Metallurgical Corporation of China Ltd. ("MCC")

Global Mining Assets

Independent Technical Review Report ("ITRR")

July 2009
Project No. 3249M

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The Directors
China Metallurgical Group Corporation
11, Gaoliangqiaoxie Street,
Haidian District,
Beijing, P.C 100081

July 21, 2009

RE: INDEPENDENT TECHNICAL REVIEW REPORT

Dear Sirs,

Minarco-MineConsult ("M-MC") has carried out an Independent Technical Review ("ITR") of the assets of "Metallurgical Corporation of China Ltd." ("MCC" or "the Company"). The results of the ITR are summarised in the attached Independent Technical Review Report ("ITRR").

The assets reviewed ("Relevant Assets") consist of International and Domestic China Assets as follows:

International Assets

- Minera Sierra Grande Iron Ore Mine in Argentina;
- Ramu Nickel Project in Papua New Guinea;
- Aynak Copper Project in Afghanistan;
- Duddar Lead-Zinc Project in Pakistan;
- Saindak Copper-Gold Mine in Pakistan; and
- Cape Lambert Iron Ore Project in Western Australia.

Domestic China Assets

- Jinchang Mining Assets;
  - Guanfen Iron Ore Mine
  - Wutaigou Iron Ore Mine
  - Songzhangzi Iron Ore Mine
  - Jinchang Processing Plant
- Hongda Iron Ore Mine;
- Xiangxi Carbon Shale — Vanadium Project; and
- Nonggeshan Lead Zinc Project.

The following report (the ITRR) has been prepared by M-MC in connection with the ITR conducted by M-MC on the Relevant Assets. The report sets out the process and conclusions of M-MC's review and M-MC consents to its inclusion as required in MCC's document.

M-MC has conducted its review and preparation of this report in accordance with the requirements of Chapter 18 of the Listing Rules of the Stock Exchange of Hong Kong Limited, with the exception of the requirements set out in Listing Rule 18.09 item (8) which relates to the provision of a two-year working capital statement. This report is also in compliance with:

- The "Australasian Code for Reporting Mineral Resources and Ore Reserves" (2004 edition published by the Joint Ore Reserves Committee ("JORC") of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and the Minerals Council of Australia" (the "JORC Code"), for determining resources and reserves; and

- The Code and Guidelines for technical assessment and/or valuation of mineral and petroleum assets and mineral and petroleum securities for Independent Expert Reports (the "Valmin Code").

M-MC carried out a review of Mineral Resources as reported under the Chinese Mineral Reporting Standards and has compared these results in broad terms with the reporting requirements of the JORC Code. Where mineral resources and reserve estimates were not in compliance with the recommendations of the JORC guidelines, M-MC have used the terms 'In Situ Quantities' and 'Mineable Quantities' respectively.

M-MC operates as an independent technical consultant providing resource evaluation, mining engineering and mine valuation services to the resources and financial services industries. This report was prepared on behalf of M-MC by technical specialists, details of whose qualifications and experience are set out in ***Annexure A***.

M-MC has been paid, and has agreed to be paid, professional fees for its preparation of this report. However, none of M-MC or its directors, staff or sub-consultants who contributed to this report has any interest in:

- The Company; or
- The Relevant Assets.

Drafts of this report were provided to the Company, but only for the purpose of confirming the accuracy of factual material and the reasonableness of assumptions relied upon in the report. The review was based mainly on information provided by MCC, either directly from the data room or from project sites and other offices. The report is based on information made available to M-MC before March 20, 2009.

The work undertaken is a technical review of the information provided as well as that obtained during such inspections as M-MC considered appropriate to prepare the report. It specifically excludes all aspects of legal issues, commercial and financing matters, land titles and agreements, excepting such aspects as may directly influence technical, operational or operation cost issues. M-MC have not given any opinion on political risk associated with the Relevant Assets.

In M-MC's opinion, the information provided by MCC was reasonable and nothing discovered during the preparation of the report suggested that there was any significant error or misrepresentation in respect of that information.

M-MC has independently assessed the Relevant Assets by reviewing pertinent data, including mineral resources, future exploration plans, development potential, and potential mining issues. All opinions, findings and conclusions expressed in the report are those of M-MC and its specialist advisors.

The key conclusions from M-MC's review of the Relevant Assets are summarised as follows:

Minera Sierra Grande Iron Ore Mine: Argentina.

MCC Equity Stake:	70.0%
Status:	Refurbishment of the mine, processing and port facilities commenced in 2005 with limited production. At the time of M-MC's last site visit in March 2009 the bulk of the refurbishment had been completed with production reaching 312kt in 2008. Port facilities will be ready for shipment in April 2009.
Commodity:	Iron ore, magnetite with some hematite
In Situ Quantities:	6 mining leases grouped in 3 deposits, i.e. South, East and North. Currently the South Deposit is the only producing area and hosts known resources of 199Mt (at approximately 57.5%TFe)
Mineable Quantities:	South Deposit estimated at 31Mt based on detailed mine drilling above the 410 mine level. Much larger reserve based on known resources below this depth has yet to be estimated
Mining Method:	Underground, sub-level stoping production at 310ktpa
Product:	Iron magnetite concentrate, delivered to owner operated Panamax vessel loading dock
Designated Production Capacity:	One crushing and ore processing plant — concentrate slurry (process capacity 3.5Mtpa)
Risks:	Lack of a geological model, and a life-of-mine (LOM) plan Current and forecast production is hampered by processing limitations linked to unresolved water and power supply constraints Iron ore prices Freight costs
Opportunities:	Significant increase in reserves, South Deposit Additional resources from East and North Deposits Additional drilling and revised estimates of resources & reserves required Increase in ROM production from 310ktpa to 2,800ktpa by 2011 achievable with current equipment.

Ramu Nickel Project: Papua New Guinea.

MCC Equity Stake:	Effective stake 51.85% (61% of an affiliated company holding an 85% stake in the project)
Status:	Mine development project, predicted production commencement late 2009/ early 2010
Commodity:	Lateritic Ni with minor Co
Mineral Resources:	143.2Mt at 1.01% Ni and 0.10% Co
Ore Reserves:	75.7Mt at 0.91% Ni and 0.10% Co
Mining Method:	Open cut mining at 3.6Mtpa

Product:	Ni-Co hydroxide
Designated Production Capacity:	31ktpa Ni and 3.2ktpa Co
Risks:	Medium to low grade lateritic ore Complex and difficult extraction of Ni and Co from lateritic ore containing 0.8 to 1% Ni Slower than expected ramp up to full production capacity Cost of sulphur Heavy oil price Nickel price
Opportunities:	Additional Mineral Resources Higher grade zones in Ore Reserves. Additional drilling and revised estimates of Resources & Reserves required Recovery and sale of chromite concentrate

Aynak Copper Project: Afghanistan.

MCC Equity Stake:	75.0% (remaining 25% is owned by the Jiangxi Copper Group of China)
Status:	Pre Feasibility
Commodity:	Cu
In Situ Quantities:	483.4Mt at 1.85% Cu (combined Middle and West District at various Cu cut-off grades)
Mineable Quantities:	Open cut (Middle District) 155.4Mt at 1.13% Cu Underground (West District) 194.1Mt at 1.3% Cu
Mining Method:	Open cut at 9.9Mtpa and underground natural and sub-level caving at 9.9Mtpa (by 2018)
Product:	Cu concentrate, cathode Cu and sulphuric acid
Designated Production Capacity:	987ktpa Cu concentrate
Risks:	Some historical data for resources is missing Geotechnical parameters for the open cut and underground block cave have not been assessed in detail Mine planning and scheduling is at a preliminary stage Metallurgical studies are in the early stages and require further confirmation Capex and Opex provided by MCC reflect the preliminary status of the study and will change as mining and processing parameters are refined
Opportunities:	Resources to date remain open at depth Open cut recovery and dilution could be improved through careful grade control, resulting in reduced ROM tonnes and improved grades. Improvements to the proposed metallurgical process are achievable with further studies

Duddar Lead-Zinc Project: Pakistan.

MCC Equity Stake:	Effective stake 40.8% (MCC holds 80% of the Duddar Mining Development Company which holds 51% of the mine)
Status:	Mine development project, ready for underground production commencement
Commodity:	Stratiform Pb-Zn
Mineral Resources:	14.48Mt at 9.9% Zn, 3.4% Pb
Mineable Quantities:	Underground 9.13 Mt at 9.3% Zn and 3.0% Pb
Mining Method:	Underground at 660ktpa
Product:	Zn and Pb concentrates
Designated Production Capacity:	Zn Concentrate 97.2ktpa at 55% Zn, Pb Concentrate 22ktpa at 67% Pb.
Risks:	Structural deformity, many faults, discontinuous ore blocks, unstable mining conditions Metal prices, particularly zinc
Opportunities:	Medium — high grade Pb-Zn Continuity of mineralisation — stratiform Additional resources and reserves at depth Additional drilling and revised estimates of Resources & Reserves required Recovery of barite concentrate

Saindak Copper and Gold Project: Pakistan.

MCC Equity Stake:	Sole exploiter through a rental agreement with the Balochistan government. (10 year mining licence)
Status:	Operating Open cut mine
Commodity:	Porphyry low to medium grade Cu-Au
Mineral Resources:	50.9Mt at 0.47% Cu and 0.46g/t Au (cog 0.25% Cu), as at December 2008.
Mineable Quantities:	Open cut: 49.7Mt at 0.45% Cu and 0.47g/t Au. (cog 0.25% Cu), as at December 2008.
Mining Method:	Open cut 5.3Mtpa
Product:	Cu concentrate and cathode copper
Designated Production Capacity:	81.5kt of concentrate per annum at 22.4% Cu, 20.6g/t Ag (based on 2008 production).
Risks:	Reserves estimates do not take into account mining loss or dilution Low to medium grade deposit Copper and precious metal prices

Opportunities:	Production grades of gold and silver are above forecasts from the model Additional Ore Reserves (UG or OC) below currently designed open cut (626mRL) Additional Resource and Reserves at other deposits (North and West) and higher grades. Additional drilling and revised estimates of Resources & Reserves for the North and West deposits required Improved metallurgy particularly copper concentrate grade

Cape Lambert Iron Ore Project: Western Australia.

MCC Equity Stake:	100%.
Status:	Preliminary Pre-Feasibility Study completed. Currently working towards a Bankable Feasibility Study due for completion in Q2 2010. Current plans are for project construction to commence in 2011, following successful completion of the relevant studies and granting of approvals.
Commodity:	Iron ore, magnetite
Mineral Resources:	1.9Bt at 30.7% Fe using a 20% Fe cut-off
Mineable Quantities:	Preliminary studies indicate a Mineable Quantities of 1.31Bt at 29.5% Fe using a 20% Fe cut-off. This estimate includes both Indicated and Inferred Resources. Inferred Resources cannot be included in Ore Reserve estimates under JORC.
Mining Method:	Open Cut, planned production of 48Mtpa.
Product:	Iron magnetite concentrate, delivered to Panamax vessels by barging through an owner operated loading dock.
Designated Production Capacity:	One crushing and ore processing plant - concentrate slurry (process capacity 48Mtpa at 29.5% Fe for 15Mt of magnetite concentrate at 65% Fe)
Risks:	40% of the Resources currently included in the Mineable Quantities are of Inferred category. These will need to be upgraded to Indicated or removed from the Mineable Quantities in later studies to meet the requirements of Ore Reserves under the JORC code. Mine planning and scheduling is at a preliminary stage and detailed analysis needs to be undertaken to confirm the project economics. Forecast concentrate grade of 65% Fe is not supported by the resource model, which estimated a concentrate Fe grade of 61.8% for the deposit based on comprehensive DTR analyses. Variations in concentrate Fe grade throughout the deposit have not been considered in the production schedule with the result that lower than average concentrate Fe grades will be produced in the initial years of the operation making the production forecasts optimistic. Processing test work and Resource estimates to date have not defined the various ore type characteristics in sufficient detail. This work is crucial to managing the process response of different forms of silica other than quartz in the final product. This could have a significant impact on the value and attractiveness of the final saleable product.

	Large scale plant may prove challenging to construct and commission. There may be a mismatch between the ramp up in mining and processing production. Infrastructure (1 rail, 2 powerlines and 1 gas pipeline) will need to be relocated to access the planned pit. Costs and approvals for this work have not been obtained by MCC to date and are not included in the CAPEX estimates provided by the company. For project evaluation longer term iron ore prices should be used (i.e. US$60/t). Current Exploration Licence (EL) does not allow for Mining and various environmental (including an Environmental Impact Statement (EIS)), native title and governmental approvals will need to be gained before a Mining Licence (ML) will be granted. This process for a large scale project can take a number of years.
Opportunities:	Recovery of the non magnetic iron oxide may be achievable with minimal CAPEX. This opportunity if confirmed by testing could enhance revenue and project economics.

Jinchang Mining Assets: China.

MCC Equity Stake:	MCC's effective equity stake in the Jinchang Assets is 85.1% (85.1% of a subsidiary owning an 100% stake in the project)
Assets:	Guanfen Mine Wutaigou Mine Songzhangzi Mine Jinchang Processing Plants

Guanfen Iron Ore Mine: China.

Status:	Dormant since late 2008
Commodity:	Iron ore, magnetite, pre-concentrate product
In Situ Quantities:	known Soft Ore resources depleted
Mining Method:	Open cut at 300ktpa (Soft Ore Only)
Product:	Magnetite powder pre-concentrate at approximately 20%TFe, sent to Jinchang Plants for further upgrading at final Fe concentrate at 65% TFe.
Risks:	Further work required to confirm viability of underground mining of Hard Ore resource Iron ore prices Cost of power
Opportunities:	Open cut depth extended to extract hard ore

Wutaigou Iron Ore Mine: China.

Status:	Dormant since late 2008
Commodity:	Iron ore, magnetite, pre-concentrate product
In Situ Quantities:	193.3kt at 29.7%TFe (Hard Ore)

Mining Method:	Open cut at 300ktpa (Soft Ore Only)
Product:	Magnetite powder pre-concentrate at approximately 20%TFe, sent to Jinchang Plants for further upgrading.
Risks:	Further work required to confirm mineable quantities of Hard Ore Iron ore prices
Opportunities:	Extend open cut depth to extract Hard Ore Improved metallurgy, particularly pre-concentrate grade

Songzhangzi Iron Ore Mine: China.

Status:	One Operating UG Mine
Commodity:	Iron ore, magnetite, concentrate product
In Situ Quantities:	475.7kt at 32.5% TFe (Hard Ore).
Mining Method:	Underground (short hole) at 100ktpa
Product:	ROM ore at approximately 30% Fe
Risks:	Thin lodes will impact on mining recovery and dilution UG development is outside mining districts (boundaries) Unreliable estimates of mining recovery factors for UG extraction Iron ore prices Cost of power
Opportunities:	Development of additional underground resources and reserves. Additional OC in situ quantities as extensions along strike of lodes and ore zones outside mining districts Exploration may identify additional lodes, thicker ore zones and deeper ore Improved metallurgy, particularly concentrate grade

Jinchang Processing Plant: China.

Status:	Operating
Commodity:	Iron ore, magnetite, pre-concentrate and ROM ore
Designated Production Capacity:	March 2009: 400ktpa ROM ore, 200ktpa magnetite concentrate powder at 65% TFe. Plans and infrastructure in place for 1.1Mt ROM ore and 500ktpa of concentrate powder
Risks:	Iron ore prices
Opportunities:	Processing third party sourced ore to improve economy of scale Improved metallurgy, particularly concentrate grade

Hongda Iron Ore Mine: China.

MCC Equity Stake:	MCC's effective equity stake in the Hongda Mine is 48.6% (90% of a subsidiary owning a 54% stake in the project)
Status:	Operating
Commodity:	Iron ore, magnetite, concentrate product

In Situ Quantities:	73.5Mt at 12.64% TFe and 5.11% mFe
Mineable Quantities:	Open cut : 60Mt (site estimated)
Mining Method:	Open cut at 12Mtpa
Product:	Magnetite powder concentrate at approximately 61%TFe
Designated Production Capacity:	672ktpa magnetite concentrate powder at 60% TFe
Risks:	Lower ore grades at base of current pit Iron ore prices
Opportunities:	Exploration for adjacent replacement ore reserves Improved metallurgy, particularly concentrate grade

Xiangxi Carbon Shale Project: China.

MCC Equity Stake:	80.0%
Status:	Project Planning
Commodity:	Vanadium (V) and Carbonaceous Shale
In Situ Quantities:	71.1Mt at 3,507J/g of Carbonaceous Shale Upper Mining section 17.147Mt at 0.79% V_2O_5 , 0.7% V_2O_5 cut off grade of Vanadium ore in Upper Mining Section
Mineable Quantities:	13.0Mt at 0.79% V_2O_5 of Vanadium ore in mining section 2
Mining Method:	Proposed open cut and underground board and pillar at 570ktpa (open cut only)
Product:	Vanadium pentoxide (V_2O_5), carbonaceous shale, cement slag
Designated Production Capacity:	2ktpa of vanadium pentoxide (V_2O_5) with mining production of 570ktpa Co-product of slag used for cement, waste heat from vanadium processing for use in power generation
Risks:	Mining method may be controlled by environmental requirements No detailed mining plans or schedules available Fissile shales may cause geotechnical issues in underground mine Validation of heat (energy) values of carbonaceous shale, definition of basis of analysis (gross or net) Suitability of ore for power generation
Opportunities:	Large additional carbonaceous shale resources (mining Section 1) for domestic power generation

Nonggeshan Lead Zinc Project: China.

MCC Equity Stake:	MCC's effective equity stake in the project is 49.9% (97.83% of a subsidiary owning a 51% stake in the project)
Status:	Project Development
Commodity:	Pb, Zn minor Ag
In Situ Quantities:	20.4Mt in Orebody I+II+III, at 1.8% Pb, 1.4% Zn, 16.6g/t Ag

Mineable Quantities:	5.709Mt at 2.5% Pb, 1.5% Zn, 17.4g/t Ag with contained metal of 141kt Pb, 84kt Zn and 99t Ag, of 111b and 122b Chinese Code Mineable Quantities, within orebody I An additional 12.2Mt of 333 Chinese Code Mineable Quantities represents additional potential Mineable Quantities within orebody I
Mining Method:	Underground at 600ktpa
Product:	2 concentrates; one Pb with Ag and minor Zn and one Zn with minor Pb
Designated Production Capacity:	12.7ktpa Pb & Ag concentrate at 62% Pb, 6% Zn and 464g/t Ag, 12.5ktpa Zn concentrate at 45% Zn.
Risks:	Only 32% of in situ quantities are 111b or 122b status Lower mining recovery and higher dilution related to fault zones bounding Orebody I Unstable wall rocks bounding orebody I Metal prices Operating costs
Opportunities:	Additional 0.68 Mt of Exploration Targets Definition of low grade zones as potential additional ore. Pre-concentration of ores to lower both Capital and Operating Costs

Yours faithfully

Andrew Ryan
General Manager North Asia
Minarco-MineConsult

1 OVERVIEW AND EXECUTIVE SUMMARY

1.1 INTRODUCTION

MCC's mines and projects are discussed throughout this report in various sections. The non-China based assets (referred to as International Assets in this report) tend to be more substantial and therefore they are described in more detail in this report than the smaller scale, China based assets (referred to as Domestic Assets). A brief summary of International and China based assets are given in ***Tables 1.1*** and ***1.2*** respectively

Table 1.1 — International Operations Project Status

International Assets	Location	Commodity	Status	Mining Method
Minera Sierra Grande Iron Ore Mine	Argentina	Iron ore	Operating	UG
Ramu Nickel Laterite Project	Papua New Guinea	Nickel laterite	Development	OC
Aynak Copper Project	Afghanistan	Copper	Development	OC/UG
Duddar Lead Zinc Project	Pakistan	Lead-Zinc	Development	UG
Saindak Copper Gold Mine	Pakistan	Copper-Gold	Operating	OC
Cape Lambert Iron Ore Project	Western Australia	Iron ore	Development	OC

Table 1.2 — Domestic China Operations Project Status

Domestic Assets	Location	Commodity	Status	Mining Method
Jinchang Mining Assets				
Guanfen Iron Ore	Liaoning Province	Iron Ore (Soft)	Dormant	OC[1]
Wutaigou Iron Ore	Liaoning Province	Iron Ore (Hard)	Dormant	OC[1]
Songzhangzi Iron Ore	Liaoning Province	Iron Ore (Hard)	Operating	UG
Hongda Iron Ore	Inner Mongolia	Iron Ore	Operating	OC
Xiangxi Carbon Shale	Hunan Province	Vanadium — Carbonaceous Shale	Exploration	OC/UG
Nonggeshan Lead Zinc	Sichuan Province	Lead — Zinc	Development	UG

*Note 1: The Jingchang opencut mines in **Table 1.2** are planning to mine deeper resources in the future, and will therefore move to underground mining over the next two years if iron prices recover.*

1.2 LIMITATIONS AND EXCLUSIONS

The review was based on various reports, plans and tabulations, which were translated into English. The data reviewed in most instances did not include detailed exploration samples or assay data as this information was unavailable to M-MC.

