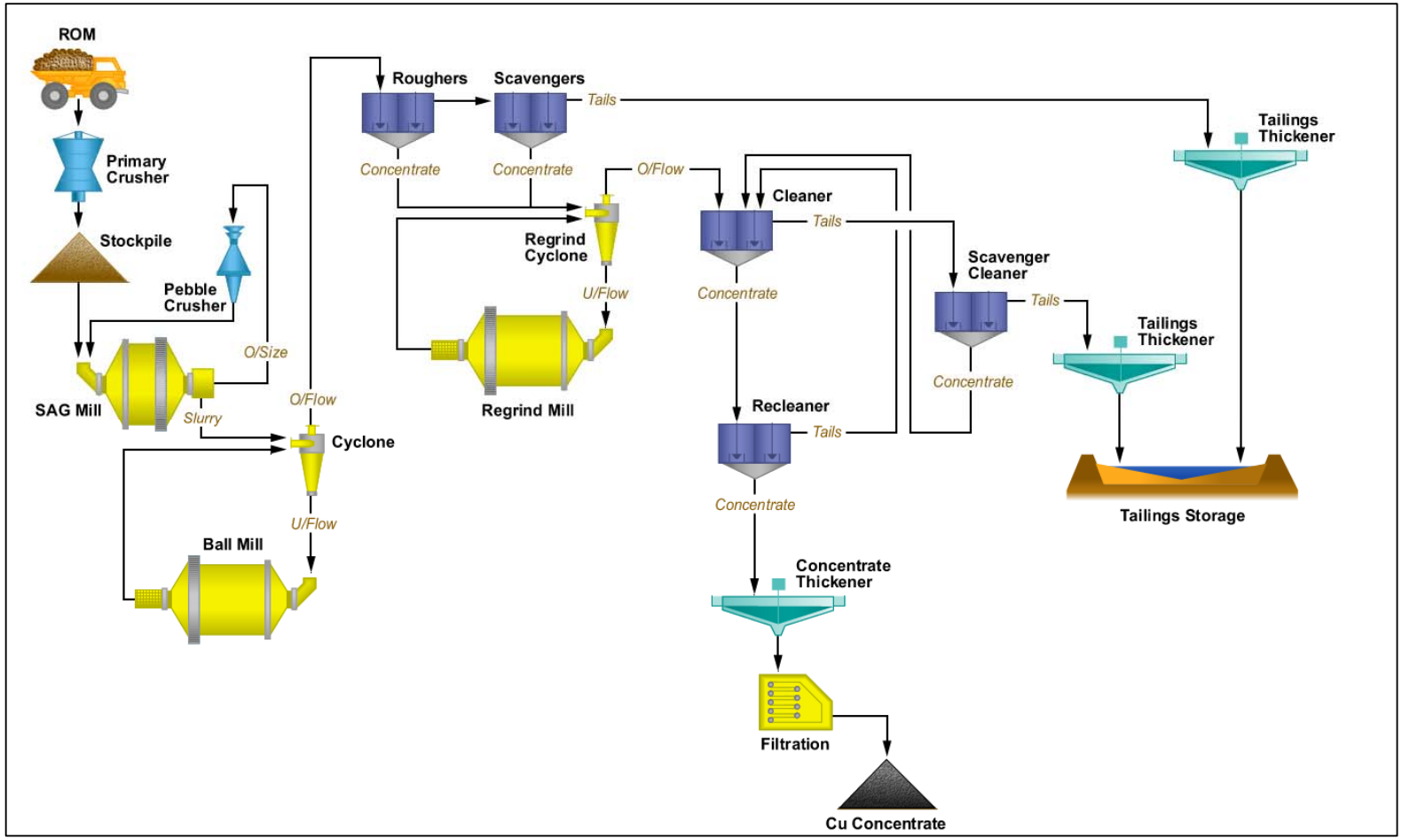
18.3.2 Sulphide ore plant

The Mina Justa Project includes, as the second of two phases, a concentrator for processing Sulphide ore, producing copper concentrates (also containing minor quantities of silver and gold) for sale on world and/or local markets. The concentrator will be built after the Oxide ore processing facilities.

The concentrator and related facilities have been designed and costed to a PFS standard rather than the DFS standard used in the design and costing of the Oxide ore processing facilities.

The overall processing flow sheet for the Sulphide ore is depicted in Figure 18.14 and the concentrator facilities are depicted in Figure 18.15.

Figure 18.14
Mina Justa Sulphide Circuit Flowsheet



18.3.2.1 Comminution

The comminution circuit is designed to treat 5 Mt/a of sulphide ore to produce a product size of 80% passing 150 µm.

Primary crushing

The Sulphide ore crushing circuit is completely separate from the Oxide ore crushing circuit.

ROM Sulphide ore is delivered to the crushing area by 220 t mine haul trucks. The trucks tip directly into the ROM bin which is configured to allow tipping from one side only. Ore that cannot be fed directly to the primary crusher, for example due to crusher maintenance, is unloaded to the adjacent ROM Sulphide ore stockpile, and later fed to the crusher by a FEL.

The primary crusher consists of a 54" x 75" gyratory crusher, which treats an average of 951 t/h, producing 5 Mt/a with an availability of 60%. A coarse ore stockpile provides surge capacity between the crushing and milling stages to account for mining trucking cycles and maintenance requirements. The coarse ore stockpile has been sized to provide a live capacity of 12 hours.

Dust control

The primary crusher will be a major source of dust, particularly the ROM bin during truck dumping. To counter this, the truck tip-point is enclosed on three sides, wetting sprays have been included to dampen the ore while on the truck, and a set of sprays has been included in the ROM bin to wet the ore stream as it is dumped.

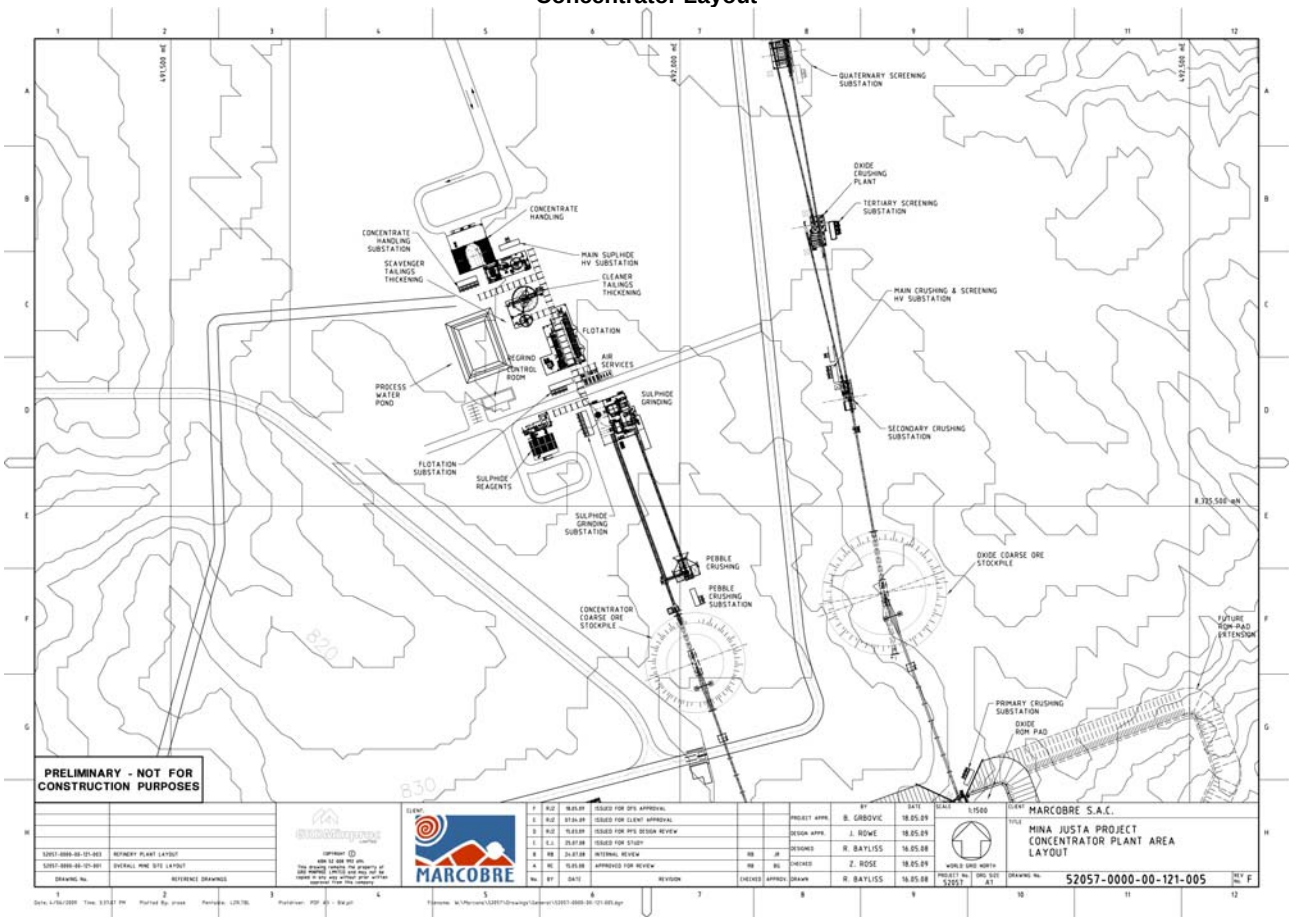
Ducted dust collection systems with high-efficiency wet scrubbers are included with extraction hoods at all major dust generating locations throughout the crushing circuit, including transfer points, crusher discharges, and feeders.

Primary grinding and pebble crushing

The primary grinding circuit consists of an open circuit semi-autogenous (SAG) mill with a pebble crushing circuit. The SAG mill feed conveyor transports crushed material reclaimed from the crushed ore stockpile to the SAG mill.

The SAG mill has a diameter of 9.15 m, with an effective grinding length of 5.3 m. It has been designed on the basis of a nominal 12% ball load. The motors are sized to take a maximum ball load of 15% at a 30% charge level and approximately 72% of critical speed. The mill is powered by twin 4000 kW hyper-synchronous wound rotor motors. The drive motors are variable speed with the adjustment achieved using slip energy recovery (SER) drives. The SAG mill is equipped with a dedicated lubrication system for mill motors, gearboxes, pinion bearings and mill bearings.

Figure 18.15
Concentrator Layout



Oversize pebbles from the SAG mill are transferred to the pebble crusher for size reduction. The pebble crushing circuit consists of a 600 kW cone crusher. The crushed pebbles are returned to the SAG mill via the SAG mill feed conveyor.

Secondary grinding and classification

The secondary grinding circuit consists of a ball mill in closed circuit with a cyclone cluster. The circuit targets a product size of 80% passing 150 µm.

The ball mill has a diameter of 6.57 m, and an effective grinding length of 10.75 m. It is powered by twin 4200 kW motors. The ball mill has a jacking cradle system and inching drive for maintenance purposes. The ball mill is equipped with a dedicated lubrication system for mill motors, gearboxes, pinion bearings and mill bearings.

18.3.2.2 Flotation

The flotation circuit comprises bulk flotation, concentrate regrind, cleaner flotation and on-stream analysis. The bulk flotation circuit produces a concentrate that is processed to liberate the locked copper minerals. Selective recovery of the copper minerals occurs in the cleaner circuit, where a concentrate of final product quality is produced.

Bulk flotation

The bulk flotation circuit aims to maximise recovery of copper minerals in the rougher-scavenger section. Cyclone overflow from the secondary grinding circuit, at a pulp density of 35% solids and pH of 9, reports to the rougher/scavenger circuit. The rougher flotation stage consists of two 70 m^3 tank cells, and the scavenger flotation stage consists of four 70 m^3 tank cells. The total installed residence time for the rougher-scavenger flotation circuit is 20 minutes.

Rougher/scavenger flotation concentrates are pumped to the regrind circuit for further grinding. Provision exists to transfer the rougher concentrate to alternative locations, such as cleaner feed. The scavenger flotation tailings are transferred to the tailings disposal circuit.

Concentrate regrind

The rougher and scavenger concentrates report to the regrind circuit for fine grinding. A single 3.8 m diameter ball mill operates in closed circuit with hydrocyclones. The regrind mill is powered by a 1300 kW motor and uses 40 mm balls as grinding media to achieve a P_{80} in regrind cyclone overflow of approximately 49 µm.

Cleaner flotation

Selective flotation is achieved through the addition of collector and frother, and by increasing the pulp pH to 11. Cleaner flotation is carried out in four 38 m^3 cells with a total nominal residence time of 10 minutes.

Cleaner concentrate is transferred for further cleaning in the recleaner circuit. The recleaners consist of three 16 m^3 u-shaped flotation cells with a total nominal residence time of 10 minutes. The recleaner concentrate is pumped to the concentrate handling area.

The cleaner flotation tailings flow to cleaner scavenger flotation. The cleaner scavengers consist of three 38 m^3 u-shaped flotation cells with a total nominal residence time of 10 minutes. The cleaner scavenger flotation tailings are transferred to the tailings disposal circuit.

Sampling and analysis

Eight sample streams are collected for on-line control of the flotation circuit. Various in-stream samplers collect samples and direct the streams to a multiple stream analyser. The analysis is undertaken using an XRF analyser. The rejects from the sampling system are pumped to their respective return points in the process.

Rougher feed and regrind overflow samples pass through an analyser for particle size determination.

18.3.2.3 Concentrate handling

The recleaner concentrate is screened to remove debris from the slurry, in order to protect the thickener and downstream filter operation. Thickening of the concentrates is conducted using a 15 m diameter high-rate thickener to produce a product at 65% solids.

The thickened stream is transferred to the filter feed-tank, which provides a storage capacity equivalent to 12 hours. The concentrate solids are dewatered by a pressure filter. The filter discharges moist concentrate directly onto a storage slab below the filter. The filtrate returns to the concentrate thickener.

Concentrate is transferred from the stockpile into a storage shed by a FEL, which is also used to load road trucks for shipment.

Tailings thickening and disposal

Two tailings streams are produced by the concentrator, the cleaner scavenger tailings (CST) stream with potential for acid generation, and the rougher scavenger (RST) tailings stream with low acid generation potential. The two streams are disposed of separately, because combination of the two could result in a single tailings stream with the potential to be a net generator of acid.

- Rougher scavenger flotation tailings

  The RST are pumped to the tailings thickener, which is a 28 m diameter high-rate unit. Thickener overflow discharges to the process water pond. The thickener underflow stream, at 60% solids, is pumped to the RST section of the tailings storage facility described in Section 18.7.2.

- Cleaner scavenger tailings

  The CST require selective disposal to control the release of elevated concentrations of controlled elements and reduce the tailings area to be treated during closure. The tailings are discharged sub-aqueously to inhibit sulphur oxidation.

  The CST report to the CST thickener, which is an 11 m diameter unit. Thickener overflow discharges to the process water pond, while the thickener underflow stream, at 60% solids, is pumped to the CST section of the tailings storage facility described in Section 18.7.2.

18.3.2.4 Reagents

Collector (Sodium Isopropyl Xanthate)

Sodium isopropyl xanthate (SIPX) is the collector used in the flotation process. Collectors promote recovery of mineral particles by creating hydrophobic surfaces for air bubble attachment during flotation. SIPX is delivered to site in 1 t bulka bags, with storage being provided for 20 bags. A 2 t monorail hoist lifts the bags into a bag splitter chute above a 6 m^3 agitated collector mixing tank. A 20% solution is prepared with raw water and pumped to the collector header tank for distribution.

Promoter (Aerofloat 3477)

Aerofloat 3477 promoter consists of dithiophosphates, which are selective for copper recovery and are generally used in conjunction with xanthate collectors. A3477 is delivered as a liquid to site in 210 L drums, with storage being provided for 75 drums. The promoter is transferred from the drums by a drum pump to a 1 m^3 promoter storage tank. The promoter is dosed into the flotation circuit from this tank by dedicated metering pumps.

Frother (Dow 250)

Frother stabilises air bubbles that reach the surface of the agitated slurry in the flotation process, creating froth. Dow 250 is received as liquid in 210 L drums and storage is provided for 75 drums. The frother is transferred from the drums using a drum pump to a 1 m^3 frother storage tank, from where it is dosed in the flotation circuit with dedicated metering pumps.

pH Modifier (Lime)

Lime is used is to regulate the pulp pH, suppressing the flotation of iron sulphides and improving copper sulphide mineral flotation. Hydrated lime (85% $Ca(OH)_2$) is delivered to site as a bulk solid and stored in a 60 t hopper. Lime solution is prepared with raw water and transferred to a 20 m^3 agitated lime storage tank prior to distribution to the plant through a ring main.

Flocculent

Flocculent is added to the concentrate and tailings thickeners, to aid settling of solids by agglomerating fine particles in the slurry. Flocculent is transported to site as a solid in 25 kg bags. Storage is provided on-site for 400 bags. Dry flocculent powder is transferred to the storage vessel when required. Flocculent is mixed in an automated system and is made up to a concentration of 0.3% w/w. Flocculent solution is delivered to the respective thickeners from the flocculent storage tank using dedicated variable-speed metering pumps. Flocculent solution is diluted to 0.03% w/w prior to dosage.

Sodium sulphide

Sodium sulphide (Na_2S) is used in the flotation circuit as a sulphidising agent to improve the flotation of partially oxidised minerals. Sodium sulphide is delivered to site in 1 t bulka bags, with storage allowance for 15 t in a secured area on-site.

A 2 t monorail hoist lifts the bulka bags into a bag splitter chute above a 6 m^3 agitated mixing tank. Sodium sulphide is made up to a 15% solution concentration with raw water before being transferred to a 10 m^3 storage tank sited in a concrete containment area capable of storing the entire contents of the tank in case of an emergency. The solution is metered to the rougher flotation feed box.

18.3.2.5 Services

Raw water
Sulphide plant raw water requirements are provided from the Oxide plant raw water pond. A set of raw water pumps is installed at the pond to supply the Sulphide plant with its raw water requirements.

Fire water
Fire water for the Sulphide circuit is supplied from the Oxide fire water system as described in Section 18.3.1.7.

Potable water
Potable water is provided from the Oxide Plant potable water system described in Section 18.3.1.7.

Process water
Process water is composed of water that has been used in the processing of ore, such as thickener overflows. The process water pond has a capacity of 4500 m^3. Provision exists to transfer raw water into the process water pond to maintain the level at a predetermined value. Process water is distributed to the process plant areas as required.

Plant and instrument air
As it would be impractical to pipe the compressed air over the distance between the two facilities, the concentrator has a dedicated plant and instrument air system.

Plant air at 750 kPag is provided from the two main plant air compressors and stored in the plant air receiver where it is reticulated to the plant air utility stations. A stream of plant air is diverted through a pair of air filters and fed to a duty/standby desiccant air drier to remove moisture from the plant air to generate instrument quality air.

Two independent flotation air blower systems are utilised for bulk flotation and cleaner flotation air supply.

18.4 GENERAL INFRASTRUCTURE

In addition to the mine and process plant, the Mina Justa Project includes significant infrastructure, principally in the areas of power, transport and water supply. Key components of these are situated off-site, with links to the Project area.

Additional on-site infrastructure includes buildings for accommodation and offices, workshops, stores and explosive magazines. Facilities for waste disposal, security and fire protection have also been considered.

The infrastructural facilities for the Mina Justa Project are depicted in Figure 18.16.

18.4.1 Access roads

A 9.6 km access road has been surveyed and designed, linking National Route 026 (connecting the Municipality of San Juan de Marcona to the Panamericana Highway) to the plant site and

accommodation camp. An extension runs to the accommodation camp. This road will be used not only by light and passenger vehicles, but also by heavy vehicles transporting fuel, reagents and spare parts to site, and hauling cathodes and concentrates to the port. Consequently, the road has been designed for sustained, long-term use, including adequate foundations and a tarmac surface.

Signage will be provided to control traffic movements, particularly where heavy vehicles are operating.

18.4.2 Internal roads

An internal road network is required to provide access from the main offices to the mine, waste and ripios dumps, stockpiles, crushing and other plant areas, and the TSF. Some of these roads will carry heavy traffic, but are more transitory in nature. Consequently, the roads will be surfaced with crushed rock and maintained by watering (to control dust), grading and periodic resurfacing.

A lighting system is provided for internal roads.

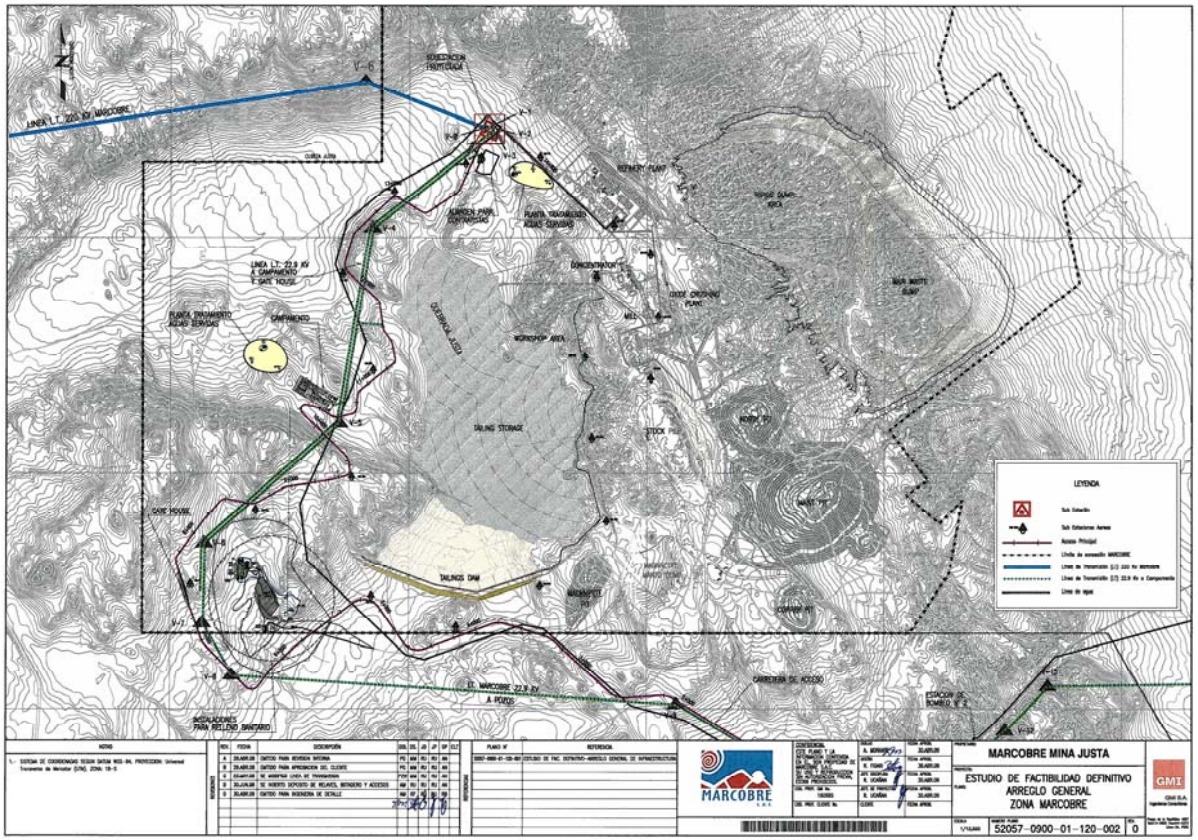
Extensive road signage is considered very important, particularly where mine haul trucks are operating.

18.4.3 Buildings

Building design has taken into account the following:

- Avoid environmental impacts from emissions produced during the process.
- Provide safe working conditions and protect workers from harmful emissions.
- Ensure that the design of permanent and temporary buildings is safe and solid, in order to avoid structural failure and to comply with applicable building codes and regulations, and in particular seismic conditions.

Figure 18.16
Mina Justa Project Plan



The list of the buildings includes the following:

- **Process Plant Site:** concentrate store, reagents store, laboratory, administration building, control room, change room and toilets, first aid and fire station, gate and sentry box, main security building, main EPCM office, workshop and offices, warehouse and offices, dining room, site toilets and sewage treatment plant.
- **Plant Substations:** main substation, crushing and screening HV substation, primary crushing substation, secondary crushing substation, tertiary screening substation, quaternary screening substation, Oxide process area substation, vat leaching substations, SX substation, EW substation, EW rectiformer building, services Oxide substation, administration/services substation and magazine store.
- **Mine Site:** mine offices, heavy vehicle workshop, mine workers change rooms and toilets, vehicle wash-down and vehicle refuelling.
- **Mine Substations:** mine office, mine workshop substation.
- **Water Supply Substations:** water borefield, pump station No. 1, pump station No. 2.

18.4.4 Construction and accommodation camp

A construction camp is situated on the site, some 4.4 km southwest of the plant area. This is designed to hold 990 persons, which is the expected maximum labour force at any one time during construction. During operation, the permanent camp holds 300 people.

The camp will be equipped with accommodation, kitchen and eating areas, medical, security, communications and recreation facilities. A gate-house is provided where the access road enters the Concession. Except for those accommodation units that will be used during operations, accommodation will be in pre-fabricated transportable units. Accommodation and other facilities to be used during operations, such as offices, change-rooms, and a cafeteria, will be constructed of masonry and structural steel with roofing and siding, concrete slabs and foundations.

18.4.5 Sewage and waste water treatment

Sewage treatment plants meeting World Health Organisation standards will be supplied by a suitably qualified contractor and installed at each of the construction camp, administration office, plant sites and mine workshop areas.

Although no liquid effluents will leave the site, water treatment plants will be installed in each of the above areas to treat wash-down and other “grey” water, which will then be re-used for dust control, plant process water and any vegetation programmes, as appropriate.

18.4.6 Other inert residual waste

A policy will be established to minimise usage and maximise recycling of domestic wastes such as paper, aluminium, glass, plastics, etc., through the provision of receptacles throughout the camp and offices, combined with instructions to all personnel (including cleaning staff) in the use of these facilities. Collection will be undertaken regularly, with separated materials transferred to a secure, central storage

facility on site for consolidation and onward transfer to reprocessors. A registered contractor will be used to undertake these transfers.

Organic wastes will be collected and composted in a suitable facility on site.

Construction materials will include a significant proportion of recyclable material such as timber and metal off-cuts, plastics and other packaging. Such materials will be collected and transferred to reprocessors, using a registered contractor.

During construction and particularly during operations, significant quantities of used oil filters and vehicle and plant parts will be generated. These items will be collected and taken to the central storage facility, recyclables separated, and the inert residual wastes landfilled in a specially designed, secure, registered, sanitary landfill site on the property.

Management of other inert residual wastes will be by burial in the sanitary landfill site on the property, with sufficient capacity for the proposed construction, operation and closure phases over the life of the mine. Relevant parts of the residual stream will be incinerated in a suitable facility and the ashes deposited in the sanitary landfill site. The remainder will be landfilled directly**.**

18.4.7 Management of dangerous waste

A study of waste management, including dangerous waste management, was undertaken for Marcobre by GMI. Dangerous wastes are defined as materials which are corrosive, reactive, explosive, toxic, inflammable and bio/pathogenic. Specific legislation covers the management, handling, transport and treatment of dangerous wastes, in order to minimise environmental, health and social impacts.

Dangerous wastes associated with the Mina Justa Project are discussed in the following sub-sections. In general, they will be collected and stored briefly at the point of generation, before being transferred to the central storage facility. Those materials that can be rendered inert will be treated at the site and then landfilled in the sanitary landfill, while others will be sent off-site either for treatment and re-usage, or for permanent disposal in a compliant dangerous materials storage site. Registered transport and disposal companies will be used for this purpose.

18.4.7.1 Contaminated soils

Soils contaminated with hydrocarbons will be treated in a lined facility at the central storage facility by evaporation and treatment with enzymes or bacteria to encourage breakdown.

18.4.7.2 Combustibles

Combustible materials and substances used on site include conveyor belts, oil, grease, fuel, tyres, synthetic liners, organics used in the SX process, various reagents, etc. Handling of combustible waste will be as follows:

- Recyclable combustible materials: waste reduction strategy includes reuse and recycling of as many materials as possible. Air filters are examples of materials that can sometimes be cleaned and reused. Used oil and coolant are examples of materials that can and will be recycled.

- Non-putrescible combustible waste: non-recyclable, non-putrescible items such as cardboard boxes, packaging materials, wooden shipping materials, plastic sample bottles and containers, tyres and other non-recyclable materials, will be handled by direct burial. Items such as clean, non-recyclable synthetic liners and other inert materials that would not benefit from burning will be placed directly into the landfill trenches. Tyres will be placed along the base of the waste rock dumps where they will be covered by the next lift or material

18.4.7.3 Laboratory reagents, chemicals, fluxes and laboratory products

Minor quantities of typical laboratory reagents will be used in the site laboratory. The waste products from the laboratory are minor in volume and generally inert, so these can be disposed of in the sanitary landfill. Non-inert wastes will be stored in sealed containers and transferred periodically to a registered dangerous materials storage site.

18.4.7.4 Explosives

ANFO is the main bulk explosive used on site. Emulsion may be used in smaller quantities as required for additional breakage, but is not required to counter wet conditions.

Explosive supply and down-the-hole service will be provided by an explosives contractor. The contactor will manage the bulk explosive yard and explosive magazines which will be designed, constructed and managed to comply with Peruvian law. The annual consumption is estimated to be about 14 500 t, requiring approximately nine deliveries of 30 t each to site per week. A maximum of 1000 t of ammonium nitrate will be stored on site at any one time. This will be kept in a remote and separate warehouse. The magazine and detonator house will be located in a separate, remote, fenced area and will be guarded by security personnel at all times.

18.4.7.5 Crud

Crud is a term used to describe extraneous matter that may contain solid, aqueous and organic phases that is present in SX circuits. The crud is collected and processed in several treatment stages to separate the material into its constituent phases. The separated aqueous and organic phases are returned to the main circuit and re-used. Contaminated solids from the crud treatment circuit are collected and placed in a skip for disposal in a registered facility off-site.

18.4.7.6 Anode sludge

Anode sludge accumulates in the bottom of EW cells from the breakdown of the lead oxide/sulphate coatings on the anodes. The cells are, therefore, cleaned periodically to prevent an excessive build up of anode sludge.

Anode sludge removed from EW cells is generally recognised as a toxic material, and stringent handling and disposal procedures are followed. Anode sludge is typically sent to a smelter for reprocessing, and not stored permanently on-site.

18.4.7.7 Other dangerous wastes

Lead-bearing products such as batteries, fluorescent products, welding rods, paint and other dangerous wastes will be generated during construction and operations, and will be managed firstly by consolidation into the central on-site facility, and thereafter by transfer to and deposition in a registered dangerous wastes facility off-site.

18.5 WATER SUPPLY SYSTEM

The Project site is characterised by:

- Extremely low rainfall, the yearly average being conservatively estimated at 27 mm[8]
- One hundred year frequency 24-hour storm event predicted to result in 11 mm of precipitation
- Evaporation greatly exceeding precipitation throughout the year
- No surface or groundwater at contemplated mining depths.

As a consequence, the Project has been designed to have zero surface discharge of mine and process water from site. A study of the water balance by Knight Piésold considers that water will be transported to site by pipeline from the Jahuay aquifer, 31 km distant, for use in mining, leaching and flotation processes, for dust control, personal consumption, and for fire protection.

The types of water considered as requiring management are:

- Water required during construction to support construction activities and construction personnel.
- Water involved in operations:
  - Mining (dust control)
  - Waste dump, ripios dump and mining facility run-off
  - Plant and infrastructure area run-off
  - Potable water for personnel requirements
  - Leaching (make-up water for sulphuric acid leach)
  - Flotation (water used in milling and flotation circuits)
  - Sewage
  - Bleed streams from the SX and EW circuits.
- Precipitation, including post-closure.

18.5.1 Project water balance

A site-wide water balance has been developed to quantify the amount of make-up water that is required to sustain operations, taking account of water that can be recovered from the tailing storage facility, if any, as well as the required pond storage capacities. In addition, the amount of water lost in the vat leach process and through evaporation has been determined.

[8] During [24] months of monitoring on site to the end of 2007, no rainfall was recorded

The water balance was completed using a Knight Piésold computer model that makes monthly calculations of the amounts of water in the system from defined or calculated inflows and outflows. A principal input is precipitation; precipitation data were selected from a separate Knight Piésold study. A second principal input is the proposed operation of the Project, including consideration of planned operation of the TSF for cleaner and rougher tails. It should be noted that the rougher tails facility is an unlined facility (as the tails will not generate acid), whereas the cleaner tails facility will be lined to prevent acid drainage from entering the ground below the facility. Additional information was supplied by GRD Minproc and Marcobre with regards to the loading, leaching and unloading of the vat leach cells and this information was input into the model.

18.5.1.1 Sulphide plant operations

The results of the water balance indicate that under normal operating conditions the volume of the supernatant pond will vary between about 79 800 to 116 200 m^3 for the rougher tailings facility and 20 280 to 243 109 m^3 for the cleaner tailings facility. The results also indicate that under an extreme precipitation event condition (Probable Maximum Precipitation or PMP), the design maximum pond size will vary between 166 200 to 220 200 m^3 for the rougher tailings facility and 34 529 to 278 885 m^3 for the cleaner tailings facility, over the life of the operation.

In addition to providing capacity to store the operation and storm water volume, 1 m of freeboard has been provided at each stage of development of the TSF.

Due to the dry climatological conditions of the site, the water balance is significantly in deficit, requiring the addition of make-up water from an outside source to sustain operations. Based on the water balance results, the supernatant water pond is not expected to cover the entire cleaner tailings beach. However, management of the supernatant water pond should provide for covering as much of the CST as possible. A reclaim water system from the supernatant water pond to the plant site was not considered to be practical.

The make-up water requirement for the sulphide circuit is estimated to be 375 m^3/hr.

18.5.1.2 Leaching operations

For the vat leach system, input information with regards to ore moisture contents at leach and during removal of the ripios from the cells were used to determine the amount of water that is recovered from the system. In general, 710 m^3/hr of water is sent to the SX/EW plants for processing. Outside source make-up is required to sustain operations for this facility as well; predicted make-up requirements are roughly 140 m^3/hr for the life of the facility.

18.5.1.3 Other areas

Apart from the process operations that require outside sources of make-up water, approximately 45 m^3/hr of “other” make-up is required for the operation of the camp and for water trucks used for dust suppression on the roads.

18.5.1.4 Total operational requirements

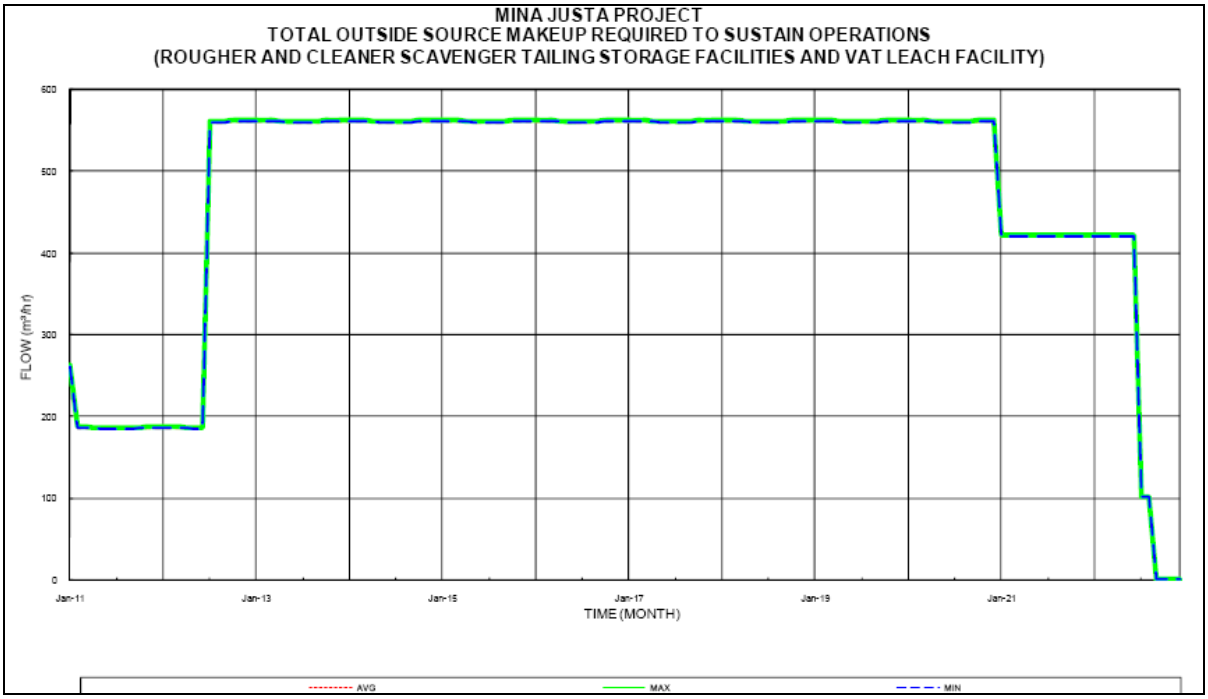
The total outside source make-up requirement is roughly 186 m^3/hr from year 1 through Q7 in year 2 (Oxides). From years 2 through 10 the requirement increases to 589 m^3/hr (Oxides + Sulphides), but it decreases for years 10 through 12 to 420 m^3/hr (Sulphides), as shown in Figure 18.17.

18.5.1.5 Start-up water requirements

The start-up water requirements in the mill are predicted to be 9 714 m^3, and an additional 45 m^3/h for "other" sources is required to sustain operations. The start-up water requirement in the vat leach facility is predicted to be 77 423 m^3.

For the start-up of the Rougher and Cleaner TSF, it is estimated that continuous flow of 319 m^3/h for the roughers and 56 m^3/h for the cleaners is required.

Figure 18.17
Water Demand



18.5.1.6 Closure and reclamation water requirements

For closure and reclamation, the CST facility will be covered to limit the ingress of air. It may be necessary to cover the entire facility with natural surficial soils to prevent the movement of tails due to wind. In order to place the cover materials on these facilities, the water contained in the supernatant pond will be evaporated and the area dried. It has been assumed that this water will be pumped into a spray system that will spray the water on the deposited tails area and left to evaporate. It has been determined that it will take approximately one year to evaporate the water from the RST and CST storage facilities.

The climatological data analyses indicate there is no net accumulation of precipitation on the site. In addition, each the tailing storage facility has capacity to store the run-off associated with the PMP with 1 m of freeboard. Based on this information, it has been determined that there is no need to regrade or construct a spillway to divert run-off from large precipitation events.

18.5.2 Hydrological testwork and studies

In March and April 2008, Ground Water International SAC (GWI) conducted a field investigation of the Jahuay and Lomas aquifers (Figure 18.18) to address the supply requirements. The program was designed to complement a previous study undertaken by Vector (2006) and included:

- Climatic water balance
- Test well drilling and installation in the Upper Jahuay aquifer, about 31 km from the project site, and in the Lomas aquifer, about 50 km from the project site

- Numerical modelling and other analysis.

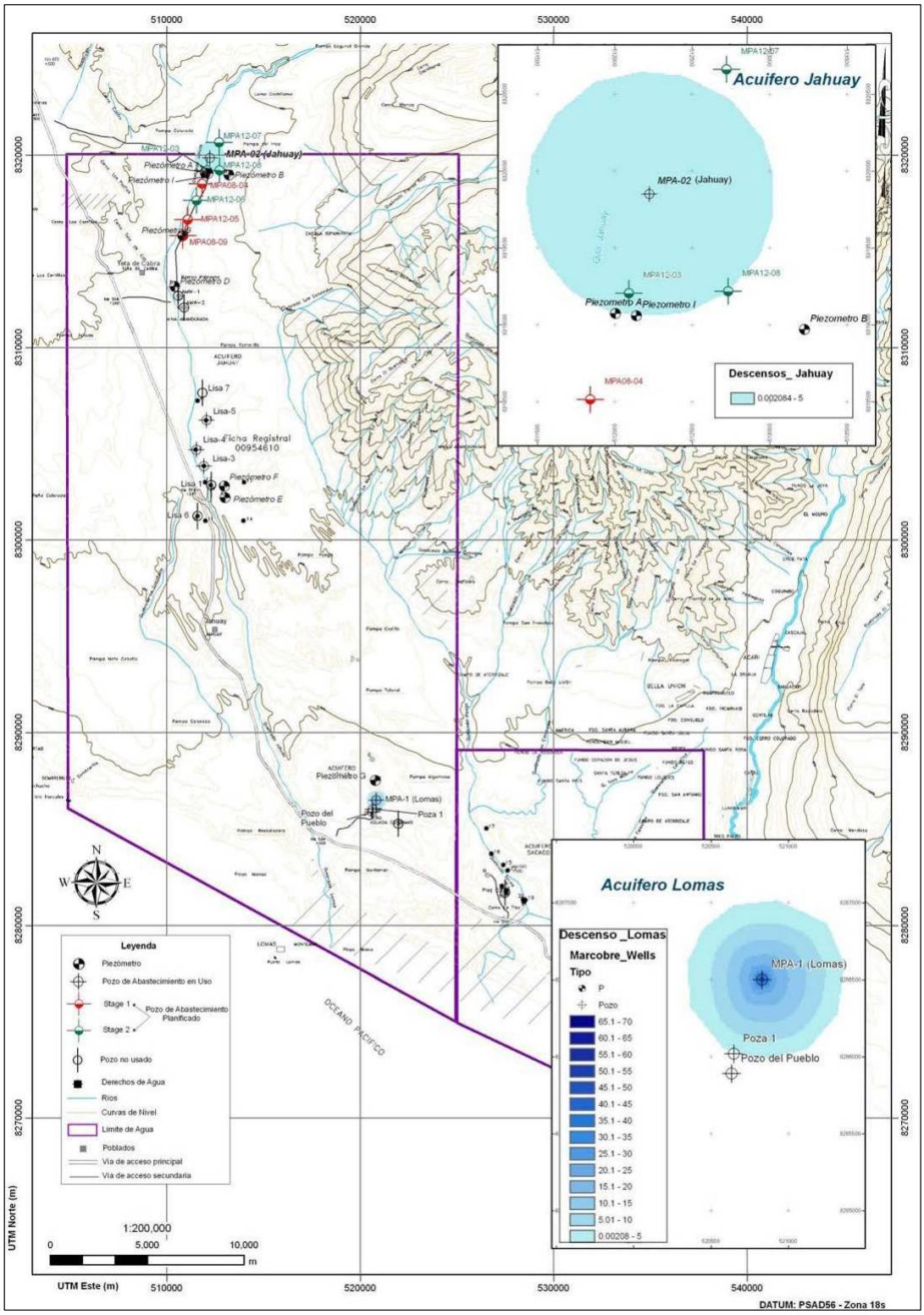
18.5.2.1 Test wells, hydraulic and hydrochemical testing

Two test wells MPA-1 (Lomas) and MPA-2 (Jahuay) were drilled by AK Drilling S.A using a Foremeost Barber dual-rotary drilling rig (Figure 18.18).

Test-production well MPA-1 with a diameter of 8" (200 mm) was installed in the Lomas aquifer to a final depth of 119 m, with a static water level at about 3 m below ground (mbg). Test pumping indicated that this well has a moderate supply potential, with a long-term safe yield estimated at about 5 L/s, with water quality suitable for the use intended. It should be noted, however, that other wells in the area have encountered slightly brackish water. There is some potential to affect neighbouring wells in the area, should significant water withdrawal occur. Overall, however, this aquifer is considered to have good water supply potential, and ranks as a potential back-up water source.

Test-production well MPA-2 with a diameter of 10" was installed in the upper Jahuay aquifer, to a depth of 245 m. An 8" telescopic screen assembly was installed between depths of 220 to 226 mbg. The well has a static water level of approximately 83 mbg and a long-term safe yield estimated at 34 L/s (not considering potential well interference from future neighbouring wells).

Figure 18.18
Locations MPA-1 Lomas, MPA-2 Jahuay and Proposed Test-Production Wells



The average hydraulic conductivity values for the aquifer are high, estimated in the range of 1 to $2x10^{-4}$ m/s, and the storativity value estimated for the pumping test was 0.00115 (0.12%).

The water is fresh, pH-neutral with slightly elevated levels of iron (0.3 – 0.6 mg/l) and potentially corrosive to mild steel over the long term, largely due to the low total dissolved solids (TDS) content. However, corrosion potential is anticipated to be minimal over the proposed life of the project.

18.5.2.2 Climatic water balance and sustainability of the source

The climatic analysis confirms that the aquifer is fed almost exclusively by water surplus generated within the Jahuay/Carbonara basin at elevations between approximately 3300 and 4200 masl.

Average annual water surplus for the basin is estimated to be in the range of 40 to 140 L/s; the significant range is largely due to the lack of reliable evaporation data at altitude for the area, and the lack of flow monitoring data in the Upper Jahuay valley. It is recommended that instrumentation be installed to provide more precise estimates of the water surplus.

It is expected that most of this water surplus recharges the aquifer via “transmission losses” (infiltration through the sandy bed of the stream) in the valley above the proposed wellfield location. Some additional water arrives in the wellfield area via groundwater flow from the upper basin, and, during peak storm events, some will run-off within the valley south of the proposed wellfield.

Numerical modelling indicates that withdrawals in the order of those required for the Mina Justa Project are sustainable in the medium-term (i.e. <50 year), due in part to the high storage capacity of the sand and gravel aquifer, since it is possible that total annual withdrawals by Marcobre, Marcona and Shougang from the aquifer will exceed the average annual water surplus for the basin. Based on the analysis of both simulations it appears that pumping of the Marcobre wellfield to supply the total water requirement should not adversely affect the Shougang/Marcona field, located 9.6 km south of the southernmost Mina Justa well modelled, even if Shougang/Marcona were to double current well usage in the near future.

Following closure of the Mina Justa operation, water levels will gradually recover to pre-operation levels.

The Upper Jahuay aquifer is the preferred source, in view of the following factors:

- A shorter pipeline length (estimated 31 km lineal distance against 48 km from Lomas)
- Higher permeability, potentially higher well yields and fewer wells required
- Lower potential for well interference with other users
- Better water quality.

The Lomas aquifer represents a viable back-up water supply source, should this be required.

18.5.2.3 Wellfield design

Based on the staged water supply requirements, two stages of well installation are proposed:

- Stage 1 (Year -1) – 3 wells
- Stage 2 (Year 2) – 5 wells.

The schedule assumes conversion of the existing test well MPA-2 in Jahuay into a production well. An average well yield of 25 L/s is used as the basis for design, and provides for one back-up well (Stage I) and two back-up wells (Stage 2). Approximate drilling locations and depths are presented in Table 18.9 and Figure 18.19.

However, prior to drilling, execution of additional surficial resistivity surveys on the east bank of the Jahuay valley is recommended to investigate the potential water resources on that side of the valley. This would reduce the need for pipeline crossings of the valley, and provide additional information for characterising the aquifer. The drilling locations should be revised at this point.

Table 18.9
Existing and Tentative Staged Drilling Locations and Depths

Well	UTM Easting (m)	UTM Northing (m)	Ground Elevation (masl)	Drilling Depth (m)	Reference
Existing					
MPA-02	512224	8319850	500	-	-
Stage 1					
MPA09-3	512094	8319207	498	184	midpt MC2 and MC3
MPA09-4	511843	8318517	505	222	MC4
MPA09-5	511088	8316661	485	200	MC8
Stage 2					
MPA13-6	511543	8317646	500	210	MC6
MPA13-7	510418	8313175	487	180	Piezo D
MPA13-8	512733	8319220	515	190	MC30
MPA13-9	512094	8319207	514	154	Piezo C
MPA13-10	511653	8318098	505	194	-
Total				**1534**	

18.5.2.4 Well design

GWI believes a test-production drilling approach may be applied to the wellfield installation program. Test-production drilling involves drilling from the outset at a diameter that permits installation of a final production well. Wells drilled using this approach should be installed with an air-rotary rig which can advance well casing, or by a cable-tool rig. The alternative is an exploration drilling approach, which involves drilling of pilot (5½") holes with an air rotary rig and subsequent drilling of large diameter holes at the best locations, using air rotary, cable tool or mud rotary drilling method.

18.5.2.5 Recommendations

With respect to the development of a water supply wellfield for the Marcobre Project in the Upper Jahuay Valley, GMI recommends the following:

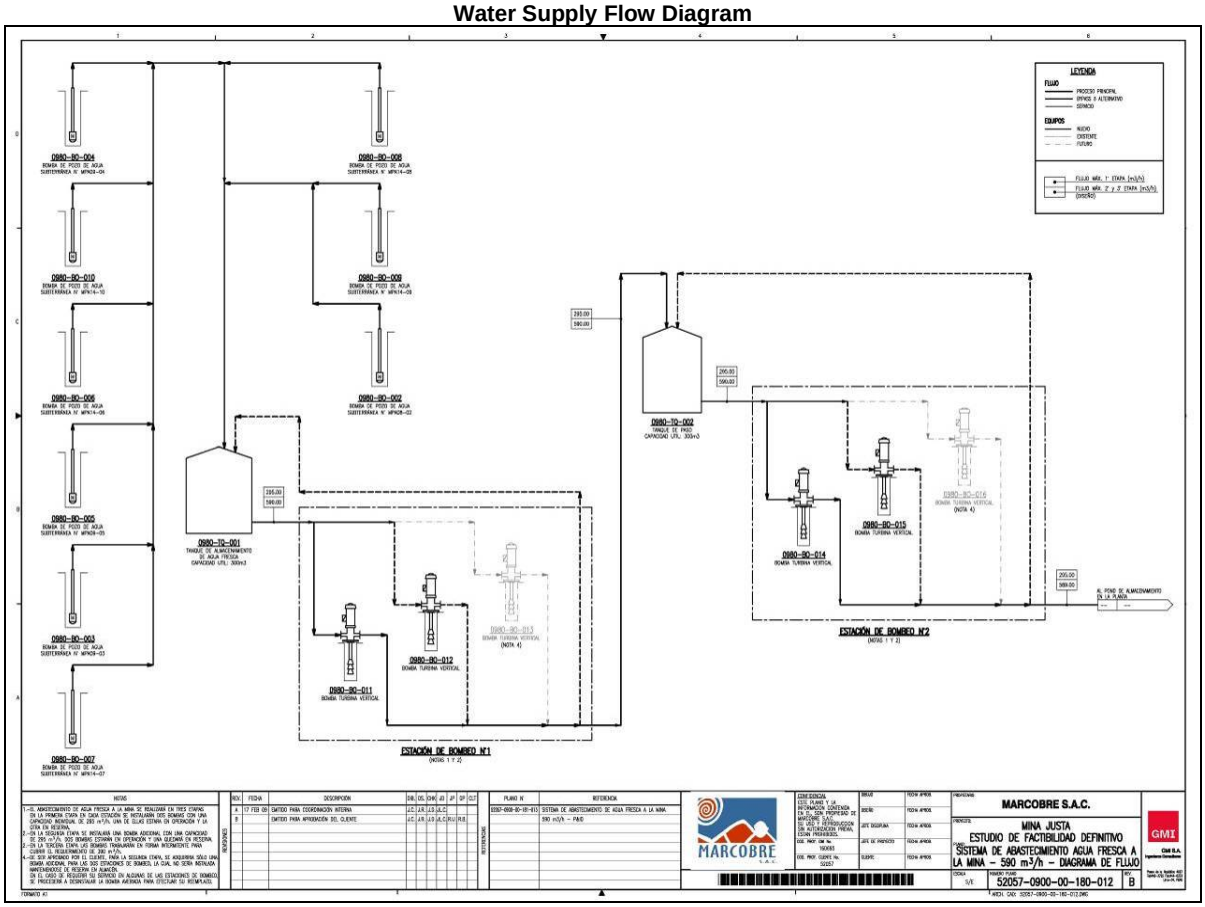
- Make application to Autoridad Local de Agua (ALA) for further ground water exploration and resource development in the Upper Jahuay area in order to confirm the availability of the water resource for the project.
- Undertake further resistivity surveys on the west bank of Quebrada Jahuay to refine the drilling locations, particularly in readily accessible areas.
- For long-term aquifer management, evaluate aquifer recharge processes in the Quebrada Jahuay. Install equipment to measure short-term flows within upper Quebrada Jahuay north of the proposed wellfield in order to better evaluate aquifer recharge through this means. Install pressure transducers/ dataloggers in existing wells close to the quebrada to monitor level responses to recharge events.
- Develop all wells to a sand-free condition and undertake test pumping to evaluate optimum well yield, pump setting, potential for well interference and water quality.

18.5.3 Water supply system

The water supply system considered for Mina Justa will take the water from Jahuay aquifer 31 km to the mine site. The water supply system includes the following:

- Borefield / water collection system, which consists of the water wells and pumps located in the Jahuay aquifer.
- Water transfer system, which consists of the pump stations and transfer pipelines, from the reception tank in the Jahuay aquifer to the pond at the mine site.
- Electrical distribution system, consisting of 22.9 kV power distribution line to the borefield and distribution to each pump station.
- Controls and instrumentation at each pump station and at the borefield location tied back via optical fibre. Each sector has a switch that connects the corresponding area with the control room.

Figure 18.19
Water Supply Flow Diagram



18.5.3.1 Water consumption

The water demand was defined from the site-wide water balance and will vary during the life of mine (refer to Section 18.5.1).

During the construction period, the water will be supplied from Nazca by truck; a well could be enabled with pump and discharge system to supply water during this stage.

18.5.3.2 Borefield / water collection system

Wells

Nine water wells will be located in the Jahuay aquifer, four in Stage 1, and five more in Stage 2, installed to produce at an average rate of 25 L/s (90 m^3/h) each.

Water wells will be activated as the water requirement for the mine increases. At the maximum requirement, all wells shall be operative.

To cover the maximum requirement during Stage 1, three well pumps will be operating with one on stand-by. During Stage 2, seven well pumps will be operating with two on stand-by. During Stage 3,

when oxide production has ceased, all pumps will remain in commission, but operating hours will be adjusted to meet the reduced demand.

Water transfer system

Based on a hydrogeology study, the pump intake is located at 150 m depth, which allows for lowering of the water table level during operations.

Pumps shall be turbine type with submersible motors/drives.

Due to their location, environment characteristics and operating conditions, the pumps have been selected as follows:

- One pump with estimated 90 m^3/h, 224 m TDH, 150 HP
- One pump with estimated 90 m^3/h, 143 m TDH, 100 HP
- Four pumps with estimated 90 m^3/h, 191 m TDH, 125 HP
- Three pumps with estimated 90 m^3/h, 121 m TDH, 75 HP.

Piping system

Water shall be transferred from the wells using one carbon steel pipeline. The pipeline runs from south to north to the collection tank located adjacent to Pump Station No. 1.

Collection tank

The collection tank receives water from the wells. It has a capacity of 300 m^3, and measures 7.60 m diameter by 8.00 m height.

18.5.3.3 Water transfer system

Pump stations

Two pump stations are located along the pipeline. Pump Station No. 1 is located at the wellfield collection tank, 31 km from the plant site, and Pump Station No. 2 is 13 km from the plant site.

Each pump station includes a collection/transfer tank and vertical turbine pumps of 295 m^3/h capacity and 326 m TDH. Motor power is estimated at 300 HP per pump.

During the initial phase of operation two pumps will be installed in the station, one operating and other stand-by. In the second phase, a third pump is installed, resulting in one pump used as a stand-by and the other two in operation.

The pump stations are permanent buildings made with masonry walls and roof of structural steel cover with metal roofing. Axial wall fan type air extractors are included to provide ventilation, especially in summer.

Transfer pipeline

The transfer pipeline has two sections. The first section (between the two transfer stations) is approximately 18 km long, while the second section (from the second pump station to the mine) is approximately 13 km long.

The system includes a 14" line for the first section and a 12" line for the second section. The pipeline is constructed of carbon steel changing to HDPE in sections further away from the pumping station where pressure decreases. Pipes are protected with venting/vacuum breaker valves.

The steel pipes have concrete sleepers installed every 6 m and anchor blocks whenever the direction changes. HDPE pipes will be restrained with earth anchors.

Fire Protection system

BC type carbon dioxide fire extinguishers are installed at both pump stations. The signal of the detection systems (smoke and/or temperature detectors, manual stations, signs and alarm devices) are transferred to local detection and alarm control panels, which will form a security net that concentrates all signals in the central control room.

Fresh water storage

The water is discharged into the 16000 m^3 raw water storage pond located at the plant site, from where will be distributed to internal facilities.

18.5.3.4 Electrical system

Electrical power supply to the water system is by means of the 22.9 kV transmission line from the Mina Justa substation.

From the transmission line, a secondary line is installed to each well using a 315 kVA transformer of 22.9/0.48 kV. From each transformer, 480 V is fed to the pump motor starter. Each motor control centre that feeds each borefield pump motor control is supplied from the overhead line via a pole-mounted on-load isolator that supplies a pad-mounted transformer.

At each pump station, a transformer of 1 500 kVA, 22.9/4.16 kV feeds the 4.16 kV motor control centre (MCC) that supplies the pumps. The substations that feed each pumping station are conventional type unitary substations, identical to those feeding the MV MCC located in the electrical room. DOL starters are included.

Auxiliary power for the control system, lighting, small power, etc. is supplied via an auxiliary transformer.

18.5.3.5 Control & instrumentation system – water collection & transfer system

Instrumentation at each borefield water collection and transfer station is installed to provide a safe control system for the pumps, and also provide adequate information for remote control/operation of the water collection and transfer system. Typically this includes instrumentation such as:

- Flowmeters
- Low flow switches on pump headers

- Alarms to warn remote operators when pumps have tripped out
- Optical fibre communication system relaying information from each borefield/water collection pump substation to the Mina Justa central control room (CCR)
- Optical fibre communication system relaying information from each water transfer substation to the Mina Justa CCR.

18.6 POWER SUPPLY SYSTEM

18.6.1 Power supply and distribution

18.6.1.1 Overview

Marcobre will enter into a long term power supply agreement with a Generator that will deliver power to the distribution grid operated by Red de Energía del Perú (REP).

A dedicated 15 km 220 kV overhead power line supported by steel towers will connect to the grid at REP's Marcona substation 220 kV bus. The 220 kV overhead power line terminates at the plant's HV switchyard on the 220 kV bus.

A 22.9 kV power line runs between site, the Jahuay borefield and an intermediate pump station. The line is supported by wooden poles.

18.6.1.2 HV switchyard

The plant site is located approximately 15 km from the existing Marcona 220 kV substation, which is connected into the regional 220 kV network. An outdoor switchyard is proposed to accommodate the incoming 220 kV overhead lines and a 220/22.9 kV transformer to supply the plant and associated HV switchgear feeding a 22.9 kV switchboard located indoors at the main plant substation.

To satisfy the power demand requirements for Mina Justa, it is necessary to upgrade the National Network System to Marcona. Discussions are underway to resolve this

18.6.1.3 22.9 kV main switchboard

The 22.9 kV main switchboard is provided with a single incomer bay. The 22.9 kV main switchboard is provided with gas-insulated switchgear bays for distribution of 22.9 kV to plant load centres, power factor correction and HV motors.

18.6.1.4 Distribution

Power is distributed from the 22.9 kV main substation switchboard to major plant loads via an overhead line to the boundary of the process plants. Within the process plants, power cables are used. Plant load centres have varying secondary voltages supplied by step-down power transformers adjacent to the each of the load centres.

18.6.2 Power supply

18.6.2.1 Oxide plant electrical load

The total connected load for the Oxide plant and mine infrastructure is 36 394 kW. The application of relevant utilisation factors results in a total running load of 30 124 kW for the Oxide process plant and mine infrastructure only. The overall uncorrected power factor for the total plant load is 0.83 lagging and results in a predicted maximum demand of 36 448 kVA. The inclusion of harmonic filtering equipment (with a total reactive power of 6 MVAr) allows for the correction of the power load to approximately 0.9 lagging and a predicted maximum demand of 33 441 kVA.

18.6.2.2 Sulphide plant electrical load

The Sulphide plant total connected load is 29 061 . The application of relevant utilisation factors results in a total running load of 22 861 kW for the sulphide process plant, tailing and associated process plant infrastructure only. The overall uncorrected power factor for the total plant load is 0.85 lagging and results in a predicted maximum demand of 26 793 kVA. The inclusion of harmonic filtering equipment (with a total reactive power of 6 MVAr) allows for the correction of the power load to approximately 0.94 lagging and a predicted maximum demand of 24 211 kVA.

18.6.2.3 Other electrical loads

In addition to process plant loads, other loads include the following:

- Camp 824 kVA
- Lighting for internal access road 190 kVA
- Wellfield substation 695 kVA
- Transfer pumping station 842 kVA.

It should be noted that the above loads are fed from the plant site via 22.9 kV overhead power line.

18.6.2.4 Power reticulation

Twelve substations service the Oxide crushing, screening and process plant facilities. The Sulphide plant requires an additional seven substations.

Power factor (p.f.) correction is provided at the main substation; 6 MVAr of correction is provided. This results in overall power factors (p.f.) and maximum demands as follows:

- Oxide only – p.f. 0.9, maximum demand (with p.f. corrected) 33 441 kVA.
- Sulphide only – p.f. 0.94, maximum demand (with p.f. corrected) 24 211 kVA.
- Combined Oxide and Sulphide – p.f. 0.88, maximum demand (with p.f. corrected) 60 160 kVA.

18.6.2.5 HV switchboards

The Oxide plant HV switchboards are located within the Oxide crushing and screening area HV substation and the Oxide process area HV substation.

For the Sulphide plant, power is distributed from the 22.9 kV main substation switchboard to major plant loads via an overhead line to the boundary of the Sulphide plant. The Sulphide plant HV switchboard is located within the Sulphide plant HV substation. It handles the Sulphide primary crusher as well.

18.6.2.6 Emergency generation

Emergency power is required to maintain the plant process in a safe state, allowing safety systems to function seamlessly and prevent long delays in restarting the plant after the restoration of power to the site. Catastrophic failure of the power system is not considered.

The emergency power is provided to a number of drives that typically require back-up emergency power for short term (4 hours or less) power outages. Individual small generators for this purpose are located at or near the required substation.

The following emergency power requirement has been identified:

- **Oxide Plant**: total 2321 kW consisting of 2 027 kW fixed loads (building loads, plant lighting and small power) and 294 kW process loads. The key process load is the trickle current to the electrowinning cells. This maintains cell polarity and prevents dissolution of the plated copper and depassivation of the lead anode surfaces.
- **Sulphide Plant**: total 922 kW consisting of 690 kW fixed loads (building loads, plant lighting and small power) and 232 kW process loads.

18.6.3 Control system

The plant is provided with a process control system (PCS) to manage the activities of the processing plant. The plant is of moderate level of control complexity.

The plant is designed to be operated primarily from the CCR located adjacent to the electrowinning building. The CCR contains four operating stations and an engineering workstation. Local field operator stations provide complete control room type information to the operators, but allow interaction from the field operators on a secured basis.

A CCR is provided for the Sulphide plant and is located near the grinding building.

18.7 WASTE DISPOSAL

18.7.1 Mine and ripios waste dumps

A design has been prepared for the two mine waste rock dumps, the ripios dump and the low grade stockpile, taking account of the physical and geochemical stability of the structures and the appropriate land use following closure.

The estimated amount of waste rock to be generated by the Project is approximately 402.5 Mt, of which 383 Mt will be placed in the main waste dump, 14 Mt in the Magnetite Manto waste dump and the remaining 5.5 Mt (non-PAG material) will be used for construction of the tailings dam.

The ripios dump has been designed with a capacity of approximately 114 Mt, and the low grade stockpile has a capacity of 20 Mt.

The location of the structures is shown in Figure 18.20, and includes:

- Main mine rock waste dump, located to the east of the concentrator area at an average elevation of 710 masl
- Ripios dump, located within the main waste dump
- Magnetite Manto waste dump, located in the southern sector of the project site, 200 m east of the Magnetite Manto open pit
- Low grade stockpile, located 300 m to the north of the Magnetite Manto waste dump.

In order to generate information for the design of these structures, a site investigation program was conducted by Knight Piésold between June 2006 and December 2007. The investigation included characterisation of the foundations, waste rock material and ripios material.

The foundation of the waste dumps and stockpile comprises three main geological units: Quaternary material (aeolian sands), overlying highly weathered and fractured bedrock (Tunga Andesites and volcano-sedimentary material of the Rio Grande Formation), which, in turn, overlies fresh, fractured bedrock with medium to high strength (Tunga Andesites and Rio Grande Formation). All three geological units are adequate for foundations. However, the aeolian material located at the toe of the final mine waste dump will be removed to improve stability.

The water level encountered during the site investigation is approximately at elevation 312 masl, or some 400 m below the main waste dump and 500 m below the Magnetite Manto waste dump and low grade stockpile.

The strength parameters for the rock waste were estimated using correlations based on the expected percentage of fines. The strength parameters for the ripios material were determined from standard laboratory testing as the ripios material comprises silty sands and fine gravels with a maximum size of 8 mm.

18.7.1.1 Ripios and mine waste dump design

As mentioned previously, the design of the mine waste dump takes account of placement of the ripios material within its boundary. The mine waste dump acts as a dyke for the ripios material in the north area; this dyke will reach an elevation of 820 masl and will surround the ripios material. The ripios material will reach a final elevation of 832 masl.

The intermediate slopes (bench slopes) for the mine waste dump have been designed to 1.4H:1V. The overall slope in the North sector will be 1.7H:1V with an intermediate berm of 40 m width; the overall slope in the South area will be 2.15H:1V and will reach a maximum elevation of 780 masl. The upstream waste dump slope in the North sector in contact with the ripios material, will have an overall

slope of 1.4H:1V with no intermediate berms. Finally, the ripios material will have an operational slope of 1.4H:1V up to the contact with the mine waste dump dyke.

The disposal of the ripios material will be managed in order to reduce the moisture content before a new layer is placed over previous placed material. This will reduce the potential for seepage down to bedrock.

The results of stability analyses for the mine waste dump indicate that the static factor of safety (FoS) during the construction and post-construction periods are over 1.3 and 1.5, respectively. From pseudo-static analysis, the overall slope FoS is over 1.0 for both operation and post-closure. Stability analyses for the ripios dump during operation indicated that a security distance of approximately 15 to 25 m from the edge of the ripios waste slopes is necessary during operation as a result of some minor (superficial) slope failures that are expected during the construction of this stage. It is recommended that this distance be respected, in order to avoid potential damage to mobile equipment in the area.

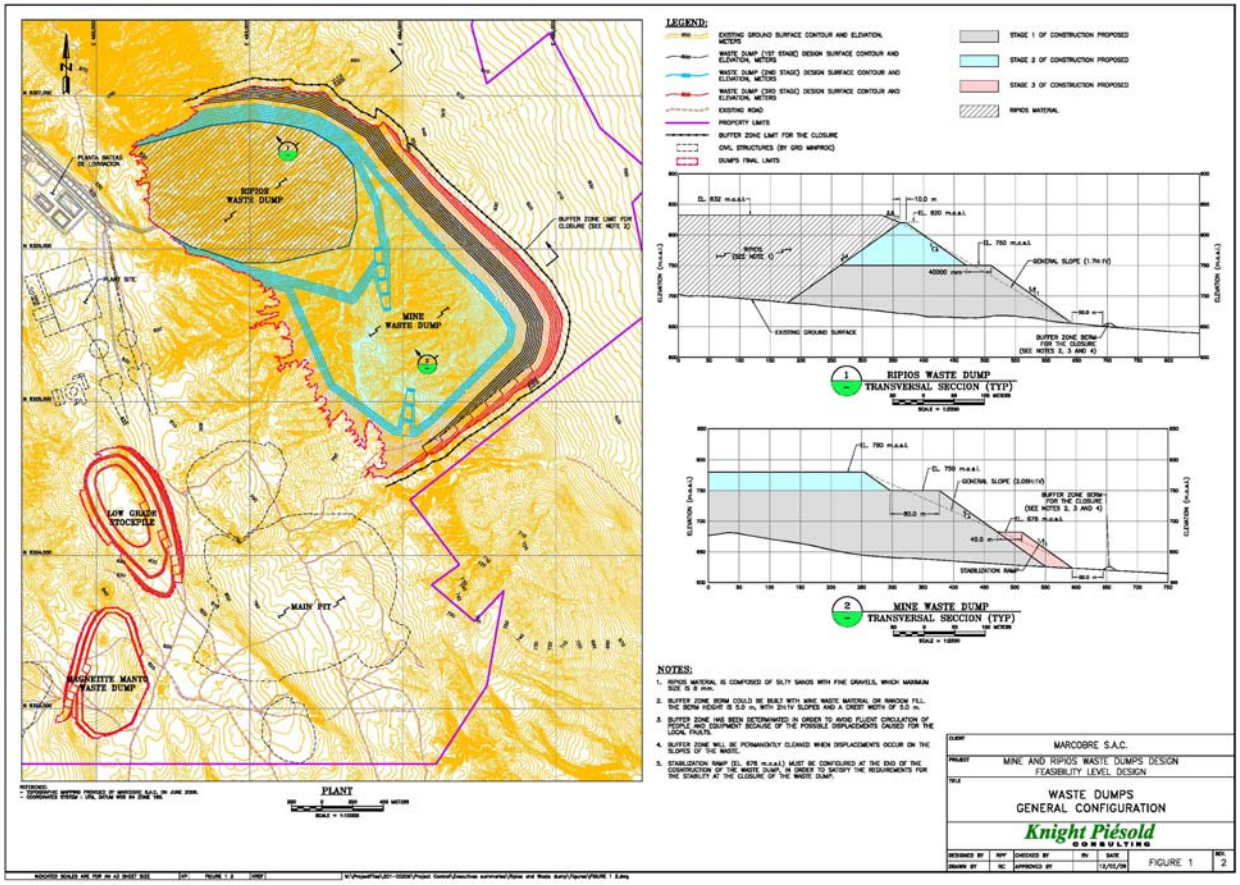
As part of the design of the main waste dump, seepage analyses were undertaken. The results indicate that the saturation degree of the soil increases 5% in the first 5 to 10 m starting from the surface as consequence of a storm event (PMP). No variation on the saturation degrees is observed below those levels. Consequently, the potential to generate seepage through the dumps is limited, and that the potential of flow, if there are, to the groundwater table is low.

In order to monitor slope movements on the mine waste dump, marker points will be installed during and after operations.

Initially, frequent monitoring of the ripios slopes will be required during the start-up of operations, in order to confirm the safety zone for the mobile equipment. To monitor seepage from the ripios area, underdrain systems will be installed at the base of the ripios dump. Potential flow will be conducted to a water monitoring station.

PAG mine waste rock is estimated at approximately 15 Mt. The waste dump design specifies that the PAG mine waste be encapsulated by non-PAG material in the main waste dump, separated from the ripios material and from the final slopes of the waste dump. At closure, any exposed PAG waste rock will be covered with a 1 m layer of non-PAG material so as to avoid potential acid dust generation and dermal contact. When the ultimate dump configuration has been reached, a security berm will be constructed at about 50 m from the final toe of the mine waste dump as a buffer zone.

Figure 18.20
Ripios and Mine Waste Dumps



18.7.1.2 Magnetite Manto waste dump and low grade stockpile design

The Magnetite Manto waste dump will cover 34 ha, and will be constructed in three layers.

The low grade stockpile will extend over 41 ha, and will also be constructed in three layers.

For both structures the bench slopes and the overall slopes will be 1.4H:1V and 2.5H:1V, respectively. Stability analyses show that the FoS at the end of construction is greater than 1.5 and 1.0 for static and pseudo-static conditions, respectively.

18.7.2 Tailings storage facility (TSF)

The life of mine for the Mina Justa Project has been estimated at about 12 years. For the first 1.5 years, Oxide ore will be mined and processed using a vat leaching system. The second stage includes processing of Sulphide ore by flotation in two stages, namely the rougher and cleaner flotation stages.

The TSF is designed to DFS level with an approximate capacity of 49 Mt of dry tailings over a period of 10 years; the tailings delivery systems have been designed to a PFS level. The TSF is located in an area called “Quebrada Justa” that lies to the west of the plant site and to the northwest of Magnetite Manto open pit, and covers a surface area of about 372 ha (Figure 18.21).

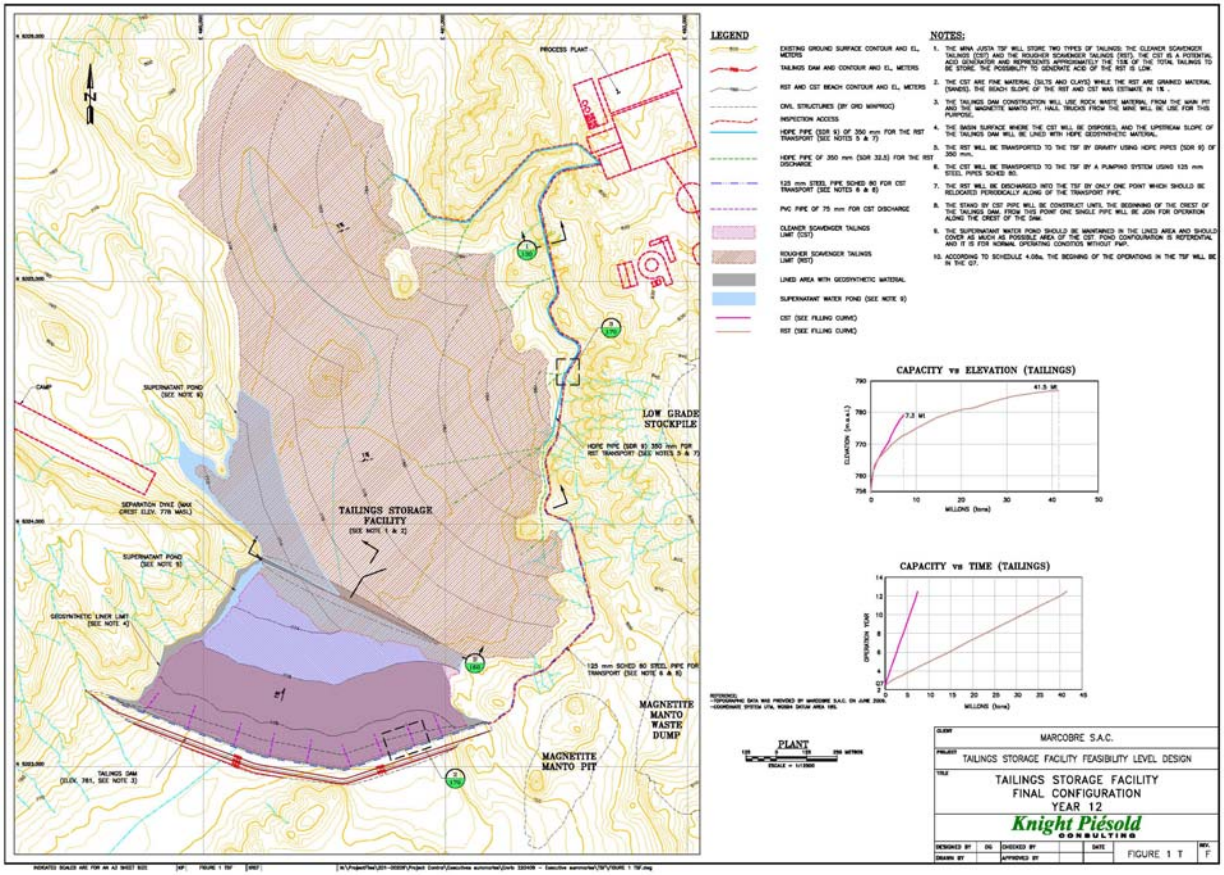
The design has been developed by Knight Piésold, based on observations made during site reconnaissance, results obtained from geotechnical investigations conducted by Knight Piésold between July 2006 and December 2007, interpretation of the sub-surface conditions, and the physical and geochemical characterisation of the tailings and mine rock waste. Additionally, the design takes into account the mining schedule for the Main (Mina Justa) and Magnetite Manto open pits.

The geotechnical investigation included logging and sampling of drillholes and test pits. Permeability tests were conducted in the drill holes and test pits, and piezometers were installed in selected drill holes. From the hydrogeological study developed by Vector, groundwater was encountered at an approximate depth of 450 m in Quebrada Justa.

Geochemical characterisation of tailings and waste material from the Mina Justa and Magnetite Manto open pits were reported in a study by Knight Piésold in April 2008. The waste production schedule for dam construction and tailing deposition was provided by GRD Minproc.

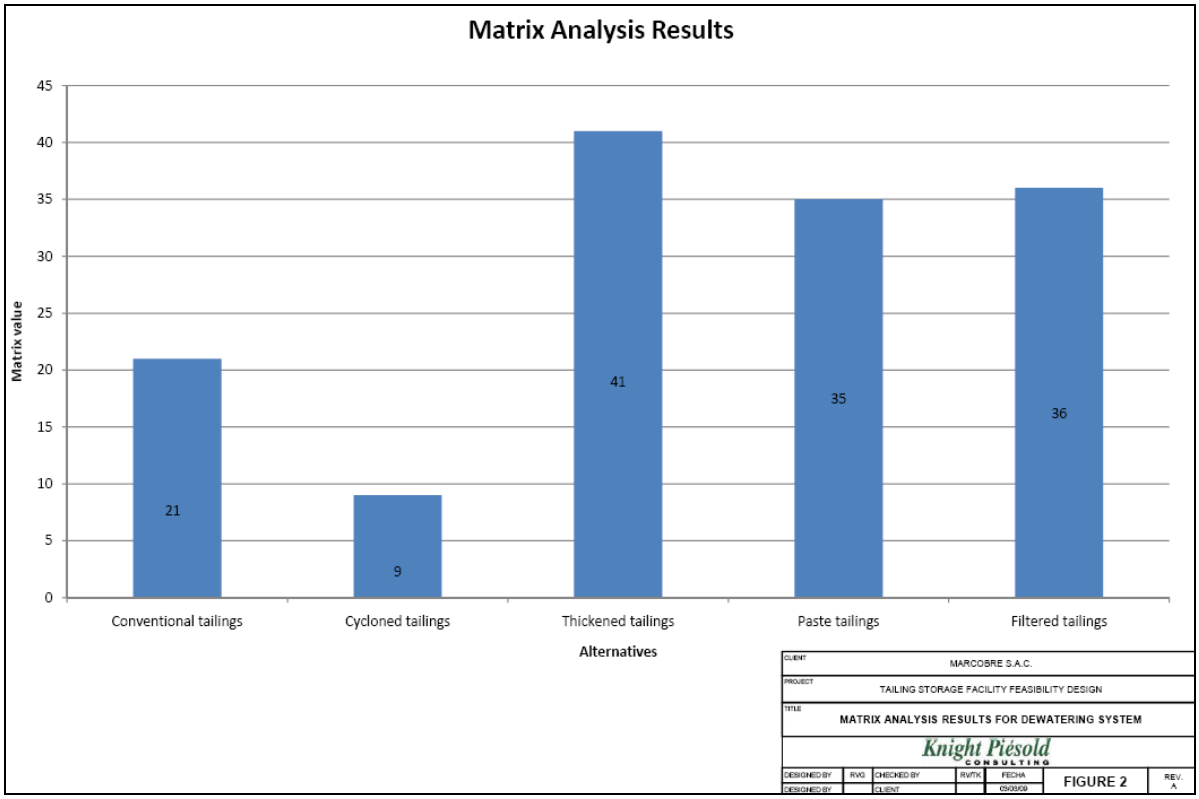
The location of the TSF (Figure 18.21) was selected based on an alternatives analysis study in which environmental, economic and technical aspects were considered. Seven options were evaluated and the preferred option was selected based on economic and technical advantages, as well as its location within the Mina Justa property limits.

Figure 18.21
Tailings Storage Facility



As part of TSF feasibility design, a preliminary study was conducted to evaluate the costs and benefits of including a tailings dewatering system in the design. The results indicated that a thickened tailings system should be incorporated in the design to recover water from the tailings (Figure 18.22).