The report is based mainly on information provided by the Company either directly from the mine sites and other offices, or from reports by other organisations whose work is the property of the Company. The Company has not advised M-MC of any material change, or any event likely to cause material change, to the operations or forecasts since the date of the most recent inspection of the Relevant Asset.

The work undertaken for this report is that required for a technical review of the information coupled with such inspections as the Team considered appropriate to prepare this report. It specifically excludes all aspects of legal issues, commercial and financing matters, land titles, agreements, excepting such aspects as may directly influence technical, operational or operational cost issues. M-MC have not given any opinion on political risk associated with the Relevant Assets.

M-MC has also specifically excluded making any comments on the competitive position of the Assets compared with other similar and/or competing assets. M-MC advises that any potential investors make their own assessment of the competitive position of the Relevant Assets in the market.

1.3 STUDY METHODOLOGY

The study was completed in several stages as follows:

- Preparatory Work and Translation — Limited technical information was made available to the Team prior to the 1st site visits in 2007 and 2008. Updated data was provided prior to the second site visit in 2009.
- Site Visits — The Assets were visited by M-MC's various professional experts in two main phases from October 2007 - April 2008 and again in March 2009. The Pakistan Assets (Duddar Pb-Zn Project, and Saindak Cu-Au Mine) and the Afghanistan Asset (Aynak Cu Project) were not visited due to perceived political instability and associated security risks at the time of compiling of this report. In addition, M-MC did not visit the Nonggeshan Pb-Zn Project because of its greenfields nature and relative remoteness.
- Document and Reports — Copies of the Geology Reports, Feasibility Studies and associated supporting documents were made available to M-MC's office for translation and review by experts in M–MC's various offices around the world.
- Preparation of the Report — M-MC prepared this report and provided drafts to the Company and its specialist advisers.

The basis for the comments and forecasts in this report is information compiled by enquiry and verbal comment from the Company, cross checked where possible with hard data or by comment from more than one source. Where there was conflicting information on issues, the Team used its professional judgment to resolve the issues where possible.

Generally, the data available was sufficient for M-MC to complete the scope of work. The quality and quantity of data available, and the co-operative assistance, in M-MC's view, showed a willingness by the Company to assist the ITR process.

1.4 DESCRIPTION OF ASSETS

The general locations of the assets are shown in ***Figure 1.1 and 1.2***. More detailed location plans for each of the assets are shown individually in **Sections 2** through to **Section 10** of the report.

1.5 SUMMARY OF RESOURCES AND ORE RESERVES

A summary of Resources and Ore Reserves has been prepared by M-MC using the guidelines of the JORC Code, with the estimates for the International and Chinese assets shown in ***Tables 1.3*** and ***1.4*** respectively**.** Resources for all mines and projects are presented together in International and Domestic groups. However, because there are different commodities, resource estimates for each project are not additive.

Resources and reserve estimates by the relevant Chinese Institute based on the Chinese Code are accepted by the Hong Kong Stock Exchange (HKEX). In M-MC's opinion, the Chinese estimates are reasonable, however without validation of exploration and estimation methods, these estimates do not comply with JORC guidelines.

The JORC Code requires that Ore Reserve estimates must apply relevant modifying factors including mining layout, pit shells, recovery and dilution factors to Indicated and/or Measured Resources. Where resource and reserve estimates are not compliant with JORC guidelines M-MC have used terms 'In Situ Quantities' and 'Mineable Quantities' respectively as follows:

Non JORC Mineral Resources	are referred to as	In Situ Quantities
Non JORC Ore Reserves	are referred to as	Mineable Quantities

Most of the projects estimated using the Chinese Code do not estimate Ore Reserves (Mineable Quantities). For these projects it is common for operational management and planning to use In Situ Quantities as indicative "ore reserves". In M-MC's opinion this is not a reasonable practice as "ore reserves" should take into account relevant modifying factors including mining layout, pit shells, recovery and dilution factors.

Where contained metal is estimated, the JORC code requires reporting together with tonnes and grade of Mineral Resource estimates or more ideally with Ore Reserve estimates.

In its report M-MC has referred to the periodic table chemical symbol as opposed to the full metal name. A definition of each symbol used in the report can be found in ***Annexure B***

Mineral Resources (In Situ Quantities) and Ore Reserves (Mineable Quantities) are discussed in more detail in each of the projects chapters.

Table 1.3 — International Assets Resource and Reserve Summary

International Assets	Mineral Resources (Mt)	Ore Reserves (Mt)	Average Grade (Ore Reserves)		Resource Reporting	Reserve Reporting
Minera Sierra Grande	199.0	31.3	57.5%TFe	1.3% P	USGS	USGS
Ramu	143.2	75.7	0.91% Ni	0.1% Co	JORC	JORC
Aynak	483.4	349.5	1.22% Cu		Russian	Chinese
Duddar	14.5	9.1	9.3% Zn	3.0% Pb	JORC	Chinese
Saindak	50.9	49.7	0.45% Cu	0.47g/t Au (2.2–2.6g/t Ag)	JORC	Chinese
Cape Lambert	1,915.0	1,310.0	29.5%Fe		JORC	Chinese

Notes: Mineral Resources include Inferred, Indicated and Measured.

Mineral Resources are inclusive of Ore Reserves (Mineable Quantities).

Minera Sierra Grande Reserve Grade based on resource grades

Saindak (Ag grades) shown as indication only based on historical production results, Ag was not assayed in the drilling or estimated and hence cannot be reported as either Resources or Mineable Quantities.

Cape Lambert Mineable Quantities includes Inferred Resources which are not reportable as Reserves under JORC.

Table 1.4 — Domestic Assets resource (In Situ Quantities) and reserves (Mineable Quantities) Summary

Domestic Assets	In Situ Quantities (kt)	Mineable Quantities (kt)	Average Grade (Mineable Quantities) Fe % #1	Pb %	Zn %	Ag g/t	V_2O_5%	J/g
Jinchang Mining Assets								
Wutaigou Iron Ore (hard)	193.3	n/a	29.7					
Songzhangzi Iron Ore (hard)	475.7	n/a	32.49					
Hongda Iron Ore#2	73,455	60,000	12.64					
Xiangxi Carbon Shale	71,114							3,507
Xiangxi Vanadium#3	17,147	13,000					0.79	
Nonggeshan Lead Zinc#4	20,416	5,709		2.5	1.5	17.4		

Notes: All results are based on the Chinese Code.

Estimates are not reported in accordance with the recommendations of the JORC Code.

In Situ Quantities are inclusive of Mineable Quantities.

Estimates are not precise calculations; therefore estimates have been rounded to appropriate significant figures.

#1 Fe grade quoted are TFe which is often much higher than the actual recovered mFe grade.

#2 Previously reported Mineable Quantities at a higher Fe price.

#3 Vanadium resource reported at $>$0.7% V_2O_5 cog.

#4 An additional 12.2Mt of 333 In Situ Quantities have been considered as Mineable Quantities by the Lanzhou Institute. M-MC has excluded these from its estimate of In Situ Quantities

The locations of the China Domestic Assets are shown in ***Figure 1.2***

Figure 1.1 — General Location Plan — All Mines and Projects

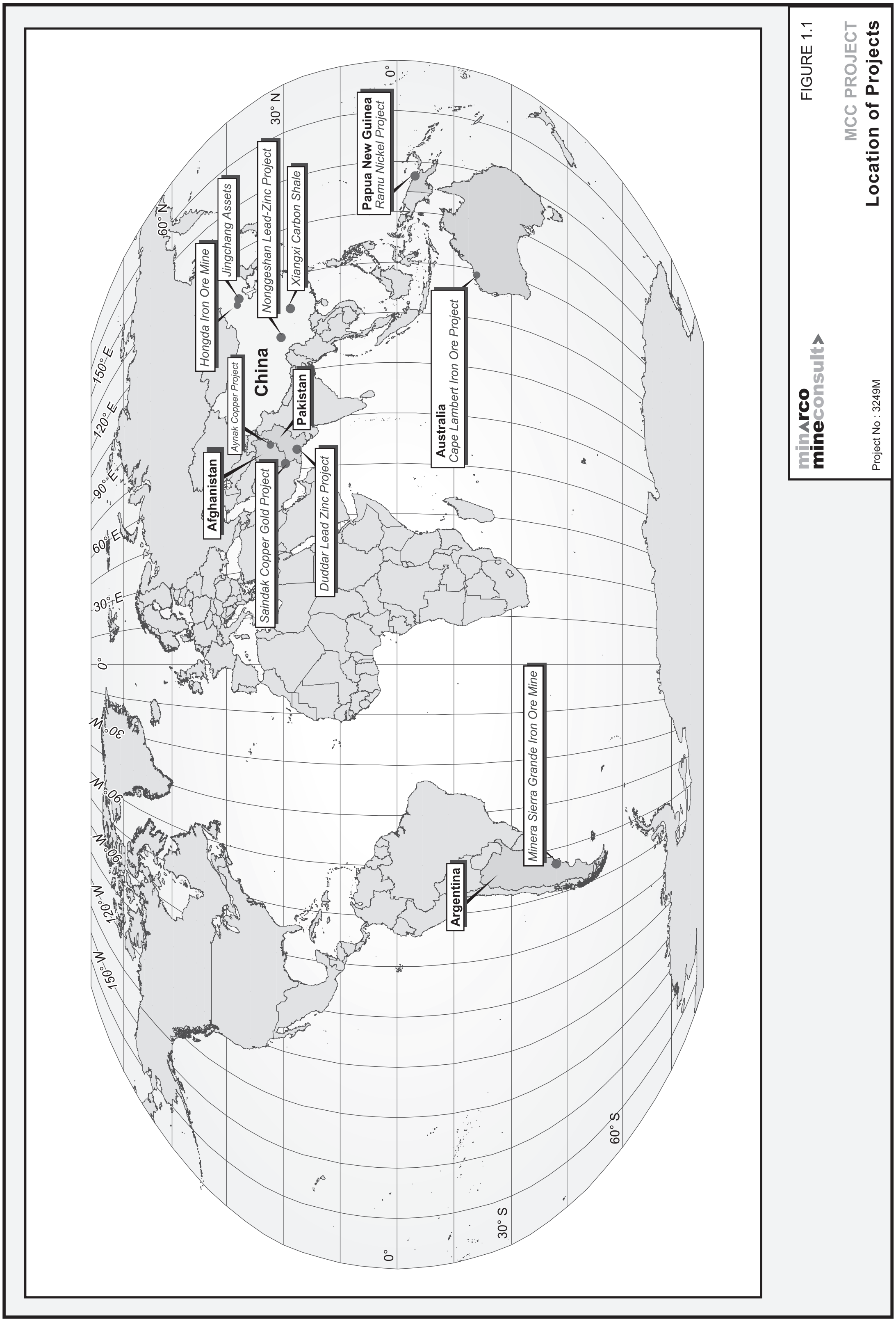


Figure 1.2 — General Location Plan — Chinese Domestic Assets

2 MINERA SIERRA GRANDE IRON ORE MINE

M-MC made a site inspection of this property in December 2007 to review the resources and in April 2008 to review processing and mining. In March 2009 M-MC carried out a final site visit to review recent exploration work on the East orebody and review progress of the refurbishment and re-commissioning. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "1980, Preliminary Technical Report Minera Sierra Grande" prepared by Hierro Patagonia Rionegrino Sociedad Anonima; and
- "2007, Feasibility Study for the Reactivation of Mining and Beneficiation of the Sierra Grande Iron Ore Mine Argentina" prepared by the Northern Engineering and Technology Corporation of MCC.

MCC's effective equity stake in the Minera Sierra Grande Project is 70%.

2.1 BACKGROUND

Minera Sierra Grande, SA (MSG) is an Argentinean company formed to acquire the iron ore mining concession owned by Hierro Patagonia Rionegrino Sociedad Anonima (HIPARSA) through a formal process of a private initiative submitted on September 24, 2004. MCC has a 70% interest in MSG, and thus is the primary owner.

The present mining operation is located approximately 8 km south of the town of Sierra Grande, in the Province of Rio Negro (Patagonia), Argentina. The mining concession, among other assets, includes six mining leases, containing the South, East and North deposits, two existing industrial areas, known as Industrial Area I and Industrial Area II, a 32km long pipeline and a port (Punta Colorada), capable of handling Panamax size vessels. The location of the mine is shown in ***Figure 2.1***.

As a part of the conditions associated with the re-activation of this operation, the local Rio Negro government agreed to the establishment of both power and water facilities. MCC is currently finalising the feasibility of constructing their own gas-fired power facility as no alternative power sources could be found. Water supply issues during M-MC's last site visit were still unresolved with the government delaying making any decisions with regards to refurbishing existing infrastructure or assisting in the sourcing of alternative water sources.

Due to the high phosphorous content of the concentrate produced through the current processing plant, MCC is planning to sell its concentrate to third parties for blending prior to selling to steel manufacturers. An alternative could be to supply a different market, servicing the high demand for magnetite concentrate as dense media in the Chinese coal washing industry, rather than providing feed stock to the iron smelting industry, as is currently planned. This is advantageous for a number of reasons, not least being that very low levels of phosphorus are difficult to achieve and that the specifications for dense media are not as strict concerning such impurities. Moreover, this type of product significantly lowers the processing costs, since some stages of processing are not required (floatation).

2.2 ASSETS

The assets and status include;

- Six (6) Iron ore mining leases — containing the North, South and East Deposits
- One (1) operating underground mine — South Deposit (31Mt Mineable Quantities, capacity 3.6Mtpa)
- One (1) crushing and ore processing plant — concentrate slurry (at 1Mtpa, capacity 3.5Mtpa)
- One (1) Ore slurry pipeline (32km)

- One (1) Slurry dewatering facility
- One (1) Port ship loading facility (loads Panamax vessels at 2,000tph)

2.3 LAND TENURE AND MINERAL RIGHTS

Tenure of the assets is subject to two (2) official documents signed on March 08, 2005 between HIPARSA and A Grade Trading Argentina S/A (MSG), registered at the Register Office of the Rio Negro Province.

The two documents are titled;

1. Transference of Assets and Mining Rights, and
2. Agreement on the Constitution of Mining Easements.

Document (1) refers to:

all iron ore mineral rights known so far in the Sierra Grande Mining Complex, that is:

- Mina Liberdad (Process # 129696-M-1948 / # Mina PASMA 20-284)
- Mina San Martin (129965-M-1948 / 20-285)
- Mina Pecheca (138102-M-1949 / 20-286)
- Mina Calfucura (44521-M-1959 / 20-287)
- Mina Namuncura (157259-M-1963 / 20-288)
- Cuenca Ferrifera (152125-M-1975)

Document (2) refers to:

use and exploitation of all existing assets necessary to the reactivation and operation of the Sierra Grande Mining Complex: buildings, industrial installations, machinery, warehouses and all other existing facilities.

The mining easements cover all the surfaces of Industrial Areas I (over 1,250ha) and II (over 40ha), in addition to all accesses and pathways, except the installations of Punta Colorada extending over the ocean, which are managed by Article 19 of the Punta Colorada Port contract, related to use of the port spare capacity by third parties.

With regards to mining rights, not only the existing mines are covered by the easements, but also all deposits to be discovered or to be incorporated for exploitation in the Sierra Grande Mining Complex.

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

2.4 EXPLORATION AND MINING HISTORY

Since privatisation a large proportion of the geological data has not been recovered. There are few remaining records of discovery or exploration history. A comprehensive drilling programme started in the 1960's followed by some drilling in the mid 1970's. Exploration and mining activities are summarised in ***Table 2.1***

Table 2.1 — Minera Sierra Grande Iron Ore Project — Exploration and Mining History

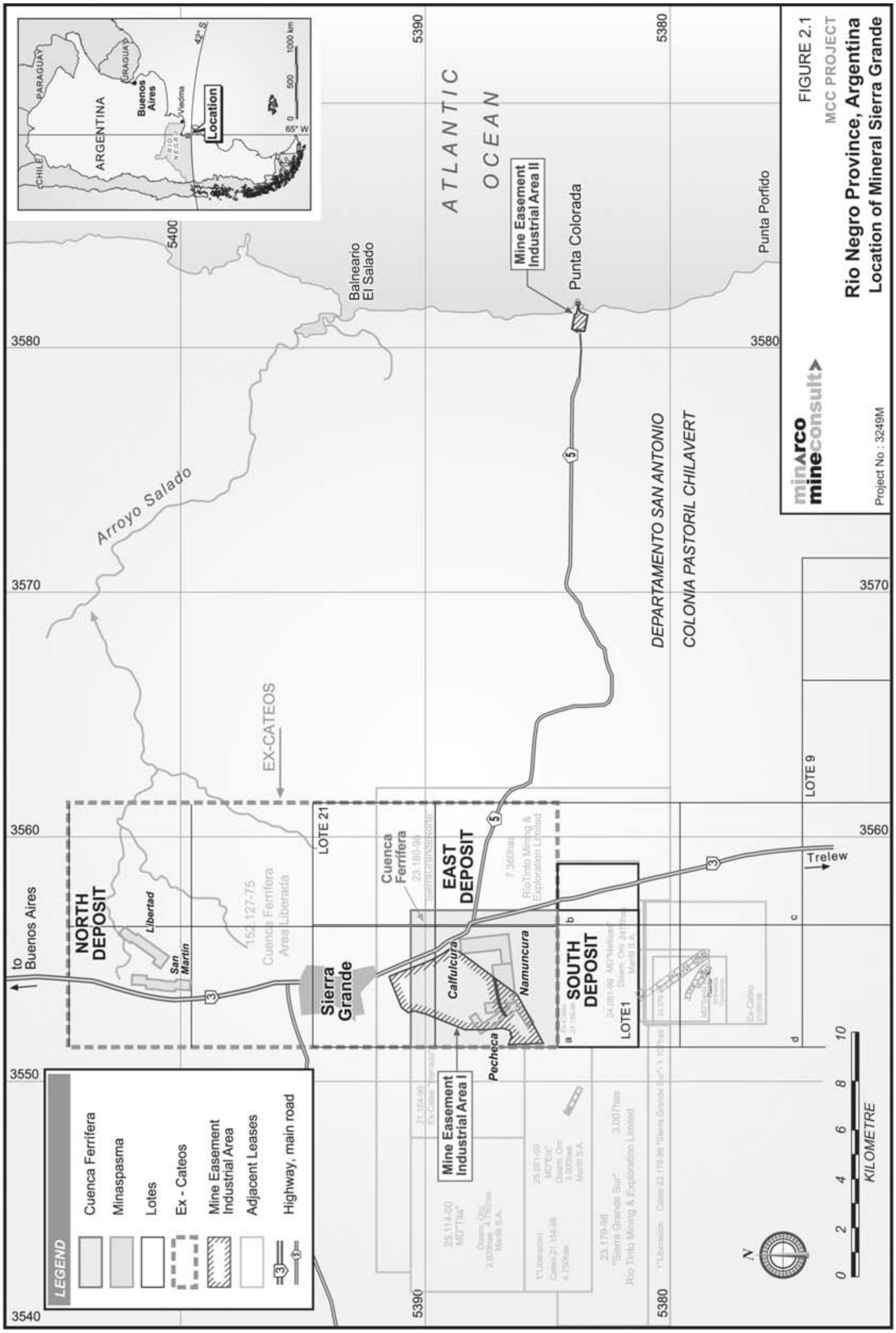
Year	Activity	Comment
1949	Regional Scale Magnetic Survey	Identified the prospectivity of the area
1949-1956	Detailed Magnetic Survey 297m of surface trenching 40 HQ Diamond Drill Holes, 4,715m of drilling	National Institute of Geology and Mineral Argentina
1958-1959	Preliminary Resource Estimate to the 450m depth	Direction General of Military Fabrication
1959-1962	7 deep drill holes, 2,116m of drilling to 480m depth Resource Estimate 48.35Mt South Deposit to 480m depth	MISIPA MISIPA
1968-1969	Further Drilling 1,575m in total to 500m depth (640 Mine Level) Resource Estimate 89.78Mt South Deposit to 500m depth	MISIPA Taken in various ore types, Oxide/Mixed Ores, Chalcopyrite Ore and Bornite Ore
1972	Commencement of mining development	
1976	Further Drilling to 780m depth (940 Mine Level)	
1979	First production	
1980	Resource Estimate 219.64Mt South Deposit to <1,000m depth (1100 Mine level)	
1991	Mining stopped	
2005	Mine refurbishment commenced	
2008	HQ diamond drilling East Deposit and northern extent of South Deposit	East Deposit 40 holes, totalling 12,000m. South Deposit 1,300m.

Source: 1980 Preliminary Technical Report and site information

Mine construction started in 1972 and with first production taking place in 1979. Operations involved underground mining and processing, a pellet production plant, and the maritime terminal (the Port). Production ceased in 1991 due to two factors: (a) the difficulty in reducing the concentrate phosphorus content to product specifications and (b) the failure to achieve capacity of the pellet plant project to 2Mtpa.

The operation was restarted in 2005 by MCC with the focus of the past 3 years being refurbishment and recommencement of mining. This involved the purchasing of new equipment and training of local staff. The 2007 total production level run-of-mine (ROM) ore was 94kt and the concentrate production reached 36.6kt. 2008 achieved a total ROM ore production of 312kt and concentrate production of 129.5kt at 67% TFe. The planned 2009 production is for 1Mt ROM ore and 450kt of concentrate. By end of February 2009 the mine had produced 80kt of ROM ore and 30kt of concentrate. The shortfall was mainly attributed to a shortened month in January due to holidays and reduced throughput capacity due to lack of water for the processing plant.

Figure 2.1 — Minera Sierra Grande — Project Location



2.5 GEOLOGY

The majority of the data related to geology was not recovered after privatisation, therefore there is no exploration data to support any estimate of Mineral Resources. This data loss includes all borehole information (logs, chemistry, structure information, etc), geological maps, horizontal and vertical geological sections and all information used to prepare the geological model (manual). Some of the historical core is available on site and could be used to check the original estimates. No resource or drilling data prior to 2008 exists in digital format.

There are two full time geologists responsible for geology, one Argentinian and one Chinese. They are further supported by local geotechnicians who assist with collecting of samples and geological information from underground.

The 2008 resource development drilling has been sampled using half core and samples collected have been shipped to the Northern Engineering & Technology Corporation, MCC Institute for analysis. The resource estimate will be completed by the same institute in March 2009.

The iron formation of Silurian Age is bounded both in its hanging wall and in its footwall by angular unconformities. The footwall is made up of clastic sediments of the lower Palaeozoic (Cambrian/ Silurian Age) and the hanging wall by carbonate rocks, shales and quartzites of Mesozoic age (Triassic/ Jurassic). The iron formation strikes NW-SE and dips NE (40^o to 60^o at the upper levels and about 30^o at the lower levels) and has a strike length continuity of 3,200m. The extent down dip of the iron formation in the South Deposit varies from 900 to 1,100m. The geological (true) thicknesses of the iron formation ranges from 5 to 15 m (average 10 m).

The main geological structure of the South Deposit is an anticline intruded by a quartz-diorite dyke. The thickest iron formation horizon, about 14 meters thick, is situated at the axis of this structure. Geological continuity between the South and East Deposits is expected. The transition between ore and waste is gradational, with an ore cut-off grade of 40% Fe.

The available exploration drilling records indicates a total of 168 boreholes in four stages, from 1960 to 2008.

M-MC reviewed drill hole plans on site as well as the recent and historical underground plans of the development. In 2008 MCC carried out validation drilling every 25m along strike, using an underground Simba percursion drill rig, from the 410 level up to the 270 level. Results from this work confirmed the original 1980 ore interpretation and volumes. M-MC considers the current ore interpretation to be reasonable.

MCC currently logs all stope rings. This results in a very detailed interpretation of the ore every 1.7m along the development. Based on this information MCC accurately defines the amount of intercalated waste rock as well as design ore loss and dilution. M-MC considers the work being carried out on site to be of a very high standard.

2.6 RESOURCES AND RESERVES

M-MC reviewed the reasonableness of the estimates and reporting categories from the 1980 report. However, M-MC could not validate the estimates due to a lack of supporting data.

2.6.1 Mineral Resources — In Situ Quantities

The most recent and relatively reliable information available is the internal report produced in February 1980, which reports the "reserve number" i.e. mineral resources of 219.6Mt for the South Deposit, down to the 1,100 level, ***Table 2.2.***

The North Deposit has an Inferred Resource of 11.3Mt in addition to an exploration target of 20.0Mt. The East Deposit has approximately 30Mt to 40Mt in the Exploration Target category only. The East Deposit in 2008 underwent further exploration with 40 drill holes totalling 12,000m completed. The revised resource for this deposit will be completed in March 2009.

The South Deposit is by far the largest, most continuous and well known of the three deposits. Of the 219.6Mt, 108.6Mt (49%) are Measured Resources (to level 640m), 38.8Mt (18%) are Indicated Resources (to level 20m) and 72.3Mt (33%) are Inferred Resources (to level 1,100 m). A summary by level and category is shown in ***Table 2.2***.

Iron Ore mineralisation is predominantly martitic (platy hematite Fe_2O_3) in near surface to 25 m depth. Surface ore average grade is 54.8% Fe, 4.8% Al_2O_3, 5.9% SiO2 and 1.43% P. Magnetite ore (Fe_3O_4) predominates below the surface ore with an average grade of 57.3% Fe. Iron ore density averages 4.5t/bcm.

The mineral resource estimation parameters included:

- Cut-off grade (cog) — >30 %Fe
- Minimum thickness — not available
- Intercalated Waste — Factor of 0.9 applied to the resource tonnes to account for barren bands within the mineralisation
- Resource Categories — Based on US Geological Services guidelines of the 1970's

In M-MC's opinion, these estimates are reasonable and have been reported in accordance with the recommendations from the USGS guidelines of the time. M-MC was not provided with the detailed data to confirm the JORC compliance of the estimate and hence refers to the resources as In Situ Quantities and to the classifications applied as JORC Equivalent.

The resources reviewed by M-MC did not clearly report the portions of the resource which are Hematite and which are Magnetite. Both of these should be reported separately as they have different metallurgical characteristics and grades. ***Table 2.2*** shows the reported resource as at February 1980.

Table 2.2 — Minera Sierra Grande — South Deposit — In Situ Quantities February 1980

	In Situ Quantities (USGS Classification)											
	Measured				Indicated				Inferred			
Level (m)	Mt	TFe %	Fe3O4%	P %	Mt	TFe%	Fe3O4%	P %	Mt	TFe %	Fe3O4%	P %
410	59.9				1.2				0.3			
620	42.6				3.4				5.4			
830	6.2				21.2				33.2			
>830					13				33.3			
Total	**108.7**	**57.3**	**68.3**	**1.29**	**38.8**	**57.8**	**67.7**	**1.33**	**72.2**	**57.8**	**69.2**	**1.3**

	All Classifications			
	Mt	TFe %	Fe3O4%	P %
Grand Total	219.7	57.5	68.5	1.3

Source: Geology Report 1980

Notes: Estimates at Feb 1980
Mineral Resource estimates inclusive of Ore Reserves
Cut-off grade >30% Fe
Intercalation factor of 0.9 applied to resource tonnes to account for waste bands within the ore.
In Situ Quantities classified using USGS 1970 classification. These are broadly similar to JORC.

The 1980 estimate of In Situ Quantities has been adjusted for depletion and reported as at June 2008 based on data provided by MCC (***Table 2.3***). Mining to date has exploited all resources above the 223 level and parts of the 246 level. Lode development is currently underway on the 293 and 316 levels with the 223 and 246 levels having being completed in 2008. In Situ Quantities remaining are mostly magnetite.

Table 2.3 — Minera Sierra Grande — South Deposit — In Situ Quantities June 2008

	In Situ Quantities				
Category	USGS Classification Chinese Code	Inferred 333 (Mt)	Indicated 332 (Mt)	Measured 331 (Mt)	Total (Mt)
Original Resource (Mt)	Iron ore	72.2	38.8	108.7	219.7
Resources Depleted (Mt)	Iron ore			20.3	
Remaining(Mt)	**Iron ore**	**72.2**	**38.8**	**88.4**	**199.4**

Source: Geology Report 1980

Note: Adjusted for depletion to June 2008

2.6.2 Reserves — Mineable Quantities

Due to the lack of reliable exploration data, there are no detailed geological models. Without this and a mine plan a Reserve Statement is unavailable. However, an operational estimate of the South Deposit above the 410 level is approximately 31.3Mt based on recent MCC resource drilling carried out on 25m profiles along the 410, 340 and 270 mine levels. This information provided by MCC is shown in ***Table 2.4***. These estimates do not meet JORC standards and therefore have been referred to as "Mineable Quantities".

Table 2.4 — Minera Sierra Grande — South Deposit — Mineable Quantities

JORC Equivalent	Mineable Quantities (Mt)
Proven	11.3
Probable	20
Total	**31.3**

Source: Client Information June 2008

Note: Estimated 2007
Mining method; sub-level stoping

This estimate provides reasonable Mineable Quantities for mining at the current rate for a period exceeding 8 years.

2.7 MINING

2.7.1 General Description

Current underground mining operations are based on the exploitation of the South Deposit only. The South Deposit was largely developed to its present state by the former operators, HIPARSA. According to the operators, over 60km of development drifts and ramps have been driven in developing the mine for production. A main haulage and service decline has been driven from surface to the bottom of the mine and a rock hoisting and service shaft has also been constructed, bottoming out at 522m below the surface. Current depleted resources as well as existing and planned development are shown in ***Figure 2.2***.

The operation is mechanised through the use of traditional, trackless underground mining equipment, quite modern considering when the operations commenced (early 70's). Sub-level stoping and sub-level caving have been the main mining methods applied to extraction of the orebody. The distance between mining levels is 70m and 23m between sub-levels. The blasted ore is transported by means of LHD to ore-passes converging to level accesses from where it is trucked to the primary crusher located at the 410m mining level. Once crushed, the ore is then transported to the central shaft lift system which consists of two 17t skips to transport the ore to the surface for secondary crushing. The former methods of operation are still viable for the current operation, however the mucking (bogging) equipment (LHD's) employed in the stopes should be equipped with remote-control units to avoid operator exposure under some of the very high open stope backs, and also to recover 100% of the broken ore from a blasted stope; only 70 to 85% of blasted stope ore is now recovered. Mining and design dilution for 2008 is reported to be 6%. Development will be a priority after 2008 and about 8,500m of development per year will be required to maintain the mine in a developed state.

Due to the lack of reliable estimates for resources or reserves, mine planning is based on volume estimate of ore and waste based on drilled out stope rings at 1.7m spacing. An operational estimate of 31.3Mt of Mineable Quantities based on 25m drill profiles undertaken by MCC using a Simba drill rig provides some reliability for mining to continue for a period of 4 to 5 years. The current mine production rate of 312ktpa is easily attainable, with the budgeted ramp-up to the maximum 2.80Mtpa by 2011 all attainable with the present shaft and hoisting equipment.

The historical production from the operation is summarised in ***Table 2.5***. Some of the mining before 1980 represented pre-production and development mining and the mined material was not treated in the plant.

Figure 2.2 — Minera Sierra Grande — Stoping and Level Development Status, June 2008

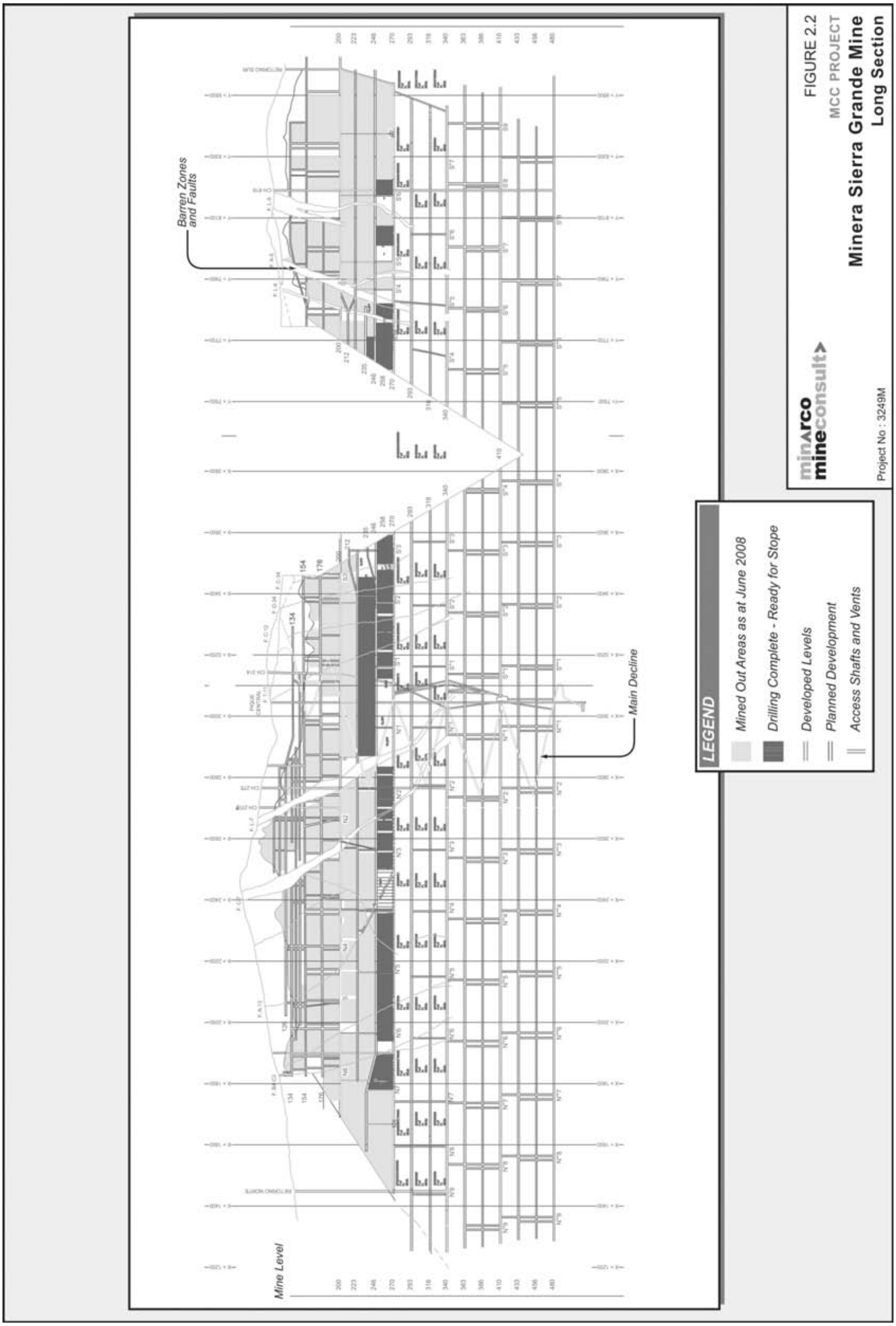


Table 2.5 — Minera Sierra Grande — Historical Production

Year units	Mine Production (kt)	Plant Feed (kt)	Pre-concentrate (kt)	Concentrate (kt)	Pellet (kt)
1972	212.1				
1973	312.1				
1974	254.6				
1975	106.7				
1976	404.7				
1977	806.2				
1978	777.4				
1979	522.8				
1980	1,056.5	736.5	506.7	337.5	311.9
1981	811.8	660.0	478.7	296.9	326.3
1982	660.9	1,181.1	867.7	523.6	Not reported
1983	657.3	1,138.0	790.9	496.7	520.8
1984	1,222.9	921.0	630.1	394.7	420.1
1985	940.2	1,052.0	740.7	463.8	509.4
1986	1,219.1	1,489.9	1,069.1	650.8	646.4
1987	756.3	1,069.1	772.6	456.5	464.7
1988	1,119.0	1,349.6	963.9	585.1	605.1
1989	1,223.3	1,305.0	954.1	567.5	591.9
1990	1,271.8	1,292.5	927.1	568.5	612.8
1991	230.2				Not reported
Total	**14,565.9**	**12,194.7**	**8,701.6**	**5,341.7**	**5,009.4**

Source: November 2007 Feasibility Report

An analysis of the historical and recent production figures indicates that there is 240kt of magnetite concentrate available for sale, of which 71kt has been moved to the port for shipment in February 2009. Using a valuation of USD49.40/t of concentrate, which is conservative in today's market, the stockpiled concentrate has a value of some USD11.9 million.

2.7.2 Forecast Production

The long-term Minera Sierra Grande production schedule is given in the ***Table 2.6***.

Table 2.6 — Minera Sierra Grande — Production Forecast

Production Rate	units	2008 (actual)	2009	2010	2011	2012	2013	2014
ROM Ore / day	**t/day**	1,020	3,270	4,900	7,520	9,150	9,150	9,150
ROM Ore /year	**ktpa**	312	1,000	1,500	2,300	2,800	2,800	2,800
Concentrate/year	**ktpa**	129.5	450	660	1,009	1,228	1,228	1,228
Concentrate	**%Tfe**	67	68.5	68.5	68.5	68.5	68.5	68.5

Source: Site Information from 2009 site visit

Note: Based on 306 working days per year

The 2009 production target of 1.0Mtpa will be reduced due to the shortage of water and power. MCC has decided to build a gas fired power station however this will not be operational for at least two years. The outstanding issue is the availability of water and both of these issues are described further under ***Section 2.9.***

Production ramp-up as presented by MCC during the 2009 site visit appears reasonable and achievable. The gradual ramp-up should be achievable based on the equipment already on site as long as operator training and processing throughput to match production output can be achieved.

The shaft is currently being used on a single 8 hour shift per day and has a capacity of 800t/hr. Hoisting operations will have to be increased to two 8 hour shifts per day to reach the target production rate. In the opinion of M-MC the hoisting capacity is sufficient to handle the ramp-up of production to 2.8Mtpa with no requirement for any further upgrades.

Based on information collected during the site visit and from the review of operational plans, the mine appears to have sufficient horizontal development and current drilled stocks in excess of 1.0Mt of ore. The size, quantity and quality of the mining equipment currently on site also appear to match the proposed mining production ratios.

2.8 MINERAL PROCESSING

2.8.1 Concentrate Production (Industrial Area I)

The operation, at a low pre-production level, was restarted in 2007. In 2009 total production level run-of-mine (ROM) ore is expected to reach 1 Mtpa and the concentrate production level around 450ktpa. Full production is reached in 2012 with a budgeted ROM ore production of 2.8 Mtpa and concentrate production of 1.23Mtpa. The following description highlights the processing stages and the associated equipment. In 2008, the actual concentrate production was 129.5kt from 311.8kt of feed.

Primary Crushing

Primary crushing is performed underground on the 410m level by 1,800mm x 1,470mm jaw crusher (220kW) with a capacity of 800tpd, which is more than capable of processing the planned mining production. At an availability of 60%, the jaw crusher would be able to handle more than 4Mtpa.

Secondary Crushing

Crushed ore is hauled to the surface and stored in a 3,000t storage bin. The ore is sized on two sets of 1,800mm x 4,800mm single deck vibrating screens at 100mm and the +100mm ore is crushed in two sets of 480mm x 120mm hydroset cone crushers (110kW) in closed circuit with the screens. The −100mm product is then conveyed and stacked onto a 20kt live capacity stockpile with a total capacity of 90kt.

Pre-concentration

Ore from the stockpile is recovered by vibro-feeders and split into three parallel streams for dry magnetic concentration. This stage rejects approximately 14% of the feed (500ktpa from a feed of 3,500ktpa) although in 2008 the level of rejection increased to 28%.

Concentration

The revised concentration plant flowsheet represents a typical flowsheet ***Figure 2.3*** used for a magnetite ore where the magnetic component is separated from the non-magnetic impurities. The process combines staged grinding with low intensity magnetic separation to produce final iron rich product. The processing plant consists of three parallel lines with line number 1 and 2 available for production. Currently, one line is in operation and operating on two 8 hour shifts. The third line is under repair and undergoing modifications to achieve a finer grind.

Liberation of the minerals is conducted in two stages using a rod mill (3.9m diameter x 5.2m) and an autogenous grinding mill (5.9m diameter x 10m) to produce a relatively fine product 85% passing 44 microns.

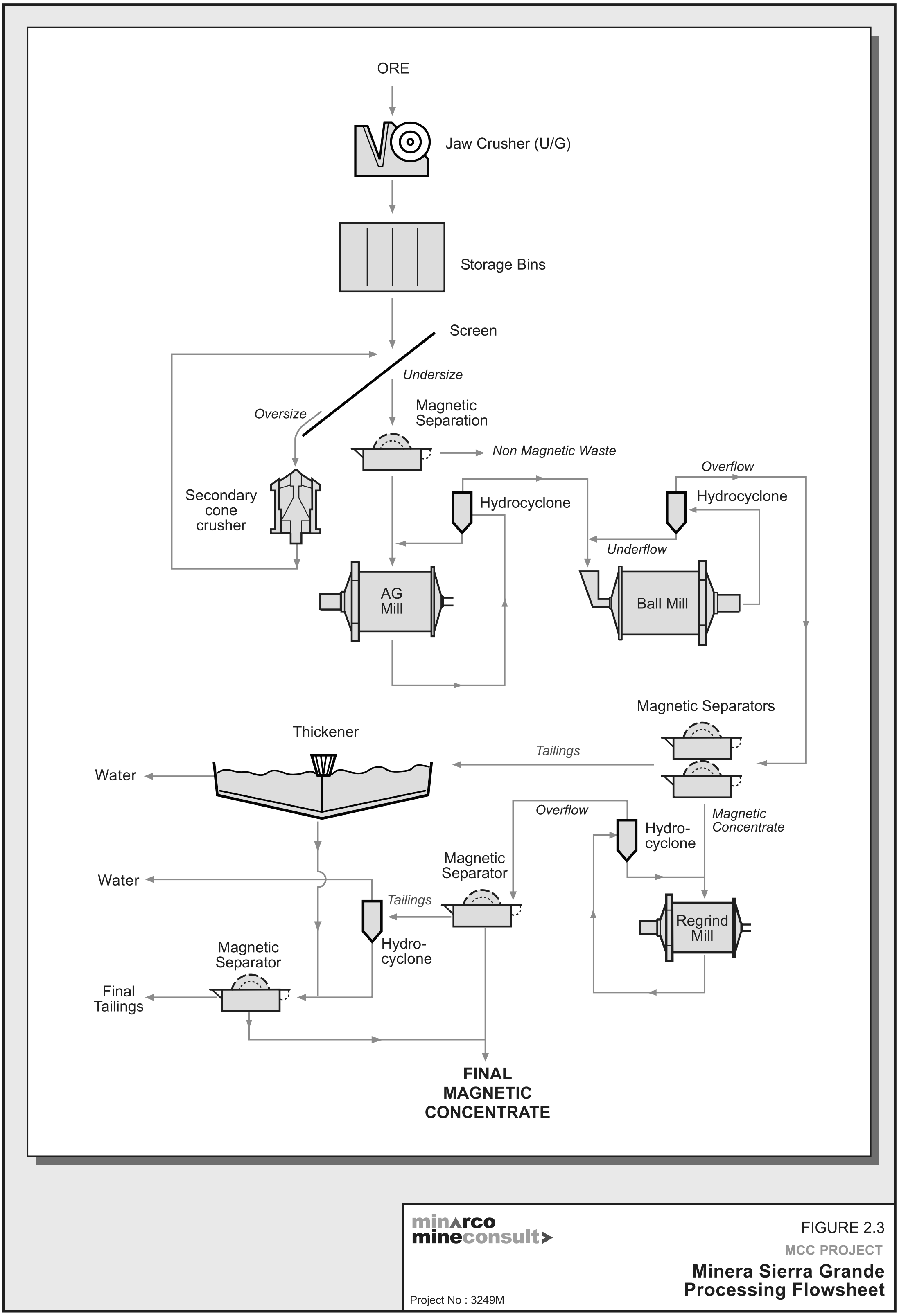
The product from the autogenous mill is classified by hydrocyclones and the overflow is subjected to two stages of magnetic separation. The subsequent magnetic concentrate is reground to 97% passing 44 microns and the mill discharge upgraded in another magnetic separation to produce a final concentrate product, which is directed to the concentrate thickener. The thickened underflow (65% solids) is pumped to the concentrate transportation pipeline. Modifications to the processing operation, which includes finer grinding of the concentrate utilising the milling circuit in line number 3 as well as fine tuning the magnetic separation operation, are being conducted to ensure that the magnetite concentrate grade averages 67+% Fe and $<$0.3% phosphorus.