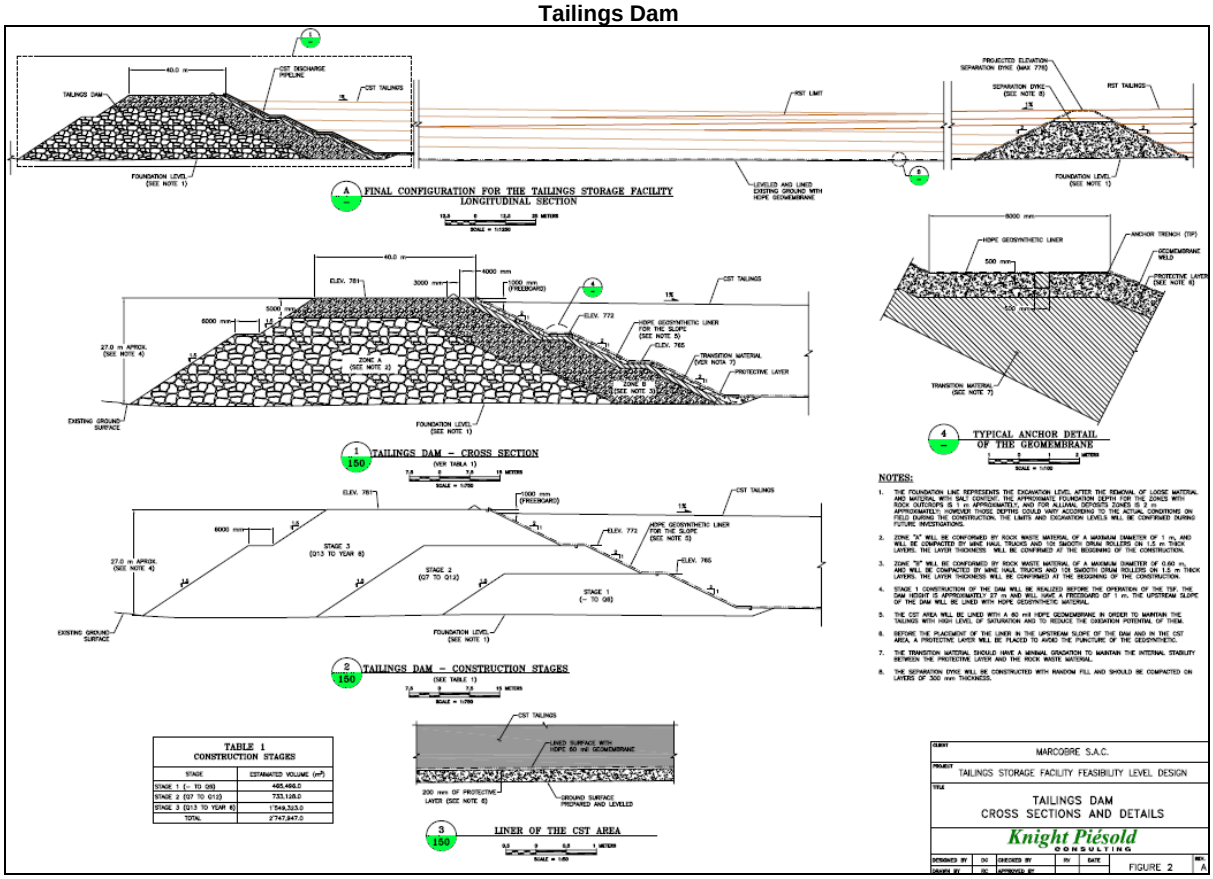
Figure 18.22
Tailings Dewatering Matrix



The Mina Justa TSF is designed to store two types of tailings, the Cleaner Scavenger Tailings (CST, potentially acid generating) and the Rougher Scavenger Tailings (RST, non-acid generating).

The tailings dam is constructed of non-acid generating mine rock waste material from the Mina Justa and Magnetite Manto open pits, and has a length and height of 1.8 km and 27 m, respectively. The dam is constructed in three stages using the downstream construction method (Figure 18.23). It is designed to maintain its physical stability during construction, operation, and post-closure. Mine haul trucks are used to transport construction materials to the dam. Compaction is provided by the mine trucks and smooth drum rollers. A geosynthetic liner is included on the upstream slope of the dam and beneath a portion of the tailings basin.

Figure 18.23
Tailings Dam



The Mina Justa tailings dam is included in the "low" incremental hazard consequence category with respect to “Human Life Safety”; no deaths are anticipated in the area of study and it is noted that there is a limited presence of population within a 25 km radius of the dam. The dam is also included in the "low" consequence category for “Socioeconomic, Financial and Environmental” impacts; in a hypothetical failure case, no water sources or villages around the TSF will be affected. Therefore, the tailings dam was given a "low" consequence classification according to the Canadian Dam Association (CDA) guidelines.

A separation dike will be constructed between the CST and the RST into the basin of the TSF. The CST are discharged from the crest of the tailings dam, while the RST are discharged from the eastern edge of the TSF. Placement is such that the CST are kept between the tailings dam and the separation dike. In the lasts years of operation the RST are discharged in a direction and sequence such that the separation dike and part of the RST contain the CST within the lined portion of the basin.

The CST and RST are produced at a rate of about 4.9 Mt dry tailings per year. CST are estimated to correspond to 15% of total production or 0.75 Mt/a of dry tailings, and RST correspond to 85% of the total production (4.2 Mt/a dry tailings).

The RST and CST are thickened at the plant to 60% solids content (by weight). The CST are delivered to the TSF via a 125 mm diameter steel pipe to the crest of the tailings dam, from where they are discharged in a south-north direction using spigots located on the upstream slope of the dam. The upstream slope and portion of the tailings basin in which the CST are placed are lined with a geosynthetic liner to keep the CST saturated without excessive water consumption, and to reduce the likelihood for seepage into the foundation.

The RST are delivered to the TSF via a 350 mm diameter HDPE pipe installed on an access road running parallel to the east edge of the TSF. The tailings are deposited from a small number of discharge points along the length of the delivery pipe. The tailings are discharged in northeast-southwest direction, and in a sequence such that the CST tailings are kept within the lined portion of the basin.

The TSF was designed using the sub-aerial deposition method with a beach slope of 1% for both the CST and RST. The concept for depositing the CST is to maintain the tailings beach wetted (i.e. with a high moisture content) to limit the potential for oxidation. The active portion of the tailings beach will be re-located regularly along the length of the dam so that fresh tailings layers are added to cover the previously placed tailings before the moisture content reduces significantly. The RST deposition concept is to develop stiff and well-drained beaches, and rotation of the discharge points will be carried out after previously placed tailings have had the opportunity to drain.

Based on the TSF water balance calculations, the supernatant water pond is not expected to cover the entire CST beach. However, management of the supernatant water pond will provide for covering as much of the CST as possible. A reclaim water system from the supernatant water pond to the plant site is not considered to be practical; it would be possible to reclaim water only for shorts periods of time during the facility life.
.

Seepage analyses were carried out to evaluate the potential for seepage from the CST area into the foundations. It is important to mention that the tailings considered potentially acid generating are usually those CST exposed to the atmosphere, not submerged under the supernatant water pond. The results of the analyses indicate that seepage from the active surface is limited; more seepage occurs through the non-acid generating RST, partly because the RST are expected to have a higher hydraulic conductivity than the CST.

In order to monitor seepage into the tailing dam, open standpipe piezometers will be instaled and potential movements will be monitored using topographic prisms. In order to monitor the water quality in the foundation below of the tailings beach, open standpipe piezometers will be placed in the separation dyke crest.

To reduce initial capital costs, the TSF is planned to be built in three stages, involving raising the tailing dam and progressively installing the liner system. The construction stages are:

- Stage 1: construction of the tailings dam to elevation 765 masl. Placement of the transition material and geosynthetic liner on the upstream slope of the dam and in the basin under the portion corresponding to CST deposition. Installation of the CST and RST delivery systems.

- Stage 2: raise the tailings dam to elevation 772 masl. Placement of the transition material and geosynthetic liner on the upstream slope of the dam, and in the basin under the portion corresponding to CST deposition. Install the second stage of the RST transport system.
- Stage 3: raise the tailings dam to elevation 781 masl. Placement of the transition material and geosynthetic liner on the upstream slope of the dam and in the basin under the portion corresponding to CST deposition. . Install the second stage of the RST transport system.

The closure plan includes removal of the tailings delivery and deposition system, followed by covering of the CST using RST in order to limit the potential for oxidation and to reduce the potential of inhalation or dermal contact with acidic tailings, If these conditions were to develop, a type of cover may be necessary, using local surface material, in order to prevent aeolian transport of tailings.

18.8 PORT AND TRANSPORT

18.8.1 Port facilities

18.8.1.1 Overview

The Government of Peru has announced plans for an international public auction of a port concession at San Juan de Marcona. Under this plan, the winning bidder would construct a new port, initially to accommodate the shipment of iron ore and base metal concentrates, and later other cargos. For the purpose of the DFS, Marcobre has assumed that the Government's proposed schedule for the start-up of the port by 2014 will be delayed by one year, following which the Mina Justa project will use San Juan de Marcona, initially for concentrates, and later for sulphuric acid supply and cathode shipment. In the meantime, the ports of San Martin and Matarani will be used.

Sandwell (2009) completed a port evaluation study to identify costs and availability of port options, following which Marcobre determined a multi-port strategy as follows:

- San Martin, 250 km by road north of Mina Justa, is selected for cathode and acid shipments for the first five years.
- Matarani, 550 km by road to the south, is used for shipment of concentrate for one year
- San Juan de Marcona, 30 km to the southeast, is selected for cathode, acid and concentrate supply and shipments for the reminder of the Project.

18.8.1.2 San Martin (Terminal Portuario General San Martin)

The port of San Martin (Terminal Portuario General San Martin - TPGSM) will be used from the start of production for five years for the import of acid and shipment of cathodes.

TPGSM is located on the northeast end of the Paracas Peninsula, some 300 km by road and 132 nautical miles to the south of Callao, the main Peruvian port. This port is located approximately 250 km by road to the north of Mina Justa. Road access is along the two-lane Panamericana Sur highway. TPGSM suffered damage from an earthquake in August 2007 as the port was being prepared for privatization.

TPGSM handled 997 740 t of cargo in 2007, down from the peak of 1 142 707 t it handled in 2006. The reason for the drop was the earthquake which closed operations while the damage was assessed and emergency repairs took place. The predominant cargo is bulk solids which accounted for 78% of the cargo in 2006 and 69% of the cargo in 2007.

The port has a 700 m long wharf divided into four berths. Ships can be moored port or starboard side depending on cargo or ship particulars. Overall berth utilization in 2006 was approximately 20%, corresponding to 292 berth-days per year total for the four berths combined, indicating that there capacity for more traffic.

If the privatization scheme goes ahead as envisaged, the two northernmost berths will be combined into a large 350 m long berth for container ships. This will decrease the number of berths available for other cargo to two or three (if the container berth is shared with other cargo) and increase pro-rata the berth utilization.

Overall, the space available at TPGSM would meet Marcobre's requirements for cargo handling even during the peak production years, the exception being the tank farm for liquid bulk storage that would have to be expanded.

According to the tender basis document for TPGSM, the winning tenderer had to make an investment in the order of US$80.4 M. This estimate is no longer valid considering the extensive damage to the port caused by the earthquake of August 2007. It is not definite yet, but it seems that the government plans are to transfer the insurance claim and the responsibility for executing the necessary repairs to the selected tenderer.

18.8.1.3 Matarani

The port of Matarani will be used at the start of concentrate production for the shipment of concentrates for one year, in order to accommodate a delay in the start-up of the port of San Juan de Marcona. The port is located approximately 550 km by road to the south of Mina Justa, road access being along the two-lane Pan Americana Sur highway.

The port of Matarani is operated by Terminal Internacional del Sur (TISUR), a private company that belongs to the Romero Group, a large Peruvian enterprise with operations in port facilities, logistics, finance and the agribusiness. Currently TISUR handles bulk copper concentrate and copper cathodes from various mines. Other cargo handled at the terminal includes sulphuric acid, soya and soya meal, liquid bulks and containers.

Copper concentrate is received by rail or truck. Rail shipments arrive in a special designed cylindrical container of 15 t net capacity. Each rail car is capable of carrying four containers. Rail cars are positioned two at time inside the unloading station, and the containers are unloaded individually on a hopper with the use of a bridge crane. Trucks are end-dumped into a hopper. Unloading rates are 350 t/h for rail shipments and 240 t/h for truck shipments.

Two storage buildings are available at the terminal. Stacking in both buildings is made by a travelling stacker mounted against one side of the building. Reclaiming of the product is done by FEL and a fixed hopper system. A pipe conveyor connects the storage buildings to the ship loading system.

The pipe conveyor feeds a travelling ship loader equipped with a tripper and a boom with a telescopic chute. All the conveyors are enclosed to reduce dust emissions. The conveyor and ship loader capacity is 1 500 t/h. Pier length is 582 m, and the port is capable of handling ships with a LOA up to 210 m and beam up to 35 m.

TISUR has 160 ha of land area available for new developments; expansion plans include the option of two new berths inside the existing harbor and, in a later phase, the construction of additional berths outside the harbor through additional land filling.

The highway connecting Brazil and Bolivia to the Pacific Ocean is under construction and one route will reach the port of Matarani. The impact on port traffic is uncertain. The existing port infrastructure and shipping facilities could handle concentrates from Mina Justa and it is likely that no additional investment in bulk copper concentrate handling facilities is required.

Marcobre could share the existing truck receiving hopper and 50 000 t capacity shed with other customers, and negotiate storage and handling charge with TISUR. The copper concentrate from Mina Justa could be loaded through the existing ship loader and berth.

18.8.1.4 San Juan de Marcona

The bay of San Juan de Marcona provides excellent marine conditions for a deep sea port which could accommodate Capesize vessels at approximately 1 000 m from shore.

There is an old pier, which is approximately 500 m long. This pier was originally constructed for Minera Acarí, transferred to Hierro Perú, and is now under the control of the Peruvian Navy. However, it is beyond repair.

One branch of the highway connecting Brazil to the Pacific Ocean will extend to San Juan de Marcona. A Supreme Decree was issued by the Government of Peru in December 2008, stating that the port will be privatised. It is expected that tenders for the port concession will be called for in late 2009.

The national port authority of Peru (Autoridad Nacional Portuario - APN) has completed preliminary studies for the development of a port at San Juan de Marcona. The APN's plan is to develop a "Mega Port" in up to six phases. At this time, the details of expected handling capacity for each phase are still somewhat uncertain. The initial phase of port development is to be focussed on the handling and shipment of iron ore, copper and possibly other bulk minerals. Subsequent phases would see the development of facilities to handle containers, liquid bulks and other materials.

The port is expected to handle up to 10 Mt/a of minerals starting in 2013 (and rising to 22 Mt/a by 2032), the majority of these shipments being iron ore.

At the APEC convention on 20-22 November 2008, a consortium between Minera Mapsa (a Peruvian company) and South Korea-based Emirates Malaysia Korea Consortium (EMKC), announced a project to build a 560 km long railroad from Cuzco to the Pacific Ocean, and a major port at the terminus in San Juan de Marcona.

Although the privatisation of a port at San Juan is faced with several challenges, there are a number of mining projects such as Mina Justa that are interested in the use of a port such that the privatisation process will need to keep pace with their requirements, or these projects will look for alternatives.

18.8.1.5 Port selection for Mina Justa Project

Marcobre has assumed that the start of operation of the bulk mineral handling facilities at the new port of San Juan de Marcona will be further delayed for one year and will start in 2015, at which time concentrates from Mina Justa will be shipped out of this port. Marcobre has assumed that two years later, starting in 2017, cathodes and acid will also be handled by the new port of San Juan de Marcona in accordance with the proposed phased development of this facility.

A contingency plan has been developed in case these assumptions do not materialize. If San Juan de Marcona is not ready by 2015, then concentrates will continue to be shipped from the port of Matarani until there is a facility in San Juan de Marcona. If there are no cathode handling and/or acid handling facilities ready in San Juan de Marcona by 2017, then these can continue to be handled from San Martin until facilities are available at San Juan de Marcona.

The report prepared by Sandwell in February 2009 also evaluates other potential ports that can be used by Mina Justa, such as the nearby Petral site for acid.

18.8.2 Transport

18.8.2.1 Land

The Mina Justa Project is located approximately 500 km by road south of Lima, and is easily accessible. The highway south of Lima, referred to as the Panamericana Sur, passes through several urban centres, but primarily passes through rural farm land and desert. Initially the highway is 4 or 6 lines wide, reducing to two-way traffic after 115 km.

At Panamericana Sur highway marker 488 there is a junction to another highway that leads to the town of San Juan de Marcona. Ten kilometres along this highway there is an exit to the Mina Justa exploration camp, which lies another three kilometres from the exit. Total travel distance from Lima to the exploration site is 501 km, a journey taking approximately 6 to 7 hours.

Road distances to other urban centres from the project site are:

- Nazca - 50 km
- San Juan de Marcona – 30 km

Road distances to facilities that will or may be used by the Mina Justa Project are:

- San Juan de Marcona Port – 30 km
- San Martin Port – 250 km

- Acid Terminal at San Nicholas Bay (Petral) – 55 km
- Matarani port - 550 km.

Between Lima and the project site there are five toll stations on the Panamericana Sur. Between the project site and San Juan de Marcona there is 1 toll station.

18.8.2.2 Air

It is expected that some of the project employees will be transported between Lima and site by aircraft. Close to the project site there are two airports:

- Nazca has a small airport that is used by small aircraft. The runway is approximately 1 km long. Flying distance from Lima to Nazca is approximately 350 km.
- On the edge of San Juan de Marcona there is a small airport owned and operated by the Peruvian Navy for training purposes. It has a runway of approximately 2 km, but only half is in good condition. Flying distance from Lima is approximately 400 km.

Neither airport has lighting for night flights, and flying is restricted to between approximately 7 am and 4 pm. Neither airport is serviced by commercial carriers at this time.

18.9 PROJECT IMPLEMENTATION PLAN

The Mina Justa Project will be implemented in two stages. The first stage involves construction of equipment, plant and facilities to mine and process oxide ore and produce copper cathode. The second stage will commence while the oxide plant is still under construction, and involves construction of a concentrator plant and related facilities such as the tailings dam to process the sulphide ore.

18.9.1 Implementation strategy and schedule

A summary of the oxide plant project implementation schedule is provided in Figure 18.24, and the sulphide plant project implementation schedule is provided in Figure 18.25.

Figure 18.24 indicates a 29 month timeframe from the commencement of detailed engineering to completion of oxide plant construction, and an additional 3 months to complete the commissioning and commence cathode production. It is assumed that all necessary permitting and environmental approvals are obtained within the timeframe indicated.

Similarly Figure 18.25 indicates a 29 month timeframe for the sulphide plant implementation from commencement of engineering to completion of construction, and 3 months for commissioning.

The key drivers of the schedules are as follows:

- Very long delivery lead times for some critical equipment (e.g. crushers and mills).
- Large quantity of concrete works for the vat leaching area.
- Construction of camp accommodation.

The strategies employed to achieve the project completion dates include the following essential elements.

- **Early award of EPCM contract:** An EPCM (Engineering, Procurement and Construction Management) approach has been assumed for the implementation of the facilities and supporting infrastructure. Early award of the EPCM contract is necessary to progress the engineering and tender the critical long lead time equipment packages and contracts.
- **Early award of critical items:** The supply of cone crushers and construction of the camp are on the critical path. The schedule relies upon award of these critical items at the Project Release and Financial Approval milestones. For the sulphide plant, the mills are critical long lead equipment and it is assumed that these will be ordered 3 months before the sulphide project release milestone.
- **Maximum Pre-assembly:** To minimise construction time, where possible, tanks, platforms, MCCs and other equipment will be pre-assembled as much as practicable before being delivered to site.

  Also the installation schedules for critical path equipment such as cone crushers, cathode stripping machine, and mills have been compressed. It is planned that purchase orders for these items will provide for maximum pre-assembly and progressive deliveries to enable assembly and installation work to commence before their final delivery dates.

Figure 18.24
Project Implementation Schedule - Oxide Project

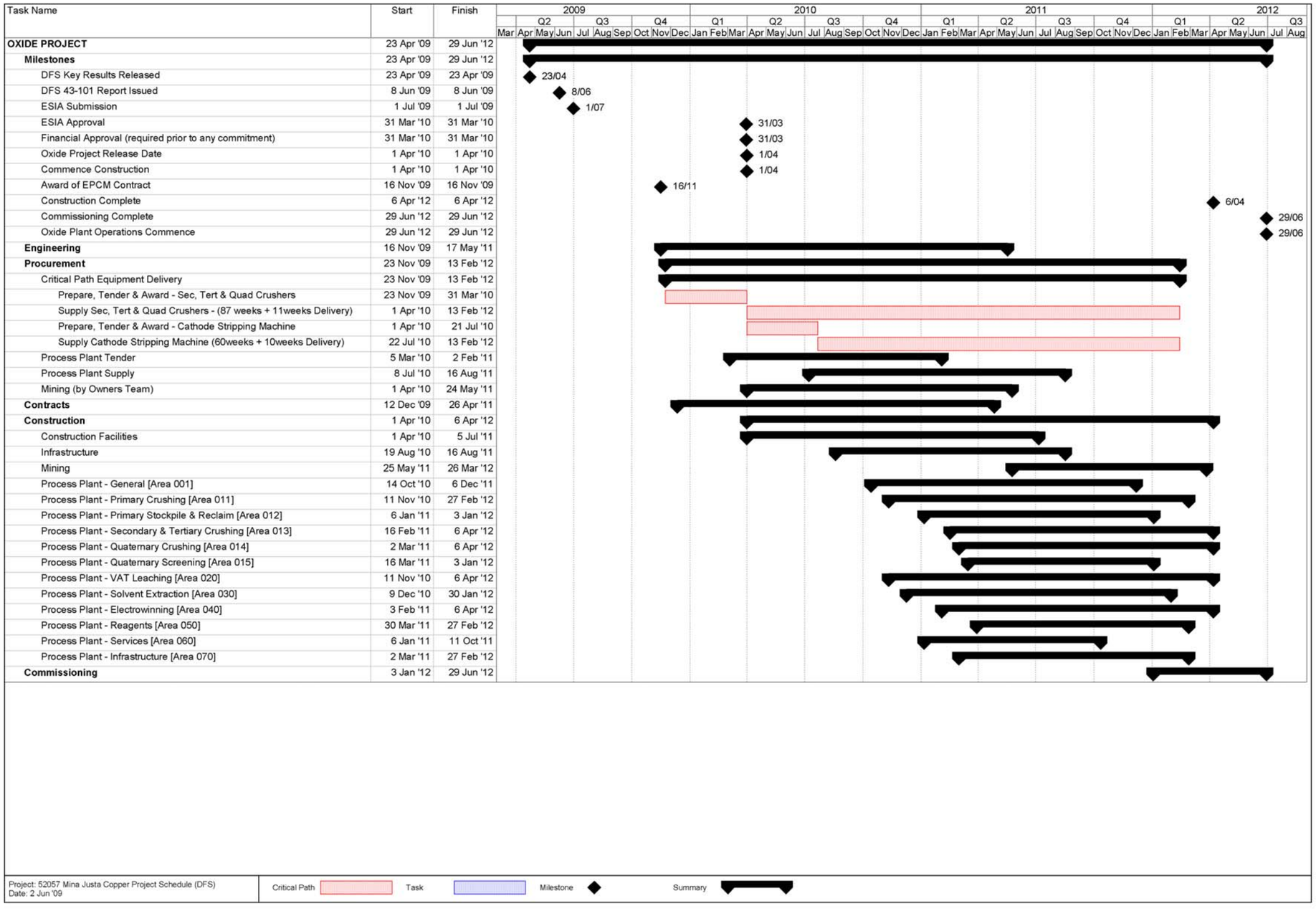
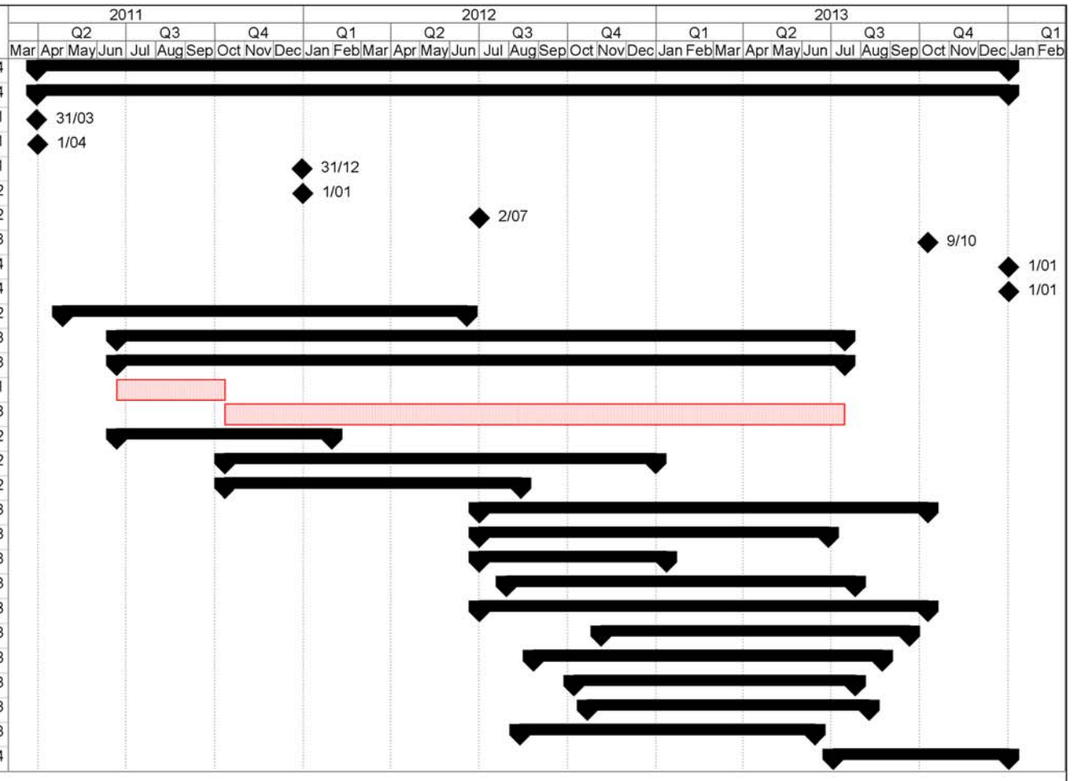


Figure 18.25
Project Implementation Schedule - Sulphide Plant

Task Name	Start	Finish
SULPHIDE PROJECT (CONCENTRATOR)	31 Mar '11	1 Jan '14
Milestones	31 Mar '11	1 Jan '14
Sulphide DFS Complete	31 Mar '11	31 Mar '11
ESIA Submission	1 Apr '11	1 Apr '11
ESIA Approval	31 Dec '11	31 Dec '11
Sulphide Project Release	1 Jan '12	1 Jan '12
Construction Start	2 Jul '12	2 Jul '12
Construction Complete	9 Oct '13	9 Oct '13
Commissioning Complete	1 Jan '14	1 Jan '14
Sulphide Plant Operations Commmence	1 Jan '14	1 Jan '14
Engineering	27 Apr '11	18 Jun '12
Procurement	22 Jun '11	15 Jul '13
Critical Path Equipment Delivery	22 Jun '11	15 Jul '13
Prepare, Tender and Award - Mills	22 Jun '11	11 Oct '11
Supply Mills (80weeks + 12 weeks Delivery)	12 Oct '11	15 Jul '13
Prepare,Tender and Award Long Lead Items	22 Jun '11	30 Jan '12
Delvery Lead Times (Delivered to site)	12 Oct '11	31 Dec '12
Contracts	12 Oct '11	13 Aug '12
Construction	2 Jul '12	9 Oct '13
Infrastructure	2 Jul '12	28 Jun '13
Process Plant - General [Area 001]	2 Jul '12	11 Jan '13
Process Plant - Primary Crushing [Area 110]	30 Jul '12	26 Jul '13
Process Plant - Grinding [Area 120]	2 Jul '12	9 Oct '13
Process Plant - Flotation [Area 130]	5 Nov '12	20 Sep '13
Process Plant - Concentrate Thickening & Filtration [Area 140]	27 Aug '12	23 Aug '13
Process Plant - Tailings Thickening & Disposal [Area 160]	8 Oct '12	26 Jul '13
Process Plant - Reagents [Area 170]	22 Oct '12	9 Aug '13
Process Plant - Services [Area 180]	13 Aug '12	14 Jun '13
Commissioning	4 Jul '13	1 Jan '14



Project: 52057 Mina Justa Copper Project Schedule (DFS)
Date: 2 Jun '09

Critical Path | Task | Milestone | Summary

18.9.1.1 Critical schedule activities

The critical schedule activities for the Oxide Project are:

- Obtaining environmental and construction approval by the nominated Project Release date to allow early site works to commence as planned, e.g. construction of the camp.
- Process design and flow sheet finalisation. The first engineering activities include generating the key engineering documents such as the process design criteria, process flowsheet, mechanical equipment list, site layout, mechanical plant general arrangements, and control philosophy. It is critical to minimise the turnaround time for the review and approval of these key documents because the engineering cannot progress if these have not been approved and locked.
- Construction of the camp. It is essential to start building the camp accommodation and facilities early enough to allow the main construction work to commence and ramp-up. The kitchen/dining facility is expected to be the critical path for the camp.
- Delivery time for cone crushers for secondary, tertiary and quaternary crushing.
- Delivery time for the cathode stripping machine is near a critical item.
- Construction time for the concrete vats in the Vat Leaching Area.

The most critical schedule activity for the Sulphide Project is delivery time for the mills.

18.9.2 Contracting strategy

The Project will be implemented with a general strategy of multiple horizontal discipline-oriented contract packages (for earthworks, concrete, structural erection/mechanical/piping, tank erection, field piping, electrical/instrumentation works, HV electrical works, process control, etc.) with the inter-discipline construction interfaces managed by an EPCM contractor.

This approach is aimed at minimising construction interface issues and risks. Key elements are as follows:

- Matching contractor skills with the work scope and minimising the impact of using too many lower tier contractors.
- Optimising interfaces by using small packages for specialised or interruptive work that, if incorporated into the larger packages, would cause disruption or loss of focus with the main contractors.
- Focusing on an orderly and timely implementation of the project philosophy, i.e. schedule to allow for progressive release of engineering, prioritised procurement and delivery of material and equipment.
- “Free-issuing” specific long lead equipment and materials to allow contract scopes to be defined without requiring contractors to procure specialised long lead material or equipment.

In general competitive tendering on a lump sum basis is preferred for the Project using well-defined detail drawings, equipment list, standards and scope of works specification. This philosophy will apply in particular to bulk equipment supply and fabrication.

In circumstances where engineering and vendor information is limited, and there are time schedule constraints due to long delivery times, it may not be appropriate to enter into lump sum contracts. In such instances fabrication or construction contracts are better based on unit rates with provisional bills of quantities extracted from the Project control estimate. Such an approach could be adopted for the earthworks packages, concrete construction packages, and structural steel fabrication packages.

There is suitable construction labour and equipment available in Peru and surrounding countries such as Chile and Bolivia to carry out the works.

18.9.3 Implementation scope of work

The project scope of work includes the provision of facilities for mining, process plant, utilities and services, waste disposal and the associated infrastructure to support the construction work and on going operations.

It is envisaged that the division of responsibilities applicable to the study work will be similar for the project implementation phase.

Marcobre's scope will include:

- Finance, insurance, governmental approvals, environmental approvals and licences.
- Land purchase, easements, rights-of-way, permits, approvals, licences, security, medical, taxes and duties.
- Mine planning and operations.
- Engagement of specialist consultants and contractors for blasting, geotechnical monitoring, hazardous and non-hazardous waste disposal, and other specialist scopes.
- Contracts for power supply, port usage, transport of cathodes, copper concentrate, acid, etc.

EPCM Contractor's scope will include:

- Engineering and procurement of the following facilities:
  - Oxide Process Plant
  - Mine Facilities
  - Infrastructure and services
  - Sulphide Plant
- Contracting and management of construction contractors
- Management of commissioning

18.9.4 Organisation

The implementation strategy is based on an organisation with a Marcobre project team and an EPCM Contractor both reporting to a Marcobre Project Manager. The Marcobre project team will assist the Marcobre Project Manager in carrying out Marcobre's scope of work.

The EPCM Project Manager will be responsible for managing the EPCM works and other specialist consultants and subcontractors. Key personnel will be nominated to ensure that assigned areas of responsibility are delivered safely, on time, on budget and in accordance with specifications and project criteria.

The EPCM Construction Manager and management team will be based on-site to manage and oversee the construction contractors who will carry out the construction. A Project Sponsor's Committee, made up of senior management from Marcobre and the EPCM Contractor, will be appointed to provide resolutions to problems or issues that cannot be resolved by the project team.

18.9.5 Health, safety, environment and community

HSEC performance is critical to the success of the Project.

The Project HSEC Management Plan will be developed prior to project execution, and will identify the HSEC requirements, allocate duties and responsibilities, and detail the processes and procedures that are used to manage HSEC during the implementation of the project.

Systems, procedures and management plans will be used to align the key stakeholders, namely the Marcobre Team, the EPCM Contractor, contractors, vendors, the workforce and the community in order to achieve the HSEC objectives.

18.9.6 Project management

18.9.6.1 Risk management

The risk review/assessment process is a fundamental tool, and it will be used to maximise the probability of achieving the project objectives.

18.9.6.2 Planning and scheduling

Progress and performance will be monitored and measured against a base-lined Master Project Schedule. Suppliers and contractors will develop their own more detailed schedules to manage and control their work scopes, and provide input at a summary level into the Master Project Schedule that will be managed by the EPCM Contractor.

18.9.6.3 Work breakdown structure

A defined work breakdown structure (WBS) is a fundamental part of the development of the project numbering system and the cost accounting system, and will be adopted to define the overall project and break it down into smaller areas and sub-areas in a hierarchical numbering system.

18.9.6.4 Project cost controls

The quantities and project costs will be controlled and managed against the control budget through the EPCM contractor's cost control system for that part of the Project under the EPCM contractor's control. Marcobre will directly manage other costs such as Owner's Costs.

The EPCM Contractor's cost control system will involve tracking of contracts, procurement and expediting for direct costs and site services, temporary works, sub-consultants, timesheet and manhour control and associated expenses for indirect costs.

18.9.6.5 Quality assurance

Project management plans and project criteria will be developed early in the project implementation stage to ensure alignment of the EPCM Contractor with Marcobre's requirements. The EPCM Contractor will have an audit system in place to identify non-compliances.

Supplier's and contractor's quality systems will be assessed at time of tender, and compliance with their systems will be monitored by the EPCM Contractor during implementation.

18.9.6.6 Finance and administration

The processes to be used by the Project for cash forecasting, paying accounts, and financial reporting will be developed prior to project implementation.

18.9.7 Project phases

18.9.7.1 Permitting

The efforts to obtain all the applicable permits for the project will be grouped into two major phases. The first phase will include obtaining the environmental approval for the project and all the construction and operating permits required to commence mining activities and construction of the oxide ore processing facilities, including all supporting site infrastructure and off-site infrastructure.

The second phase (concentrator) will begin during the execution of the first phase. The construction of the concentrator is scheduled to commence immediately after commissioning of the oxide plant, and by that time all relevant construction and operating permits required for construction of the sulphide ore processing plant will be obtained, to the extent legally possible.

Note that certain key permits required for operations, such as the water license and the beneficiation concession, can only be obtained after the corresponding facilities have first been constructed, inspected, and found to be in accordance with the associated construction authorisation. Under the existing Peruvian legal and regulatory regime, it is therefore impossible to obtain all necessary operating permits and approvals prior to the commencement of construction.

In accordance with applicable law, the Mine Closure Plan will be submitted within 1 year following the approval of the ESIA and will be updated as required.

Required Permits and Approvals

The purpose of preparing a strategy for the procurement of government permits and approvals is to assure that they will be obtained in a timely manner and in compliance with the applicable laws and regulations. The dates in which the applications requesting the approvals should be filed with the authorities are directly related with the timeline of the construction schedule. The correct outcome of the procedure depends on the correct and timely preparation of the technical documents that support the applications, in accordance with the applicable regulations and guidelines.

With regard to the environmental approvals required for the Project, Peruvian law has established the (*ventanilla única*) principle through Legislative Decree Nº 757, (*Ley Marco para el Crecimiento de la Inversión Privada*), modified by Law Nº 26734. According to this principle, the ministry and the supervising agencies in charge of regulating an industry sector are also the environmental authorities for that sector.

The competent environmental authorities for the mining sector are the Ministry of Energy and Mines (MEM) and the Bureau for the Supervision of the Investments in Energy and Mines (OSINERGMIN). The former approves the environmental management instruments, in this specific case, the ESIA and the Mine Closure Plan for the Mina Justa Project. The latter is in charge of supervising compliance with the legal obligations in environmental matters.

Marcobre will have to obtain other governmental consents in order to develop activities that are regulated by Peruvian legislation, i.e. mining and mineral processing, construction of hydraulic infrastructure and water use, electrical transmission, archeological evaluation projects, storage of fuel and the use of restricted chemicals, explosives, telecommunication equipment, radioactive substances, etc. These consents are issued by authorities within the Ministry of Energy and Mines, as well as within other ministries.

The strategy to obtain the governmental consents has been structured using two main tools: i) a Gantt diagram called *Mina Justa Project – Gantt Chart,* which summarizes each administrative procedure and the timeframe in which it will have to be started; and, ii) a spreadsheet called *Mina Justa Project – Cost Chart*, which summarizes the costs of following the administrative procedures to obtain the consents. Refer to DFS Appendix 20.2.

The Gantt Chart should be updated following the considerations of the *Mina Justa Project – Permits List and Assumptions* document, which establishes the fundamentals of the strategy as well as any assumption(s) required to support a decision on the Gantt Chart. This document must be reviewed periodically in order to revalidate the criteria used. Changes in any of the aforementioned documents may determine the need to review and accordingly update the others.

The following subsections describe the main consents to be obtained in order to construct and operate the facilities in each phase. Renewals or modifications of some of the permits and approvals may be required from time to time in the ordinary course of business.

(1) First phase: construction and start-Up

Before the start of the first phase it is necessary to obtain the general Project permits and approvals, including approval of the ESIA, and obtaining the Certificate of Nonexistence of Archeological Remains

(CIRA) for the area that will be involved during the Project implementation.[9] Marcobre will have to apply for the approval of the Mine Closure Plan within one year after the approval of the ESIA.

During this phase the following are the principal permits and approvals to be obtained:

- Beneficiation concession construction authorisation, and following construction and inspection, the beneficiation concession itself, which constitutes the principal approval for the operation of the oxide ore processing facilities.
- Authorisation to start mining exploitation activities in the Mina Justa Pit, the Magnetite Manto Pit and the construction material quarries.
- Authorisation to construct hydraulic infrastructure for the wellfield facilities and the water pipeline to the mine site.
- Licence to use groundwater for domestic and mining purposes, which can only be issued after the construction and inspection of the hydraulic infrastructure.
- Definitive Electrical Transmission Concessions for the 220 kV and the 22.9 kV electrical lines.
- Effluent Treatment and Reuse (in case Marcobre decides to discharge effluents into the environment).

The rest of the permits, approvals and consents that Marcobre is required to obtain and/or maintain during this phase are identified in the Gantt chart, cost chart and assumptions documents. Refer to DFS Appendix 20.2.

(2) Second phase: concentrator construction and ongoing operations

The second phase of the Project will commence during the construction and operation of the oxide plant (phase 1) and will include the construction and start-up of a concentrator plant that will process copper sulphide ore from Mina Justa.

This phase should be developed once the consents required for its construction are obtained. During this phase the following are the principal consents to be obtained:

- Modification of the beneficiation concession (includes construction authorisation).
- Effluent Treatment and Reuse (in case Marcobre decides to discharge effluents into the environment).

The rest of the consents that Marcobre is required to obtain and/or maintain during this phase are identified in the Gantt Chart, cost chart and assumptions documents. Refer to DFS Appendix 20.2.

[9] Permits and approvals that will be requested are: Authorisation to execute the Archaeological Evaluation Project (Mina Justa, Water infrastructure and 220 kV. areas), Approval of the Final Report of the Archaeological Evaluation Project (Mina Justa, Water infrastructure and 220 kV. areas) and finally the Certificate of Nonexistence of Archaeological Remains (Mina Justa, Water infrastructure and 220 kV. areas).

18.9.7.2 Engineering

The engineering plan will be based on conducting the engineering from a well equipped and resourced Project office, by a competent group of engineers with the appropriate level of skills and experience, utilising standard documents and procedures.

To ensure Marcobre has input to process and plant control issues, meetings will be conducted progressively with Marcobre representatives during the development of:

- Process flow sheets
- Process control philosophy
- P&IDs
- Design criteria.