Tailings from the two coarser magnetic separation stages are directed to a thickener while tailings from the finer magnetic separation stages are dewatered in hydrocyclones. The underflows from both the hydrocyclone and thickener are further scavenged for any remaining magnetite. The magnetic concentrate is directed back to the rod mill discharge while the tailings are stored in a tailings dam called Laguna Blanca. The tailings dam has considerable capacity for many years of operations.

Concentrate Pipeline

The slurried concentrate is stored in two agitated tanks (3,000t capacity) before being pumped down a 32.4 km, 200 mm diameter pipeline to the dewatering facility (Industrial Area II) located near the port. At this pipe diameter, the two sets of piston pumps can transport up to 2Mt of concentrate per annum (290tph). These pumps are in the process of being upgraded with pumps currently being refurbished in Brazil. Currently the magnetite product is being stored in a dam, which will be recovered and transported to the port when the pumps have been refurbished. Currently 170,000t is stored in the concentrate dam and another dam is under construction.

Figure 2.3 — Minera Sierra Grande — Processing Flowsheet



2.8.2 Dewatering (Industrial Area II)

Only the dewatering and the storage and loadout facilities are used at the pelletising facility located at the port (Punta Colorada). The facility consists of a 25m diameter thickener and 4 new ceramic disc filters, with provision for two more sets should the fineness of the magnetite concentrate limit the production rate. It is planned to achieve a concentrate cake with 8% moisture which will be stored until reclaimed for loading onto ships.

2.9 INFRASTRUCTURE AND SERVICES

There are high quality workshop facilities and equipment at both the main plant and the port. A potential operationally disruptive feature is the general lack of spares, since there is a significant delivery time of 6 months for all items. Pump liners and flotation cell spares are available from the previous operation, while some critical items have been sourced and held in stock (e.g. conveyor belts). Other items have been ordered (e.g. screens), however more spares need to be ordered.

The assay facility and metallurgical laboratory were examined and found to be in reasonable condition. Mill discharge, concentrate and tailings samples are taken four times a shift and checked for magnetite content using a Satmagan. Elements such as silicon, calcium, aluminium and magnesium are determined by wet chemical means. Phosphorus is measured using spectrometry.

Port

The port is located at Punta Colorada and can accommodate Panamax size and larger ships. It consists of a 500m conveyor belt linking Industrial Area II to the pier, which comprises a 1,000m conveyor belt out to the sea and two pairs of mooring dolphins; one pair on the north side and the other on the east side, allowing ships to berth on either side.

The maximum loading rate of the system is 2,000tph. In 2006, some 60,000t of magnetite concentrate was loaded onto a ship demonstrating that the system had been suitably restored. The concentrate bucket reclaim units have been replaced.

The water recovered from filtration is reclaimed in a collection and storage dam, and pumped back to the plant site. This project has been completed and consists of a pumping station, pipeline and associated tankage and control systems. The recovered water is stored in two recently constructed 2,000 m^3 concrete tanks located above the plant site.

Power

Electric power to the mine is provided by EDERSA, a local Argentine power company from the Patagonian power grid. Currently the operation has 14.0MW of power available, which is sufficient for two production lines in the concentrator as well as the mining equipment. Both the provincial government and the Argentine Production Management Department have been onsite to discuss the electrical supply problem with MCC. There is a growing demand for electric power in the Sierra Grande Region, and while there are plans to construct power stations, no timetables have been sighted. As a result, MCC has decided to construct a gas fired power station, which is expected to be operational within a few years.

Water

There is currently insufficient water to operate the mine, beneficiation plant and ancillary facilities at full capacity. Water to the area is supplied through two 110km long aqueducts originating in the Andes Mountains. The aqueducts are in poor condition with many leaks and currently deliver $120m^3$/sec to the Sierra Grande Region. Only

24m^3/sec is delivered to the mine operation. One solution proposed by the local authority is the construction of a desalination plant, however this will only produce 35 l/sec of potable water. Both the provincial government and the Argentine Production Management Department have been onsite to discuss the water supply problem with MCC. The government has made a commitment to meet the 2009 water requirements and if necessary, supply water condensed from gas exhaust used to generate power.

M-MC believes that, as an option to achieve the full plant capacity, MCC should consider the use of sea water in the process plant and transport the concentrate with fresh water after filtration. Another solution may lay in assisting with the restoration of the aqueducts to their full capacity (240m^3/sec).

Labour

In March 2009, there were 385 people employed at the mine and the current plan is to increase the workforce to a maximum of 540 staff in 2011 when the mine reaches full production. The mine engineering group and mine operations groups consist mainly of Chinese expatriate engineers. With the exception of a geotechnical engineer the operation appears to have sufficient personnel in all key areas; mining, processing, maintenance and administration.

2.10 CAPITAL AND OPERATING COSTS

The forecast mining and processing costs until 2014 are presented in ***Tables 2.7*** and ***2.8***. Mining costs are fixed at USD8.65/ROM t while processing costs are USD13.03/ROM t. Administration and other costs are estimated at USD1.59/ROM t. Finance costs were estimated at USD7.29/ROM t in the Feasibility Study; however the revised figure has not been made available. Note that 2.28 ROM t are required to produce one tonne of iron concentrate.

Table 2.7 — Minera Sierra Grande — Forecast Mining Costs

Cost Centre	Unit	2009	2010	2011	2012	2013	2014
Auxiliary Material	USD/ROM t	1.11	1.11	1.11	1.11	1.11	1.11
Water & Power	USD/ROM t	1.10	1.10	1.10	1.10	1.10	1.10
Labour	USD/ROM t	1.28	1.28	1.28	1.28	1.28	1.28
Repair & Maintenance	USD/ROM t	0.32	0.32	0.32	0.32	0.32	0.32
Mine Development	USD/ROM t	2.40	2.40	2.40	2.40	2.40	2.40
Depreciation	USD/ROM t	1.55	1.55	1.55	1.55	1.55	1.55
Others	USD/ROM t	0.89	0.89	0.89	0.89	0.89	0.89
Total	**USD/ROM t**	**8.65**	**8.65**	**8.65**	**8.65**	**8.65**	**8.65**

Source: MCC provided Capex and Opex figures February 09

Table 2.8 — Minera Sierra Grande — Forecast Processing Costs

Cost Centre	Unit	2009	2010	2011	2012	2013	2014
Auxiliary Material	USD/ROM t	8.20	8.20	8.20	8.20	8.20	8.20
Water & Power	USD/ROM t	2.09	2.09	2.09	2.09	2.09	2.09
Labour	USD/ROM t	0.13	0.13	0.13	0.13	0.13	0.13
Maintenance	USD/ROM t	0.49	0.49	0.49	0.49	0.49	0.49
Depreciation	USD/ROM t	1.71	1.71	1.71	1.71	1.71	1.71
Others	USD/ROM t	0.41	0.40	0.40	0.40	0.40	0.40
Total	**USD/ROM t**	**13.03**	**13.03**	**13.03**	**13.03**	**13.03**	**13.03**

Source: MCC provided Capex and Opex figures February 09

The source and indeed nature of the Auxiliary Material cost in ***Table 2.8*** is unclear and is remarkably similar to the mining cost (USD8.65). M-MC believes that the underlying processing cost is USD10.20/ROM t, which is reasonable.

M-MC believes that these forecast costs do not take into account local issues such as inflation, increased wages and other local charges and costs. While these potential increases in costs may be offset to some extent by a change in the exchange rate, the future cost of imported goods or parity priced materials (e.g. fuel) may contribute to increased operating costs. M-MC therefore believes that future operating costs could be 20% higher than MCC has forecast by 2010. Nonetheless, it should be noted that operating costs at the Sierra Grande operation would still be considered relatively low.

An important operating cost that has not been included is the concentrate ocean freight costs. The Feasibility Study reported that this cost ranged between USD37.5-46.5/t in 2004 and USD28-32/t in 2005 for Brazilian pellets. While MCC staff indicated that freight costs may be as high USD80-90/t for a 150,000t ship, in the current market, this figure is USD10-12/t from Brazil.

MCC plan to sell the concentrate for blending with other iron concentrates to supply the steel manufacturing industry. Three potential buyers have been identified from Paraguay, Brazil and China. The product has a high iron content (67% Fe) and a higher than normal phosphorus content. This product will compete in the iron ore fines market, where spot prices are currently USD70/t. The Feasibility Study used a value of USD49.40/t based on the then prevailing value for Brazilian pellets, depending upon the actual price realised for the product as well as the cost of freight. It may be anticipated that revenues may well exceed that predicted in the Feasibility Study.

Capital Expenditure

MCC plan to spend some USD93 million to fully re-activate this project of which MCC has supplied nearly USD21 million while USD72 million will be raised through a loan in Argentina. This cost is somewhat less than that estimated by a Japanese study, which estimated that around USD120 million would be required.

The operation to May 2009 has spent USD59 million. The operators advised that the capital estimate from the Feasibility Study was the most relevant. In M-MC's opinion, the capital cost estimate of USD93 million as stated in the Feasibility Study appears reasonable. This is summarised in ***Table 2.9***.

Table 2.9 — Minera Sierra Grande — Current Expenditure

Cost Centre	Expenditure (USD)
Construction (building)	4,705,010
Construction (erection)	802,390
Equipment	14,055,720
Other	818,300
Financial Reserve	611,440
Total	**20,992,860**

Source: November 2007 Feasibility Report

The Planned expenditure is given in ***Table 2.10***.

Table 2.10 — Minera Sierra Grande — Planned Expenditure

Cost Centre	Expenditure (USD)
Mining (shafts & tunnels)	4,890,820
Construction (building)	34,104,120
Construction (erection)	1,437,720
Equipment	20,182,350
Other	3,677,530
Financial Reserve	7,715,110
Total	**72,007,650**

Source: November 2007 Feasibility Report

This planned capital expenditure does not make any provision for other construction activities that may become necessary, in order for the operation to realise its potential. Notably, these activities concern power and water and the construction of a power station may be required. This power station would presumably operate on natural gas and supply power to general region. While it is difficult to estimate how large the power station would be and thus the associated cost, at least USD20-30 million be need to be allocated for such a venture. On the water front, a number of solutions are possible, the least expensive being the repair and reconstruction of the aquaducts. Perhaps the expenditure of USD5 to 10 million may have a significant impact upon the amount of water that would become available. A desalination plant is expensive in terms of a capital and operating costs is not a recommended solution. Another solution involves the use of saline water in the processing plant. This would require that the final magnetite concentrate is washed to remove the salt content before re-slurrying and pumping to the port. This option would cost less than USD1 to 2 million.

2.11 SAFETY AND ENVIRONMENT

Management systems for underground risks such as rockfalls are in place with MCC undertaking regular scaling of active mining areas. These could be better improved with the addition of a geotechnical engineer to the operation.

MCC has removed PCB based transformers both underground and in the plant and replaced them with new transformers.

Most of the water used in the process is recycled using thickeners and filters, and only a small amount of water is discharged into the environment with the tailings stream. With the elimination of the need for the ore to be treated by flotation, it would appear to pose no chemical threat. This arises since many reagents or chemicals are now not required.

There is little discharge of dust during processing due to the use of dust collectors in appropriate places. There may be some dust issues with the storage and reclaiming of magnetite concentrate at the port.

While the environmental monitoring program and associated environmental monitoring facilities were not inspected during the visit in April 2008, MCC are aware of the potential environmental issues and are addressing such matters. The Argentine Environmental Protection Agency has visited the operation on several occasions to inspect MCC's environmental monitoring practices and the interaction is on-going.

3 RAMU NICKEL LATERITE PROJECT

M-MC made a site inspection of this property in April 2008 to review geology, mining and processing. In March 2009 M-MC carried out a final site visit to review the status of the processing and mining construction works. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "2007, Basic Design of Ramu Nickel and Cobalt Project" prepared by ENFI.
- "2008, Basic Design Updated Processing Report" prepared by ENFI.

MCC's effective equity stake in the Ramu Project is 51.85%. (61% of an affiliated company holding an 85% stake in the project)

3.1 BACKGROUND

The Ramu Nickel Laterite Mine is located near the town of Usino, in the Madang Province, Papua New Guinea (***Figure 3.1***). The mine area is located approximately 80km west of the coastal town of Madang in the Highlands of the Bismarck Mountains at an elevation of 700m above sea level. Access is approximately 20km west from the Madang-Lae Highway. The proposed slurry pipeline route (135km) follows the highway to the coast then eastwards along the coast to a proposed refinery site at Basamuk, where clearing and site preparation has commenced in preparation of civil works.

The topography at the mining area is rugged and is covered with dense tropical forest.

3.2 ASSETS

The assets and status include;

- A mining project in development (commissioning to be completed in late 2009)
- An Exploration Licence: EL 193
- Mineral Resources of 143.2Mt at 1.01% Ni and 0.10% Co.
- A mining Feasibility Study 2007
- Ore Reserves of 75.7Mt at 0.91% Ni and 0.10% Co.
- 135km Slurry pipeline with pumping stations
- Ore washing and chromite plant
- HPAL plant
- Refinery
- Two power stations
- Acid plant
- Limestone preparation plant (including kiln)
- Port with gantry cranes
- Accommodation facilities

3.3 LAND TENURE AND MINERAL RIGHTS

The project is located in exploration Licence EL.193. The Kurumbukari proposed Special Mining Lease (SML) is approximately 249 km^2 and is within the EL shown in ***Figure 3.1.***

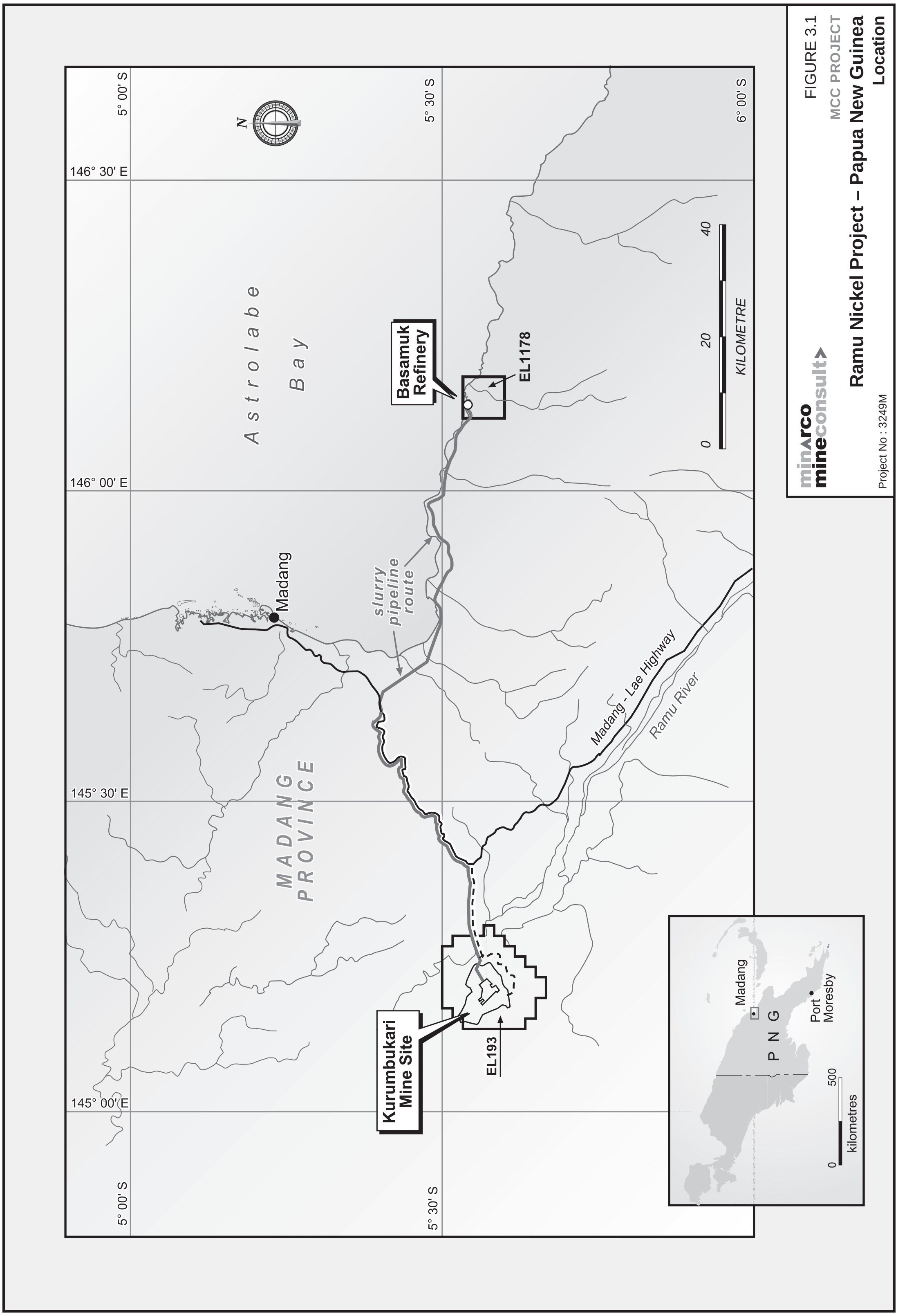
There are three separate mining blocks within the Kurumbuki SML namely, Ramu West, Kurumbukari and Great Ramu,

3.4 EXPLORATION AND MINING HISTORY

The Ramu Nickel Laterite deposit was discovered by the Australian Bureau of Mineral Resources (BMR) in 1962. This was followed by exploration by many other exploration companies from 1962 to 1999. Initial exploration efforts from 1962 to 1970 were preliminary in nature and the project was considered un-economic at that time. After 1970, comprehensive studies were made by various exploration companies to evaluate the Ramu deposit. These exploration programmes were conducted in four stages.

- Stage I, 1970 to 1982: The exploration companies involved were Carpentaria Exploration Company (CEC), Eastern Pacific Mines Ltd (EPML) and Nord Resources Corporation (Nord). Exploration included several auger, Banka and diamond drilling methods. Other studies included heavy mineral analysis, ore dressing, and smelting. A preliminary Mineral Resource was estimated at 69Mt of Ni (1.29%) and Co (0.105%) ore.
- Stage II, 1989 to 1990: CEC was joined by Highland Gold Properties Pty Ltd (HGP) and Nord during this stage of exploration.
- Stage III: 1992 to 1994: HGP bought 60% share of this project and continued its exploration work on a large scale. The area covered was 56.5 km^2. Exploration included 10,200 meters of drilling, environmental monitoring drilling, earthquake analysis, hydrological studies, re-sampling of old drill holes, and heavy mineral analysis. Mineral Resources of 24.2Mt of Ni (0.90%) and Co (0.08%) were estimated based on this exploration.
- M-MC assume this exploration was for a different or smaller area than Stage I.
- Stage IV: 1997 to 1999: HGP and Nord. The objective was to prepare a Feasibility Study report. Detailed exploration extended over a 29.9km^2 area and formed the basis for estimating Ore Reserves of approximately 75Mt.

Figure 3.1 — Ramu Nickel Laterite — Mine Location Plan



3.5 GEOLOGY

3.5.1 Regional Geology

The Ramu Nickel Laterite deposit is located in the central part of the Marum Formation of Miocene age. The ultrabasic Marum Formation is divided into the lower and upper zones. The lower zone consists of norite (hypersthene gabbro), gabbro and peridotite. The upper zone is dominated by dunite, serpentinite, harzburgite, and pyroxenite. The Marum Formation was later subjected to intense chemical weathering processes (residual weathering) in this tropical humid environment. As a result, a lateritic surface was formed over the bed rocks. This laterite cover ranges in thickness from a few meters to approximately 60 meters. Nickel (Ni) and cobalt (Co) mineralisation is associated with this lateritic zone.

The major structural lineation is north-west to south-east. Structurally this area is affected by two major fault zones. These fault zones form a "horst" structure surrounded by rift valleys (grabens) on either side. The vertical displacement is estimated to be approximately 400 meters. The Bundi Fault Zone runs from south to the west. The Ramu-Markham Fault Zone runs almost parallel to Bundi Fault in the north-east part of the area. Several parallel minor faults and associated fractures occur in this area surrounding these fault zones. ***Figure 3.2*** shows the regional geology of the area.

3.5.2 Local Geology

The Ni laterite is formed as a weathering profile over the Marum ultrabasic bedrock.

The laterite zone consists of two distinct horizons. The top red limonite horizon occurs with humus soil cover ranging in thickness from several centimetres to around 10 meters. It consists of hematite and iron clays. The Ni content is very low in this horizon. The red limonite horizon grades downward into yellow limonite horizon. The principal mineral of this horizon is goethite, and is often associated with manganese minerals. The colour varies from light yellow-brown, light red-brown to almost orange-brown. The thickness varies from few meters to approximately 30 meters. This horizon is enriched in Ni content.

The yellow limonite grades downward into the saprolite zone (weathered base rock). This is the main zone (the bulk of the orebody) which contains Ni and Co mineralisation. The upper contact with yellow limonite is gradational. The saprolite zone characterizes the intense weathering environment of the basement rocks (dunite). The seasonally fluctuating water table is responsible in changing the Eh-pH conditions of the sub-surface environment and structurally transforming the original dunite into saprolite. This zone ranges in thickness from few meters to approximately 17 meters. It contains goethite, hematite, garnierite, serpentinite, quartz, magnesium (asbolite — a hydrated oxide of manganese and Co), Mg-silicate minerals, etc. The saprolite zone is characterized by Ni, Co, silica, and magnesium enrichment. The magnesium content is higher in this zone than the upper yellow limonite zone.

The saprolite zone grades into rocky saprolite (boulders of bed rock dunite covered with saprolite). The contact of the overlying saprolite with rocky saprolite is transitional and undulating in nature. The Ni mineralisation still occurs in this partially weathered saprolitic dunite boulders or gravels. But it is very low grade or marginal in nature. The rocky saprolite grades into the bedrock dunite. A geophysical method, Ground Penetrating Radar (GPR) survey was used to estimate the depth profile of rocky saprolite.

Structurally, the mining area is affected by Ramu-Markham Fault in the northeast, forming a series of rift valleys. A few southeast and northwest trending faults are also interpreted from satellite images. Several parallel small scale (sympathetic) minor faults and associated fractures were also mapped using aerial photos, remote sensing, and magnetic survey.

The fault surfaces are characterized by intense brecciation, silicification and some serpentinisation (low grade metamorphism). The Ni-Co mineralisation is associated with several fracture zones (shear zones) and joints. The mineralisation occurs as cavity and vein fillings, coatings and breccia.

Figure 3.2 — Ramu Nickel Laterite — Regional Geology

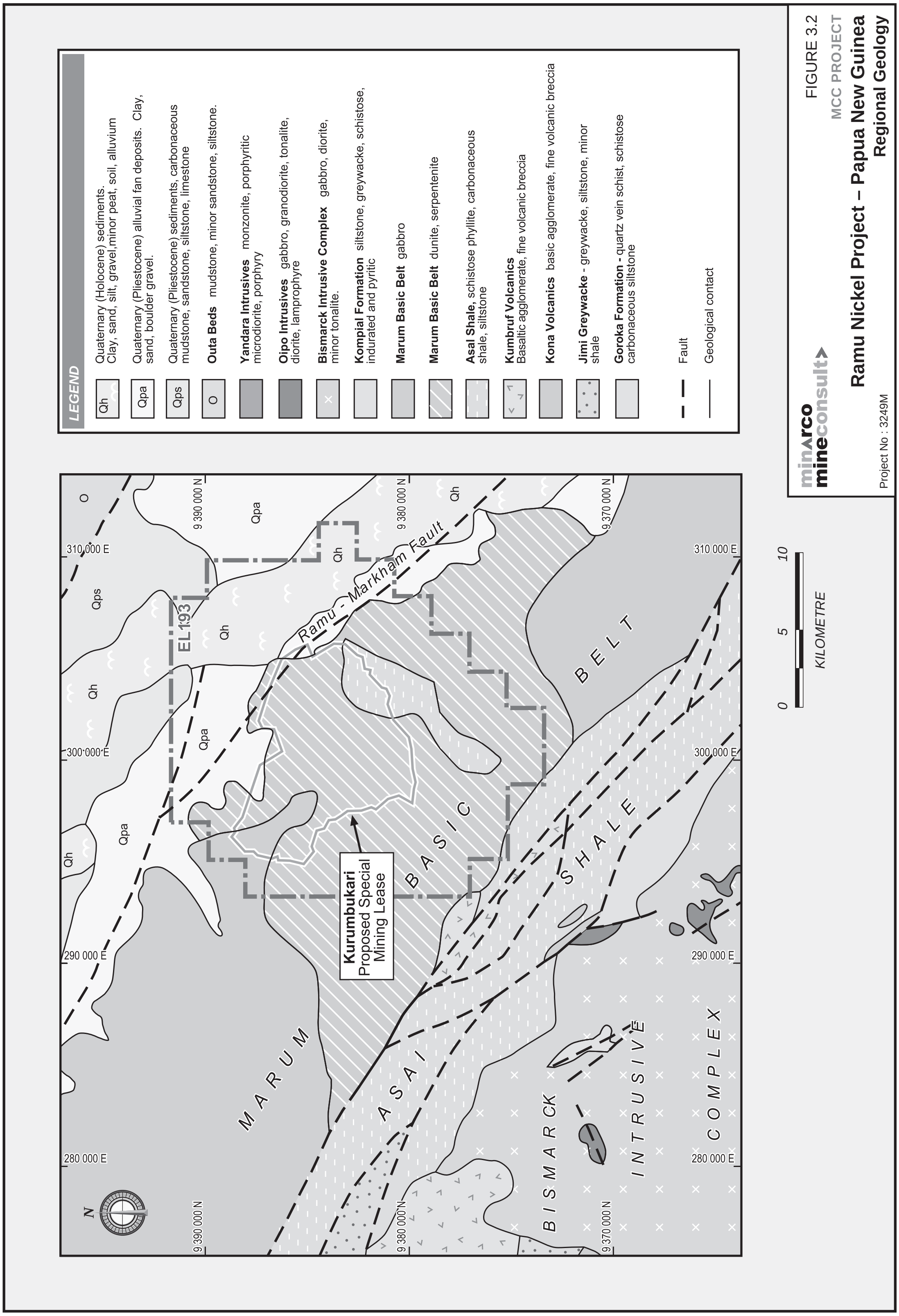


Figure 3.3 — Ramu Nickel Laterite — Geology with Laterite Extent and Resource Areas

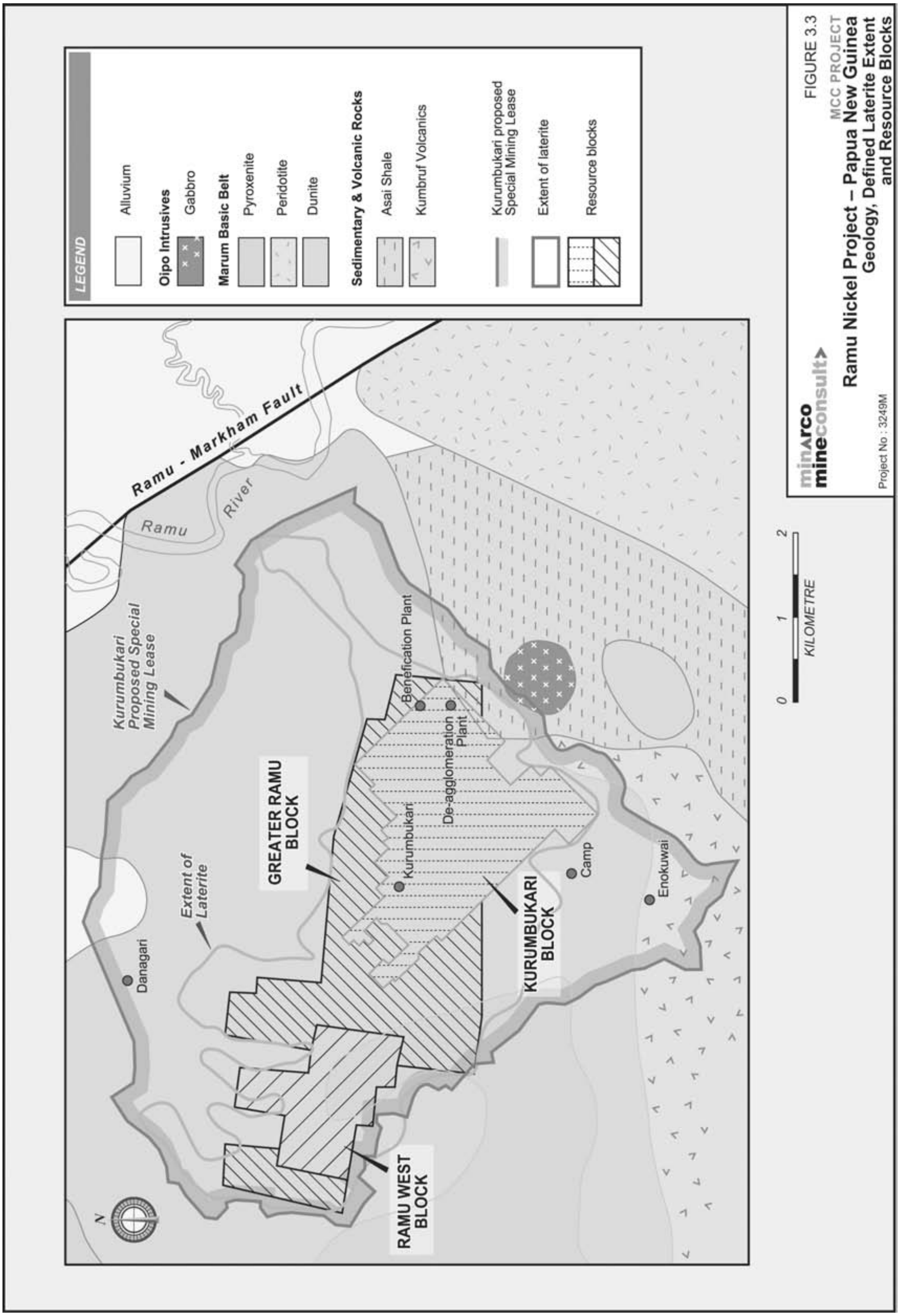
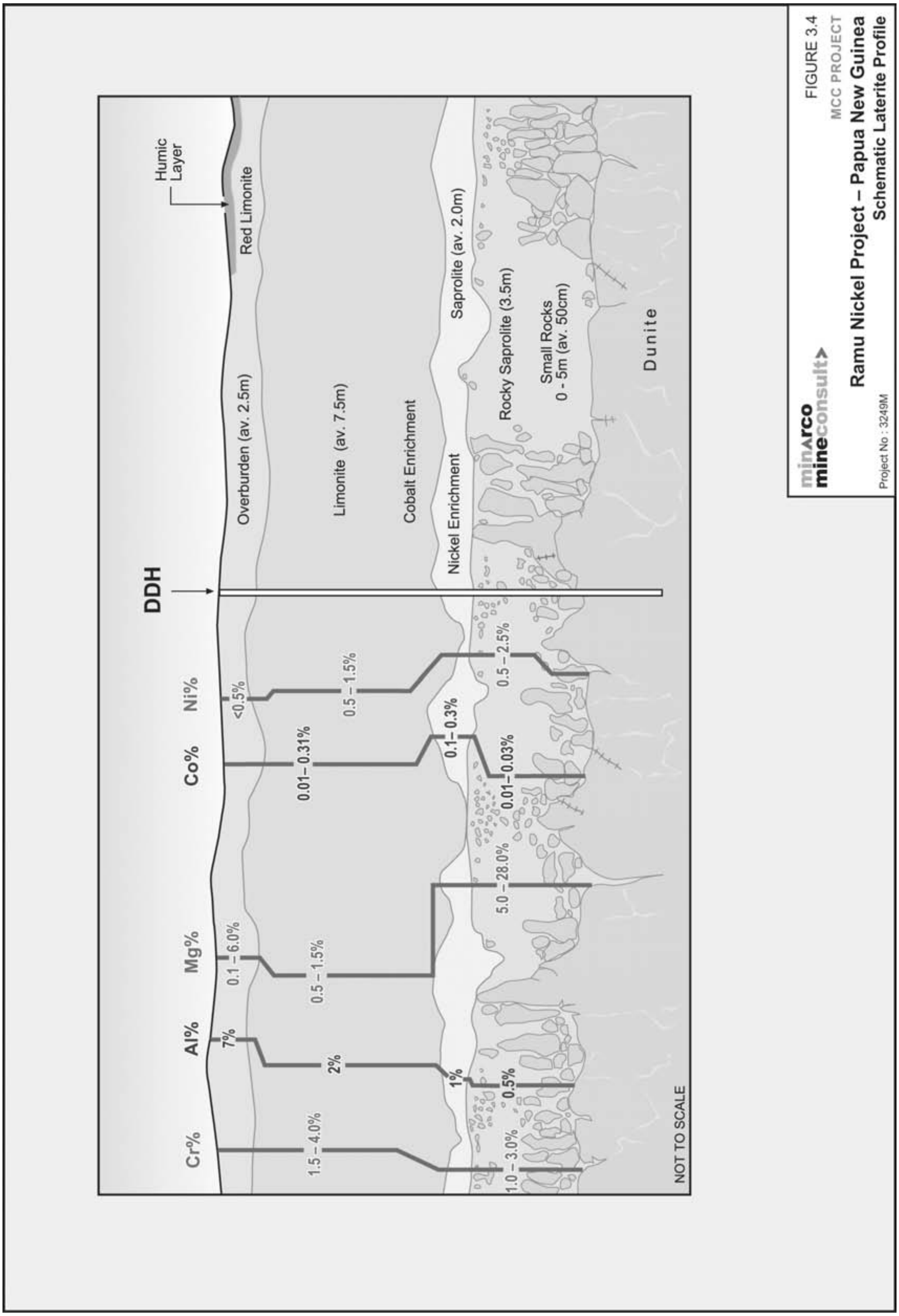


Figure 3.4 — Ramu Nickel Laterite — Laterite Profile



3.6 RESOURCES AND RESERVES

The Ramu Nickel Laterite deposit is similar to other operating lateritic nickel mines such as Murrin in Western Australia, Soroako in Indonesia and Goro in New Caledonia. The Goro deposit is relatively large and has approximately 165Mt at an average of 1.6% Ni and 0.15% Co, with proven Reserves of 47Mt.

M-MC reviewed the reasonableness of the Ramu estimates and reporting categories. However, M-MC did not validate the estimates.

3.6.1 Mineral Resources — In Situ Quantities

Mineral Resource estimates were prepared based on reliable data from Exploration Stages 3 and 4. This data was validated several times by others before preparing the geological and grade block model. Grade estimation procedures were reasonable and well documented in reports.

The cut-off grade (cog) for Ni used in the resource estimation process was 0.5% Ni. No grade variograms were available for review. For the Kurumbukari area, grade interpolation used a spherical model and Ordinary Kriging (OK). This was possible due to reasonable data density (infill drilling). For the other areas, Inverse Distance Squared (ID2) method was used to interpolate grades. Separate block models were developed for all three major ore bodies — Kurumbukari, Ramu West and Great Ramu.

The Ramu Project contains a total identified Mineral Resource of 143.2Mt at 1.01% Ni and 0.10% Co. This estimation is reported in accordance with the recommendations of the JORC Code. Approximately 50% of Mineral Resources are Inferred status. Measured and Indicated Resources total approximately 72Mt at 0.99% Ni and 0.11% Co and are contained within the Kurumbukari and Ramu West areas. The Limonite Zone comprises approximately 65% of Mineral Resources.

Mineral Resource estimate parameters included:

- Cut-off grade 0.5% Ni
- Minimum thickness not available
- Measured Resources <100 m x 100 m data spacing (limonite and saprolite only)
- Indicated Resources <200 m x 200 m data spacing (rocky saprolite)
- Inferred Resources >200 m x 200 m data spacing

The following ***Table 3.1*** summarises the estimates of Mineral Resources. Mineral Resources are inclusive of Ore Reserves. For Ramu, the Mineral Resource estimates excluded +2mm internal boulders. Reserves estimates included these boulders as dilution. The result is that Mineral Resource Estimates may report volumes less than ROM Ore Reserves.

Table 3.1 — Ramu Nickel Laterite — Mineral Resources

JORC Category	Tonnes	Average Grade (cog >0.5% Ni)	
	Mt	Ni %	Co %
Measured	42.4	0.93	0.11
Indicated	29.8	1.07	0.11
Inferred	71.0	1.04	0.10
Totals	**143.2**	**1.01**	**0.10**

Source: Company website
Notes: Mineral Resource estimates inclusive of Ore Reserves
Measured and Indicated Mineral Resources reported for Kurumbukari and Ramu West areas only.
Excluding +2 mm internal boulders.

MCC in 2006 as part of its due diligence on the project completed 10 confirmation drill holes over the Kurumbukari and Ramu West areas. Results from this work confirmed the grade, thickness and density of the laterite units, Small variations were recorded in the moisture content, most likely linked to the methodology employed. A 100t metallurgical sample was collected from 6 individual pits over the Kurumbukari area. Average tonnes and grades of material recovered from these pits reconciled well with the underlying drill holes.

M-MC was informed that recent grade control drilling on 25m by 25m spacing over the first planned mining area has so far reconciled well with the underlying model.

Based on the standard of the original work, the volumes of data and the recent confirmation work carried out by MCC, M-MC considers the Mineral Resource estimate to be reasonable and in line with JORC guidelines.

3.6.2 Reserves — Mineable Quantities

Ore Reserves were estimated by HGP and reported in the 2000 Feasibility Study. The estimates were prepared in January 2000 from the geological model and detailed mine plans. The Ore Reserves have since been reviewed and confirmed by SRK Consulting.

In M-MC's opinion, estimation of Ore Reserves is reasonable and conforms with JORC guidelines. M-MC assume that the reason Ore Reserves estimates are greater than Measured and Indicated Mineral Resources is that they include internal boulders of +2mm.

Ore Reserve estimates total 75.7Mt at 0.91% Ni and 0.10% Co. A summary is shown in ***Table 3.2***

Table 3.2 — Ramu Nickel Laterite — Ore Reserves

JORC Category	Tonnes	Average Grade (Cog >0.5% Ni)	
	Mt	Ni %	Co %
Proved	39.7	0.88	0.10
Probable	36.0	0.94	0.09
Totals	**75.7**	**0.91**	**0.10**

Source: *Company website*
Notes: *Including +2 mm internal boulders*

3.7 MINING

All of the engineering work associated with the mine and associated processing plants, slurry pipeline and port infrastructure has been completed by China Enfi Engineering Corporation ("ENFI") which has a Class A rating in China.

The most recent Mining Feasibility report was finalised in December 2007 and titled "Ramu Nickel and Cobalt Management (China Metallurgy) Co., Ltd., Basic design of Ramu Nickel and Cobalt Project". During M-MC's site visit, the company's management advised M-MC that all the construction work was being completed in accordance with this original Feasibility Study.

3.7.1 General Description

The proposed mining areas are split into three separate zones as follows:

- Kurumbukari (7.2km^2) (initial mining area, highest exploration confidence),
- Ramu West (3 km^2) (secondary mining area, lower exploration confidence), and
- Great Ramu (7.9 km^2) (future potential mining area, exploration confidence at an 'Exploration Target' level)

Current Status

The minesite area is currently in the early stages of construction. The most substantial infrastructure being completed is as follows:

- Access road from current work area to the mine site — completed 2008
- Construction of new bridge across the Ramu River — completed 2008
- New camp facilities at mine area — completed July 2009
- Agglomeration plant and slurry pipeline main pumping system — planned finish date, 2009
- Mine Workshops — Under construction
- Power Plant at minesite — To be build

The Company advised that the minesite commenced pre-stripping and trial ore mining in June 2009. February 2010 is the expected date for commencement of operations of the refinery. The mine will have completed all pre-strip requirements, trial ore production, trial operations of agglomeration and slurry pumping facilities.