The detailed design will take into account the desert conditions and earthquake exposure. The basic requirement for the plant is for reliable operation in compliance with the law and applicable standards including Equator Principles.

18.9.7.3 Procurement & contracting

Procurement and contracting will be undertaken by the EPCM team. Vendor and contractor lists will be prepared for Marcobre approval, with consideration given to capability, product quality and performance and delivery record. Highest priority will be placed on sourcing materials and services in Peru. Each package will be competitively tendered and awarded on the basis of a uniformly applied evaluation and adjudication process.
The approach to procurement and contracting will be that all packages will be tendered, awarded, and administered through to close-out in the name of Marcobre, with the EPCM Contractor acting as the Marcobre's representative for execution of the work.

Tenderers for site construction work will be prequalified on the basis of key criteria such as HSE&C, capacity to complete the works, and track record in similar environments.

Invoices and progress claims will be forwarded to the EPCM Project Manager via specific project personnel for quantity/delivery/condition verification and approval where applicable. Marcobre will control final authorisation and payment.

The nature and magnitude of the Mina Justa Project is such that some major items will be procured overseas, and a myriad of fabricated items will be produced in Peru. As part of the procurement plan, responsibilities for timely and undamaged deliveries will be set out. Peru-based transport companies familiar with local law, customs requirements and the transportation of processing equipment will be used to manage and coordinate logistics.

18.9.7.4 Construction

The EPCM Contractor will manage the construction contractors on behalf of Marcobre. The EPCM Contractor will be responsible for ensuring that no activity commences without all applicable permits and approvals required for the commencement of that activity being in-hand.

The first construction activity typically involves clearing of vegetation and the recovery and placement of topsoil into stockpiles for future rehabilitation. However, given the extremely sparse vegetation and poor development of topsoil at the Project site, this activity will be limited to local areas as required.

The principal construction materials are cement, sand, and aggregate for concrete works. Sand of suitable quality is in plentiful supply in the project area, and will be accessed from a borrow pit in the mine area. A small quarry will be established within a waste area of one of the initial open pits to provide feed to a mobile crusher to produce aggregate.

Material for the base of the access road and mine area roads will be locally derived, using crushed rock from waste areas of the proposed open pits as required to supplement broken material at surface.

Prior to commencement of construction, tests will be done on the mine waste material to determine its suitability for concrete and fill. In case mine waste material is not suitable, the quarry sites identified by Knight Piésold will be used once Marcobre carves out the relevant area of the TA1 metallic mineral mining concession, converting such area to a non-metallic minerals concession. The procedure for converting a portion of a metallic minerals mining concession to a non-metallic minerals mining concession is inexpensive and uncomplicated. Knight Piésold has done adequate testing on the sand and aggregate materials in these quarry sites to confirm their suitability for concrete and fill.

Early activities comprise the construction of the access road and internal roads, and establishment of temporary accommodation for the workforce, temporary power and temporary water supplies. Permanent power and water supplies will be developed during the construction period, but are not required until production machinery and equipment have been installed and are ready for testing.

Depending on local conditions, which have been determined by geotechnical surveys, ground preparation will be required to form firm foundations, particularly for larger machinery such as crushers, mills and vats. A batch plant for concrete will be established on site for use during the construction period, but will not be a permanent facility.

Construction will be undertaken in an orderly manner, with equipment brought to site as required and stored in a temporary lay-down area.

Once the process equipment has been installed, and power and water brought to and reticulated around the site, the equipment will be commissioned. Prior to this, the mining fleet and support facilities will be established on-site, pre-stripping of waste will commence (about 9 months prior to start-up of the processing facilities), and initial ore stockpiles will be established to allow commissioning and subsequent ramp-up of the plant to full production.

Labour requirements have been estimated for construction and operation of the Project. The intention is to maximise local employment during both construction and operation of the Project, but it is recognised that certain skills may need to be obtained from elsewhere in Peru.

Numbers will vary greatly over time during construction, depending on the activities being undertaken, but it is estimated that the peak construction labour force will number approximately 1 400 workers, not including Marcobre employees or support services contractor employees such as catering, security, explosives, heavy vehicle maintenance, and laboratory. Of these, it is estimated that up to 70% will be brought in from outside the area and will require accommodation on site during specific construction phases. The camp has been designed to accommodate up to 980 people during construction, sufficient to accommodate all those on site at any one time considering the expected proportion of workers resident in the immediate vicinity of the mine, the typical Peruvian construction work schedule of 21 days on, 7 days off, and a hotelling approach to camp accommodation.

The camp will be renovated and upgraded to provide offices and a permanent camp for Marcobre employees not living in San Juan de Marcona or Nazca.

18.9.7.5 Commissioning

The Commissioning Manager will lead the commissioning team, with discipline engineers reporting directly to the Commissioning Manager. It will be necessary to overlap construction, pre-commissioning and commissioning activities, particularly considering that two processing plants are being built in sequence. Responsibilities will be clearly defined together with authorised signatories for documentation such as check lists, test documentation, punch lists and handover certificates.

As sections of plant become available, wet commissioning will begin and areas of plant will begin to be considered mechanically as well as electrically live. Wet commissioning will culminate with running of individual sections of the plant on water, solvent and acid, as the case may be, to confirm process integrity. Practical completion is typically achieved at the end of this phase and in effect is the shift of responsibility from the EPCM Contractor to the Owner. Ore commissioning, under the management of the Marcobre's Project Team, with assistance from Operations, will commence after each system of the plant has been successfully wet commissioned.

18.9.8 Project implementation risks

The following risks are recommended to be addressed for the implementation of the project.

- Delay in the critical schedule activities will directly impact the progress of the project. It is recommended to carefully plan and manage these activities, and to actively pursue schedule savings.
- The camp is essential for the project because there is insufficient existing accommodation near the site. It is planned to commence the other construction works while in the process of building the camp. So it is recommended to plan and closely manage the camp construction and contractor mobilisation schedules to ensure that accommodation and facilities are made available ahead of demand in order to avoid constraining the build-up of the main construction workforce.

An allowance has not been made specifically for inclement weather, industrial strife, civil unrest, earthquakes, unidentified community festivals, and the like. There are more than 3,000 community festivals celebrated in Peru each year and those that apply locally have not yet been identified. It is recommended, at later stage of the project (e.g. when more specific information is available regarding contracting strategy, local industrial conditions, etc.), to review these risks and if necessary make a contingency allowance for them.

18.10 PROJECT OPERATIONAL PLAN

18.10.1 Production schedule

The mine production schedule is shown in Table 18.3. Oxide ore is mined at an annual rate of 12 Mt over the period 2013-2021, while treatment of sulphide ore commences in 2013 and continues at an annual rate of 5 Mt until closure. Quantities of waste rock mined with the ore will vary from year to year, but the overall average will be 2.46 t of waste to 1 t of ore (including pre-production stripping).

18.10.2 Operational labour levels and sourcing

The workforce required to operate the Project is shown in Table 18.10. The expectation is that 60% of the workforce will be drawn from the towns of Marcona and Nazca, and will be bussed to site each day. The remainder will be based outside the region and will require accommodation in the camp. The operations camp is designed to hold 300 people in single person quarters.

Initially, it is expected that local skills will be limited to security, clerks, general labouring, drivers, technical assistants and plant operators. However, Marcobre intends to employ full-time personnel officers to conduct training courses aimed specifically at building up the skills base of local labour so that over time the proportion of local labour will rise to 90% of the total work force and will be strongly represented in management and technical areas.

Table 18.10 Summary Operations Manning Levels			
	Phase 1 Oxide	**Phase 2 Oxide + Sulphide**	**Phase 3 Sulphide**
Mine			
All manning excl. maintenance	227	236	97
Maintenance manning	65	65	65
Mine - Total:	**292**	**301**	**162**
Process Plants			
Oxide	124	124	0
Sulphide	0	104	104
Subtotal (excl. Maintenance):	**124**	**228**	**104**
Cleaners	8	8	8
Maintenance manning	71	107	70
Plants - Total:	**203**	**343**	**182**
Site Organisation (excl. Maintenance)	75	84	75
Total Site Manning	**570**	**728**	**419**

18.10.3 Closure/post-closure plan

A preliminary closure and post-closure plan has been prepared as part of the DFS. The intention is that the Project will have no material, long term, negative impacts on the environment of the project area. The focus is on addressing potential impacts from waste rock, ripios and tailings disposal, and the closure plan ensures that any harmful components of these three waste streams are shielded permanently from the environment. All surface buildings and equipment will be removed from site unless otherwise agreed with the Peruvian authorities in accordance with applicable regulations (e.g. power lines and water supply lines may be retained).

Monitoring of the effectiveness of the closure plan will continue for a period of time to be determined in the corresponding Mine Closure Plan as approved by the regulatory authorities.

It is expected that closure of the operations will take place in stages. In particular, vat leaching, SX/EW facilities and the ripios dump would be decommissioned and the sites reclaimed while the mine operation continues to mine and process Sulphide ore. There are obvious advantages in being able to decommission facilities and reclaim the sites on a progressive basis, including the continued availability of equipment and utilities during the initial closure activities, as well as an enhanced ability to conduct post-closure monitoring due to continued presence of project personnel on site.

18.11 ENVIRONMENTAL CONSIDERATIONS

The Environmental and Social Impact Assessment (ESIA) for the Mina Justa Project forms the principal mechanism for identifying baseline conditions and evaluating the impact of the project. The ESIA has been designed to satisfy the requirements of Peruvian Legislation and to comply with internationally accepted guidelines for social and environmental protection followed by such organizations as the

World Bank and International Finance Corporation, and followed by commercial banks through the Equator Principles.

Currently the ESIA-related work is in its final stage; more specifically, parameters determined during the baseline study are being compared with the ones developed in the project description, so that mitigation measures that need to be considered for approval, can be elaborated.

18.11.1 Legal framework

The legal and institutional framework in Peru is represented by a number of authorities that have the jurisdiction to permit and regulate implementation of mining projects. Primary among these is MEM. The legal framework applicable to the Mina Justa Project is outlined by a number of environmental protection laws and documents. Key among these is Peruvian General Environmental Law (Law 28611).

The Peruvian environmental legislation is in the process of being updated, primarily through the creation of the Ministry of Environment, which, in the future, will be the entity responsible for monitoring, controlling and promoting the care of the environment in the country. The intention is that in the future all the controls, permissions and authorisations are centralized in this Ministry, but the process of developing and adapting the legislation and other organisms of the State for this purpose is still underway.

It is clear that under current legislation, MEM is the responsible environmental authority for approving the ESIA and authorising project development. However, the ESIA now being finalised will fulfill all requirements that the new authority is likely to request.

18.11.2 Permitting

The ESIA is submitted to regional and central offices of the MEM. The central office in Lima is in charge of conducting the evaluation process and, eventually, issuing permits. Depending on the location of the project and its characteristics, other agencies may be involved in the evaluation of the ESIA. The evaluation process also includes making the ESIA availabe to affected local communities for review and comment, publication of findings by the agencies, a period for the applicant to respond, and then a period of final evaluation before approval and issuance of a concession to operate.

Given the presence to the north of the project of the San Fernando Reserve, MEM will seek the opinion of the National Institute of Natural Resources, prior to issung permits. The evaluation of the impacts from construction, operation and closure of the Project from the environmental study concludes that there is no direct or indirect influence on the San Fernando Reserve, and no complications are expected in obtaining the permissions of that authority.

In addition, it is necessary to obtain agreement to the results of the archaeological evaluation from the National Institute of Culture, which is authorised to issue the required Certificate of Nonexistence of Archaeological Remains.

18.11.3 ESIA scope

The key objectives of the ESIA now nearing completion are:

- Determine baseline environmental conditions in the Project area, i.e. establish the physical, biological and socio-cultural conditions existing before the establishment of the Project.
- Identify environmental and socio-economic resources that could potentially be affected by the Project.
- Predict positive and negative effects resulting from the Project, and determine to what degree the negative effects can be mitigated.
- Quantify and evaluate the significance of the effects wherever possible.
- Outline requirements for monitoring of the resources that could be affected by the project.
- Provide a conceptual closure plan for the mine site and associated facilities.
- Complete a cost-benefit analysis of the project.

18.11.4 Baseline studies

A detailed description of environmental and social aspects of the project area was developed. The studies began in 2006 and were completed in 2008. All of the baseline studies were developed by local professionals.

The baseline study has not been restricted to the Mina Justa Project area, but covers the district of San Juan de Marcona.

The environmental and socio-economic impacts were identified by measuring the existing characteristics of the area, and comparing them with results anticipated following Project implementation. In some cases (particularly in air and water studies), models were developed to evaluate the magnitude and extent of potential effects.

The results of the studies indicate that Mina Justa Project site conditions are typical of a desert, with no surface water, saline and poor soils, generally unsuitable for activities other than mining.

Surveys have recorded scant presence of flora and fauna typical of the desert environment, which are not unique to the site but are represented throughout the San Juan de Marcona district.

No communities or population centres occur inside the zone of direct environmental influence of the Project.

Some archaeological vestiges have been recorded, and these will have to receive the treatment that the legislation dictates in order to delimit or preserve them as required.

18.11.5 Community relations and public consultation

In order to optimize relations between the community and the project, an integrated community relations program has been developed with the following objectives:

- Establishment of ties with community leaders to enhance understanding of the social conditions of the neighbouring populations, their concerns and hopes for development.
- Disclosure and consultation regarding the technical and economic aspects of the project.
- Identification and establishment of mechanisms to support local development processes throughout and after operations.
- Strengthening of the institutions through development of consensual programs based on mutual respect and transparency.

From the beginning of the environmental studies, Marcobre has implemented a policy of involvement with representative sectors of the community as part of the ESIA process.

Peruvian legislation recommends a minimum of three public consultation meetings during the elaboration of the ESIA. The initial meeting is designed to introduce the communities to the ESIA process, help them understand their rights and responsibilities, and to describe the baseline studies that form a part of the permitting process. Information about the general characteristics of the project (scale, lifecycle, etc), the complexity of the mining activity, and the relations that will be established with the local community are shared in each of these meetings. The local population is involved in the baseline studies, with community members participating in the field teams specialising in fauna, flora, water and soil surveys.

Marcobre has successfully conducted workshops over three years, which have allowed it to communicate the development of the feasibility study, and to receive contributions and suggestions from the community.

Marcobre's Office of Community Relations located in San Juan de Marcona has permanent contact with the community and has joined in the life of the population, continuously informing the community about the project and providing feed-back to Marcobre.

18.11.6 Identification and evaluation of effects

The environmental and socio-economic impacts were identified and compared with the anticipated impacts of the implementation of the project. The main effects and corresponding mitigation measures for the construction and operating stages are related mainly to water and land usage for the mine site.

Some of the impacts identified are:

- Changes in the current use of the soils.
- Changes in the topography and soil due to the presence of tailings and ripios, open pit and waste rock facilities.
- Generation of dust during the construction and operational phases.
- Generation of noise in the construction and operating areas.
- Minor loss of vegetation coverage.
- Migration of some fauna species due to the presence of the operation.

- An increase in immigrants during the construction phases; a reduction is expected during mine operation and closure.

18.11.7 Environmental management

Marcobre has committed to instituting best practices for the environmental management of the project. The implementation process will begin once the authorities have granted permission to proceed with the project. This will provide a global mechanism to ensure that appropriate environmental management is maintained during the life of the mine.

Environmental management is conceived for three principal stages of the project: construction, operation and closure.

The principal components of the Plan of Environmental Management are:

- Monitoring program
- Program of management of domestic and industrial residues
- Program of management of domestic and industrial effluents
- Policy regarding the behaviour of Marcobre and contractor personnel.
- Contingency plans.

Marcobre will establish the position of Environmental Manager, responsible for the control and environmental management of operations. This area will report directly to the Central Manager , and will be in charge of supervising and controlling all of the environmental programs related to the Project.

18.11.8 Mine closure

Closure legislation (Peruvian Law 28090) requires that every operation must have an approved closure plan and financial guarantees of ability to cover the estimated closure costs. The closure plan must be developed within a year following the approval of the ESIA, and it must be approved by the MEM prior to receipt of permission to operate.

The ESIA has developed a conceptual closure plan for the operation, with the objectives of ensuring the physical and chemical stability of the diverse components of the project after closure, and returning the environment to a condition similar to that found before implementation of the Project.

The principal closure activities relate to the reduction in slopes of waste dumps, in order to assure physical stability, and covering potential acid generating material with inert material.

In addition, the closure plan confirms the demolition of infrastructure and levelling of the involved areas.

Finally, and depending on the requirements of Government regulators and the local communities, it is possible that ownership of some of the infrastructure, e.g. the water pipeline and/or the electrical transmission line, might be transferred to the community for its use post-closure.

18.11.9 Socio-economic conditions

Details of the socio-economic conditions, as well as the projected benefits of the Mina Justa Project and Marcobre´s community relations efforts, are detailed in Subsection 15.8 of Volume 9 of the DFS report.

The Mina Justa Project is located in the Department of Ica, an important agricultural region, where grapes, cotton, asparagus, olives and other produce are cultivated, and a place where the biggest deposits of iron on the Pacific coast have been formed. Ica has experienced approximately 1.8% population growth between 1993 and 2005; however, a portion of the population still lacks access to basic services. Ica also has significant poverty rates, with about 29% of the population classified below the poverty line (INEI 2007). More than half the population earns its living through agriculture and fishing. Mining is also a significant contributor to the economy.

Most of the affected local people live in the town of San Juan de Marcona, which has a population of approximately 11 600 habitants and is located approximately 24 km from the Mina Justa Project.

The Mina Justa Project will contribute to the local area through jobs, local purchases of goods and services, and through taxes.

18.12 CAPITAL COST

18.12.1 Project capital

GRD Minproc developed or supervised the capital cost estimates for the mining equipment, mine development, process plant, and associated in-plant and ex-plant infrastructure. Marcobre developed the Owner's Cost estimates, which have been incorporated into the capital cost estimates by GRD Minproc. These are presented as an Oxide Plant DFS estimate (which includes the Mine DFS estimate) and as a Sulphide Plant PFS estimate.

The capital cost estimates are structured to encompass the following major categories:

- **Direct capital costs** include expenditures incurred for the construction of the process plant and infrastructure, mining and associated capital costs as defined in the Oxide Plant (DFS) and Sulphide Plant (PFS) scope of work. The costs include permanent materials and equipment, freight to site, construction labour and equipment (including contractors' supervision, overheads and profit), temporary construction facilities, construction mobile equipment, and commissioning assistance. (Note: GMI has considered the vendor representatives, first fill consumables and start-up spares within its Indirect costs for the infrastructure components.)
- **Indirect capital costs** are the expenditures related to the engineering design, procurement, project management, site construction management and commissioning supervision by the EPCM contractor. Indirect costs also provide for consultants required to supplement design engineering and construction activities.
- **The accuracy provisions** reflect the level of definition available relating to the scope of work, process design, conceptual engineering design and cost data at the time of the capital estimate development. These make appropriate allowances for uncertain elements of cost, for estimating anomalies and omission in quantification, thereby reducing the risk of cost variation within the required accuracy level.

- **Owner's costs** include customs duties, insurance, Owner's project team, Owner's operating team prior to production, property costs (surface rights and mineral rights) and other Owner's intangibles, excluding sunk costs.

The estimated total costs are summarised in the following tables:

- Table 18.11: Oxide Plant DFS Capital Cost Estimate, Summarised by Area.
- Table 18.12: Sulphide Plant PFS Capital Cost Estimate, Summarised by Area.

Table 18.11
Oxide Plant DFS Capital Cost Estimate, Summarised by Area

Area No.	Area Description	Bare Cost ($)	Accuracy Provision (%)	Accuracy Provision ($)	Total Cost ($)
001	General Plant	11,055,834	11.4%	1,261,807	**12,317,641**
010	Crushing and Screening	1,097,296	10.0%	109,731	**1,207,027**
011	Primary Crushing	12,011,627	8.9%	1,063,609	**13,075,236**
012	Primary Stockpile and Reclaim	5,399,360	9.3%	501,547	**5,900,907**
013	Secondary Screening/Crushing and Tertiary Crushing	20,287,527	6.1%	1,228,564	**21,516,091**
014	Tertiary Screening and Quaternary Crushing	22,968,439	6.0%	1,387,639	**24,356,078**
015	Quaternary Screening	10,418,274	7.3%	758,199	**11,176,473**
020	Vat Leaching	67,288,240	9.3%	6,248,419	**73,536,659**
030	Solvent Extraction	18,772,003	11.2%	2,094,912	**20,866,915**
040	Electrowinning	29,898,230	7.6%	2,262,348	**32,160,578**
050	Reagents – Oxide	2,391,151	10.3%	246,951	**2,638,102**
060	Services – Oxide	3,469,873	11.4%	395,997	**3,865,870**
070	Infrastructure – Oxide	21,533,336	9.7%	2,097,806	**23,631,142**
079	Mobilisation and Demobilisation	2,949,703	10.7%	315,812	**3,265,515**
080	Temporary Facilities	4,354,608	10.0%	435,461	**4,790,069**
081	Commissioning – Oxide	2,020,961	10.0%	202,096	**2,223,057**
082	Vendor Representatives	1,058,439	10.0%	105,844	**1,164,283**
083	First Fills and Spares	11,512,650	10.0%	1,151,265	**12,663,915**
084	Loose Tools and Equipment	1,221,938	10.0%	122,193	**1,344,131**
095	Power Supply	11,545,443	10.0%	1,154,545	**12,699,988**
096	Plant Access Road	7,134,713	13.6%	968,895	**8,103,608**
097	Construction Camp and Village	17,008,679	10.0%	1,700,868	**18,709,547**
098	Water Supply	16,576,532	13.7%	2,270,740	**18,847,272**
200	Mining	123,150,502	0.2%	224,776	**123,375,278**
	Direct Costs – Subtotals	**425,125,358**	**6.7%**	**28,310,024**	**453,435,382**
	EPCM	51,080,140	10.0%	5,108,014	56,188,154
	Indirect Costs – Subtotals	**476,205,498**	**7.0%**	**33,418,038**	**509,623,536**
	Owner's Costs	37,242,013	0.0%	0	37,242,013
	Totals	**513,447,511**	**6.5%**	**33,418,038**	**546,865,549**

The DFS capital cost estimate for the oxide plant has a level of accuracy of ±10% and is expressed in 1Q09 US dollars (1Q09).

Table 18.12
Sulphide Concentrator PFS Capital Cost Estimate, Summarised by Area

Area No.	Area Description	Bare Cost ($)	Accuracy Provision (%)	Accuracy Provision ($)	Total Cost ($)
001	General Plant	4,651,645	18.5%	860,088	**5,511,733**
098	Water Supply	3,598,297	12.8%	460,312	**4,058,609**
110	Sulphide Primary Crushing	15,066,043	14.3%	2,157,212	**17,223,255**
120	Sulphide Grinding	36,804,910	9.6%	3,545,430	**40,350,340**
130	Sulphide Flotation	17,230,211	15.2%	2,615,714	**19,845,925**
140	Sulphide Concentrate Thickening and Filtration	7,579,799	13.8%	1,045,210	**8,625,009**
160	Sulphide Tailings Thickening and Disposal	15,102,265	18.2%	2,753,420	**17,855,685**
170	Sulphide Reagents	2,444,480	16.9%	414,113	**2,858,593**
180	Sulphide Services	7,592,634	16.7%	1,265,847	**8,858,481**
188	Mobilisation and Demobilisation	2,136,838	13.8%	295,244	**2,432,082**
190	Temporary Facilities	2,125,302	15.0%	318,795	**2,444,097**
191	Commissioning	485,763	14.3%	69,596	**555,359**
192	Vendor Representatives	535,528	15.0%	80,329	**615,857**
193	First Fills and Spares	3,681,004	19.9%	732,099	**4,413,103**
	Direct Costs – Subtotals	**119,034,719**	**14.0%**	**16,613,409**	**135,648,128**
	EPCM	22,129,233	0.0%	0	22,129,233
	Indirect Costs – Subtotals	**141,163,952**	**11.8%**	**16,613,409**	**157,777,361**
	Owner's Costs	10,529,709	0.0%	0	10,529,709
	Totals	**151,693,661**	**11.0%**	**16,613,409**	**168,307,070**

The PFS capital cost estimate for the sulphide plant has a level of accuracy of ±20% and is expressed in 1Q09 US dollars.

18.12.1.1 Estimation methodology

The estimate methodology applies to the GRD Minproc scope of work only, unless otherwise stated.

Generally, for earthworks, concrete, structural steelwork and platework fabrication and installation supply rates and unit man-hours are based on information provided by GMI, the Marcobre approved contractor for the off-site infrastructure. For consistency, GRD Minproc and Indec were requested to use these rates where possible. GRD Minproc has independently checked GMI-provided rates against other Peruvian contractors. For these disciplines, quantities were determined from material take-offs based on preliminary designs and layout drawings.

Equipment specifications were prepared and issued with tender packages for all major equipment items and packages. Budget equipment prices were obtained for all major items of equipment. Where budget quotes were not received, the balance of costing was derived from GRD Minproc's database and from allowances based on the database.

In-plant piping is derived from the actual costs of similar type plants completed by, or currently in progress, designed by GRD Minproc and adjusted to Peruvian costs and productivity. The basis of the piping estimate is installed piping to number of pumps per area, for equivalent type areas.

Electrical equipment prices were obtained from multiple Peruvian suppliers for all major items of electrical equipment; the balance of pricing was based on quotes from suppliers for recent GRD Minproc projects. Some minor items were based on GRD Minproc electrical estimating database. The majority of unit rate items were built up from unit rates supplied by GMI. Site installation hours have been calculated from the GRD Minproc in-house database, using Peruvian norms as the basis.

For the Buildings a budget quotation was received from a local contractor and used within the estimate. Building area costs were compared and verified against similar buildings on current projects.

Transport of concrete-related bulk materials is included in the all-in concrete rate (reinforcing, cast-in steelwork, culverts, etc). Transport rates for steelwork and platework bulk materials were derived from rates received from installation contractors.

Transport for all equipment items is based on information received for steelwork and platework transport. Where this method of calculating freight costs was inappropriate, an allowance varying from 5% to 12% was applied. This is based on historical information, depending on original source of equipment, volume, weight, etc.

No equipment suppliers provided freight costs with their quotes other than to their closest port of departure in some instances only.

Subsequent to original pricing requests some transport information has been received from suppliers and incorporated into the estimate.

18.12.1.2 Sunk costs

In addition to the costs presented in Table 18.11 and Table 18.12, the project carries sunk costs of US$99.6 M to 31 December 2008, and is projected to incur additional costs through 30 September 2009 of $6.4 M. Total sunk costs at 30 September 2009 would be $106.0 M. These sunk costs are not considered in the economic analysis, other than the Peruvian income tax and IGV tax benefits associated therewith that would otherwise be unused.

GRD Minproc did not prepare estimates for the following key components that apply to both plants:

- Owner's Costs (including all taxes, import duties, statutory charges, etc), which were prepared by Marcobre.
- Project Contingency: GRD Minproc strongly recommends that adequate allowances are made by Marcobre for items not included in the estimates as presented by GRD Minproc, such as change of scope, abnormal or inclement weather, acts of God, industrial disturbances, foreign currency rate of exchange variations, or variances to the current market situation. Marcobre has elected to determine the Project Contingency allowance as nil for the purpose of economic assessment.
- Escalation: at Marcobre's request, escalation has been excluded from the project estimates.

18.12.2 Sustaining capital

Sustaining capital represents the amount of capital investment, at current day costs (1Q09), required to sustain and support the process plant operation at its most productive and efficient level, by replacing support equipment or extending permanent structures at various pre-determined intervals. This includes deferred capital such as a tailings dam lift, and sustaining capital such as the replacement of mining equipment, other heavy mobile equipment, computer hardware, light vehicles, etc. that have an expected service life shorter than the mine life.

The summary of the sustaining capital requirements is presented in Table 18.13.

Table 18.13
Sustaining/Deferred Capital Summary

PLANT	DESCRIPTION	TOTAL COST ($)
	Deferred Capital	
Sulphide	Recleaner concentrate pump 2	28,600
Sulphide	Tailings Storage Facility – Phase 2	3,220,000.00
Sulphide	Tailings Storage Facility – Phase 3	4,640,000
Sulphide	Pressure Filter Upgrade	510,000
	Sub-Total Deferred Capital	**8,398,600**
	Sustaining Capital	
Oxide/Sulphide	Replacement of computers	1,411,000
Oxide/Sulphide	Mining Sustaining Capital	15,185,638
Oxide/Sulphide	Vehicles Sustaining Capital	7,859,997
	Sub-Total Sustaining Capital	**24,456,635**
	TOTAL	**32,855,235**

18.13 OPERATING COSTS

The operating costs for the project have been determined for the following categories:

- Mining
- Oxide plant
- Sulphide plant
- General and administration (site and Lima office)
- Land transport, port, ocean freight, marketing, treatment and refining charges.

The operating cost estimate has an accuracy of ±10% (except the Sulphide Plant which was developed to a PFS level of ±20%). Operating costs are in United States Dollars and reflect an estimate base date of 1Q09 unless otherwise stated.

A summary of Project unit operating costs per ROM tonne is provided in Table 18.16, while Project closure costs are shown by year in Table 18.17.

Costs originally in currencies other than US Dollars have been converted to US Dollars at the exchange rates shown in Table 18.14.

Table 18.14
Exchange Rates

Currency Unit		Units per US$
AUD	Australia Dollars	1.54
CLP	Chile Pesos	595
EUR	Euro	0.785
JPY	Japan Yen	91.7
PEN	Peru Nuevos Soles	3.26
USD	United States Dollars	1
ZAR	South Africa Rand	10.1
CAD	Canadian Dollar	1.24
Note: Base date 16 February 2009		

IGV tax is not included in the operating cost estimates. It is expected to be fully recovered by Marcobre with a three month lag.

Except where specifically noted, no allowance has been made in the operating cost estimate for the following items:

- Financing charges
- Contingency
- Escalation or exchange rate variations
- Depreciation and accounting effects
- Sustaining capital (which is included in capital expenditures)
- Exploration on other prospects, whether part of the Marcona Copper Property or otherwise.

Key unit operating costs provided by Marcobre are summarised in Table 18.15.

Table 18.15
Key Unit Costs Provided by Marcobre

Item	Unit	Cost
Diesel (delivered to site, including site storage and dispensing)	US$/litre	0.636
Electricity (including transmission)	US$/MWh	55.00
Sulphuric acid (delivered to site)		
Via San Martin	US$/tonne	90.40
Via San Juan	US$/tonne	62.50

18.13.1 Mining cost

Mine operating costs have been determined based on equipment and labour requirements to achieve the mine schedule.

The mining unit cost includes appropriate allowances for technical labour and materials to control the mining operation, including grade control, giving due consideration to the geology and mineralogy of the deposit and, specifically, the widespread presence of non-mineralised dykes in certain parts of the deposit.

The accuracy of the Mining operating cost estimate is ±10%.

Mining costs are determined on the basis of Owner mining except for:

- Mining equipment maintenance and repair costs, which are assumed to be carried out under a maintenance and repair contract between Marcobre and the vendor of the mining equipment.
- Blasting, which is assumed to be carried out by the explosives supplier under contract with Marcobre.

Mining costs have been derived considering the following areas:

- Drilling
- Grade control
- Blasting
- Loading
- Hauling
- Dumping (including dump construction)
- Services (dust control, geotechnical, safety)
- Management (supervision, planning).

Within these areas, mining costs have been developed considering the following cost categories:

- Labour
- Power
- Explosives
- Drilling consumables
- Ground-engaging tools
- Other consumables
- Fuel and lubes
- Maintenance materials
- Maintenance contracts
- Miscellaneous items.

18.13.2 Oxide plant

The Mina Justa Oxide Plant is designed to process 12 Mt/a of oxide ore by four stages of crushing, vat leaching, solvent extraction, and electrowinning to produce up to 52 000 t/a cathode copper for sale on world markets.

Operating costs have been developed for the Oxide Plant under the following categories:

- Labour
- Power
- Maintenance materials
- Reagents
- Consumables
- Miscellaneous items.

The accuracy of the Oxide Plant operating cost estimate is ±10% and reflects the plant operating at design capacity.

18.13.3 Sulphide plant

The Mina Justa Sulphide Plant is designed to process 5 Mt/a of sulphide ore to produce copper concentrates (also containing silver and lesser gold) for sale on world markets.

Operating costs have been developed for the Sulphide Plant under the following categories:

- Labour
- Power
- Maintenance materials
- Reagents
- Consumables
- Miscellaneous items.

The accuracy of the Sulphide Plant operating cost estimate is ±20% and reflects the plant operating at design capacity.

18.13.4 Transport

San Martin port is selected for cathode and acid shipments initially, and Matarani port for copper concentrates, switching to the Port of San Juan once available.

Reagents and consumables have been priced by suppliers on the basis of delivery to site where possible, or shipped via the port of Callao. Additional road transport costs from Callao to Mina Justa site have been estimated on the basis of the transportation cost assessment undertaken by Marcobre and Sandwell.

18.13.5 General and administration

The General and Administration costs cover labour and miscellaneous items required to support site operations.

The Labour cost in this category includes administration, community relations, environmental, safety, security, accounting, logistics, laboratory and centralised maintenance personnel.

Miscellaneous items include administration costs, insurance, personnel transport and accommodation, site services, administration vehicles, road maintenance, consultants and health, safety and environmental management related costs.

General and Administration costs encompass both site administration and Lima corporate office costs, but exclude costs attributable to exploration on Marcobre's other prospects on the Marcona Copper Property.

18.13.6 Environmental

The environmental costs considered are related to the phases of construction, operation, closure and post-closure phases of the project.

The cost of environmental monitoring is based on Vector's work, which includes costs associated with:

- Atmospheric condition monitoring
- Air quality and noise
- Biological monitoring
- Water quality (TSF seepage)
- Inspection and audit costs by governmental authorities, related to health, safety and environment
- Other costs including dust control program, management of efluents and management of domestic and industrial solid wastes.

Closure costs were estimated by Vector, following review of Knight Piewsold designs.

Table 18.16
Summary of Project Operating Costs (US$/t ROM processed) Model 090821

Area	Period												
	Yr -1	2012	2013	2013	2015	2016	2017	2018	2019	2020	2021	2022	2023
Mining	0	9.88	5.47	3.72	3.60	3.68	3.84	3.74	3.74	2.98	1.85	2.24	2.07
Oxide Plant	0	5.76	5.68	5.68	5.69	5.69	4.50	4.49	4.52	4.52	4.46	4.57	-
Sulphide Plant	0	-	-	4.90	4.82	5.20	5.12	5.06	4.81	5.24	4.80	5.16	4.92
General and Administration	0	2.14	1.30	1.08	1.04	1.05	1.05	1.05	1.05	1.04	1.03	1.88	2.95
Corporate Office (Lima)	0	0.25	0.15	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.11	0.22	0.38
Transport/ Marketing	0	0.23	0.23	3.42	3.08	4.74	4.88	2.65	2.60	2.66	2.61	7.18	11.92
Total	**0**	**18.27**	**12.84**	**18.91**	**18.34**	**20.46**	**19.49**	**17.10**	**16.83**	**16.54**	**14.85**	**21.24**	**22.25**

Table 18.17
Summary of Project Closure Costs

	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15
Progressive Closure [1)]	688 468	323 212	155 209	1 656 698	851 302	-	-	-
Final Closure [2)]	-	-	-	-	3 008 260	5 556 186	3 217 554	144 125
Total	**688 468**	**323 212**	**155 209**	**1 656 698**	**3 859 563**	**5 556 186**	**3 217 554**	**144 125**

[1)] Progressive closure costs include costs related to pits mined out before cessation of production and closure of the oxide treatment facilities (including the ripios dump) which cease operation prior to the concentrator.

[2)] Final closure costs include sulphide processing facilities, waste dumps, tailings pond, concentrator facilities, camp and infrastructure.

18.14 MARKETING AND PRODUCT PRICING

18.14.1 Copper cathode sales contract

Marcobre's marketing approach for its copper cathode production considers requirements under the Shareholders Agreement as well as financing requirements. Current plans would commit 100% of the first 10 years' production of copper cathodes to LS-Nikko Copper and Norddeutsche Affinerie AG (NA).

18.14.1.1 LS-Nikko copper

Pursuant to the Shareholders Agreement, and subject to finalisation of the formal sales contract, Marcobre has agreed to sell 70% of its copper cathode production to LS-Nikko Copper. The Shareholders Agreement specifies that the contract will contain the following principal terms:

- Quantity: 70% of annual production reducing to 35% of annual production if the joint interest of LS-Nikko and Korea Resources in Marcobre is less than 20% but at least 15%, reducing to nil if their joint interest is below 15%.
- Term: 120 months from commencement of cathode production.
- Marketing Fee and Cathode Premium: Annual negotiation, with dispute settlement by arbitration.
- Other terms and conditions will be on standard industry terms and conditions for the sale of copper cathodes on a long-term basis.

18.14.1.2 NA

Marcobre has entered into a letter of intent with NA covering the remaining 30% of annual production for 10 years from the commencement of copper cathode production. The key terms of the letter of intent are as follows:

- Quantity: 30% of annual production.
- Term: 120 months from commencement of cathode production.
- Shipment: equal monthly shipments.
- Basis: CIF landed into customer's barge along-side vessel.
- Marketing fee: to be agreed.
- Cathode premium: related to the benchmark premium/discount of major producers.
- Quotational Period: either the month prior to the month of shipment or the month following the month of arrival at the port of discharge.
- Other terms and conditions: on a basis that is standard and normal in the industry for the long-term sale of copper cathodes.

18.14.2 Copper concentrates

18.14.2.1 LS-Nikko

Pursuant to the Shareholders Agreement, it was agreed that Marcobre would sell 90% of its copper concentrate production to LS-Nikko for 10 years from the commencement of copper concentrate

production. The sales contract has not been finalised, but the Shareholders Agreement specifies that the contract will contain the following principal terms:

- Quantity: 90% of annual production, reducing to 45% of annual production if the joint interest of LS-Nikko and Korea Resources in Marcobre is less than 20% but at least 15%, further reducing to nil if their joint interest is less than 15%.
- Term: 120 months from commencement of concentrate production.
- Delivery: CIF FO Onsan, Korea, Hibi Port, Japan, or Saganoseki, Japan, at Buyer's option.
- Payable Cu: for copper concentrates grading 35% Cu or less, pay 96.5%; for copper concentrates grading more than 35%, pay 96.75%; subject to a minimum deduction of 1 unit.
- Payable Ag: less than 30 g Ag/dmt, 0; 30 g Ag/dmt or more, 90%.
- Payable Au: less than 1 g Au/dmt, 0; between 1 g Au/dmt and 3 g, 90%; 3 g Au/dmt or more, to be agreed in the long term sales contract.
- Pricing Basis:
  - Copper: average of the daily LME Grade "A" settlement quotations during the Quotational Period.
  - Silver: average of the daily London Bullion Market fine silver spot quotations during the Quotational Period.
  - Gold: average of the London Bullion Market a.m. and p.m. quotations during the Quotational Period.
- Quotational Period: third month after month of arrival.
- Allocation to Market Terms and Price Sharing Terms: 60% Market-Related Terms, 40% Price Sharing Terms.
- Copper Treatment and Refining Charges, Market-Related Terms:
  - Copper treatment charge and refining charge: negotiated annually on a two year block basis, based on then current international transactions and agreements for similar quantities between major mines and major custom smelters in Korea and Japan for quantities of not less than 30 000 t/a.
  - Copper Price participation, ±10% on the basis of ¢90/lb for first two years, and thereafter subject to biannual negotiation based on market terms.
- Copper Treatment and Refining Charges, Price Sharing Terms:
  - Combined treatment and refining charge: as a percentage of LME Copper Grade "A" settlement price for the Quotational Period, 24% for first 60 months, and 25% for second sixty months, subject to a minimum of US$0.19 per pound of payable copper, no maximum.
- Refining charge for gold and silver: annual negotiation based on market terms.
- Payment Terms: 90% of provisional invoice on second business day after arrival of vessel at port of discharge; 10% of provisional invoice 60 days after arrival; final payment on 3rd day after Buyer's confirmation of Seller's final invoice; all payments made under irrevocable letter of credit.