Mining

Mining is proposed to be by open cut methods using 5.5 cubic metre excavators and backhoes and 40t articulated trucks for haulage. No drilling and blasting is envisaged because of the soft nature of the rocks but provisions will be made where harder material is encountered. A typical 15m depth lateritic ore profile will be mined by using a face excavator up to a maximum height of 10m in advancing mode and 5m in retreating mode using backhoe configuration. The upper contact zones between the ore and overburden and the lower contact zone with the dunite base rock will be cleaned using 1.6 cubic metre backhoes to minimize mining losses and ore dilution. The overburden and topsoil are to be removed by dozing and used for rehabilitation as soon as the bench has been mined. Articulated trucks are preferred over rigid trucks for their road handling abilities based on experience in laterite deposits under similar conditions.

More than one mining face will be operated depending on the quality of available ore and requirements for blending. The average grade mined is expected to be 1.09% Ni and 0.10% Co with a cut of grade of 0.7% Ni. The average lead distance from the mining to the proposed de-agglomeration plant is expected to vary between 0.5km and 1.2km in the first few years. Blending stockpiles are also proposed near the de-agglomeration plant for blending the ores. Oversize from the de-agglomeration plant will be used for road making in the mine by transporting in empty returning trucks from the production cycle.

A schematic representation of the mining method is provided in ***Figure 3.5***.

Figure 3.5 — Ramu Nickel Laterite — Mining Method Schematic

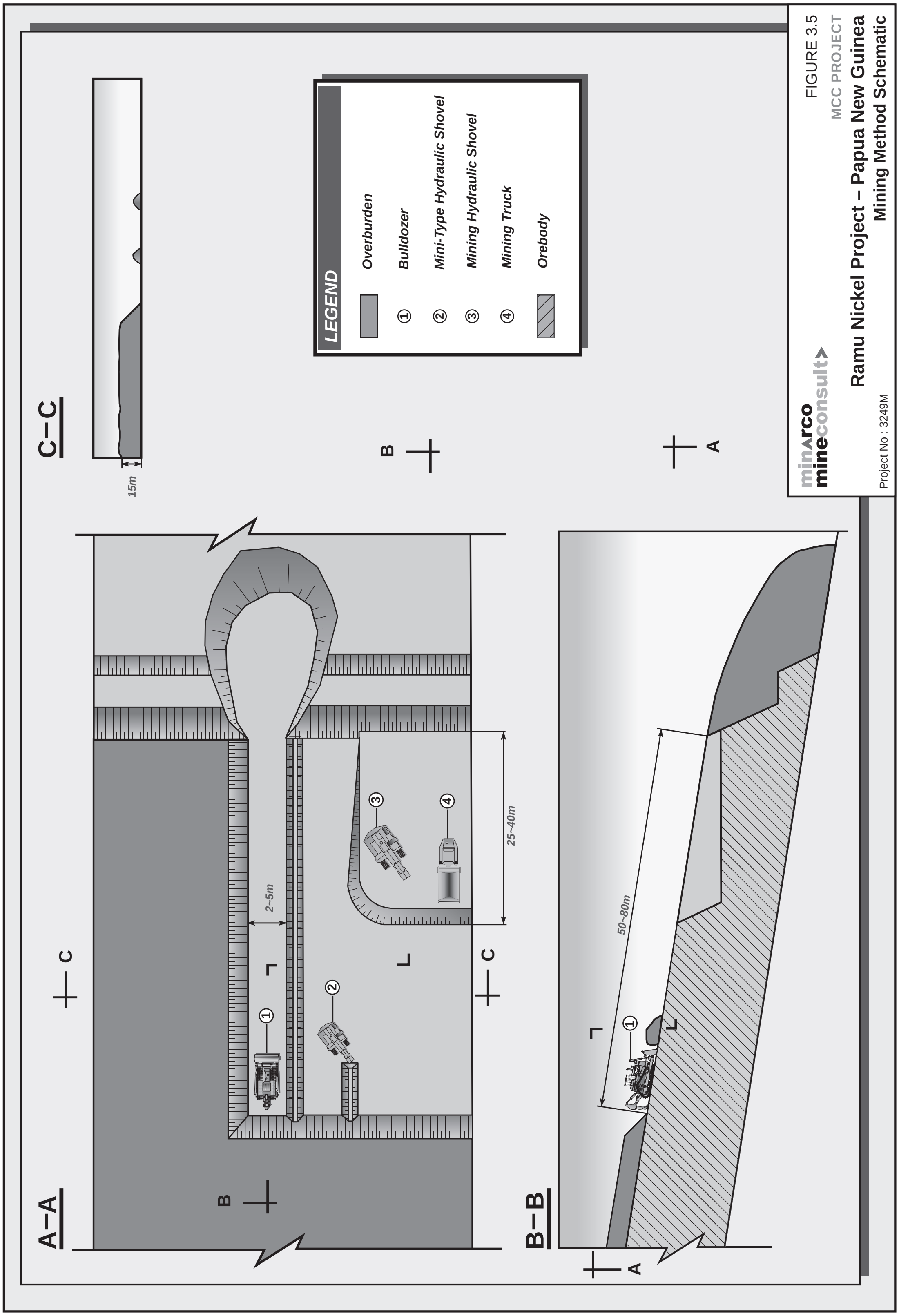
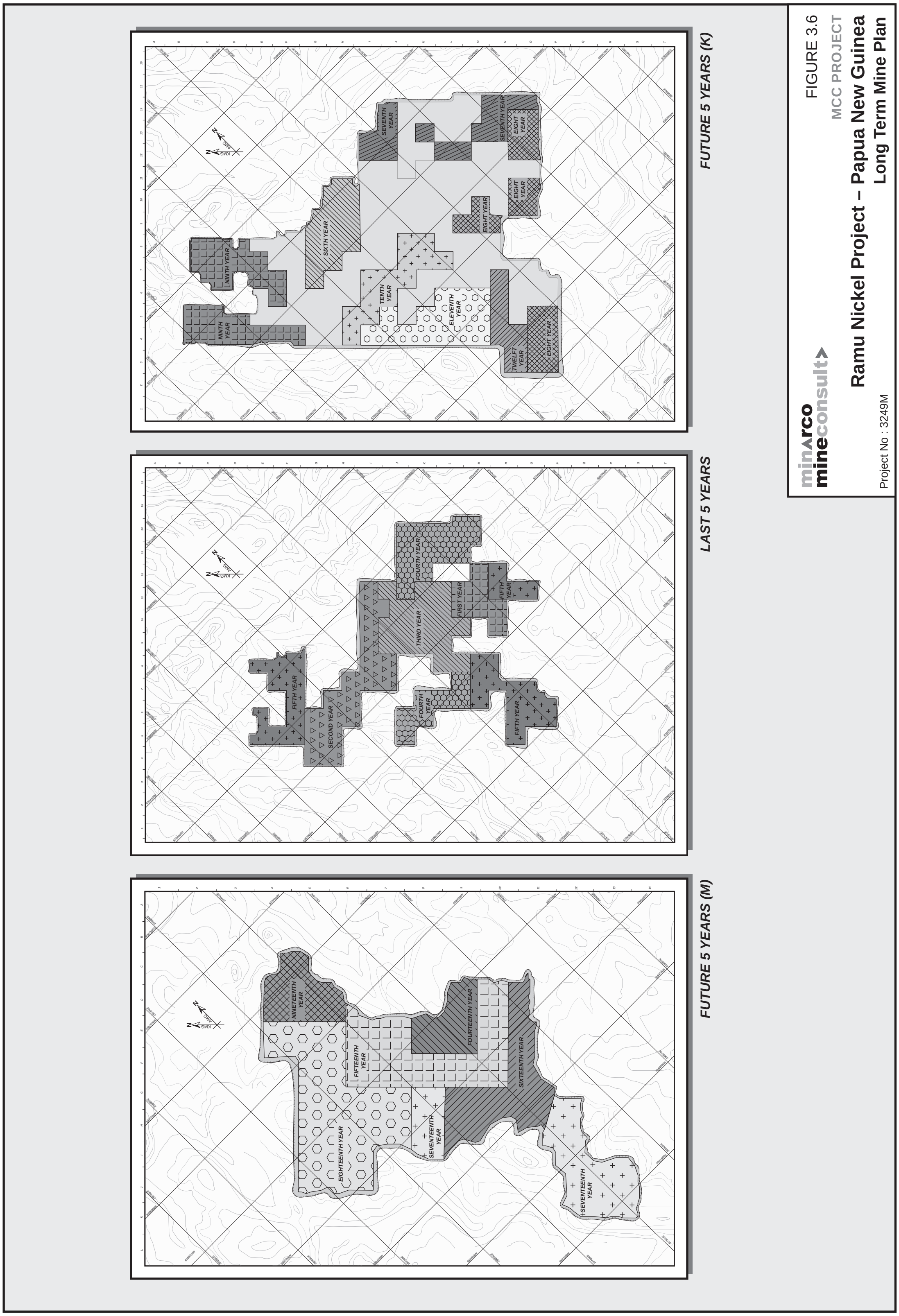


Figure 3.6 — Ramu Nickel Laterite — Mine Plan



3.7.2 Forecast Production

Based on the design specifications, the following production levels are planned:

According to the Company, ground preparation, overburden removal and incidental mining production will be completed during the first half of 2009 prior to commencement of full scale production in the second half of 2009. It is difficult to see this timetable being fully met and it is unlikely that more than 450,000 tonnes of material would be mined and processed during 2009. The planned production profile is shown in ***Table 3.3***.

Table 3.3 — Ramu Nickel Laterite — Forecast Production

Production	2009	2010	2011	2012	2013	2014
ROM ktpa	900	1,800	3,600	3,600	3,600	3,600
Ni Grade %	1.15	1.15	1.12	1.11	1.07	1.05
Co Grade %	0.12	0.12	0.11	0.11	0.13	0.1
Output						
MHP tpa	19,933	39,863	77,535	76,804	73,749	72,418
Ni metal tpa	8,571	17,141	33,340	33,026	31,712	31,140
Co metal tpa	869	1,737	3,192	3,192	3,757	2,910

Source: MCC provided Capex and Opex figures February 09

The above forecast is based on the availability of proper infrastructure and equipment and with no undue constraints during the development and mining phases.

The mine plan showing the location of the working areas for years 1 to 19 are shown in ***Figure 3.6***.

3.8 MINERAL PROCESSING

General Overview of Processing

The processing stages are as follows:

- Ore Washing — where the mined ore is screened and scrubbed to remove large sized material and gravel.
- Chromite Recovery — recovery of the chromite from the washed ore.
- Refinery: the slurried ore is transported to the refinery by pipeline and leached with sulphuric acid under pressure and high temperature (HPAL). The Ni and Co are recovered as hydroxides (known as Mixed Hydroxide Product or MHP) with the addition of sodium hydroxide and calcium hydroxide, after the iron, aluminium, magnesium and manganese have been removed with limestone. The selective removal of these impurities before the precipitation of the Ni and Co is achieved by carefully controlling the pH of the slurry.

The final product from the operation is a MHP, often considered an intermediate product. Along with Mixed Sulphide Product (MSP), the original proposed product, these products are sold to nickel refineries for the subsequent extraction of the nickel and cobalt. A comparison between the two Intermediate products is shown in ***Table 3.4***, noting the higher moisture content associated with the MHP product. MHP is commonly produced in many laterite processing plants, such as several of the Australian operations as well as Moa Bay and Goro, who then further process (typically ammonia, solvent extraction and electrowinning) the product on site to yield nickel and cobalt. There are a number of nickel refineries around the world that take this product, depending upon market conditions. The production of MHP has a number of advantages, such as safety (no hydrogen sulphide involved) and being an established and well known process. In MCC's case, the off-take agreements with the Chinese nickel producers (and fellow joint venture partners) specify MHP, since their nickel refineries are designed to process this product type. This saves the need for an expensive nickel refinery to be constructed at Basamuk.

Table 3.4 — Ramu Nickel Laterite — Comparison between Intermediate Products

Product	Ni (%)	Co (%)	S (%)	Fe (%)	Mg (%)	Zn (%)	H_2O (%)
MSP	50-55	5-5.5	35	1.1	0.01	0.10	20
MHP	43	4.3	2-3	0.24	1.5	1.2-1.4	35-40

Source: 2008 Update Feasibility Report

Construction is well underway, with completion of most facilities planned for Q3 2009. This may well prove to be challenging given the shortage of skilled labour and the current wet season (January to May). MCC plan to introduce an additional shift in April; however it is unlikely to meet the proposed deadlines as stated by the company. Assuming that the equipment is available for installation at Basamuk and the mine site, it would appear that the refinery and processing plant would not be ready before September at the earliest for commissioning. For example, during the M-MC visit, no work had commenced on either power stations and it is difficult to imagine that they would be completed and operating satisfactorily within 4 months. Access to the mine site is currently difficult due to the wet and slippery nature of the road and deliveries of equipment, construction materials and consumables (e.g. fuel) was difficult to achieve. It is possible that the completion of construction may slip to December. The company believes that production of MHP is achievable by March 2010, which is reasonable; however it is unlikely to be produced in the quantities forecast by MCC. It appears that substantially less than 19,000 tonnes of MHP would be produced in 2009 (probably around a thousand tonnes at best) while during 2010, assuming no difficulties are experienced, possibly 20,000 to 30,000 tonnes of MHP will be produced.

M-MC also suggests that both the commissioning schedule (currently 2 months), and the ramp up to full production (currently planned for two years), may also prove challenging given the difficult environment in which they operate, and the historical difficulties that other nickel laterite projects have experienced.

Ore Washing and Chromite Recovery

After mining, the ore will be screened and scrubbed to remove the coarse material as aggregate and mine-fill. The chromite will also be removed from the slurried ore and upgraded to a marketable product using gravity and magnetic separation. Based on mining 3.6Mtpa the annual projected volumes of these materials are shown in ***Table 3.5***.

Table 3.5 — Ramu Nickel Laterite — Ore Washing Products

Process Stream	Capacity (ktpa)
Washing Plant Feed	3,600
Chromite (−3mm)	135
Chromite tailings	8
Aggregate (+ 50mm)	539
Mine fill (−50mm/+3mm)	215
HPAL Feed	2,703

Source: Pro-Rated by MMC from 2008 Update Feasibility Report

The removal of the coarse material and the chromite will minimise the wear on the pipeline as well as reducing the acid consumption in the HPAL. This is due to the fact that most of the magnesium minerals, which consume significant amounts of acid, are associated with the chromite.

Ore from mining is delivered to a 40kt stockpile and during the wet season to a covered 20kt stockpile, which provides about 36 hours of production. The ore washing plant will treat 570tph of ore and will be conducted in three parallel scrubbing and screening operations, each consisting of a rotary screen (3m Ø x 10 m) to remove the coarse material (+50mm) followed by two scrubbing screens. The undersized stream (−50mm) will be further screened to remove material coarser than 3 mm and the −3mm fraction will be stored in three intermediate storage tanks (each 275m^3) for chromite removal. This stream is further classified at 53 microns with hydrocyclones, with the material coarser than 53 microns becoming the feed for the chromite recovery section, while the underflow wil be directed to the thickener feed storage tanks (two tanks each 196m^3). The chromite recovery section will employ gravity separation using spirals followed by shaking tables to upgrade the concentrate before final cleaning in a magnetic separation circuit. The coarse fraction in the tailings (+150 microns) from this section will be separated by hydrocyclones and ground in a ball mill (2.7mØ x 4m) until finer than 53 microns before being transferred to the thickener feed storage tanks. Two 43m diameter thickeners will upgrade the slurry solids content from around 4-5% to desired 18.3% and stored in four storage tanks (each 1885m^3) which will provide 10-12 hours of storage before pumping to the refinery. Approximately 50g/t of flocculant will be required to thicken the slurry to this solids density. The chromite concentrate will be stored in a 160kt tonne stockpile before transportation to the markets.

The pumping station will consist of six pumps (2 standby) that will feed a 610mmØ pipeline that traverses 135 kilometres from the ore washing and chromite recovery area to the refinery site at Basamuk, dropping 700 metre in height (see ***Figure 3.6***), The slurry density of the thickened slurry can be adjusted as required while the pressure in the pipeline is monitored and controlled by five stations along the length of the pipeline.

Figure 3.7 shows the washing/scrubbing flowsheet while ***Figure 3.8*** presents the slurry pipeline pumping flowsheet.

Figure 3.7 — Ramu Nickel Laterite — Washing/scrubbing flowsheet

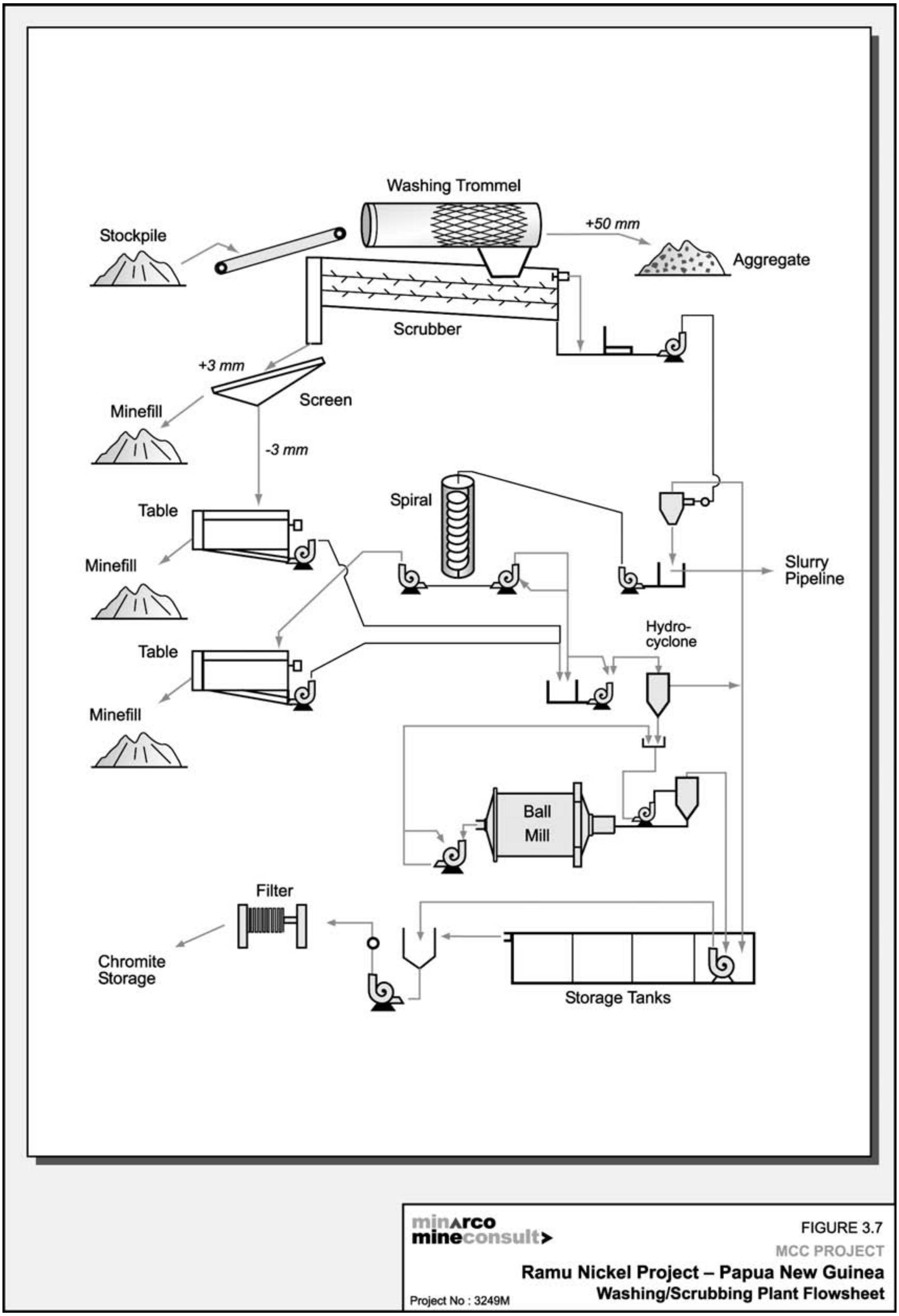
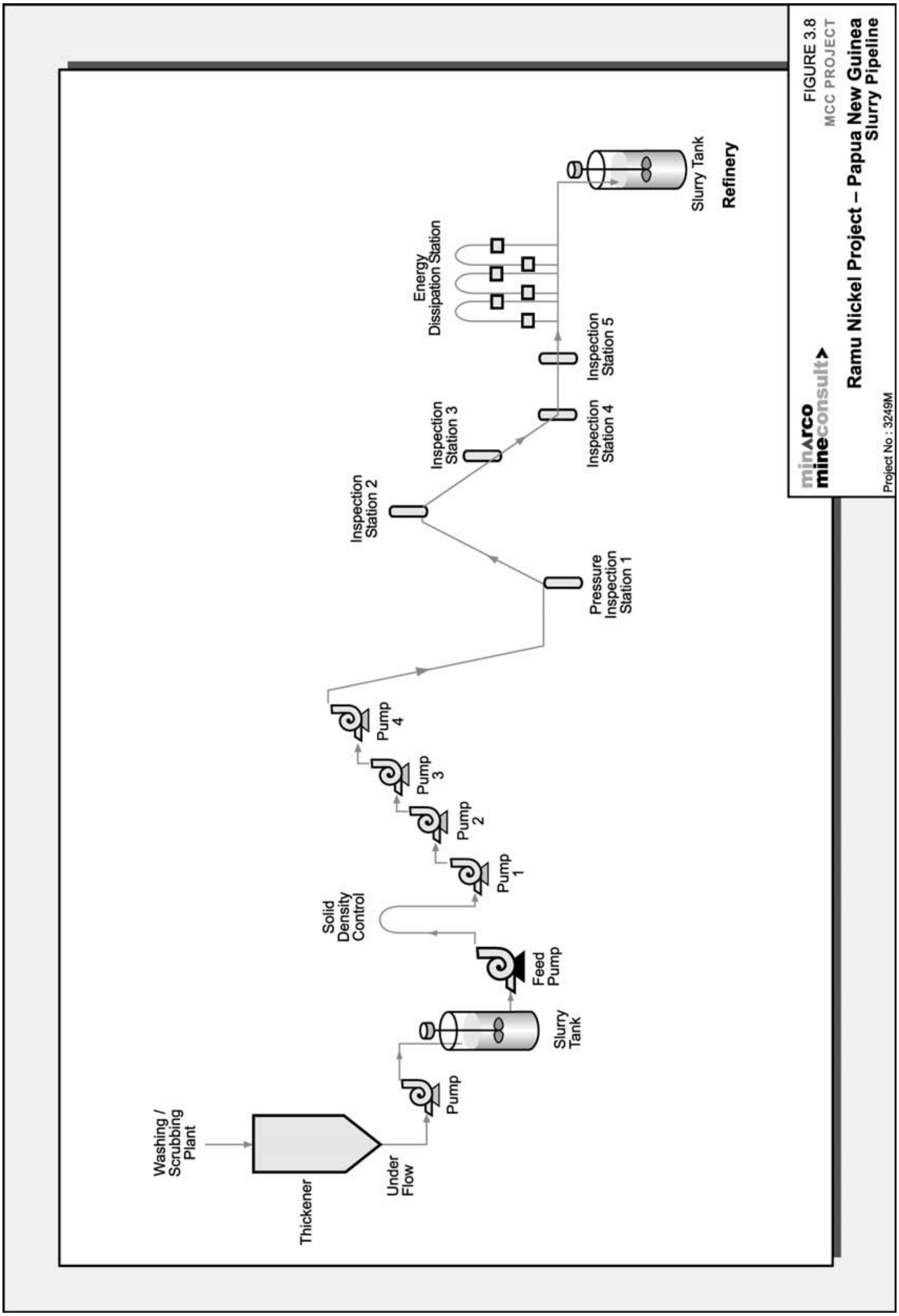


Figure 3.8 — Ramu Nickel Laterite — Slurry pipeline pumping flowsheet



HPAL/Refinery

At the Basamuk processing site, the HPAL/Refinery will consist of three stages. The first stage will dissolve all of the base metals using sulphuric acid (HPAL), followed by selective removal of the iron (as hematite), aluminium and other elements and finally recovery of the nickel and cobalt as hydroxides.

In the first five years, an average of 2Mt of slurry averaging 1.09% Ni and 0.11% Co will be treated annually at the refinery to produce an average of 57.6kt of a mixed Ni-Co hydroxide product. Eighty-nine percent of the Ni and eighty-eight percent of the Co will be recovered to a product assaying 43% Ni and 4.3% Co on a dry basis. The final product will also contain minor quantities of other elements such as Zn (1.2-1.4%) and Mn (2.5%) and typically contain 45% moisture.

The slurry from the ore washing stage is stored in six stirred tanks with a total residence time of 40 hours. The slurry is then thickened to 32% solids and pumped to three parallel horizontal, titanium clad autoclaves (5.4m diameter by 40m long) where sulphuric acid and steam are added to extract the base metals. The autoclaves have seven stirred chambers, providing a residence time of 1 hour and operate at 255^{o}C and 4.7 Mpa. The typical acid consumption will be 240kg/t, which is low by world standards, and in the later years, when less limonite will be present in the blend, is expected to increase to 260kg/t. The sulphuric acid will be produced on site by an acid plant burning sulphur while the large quantities of steam and power required to run the operations will be provided by a power station running on heavy oil (refer to Section 3.9).

After digestion in the HPAL, the slurry will be cooled down to 100^{o}C and depressurised and the heat recovered to pre-heat in the incoming HPAL slurry from 25^{o}C to 195/215^{o}C. The cooled slurry will be treated with sodium hydrosulfide to assist in the removal of any dissolved chromium while the pH will be adjusted to between 2 and 2.5 using limestone. The undissolved solids will be removed from the solution containing the dissolved metals in a processing stage known as counter current decantation (CCDs) employing a series of six thickeners and one large diameter thickener (47mØ). Flocculants will be used to improve the separation between solids and liquids. The solids from this stage, as well as the precipitated materials from the following purification stages, will be neutralised with limestone (pH 7-8) and disposed of in a deep sea burial process.

The first elements to be removed in the purification process are aluminium and iron and two stages will be required to remove all of the iron and ninety-nine of the aluminium. In the first stage, the slurry pH will be adjusted to between 3.5 and 3.8 with limestone, which will remove all of the ferric iron and 70% of the aluminium. The remaining iron and aluminium will be precipitated by adjusting the pH to 4.5 to 4.8 with limestone and applying aeration. The precipitated solids will be separated from the solution using a CCD circuit.

After having purified the solution by removing the unwanted metals, the Ni and Co will be recovered in two stages, firstly using sodium hydroxide to adjust the pH to 7.2 to recover 90% of the two metals. The residual Ni and Co is recovered with calcium hydroxide at pH 7.8.

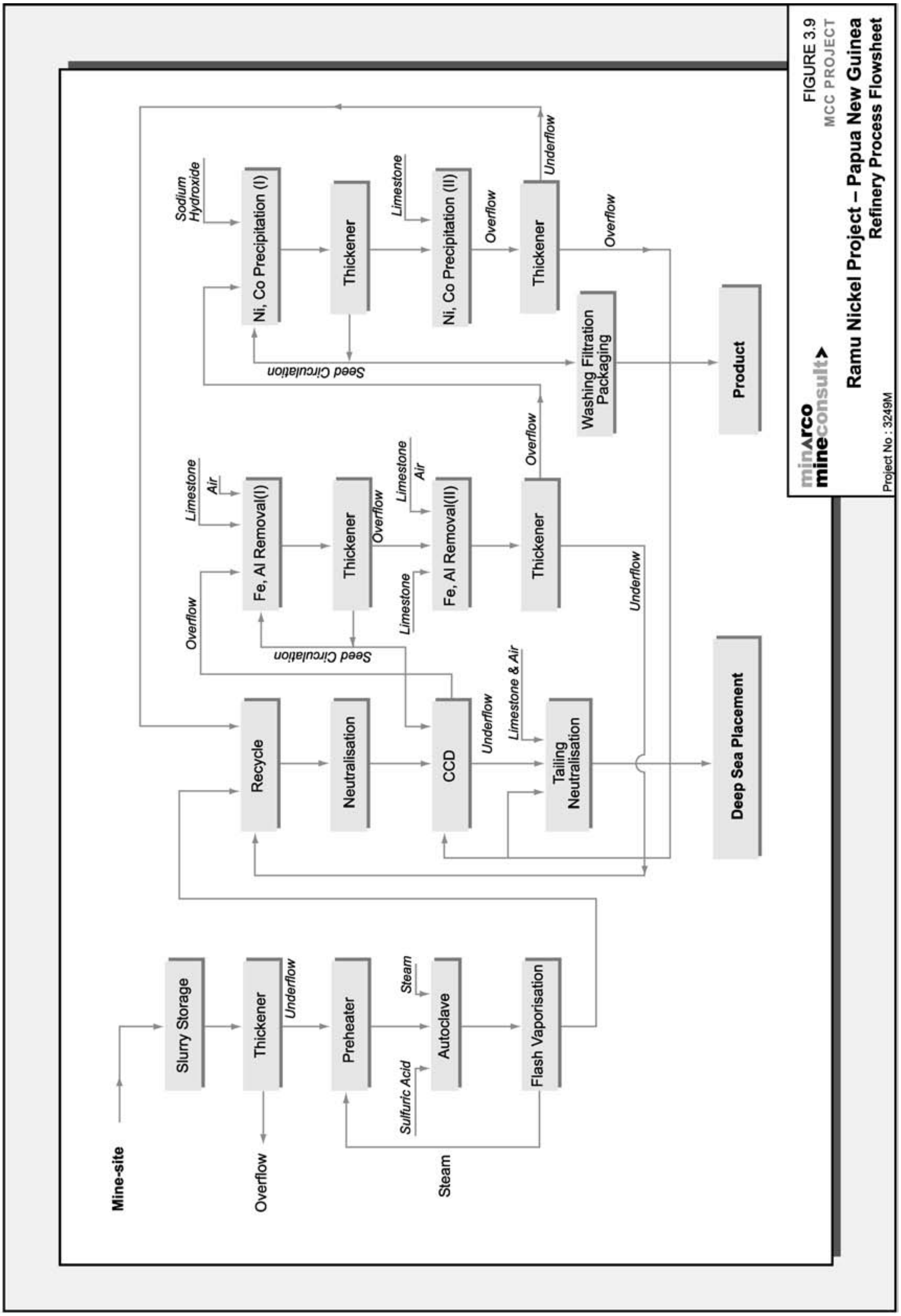
The refinery flowsheet is shown in ***Figure 3.9***.

M-MC makes the following comments based on its review of the technology:

- Adequate surge and storage capacity has been applied to all stages of processing, notably the ore washing, scrubbing and thickening stages as well as the pumping and leaching stages
- The operability and efficiency of the scrubbing equipment at the full scale duty remains to be confirmed
- Low acid consumption rates are expected with the treatment of the Ramu ore blends over the life of the project

- The process employs three autoclaves with each autoclave approximately 40m long with reaction temperatures going from 195^{o}C at the feed to 255^{o}C at the discharge. This technology is in use at a number of lateritic nickel refineries worldwide and is considered to be appropriate for the stated process. The primary CCD circuit consisting of seven thickeners for removal of the leached solids appears appropriately sized for the duty. It should be noted that the start up of a refinery of this type is a slow process where the heat built up in the CCD circuit takes between 7 and 10 months to reach equilibrium.
- The details of the other CCD circuits, namely that for the iron-aluminium removal and the recovery of the MHP product were not available and a comment on their adequacy cannot be made
- The process has been fully modeled and audited by several groups of specialists. While details of the modeling and the audit reports were not sighted by M-MC, it is reasonable to believe that the basis for the process and the subsequent process flowsheet and equipment meet the project needs based on the design criteria used in the modeling.
- The refining and purification process is based on well established precipitation technologies and is a relatively simple process to operate
- A process producing MHP as the final product is a far safer operation than the previous design where hydrogen sulfide gas was used to produce a MSP. More importantly, it meets the specifications of the buyers.
- The entire Ni and Co production is taken up in agreed take-offs and effectively does not have to compete in the world market
- The project has an ambitious timetable in terms of construction, commissioning and ramp up to full production. It is possible that the plant may produce MHP in the 4th quarter 2009, however not in substantial quantities until the latter part of 2010.

Figure 3.9 — Ramu Nickel Laterite — Refinery flowsheet



3.9 INFRASTRUCTURE AND SERVICES

The Ramu Ni-Co Project is being developed in a region where there is very limited infrastructure and has required the development of temporary infrastructure such as roads and bridges to permit construction at the minesite to begin as well as the installatioin of permament infrastructure such as the port at Basamuk to allow equipment and pre-fabricated plant to be delivered.

Both the mine-site and the refinery require independent and stand-alone infrastructure, which includes power, communications, accommodation, workshops and spares. The infrastructure for all aspects of this project has been carefully considered and both suitable and appropriate for this project.

The headoffice at Madang requires substantially less infrastructure, namely offices, accommodation and communications. Additionally, Basamuk is accessed by boat from Madang, where MCC operates two boats.

Power

Power on the mine-site will be supplied by a 22MW facility consisting of 6 diesel fired 3.78MW generator sets, with 2 on standby. At the refinery, a 60MW facility will supply both power and steam for the leaching operation. This will consist of 8 heavy oil fired 7.5MW generating sets, with three units (1 standby) dedicated to power generation and three units and 1 standby to steam raising (104tph of steam). Power will be generated at 10.kV and transformed where required as reticulation as 380/220V. Steam will also be captured for pressurizing the autoclave from the sulphur burning facility.

Power costs are expected to be around USD140-190MWhr, depending upon heavy oil and diesel prices. Power production will be 398.27GWh/a, with 337.5GWh/a being consumed at the refinery and 60.77GWh/a at the mine-site.

M-MC is co-operating in the development of the second phase of the Ramu Hydroelectric Yankee Dam Project which should be completed in the next 4 years. This would allow the project to have access to significantly cheaper power.

Water

Water for the mine-site operations will be sourced from the Gagaiyo River and supplemented by the capture of the mining site water run-off. A dam has been constructed on this river and water delivered to water storage facilities by pipeline and a pumping station at the mine-site. 58.22ML/day (0.67m^3/s) will be required to operate the washing and scrubbing facilities on the mine-site. Water will also be recovered at the mine-site for re-use after the scrubbed slurry is thickened for pumping to the refinery.

At the refinery site, some 30ML/day (0.35m^3/s) will be required and sourced from the nearby Yaganon River. An extensive system of large deep wells, have been developed in the delta of the Yaganon River with submersible pumps to capture the water. Additionally, water will be also recovered for re-use at the refinery when the slurry from the mine-site is further thickened for leaching in the autoclaves.

For the preparation of sulphuric acid, water from the Yaganon River will be treated at the site to remove fine solids and contaminants.

Drinking water will be supplied from both the collection of rainwater as well as the Gagaiyo and Yaganon rivers at both the mining and refinery sites and treated to produce potable water.

Port

A large concrete wharf at Basamuk has been constructed capable of birthing 60,000 dwt ships. The gantry cranes await installation.

Other facilities

The other facilities are a double contact sulphuric acid production and a limestone preparation plant including a calcining kiln.

The sulphuric acid plant will burn 304ktpa of sulphur to produce around 900ktpa of 98.5% sulphuric acid.

The limestone preparation will receive locally mined limestone and consists of a crushing, screening and ball milling equipment to produce a finely ground slurry (P_{95} = 74 microns, 30% solids) for precipitation and neutralisation. 70ktpa of the −40mm/+15mm limestone will be calcined (400tpd) and then slaked as a slurry (20%) for precipitation of the last quantities of Ni and Co.

3.10 CAPITAL AND OPERATING COSTS

The projected capital are summarised in ***Table 3.6***, based on the 2008 Feasibility Study. The total project costs are estimated at USD1.37 billion, which is relatively low compared to many other nickel laterite of similar production rate. Site personnel believe that this estimate will not blow out.

Table 3.6 — Ramu Nickel Laterite — Forecast Capital Costs

Cost Centre	USD(000's)	% of Total
Mine	128,938	9%
Slurry Pipeline	82,370	6%
Refinery	588,159	43%
Madang Office & communications	16,046	1%
Other Costs	301,700	22%
Project Contingencies	166,091	12%
Loan Interest (construction period)	74,277	5%
Working Capital	16,303	1%
Total	**1,373,884**	**100%**

Source: Costs provided by company as at January 2008.
Other costs include fees, insurances, commissioning, taxes, owner's costs, study costs, environmental, social, PR, HSE and spare parts

40% of the equipment spend has already been made, with 60% outstanding mainly in the form of progressive payments or monies retained for successful completion and operation.

The operation will employ some 1,205 people and the forecast mining operating costs are shown in ***Table 3.7***.

Table 3.7 — Ramu Nickel Laterite — Forecast Mining Operating Costs

Cost Centre	Units	2009	2010	2011	2012	2013	2014
Auxiliary Material	USD (000's)	4,945	8,992	16,348	14,862	15,605	16,385
Labour	USD (000's)	5,731	4,585	6,113	4,585	4,585	4,585
Others	USD (000's)	1,355	2,710	5,420	5,420	5,420	5,420
Total	**Total USD**	12,031	16,286	27,881	24,867	25,610	26,390

Source: Costs provided by company as at February 2009.

The estimated processing operating costs are summarised in ***Table 3.8*** before Co and chromite credits. The main costs are associated with raw materials or consumables such as:

- Sulphuric acid (240 — 260kg/t to dissolve the Ni and Co)
- Sulphur (304.5ktpa at USD60/t to make sulphuric acid)
- Heavy oil (88.7ktpa at USD315/t to provide power and steam)
- Limestone (800ktpa for precipitation and neutralisation)
- Sodium Hydroxide (53kt [1.65kg/t] at USD370/t)
- Water (33.685 ML/day or 3.83m3/t)
- Grinding media (0.9ktpa [279g/t] at USD980/t)
- Flocculant (1.7ktpa [53g/t] at USD3,500/t)

The development of the capital cost is thorough and based on Chinese cost structures, reasonable. The actual capital cost may not increase due to the decrease in the cost of some items towards the end of the construction period. The costs of consumables used in the development of the operating costs reflect current prices however no allowance has been made for inflation. The actual operating cost will depend upon the actual consumption rates of the consumables, which may be higher during commissioning and the early stages of the ramp up period.

Table 3.8 — Ramu Nickel Laterite — Forecast Processing Operating Costs

Cost Centre	Units	2009	2010	2011	2012	2013	2014
Auxiliary Material	USD (000's)	27,140	49,342	87,246	78,568	151,044	148,320
Labour	USD (000's)	8,903	5,935	7,696	5,717	12,413	12,189
Others	USD (000's)	3,711	7,421	14,434	14,298	13,729	13,481
Administration	USD (000's)	8,019	16,036	31,191	30,898	29,668	29,133
Total	**USD (000's)**	**47,772**	**78,734**	**140,567**	**129,481**	**206,854**	**203,124**
USD/lb Ni		2.53	2.08	1.91	1.78	2.96	2.96

Source: Costs provided by company as at February 2009.

3.11 SAFETY AND ENVIRONMENT

The major environmental issue for this project relates to the tailings disposal. Due to the high rainfall in the area, deep sea tailings placement ("DSTP") has been selected as the preferred option. The DSTP method is similar to that used in other comparable tropical locations in Papua New Guinea such as Lihir and Misima. The tailings will be disposed approximately 2km offshore from the processing plant facilities at a depth of 150m below sea level beneath the phototropic zone as well as the mixing zone. The slurry, after de-aeration and being diluted with sea water, will be placed in a deep marine valley and will be moderately small in quantity compared to the volumes of sediment deposited annually by the rivers. Extensive studies have concluded that the proposed tailings disposal method is the most appropriate for the circumstances and presents low and manageable environmental risks. The project is supported by an approved and extensive Environmental Management Plan covering all aspects of the mining and processing operations.

M-MC makes the following observations with regard to the environmental situation:

- New EMP produced by MCC, involving NSR Consultants and specialist marine biological consultants
- New monitoring program undertaken to prepare report from October 2006 to June 2007

Memorandum of Agreement (MOA):

- MOA involves all key stakeholders (4 landowner associations), shareholders, National and Provincial Governments. It includes the Local Economic Development Plan and is managed by the Community Affairs Department.
- Four landowner associations: Coastal — refinery area; Coastal — pipeline; Inland — Pipeline and Mine area.
- As a result of MOA, MCC involved in meeting its various obligations including sponsorship, culture, sports, education and gender issues. MCC has worked with schools, churches, etc. to carry out renovations, AIDS clinics, etc.
- A key element of the MOA related work is to develop the local economy including agriculture development (e.g. rice growing near the mine site area, poultry, etc).
- Two key roads have been built by MCC in the province as well as one 15km road linking the mine from the highway. A permanent steel and concrete bridge has been built over the Ramu River to access the minesite as well as a smaller bridge locally.
- There is a focus from Community Affairs Department in Madang on gender and youth issues for MCC.

Two dams have been built at the mine-site to capture run-off from the mine, particularly during stripping, and prevent pollution of the nearby Gagaiyo River. These dams will allow the fine solids to settle while the water will be used in the washing and scrubbing processes.

4 AYNAK COPPER PROJECT

M-MC did not make a site inspection of this property due to perceived political instability in Afghanistan. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "1978, Aynak Geology Report" prepared by the Soviet Union Geological Prospecting Group. (SUGP)
- "1987, Aynak Copper Mine West section, Reserve Estimate Survey Results Report" prepared by the Russian Geological Research Construction Bureau. (RGRCB)
- "2008 Afghanistan Aynak Copper Feasibility Report" prepared by China Central Engineering Institute for Non-ferrous Metallurgical Industries (ENFI)
- "Revised 2008 Afghanistan Aynak Copper Feasibility Report" prepared by China ENFI Engineering Corp (Revised 2008 Feasibility Report)
- "Minerals in Afghanistan, The Aynak Copper Deposit Report" prepared by the Afghanistan Geological Survey Kabul, Afghanistan

MCC's equity stake in the Aynak Project is 75% with Jiangxi Copper Group owning the remaining 25%.

4.1 BACKGROUND

The Aynaka (Aynak) Copper (Cu) Mine (69° 18' 18" longitude and 34° 15' 58" latitude) is located in the Islamic Republic of Afghanistan approximately 35km south of the city of Kabul ***Figure 4.1***. The mine is located in a mountain basin with elevations ranging from 2,275m to 2,675m and the surrounding mountains reach a maximum elevation of 3,450m.