- Insurance: Seller pays for insurance at 110% of value stated in the provisional invoice, subject to adjustment to 110% of final value.
- Other terms: on terms standard and normal in the industry for the sale of copper concentrates on a long-term basis.

18.14.2.2 Other copper concentrate contracts

The 10% balance of the copper concentrates not committed to LS-Nikko would constitute one or two shipments a year in most years, insufficient to support a long term sales contract. Marcobre intends to sell the uncommitted production on a spot basis, most likely to smelters in the Far East to take advantage of freight savings from combining shipments with the volume going to LS-Nikko. For the purposes of the DFS, it has been assumed that the 10% balance of production is sold to LS-Nikko on the same terms as the market portion of the long-term contract.

18.14.3 Market review (copper and sulphuric acid)

Marcobre contracted Brook Hunt and Associates Limited (Brook Hunt) to provide a market review of the following:

- Supply and demand for copper cathodes and copper concentrates.
- Copper price forecast.
- Copper concentrate treatment and refining charge forecast.
- Copper cathode premium forecast.
- Penalty elements and standard penalty rates.
- Freight rate forecast for copper concentrates to South Korea and copper cathodes to South Korea and Northern Europe.
- Sulphuric acid supply, demand and price forecast in the Chile-Peru market.
- Elemental sulphur price forecast FOB Vancouver.
- Sulphur freight rate forecast from Vancouver to Major Peruvian Ports.

The initial report was completed in August 2008. Owing to the intervening financial and commodity market disruption, Marcobre requested Brook Hunt to prepare an update, which is dated February 12, 2009.

The Brook Hunt projections are in Q1 2008 U.S. Dollar terms. US CPI inflation from Q1 2008 to Q1 2009 was -0.04%; accordingly, Marcobre determined that no inflation adjustment was required to convert the Brook Hunt forecasts to a 1Q09 basis.

Table 18.18 summarises the key forecasts from the updated Brook Hunt report, and compares the forecasts with the assumptions used in the DFS financial analysis.

Table 18.18
Summary of Forecast Prices and Terms
Q1 2009 US$ terms, 2012 to 2023 Average

	Brook Hunt Cases [1]			DFS
	Low	Base	High	Case
Price				
LME Grade "A" Copper, $/lb	1.82	1.93	2.12	2.00
Cathode premium, $/tonne				
South Korea	63.82	69.17	77.50	69.55
Germany	82.08	89.17	100.42	89.55
Reagents				
Sulphuric Acid, $/t, CIF Main Peruvian Ports	33.33	44.17	57.50	45.00
Ocean Freight Rates				
Cathodes: South Korea, $/t	53.83	57.08	61.50	56.91
Cathodes: Germany, $/t	59.58	64.00	67.58	63.64
Concentrates: South Korea, $/t	41.83	45.17	61.00	45.50
Market Treatment and Refining Charges, Copper Concentrates				
Treatment Charge US$/t	79.17	86.25	87.50	87.00
Refining Charge c/lb	0.07917	0.08625	0.0875	0.087
Price Participation Base commencing 2015	1.50	1.60	1.75	1.60
Precious Metal Refining Charges				
Au (US$/payable oz)	5.00	5.00	5.00	5.00
Ag (US$/payable oz)	0.35	0.35	0.35	0.35

[1] *Brook Hunt cases copyright Brook Hunt and Associates Limited*

The Brook Hunt forecasts are cyclical, as shown in Table 18.19. For the DFS base case cash flow projections, Marcobre has elected to use the simple average of the Brook Hunt base case projections over the relevant period, being 2012 to 2022 for costs affecting cathode production, and 2014 to 2023 for costs affecting copper concentrates. The assumptions derived in this matter are shown in the DFS Case column in Table 18.18.

Table 18.19
Brook Hunt Forecasts Base Case

Years	Units	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	Avg.
Base Case														
LME Grade 'A' Copper	$/lb	1.75	1.90	2.05	2.25	2.45	2.60	2.05	1.80	1.45	1.60	1.60	1.60	1.93
Cathode Premium, Hamburg	$/t	70	75	80	95	110	115	100	85	85	85	85	85	89
Cathode Premium, Korea	$/t	50	55	60	75	90	95	80	65	65	65	65	65	69
Cu Concentrate TC	$/t	75	90	105	80	75	90	100	95	85	80	80	80	86
Cu Concentrate RC	$/lb	0.075	0.090	0.105	0.080	0.075	0.090	0.100	0.095	0.085	0.080	0.080	0.080	0.086
Cu Price Participation Base	$/lb	-	-	-	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60	N.M.
Ocean Freight		43	44	41	41	44	43	44	47	48	49	49	49	45
Concentrate, Korea	$/t	43	44	41	41	44	43	44	47	48	49	49	49	45
Cathodes, Hamburg	$/t	61	59	61	60	61	62	65	65	70	68	68	68	64
Cathodes, Korea	$/t	55	52	55	54	55	55	60	60	62	59	59	59	57
Sulphuric Acid CIF Peruvian Port	$/t	50	45	45	55	50	60	45	40	35	35	35	35	44

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18.15 TRANSPORT, MARKETING AND REALISATION COSTS

The operating costs covered under this category include the following:

- Product trucking
- Port storage and handling
- Ship loading
- Ocean freight
- Marketing fee in the case of copper cathodes
- Copper concentrate treatment and refining charges
- Marine Cargo Insurance for copper cathodes and copper concentrates.

Mina Justa copper cathodes and copper concentrates will be trucked to port and shipped to overseas customers, while sulphuric acid will be transported by ship to port and trucked to site. For the first five years of operations, (2012 to 2016), transport of cathodes and acid are assumed to be via Port of San Martin, switching to the Port of San Juan in 2017. Concentrates are assumed to be shipped via the Port of Matarani in the first year of production (2014), switching to San Juan in 2015.

Cost estimates are shown in Table 18.20.

Table 18.20
Transportation, Marketing and Realisation Costs

Category	Units	$
Copper Cathodes		
Truck to port, San Martin/San Juan	US$/t	19.59/5.48
Port handling and storage charges, Martin/San Juan	US$/t	18.20/14.50
Ocean freight: South Korea	US$/t	56.91
Ocean freight: Northern Europe	US$/t	63.64
Marine cargo insurance	% of CIF value	0.15125%
Marketing fee	US$/t	13.00
Copper Concentrates		
Truck to port, Matarani/San Juan	US$/wmt	35.72/5.48
Port handling and storage charges, Matarani/ San Juan	US$/wmt	10.50/22.00
Ocean freight	US$/wmt	45.50
Marine cargo insurance	% of CIF value	0.15125%
Market Portion (60% of LS-Nikko and 100% of other)		
Copper concentrate treatment charge	US$/dmt	87.00
Copper refining charge	US¢/payable lb Cu	8.7
Price-Sharing Portion (40% of LS-Nikko)		
Combined treatment and refining charge	% of Copper Price	24.5% Note 1
Gold refining charge	US$/payable oz Au	5.00
Silver refining charge	US$/payable oz Ag	0.35
Note 1: 24% for first 5 years, 25% for second five years, floor of 19 cents per payable pound.		

18.16 PROJECT FINANCIAL ANALYSIS

18.16.1 Background

GRD Minproc provided the capital cost, operating cost and production plan inputs for the financial analysis but expresses no opinion on the financial analysis prepared by Marcobre.

18.16.2 Key project assumptions

A production schedule is provided in Table 18.21. Annual production quantities differ from those presented in Table 18.3 due to the project schedule as outlined in Section 18.16.3 offsetting quantities by 6 months. Figure 18.27 displays changes in Project annual head grade, by process, over time, while Figure 18.28 shows annual production of payable copper, again by process.

The following key assumptions and parameters were used in preparing the project cash flow projections:

- Capital costs as described in this study.
- Operating costs as described in this study including the following key unit operating costs:
  - Diesel, US$0.636/L delivered to site, including site storage and dispensing
  - Electricity, US$55/MWh at the busbar at the San Juan de Marcona Substation of Red Energia del Perú (REP)
  - Sulphuric Acid, US$45/t CIF main Peruvian ports.
- Project ramp-up and production as illustrated in Figure 18.26.
- Corporate income tax rate of 32%, the rate applicable under a mining stability agreement, which Marcobre expects to qualify for and obtain.
- Tax depreciation at rates applicable under a mining stability agreement, being 5% for buildings, 20% for machinery, equipment and other fixed assets, and life of mine for intangibles.
- Temporary Net Assets Tax (ITAN) of 0.4% of net assets.
- Statutory employee profit sharing, 8% of pre-tax profits.
- Dividend withholding tax of 4.1%.
- Mining royalty (payable to the Government of Peru) of 1% on the first $60 M of annual sales, 2% on the next $60 M in annual sales, and 3% on annual sales in excess of $120 M, in accordance with applicable Peruvian law.
- Financial transactions tax (ITF) of 0.05% on all receipts and payments.
- Metal prices:
  - Cu price of US$2.00 /lb.
  - Silver price of US$11.00/oz.
  - Gold price of US$700.00/oz.
- Road transport, port and ship loading costs for copper cathodes and copper concentrates as outlined in section 18.15.

Figure 18.26
Project ramp-up schedule

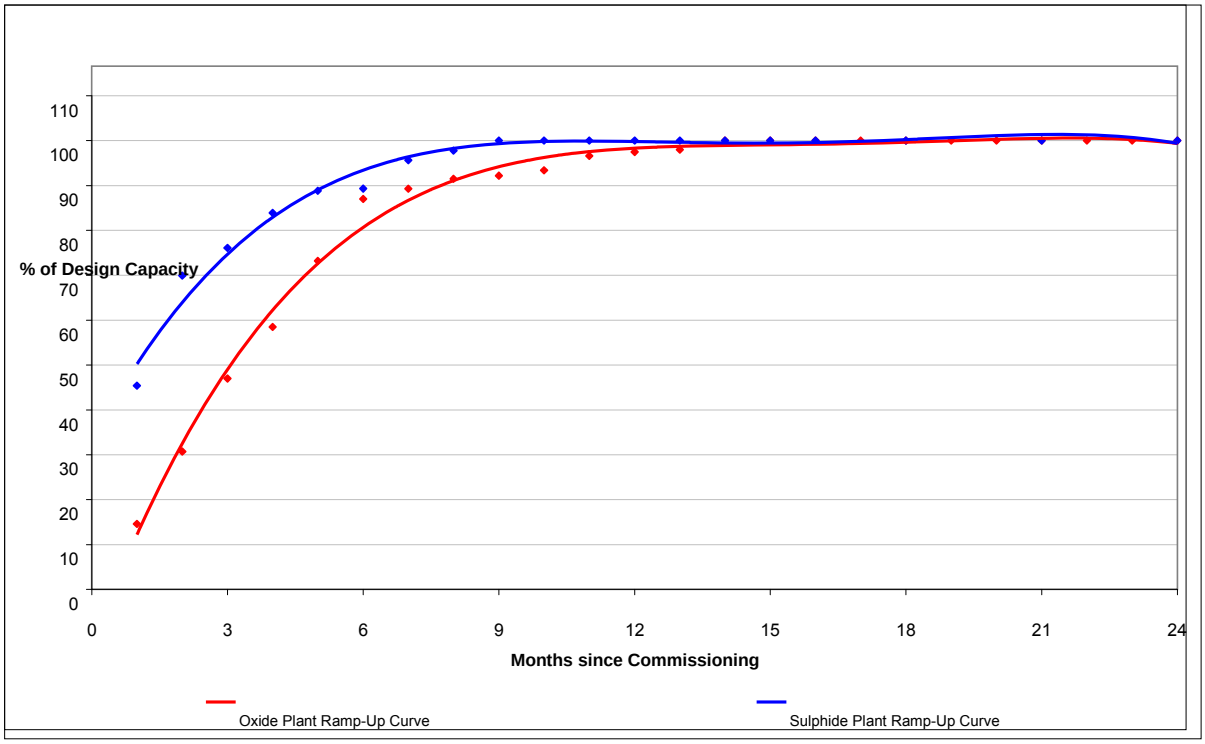


Table 18.21
Annual Production Schedule

Years	Units	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	Total (*Average in (italics)*
Mine Production															
Oxide ore mined	000 t	215	10 845	20 157	13 846	7 768	11 329	14 923	10 484	9 109	9 411	5 750	765	-	114 602
Sulphide ore mined	000 t	-	-	11	4 901	5 038	5 062	4 803	5 172	5 024	5 010	5 048	5 632	3 092	48 794
Total ore mined	000 t	215	10 846	20 169	18 747	12 806	16 391	19 726	15 657	14 133	14 420	10 798	6 398	3 092	163 396
Waste mined	000 t	4 171	47 190	39 764	41 726	47 807	43 723	40 669	44 743	46 097	29 740	11 308	4 440	984	402 363
Total mined	000 t	4 386	58 036	59 932	60 474	60 612	60 114	60 395	60 400	60 230	44 160	22 106	10 838	4 075	565 769
Closing ore stockpile	000 t	215	7 511	15 980	18 307	14 112	13 502	16 229	14 886	12 018	9 438	3 236	1 279	-	-
Oxide Plant															
Oxide ore processed	000 t	-	3 550	11 699	12 001	11 999	12 001	11 999	12 000	12 000	12000	12000	3 335	-	114 602
Feed grade	% Cu	-	0.549	0.575	0.613	0.560	0.540	0.593	0.592	0.568	0.522	0.513	0.481	-	*0.564*
Cu recovery	%	-	79.0	75.4	70.7	79.2	75.7	71.8	71.9	76.3	78.5	77.0	74.0	-	*74.5*
Cathode production	t	-	15 390	50 708	51 995	48 989	49 001	51 075	51 079	52 000	52 000	47 418	11 940	-	481 596
Concentrator															
Sulphide ore milled	000 t	-	-	-	4 420	5001	5 001	5 000	5 000	5 000	5 000	5 000	5 000	4 371	48 794
Cu feed grade	% Cu	-	-	-	1.366	1.275	1.986	1.996	1.179	1.030	1.042	1.086	1.418	1.304	1.369
Cu Recovery	%	-	-	-	91.0	91.9	94.7	94.4	92.4	91.6	92.1	92.4	93.9	93.3	93.0
Recovered copper	t	-	-	-	54 947	58 595	94 069	94 189	54 499	47 188	47 979	50 181	66 577	53 148	621 373
Copper concentrate grade	% Cu	-	-	-	38.5	41.4	43.8	38.3	35.9	37.5	34.9	33.3	35.5	36.5	37.8
Copper concentrate production	t	-	-	-	142 598	141 683	214 802	245 863	15 1612	125 671	137 575	150 551	187 748	145 638	1 643 741
Au feed grade	g/t	-	-	-	0.02	0.02	0.03	0.03	0.02	0.02	0.03	0.04	0.05	0.04	*0.03*
Au recovery	%	-	-	-	80	80	80	80	80	80	80	80	80	80	*80*
Recovered gold	000 oz	-	-	-	1.8	2.3	3.4	4.3	3.2	2.8	3.5	5.0	6.4	4.4	37.0
Ag feed grade	g/t	-	-	-	10.95	11.18	20.0	22.86	12.43	11.23	11.27	10.24	15.21	15.69	*14.12*
Ag recovery	%	-	-	-	80	80	80	80	80	80	80	80	80	80	*80*
Recovered silver	000 oz	-	-	-	1 245	1 438	2 572	2 940	1 599	1 445	1 450	1 317	1 957	1 763	17 725
Payable Metal in Concentrate															
Copper	t	-	-	-	53 190	56 644	90 961	90 957	52 583	45 558	46 275	48 359	64 212	51 294	599 953
Gold	000 oz	-	-	-	-	-	-	-	-	-	1.2	3.8	4.5	2.2	11.6
Silver	oz	-	-	-	1 120	1 294	2 315	2 646	1 439	1 300	1 305	1 186	1 761	1 587	15 953
Total Payable Copper Production	000 t	-	15 390	50 708	105 104	105 633	139 963	142 033	103 661	97 558	98 275	95 777	76 153	51 294	1 081 549

- Working capital:
  - Receivables based on expected sailing times (30 days) and contract payment terms (90% 2 business days after arrival at port of discharge and 10% 60 days later)
  - Supplies equal to 30 days of operating costs other than labour and power
  - Finished product inventories averaging 3100 t of cathode and 11 250 t of concentrates
  - Early recovery of value added tax (IGV) during the construction period as permitted under an Investment Agreement which Marcobre would qualify for and plans to obtain
  - Payables equal to 30 days operating costs, except labour in which case 15 days.
- Oxide ore treatment at 12 Mt/a for approximately ten years at an average head grade of 0.56% Cu and Sulphide ore processing at 5 Mt/a with an average head grade of 1.37% Cu, 14.1 g/t Ag, and 0.03 g/t Au from production quarter 7. Sulphide ore treatment continues for approximately ten years until production Year 12.

A number of criteria were considered in determining the viability of the Mina Justa Project, including:

- After-tax net present value using a real-terms discount rate of 8%;
- C1 cash cost per pound of payable copper
- Cash breakeven price
- Economic breakeven price
- Price and cost sensitivity, including sensitivity to the price of key inputs.

Figure 18.27
Project Head Grade (CuT%)

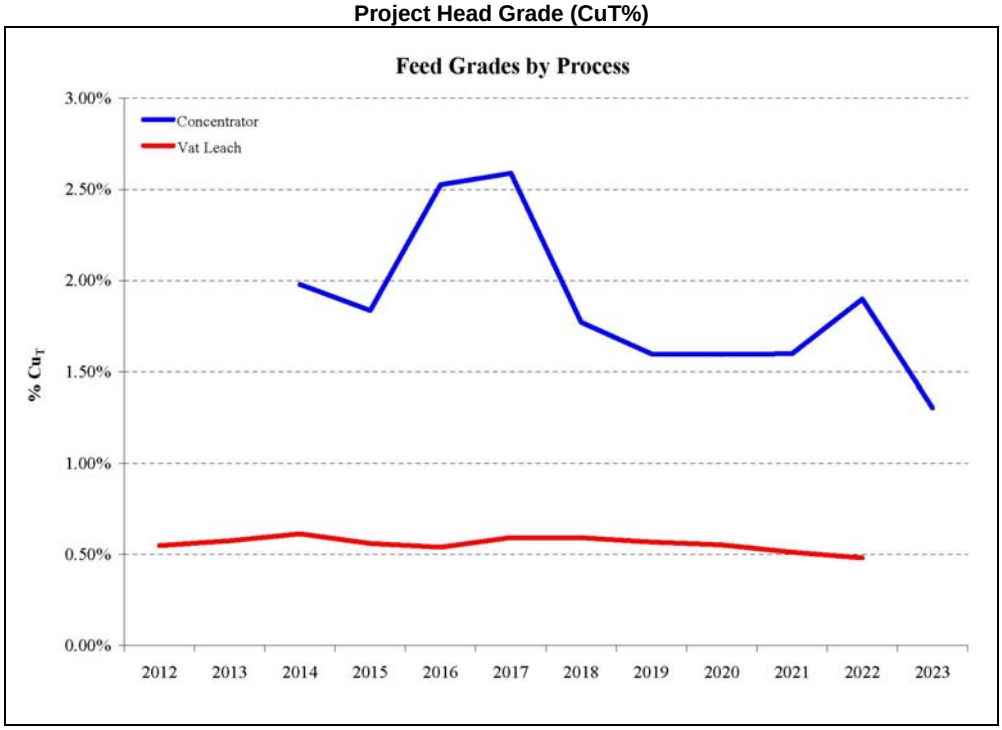
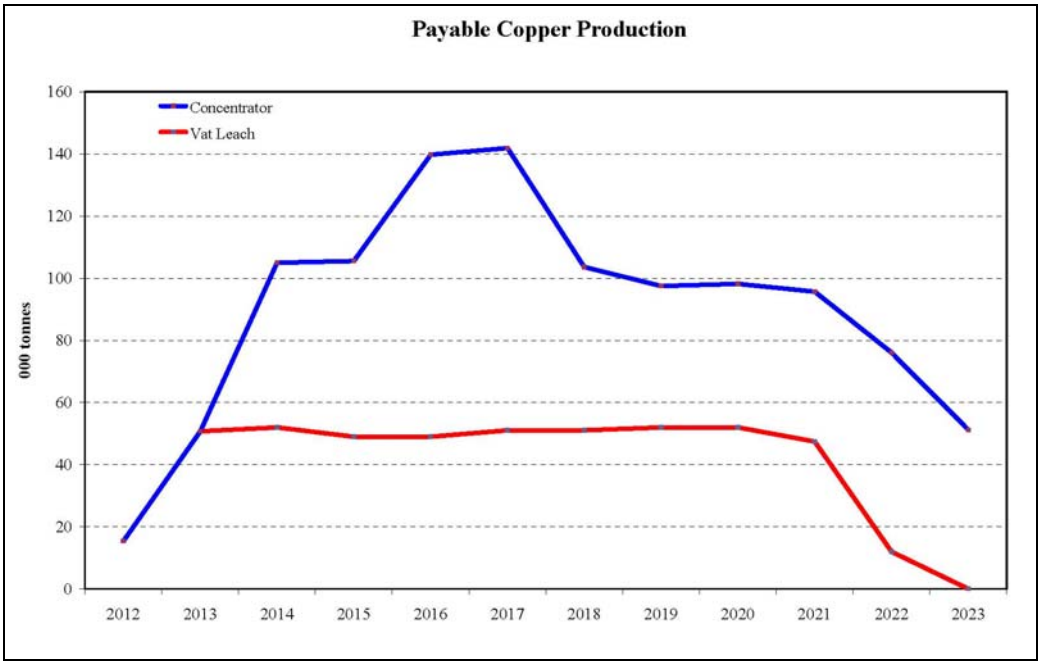


Figure 18.28
Production Data



18.16.3 Key cash flow assumptions

The cash flow projections and the financial evaluations based on those projections have been prepared considering the following schedule:

- ESIA submission in Q3 2009
- Other permitting and engineering commencing Q4 2009
- Project release on April 1, 2010
- Oxide ore treatment commencing July 1, 2012 and ending in Q1 2022
- Sulphide ore treatment commencing 1 January 2014 and ending in Q4 2023
- Closure of mined-out pits and the oxide treatment facilities taking place progressively, with final closure and post-closure monitoring lasting for 3.5 years.

The analysis has been conducted on a before-tax and after-tax basis. Escalation and inflation have been excluded, although there is provision within the model to separately escalate metal prices and costs. The currency adopted for the analysis is the US$.

Costs from October 1, 2009 are considered project costs for the purpose of the evaluation, and cash flows are discounted back to that date.

Corporate income tax, statutory employee profit sharing, mining royalties, financial transaction tax, temporary net assets tax and dividend withholding tax have been included in the calculation of after-tax cash flow. The effects of debt financing have not been assessed.

Sensitivity analyses are performed to demonstrate the effect of variations in key parameters on Project economics.

Working capital is provided for as set out in Sub-section 18.16.2.

Hedging has not been considered in the financial analysis.

The financial analysis is on a 100% equity basis.

18.16.4 Summary of results

Based on the assumptions set out above, the 100% equity basis after-tax IRR is 15.6% and the NPV at 8% is $333.0M. The project is expected to pay back initial capital, including the concentrator project capital, 4.6 years after the commencement of cathode production or 3.1 years after the commencement of concentrate production, determined on an after-tax basis.

The cash break-even copper price (the price at which revenues equal the sum of operating costs, sustaining and deferred capital, and closure costs) is US$1.059 per pound. The economic break-even copper price (the price at which the 100% equity basis NPV at 8% is equal to zero) is US$1.520 per pound.

The 114 6 Mt of oxide ore treated produces 481 596 t of copper cathodes. The 48.8 Mt of sulphide ore treated produces 599 953 t of payable copper in concentrates.

At constant prices over the life of the project of $2.00/lb of copper, $700/oz gold and $10/oz silver, it is anticipated that $4541 M would be realised in net smelter returns (96.1% copper, 3.7% silver, 0.2% gold).

On-site operating costs over the life of the project are anticipated to average $0.677/lb of payable copper. Total operating costs (including the mining royalty, transportation, marketing fees, and, in the case of copper concentrates, treatment and refining charges), are anticipated to average $0.978 lb of payable copper. After silver and gold by-product credits of $0.077 per pound of payable copper, the C1 cash cost is estimated at $0.902 per pound. Over the life of the Project, $1 615 M is expected to be paid in on-site operating costs and $748 M in realisation costs.

The key project statistics are summarised in Table 18.22.

Table 18.22
Key Project Economic Statistics

Project Economics		
Pre-tax		
Net Present Value at 8%	$ M	609.2
Internal Rate Of Return	%	20.2%
Capital Recovery Period: from concentrator start-up	years	2.9 years
: from SX-EW start-up	years	4.4 years
After Tax		
Net Present Value at 8%	$ M	333.0
Internal Rate Of Return	%	15.6%
Capital Recovery Period: from concentrator start-up	years	3.1 years
: from SX-EW start-up	years	4.6 years
Break-even Prices		
Cash-breakeven copper price	$/lb	1.059
Economic breakeven copper price	$/lb	1.520
Total Operating Costs		
Site production and G&A costs	$ M	1 615
Product transport and insurance	$ M	184
Treatment and refining charges	$ M	441
Mining Royalty	$ M	122
Financial Transactions Tax	$ M	2
Total Operating Costs	$ M	2 363
Capital Costs		
Pre-production stripping	$ M	28.9
Oxide project capital and owner's costs	$ M	546.9
Sulphide project capital and owner's costs	$ M	168.3
Total Initial Capital	$ M	744.7
Sustaining and deferred capital	$ M	32.8
Life of Mine Capital	$ M	777.5
Working Capital: End of Q1 of concentrator operations	$ M	62.8
: Maximum	$ M	70.3
Closure costs	$ M	15.6
Total Project Cash Flows		
Gross revenue	$ M	4 982
Total Operating Costs	$ M	-2 363
Operating margin	$ M	2 618
Interest expense	$ M	0
Interest income	$ M	4
Statutory employee profit sharing	$ M	-140
Subtotal	$ M	2 482
Working capital	$ M	10
Capital expenditures	$ M	-778
Closure costs	$ M	-16
Pre-tax cash flow	$ M	1 699
Taxes: Income tax and net asset tax	$ M	-514
: Interest withholding tax	$ M	0
: Dividend withholding tax	$ M	-43
Project After-Tax Cash Flow	$ M	1 142
Costs per payable pound of copper		
Cathodes		
Site costs	¢/lb	106.4
Off-site costs	¢/lb	1.5
Mining royalty and Financial Transactions Tax	¢/lb	4.9
Total cash cost per pound, cathode copper	¢/lb	112.8
Copper concentrates		
Site costs	¢/lb	36.7
Off-site costs	¢/lb	43.9
Mining royalty	¢/lb	5.3
Net by-product credits	¢/lb	-13.8
Total cash cost per pound, copper in concentrates	¢/lb	72.1
Total Copper		
Site costs	¢/lb	67.7
Off-site costs	¢/lb	25.0
Mining royalty	¢/lb	5.1
Net by-product credits	¢/lb	-7.7
Total cash cost per pound, total copper	¢/lb	90.2

18.16.5 Sensitivity analysis

18.16.5.1 Prices, operating costs and capital costs

Table 18.23 illustrates the sensitivity of the economics to copper price, by-product (silver and gold) prices, operating costs and capital costs.

The financial indicators are all very sensitive to copper price. A 1% change in copper price from the base case $2.00 per pound results in a 4.1% change in after-tax NPV at 8%. The project is less sensitive to capital costs (around a 1.4% change in NPV for a 1% change in capital) and total operating costs, i.e. site and off-site, excluding taxes (1.3% change in NPV for a 1% change in operating costs). With silver and gold credits amounting to only 3.7% of gross revenue, there is very little sensitivity to by-product prices (only a 0.2% change in NPV for a 1% change in by-product prices).

Regarding cost components, acid, diesel and electricity are the largest, representing 21%, 13% and 12% of site costs respectively. A 1% change in acid price results in a 0.4% change in NPV, while a 1% change in diesel prices results in a 0.2% change in NPV. The result for electricity is similar to diesel.

Table 18.23 Sensitivities				
% of Base Case	After-Tax IRR	After-Tax NPV at 8%: $ M	Payback Period: Years	C1 Cash Cost, ¢/lb
Copper, percent of $2.00 base price				
80%	9.4%	56.6	4.16	85.6
90%	12.6%	196.1	3.57	87.9
100%	15.6%	333.0	3.15	90.2
110%	18.3%	469.5	2.93	92.4
120%	20.9%	605.9	2.75	94.7
Silver and Gold, percent of base prices ($11/oz and $700/oz)				
80%	15.4%	322.5	3.17	91.6
90%	15.5%	327.8	3.16	90.9
100%	15.6%	333.0	3.15	90.2
110%	15.7%	338.3	3.14	89.4
120%	15.8%	343.6	3.13	88.7
Capital Costs, percent of base case				
80%	19.1%	426.4	2.89	90.2
90%	17.2%	379.7	3.02	90.2
100%	15.6%	333.0	3.15	90.2
110%	14.1%	286.3	3.30	90.2
120%	12.8%	239.3	3.53	90.2
All Operating Costs (site and off-site), percent of base case				
80%	17.4%	421.0	2.99	79.0
90%	16.5%	377.0	3.07	84.6
100%	15.6%	333.0	3.15	90.2
110%	14.6%	289.0	3.24	95.7
120%	13.7%	244.9	3.40	101.3
Acid Price, % of avg. delivered price of $76.46/t				
80%	16.1%	358.1	3.09	87.2
90%	15.8%	345.6	3.12	88.7
100%	15.6%	333.0	3.15	90.2
110%	15.3%	320.5	3.18	91.6
120%	15.0%	307.9	3.21	93.1
Diesel Price, % of avg. delivered price of $0.636/l				
80%	15.9%	347.3	3.12	88.7
90%	15.7%	340.2	3.13	89.4
100%	15.6%	333.0	3.15	90.2
110%	15.4%	325.9	3.17	90.9
120%	15.2%	318.8	3.18	91.7

18.16.5.2 Brook Hunt scenarios

As discussed in Section 18.14.3, Marcobre commissioned Brook Hunt to provide forecasts of copper prices, copper realisation costs, ocean freight and sulphuric acid prices. With the exception of copper price (which Marcobre chose to set at $2.00), the DFS assumptions for these factors are set at the average over the relevant period of the Brook Hunt forecasts. The Brook Hunt forecasts are cyclical, however, and the use of averages can potentially obscure some important effects. Furthermore, Brook Hunt provided base, high and low case forecasts, which are useful in assessing the aggregate effect of a set of internally consistent changes in key prices and costs, rather than relying solely on the sensitivity analysis as set forth in the previous section.

Table 18.24 sets out the key results from employing the Brook Hunt forecasts in the financial model.

Table 18.24
Brook Hunt Scenarios

Brook Hunt Case	After-Tax IRR	After-Tax NPV at 8%: $ M	Payback Period: Years	C1 Cash Cost, ¢/lb
Low	15.6%	317.9	3.02	86.3
Base	16.7%	368.4	2.92	90.3
High	18.9%	476.3	2.82	94.9

There is a cyclical copper price peak in the Brook Hunt scenarios in 2016 and 2017. This just happens to coincide with the highest sulphide copper grades in the Mina Justa mine plan. The result is that the weighted average copper prices realised on Mina Justa production is considerably higher than the simple average price over the period. For example, in the Brook Hunt Base Case, the average copper price over the period 2012 to 2023 is $1.93/lb, while the weighted average realised price is $2.01/lb, an effective 4% increase in price.

18.17 RISK ASSESSMENT

18.17.1 Hazard identification (HAZID)

A hazard identification (HAZID) and risk assessment study was conducted to identify issues affecting the safety and health of people working in the Oxide plant (note that the Sulphide facility was not considered in this study as work was to a PFS level). The Sulphide facility was not considered in this study as design work was to a PFS level. Marcobre intends that a HAZID and risk assessment study would be performed for the Sulphide facility during the DFS for such facility.

The focus of the study was the identification of process hazards applicable during the operational phase of the Project and a review of the engineered safeguards included in the design that mitigate or reduce the risks of those hazards causing harm to personnel, members of the public or the environment.

The HAZID study addressed the following:

- Identify potential hazards associated with the proposed design and equipment in all process areas.

- Assess the credibility of each potential hazard, the impact of any consequences and the adequacy of existing hazard controls.
- Record all credible hazards with potentially significant impacts on health and safety, the environment or production, together with their causes, consequences and any existing controls included in the design.
- Record any recommended actions and assign responsible individuals from within the HAZID team to address potential hazards where the existing controls were deemed inadequate by the team.
- Determine the likelihood of each risk and the severity of its impact.
- Ensure all relevant hazards raised were considered and addressed in the HAZID study.

The vat leaching system was designed by Indec SA. During the Preliminary Design Phase HAZID study, the GRD Minproc team identified a number of potential hazards, based on its understanding of the Indec design. As the study team was not directly involved in the vat leach design, the HAZID study could not be completed to the same detail as by Indec. Accordingly Indec conducted its own review and produced a separate Hazard Identification report.

The HAZID has not attempted to document “known and well controlled” hazards that are not unique to the process or equipment proposed. Rather, the focus was agreed to be on unique or particularly acute hazards that warrant special attention during the subsequent study and design phases of the Project.

Following consideration of each identified hazard and the current design response to that hazard, the workshop study group determined whether any specific recommendation should be made to either further define the hazard or to apply additional controls to the hazard. Where relevant, recommendations were devised by the workshop study group and recorded.

Seventy-four risks were identified during the Preliminary Design Phase Hazard Identification and Risk Assessment.

Thirty-three actions were logged by the review team to address the potential inadequacies in the existing controls detailed in the design documentation presented for examination. The actions recommended to address perceived inadequacies were determined.

Seven extreme risk topics were identified and recorded during the workshop. These risks are:

- Vehicle collisions

  Vehicle collisions are a significant hazard both with equipment operations within the plant and with other vehicles on and off-site, resulting in potential equipment damage and uncontrolled release of leach solution/acid to the environment. The key risk is the potential for human fatalities.

  This risk will be mitigated by adoption by Marcobre and its contractors of a transport logistics plan and site traffic management plan with relevant procedures, as the project moves into the implementation phase. The site layout and mine-haul and access roads are designed to minimize the risks for collision.
- Seismic events

The region has a high occurrence of seismic events. If the earthquake protection features in the design and construction of pits, dumps, plant buildings and other infrastructure fail, catastrophic consequences in terms of production, environmental and safety impacts could occur. To mitigate this risk, the design criteria are established to comply with all relevant standards for the type of facilities and the location, with input provided from Peruvian consultants that have relevant experience with such events. Mine and plant are designed for a 500-year earthquake occurrence. The plant and building heights are minimized to mitigate potential impacts of earthquakes.

- Ripios disposal system hazards.

  The ripios disposal conveyor system will require frequent relocation of the conveyor belts and the associated power supply. Electrocution hazards are associated with poor earthing due to very low soil moisture, poor isolation and damage to electric cables caused by mobile equipment such as bulldozers and cranes. Poor coordination of the interlocks from vendor packages is a hazard that could result in production interruptions. Personnel fatigue, resulting from working continuous 12 hour production shifts and moving gear on ripios dumps during night shift and under poor lighting, creates the potential for serious safety incidents.

  The ripios disposal system has subsequently been modified and risk mitigated to include mine trucks in place of the movable conveyors system. For existing fixed conveyors, the risks will be mitigated by implementing relevant operation and maintenance procedures, and having a properly trained specialist HV electrician. The design of the conveying system incorporates international standard HV protection systems.

- Acid delivery traffic volume

  Acid transport and delivery will result in high traffic volumes. As with the vehicle collisions risk, this risk will be mitigated through adoption by Marcobre and its contractors of the transport logistics plan, site traffic management plan and relevant procedures during an early stage of the Project.

- Unsafe equipment operation

  Inadequate or poorly enforced site-wide project standards and equipment naming conventions increase the likelihood of incorrect, unsafe operation of equipment with potential significant production and safety impacts resulting from poor isolation. In addition, it may also create the need for holding excessive operating spares.

  This risk will be addressed during the engineering and procurement phases of the project, and by diligent training of personnel.

- SX fire

  A high risk of serious fire in the SX plant and fuel storage area was identified. Water shortages and inadequate training of fire fighters increases the likelihood of fires causing death or serious injuries. Adequate allowances for the fire detection and suppression systems have been made in the study, and correct equipment and materials of construction are selected. This risk will be addressed and details of the system further developed during the engineering phase of the project, and during the training of personnel.

The HAZID study recommended the following:

- All actions raised in the HAZID have been regularly reviewed and will need to be followed up during the implementation phase of the project by the Design Team and Marcobre, to ensure they have been adequately addressed in a timely manner.
- Some of the recommended actions may require further risk assessment by the Project Team to determine the appropriate level of action required to reduce the risk to an acceptable level. These should be identified based on the estimated cost to implement and the level of risk reduction that can be achieved by each action. Only those incurring significant costs to implement warrant the added expense of a formal risk assessment. All low cost actions should be implemented unless it is demonstrated that they will not be effective in reducing the hazard they were intended to control.
- Any subsequent changes made to processes, plant and equipment reviewed in this HAZID must be examined by suitably experienced and qualified members of the Design Team and by Marcobre, to verify that new hazards or operability issues have not been introduced inadvertently by the changes.

18.17.2 Technical risks

BatteryLimits Pty Ltd (BatteryLimits) has conducted a Technical Risk Assessment for the Project at its current level of development.

Thirty-six risks were identified from discussions with the various participants and during the workshop.

The technical risk profile contains a low number (two) of top priority risks. This reflects the considerable amount of work that has gone into the project to date. The key issues identified are:

- The study is at DFS level for the Oxide project and at PFS level for the Sulphide project. The Sulphide project is expected to be a major contributor to project economics, but more work is required to support a DFS-level study.
- Testwork undertaken to date on Sulphide ore does not accurately represent recovery from differing ore grades and ore types.

Proving the viability of the Sulphide project to DFS level is a timing issue and this risk is expected to be mitigated as more data and information are developed and uncertainty is reduced.

A second priority group of risks incorporated issues that need on-going attention from Marcobre, including transport logistics, lease restrictions, port availability and project execution.

Technical and process issues are expected to be resolved as the project progresses.

18.17.3 Opportunities

The following Project opportunities may result in improved economics or reduced technical or commercial risk.

- **Additional ore reserves**

  Two whittle runs were performed to test the potential to expand the reserve, based on the current DFS parameters. These parameters include the selling prices of copper, process recoveries and

concentrate grades and costs associated with the transportation, smelting and refinery of copper, etc.

Table 18.25 shows the results of the two runs and comparison of the results with the DFS case.

- Run 5 - Whittle results used for DFS
- Run 6- DFS parameters with copper price of $1.65/lb
- Run 7- DFS parameters with copper price of $2.00/lb

Table 18.25
Mina justa optimisation using final DFS parameters

Scenario	Oxide Ore				Sulphide Ore			Rec.Metal		Increase	Total
	Feed (kt)	Cu (%)	Cu_SS (%)	Cu Rec t	Feed (kt)	Cu (%)	Cu Rec t	Cu kt	Cu Mlb	Rec. Metal %	Mining kt
Run5	120,136	0.57	0.46	507,938	48,404	1.39	603,013	1,111	2,449		544,213
Run6	141,648	0.53	0.43	563,245	71,424	1.16	762,703	1,326	2,923		703,715
RUN7	171,970	0.49	0.40	630,451	82,965	1.064	810,433	1,441	3,177		795,906
Diff-Run6 & 5	21,512	0.31	0.29	55,307	23,019	0.69	159,690	215	474	19%	159,501
Diff-Run7 & 5	51,834	0.32	0.26	122,513	34,560	0.61	207,420	330	727	30%	251,693

If DFS recoveries and costs are used at the $1.65 Cu price then the potential ore reserves could result in a 19% increase in recovered metal. However, the benefit associated with additional oxide may be reduced due to its lower average grade. The mining (total rock handling) increases by 29%.

For $2.00/lb copper case (with the DFS parameters), further marginal grade material could be added with a potential of 30% more recoverable copper for a 40% increase in total mining.

Additional reserves have not been determined, but the potential to increase the life of the operation constitutes an obvious opportunity to improve project economics.

- Sulphide-Only Option

This option was first conceived when sulphuric acid prices increased sharply in 2007 and 2008, and concerns were expressed about the long-run cost and availability of acid. Since that time, acid prices have returned to more normal levels and long term supply shortages are no longer a concern. Nevertheless, this option has been examined in some detail.

A Sulphide-Only option would have relatively high pre-production stripping costs to develop the sulphide ore at depth. However, this would be offset by the capital cost reduction arising from not having to build the oxide processing facilities. Furthermore, in this case, the oxide material mined to access the sulphide ore would not necessarily be wasted; for a modest additional cost it could be stockpiled for possible future treatment.

A Sulphide-Only mine plan was developed based on the DFS Base Case pit shells but excluding those parts of the deposit such as Magnetite Manto and the Northern Oxides. Mining capital and operating costs were then developed for this mine plan. Capital costs for process and infrastructure were factored from the DFS capital costs, eliminating those portions of the Project

which relate solely to oxide processing. Process and G&A operating costs for the Sulphide-Only case were already available because Phase 3 of the combined oxide/sulphide project is sulphide-only.

The preliminary analysis of the Sulphide-Only case indicates that this alternative is feasible. Optimisation studies will be initiated after completion of the DFS.

- Acid import port change

  While the DFS shows acid imported through San Martin port, 250 km by road from site, the opportunity to import acid from a purpose-built facility at San Nicholas Bay, 30 km from site, is being investigated by Petral S.A.C. Petral estimates possible savings of $7-8 M per year for the first five years of operation, compared to San Martin. Realization of savings after five years arising from the use of the Petral site would, in part, be dependent on the specific volumes of acid that Marcobre contracts with the operators of this site. The impact of possible savings for the first five years of operation would be that the life-of-mine C1 cash cost would decrease from 90.2 cents per payable pound to 88.9 cents, and the after-tax NPV at 8% would increase from $333.0 M to $346.2 M.

- Modified terms and conditions for copper concentrate sales.–

  As discussed in Section 18.14.2.1, the Shareholders' Agreement specifies that 90% of copper concentrates will be sold to LS-Nikko for the first 10 years of copper concentrate production, and of these concentrates, 40% are to have copper treatment charges determined on a price-sharing basis, with a combined TC/RC per payable pound of copper equal to 24% of the copper price during the quotational period for the first 60 months of concentrate production, increasing to 25% during the second 60 months of production, subject to a floor of 19 cents per pound. The Shareholders' Agreement specifies that treatment and refining charges for the other 60% of shipments to LS-Nikko are to be determined annually on the basis of market terms, except that during the first two years of production, price participation will be on the basis of ±10% off a base copper price of $0.90/lb. The DFS Base Case financial projections have been prepared on this basis.

  It is within the remit of Marcobre and its shareholders to agree to alter these terms, and in a polar case, possibly even convert all the price sharing volume to market terms, and to switch price participation for the first two years of concentrate production over to market terms as well.

  Table 18.26 compares the results for the key financial parameters for this "Market Terms" scenario with the DFS Base Case results.

Table 18.26
Marketing scenarios

Case	After-Tax IRR	After-Tax NPV at 8%: $ M	Payback Period: Years	C1 Cash Cost, ¢/lb
DFS Base Case	15.6%	333.0	3.15	90.2
Market Terms Case	16.4%	373.9	3.07	84.8

Using the DFS Base Case copper price of $2.00/lb, the effect of the replacing the price sharing portion of the volume committed to LS-Nikko with market terms, and also adopting market price participation on the market portion, is to reduce the C1 cost per payable pound of copper in

concentrates from $0.697 to $0.600 per pound, reducing the overall C1 cost from $0.885 to $0.831. The effect on the after-tax NPV at 8% is an increase of $40.4 M.

- Definitive Feasibility Study for Sulphide Plant

  Limited and variable test data on ore competency and grindability was available for the sulphide plant PFS. The existing Bond Ball Mill Work Index (BBWi), Bond Rod Mill Work Index (BRWi), Bond Crusher Work Index (CWi), Bond Abrasion Index (Ai), Unconfined Compressive Strength (UCS) and Semi Autogenous Grinding (SAG) specific test database will be expanded during a detailed feasibility study by evaluating core samples taken from the full spatial range of the proposed mining areas. This may indicate a decrease in ore competency, which will impact on the comminution circuit design and operating philosophy i.e. in a reduction in the proposed comminution circuit size.

  The TSF design incorporates separate disposal of rougher and cleaner tailings, based on the acid generation potential of the cleaner tailings. Additional tailings characterisation studies are required to investigate if combined tailings deposition will result in non-acid generating tailings. If this is the case, a simplified tailings deposition system will result in capital, operating and closure cost savings.

- Explore second-hand plant and cancelled orders

  In the current environment, there is likely to be a surplus of second-hand equipment and cancelled orders building up, and there is potential to take advantage of this. This may reduce capital by as much as 20%.

  However, it was not possible to justify incorporation of such assumption into the DFS estimate without making a commitment to secure specific equipment. Therefore, at this stage it only represents an opportunity.

19. INTERPRETATION AND CONCLUSIONS

19.1 GEOLOGY AND RESOURCES

Extensive drilling, geological logging, geological interpretation and assaying has been completed by Marcobre. Snowden has validated the data, and developed a block model incorporating geology, mineralogy and copper, silver and gold information. This model is dated October 2008, and is based on drilling data available as of 23 May 2008. No further modelling has been undertaken since October 2008.

Snowden has reported resources at a range of cut-off grades, classified according to CIM categories
The October 2008 Mina Justa Prospect mineral resource is estimated to contain approximately 5650 M lbs of copper, 45 530 koz of silver and 291 000 oz of gold in the Indicated category, and approximately 1 170 M lbs of copper, 9 430 koz of silver and 148 500 oz of gold in the Inferred category at a 0.3% total copper cut-off grade. Snowden considers that the majority of the resources are of acceptable quality – taking account of data quality, geological interpretation and geological continuity, and grade continuity – to be classified as Indicated and thus suitable for use in a DFS.

Although drill spacing is considered to be sufficently close to allow a proportion of the Indicated resources to be upgraded to Measured status, Snowden believes that expansion of the bulk density database is required before this can be achieved.

19.2 MINING

A DFS-level study of open pit mining of the Mina Justa and Magnetite Manto deposits has been completed, based on Indicated resources in the October 2008 resource model, leading to reporting of Probable Mineral Reserves.

Pit optimisation was based on available process parameters, mining and plant operating cost estimates and metal prices supplied by Marcobre. Subsequently some changes have been made to these inputs based on on-going studies, but sensitivity analysis indicates little impact on final pit shells.

Pit design is based on geotechnical recommendations following geotechnical drilling, logging, mapping, laboratory testwork and modelling. Suitable waste dump areas have been located and dump design supported by geotechnical analysis. Only a very limited proportion of the waste has PAG characteristics.

An equipment fleet, capital and operating costs have been developed from first principles, taking account of final process plant requirements and haulage profiles.

Examination of other production scenarios is being considered, but no further work is required on the DFS Base Case.

19.3 METALLURGICAL INFORMATION AND PROCESS DESIGN

19.3.1 Heap leach, SX/EW

A prolonged period of metallurgical testing has confirmed that vat leaching of the Oxide mineralization provides better economic returns than heap leaching. Testwork has confirmed optimum crush size, leaching cycle and acid addition for a representative range of ore types.

Flowsheet design and equipment specification has been completed to a DFS level of accuracy, and this provides a suitable basis for estimation of capital and operating costs to an accuracy of ±10%.

Leach residues have been analysed, and their characteristics taken into account in the design of the ripios dump.

Testwork and simulation support the design of a conventional SX/EW facility to recover copper from the PLS.

19.3.2 Flotation and flotation product processing

A conventional comminution and flotation circuit has been proposed, supported by PFS-level metallurgical testwork. Additional sampling and testwork are required to optimise the circuit for a full range of material types. Further work is required to define a magnetite separation circuit and confirm that product quality meets customer requirements.

Tailings testwork indicates that the RST are non-acid generating, whereas the CST are potentially acid generating. The design of the TSF is to DFS level, and allows for control of acid seepage from the CST.

19.3.3 Production plan

The production plan balances the plant demand objective of maximising production against mining costs to provide continuous ore supply without excessive stripping and stockpiling of low grade ore. Optimum throughput is 12 Mt/a of leach ore and 5 Mt/a of flotation ore. The start-up date for the Sulphide circuit has been advanced as far as possible to allow early treatment of the more profitable Sulphide ore.

19.4 PLANT DESIGN

Plant design has been completed to a DFS level for the Oxide circuit and to a PFS level for the sulphide circuit. Site investigations support plant locations and foundation designs. The designs allow for the high seismicity of the area.

Full services have been considered, i.e. plant compressed air, potable water, process water, fire water, communications and control systems, and emergency power systems. Specific attention has been given to limiting and managing fire hazards associated with the SX plant.

19.5 GEOTECHNICAL STUDIES

Extensive DFS-level geotechnical investigations have been completed to:

- Review seismic conditions in the area and provide input to structural designs.
- Provide recommendations for pit wall design.
- Confirm safe waste dump design.
- Identify suitable areas and confirm foundation requirements for plant, infrastructure and waste dumps.
- Complete a trade-off study comparing several potential TSF sites in the vicinity, followed by DFS-level design and costing of the tailings dam.
- Determine the site water balance.
- Undertake hydrological and hydrogeological studies of the site.

19.6 WATER SUPPLY

Investigations of the Jahuay aquifer, 31 km southeast of the site, have confirmed the presence of adequate water to supply plant requirements for leaching and flotation. Pump testing has been completed, providing a basis for design of the wellfield.

A pipeline has been designed to supply the site. Surveys have been completed to confirm the route, pumping requirements and the provision of a 22.9 kV powerline to operate the pumps.

Acquisition of the right to withdraw from Jahuay aquifer is a critical step in underpinning the Project.

19.7 WATER MANAGEMENT

Unlike many other sites in Peru, the area is extremely arid, and stormwater management is effectively a non-issue. PMP for a 24-hour event (100-year return period) is conservatively estimated at 17 mm, and no surface run-off is anticipated. The TSF has provision to retain rainfall from a PMP event falling on the TSF basin.

Modelling of rainfall on waste dumps and the ripios dump indicates that there should be no seepage into underlying rock. The water table lies some 450 m below surface and is extremely unlikely to be affected in any way.

19.8 TRANSPORT

The site is well-serviced from Lima by the Pan Americana Sur highway, however there are no rail or scheduled air services.

Bulk transport of acid and mineral products will make use of existing port facilities at San Martin (250 km north) and Matarani (550 km south), until the proposed port at San Juan de Marcona 30 km by road to the southwest, becomes available. The cost savings associated with this move will be very considerable.

19.9 POWER SUPPLY

Total power demand has been estimated at 59 385 kVA when both process circuits are in operation. The site will obtain power from the REP's Marcona substation, 15 km to the southwest. A 220 kV overhead powerline will be constructed to connect the site to the national grid.

Internal power will be distributed around the site and to the Jahuay aquifer by 22.9 kV overhead line, with power cables supplying power within the plant areas.

19.10 ENVIRONMENTAL AND SOCIAL ISSUES

An ESIA is nearing completion. This document records base-line environmental and social conditions as determined by Vector. Potential project impacts have been identified, and mitigated as far as possible.

The Mina Justa site is considered very favourable for mine development, given the arid nature of the environment, the absence of surface or sub-surface water resources and the absence of local inhabitants. Dust (from primary crushers and open pits) appears to be the single largest contaminant likely to escape the immediate project area; strict dust control has been included within the design of the crushers.

Potentially the major impact from the project will be socio-economic, with the project creating a very significant increase in employment opportunities in the district. Marcobre intends to maximize the proportion of local people employed during construction and operations, but it will be necessary to import much of the skilled labour initially.

Marcobre has established a permanent office at San Juan de Marcona town, in order to facilitate liaison with the local population through the exploration phase, promote understanding of the activities involved in exploration and development, and identify local concerns.

19.11 PROJECT IMPLEMENTATION

An aggressive project implementation schedule has been prepared for construction of the Oxide plant over a 30-month period, to be followed by construction of the Sulphide plant over a second 30-month period. Key drivers of the schedules are:

- Very long lead times for some critical equipment items
- Large quantities of concrete required for the vat leach area
- Construction of camp accommodation.

An EPCM approach has been assumed for project construction. In order to achieve the schedules, it is important that the EPCM contract is awarded as soon as possible, followed by detailed engineering, tendering and early award of long-lead items.

Potential risks to the schedule include the potential for delays in obtaining the necessary environmental, access, construction and operating permits and licences.

19.12 FEASIBILITY STUDY RESULTS

19.12.1 Capital costs

Capital cost estimates have been completed to an accuracy of ±10% for the Oxide plant, infrastructure and mine, and ±20% for the Sulphide circuit. Total capital cost (excluding pre-production stripping), in 1Q09 dollars, is estimated to be $546.9 M for the Oxide plant etc, and $168.3 M for the sulphide plant. No contingency has been included.

Additional Sustaining/Deferred capital of $32.9M is required over the life of the project.

Capital costs for the Sulphide circuit will need to be reviewed and revised as appropriate during the DFS for this part of the Project. The capital cost for the Sulphide circuit includes $2.9 m for a DFS.

19.12.2 Operating costs

Total operating costs have been estimated to be $2363 M, including $123 M in State royalties. The on-site component is $1615 M, and off-site realization costs are $748 M. The accuracy of the estimate is as for the capital cost estimate.

Average annual costs per tonne of ore processed range from $12.73/t to $22.90/t, depending on the ore types being processed and the amount of waste stripping.

19.12.3 Cash flow modelling

A cash flow model has been developed by Marcobre, based on the mine and plant production schedules, capital and operating costs, and closure costs. Taxes, royalties and other government charges have been considered, as have working capital requirements.

Key inputs include a copper price of $2.00/lb, an acid price of $45/tonne CIF Peruvian port, electricity price of $55/MWh, and a diesel price of $0.636/l including delivery, storage and dispensing.

An NPV of $333.0 M has been determined at a discount rate of 8%, on a 100% equity, after-tax basis. The IRR is 15.6%, with an after-tax payback of 4.6 years from commencement of cathode production (3.1 years from commencement of concentrate production).

The C1 cash cost per payable pound of copper averages $0.902/lb, after by-product credits.

Sensitivity analysis confirms that the Project is highly sensitive to copper price, and moderately sensitive to capital and operating costs.

20. RECOMMENDATIONS FOR FURTHER WORK

20.1 RESOURCES

There are currently insufficient spatially and geologically representative density data to support a Measured classification for the Mina Justa mineral resource. Snowden recommends that Marcobre collects substantially more density data for both the Mina Justa and Magnetite Manto parts of the deposit. Care should be taken to ensure that samples are representative of all styles of lithology, alteration, weathering, mineralisation and structural intensity (e.g. extremely fractured, brecciated or vuggy material, etc.), and that the density data also be spatially representative of the deposit. Snowden has not seen any QAQC data (standards and duplicate analyses) for the density data. Marcobre is recommended to collect and analyse such data to assess accuracy and precision in its sample-scale density estimates.

In addition to the need for a representative density database, additional infill drilling to drill hole spacings of 25 m (oxidised material), 30 m (chalcopyrite sulphide material) and 35 m (transition and bornite-chalcocite sulphide material) are required to raise the confidence in parts of the deposit.

Marcobre is advised to refine and improve the current domain definition for the Mina Justa deposit through detailed section-by-section geological interpretation. Improved domain definition would ultimately result in increased confidence in grade and tonnage estimates for the deposit.

Snowden recommends that Marcobre conduct a conditional simulation study on the Mina Justa deposit to quantify spatial uncertainty and mining risk associated with the resource model. Such a study is well suited to both long term and short term planning, which will become more important as the work progresses beyond a Feasibility Study level. The simulation study should cover the following:

- Quantification of resource classification risk
- Quantification of mineral reserve risk
- Stockpile planning for high risk areas of the deposit
- Quantification of mill feed variability
- Optimisation of SMU and/or bench height
- Optimisation of grade control configuration.

20.2 MINING

The mining study has been completed to DFS level, but would require revision should the resource model be updated.

A sulphide-only mining schedule has been developed, and other options could be analysed if there are material changes to metal prices, metallurgical recoveries or major cost inputs (acid and diesel).

20.3 METALLURGICAL TESTWORK

Additional metallurgical testwork is required to support the design and costing of the sulphide concentrator at a DFS level. This work would be wide-ranging, including:

- Final definition of comminution characteristics
- Optimisation of sulphide flotation parameters such as grind size, kinetics and reagents for the three ore types. Further attention should be given to the cleaner/recleaner circuit and to the quality of concentrates from each of the three ore types.
- Final definition of tailings characteristics. Tests are required to confirm PAG capacity of CST and combined tails, and the neutralising capacity of RST.

20.4 PLANT ENGINEERING

Once the concentrator flowsheet has been defined, DFS-level plant engineering and capital and operating cost estimation is required.

20.5 GEOTECHNICAL STUDIES

Geotechnical studies have been completed, subject to any changes arising in other areas.

20.6 PORT

Significant operating costs are associated with the use of the more distant ports of San Martin and Matarani during the early years of operation, and there are material cost (and safety) benefits to the early establishment of port facilities at San Juan de Marcona. Marcobre should continue to investigate all avenues to promote and support this development, and continue to investigate other options.

20.7 ESIA AND PERMITTING

The ESIA is in the process of being completed and will be submitted in the coming months.

Assuming that Marcobre decides to develop the Mina Justa Project, the process of obtaining the necessary construction and operating permits and other consents should continue, following the path established by Vector.

Priority areas include acquisition of water rights for the Jahuay aquifer, and establishing surface rights for the water pipeline and power-lines.

20.8 PROJECT IMPLEMENTATION

Strategies to achieve the proposed 30-month implementation schedule for the Oxide plant and infrastructure include early award of the EPCM contract, allowing detained engineering and early ordering of long-lead items such as cone crushers and the construction camp. Marcobre will need to determine whether and when to commence the project and prepare a Request for Tender for the EPCM contract. Alternatively, detailed engineering of long-lead items could commence under Marcobre's direction prior to award of the EPCM contract.

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22. DATE AND SIGNATURE PAGE

Title Page

Project Name: Mina Justa Copper Project

Title: Definitive Feasibility Study Technical Report

Location: Province of Nazca, Peru

Effective Dates:

Effective Date of Technical Report: 8 June 2009

Effective Date of Mineral Reserves: 8 June 2009

Effective Date of Mineral Resources: 8 June 2009

Qualified Persons:

- David D. (Dan) Greig, B.Sc, M.A.I.G. (1684-1722), employed by GRD Minproc Limited as Principal Geologist, was responsible for overall preparation of the report.

  Signed: Date:

  Dan Greig

- Warwick Board, B.Sc (Hons.), M.Sc., Ph.D. Geology, P.Geo. (#31256), P.Geol. (#96031) Aus.I.M.M. (#212259), Pri.Sci.Nat. (#400009/03), employed by Snowden Mining Industry Consultants Inc. as Principal Consultant in the Resource Division, was responsible for the preparation of Sections 14 (Data Verification) and 17.1 and 17.2 (Mineral Resources), and for supervision of the preparation of Sections 7 through 13 and Section 15 of this report.

  Signed: Date:

  Dr. Warwick Board

- Ross Oliver, B.Eng (Mining), Aus.I.M.M. (#105137), employed by GRD Minproc Limited as Manager Mining & Geology, was responsible for the preparation of the whole or parts of Sections 17.1 and 18.1, on Mineral Reserve and Mining.

  Signed: Date:

  Ross Oliver

- Adam Johnston, B.Eng (Extractive Metallurgy), M.Aus.I.M.M (#112641), employed as Chief Metallurgist by Transmin Metallurgical Consultants, was responsible for supervision of Metallurgical Testwork and interpretation of testwork results as reported in Section 16.

  Signed: Date:

  Adam Johnston

- Joseph Schlitt, B.SC. (Met. Eng.), Ph.D., Qualified Professional member of the Mining and Metallurgical Society of America No. 01003QP and a Registered Professional Metallurgical Engineer (Texas No. 53603), President of Hydrometal, Inc., employed as a Metallurgical Consultant on behalf of GRD Minproc Limited, was responsible for Metallurgical Testwork and flowsheet design for the Mina Justa Project Oxide circuit (parts of Section 16 and Section 18).

  Signed: Date:

  Dr. Joseph Sclitt

- Dean David, M.Aus.I.M.M. (#102351) Process Consultant for GRD Minproc Limited, was responsible for Mineral Processing and Metallurgical Testwork of Sulphide ores (Section 16.2), Sulphide plant design (Section 18.3) and for Plant Operating Costs (Section 18.13).

  Signed: Date:

  Dean David

- Daniel Y. Yang, Senior Geotechnical Engineer, M.Eng., P.Eng. 28936, employed by Knight Piésold Ltd. was responsible for geotechnical studies, analysis and conclusions contained in Section 18 (specifically in 18.1.3, 18.2 and 18.7.1), relating to pit slope design criteria, waste dump design, geotechnical site investigations and sourcing of construction materials.

  Signed: Date:

  Daniel Yang

- Thomas F. Kerr, M.Sc., President of Knight Piésold and Co. USA, Registered Professional Engineer (Civil and Geotechnical), P.Eng., in British Columbia (#14906) and Ontario (#90407230) and P.E. in California (#C49260) and Alaska (#10969), was responsible for geotechnical matters related to the design and costing for the Tailings Storage Facility in Section 18.7.2.

  Signed: Date:

  Dr. Thomas Kerr

- Branislav Grbovic, B.Sc, M.Sc, Mech Engineer, M.Aus.I.M.M. (302515), Senior Project & Design Manager, GRD Minproc Limited, was responsible for engineering aspects of the Project as contained in Section 18.

  Signed: Date:

  Branislav Grbovic

- Anthony Sanford, Manager Environmental Services for Vector Perú S.A.C., member of the Geological Society of South Africa, Professional Natural Scientist Registration Nº400089/93, was responsible for those sections of the Technical Report related to environmental and social studies in section 18.

Signed Date

Anthony Sanford

23. ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON PRODUCTION AND DEVELOPMENT PROPERTIES

a) Mining operations

Currently there is no mining operation.

The proposed mining method is by open pit mining at Mina Justa and Magnetite Manto deposits. Further information is contained in section18.1

It is proposed to process Oxide ores by vat leaching and SX/EW recovery to copper cathodes. Sulphide ore will be processed by flotation to produce copper concentrates. Further details of metallurgical processes are disclosed in section 16 and setion 18.3.

Production forecasts are displayed in Table 18.20.

b) Recoverability

Information regarding metallurgical testwork is provided in section 16.

Copper recovery from Oxide ore varies according to grade, but is predicted to average approximately 74.5% of total copper (92% of acid soluble copper) over the life of the mine.

Copper recovery from sulphide ores varies according to ore type. Average recoveries of 855 and 94% are anticipated from Transitional and Primary ores, respectively. For Secondary (Bn-Cc) ores, recoveries are related to head grade, and range up to 96%.

Precious metals recovery to concentrates is expected to be 805 for both silver and gold.

c) Markets

Market information was provided by Marcobre, and is contained in section 18.14.

d) Contracts

At this stage, no contracts are in place regarding development or operation of the Project.

e) Environmental considerations

The environmental conditions relating to the project are the subject of an ESIA, which is nearing completion (section 18.11). No issues have been identified that are likely to cause significant impact on the environment during construction, operations or closure. There are no dwellings within 25 km of the site, but there will be a significant impact on the population in the district which will require management by Marcobre.

Remediation and reclamation is expected to be straight forward, involving covering of a limited amount of potentially acid generating material (CST) in the tailings dam, and a minor quantity of PAG rock waste from the open pits.

No bonds have been posted.

f) Taxes

Marcobre has provided information regarding royalties, taxes and other charges in section 18.16.

g) Capital and operating cost estimates

These have been prepared and reported in sections 18.12 and 18.13. In both cases, accuracy levels are ±10% for the Oxide plant, mine and infrastructure, and ±20% for the Sulphide plant.

Summaries of the capital costs are provided in Table 18.11 and Table 18.12. Oxide plant and infrastructure capital cost is estimated to be $546.9 M, while the Sulphide plant capital is an additional $168.3 M. Deferred and sustaining capital over the Project life is a further $ Table 18.13

Operating costs are summarised in Table 18.16 and range from $12.84 to $22.25 per tonne of ROM ore.

Closure costs are reported in Table 18.17. Closure activities commence in year 8 and continue to year 15, for a total cost of $15.6 M.

h) Economic analysis

Results of Project economic analysis by Marcobre, on a 100% equity basis, are provided in section 18.16, and include cashflow forecasts and sensitivity analyses. Schedules are provided in Appendix 1.

The Base Case indicates an after-tax IRR is 15.6% and the NPV at 8% is $333.0M. The cash break-even copper price (the price at which revenues equal the sum of operating costs, sustaining and deferred capital, and closure costs) is US$1.059 per pound. The economic break-even copper

price (the price at which the 100% equity basis NPV at 8% is equal to zero) is US$1.520 per pound.

On-site operating costs over the life of the project are anticipated to average $0.677/lb of payable copper. Total operating costs (including the mining royalty, transportation, marketing fees, and, in the case of copper concentrates, treatment and refining charges), are anticipated to average $0.978 lb of payable copper. After silver and gold by-product credits of $0.077 per pound of payable copper, the C1 cash cost is estimated at $0.902 per pound. Over the life of the Project, $1 615 M is expected to be paid in on-site operating costs and $748 M in realisation costs.

Sensitivity analysis confirms that the Project is very sensitive to copper price. A 1% change in copper price from the base case $2.00 per pound results in a 4.1% change in after-tax NPV at 8%. The project is less sensitive to capital costs (around a 1.4% change in NPV for a 1% change in capital) and total operating costs, i.e. site and off-site, excluding taxes (1.3% change in NPV for a 1% change in operating costs). With silver and gold credits amounting to only 3.7% of gross revenue, there is very little sensitivity to by-product prices (only a 0.2% change in NPV for a 1% change in by-product prices).

Regarding cost components, acid, diesel and electricity are the largest, representing 21%, 13% and 12% of site costs respectively. A 1% change in acid price results in a 0.4% change in NPV, while a 1% change in diesel prices results in a 0.2% change in NPV. The result for electricity is similar to diesel.

i) Payback

The project is expected to pay back initial capital, including the concentrator project capital, 4.6 years after the commencement of cathode production or 3.1 years after the commencement of concentrate production, determined on an after-tax basis.

j) Mine life

Project operating life is estimated to be approximately 12 years. Both process circuits operate for 10 years, with the introduction of the Sulphide concentrator occurring two years after the Oxide plant.

24. ILLUSTRATIONS

Illustrations have been included in the body of the report. A complete list of drawings is included in the table of contents.

25. ANNEXURES

Printouts from the financial model are included as Appendix 1.

Marcobre S.A.C.
Mina Justa Project: Annual KPI's Schedule

Years ending	Total	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Mine production														
Vat leach oxide ore	114,602	215	10,846	20,157	13,846	7,768	11,329	14,923	10,484	9,109	9,411	5,750	765	-
Sulphide ore	48,794	-	-	11	4,901	5,038	5,062	4,803	5,172	5,024	5,010	5,048	5,632	3,092
Total ore	163,396	215	10,846	20,169	18,747	12,806	16,391	19,726	15,657	14,133	14,420	10,798	6,398	3,092
Waste	402,363	4,171	47,190	39,764	41,726	47,807	43,723	40,669	44,743	46,097	29,740	11,308	4,440	984
Total	565,759	4,386	58,036	59,932	60,474	60,612	60,114	60,395	60,400	60,230	44,160	22,106	10,838	4,075
Rehandle														
Oxide ore	30,235	-	1,197	116	2,240	5,676	1,107	703	3,036	3,304	3,303	6,964	2,589	-
Sulphide ore	2,682	-	-	-	-	-	-	694	117	-	-	-	-	1,871
Total rehandle	32,917	-	1,197	116	2,240	5,676	1,107	1,397	3,153	3,304	3,303	6,964	2,589	1,871
Total moved	598,676	4,386	59,233	60,048	62,714	66,288	61,221	61,792	63,553	63,534	47,463	29,070	13,427	5,946
Strip Ratio	2.46	19.44	4.35	1.97	2.23	3.73	2.67	2.06	2.86	3.26	2.06	1.05	0.69	0.32
Process feed														
Vat leach	114,602	-	3,550	11,699	12,001	11,999	12,001	11,999	12,000	12,000	12,000	12,000	3,355	-
Concentrator	48,794	-	-	-	4,420	5,001	5,001	5,000	5,000	5,000	5,000	5,000	5,000	4,371
Total	163,396	-	3,550	11,699	16,421	17,000	17,001	16,999	17,000	17,000	17,000	17,000	8,355	4,371
Closing Stockpiles														
Vat Leach ore	(0)	215	7,511	15,969	17,815	13,583	12,912	15,835	14,320	11,428	8,839	2,589	(0)	(0)
Sulphide ore	(0)	-	-	11	492	529	590	394	566	590	599	647	1,279	(0)
Total	(0)	215	7,511	15,980	18,307	14,112	13,502	16,229	14,886	12,018	9,438	3,236	1,279	(0)
Process Feed Grades														
Vat leach: Cu_T (%)	0.564%	0.000%	0.549%	0.575%	0.613%	0.560%	0.540%	0.593%	0.592%	0.568%	0.552%	0.513%	0.481%	0.000%
Vat leach: Cu_{SS}/Cu_T (%)	81.0%	0.0%	86.7%	82.2%	77.0%	79.7%	82.7%	77.8%	77.5%	82.2%	85.2%	83.9%	81.2%	0.0%
Concentrator: Cu_T (%)	1.369%	0.000%	0.000%	0.000%	1.366%	1.275%	1.986%	1.996%	1.179%	1.030%	1.042%	1.086%	1.418%	1.304%
Concentrator: Cu_{SS}/Cu_T (%)	12.3%	0.0%	0.0%	0.0%	20.0%	14.9%	10.8%	11.5%	12.5%	13.0%	11.6%	10.0%	9.1%	11.7%
Concentrator: Au (g/t)	0.03	-	-	-	0.02	0.02	0.03	0.03	0.02	0.02	0.03	0.04	0.05	0.04
Concentrator: Ag (g/t)	14.12	-	-	-	10.95	11.18	20.00	22.86	12.43	11.23	11.27	10.24	15.21	15.69
Concentrator: Magnetite (%)	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%
Vat Leach and Concentrator: Cu_T (%)	0.804%	0.000%	0.549%	0.575%	0.816%	0.770%	0.965%	1.006%	0.765%	0.704%	0.696%	0.681%	1.041%	1.304%
Contained Metal in Process Feed														
Vat leach: Cu_T (000 lbs)	1,425,179	-	42,955	148,327	162,121	148,140	142,780	156,820	156,550	150,260	146,041	135,624	35,560	-
Vat leach: Cu_{SS} (000 lbs)	1,154,083	-	37,249	121,920	124,781	118,094	118,008	122,084	121,339	123,451	124,454	113,826	28,878	-
Concentrator: Cu_T (000 lbs)	1,472,745	-	-	-	133,137	140,594	218,950	220,034	130,002	113,523	114,894	119,690	156,270	125,652
Concentrator: Cu_{SS} (000 lbs)	181,815	-	-	-	26,650	20,880	23,657	25,315	16,299	14,714	13,384	11,950	14,277	14,690
Concentrator: Au (000 oz)	46.222	-	-	-	2.239	2.827	4.254	5.406	4.010	3.455	4.338	6.226	7.973	5.495
Concentrator: Ag (000 oz)	22,157	-	-	-	1,556.082	1,797.293	3,215.345	3,674.456	1,999.041	1,805.907	1,811.877	1,646.689	2,445.843	2,204.098
Concentrator: Magnetite (000 dmt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Recoveries														
Vat leach: Cu %	74.5%	0.0%	79.0%	75.4%	70.7%	72.9%	75.7%	71.8%	71.9%	76.3%	78.5%	77.0%	74.0%	0.0%
Concentrator: Cu %	93.0%	0.0%	0.0%	0.0%	91.0%	91.9%	94.7%	94.4%	92.4%	91.6%	92.1%	92.4%	93.9%	93.3%
Concentrator: Au (g/t)	80.0%	0.0%	0.0%	0.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Concentrator: Ag (g/t)	80.0%	0.0%	0.0%	0.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Concentrator: Magnetite (%)	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%
Production														
Cu in cathodes (t)	481,596	-	15,390	50,708	51,995	48,989	49,001	51,075	51,079	52,000	52,000	47,418	11,940	-
Cu in cathodes (000 lbs)	1,061,736	-	33,930	111,793	114,629	108,002	108,030	112,602	112,609	114,640	114,640	104,538	26,324	-
Cu in Cu concentrates (000 payable lbs)	1,322,669	-	-	-	117,084	124,879	200,535	200,527	115,925	100,439	102,018	106,614	141,564	113,084
Total Payable Cu (000 payable lbs)	2,384,405	-	33,930	111,793	231,714	232,881	308,565	313,128	228,534	215,079	216,658	211,153	167,887	113,084
Cu Concentrates (dmt)	1,643,741	-	-	-	142,598	141,683	214,802	245,863	151,612	125,671	137,575	150,551	187,748	145,638

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Years ending	Total	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Concentrate Grades														
Cu (%)	37.8%	-	-	-	38.5%	41.4%	43.8%	38.3%	35.9%	37.5%	34.9%	33.3%	35.5%	36.5%
Au (g/t)	0.70	-	-	-	0.4	0.5	0.5	0.5	0.7	0.7	0.8	1.0	1.1	0.9
Ag (g/t)	335.4	-	-	-	271.5	315.6	372.5	371.9	328.1	357.6	327.7	272.2	324.2	376.6
Contained metal in Cu concentrates														
Cu (t)	621,373	-	-	-	54,947	58,595	94,069	94,189	54,499	47,188	47,979	50,181	66,577	53,148
Au (000 oz)	37.0	-	-	-	1.8	2.3	3.4	4.3	3.2	2.8	3.5	5.0	6.4	4.4
Ag (000 oz)	17,725	-	-	-	1,245	1,438	2,572	2,940	1,599	1,445	1,450	1,317	1,957	1,763
Payable metal in Cu concentrates														
Cu (t)	599,953	-	-	-	53,109	56,644	90,961	90,957	52,583	45,558	46,275	48,359	64,212	51,294
Au (000 payable oz)	11.636	-	-	-	-	-	-	-	-	-	1.202	3.767	4.486	2.182
Ag (000 payable oz)	15,953	-	-	-	1,120	1,294	2,315	2,646	1,439	1,300	1,305	1,186	1,761	1,587
Total Payable Cu Production (t)	1,081,549	-	15,390	50,708	105,104	105,633	139,963	142,033	103,661	97,558	98,275	95,777	76,153	51,294
Magnetite Concentrates (dmt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Key Input Quantities														
Sulphuric Acid (t)	4,659,953	-	142,260	471,022	483,920	489,009	489,467	487,164	489,459	487,748	485,182	494,197	140,525	-
Diesel (000 litres)	338,563	2,544	30,410	35,069	32,966	33,641	35,168	36,972	35,805	35,541	28,341	17,397	9,851	4,857
Power (MWh)	3,655,892	-	59,182	191,741	361,557	374,218	376,124	381,586	379,267	380,737	381,021	370,701	235,904	163,853
Sales														
Cu in cathodes (000 lbs)	1,061,736	-	26,455	112,436	112,436	112,436	105,822	112,436	112,436	112,436	119,049	105,822	29,974	-
Cu in Cu concentrates (000 payable lbs)	1,322,525	-	-	-	108,664	115,464	206,098	202,206	107,947	103,670	103,050	100,280	143,293	131,854
Total Payable Cu (000 payable lbs)	2,384,261	-	26,455	112,436	221,100	227,900	311,919	314,641	220,382	216,106	222,099	206,102	173,267	131,854
Au in Cu concentrates (000 payable oz)	11	-	-	-	-	-	-	-	-	-	1	3	4	3
Ag in Cu concentrates (000 payable oz)	15,953	-	-	-	1,027	1,200	2,382	2,632	1,360	1,318	1,317	1,154	1,732	1,831
Cu Concentrates (dmt)	1,643,741	-	-	-	130,130	130,130	220,220	250,250	140,140	130,130	140,140	140,140	190,190	172,271
Magnetite Concentrates (dmt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Product prices (Q1 2009 US$)														
Cu, $/lb	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00
Au, $/oz	700	700	700	700	700	700	700	700	700	700	700	700	700	700
Ag, $/oz	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00	11.00
Magnetite Concentrates, $/dmt	-	10.50	10.50	10.50	10.50	10.50	10.50	10.50	10.50	10.50	10.50	10.50	10.50	10.50

Years ending	Total	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Gross Revenue														
Copper cathodes														
LME Grade A Cathode	2,123,473	-	52,911	224,871	224,871	224,871	211,644	224,871	224,871	224,871	238,099	211,644	59,948	-
Cathode Premium	36,345	-	895	3,847	3,847	3,847	3,638	3,847	3,847	3,847	4,055	3,638	1,037	-
Marketing fee	(6,213)	-	(155)	(658)	(658)	(658)	(619)	(658)	(658)	(658)	(697)	(619)	(175)	-
Gross revenue	2,153,605	-	53,651	228,060	228,060	228,060	214,663	228,060	228,060	228,060	241,458	214,663	60,810	-
Ocean freight	(28,366)	-	(703)	(3,003)	(3,003)	(3,003)	(2,833)	(3,003)	(3,003)	(3,003)	(3,174)	(2,833)	(805)	-
Insurance	(3,259)	-	(81)	(346)	(346)	(345)	(325)	(345)	(345)	(345)	(365)	(325)	(92)	-
Net smelter return	2,121,979	-	52,866	224,711	224,711	224,712	211,505	224,712	224,712	224,712	237,919	211,505	59,914	-
Copper concentrates														
Cu	2,645,050	-	-	-	217,329	230,928	412,195	404,412	215,894	207,340	206,099	200,560	286,585	263,708
Au	7,618	-	-	-	-	-	-	-	-	-	407	2,192	3,058	1,961
Ag	175,480	-	-	-	11,302	13,196	26,203	28,948	14,965	14,501	14,484	12,692	19,049	20,141
Gross revenue	2,828,148	-	-	-	228,631	244,124	438,398	433,360	230,858	221,841	220,990	215,444	308,693	285,809
Cu Treatment and Refining Charges	(392,681)	-	-	-	(32,329)	(33,883)	(58,351)	(60,192)	(31,622)	(31,784)	(32,114)	(31,428)	(43,633)	(37,345)
Cu Price Participation	(42,803)	-	-	-	(6,606)	(7,380)	(5,406)	(5,153)	(2,890)	(2,634)	(2,622)	(2,555)	(3,742)	(3,815)
Total Cu Smelter Charges	(435,484)	-	-	-	(38,935)	(41,263)	(63,757)	(65,345)	(34,512)	(34,417)	(34,736)	(33,982)	(47,375)	(41,160)
Au Refining Charge	(54)	-	-	-	-	-	-	-	-	-	(3)	(16)	(22)	(14)
Au Refining Charge	(5,583)	-	-	-	(360)	(420)	(834)	(921)	(476)	(461)	(461)	(404)	(606)	(641)
Gross revenue less treatment and refining charges	2,387,027	-	-	-	189,336	202,441	373,807	367,094	195,870	186,962	185,790	181,042	260,689	243,995
Ocean freight	(82,187)	-	-	-	(6,507)	(6,507)	(11,011)	(12,513)	(7,007)	(6,507)	(7,007)	(7,007)	(9,510)	(8,614)
Insurance	(3,610)	-	-	-	(286)	(306)	(565)	(555)	(296)	(283)	(281)	(274)	(394)	(369)
Net smelter return	2,301,229	-	-	-	182,543	195,628	362,231	354,026	188,567	180,173	178,502	173,762	250,786	235,012
Total products														
Gross revenue	4,981,753	-	53,651	228,060	456,691	472,184	653,060	661,420	458,919	449,901	462,448	430,107	369,503	285,809
Realization costs	(558,545)	-	(785)	(3,349)	(49,437)	(51,844)	(79,324)	(82,682)	(45,639)	(45,016)	(46,027)	(44,840)	(58,804)	(50,797)
Net smelter return	4,423,208	-	52,866	224,711	407,254	420,340	573,736	578,738	413,279	404,885	416,421	385,267	310,699	235,012
Total Copper Unit Cost Calculations														
Nominal terms, $ 000														
Ore mining	180,460	287	10,909	21,952	19,161	12,541	17,179	21,713	16,374	14,659	16,225	13,026	10,235	6,198
Waste mining	423,488	5,678	47,224	43,100	41,758	46,806	45,693	44,673	46,373	47,648	32,728	13,641	6,193	1,972
Rehandle	18,195	-	618	61	1,152	2,781	556	709	1,569	1,643	1,870	4,423	1,680	1,133
Less: capitalized preproduction stripping	(28,885)	(5,678)	(23,207)	-	-	-	-	-	-	-	-	-	-	-
Total mining	593,258	287	35,545	65,113	62,072	62,129	63,428	67,095	64,316	63,950	50,823	31,090	18,108	9,303
Processing	808,801	-	20,069	65,120	88,046	90,465	92,461	78,748	78,366	77,473	79,574	76,700	40,657	21,122
Site G&A	191,919	-	7,286	14,568	17,744	17,744	17,756	17,753	17,753	17,753	17,672	17,567	15,571	12,754
Lima G&A	20,634	-	904	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,657
Land transport and port	66,209	-	582	1,916	9,371	6,130	8,338	8,445	5,599	4,833	5,193	5,494	5,908	4,398
Subtotal	1,680,821	287	64,385	148,524	179,040	178,275	183,790	173,848	167,841	165,816	155,070	132,658	82,051	49,234
Insurance, shiploading, ocean freight & FTT	119,160	-	892	3,555	10,314	10,332	14,914	16,590	10,812	10,296	10,974	10,563	10,876	9,029
Treatment and refining costs	441,122	-	-	-	39,295	41,683	64,591	66,266	34,988	34,879	35,200	34,402	48,003	41,815
Marketing fee less cathode premium	(30,132)	-	(740)	(3,189)	(3,189)	(3,189)	(3,019)	(3,189)	(3,189)	(3,189)	(3,359)	(3,019)	(862)	-
Mining royalty	122,210	-	517	4,788	11,220	11,781	17,009	17,201	11,384	11,152	11,492	10,535	8,758	6,373
Subtotal	2,333,181	287	65,054	153,679	236,680	238,883	277,286	270,717	221,836	218,954	209,377	185,139	148,827	106,451
By-product credits														
Au	7,618	-	-	-	-	-	-	-	-	-	407	2,192	3,058	1,961
Ag	175,480	-	-	-	11,302	13,196	26,203	28,948	14,965	14,501	14,484	12,692	19,049	20,141
Magnetite	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	183,098	-	-	-	11,302	13,196	26,203	28,948	14,965	14,501	14,891	14,884	22,107	22,102
C1 Costs	2,150,082	287	65,054	153,679	225,378	225,687	251,083	241,768	206,872	204,453	194,486	170,255	126,720	84,349
Depreciation, amortization and other non-cash charges	886,727	-	12,891	56,939	78,895	92,225	95,711	90,071	89,432	94,893	99,944	100,667	52,466	26,029
C2 Costs	3,036,810	287	77,945	210,618	304,273	317,911	346,794	331,839	296,304	299,346	294,430	270,922	179,186	110,378
Interest	-	-	-	-	-	-	-	-	-	-	-	-	-	-
C3 Costs	3,036,810	287	77,945	210,618	304,273	317,911	346,794	331,839	296,304	299,346	294,430	270,922	179,186	110,378

Years ending	**Total**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**
Deflator	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Real terms, $000														
Ore mining	180,460	287	10,909	21,952	19,161	12,541	17,179	21,713	16,374	14,659	16,225	13,026	10,235	6,198
Waste mining	423,488	5,678	47,224	43,100	41,758	46,806	45,693	44,673	46,373	47,648	32,728	13,641	6,193	1,972
Rehandle	18,195	-	618	61	1,152	2,781	556	709	1,569	1,643	1,870	4,423	1,680	1,133
Less: capitalized preproduction stripping	(28,885)	(5,678)	(23,207)	-	-	-	-	-	-	-	-	-	-	-
Total mining	593,258	287	35,545	65,113	62,072	62,129	63,428	67,095	64,316	63,950	50,823	31,090	18,108	9,303
Processing	808,801	-	20,069	65,120	88,046	90,465	92,461	78,748	78,366	77,473	79,574	76,700	40,657	21,122
Site G&A	191,919	-	7,286	14,568	17,744	17,744	17,756	17,753	17,753	17,753	17,672	17,567	15,571	12,754
Lima G&A	20,634	-	904	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,807	1,657
Land transport and port	66,209	-	582	1,916	9,371	6,130	8,338	8,445	5,599	4,833	5,193	5,494	5,908	4,398
Subtotal	1,680,821	287	64,385	148,524	179,040	178,275	183,790	173,848	167,841	165,816	155,070	132,658	82,051	49,234
Insurance, shiploading, ocean freight & FTT	119,160	-	892	3,555	10,314	10,332	14,914	16,590	10,812	10,296	10,974	10,563	10,876	9,029
Treatment and refining costs	441,122	-	-	-	39,295	41,683	64,591	66,266	34,988	34,879	35,200	34,402	48,003	41,815
Marketing fee less cathode premium	(30,132)	-	(740)	(3,189)	(3,189)	(3,189)	(3,019)	(3,189)	(3,189)	(3,189)	(3,359)	(3,019)	(862)	-
Mining royalty	122,210	-	517	4,788	11,220	11,781	17,009	17,201	11,384	11,152	11,492	10,535	8,758	6,373
Subtotal	2,333,181	287	65,054	153,679	236,680	238,883	277,286	270,717	221,836	218,954	209,377	185,139	148,827	106,451
By-product credits														
Au	7,618	-	-	-	-	-	-	-	-	-	407	2,192	3,058	1,961
Ag	175,480	-	-	-	11,302	13,196	26,203	28,948	14,965	14,501	14,484	12,692	19,049	20,141
Magnetite	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	183,098	-	-	-	11,302	13,196	26,203	28,948	14,965	14,501	14,891	14,884	22,107	22,102
C1 Costs	2,150,082	287	65,054	153,679	225,378	225,687	251,083	241,768	206,872	204,453	194,486	170,255	126,720	84,349
Depreciation, amortization and other non-cash charges	886,727	-	12,891	56,939	78,895	92,225	95,711	90,071	89,432	94,893	99,944	100,667	52,466	26,029
C2 Costs	3,036,810	287	77,945	210,618	304,273	317,911	346,794	331,839	296,304	299,346	294,430	270,922	179,186	110,378
Interest	-	-	-	-	-	-	-	-	-	-	-	-	-	-
C3 Costs	3,036,810	287	77,945	210,618	304,273	317,911	346,794	331,839	296,304	299,346	294,430	270,922	179,186	110,378
Total Copper Unit Costs														
(Q1 2009 US Cents/payable lb Cu)														
Ore mining	7.6	-	32.2	19.6	8.3	5.4	5.6	6.9	7.2	6.8	7.5	6.2	6.1	5.5
Waste mining	17.5	-	139.2	38.6	18.0	20.1	14.8	14.3	20.3	22.2	15.1	6.5	3.7	1.7
Rehandle	0.8	-	1.8	0.1	0.5	1.2	0.2	0.2	0.7	0.8	0.9	2.1	1.0	1.0
Less: capitalized preproduction stripping	(1.0)	-	(68.4)	-	-	-	-	-	-	-	-	-	-	-
Total mining	24.9	-	104.8	58.2	26.8	26.7	20.6	21.4	28.1	29.7	23.5	14.7	10.8	8.2
Processing	33.9	-	59.1	58.3	38.0	38.8	30.0	25.1	34.3	36.0	36.7	36.3	24.2	18.7
Site G&A	8.0	-	21.5	13.0	7.7	7.6	5.8	5.7	7.8	8.3	8.2	8.3	9.3	11.3
Lima G&A	0.9	-	2.7	1.6	0.8	0.8	0.6	0.6	0.8	0.8	0.8	0.9	1.1	1.5
Land transport and port	2.8	-	1.7	1.7	4.0	2.6	2.7	2.7	2.5	2.2	2.4	2.6	3.5	3.9
Subtotal	70.5	-	189.8	132.9	77.3	76.6	59.6	55.5	73.4	77.1	71.6	62.8	48.9	43.5
Insurance, shiploading, ocean freight & FTT	5.0	-	2.6	3.2	4.5	4.4	4.8	5.3	4.7	4.8	5.1	5.0	6.5	8.0
Treatment and refining costs	18.5	-	-	-	17.0	17.9	20.9	21.2	15.3	16.2	16.2	16.3	28.6	37.0
Marketing fee less cathode premium	(1.3)	-	(2.2)	(2.9)	(1.4)	(1.4)	(1.0)	(1.0)	(1.4)	(1.5)	(1.6)	(1.4)	(0.5)	-
Mining royalty	5.1	-	1.5	4.3	4.8	5.1	5.5	5.5	5.0	5.2	5.3	5.0	5.2	5.6
Subtotal	97.8	-	191.7	137.5	102.1	102.6	89.9	86.5	97.1	101.8	96.6	87.7	88.6	94.1
By-product credits														
Au	0.3	-	-	-	-	-	-	-	-	-	0.2	1.0	1.8	1.7
Ag	7.4	-	-	-	4.9	5.7	8.5	9.2	6.5	6.7	6.7	6.0	11.3	17.8
Magnetite	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	7.7	-	-	-	4.9	5.7	8.5	9.2	6.5	6.7	6.9	7.0	13.2	19.5
C1 Costs	90.2	-	191.7	137.5	97.3	96.9	81.4	77.2	90.5	95.1	89.8	80.6	75.5	74.6
Depreciation, amortization and other non-cash charges	37.3	-	38.0	50.9	34.0	39.6	31.0	28.8	39.1	44.1	46.1	47.7	31.3	23.0
C2 Costs	127.5	-	229.7	188.4	131.3	136.5	112.4	106.0	129.7	139.2	135.9	128.3	106.7	97.6
Interest	-	-	-	-	-	-	-	-	-	-	-	-	-	-
C3 Costs	127.5	-	229.7	188.4	131.3	136.5	112.4	106.0	129.7	139.2	135.9	128.3	106.7	97.6

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	Total	31-Dec-09	31-Mar-10	30-Jun-10	30-Sep-10	31-Dec-10	31-Mar-11	30-Jun-11	30-Sep-11	31-Dec-11	31-Mar-12	30-Jun-12	30-Sep-12	31-Dec-12
Gross revenue	4,981,753	-	-	-	-	-	-	-	-	-	-	-	-	53,651
Realization costs	(558,545)	-	-	-	-	-	-	-	-	-	-	-	-	(785)
Net revenue	4,423,208	-	-	-	-	-	-	-	-	-	-	-	-	52,866.1
Cash production cost of sales	(1,680,820)	-	-	-	-	-	-	-	-	-	-	-	-	(28,762)
Operating margin	2,742,388	-	-	-	-	-	-	-	-	-	-	-	-	24,104
Interest expense	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Interest income	3,617	-	-	-	-	-	-	-	-	-	-	13	13	13
Sub-total	2,746,004	-	-	-	-	-	-	-	-	-	-	13	13	24,117
Working capital														
Decrease (increase) in current assets														
Cash and equivalents	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Restricted funds	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Accounts receivable	0	-	-	-	-	-	-	-	-	-	-	-	-	(22,723)
IGV recoverable	9,593	(918)	(1,582)	(6,965)	(9,526)	(3,080)	(414)	(41)	(1,322)	(14,853)	(10,157)	12,046	9,042	9,433
Interest receivable	-	-	-	-	-	-	-	-	-	-	-	(13)	(0)	-
Supplies	-	-	-	-	-	-	-	-	(32)	(1,649)	(2,221)	(662)	(2,490)	(2,580)
Finished product inventory (cash)	(0)	-	-	-	-	-	-	-	-	-	-	-	(9,761)	1,635
Work in process inventory (cash)	(0)	-	-	-	-	-	-	-	-	-	-	-	(0)	0
Deposits, prepaids, others	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Increase (decrease) in current liabilities	-													
Accrued charges	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Accounts payable	-	-	-	-	-	-	-	-	32	1,765	2,253	671	2,701	3,067
IGV payable	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Decrease (increase) in working capital	9,592	(918)	(1,582)	(6,965)	(9,526)	(3,080)	(414)	(41)	(1,322)	(14,737)	(10,126)	12,042	(509)	(11,168)
Decrease (increase) in ore in stockpile	0	-	-	-	-	-	-	-	-	(287)	(1,782)	(2,539)	(15,425)	(7,750)
Fixed capital														
Mine development - cash	(28,885)	-	-	-	-	-	-	-	(97)	(5,581)	(10,952)	(12,255)	-	-
Exploration and Feasibility Study	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Oxide project	(546,866)	(4,832)	(8,328)	(39,659)	(49,065)	(51,799)	(51,600)	(51,706)	(57,099)	(108,490)	(83,175)	(27,587)	(9,785)	(3,741)
Sulphide project	(168,307)	-	-	-	(1,073)	(1,073)	(715)	(1,383)	(2,075)	(17,621)	(17,621)	(21,282)	(21,929)	(18,240)
Sustaining capital	(32,855)	-	-	-	-	-	-	-	-	-	-	-	-	-
Fixed capital	(776,913)	(4,832)	(8,328)	(39,659)	(50,138)	(52,872)	(52,315)	(53,089)	(59,270)	(131,693)	(111,749)	(61,124)	(31,715)	(21,981)
Closure costs	(15,601)	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine closure guarantee security	-	-	-	-	-	-	-	-	-	-	(1,109)	-	-	-
Pre-finance, pre-tax cash flow	1,963,083	(5,750)	(9,910)	(46,625)	(59,664)	(55,952)	(52,729)	(53,131)	(60,592)	(146,717)	(124,765)	(51,608)	(47,635)	(16,783)
Tax payments														
Mining royalty	(122,210)	-	-	-	-	-	-	-	-	-	-	-	-	(517)
Employee profit sharing	(139,721)	-	-	-	-	-	-	-	-	-	-	-	-	-
Income tax and Net Asset Tax (ITAN)	(514,174)	-	-	-	-	-	-	-	-	-	-	-	-	-
Financial Transactions Tax (ITF)	(2,362)	(6)	(8)	(37)	(50)	(53)	(52)	(53)	(59)	(131)	(113)	(63)	(53)	(54)
Interest withholding tax	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	(42,581)	-	-	-	-	-	-	-	-	-	-	-	-	-
Total taxes	(821,048)	(6)	(8)	(37)	(50)	(53)	(52)	(53)	(59)	(131)	(113)	(63)	(53)	(571)
Pre-finance, post-tax cash flow	1,142,035	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
Senior Loans														
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Senior Loan Payments	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	1,142,035	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	Total	31-Dec-09	31-Mar-10	30-Jun-10	30-Sep-10	31-Dec-10	31-Mar-11	30-Jun-11	30-Sep-11	31-Dec-11	31-Mar-12	30-Jun-12	30-Sep-12	31-Dec-12
Other Loans														
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Finance Payments	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	1,142,035	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
Equity Cash Flow	0													
Shareholder loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Equity contributions	(766,545)	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
Dividends	1,038,549	-	-	-	-	-	-	-	-	-	-	-	-	-
Return of capital	870,031	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Equity Cash Flow	1,142,035	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
Check	0	-	-	-	-	-	-	-	-	-	-	-	-	-
Deflator, October 1, 2009=1.000		1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	1,142,035	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
Equity IRR, deflated		15.6%	15.7%	16.1%	17.7%	20.1%	22.7%	25.6%	29.0%	33.7%	51.9%	95.1%	152.2%	430.6%
Equity NPV ($000)														
7.0%		400,050	412,679	429,721	484,109	552,589	618,496	682,276	747,550	821,468	983,576	1,126,293	1,197,616	1,266,139
8.0%		333,030	345,311	362,034	416,179	484,555	550,515	614,501	680,136	754,586	917,513	1,061,422	1,134,212	1,204,395
9.0%		272,712	284,469	300,690	354,405	422,484	488,297	552,283	618,071	692,839	856,366	1,001,247	1,075,283	1,146,906
10.0%		218,376	229,466	245,037	298,166	365,786	431,282	495,096	560,855	635,758	799,696	945,354	1,020,440	1,093,307
11.0%		169,388	179,697	194,496	246,911	313,936	378,974	442,466	508,044	582,921	747,105	893,371	969,336	1,043,273
12.0%		125,190	134,625	148,554	200,150	266,468	330,929	393,974	459,238	533,951	698,239	844,964	921,654	996,509
Pre-tax Equity Cash Flow														
After-tax equity cash flow	1,142,035	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
Income tax	514,174	-	-	-	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	42,581	-	-	-	-	-	-	-	-	-	-	-	-	-
Pre-tax equity cash flow	1,698,789	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
After tax equity cash flow		1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	1,698,789	(5,755)	(9,919)	(46,661)	(59,714)	(56,005)	(52,781)	(53,184)	(60,651)	(146,848)	(124,878)	(51,671)	(47,689)	(17,354)
Equity IRR, deflated		20.2%	20.4%	20.8%	22.6%	25.3%	28.1%	31.3%	34.9%	39.9%	59.4%	105.1%	165.8%	473.2%
Equity NPV ($000)														
7.0%		700,136	717,884	740,133	799,816	873,681	945,065	1,014,416	1,085,356	1,165,037	1,333,005	1,481,683	1,559,068	1,633,757
8.0%		609,174	626,820	649,011	708,731	782,791	854,544	924,436	996,093	1,076,681	1,245,864	1,396,152	1,475,445	1,552,258
9.0%		527,129	544,426	566,309	625,808	699,799	771,650	841,808	913,901	995,112	1,165,222	1,316,829	1,397,738	1,476,383
10.0%		453,049	469,798	491,164	550,228	623,926	695,647	765,836	838,124	919,712	1,090,498	1,243,168	1,325,436	1,405,657
11.0%		386,097	402,134	422,812	481,263	554,483	625,880	695,898	768,174	849,928	1,021,170	1,174,680	1,258,081	1,339,651
12.0%		325,533	340,725	360,577	418,266	490,852	561,761	631,439	703,527	785,261	956,770	1,110,925	1,195,258	1,277,976

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-13	30-Jun-13	30-Sep-13	31-Dec-13	31-Mar-14	30-Jun-14	30-Sep-14	31-Dec-14	31-Mar-15	30-Jun-15	30-Sep-15	31-Dec-15
Gross revenue	53,651	53,651	67,048	53,711	71,894	119,732	125,878	139,188	124,574	131,122	103,448	113,040
Realization costs	(785)	(785)	(976)	(804)	(4,555)	(11,979)	(14,173)	(18,730)	(12,667)	(16,475)	(9,947)	(12,755)
Net revenue	52,866.1	52,866.1	66,072.6	52,906.2	67,339.0	107,753.1	111,704.4	120,457.6	111,907.0	114,647.5	93,500.3	100,285.1
Cash production cost of sales	(28,981)	(28,998)	(35,479)	(27,667)	(30,762)	(44,548)	(40,525)	(45,112)	(45,649)	(45,240)	(43,988)	(46,752)
Operating margin	23,885	23,868	30,594	25,240	36,577	63,205	71,180	75,346	66,258	69,407	49,512	53,534
Interest expense	-	-	-	-	-	-	-	-	-	-	-	-
Interest income	13	25	25	25	24	37	37	37	36	49	49	49
Sub-total	23,897	23,893	30,619	25,265	36,602	63,242	71,217	75,383	66,294	69,456	49,562	53,583
Working capital												
Decrease (increase) in current assets												
Cash and equivalents	-	-	-	-	-	-	-	-	-	-	-	-
Restricted funds	-	-	-	-	-	-	-	-	-	-	-	-
Accounts receivable	(0)	-	(5,700)	5,775	(6,406)	(17,626)	(6,084)	473	3,975	(1,298)	4,461	1,814
IGV recoverable	6,767	(422)	2,543	2,136	5,701	13,113	-	-	(882)	116	766	-
Interest receivable	0	(12)	(0)	-	1	(12)	(0)	-	1	(12)	(1)	-
Supplies	(508)	(237)	4	(119)	(927)	367	(41)	(536)	278	(323)	171	252
Finished product inventory (cash)	(285)	(1,497)	4,941	(2,177)	(6,076)	3,404	(2,629)	(1,036)	1,525	(1,026)	(2,586)	837
Work in process inventory (cash)	(0)	0	(6)	6	(2)	1	2	(0)	(0)	(0)	(0)	(0)
Deposits, prepaids, others	-	-	-	-	-	-	-	-	-	-	-	-
Increase (decrease) in current liabilities												
Accrued charges	-	-	-	-	-	-	-	-	-	-	-	-
Accounts payable	576	263	12	119	1,569	(181)	104	536	(238)	281	(214)	(277)
IGV payable	-	-	-	-	-	278	4,063	(3,065)	(1,276)	-	18	375
Decrease (increase) in working capital	6,551	(1,905)	1,793	5,740	(6,140)	(655)	(4,586)	(3,628)	3,383	(2,263)	2,616	3,002
Decrease (increase) in ore in stockpile	(6,663)	(6,625)	(7,010)	(8,083)	(6,443)	(2,799)	(1,917)	(598)	8	1,012	1,689	1,895
Fixed capital												
Mine development - cash	-	-	-	-	-	-	-	-	-	-	-	-
Exploration and Feasibility Study	-	-	-	-	-	-	-	-	-	-	-	-
Oxide project	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide project	(17,825)	(18,694)	(20,169)	(8,606)	-	-	-	-	-	-	-	-
Sustaining capital	-	(4,091)	-	-	(4)	(478)	(1,258)	(805)	(1,251)	(805)	(1,508)	(255)
Fixed capital	(17,825)	(22,785)	(20,169)	(8,606)	(4)	(478)	(1,258)	(805)	(1,251)	(805)	(1,508)	(255)
Closure costs	-	-	-	-	-	-	-	-	-	-	-	-
Mine closure guarantee security	(1,055)	-	-	-	(1,055)	-	-	-	(1,055)	-	-	-
Pre-finance, pre-tax cash flow	4,904	(7,422)	5,232	14,316	22,960	59,310	63,455	70,352	67,378	67,401	52,358	58,225
Tax payments												
Mining royalty	(517)	(950)	(1,773)	(1,548)	(766)	(2,901)	(3,586)	(3,967)	(1,788)	(3,771)	(2,968)	(3,254)
Employee profit sharing	-	-	-	-	-	-	-	-	-	(3,168)	-	-
Income tax and Net Asset Tax (ITAN)	-	(1,112)	(1,112)	(1,112)	(1,315)	86	(3,320)	(3,495)	(3,228)	(4,610)	(3,718)	(3,917)
Financial Transactions Tax (ITF)	(51)	(57)	(55)	(44)	(40)	(42)	(44)	(46)	(43)	(44)	(44)	(41)
Interest withholding tax	-	-	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	(171)	-	(90)	(457)	(821)	(1,184)	(1,160)	(1,232)	(1,012)	(1,049)	(603)	(643)
Total taxes	(738)	(2,118)	(3,030)	(3,161)	(2,942)	(4,041)	(8,111)	(8,739)	(6,069)	(12,642)	(7,333)	(7,856)
Pre-finance, post-tax cash flow	4,166	(9,540)	2,202	11,155	20,018	55,269	55,345	61,613	61,309	54,759	45,025	50,369
Senior Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Senior Loan Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	4,166	(9,540)	2,202	11,155	20,018	55,269	55,345	61,613	61,309	54,759	45,025	50,369

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-13	30-Jun-13	30-Sep-13	31-Dec-13	31-Mar-14	30-Jun-14	30-Sep-14	31-Dec-14	31-Mar-15	30-Jun-15	30-Sep-15	31-Dec-15
Other Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Finance Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	4,166	(9,540)	2,202	11,155	20,018	55,269	55,345	61,613	61,309	54,759	45,025	50,369
Equity Cash Flow												
Shareholder loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Equity contributions	-	(9,540)	-	-	-	-	-	-	-	-	-	-
Dividends	4,166	-	2,202	11,155	20,018	28,887	28,283	30,039	24,671	25,585	14,705	15,685
Return of capital	(0)	-	(0)	(0)	0	26,383	27,062	31,574	36,638	29,173	30,320	34,684
Total Equity Cash Flow	4,166	(9,540)	2,202	11,155	20,018	55,269	55,345	61,613	61,309	54,759	45,025	50,369
Check	0	-	(0)	-	-	-	-	-	-	-	-	-
Deflator, October 1, 2009=1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	4,166	(9,540)	2,202	11,155	20,018	55,269	55,345	61,613	61,309	54,759	45,025	50,369
Equity IRR, deflated	N.M.	2254.4%	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.
Equity NPV ($000)												
7.0%	1,305,238	1,323,302	1,355,497	1,376,399	1,388,629	1,392,129	1,360,138	1,327,525	1,288,034	1,248,176	1,214,245	1,189,551
8.0%	1,245,314	1,265,300	1,299,512	1,322,534	1,336,964	1,342,725	1,313,006	1,282,633	1,245,342	1,207,633	1,175,805	1,153,187
9.0%	1,189,425	1,211,117	1,247,136	1,272,071	1,288,498	1,296,325	1,268,689	1,240,374	1,205,107	1,169,379	1,139,494	1,118,798
10.0%	1,137,232	1,160,439	1,198,076	1,224,738	1,242,982	1,252,697	1,226,973	1,200,551	1,167,150	1,133,250	1,105,162	1,086,247
11.0%	1,088,432	1,112,982	1,152,067	1,180,289	1,200,187	1,211,631	1,187,663	1,162,986	1,131,305	1,099,095	1,072,670	1,055,409
12.0%	1,042,747	1,068,487	1,108,869	1,138,501	1,159,905	1,172,934	1,150,583	1,127,514	1,097,423	1,066,776	1,041,893	1,026,167
Pre-tax Equity Cash Flow												
After-tax equity cash flow	4,166	(9,540)	2,202	11,155	20,018	55,269	55,345	61,613	61,309	54,759	45,025	50,369
Income tax	-	1,112	1,112	1,112	1,315	(86)	3,320	3,495	3,228	4,610	3,718	3,917
Dividend withholding tax	171	-	90	457	821	1,184	1,160	1,232	1,012	1,049	603	643
Pre-tax equity cash flow	4,337	(8,428)	3,404	12,724	22,154	56,367	59,825	66,339	65,548	60,418	49,346	54,929
After tax equity cash flow	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	4,337	(8,428)	3,404	12,724	22,154	56,367	59,825	66,339	65,548	60,418	49,346	54,929
Equity IRR, deflated	N.M.	3130.9%	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.
Equity NPV ($000)												
7.0%	1,679,128	1,703,397	1,740,955	1,767,220	1,784,534	1,792,633	1,766,367	1,736,166	1,698,879	1,661,755	1,629,171	1,607,197
8.0%	1,599,934	1,626,636	1,666,746	1,695,688	1,715,783	1,726,746	1,703,379	1,676,067	1,641,646	1,607,356	1,577,579	1,558,403
9.0%	1,526,077	1,554,929	1,597,312	1,628,657	1,651,265	1,664,832	1,644,112	1,619,445	1,587,655	1,555,973	1,528,787	1,512,201
10.0%	1,457,114	1,487,862	1,532,269	1,565,773	1,590,653	1,606,590	1,588,288	1,566,045	1,536,674	1,507,395	1,482,602	1,468,416
11.0%	1,392,644	1,425,062	1,471,270	1,506,712	1,533,648	1,551,743	1,535,653	1,515,635	1,488,488	1,461,425	1,438,844	1,426,883
12.0%	1,332,303	1,366,190	1,414,000	1,451,182	1,479,979	1,500,040	1,485,976	1,468,001	1,442,901	1,417,883	1,397,349	1,387,455

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-16	30-Jun-16	30-Sep-16	31-Dec-16	31-Mar-17	30-Jun-17	30-Sep-17	31-Dec-17	31-Mar-18	30-Jun-18	30-Sep-18	31-Dec-18
Gross revenue	133,811	132,133	189,613	197,504	197,903	216,322	113,807	133,389	103,264	133,273	117,658	104,724
Realization costs	(15,290)	(17,616)	(20,944)	(25,475)	(26,520)	(27,359)	(12,630)	(16,174)	(10,232)	(16,146)	(8,620)	(10,641)
Net revenue	118,520.9	114,517.2	168,669.0	172,029.0	171,382.9	188,962.8	101,176.8	117,215.4	93,032.1	117,126.3	109,037.9	94,082.9
Cash production cost of sales	(51,054)	(41,353)	(54,144)	(45,046)	(42,232)	(47,371)	(36,898)	(41,195)	(35,857)	(41,388)	(44,555)	(39,455)
Operating margin	67,467	73,165	114,525	126,983	129,151	141,592	64,279	76,020	57,175	75,738	64,483	54,628
Interest expense	-	-	-	-	-	-	-	-	-	-	-	-
Interest income	49	61	61	61	60	73	73	73	72	85	86	86
Sub-total	67,516	73,225	114,587	127,045	129,211	141,665	64,353	76,093	57,247	75,823	64,568	54,714
Working capital												
Decrease (increase) in current assets												
Cash and equivalents	-	-	-	-	-	-	-	-	-	-	-	-
Restricted funds	-	-	-	-	-	-	-	-	-	-	-	-
Accounts receivable	(7,913)	1,553	(33,051)	3,305	59	(7,316)	42,783	(7,001)	10,656	(10,610)	(5,892)	16,049
IGV recoverable	-	(189)	(70)	259	-	(322)	322	-	(93)	93	-	(2)
Interest receivable	1	(12)	(1)	-	1	(13)	(1)	-	2	(13)	(1)	-
Supplies	(514)	(9)	503	(620)	999	(405)	787	(215)	(328)	520	188	(156)
Finished product inventory (cash)	3,523	(6,011)	7,896	(2,724)	(330)	4,865	(3,571)	(70)	(5,768)	978	2,377	(3,915)
Work in process inventory (cash)	0	0	0	0	0	(4)	4	0	(0)	(0)	(5)	(6)
Deposits, prepaids, others	-	-	-	-	-	-	-	-	-	-	-	-
Increase (decrease) in current liabilities												
Accrued charges	-	-	-	-	-	-	-	-	-	-	-	-
Accounts payable	539	27	(484)	642	(956)	385	(842)	215	368	(541)	(195)	156
IGV payable	(336)	(57)	-	87	260	(347)	190	(34)	(156)	132	(71)	(61)
Decrease (increase) in working capital	(4,701)	(4,697)	(25,207)	949	33	(3,158)	39,673	(7,104)	4,680	(9,440)	(3,599)	12,066
Decrease (increase) in ore in stockpile	2,212	1,637	1,065	208	(1,111)	(2,802)	(1,637)	(1,496)	(1,399)	(1,406)	924	1,634
Fixed capital												
Mine development - cash	-	-	-	-	-	-	-	-	-	-	-	-
Exploration and Feasibility Study	-	-	-	-	-	-	-	-	-	-	-	-
Oxide project	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide project	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital	-	(798)	(4,136)	-	-	(4,813)	(1,613)	(1,160)	(2,226)	-	(819)	-
Fixed capital	-	(798)	(4,136)	-	-	(4,813)	(1,613)	(1,160)	(2,226)	-	(819)	-
Closure costs	-	-	-	-	-	-	-	-	-	-	-	-
Mine closure guarantee security	(1,055)	-	-	-	(1,056)	-	-	-	(1,056)	-	-	-
Pre-finance, pre-tax cash flow	63,972	69,367	86,308	128,201	127,077	130,892	100,775	66,333	57,246	64,977	61,074	68,414
Tax payments												
Mining royalty	(2,061)	(3,796)	(5,468)	(5,685)	(3,887)	(6,227)	(3,259)	(3,828)	(1,372)	(3,613)	(3,390)	(3,010)
Employee profit sharing	-	(7,688)	-	-	-	(19,040)	-	-	-	(27,559)	-	-
Income tax and Net Asset Tax (ITAN)	(7,993)	(21,686)	(12,254)	(12,456)	(20,915)	(52,636)	(13,114)	(15,102)	(16,262)	(49,652)	(19,852)	(17,193)
Financial Transactions Tax (ITF)	(43)	(44)	(48)	(46)	(42)	(49)	(41)	(42)	(42)	(40)	(39)	(39)
Interest withholding tax	-	-	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	(971)	(1,207)	(2,000)	(2,437)	(2,538)	(2,085)	(1,616)	(1,219)	(880)	-	(1,488)	(1,347)
Total taxes	(11,067)	(34,421)	(19,770)	(20,623)	(27,383)	(80,038)	(18,029)	(20,191)	(18,556)	(80,864)	(24,770)	(21,589)
Pre-finance, post-tax cash flow	52,905	34,946	66,538	107,577	99,694	50,855	82,746	46,143	38,690	(15,887)	36,304	46,825
Senior Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Senior Loan Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	52,905	34,946	66,538	107,577	99,694	50,855	82,746	46,143	38,690	(15,887)	36,304	46,825

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-16	30-Jun-16	30-Sep-16	31-Dec-16	31-Mar-17	30-Jun-17	30-Sep-17	31-Dec-17	31-Mar-18	30-Jun-18	30-Sep-18	31-Dec-18
Other Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Finance Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	52,905	34,946	66,538	107,577	99,694	50,855	82,746	46,143	38,690	(15,887)	36,304	46,825
Equity Cash Flow												
Shareholder loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Equity contributions	-	-	-	-	-	-	-	-	-	(15,887)	-	-
Dividends	23,672	29,434	48,777	59,433	61,910	50,855	39,403	29,744	21,461	-	36,304	32,854
Return of capital	29,233	5,512	17,761	48,145	37,784	(0)	43,343	16,399	17,229	-	(0)	13,971
Total Equity Cash Flow	52,905	34,946	66,538	107,577	99,694	50,855	82,746	46,143	38,690	(15,887)	36,304	46,825
Check	-	-	-	-	-	0	-	0	-	-	-	-
Deflator, October 1, 2009=1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	52,905	34,946	66,538	107,577	99,694	50,855	82,746	46,143	38,690	(15,887)	36,304	46,825
Equity IRR, deflated	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	13657.0%	N.M.	N.M.
Equity NPV ($000)												
7.0%	1,159,046	1,125,463	1,109,419	1,061,241	970,853	886,874	850,716	781,778	748,580	722,331	750,675	726,868
8.0%	1,124,733	1,093,167	1,079,119	1,032,901	944,350	862,038	827,438	759,966	728,141	703,222	732,923	710,507
9.0%	1,092,249	1,062,558	1,050,374	1,005,990	919,156	838,400	805,254	739,148	708,603	684,926	715,901	694,796
10.0%	1,061,468	1,033,525	1,023,082	980,417	895,191	815,889	784,099	719,268	689,917	667,399	699,570	679,701
11.0%	1,032,276	1,005,963	997,149	956,095	872,378	794,435	763,912	700,272	672,034	650,600	683,893	665,190
12.0%	1,004,569	979,778	972,489	932,948	850,647	773,976	744,637	682,109	654,911	634,489	668,836	651,235
Pre-tax Equity Cash Flow												
After-tax equity cash flow	52,905	34,946	66,538	107,577	99,694	50,855	82,746	46,143	38,690	(15,887)	36,304	46,825
Income tax	7,993	21,686	12,254	12,456	20,915	52,636	13,114	15,102	16,262	49,652	19,852	17,193
Dividend withholding tax	971	1,207	2,000	2,437	2,538	2,085	1,616	1,219	880	-	1,488	1,347
Pre-tax equity cash flow	61,869	57,838	80,793	122,470	123,147	105,576	97,476	62,464	55,832	33,765	57,645	65,365
After tax equity cash flow	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	61,869	57,838	80,793	122,470	123,147	105,576	97,476	62,464	55,832	33,765	57,645	65,365
Equity IRR, deflated	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.
Equity NPV ($000)												
7.0%	1,579,218	1,543,762	1,511,767	1,456,077	1,357,405	1,256,367	1,171,327	1,093,004	1,048,655	1,010,237	993,418	952,230
8.0%	1,533,217	1,500,535	1,471,288	1,418,296	1,322,195	1,223,543	1,140,716	1,064,459	1,022,070	985,553	970,607	931,261
9.0%	1,489,610	1,459,512	1,432,833	1,382,370	1,288,679	1,192,263	1,111,509	1,037,185	996,632	961,900	948,718	911,110
10.0%	1,448,237	1,420,549	1,396,272	1,348,180	1,256,752	1,162,434	1,083,624	1,011,110	972,279	939,223	927,701	891,736
11.0%	1,408,950	1,383,512	1,361,485	1,315,620	1,226,318	1,133,970	1,056,983	986,167	948,950	917,469	907,512	873,099
12.0%	1,371,614	1,348,279	1,328,362	1,284,590	1,197,288	1,106,791	1,031,515	962,292	926,591	896,590	888,108	855,163

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-19	30-Jun-19	30-Sep-19	31-Dec-19	31-Mar-20	30-Jun-20	30-Sep-20	31-Dec-20	31-Mar-21	30-Jun-21	30-Sep-21	31-Dec-21
Gross revenue	104,757	138,582	103,355	103,207	116,488	122,571	94,510	128,879	115,572	115,640	94,715	104,179
Realization costs	(10,635)	(14,549)	(9,419)	(10,413)	(10,574)	(13,966)	(8,256)	(13,232)	(13,041)	(13,050)	(8,241)	(10,507)
Net revenue	94,121.8	124,033.4	93,935.9	92,794.0	105,914.2	108,605.6	86,254.2	115,646.8	102,531.1	102,589.8	86,474.1	93,671.9
Cash production cost of sales	(39,695)	(52,909)	(40,381)	(40,262)	(48,602)	(45,477)	(37,993)	(48,807)	(39,427)	(39,259)	(39,186)	(40,531)
Operating margin	54,427	71,124	53,555	52,532	57,312	63,128	48,261	66,839	63,104	63,331	47,288	53,141
Interest expense	-	-	-	-	-	-	-	-	-	-	-	-
Interest income	84	97	98	98	97	109	110	110	108	121	122	122
Sub-total	54,511	71,221	53,652	52,629	57,408	63,237	48,371	66,949	63,212	63,452	47,410	53,263
Working capital												
Decrease (increase) in current assets												
Cash and equivalents	-	-	-	-	-	-	-	-	-	-	-	-
Restricted funds	-	-	-	-	-	-	-	-	-	-	-	-
Accounts receivable	(8)	(12,930)	8,296	5,186	(5,611)	(1,277)	4,824	(7,882)	5,626	(25)	2,217	1,757
IGV recoverable	(28)	(51)	81	-	(122)	122	-	-	-	(134)	134	-
Interest receivable	2	(13)	(1)	-	1	(12)	(1)	-	2	(13)	(1)	-
Supplies	195	(606)	520	16	(856)	768	2,500	(756)	619	(476)	1,124	289
Finished product inventory (cash)	(2,924)	7,487	(2,641)	(2,878)	2,978	2,206	(4,568)	5,447	(116)	(1,071)	(1,289)	1,427
Work in process inventory (cash)	(6)	(6)	2	2	2	2	(32)	(29)	(28)	(26)	129	0
Deposits, prepaids, others	-	-	-	-	-	-	-	-	-	-	-	-
Increase (decrease) in current liabilities												
Accrued charges	-	-	-	-	-	-	-	-	-	-	-	-
Accounts payable	(154)	585	(541)	(16)	876	(768)	(2,548)	756	(579)	456	(1,231)	(292)
IGV payable	-	-	26	61	(87)	76	383	(415)	(38)	(7)	88	152
Decrease (increase) in working capital	(2,923)	(5,533)	5,744	2,371	(2,819)	1,117	559	(2,878)	5,487	(1,296)	1,171	3,333
Decrease (increase) in ore in stockpile	2,176	2,796	1,627	1,795	2,032	2,009	8,642	7,121	5,746	4,814	8,252	7,906
Fixed capital												
Mine development - cash	-	-	-	-	-	-	-	-	-	-	-	-
Exploration and Feasibility Study	-	-	-	-	-	-	-	-	-	-	-	-
Oxide project	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide project	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital	(1,428)	(353)	(763)	-	-	-	(1,035)	-	(149)	(353)	(1,735)	-
Fixed capital	(1,428)	(353)	(763)	-	-	-	(1,035)	-	(149)	(353)	(1,735)	-
Closure costs	-	-	(172)	(172)	(172)	(172)	(81)	(81)	(81)	(81)	(39)	(39)
Mine closure guarantee security	(1,056)	-	-	-	(1,056)	-	-	-	(1,057)	-	-	-
Pre-finance, pre-tax cash flow	51,280	68,131	60,088	56,624	55,394	66,191	56,456	71,111	73,158	66,536	55,059	64,463
Tax payments												
Mining royalty	(1,409)	(3,806)	(2,972)	(2,965)	(1,637)	(3,451)	(2,701)	(3,703)	(1,612)	(3,225)	(2,708)	(2,990)
Employee profit sharing	-	(16,646)	-	-	-	(15,053)	-	-	-	(15,301)	-	-
Income tax and Net Asset Tax (ITAN)	(13,960)	(5,734)	(14,487)	(14,315)	(14,499)	(8,914)	(12,170)	(16,240)	(13,849)	(12,223)	(11,913)	(12,899)
Financial Transactions Tax (ITF)	(39)	(41)	(40)	(39)	(41)	(39)	(32)	(34)	(32)	(34)	(31)	(29)
Interest withholding tax	-	-	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	(762)	(950)	(700)	(675)	(729)	(864)	(595)	(865)	(925)	(892)	(473)	(557)
Total taxes	(16,170)	(27,178)	(18,199)	(17,994)	(16,906)	(28,321)	(15,498)	(20,843)	(16,417)	(31,675)	(15,125)	(16,475)
Pre-finance, post-tax cash flow	35,109	40,953	41,889	38,629	38,488	37,870	40,958	50,268	56,741	34,861	39,934	47,988
Senior Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Senior Loan Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	35,109	40,953	41,889	38,629	38,488	37,870	40,958	50,268	56,741	34,861	39,934	47,988

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-19	30-Jun-19	30-Sep-19	31-Dec-19	31-Mar-20	30-Jun-20	30-Sep-20	31-Dec-20	31-Mar-21	30-Jun-21	30-Sep-21	31-Dec-21
Other Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Finance Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	35,109	40,953	41,889	38,629	38,488	37,870	40,958	50,268	56,741	34,861	39,934	47,988
Equity Cash Flow												
Shareholder loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Equity contributions	-	-	-	-	-	-	-	-	-	-	-	-
Dividends	18,579	23,178	17,085	16,473	17,778	21,078	14,503	21,102	22,569	21,758	11,525	13,596
Return of capital	16,531	17,775	24,804	22,157	20,710	16,792	26,455	29,167	34,172	13,104	28,408	34,392
Total Equity Cash Flow	35,109	40,953	41,889	38,629	38,488	37,870	40,958	50,268	56,741	34,861	39,934	47,988
Check	-	-	-	-	-	-	-	-	-	-	-	-
Deflator, October 1, 2009=1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	35,109	40,953	41,889	38,629	38,488	37,870	40,958	50,268	56,741	34,861	39,934	47,988
Equity IRR, deflated	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.
Equity NPV ($000)												
7.0%	692,045	668,442	638,544	607,192	578,592	549,647	520,831	488,410	446,047	396,433	368,038	334,043
8.0%	677,032	654,736	626,106	595,975	568,550	540,734	513,003	481,615	440,217	391,479	363,886	330,636
9.0%	662,595	641,535	614,110	585,141	558,837	532,102	505,410	475,016	434,548	386,656	359,838	327,309
10.0%	648,705	628,817	602,536	574,674	549,439	523,739	498,044	468,605	429,034	381,959	355,890	324,060
11.0%	635,334	616,557	591,364	564,556	540,343	515,632	490,895	462,376	423,669	377,382	352,039	320,886
12.0%	622,457	604,734	580,575	554,772	531,535	507,772	483,954	456,319	418,447	372,922	348,280	317,785
Pre-tax Equity Cash Flow												
After-tax equity cash flow	35,109	40,953	41,889	38,629	38,488	37,870	40,958	50,268	56,741	34,861	39,934	47,988
Income tax	13,960	5,734	14,487	14,315	14,499	8,914	12,170	16,240	13,849	12,223	11,913	12,899
Dividend withholding tax	762	950	700	675	729	864	595	865	925	892	473	557
Pre-tax equity cash flow	49,832	47,638	57,076	53,620	53,716	47,648	53,722	67,374	71,515	47,976	52,319	61,445
After tax equity cash flow	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	49,832	47,638	57,076	53,620	53,716	47,648	53,722	67,374	71,515	47,976	52,319	61,445
Equity IRR, deflated	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.
Equity NPV ($000)												
7.0%	902,554	867,696	834,455	791,129	750,549	709,180	673,224	630,530	573,340	510,998	471,331	426,608
8.0%	883,356	850,203	818,621	776,896	737,850	697,949	663,399	622,045	566,104	504,895	466,264	422,496
9.0%	864,880	833,344	803,339	763,140	725,559	687,063	653,861	613,797	559,061	498,947	461,317	418,478
10.0%	847,091	817,088	788,583	749,838	713,657	676,506	644,600	605,777	552,203	493,147	456,487	414,549
11.0%	829,955	801,407	774,328	736,971	702,127	666,265	635,603	597,976	545,525	487,491	451,770	410,706
12.0%	813,441	786,275	760,553	724,519	690,954	656,328	626,861	590,386	539,018	481,973	447,163	406,947

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-22	30-Jun-22	30-Sep-22	31-Dec-22	31-Mar-23	30-Jun-23	30-Sep-23	31-Dec-23	31-Mar-24	30-Jun-24	30-Sep-24	31-Dec-24
Gross revenue	101,166	81,437	103,188	83,712	115,406	83,947	37,853	48,603	-	-	-	-
Realization costs	(12,708)	(12,414)	(17,587)	(16,095)	(21,204)	(16,115)	(6,060)	(7,419)	-	-	-	-
Net revenue	88,458.0	69,022.8	85,601.4	67,616.9	94,202.6	67,832.3	31,793.5	41,183.7	-	-	-	-
Cash production cost of sales	(36,357)	(27,046)	(16,888)	(13,799)	(18,893)	(13,636)	(10,861)	(17,802)	-	-	-	-
Operating margin	52,101	41,976	68,714	53,818	75,310	54,197	20,932	23,382	-	-	-	-
Interest expense	-	-	-	-	-	-	-	-	-	-	-	-
Interest income	119	133	134	134	131	136	137	137	-	-	-	-
Sub-total	52,221	42,109	68,848	53,953	75,441	54,332	21,069	23,519	-	-	-	-
Working capital												
Decrease (increase) in current assets												
Cash and equivalents	-	-	-	-	-	-	-	-	-	-	-	-
Restricted funds	-	-	-	-	-	-	-	-	-	-	-	-
Accounts receivable	2,073	8,327	(17,128)	17,626	(16,641)	16,550	11,343	(11,195)	29,588	-	-	-
IGV recoverable	(56)	56	-	(204)	24	172	8	-	-	(213)	(264)	(264)
Interest receivable	3	(13)	(2)	-	3	(5)	(2)	-	137	-	-	-
Supplies	(718)	4,077	976	(456)	75	(74)	1,420	703	1,159	-	-	-
Finished product inventory (cash)	(4,098)	8,725	3,063	(1,634)	3,941	(1,686)	(4,171)	9,284	-	-	-	-
Work in process inventory (cash)	0	(0)	-	-	-	-	-	-	-	-	-	-
Deposits, prepaids, others	-	-	-	-	-	-	-	-	-	-	-	-
Increase (decrease) in current liabilities												
Accrued charges	-	-	-	-	-	-	-	-	-	-	-	-
Accounts payable	757	(4,927)	(1,039)	456	(56)	64	(1,532)	(1,088)	(1,533)	-	-	-
IGV payable	(240)	798	(395)	(404)	-	-	363	(5)	(285)	(72)	-	-
Decrease (increase) in working capital	(2,279)	17,043	(14,524)	15,384	(12,655)	15,022	7,430	(2,302)	29,065	(285)	(264)	(264)
Decrease (increase) in ore in stockpile	7,584	720	(1,265)	(1,057)	(1,090)	(1,055)	4,500	2,235	-	-	-	-
Fixed capital												
Mine development - cash	-	-	-	-	-	-	-	-	-	-	-	-
Exploration and Feasibility Study	-	-	-	-	-	-	-	-	-	-	-	-
Oxide project	-	-	-	-	-	-	-	-	-	-	-	-
Sulphide project	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital	-	-	(231)	-	(41)	-	(749)	-	-	-	-	-
Fixed capital	-	-	(231)	-	(41)	-	(749)	-	-	-	-	-
Closure costs	(39)	(39)	(414)	(414)	(414)	(414)	(213)	(643)	(1,502)	(1,502)	(1,389)	(1,389)
Mine closure guarantee security	(1,057)	-	-	-	(261)	-	-	-	430	-	-	-
Pre-finance, pre-tax cash flow	56,430	59,834	52,414	67,865	60,981	67,886	32,038	22,810	27,993	(1,787)	(1,653)	(1,653)
Tax payments												
Mining royalty	(1,331)	(2,094)	(2,951)	(2,382)	(1,601)	(2,290)	(1,084)	(1,398)	-	-	-	-
Employee profit sharing	-	(14,642)	-	-	-	(14,280)	-	-	-	(6,343)	-	-
Income tax and Net Asset Tax (ITAN)	(12,359)	(10,825)	(11,993)	(9,441)	(15,950)	(20,494)	(5,415)	(7,093)	-	16,568	-	-
Financial Transactions Tax (ITF)	(30)	(16)	(14)	(15)	(16)	(15)	(9)	(7)	(2)	(2)	(1)	(1)
Interest withholding tax	-	-	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	(681)	(605)	(1,416)	(1,122)	(1,620)	(1,142)	(358)	(362)	(8)	(9)	-	-
Total taxes	(14,402)	(28,182)	(16,374)	(12,961)	(19,187)	(38,222)	(6,867)	(8,859)	(10)	10,214	(1)	(1)
Pre-finance, post-tax cash flow	42,028	31,651	36,040	54,904	41,793	29,664	25,171	13,951	27,983	8,427	(1,654)	(1,654)
Senior Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Senior Loan Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	42,028	31,651	36,040	54,904	41,793	29,664	25,171	13,951	27,983	8,427	(1,654)	(1,654)

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-22	30-Jun-22	30-Sep-22	31-Dec-22	31-Mar-23	30-Jun-23	30-Sep-23	31-Dec-23	31-Mar-24	30-Jun-24	30-Sep-24	31-Dec-24
Other Loans												
Loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Net Finance Payments	-	-	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	42,028	31,651	36,040	54,904	41,793	29,664	25,171	13,951	27,983	8,427	(1,654)	(1,654)
Equity Cash Flow												
Shareholder loan disbursements	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-	-	-	-	-	-	-	-
Equity contributions	-	-	-	-	-	-	-	-	-	-	(1,654)	(1,654)
Dividends	16,615	14,766	34,528	27,377	39,524	27,865	8,733	8,834	200	219	-	-
Return of capital	25,413	16,885	1,512	27,527	2,270	1,799	16,438	5,117	27,783	8,207	-	-
Total Equity Cash Flow	42,028	31,651	36,040	54,904	41,793	29,664	25,171	13,951	27,983	8,427	(1,654)	(1,654)
Check	-	-	-	-	-	-	-	-	-	-	-	-
Deflator, October 1, 2009=1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	42,028	31,651	36,040	54,904	41,793	29,664	25,171	13,951	27,983	8,427	(1,654)	(1,654)
Equity IRR, deflated	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	35.8%	1216.8%
Equity NPV ($000)												
7.0%	291,346	253,931	226,342	193,857	141,794	102,064	73,889	49,765	36,545	8,948	602	2,281
8.0%	288,607	251,779	224,713	192,690	140,998	101,540	73,562	49,577	36,455	8,909	574	2,256
9.0%	285,929	249,673	223,116	191,545	140,218	101,025	73,240	49,392	36,366	8,872	547	2,231
10.0%	283,311	247,611	221,551	190,422	139,451	100,519	72,924	49,210	36,279	8,835	520	2,207
11.0%	280,750	245,592	220,016	189,319	138,698	100,022	72,613	49,031	36,193	8,799	494	2,183
12.0%	278,245	243,613	218,511	188,237	137,958	99,533	72,306	48,854	36,109	8,764	469	2,160
Pre-tax Equity Cash Flow												
After-tax equity cash flow	42,028	31,651	36,040	54,904	41,793	29,664	25,171	13,951	27,983	8,427	(1,654)	(1,654)
Income tax	12,359	10,825	11,993	9,441	15,950	20,494	5,415	7,093	-	(16,568)	-	-
Dividend withholding tax	681	605	1,416	1,122	1,620	1,142	358	362	8	9	-	-
Pre-tax equity cash flow	55,068	43,082	49,449	65,468	59,364	51,301	30,944	21,405	27,991	(8,132)	(1,654)	(1,654)
After tax equity cash flow	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	55,068	43,082	49,449	65,468	59,364	51,301	30,944	21,405	27,991	(8,132)	(1,654)	(1,654)
Equity IRR, deflated	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.	38.5%	1280.2%
Equity NPV ($000)												
7.0%	371,918	322,727	284,785	239,774	177,840	121,006	71,334	41,343	20,461	(7,419)	656	2,336
8.0%	368,665	320,227	282,950	238,520	177,053	120,555	71,100	41,238	20,426	(7,439)	628	2,310
9.0%	365,481	317,777	281,149	237,288	176,278	120,110	70,870	41,134	20,392	(7,458)	600	2,285
10.0%	362,364	315,374	279,381	236,076	175,516	119,673	70,643	41,032	20,359	(7,477)	573	2,261
11.0%	359,311	313,018	277,644	234,885	174,766	119,243	70,421	40,931	20,327	(7,495)	546	2,237
12.0%	356,320	310,706	275,939	233,715	174,029	118,819	70,201	40,833	20,295	(7,512)	520	2,213

Marcobre S.A.C.
Mina Justa Project: Quarterly Cash Flow Schedule

Quarter ending	31-Mar-25	30-Jun-25	30-Sep-25	31-Dec-25	31-Mar-26	30-Jun-26	30-Sep-26	31-Dec-26	31-Mar-27	30-Jun-27
Gross revenue	-	-	-	-	-	-	-	-	-	-
Realization costs	-	-	-	-	-	-	-	-	-	-
Net revenue	-	-	-	-	-	-	-	-	-	-
Cash production cost of sales	-	-	-	-	-	-	-	-	-	-
Operating margin	-	-	-	-	-	-	-	-	-	-
Interest expense	-	-	-	-	-	-	-	-	-	-
Interest income	-	-	-	-	-	-	-	-	-	-
Sub-total	-	-	-	-	-	-	-	-	-	-
Working capital										
Decrease (increase) in current assets										
Cash and equivalents	-	-	-	-	-	-	-	-	-	-
Restricted funds	-	-	-	-	-	-	-	-	-	-
Accounts receivable	-	-	-	-	-	-	-	-	-	-
IGV recoverable	(264)	(264)	(153)	(153)	(153)	(153)	(7)	(7)	(7)	(7)
Interest receivable	-	-	-	-	-	-	-	-	-	-
Supplies	-	-	-	-	-	-	-	-	-	-
Finished product inventory (cash)	-	-	-	-	-	-	-	-	-	-
Work in process inventory (cash)	-	-	-	-	-	-	-	-	-	-
Deposits, prepaids, others	-	-	-	-	-	-	-	-	-	-
Increase (decrease) in current liabilities										
Accrued charges	-	-	-	-	-	-	-	-	-	-
Accounts payable	-	-	-	-	-	-	-	-	-	-
IGV payable	-	-	-	-	-	-	-	-	-	-
Decrease (increase) in working capital	(264)	(264)	(153)	(153)	(153)	(153)	(7)	(7)	(7)	(7)
Decrease (increase) in ore in stockpile	-	-	-	-	-	-	-	-	-	-
Fixed capital										
Mine development - cash	-	-	-	-	-	-	-	-	-	-
Exploration and Feasibility Study	-	-	-	-	-	-	-	-	-	-
Oxide project	-	-	-	-	-	-	-	-	-	-
Sulphide project	-	-	-	-	-	-	-	-	-	-
Sustaining capital	-	-	-	-	-	-	-	-	-	-
Fixed capital	-	-	-	-	-	-	-	-	-	-
Closure costs	(1,389)	(1,389)	(804)	(804)	(804)	(804)	(36)	(36)	(36)	(36)
Mine closure guarantee security	5,786	-	-	-	5,710	-	-	-	-	-
Pre-finance, pre-tax cash flow	4,133	(1,653)	(957)	(957)	4,753	(957)	(43)	(43)	(43)	(43)
Tax payments										
Mining royalty	-	-	-	-	-	-	-	-	-	-
Employee profit sharing	-	-	-	-	-	-	-	-	-	-
Income tax and Net Asset Tax (ITAN)	-	-	-	-	-	-	-	-	-	-
Financial Transactions Tax (ITF)	(1)	(1)	(1)	(1)	(1)	(1)	(0)	(0)	(0)	(0)
Interest withholding tax	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	(28)	-	-	-	(30)	-	-	-	-	-
Total taxes	(30)	(1)	(1)	(1)	(31)	(1)	(0)	(0)	(0)	(0)
Pre-finance, post-tax cash flow	4,104	(1,654)	(958)	(958)	4,722	(958)	(43)	(43)	(43)	(43)
Senior Loans										
Loan disbursements	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-
Net Senior Loan Payments	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	4,104	(1,654)	(958)	(958)	4,722	(958)	(43)	(43)	(43)	(43)

Quarter ending	31-Mar-25	30-Jun-25	30-Sep-25	31-Dec-25	31-Mar-26	30-Jun-26	30-Sep-26	31-Dec-26	31-Mar-27	30-Jun-27
Other Loans										
Loan disbursements	-	-	-	-	-	-	-	-	-	-
Financing fees	-	-	-	-	-	-	-	-	-	-
Interest	-	-	-	-	-	-	-	-	-	-
Loan repayments	-	-	-	-	-	-	-	-	-	-
Net Finance Payments	-	-	-	-	-	-	-	-	-	-
Post-Senior Loan, post-tax cash flow	4,104	(1,654)	(958)	(958)	4,722	(958)	(43)	(43)	(43)	(43)
Equity Cash Flow										
Shareholder loan disbursements	-	-	-	-	-	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-	-	-	-	-	-
Equity contributions	-	(1,654)	(958)	(958)	-	(958)	(43)	(43)	(43)	(43)
Dividends	687	-	-	-	736	-	-	-	-	-
Return of capital	3,417	-	-	-	3,986	-	-	-	-	-
Total Equity Cash Flow	4,104	(1,654)	(958)	(958)	4,722	(958)	(43)	(43)	(43)	(43)
Check	-	-	-	-	-	-	-	-	-	-
Deflator, October 1, 2009=1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	4,104	(1,654)	(958)	(958)	4,722	(958)	(43)	(43)	(43)	(43)
Equity IRR, deflated	N.M.	1.4%	601.8%	49281.9%	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.
Equity NPV ($000)										
7.0%	3,988	(83)	1,584	2,577	3,588	(1,113)	(166)	(126)	(84)	(43)
8.0%	3,970	(97)	1,572	2,570	3,587	(1,111)	(165)	(125)	(84)	(43)
9.0%	3,952	(110)	1,560	2,562	3,586	(1,109)	(164)	(125)	(84)	(42)
10.0%	3,934	(124)	1,547	2,554	3,585	(1,107)	(164)	(124)	(84)	(42)
11.0%	3,917	(137)	1,536	2,547	3,585	(1,104)	(163)	(124)	(84)	(42)
12.0%	3,900	(150)	1,524	2,539	3,584	(1,102)	(162)	(123)	(83)	(42)
Pre-tax Equity Cash Flow										
After-tax equity cash flow	4,104	(1,654)	(958)	(958)	4,722	(958)	(43)	(43)	(43)	(43)
Income tax	-	-	-	-	-	-	-	-	-	-
Dividend withholding tax	28	-	-	-	30	-	-	-	-	-
Pre-tax equity cash flow	4,132	(1,654)	(958)	(958)	4,752	(958)	(43)	(43)	(43)	(43)
After tax equity cash flow	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Deflated Cash Flow	4,132	(1,654)	(958)	(958)	4,752	(958)	(43)	(43)	(43)	(43)
Equity IRR, deflated	N.M.	3.3%	616.1%	50678.9%	N.M.	N.M.	N.M.	N.M.	N.M.	N.M.
Equity NPV ($000)										
7.0%	4,044	(54)	1,613	2,607	3,617	(1,113)	(166)	(126)	(84)	(43)
8.0%	4,025	(68)	1,601	2,599	3,617	(1,111)	(165)	(125)	(84)	(43)
9.0%	4,007	(82)	1,588	2,591	3,616	(1,109)	(164)	(125)	(84)	(42)
10.0%	3,989	(96)	1,576	2,583	3,615	(1,107)	(164)	(124)	(84)	(42)
11.0%	3,972	(109)	1,564	2,576	3,614	(1,104)	(163)	(124)	(84)	(42)
12.0%	3,955	(123)	1,552	2,568	3,614	(1,102)	(162)	(123)	(83)	(42)

Balance Sheets as at December 31	**2009**	**2010**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**
(in thousands of U.S. dollars)													
Assets													
Current Assets													
Cash and equivalents	-	-	-	-	-	-	-	-	-	-	-	-	-
Restricted funds	-	-	-	-	-	-	-	-	-	-	-	-	-
Accounts receivable	-	-	-	22,723	22,648	52,290	43,339	79,445	50,920	40,717	40,172	50,117	40,543
Taxes receivable	-	-	-	-	3,335	-	-	-	-	-	-	-	-
IGV recoverable	12,418	33,573	50,202	29,839	18,814	-	-	-	-	2	-	-	-
Interest receivable	-	-	-	13	25	37	49	61	73	86	98	110	122
Supplies	-	-	1,680	9,633	10,494	11,630	11,252	11,892	10,726	10,501	10,375	8,718	7,161
Finished product inventory	-	-	-	11,743	10,573	19,712	21,765	17,586	17,169	26,305	28,130	19,087	21,542
Work in process inventory	-	-	-	0	0	0	0	0	0	18	30	123	0
Ore in stockpile	-	-	369	34,949	73,781	93,064	88,157	81,999	90,656	91,127	79,481	54,634	16,846
Deposits, prepaids, others	-	-	-	-	-	-	-	-	-	-	-	-	-
Total current assets	12,418	33,573	52,251	108,901	139,670	176,734	164,563	190,983	169,544	168,755	158,284	132,789	86,215
Non-current assets													
Mine closure guarantee security	-	-	-	1,109	2,164	3,219	4,274	5,329	6,385	7,441	8,496	9,552	10,609
Capital assets													
Land and mining concessions	33,414	36,414	36,414	35,623	33,015	29,356	25,567	21,779	17,990	14,202	10,413	6,624	2,836
Construction in progress	1,124	136,758	356,764	98,292	160,638	-	-	-	-	-	-	-	-
Mine development	5,659	5,659	12,922	40,180	37,240	33,112	28,839	24,565	20,292	16,019	11,745	7,472	3,199
Plant and equipment	175	175	58,395	464,831	417,118	505,229	439,392	374,279	312,893	246,721	179,935	112,156	46,721
Preproduction intangibles	58,816	71,327	82,419	111,274	104,749	100,257	87,213	74,169	61,126	48,083	35,039	21,996	8,952
ARO asset	-	-	-	2,533	10,090	8,971	7,813	6,656	5,498	4,340	3,182	2,024	867
Net capital assets	99,187	250,332	546,914	752,733	762,850	676,924	588,824	501,447	417,799	329,364	240,315	150,272	62,575
Total non-current assets	99,187	250,332	546,914	753,842	765,014	680,143	593,099	506,776	424,184	336,805	248,811	159,824	73,183
Total assets	111,605	283,904	599,165	862,743	904,684	856,877	757,661	697,759	593,728	505,560	407,095	292,613	159,399
Liabilities													
Current liabilities													
Accrued charges	-	-	-	-	-	-	-	-	-	-	-	-	-
Accounts payable	-	-	1,797	10,489	11,459	13,487	13,040	13,764	12,566	12,354	12,228	10,543	8,897
Advances	-	-	-	-	-	-	-	-	-	-	-	-	-
Interest payable	-	-	-	-	-	-	-	-	-	-	-	-	-
IGV payable	-	-	-	-	-	1,276	394	87	156	-	87	45	240
Taxes payable	-	-	-	-	-	3,445	20,786	47,816	55,987	3,372	8,676	13,409	15,752
Senior debt	-	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loans	-	-	-	-	-	-	-	-	-	-	-	-	-
Total current liabilities	-	-	1,797	10,489	11,459	18,209	34,219	61,667	68,709	15,726	20,991	23,998	24,888
Non-current liabilities													
Long-term debt	-	-	-	-	-	-	-	-	-	-	-	-	-
Future income and related taxes	-	-	-	4,000	19,728	63,745	78,527	90,960	76,488	55,624	32,585	7,845	(16,968)
Asset retirement obligation	-	-	-	2,680	10,613	11,366	12,173	13,040	13,966	14,958	15,533	15,908	16,692
Total non-current liabilities	-	-	-	6,680	30,341	75,111	90,701	104,000	90,453	70,581	48,118	23,753	(276)
Total liabilities	-	-	1,797	17,169	41,800	93,320	124,919	165,667	159,163	86,308	69,109	47,750	24,612
Shareholders' Equity													
Capital stock	111,681	283,981	597,444	839,035	848,575	763,557	632,742	532,092	434,565	419,253	337,986	244,862	134,786
Retained earnings	(76)	(76)	(76)	6,539	14,309	-	-	-	-	-	-	-	-
Total shareholders' equity	111,605	283,904	597,368	845,574	862,884	763,557	632,742	532,092	434,565	419,253	337,986	244,862	134,786
Total liabilities and shareholders' equity	111,605	283,904	599,165	862,743	904,684	856,877	757,661	697,759	593,728	505,560	407,095	292,613	159,399

Statements of Earnings at December 31: (in thousands of U.S. dollars)	**2009**	**2010**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**
Gross revenue	-	-	-	53,651	228,060	456,691	472,184	653,060	661,420	458,919	449,901	462,448	430,107
Realization costs	-	-	-	785	3,349	49,437	51,844	79,324	82,682	45,639	45,016	46,027	44,840
Net revenue	-	-	-	52,866	224,711	407,254	420,340	573,736	578,738	413,279	404,885	416,421	385,267
Costs													
Production costs	-	-	-	28,762	121,125	160,946	181,629	191,596	167,696	161,255	173,247	180,880	158,402
Government royalty	-	-	-	517	4,788	11,220	11,781	17,009	17,201	11,384	11,152	11,492	10,535
Asset retirement obligation accretion	-	-	-	91	190	754	807	867	926	992	1,058	1,083	1,119
Depreciation and amortization	-	-	-	12,800	56,749	78,141	91,417	94,844	89,145	88,441	93,973	99,063	99,644
Costs of Sales	-	-	-	42,170	182,852	251,061	285,635	304,316	274,968	262,072	279,431	292,518	269,700
Earnings before other income, expenses and taxes	-	-	-	10,696	41,859	156,193	134,705	269,420	303,769	151,207	125,454	123,903	115,567
Other income and expenses													
Interest expense	-	-	-	-	-	-	-	-	-	-	-	-	-
Interest income	-	-	-	(26)	(87)	(135)	(183)	(232)	(279)	(328)	(376)	(425)	(472)
Other expenses (income)	-	-	-	107	206	172	171	181	174	160	21	(55)	29
Other income and expenses	-	-	-	82	119	38	(12)	(51)	(105)	(167)	(355)	(480)	(443)
Earnings before income and related taxes	-	-	-	10,615	41,740	156,156	134,717	269,471	303,874	151,374	125,810	124,383	116,010
Income and related taxes													
Current	-	-	-	-	-	14,825	35,981	89,108	128,978	77,904	70,446	71,610	68,527
Future	-	-	-	4,000	15,728	44,017	14,782	12,433	(14,473)	(20,864)	(23,039)	(24,740)	(24,812)
Total income and related taxes	-	-	-	4,000	15,728	58,842	50,763	101,541	114,505	57,040	47,407	46,870	43,714
Net earnings	-	-	-	6,615	26,012	97,314	83,953	167,930	189,370	94,334	78,402	77,514	72,296

Statements of Retained Earnings at December 31: (in thousands of U.S. dollars)	**2009**	**2010**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**
Retained earnings (deficit), beginning of period	(76)	(76)	(76)	(76)	6,539	14,309	-	-	-	-	-	-	-
Net earnings (loss)	-	-	-	6,615	26,012	97,314	83,953	167,930	189,370	94,334	78,402	77,514	72,296
Dividends	-	-	-	-	(18,241)	(111,623)	(83,953)	(167,930)	(189,370)	(94,334)	(78,402)	(77,514)	(72,296)
Retained earnings (deficit), end of period	(76)	(76)	(76)	6,539	14,309	-	-	-	-	-	-	-	-

Statements of Cash Flows at December 31: (in thousands of U.S. dollars)	**2009**	**2010**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**
Operating activities													
Net earnings (loss)	-	-	-	6,615	26,012	97,314	83,953	167,930	189,370	94,334	78,402	77,514	72,296
Items not affecting cash													
Depreciation and amortization	-	-	-	12,800	56,749	78,141	91,417	94,844	89,145	88,441	93,973	99,063	99,644
ARO accretion	-	-	-	91	190	754	807	867	926	992	1,058	1,083	1,119
Future income and related taxes	-	-	-	4,000	15,728	44,017	14,782	12,433	(14,473)	(20,864)	(23,039)	(24,740)	(24,812)
Loss (gain) on settlement of ARO	-	-	-	-	-	-	-	-	-	-	(138)	(202)	(96)
	-	-	-	23,505	98,679	220,226	190,960	276,074	264,968	162,903	150,257	152,717	148,150
Change in working capital:													
Accounts receivable	-	-	-	(22,723)	75	(29,643)	8,952	(36,106)	28,525	10,203	545	(9,945)	9,574
Interest receivable	-	-	-	(13)	(12)	(12)	(12)	(12)	(12)	(12)	(12)	(12)	(12)
Supplies	-	-	(1,680)	(7,953)	(861)	(1,136)	378	(640)	1,166	225	126	1,657	1,557
Finished product inventories (cash)	-	-	-	(8,126)	982	(6,337)	(1,249)	2,684	894	(6,328)	(956)	6,063	(1,049)
Work in process inventory (cash)	-	-	-	(0)	0	(0)	(0)	0	0	(11)	(8)	(57)	75
IGV recoverable	(1,610)	(21,154)	(16,630)	20,363	11,024	20,090	(883)	(306)	69	(158)	89	(42)	195
Deposits, prepaids and others	-	-	-	-	-	-	-	-	-	-	-	-	-
Accrued charges	-	-	-	-	-	-	-	-	-	-	-	-	-
Accounts payable	-	-	1,797	8,692	970	2,028	(448)	724	(1,198)	(211)	(126)	(1,685)	(1,647)
Interest payable	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxes payable	-	-	-	-	(3,335)	6,780	17,340	27,030	8,171	(52,615)	5,304	4,734	2,342
Total change in working capital	(1,610)	(21,154)	(16,513)	(9,761)	8,844	(8,228)	24,078	(6,626)	37,615	(48,908)	4,962	712	11,036
Decrease (increase) in ore in stockpile	-	-	(287)	(27,496)	(28,381)	(11,757)	4,604	5,122	(7,046)	(247)	8,395	19,804	26,718
Closure and rehabilitation costs	-	-	-	-	-	-	-	-	-	-	(344)	(506)	(239)
Cash provided by operating activities	(1,610)	(21,154)	(16,800)	(13,752)	79,142	200,241	219,642	274,570	295,538	113,748	163,269	172,728	185,665
Financing activities													
Advances	-	-	-	-	-	-	-	-	-	-	-	-	-
Repayment of advances	-	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loan drawdowns	-	-	-	-	-	-	-	-	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-	-	-	-	-	-	-	-	-
Senior debt drawdowns	-	-	-	-	-	-	-	-	-	-	-	-	-
Senior debt repayments	-	-	-	-	-	-	-	-	-	-	-	-	-
Equity contribution	12,188	172,299	313,464	241,591	9,540	-	-	-	-	15,887	-	-	-
Return of capital	-	-	-	-	-	(85,019)	(130,815)	(100,651)	(97,526)	(31,200)	(81,267)	(93,123)	(110,076)
Dividends	-	-	-	-	(18,241)	(111,623)	(83,953)	(167,930)	(189,370)	(94,334)	(78,402)	(77,514)	(72,296)
Cash provided by (used in) financing activities	12,188	172,299	313,464	241,591	(8,701)	(196,641)	(214,768)	(268,581)	(286,896)	(109,647)	(159,669)	(170,637)	(182,372)
Investing activities													
Mine closure guarantee security	-	-	-	(1,109)	(1,055)	(1,055)	(1,055)	(1,055)	(1,056)	(1,056)	(1,056)	(1,056)	(1,057)
Restricted funds	-	-	-	-	-	-	-	-	-	-	-	-	-
Capital expenditures	(10,577)	(151,145)	(296,663)	(226,731)	(69,385)	(2,545)	(3,819)	(4,934)	(7,586)	(3,045)	(2,544)	(1,035)	(2,236)
Cash provided by (used in) investing activities	(10,577)	(151,145)	(296,663)	(227,839)	(70,440)	(3,600)	(4,874)	(5,989)	(8,642)	(4,101)	(3,600)	(2,090)	(3,293)
Increase (decrease) in cash and equivalents	-	-	-	-	-	(0)	0	(0)	(0)	(0)	(0)	(0)	0
Cash and equivalents - beginning of period	0	0	0	0	0	0	(0)	0	0	(0)	(0)	(0)	(0)
Cash and equivalents - end of period	0	0	0	0	0	(0)	0	0	(0)	(0)	(0)	(0)	(0)

Balance Sheets as at December 31	**2022**	**2023**	**2024**	**2025**	**2026**
(in thousands of U.S. dollars)					
Assets					
Current Assets					
Cash and equivalents	-	-	-	-	-
Restricted funds	-	-	-	-	-
Accounts receivable	29,645	29,588	(0)	(0)	(0)
Taxes receivable	-	10,225	0	0	0
IGV recoverable	204	-	741	1,575	1,894
Interest receivable	134	137	-	-	-
Supplies	3,283	1,159	-	-	-
Finished product inventory	10,433	0	0	0	0
Work in process inventory	1	1	1	1	1
Ore in stockpile	6,375	(0)	(0)	(0)	(0)
Deposits, prepaids, others	-	-	-	-	-
Total current assets	50,074	41,109	742	1,575	1,895
Non-current assets					
Mine closure guarantee security	11,666	11,926	11,496	5,710	-
Capital assets					
Land and mining concessions	974	-	-	-	-
Construction in progress	-	-	-	-	-
Mine development	1,099	-	-	-	-
Plant and equipment	15,411	749	749	749	749
Preproduction intangibles	2,909	(0)	(0)	(0)	(0)
ARO asset	298	(0)	(0)	(0)	(0)
Net capital assets	20,690	749	749	749	749
Total non-current assets	32,356	12,675	12,245	6,459	749
Total assets	82,430	53,784	12,987	8,034	2,643
Liabilities					
Current liabilities					
Accrued charges	-	-	-	-	-
Accounts payable	4,144	1,533	-	-	-
Advances	-	-	-	-	-
Interest payable	-	-	-	-	-
IGV payable	-	357	-	-	-
Taxes payable	23,322	-	-	-	-
Senior debt	-	-	-	-	-
Shareholder loans	-	-	-	-	-
Total current liabilities	27,466	1,890	-	-	-
Non-current liabilities					
Long-term debt	-	-	-	-	-
Future income and related taxes	(25,079)	(1,290)	(751)	(237)	(10)
Asset retirement obligation	16,594	15,359	8,138	2,383	98
Total non-current liabilities	(8,485)	14,068	7,387	2,146	88
Total liabilities	18,980	15,958	7,387	2,146	88
Shareholders' Equity					
Capital stock	63,449	37,825	5,143	5,297	2,355
Retained earnings	-	-	456	591	200
Total shareholders' equity	63,449	37,825	5,600	5,888	2,555
Total liabilities and shareholders' equity	82,430	53,784	12,987	8,034	2,643

Statements of Earnings at December 31: (in thousands of U.S. dollars)	**2022**	**2023**	**2024**	**2025**	**2026**
Gross revenue	369,503	285,809	-	-	-
Realization costs	58,804	50,797	-	-	-
Net revenue	310,699	235,012	-	-	-
Costs					
Production costs	94,089	61,192	-	-	-
Government royalty	8,758	6,373	-	-	-
Asset retirement obligation accretion	1,170	1,122	874	387	69
Depreciation and amortization	51,658	25,581	-	-	-
Costs of Sales	155,676	94,268	874	387	69
Earnings before other income, expenses and taxes	155,023	140,744	(874)	(387)	(69)
Other income and expenses					
Interest expense	-	-	-	-	-
Interest income	(520)	(541)	-	-	-
Other expenses (income)	(287)	(627)	(2,307)	(1,750)	(671)
Other income and expenses	(807)	(1,168)	(2,307)	(1,750)	(671)
Earnings before income and related taxes	155,830	141,913	1,433	1,364	602
Income and related taxes					
Current	66,831	29,686	-	-	-
Future	(8,111)	23,789	540	514	227
Total income and related taxes	58,719	53,475	540	514	227
Net earnings	97,111	88,438	893	850	375

Statements of Retained Earnings at December 31: (in thousands of U.S. dollars)	**2022**	**2023**	**2024**	**2025**	**2026**
Retained earnings (deficit), beginning of period	-	-	-	456	591
Net earnings (loss)	97,111	88,438	893	850	375
Dividends	(97,111)	(88,438)	(437)	(715)	(766)
Retained earnings (deficit), end of period	-	-	456	591	200

Statements of Cash Flows at December 31: (in thousands of U.S. dollars)	**2022**	**2023**	**2024**	**2025**	**2026**
Operating activities					
Net earnings (loss)	97,111	88,438	893	850	375
Items not affecting cash					
Depreciation and amortization	51,658	25,581	-	-	-
ARO accretion	1,170	1,122	874	387	69
Future income and related taxes	(8,111)	23,789	540	514	227
Loss (gain) on settlement of ARO	(362)	(674)	(2,313)	(1,755)	(672)
	141,466	138,256	(6)	(4)	(2)
Change in working capital:					
Accounts receivable	10,898	57	29,588	-	-
Interest receivable	(12)	(3)	137	-	-
Supplies	3,878	2,124	1,159	-	-
Finished product inventories (cash)	6,056	7,368	-	-	-
Work in process inventory (cash)	(0)	-	-	-	-
IGV recoverable	(444)	561	(1,099)	(834)	(319)
Deposits, prepaids and others	-	-	-	-	-
Accrued charges	-	-	-	-	-
Accounts payable	(4,753)	(2,611)	(1,533)	-	-
Interest payable	-	-	-	-	-
Taxes payable	7,570	(33,546)	10,225	-	-
Total change in working capital	23,194	(26,050)	38,477	(834)	(319)
Decrease (increase) in ore in stockpile	5,982	4,590	-	-	-
Closure and rehabilitation costs	(906)	(1,684)	(5,782)	(4,387)	(1,681)
Cash provided by operating activities	169,736	115,112	32,688	(5,225)	(2,002)
Financing activities					
Advances	-	-	-	-	-
Repayment of advances	-	-	-	-	-
Shareholder loan drawdowns	-	-	-	-	-
Shareholder loan repayments	-	-	-	-	-
Senior debt drawdowns	-	-	-	-	-
Senior debt repayments	-	-	-	-	-
Equity contribution	-	-	3,309	3,570	1,044
Return of capital	(71,337)	(25,624)	(35,990)	(3,417)	(3,986)
Dividends	(97,111)	(88,438)	(437)	(715)	(766)
Cash provided by (used in) financing activities	(168,448)	(114,062)	(33,118)	(562)	(3,708)
Investing activities					
Mine closure guarantee security	(1,057)	(261)	430	5,786	5,710
Restricted funds	-	-	-	-	-
Capital expenditures	(231)	(790)	-	-	-
Cash provided by (used in) investing activities	(1,288)	(1,050)	430	5,786	5,710
Increase (decrease) in cash and equivalents	(0)	0	(0)	-	-
Cash and equivalents - beginning of period	(0)	(0)	(0)	(0)	(0)
Cash and equivalents - end of period	(0)	(0)	(0)	(0)	(0)