The Aynak deposit is situated on an anticline structure. For this reason, the Aynak Cu Mine has been divided into two (2) separate mining areas namely the Middle District and the Western District. The Middle District is situated on the shallow dipping eastern limb of the anticline and the Western District is situated on the periclinal closure at the western end of the structure ***Figure 4.2***.

Small historical excavations and pits have been discovered in the Aynak area along with the remains of smelting furnaces. Russian geologists re-discovered the Aynak area in 1974 and detailed exploration of the area was carried out phases from 1974 to 1976 and again from 1978 to 1989. The civil war that began in 1989 halted any further work in the area until the mine was acquired by MCC and the Jiangxi Copper Group in November 2007.

The potential for this project is very large and MCC has significant plans for the development of this project. These plans include the initial development of an open cut and mineral processing facility to produce high grade copper concentrates. In later years, it is planned to increase both the mining and treatment capacity and produce 220ktpa of cathode copper on site.

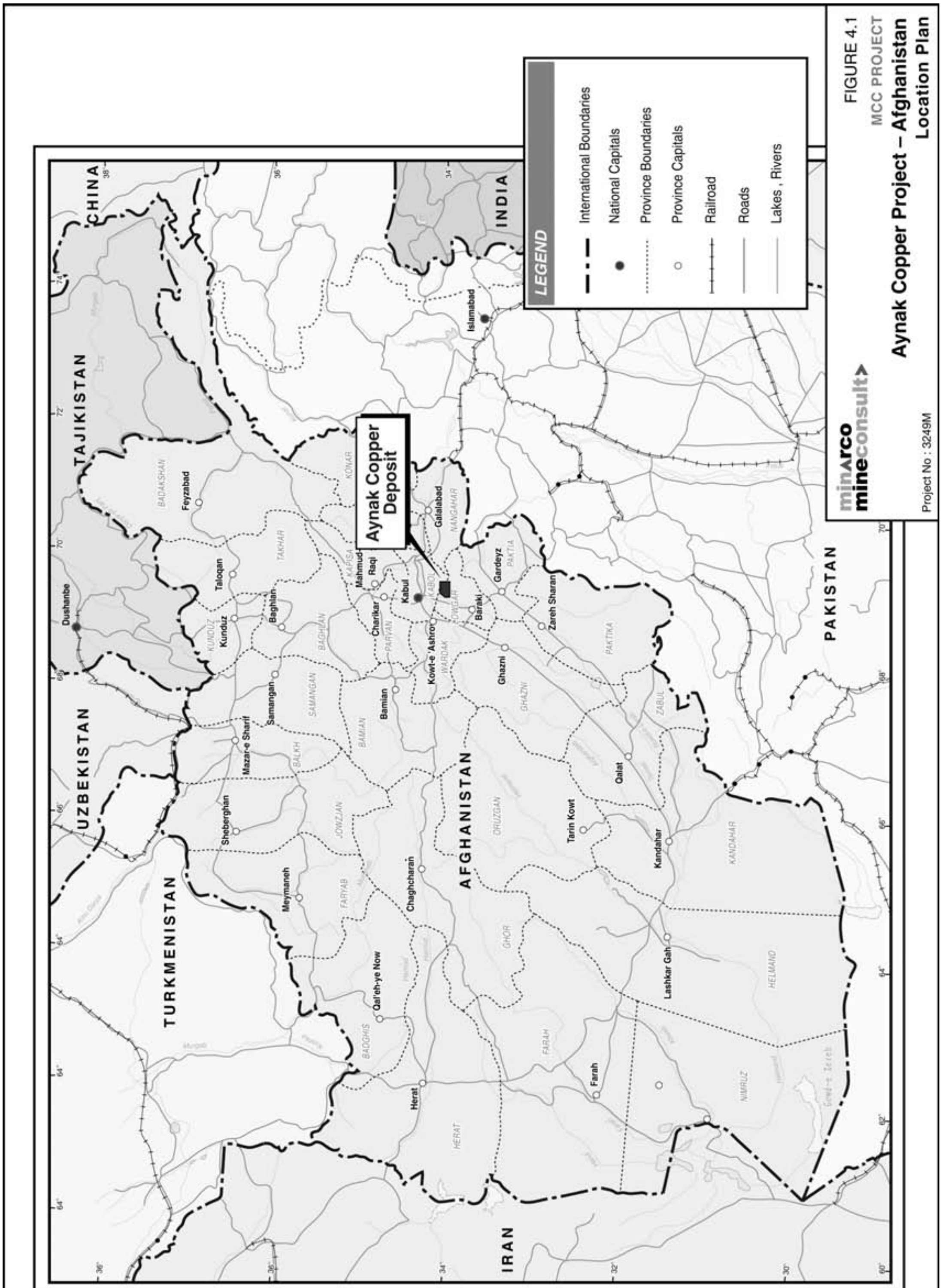
It is relatively early in the development cycle of this project, and while ENFI has access to previous Russian studies, evaluation of the three ore bodies, process behaviour of ore samples and the design of the processing facilities has yet to be completed. A Pre-Feasibility study has been conducted by ENFI and has provided an order of magnitude estimate of the economic requirements of the project. A number of infrastructure and other issues need to be addressed before the potential of the project can be fully realised. It would seem likely that this project is some years away from construction and subsequent operation.

4.2 ASSETS

The assets and status include;

- Mine development project, with some existing administration infrastructure, commencement planned for 2010.
- Russian Resource Code Compliant estimate of:
  - Middle District: 185.1Mt at 2.37% Cu (variable Cu cog).
  - West District: 298.3Mt at 1.5% Cu (variable Cu cog).
- Mineable Quantities of 349.5Mt at 1.22% Cu (variable Cu cog).
- A preliminary feasibility study "Revised 2008 Afghanistan Aynak Copper Feasibility Report" by ENFI

Figure 4.1 — Aynak Copper Project — Location Plan



4.3 LAND TENURE AND MINERAL RIGHTS

Details of the "Mining License" for Aynak are shown in ***Table 4.1***. These are valid to 2013.

Table 4.1 — Aynak Copper Project — Lease 03/87

Mine/Project	Aynak
Title	Mining License for Exploration and Exploitation
No	03/87
Owner	MCC - JCL Aynak Minerals Company Ltd
Mine/Project Name	Aynak Copper Project
Mine Method	Opencut and Underground
Permit Capacity	n/a
Permit Area	Exploration Area-D: 106.332km^2 ; Exploitation Area-E: 28.357 km^2
Permit Depth	n/a
Valid Date	Sep, 2008 - Sep, 2013
Issue Date	Nov, 2008
Issuer	Ministry Of Mines, Islamic Republic of Afghanistan

Source: Formal documentation

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

4.4 EXPLORATION AND MINING HISTORY

Mining in the area has been ongoing since ancient times. Ancient civilizations were producing Cu slag from the near surface oxide ore with dumps and shallow pits excavated over the Aynak area.

The Aynak area was first mapped in 1881 by various European geologists. During the 1970s, the Aynakar Geological Prospecting Group of Afghanistan (AGPG) started preliminary exploration in the Aynak area with the assistance of the Soviet Union Geological Prospecting Group. This work continued in a phased approach over the entire exploration area with underground development and sampling undertaken over the most advanced Middle District from 1974-1977 and West District from 1978-1984. Exploration activities are summarised in ***Table 4.2***.

Due to the break out of the Afghan conflict in the 1989 all development work stopped.

Table 4.2 — Aynak Copper Project — Exploration and Mining History

Year	Activity	Comment
1881 – 1961	Geological Regional mapping	Various geologists, Identified the Aynak Project Area around site of historical mining and processing dumps
1973	Geological Survey Report 1:100,000 Scale	Conducted by Afghan and Soviet geologists.
1974 – 1977	Preliminary Resource Delineation Work 66,480m of Diamond Drilling 3,796m of Underground Development 20,677m^3 of Trench/Pit excavation and sampling 23,498 assay samples	Most samples assayed by AAS in Russia for Cu, Ni, Co, Mo, Pb, Zn, Ag, AS. Sulphide and Oxide Cu was determined in most samples within the oxidation and mixed zone.
	Assay External Controls 1976-1978	1,111 sulphide ore and 313 oxide ore samples collected. Average relative error <5%.
	289 density and moisture samples	Taken in various ore types, Oxide/Mixed Ores, Chalcopyrite Ore and Bornite Ore
	Multiple ore and petrological samples	
	Geophysics survey : 1:2000 scale IP	Poor result
	Topographical Surveys	1:10,000 Scale over broader 196 sqkm exploration area, 1:5,000 scale near mine area and 1:500 in mine area.
	Hydrogeology work	1:10,000 scale survey including some drilling in mine area
1978	Preliminary Resource Calculation	RGRCB
1978 – 1984	Detailed Resource Delineation Work 64,508m of Diamond Drilling 6,194m^3 of Trench/Pit excavation and sampling	
	39,412 assay samples	Most samples assayed by AAS in Russia for Cu, Ni, Co, Mo, Pb, Zn, Ag, AS. Sulphide and Oxide Cu was determined in most samples within the oxidation and mixed zone.
	Assay External Controls 1976-1978	>1,500 internal and external samples collected. Average relative error <5%.
	187 density and moisture samples	Taken in various ore types, Oxide/Mixed Ores, Chalcopyrite Ore and Bornite Ore
	Multiple ore and petrological samples Downhole Geophysics, Gamma, IP	
1987	Preliminary Resource Calculation	RGRCB
2008	China ENFI Co Feasibility Report	

Source: 1978 and 1987 Russian Resource Reports

4.5 GEOLOGY

4.5.1 Regional Geology

Afghanistan sits astride the collision zone of the Indo-Pakistan and Asian crustal plates, which has given rise to the Himalayas. It has a very complex geological history, with a number of small blocks or 'terranes' which split off around 250 million years ago from the margin of the Gondwana supercontinent. These terranes then successively accreted on to the southern margin of the Asian continent.

The Kabul Block is interpreted to be a one of these fragments and is bound by two major faults, on the west by the Pagman Fault and on the east by the Altimur Fault.

4.5.2 Local Geology

Lithology and Structure

The structure at Aynak is dominated by the Aynak anticline. The anticline is asymmetrical and approximately 4 km in length and up to 2.5 km wide. The south-eastern limb dips gently to the south-east but the north-western limb is steeply dipping and, in places overturned, with dips of 45 – 70° to the south-east. The periclinal closure of the anticline at its western end is asymmetrical. Here, the southern limb is overturned and the axial plane is inclined towards the north-north-east. Several sets of later faults cut across the folds.

The oldest rocks exposed in the area belong to the metavolcanic Welayati Formation, composed of gneiss and amphibolites, and are exposed in the core of the anticline ***Figure 4.2***. This formation is overlain by the thick metasedimentary sequence of the Loy Khwar Formation, which is a cyclical sequence of dolomite marble, carbonaceous quartz schist and quartz-biotite-dolomite schist and hosts the copper mineralisation ***Figure 4.2***. Seven members were originally defined during the first phase of exploratory drilling within the Middle District. The Loy Khwar Formation is post-dated by basaltic to dacitic metavolcanic rocks of the Gulkhamid Formation, which are also of Vendian – Cambrian age. As a result of folding, the copper deposit is divided into two prospects, with the Middle District located on the shallow-dipping eastern limb of the anticline and Western District occurring in the area of the periclinal closure at the western end of the structure ***Figure 4.2***.

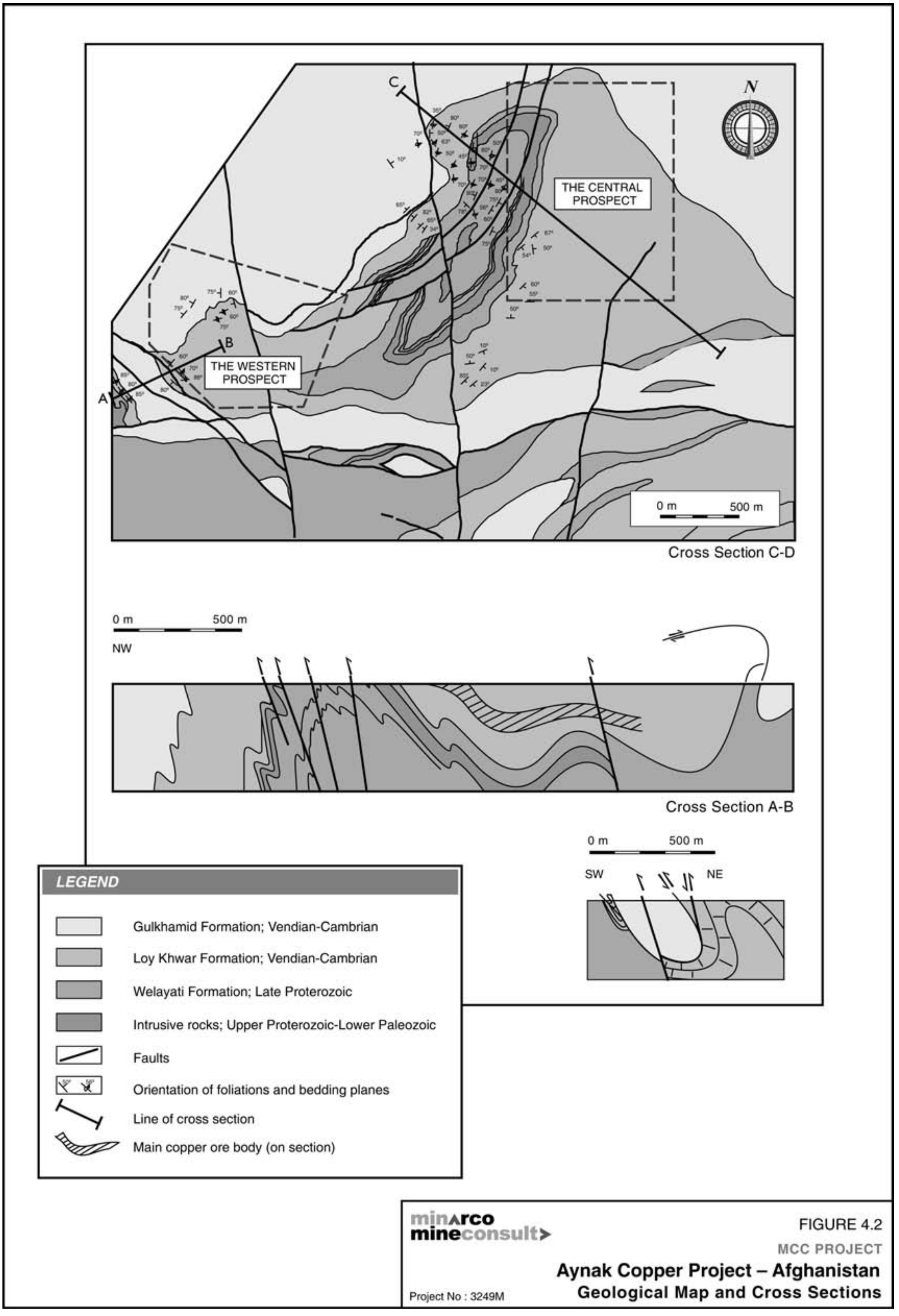
Mineralisation

The copper mineralisation at Aynak is stratabound and characterised by bornite and chalcopyrite disseminated in dolomite marble and quartz-biotite-dolomite schists of the Loy Khwar Formation. The mineralisation is mainly concentrated in the dolomite marble members of the Formation. The main zone of mineralisation in the Middle District is dominated by bornite. Chalcopyrite occurs in only minor amounts in the middle and lower parts of the orebody, but increases in the upper parts. Cobalt concentrations are very low but, like zinc, increase peripherally in some parts of the deposit. The depth of the oxidised zone is variable with the deepest oxidation occurring 250 m below the surface in the northern part of Middle District, beneath thick Neogene deposits.

The oxidised zone, with chalcocite and native copper, passes downwards into a mixed zone of oxidised and primary sulphides. No evidence for a supergene-enriched zone occurs.

Original resource estimations carried out by Soviet geologists, at varying cut-off grades, delineated several large ore bodies and a number of smaller lenses. At a 0.4% Cu cutoff grade, the Middle District orebody extends 1,850m along strike and 1,200m down dip and has a maximum thickness of 210 m. In the West District the main body extends 2,230m along strike and 1,640m down dip, and has a maximum thickness of 214 m, based on a similar cut-off.

Figure 4.2 — Aynak Copper Project — Geology Map and Cross Sections



4.6 RESOURCES AND RESERVES

M-MC has reviewed both the 1978 and 1987 Russian Resource reports as well as the validation work carried out by China ENFI Engineering Corp for their 2008 report. The methodology applied in the Russian estimates is in line with the resource reporting guidelines of Russia with the ore bearingness (polygonal) methodology based on sectional interpretations used in both reports.

MCC through ENFI carried out a detailed due diligence leading to the compilation of the 2008 Feasibility study. As part of this due diligence ENFI attempted to create digital databases for all drill holes contained within the Russian resources. However original drill data records have been lost during the war. Where the original data was unavailable ENFI digitised the information from the geological sections. This will result in inaccuracy in the hole location and subsequent lode location.

M-MC was unable to review the actual drill data and resource sections from the Russian estimates. For this reason M-MC can only make general comments based on the detailed resource reports and its experience with Russian estimates. M-MC, based on the reports presented considers the methodology employed to calculate the Russian resources reasonable. Extensive internal and external controls were undertaken of all assaying. Results from these controls are in line with the requirements of the Russian resource reporting guidelines and indicate a good level of precision and accuracy throughout the resource drilling and assaying program. Based on the data and results as presented, M-MC considers the estimates to be reasonable representations of the global tonnes and metal contents of the deposit. Polygonal estimates in their nature are poor estimators of local grade variability and these should be updated prior to undertaking any mine planning work.

M-MC reviewed the models created by ENFI for their due diligence. The result from this work appears reasonable and on the whole reflects the available underlying drill data. Both the West District and Middle District estimates correlates reasonably well with the Russian estimates.

Due to the loss of original data and lack of backing documentation such as resource calculation plans and models the current estimates do not meet the recommendations of the JORC code. Whilst the estimates appear reasonable M-MC has referred to the resources and reserves for Aynak as In Situ Quantities and Mineable Quantities accordingly.

A total of 150 holes have been completed over the Middle District, delineating the resource with a drillhole spacing of 100-150m by 100-150m. 170 holes have been drilled over the West District with uneven drill spacing of 100m by 100m in parts and opening out to 200m by 400m and greater in parts. This results in lower confidence in the interpretation and estimate of the West District. In total 130,498m of drilling has been carried out over the 2 mining districts using a combination of vertical and angled holes.

The drill holes were surveyed (down hole) to locate the drill-hole data in three-dimensional space. The drill-hole cores were logged and recorded digitally for lithology and structure using a consistent lithological code and format. The data was stored for manipulation using industrial software Techbase® software. An industry standard core-logging procedure was followed to log the geotechnical information of the diamond-drill cores.

Exploration methods are summarised in ***Table 4.3.***

In M-MC's opinion, exploration methods were appropriate for the style of mineralisation and sampling and assay techniques are reliable.

Table 4.3 — Aynak Copper Project — Exploration Methods — Summary

Exploration Methods	Purpose	Comments
Aerial Photography	Structural mapping	1:44,000 scale was later enlarged to 1:10,000 scale
Geological Mapping	Geology	
Geophysical mapping	Physical properties of the mineralisation quantified	Loop E.M, gradient array IP,
Soil Sampling Mobile Metal Ion (MMI)	To test possible MMI response	62 MMI samples, No obvious anomaly in the prospect, but comparatively higher MMI response for multiple metals indicating faults controlling mineralisation
Elemental concentration	Zn, Pb and Cu analyzed	62 samples analyzed; strong correlation with MMI data
Diamond Drilling	Resource definition	46,426 m drilling, poorly designed grids
Sampling	Geological logging and assaying	Good core recovery >98%, poor spatial mapping of some drill holes Down-hole surveys carried out at 50m Increments Database in good order, some issues with un-sampled intervals. 0.5-1m sample length based on geology.
Geochemical analysis	Cu, Pb, Zn, Ag, Ba and Fe analyzed	AAS technique was used with sulphide and oxide Cu being determined in most samples within the oxidation and mixed zone.
Duplicates and assay checks	Cu, Pb, Zn, Ag, Ba and Fe analyzed	AAS, XRF, ICP-OES techniques used, satisfactory correlation was found
Specific Gravity analysis	Ore Density	Diamond core samples. No bulk sampling reported
Geotechnical	Material characteristics	Poor sampling due to lack of training in the first phase, later revised.

Source: 1978 and 1987 Russian Resource Reports

4.6.1 Mineral Resources — In Situ Quantities

The Aynak deposit is a stratiform sulphide deposit with high grade Cu. Mineral Resource estimates for this project have been carried out in 2 main phases as drilling and understanding progressed from 1978 to 1987. The 1978 estimate was carried out by SUGP and centered on the more advanced Middle District. The 1987 estimate was completed by RGRCB over the West District. Both resource estimates were carried out according to the Soviet Union standard for reporting of resources and in MCC's opinion are reasonable. These estimates are summarised in ***Table 4.4.***

Table 4.4 — Aynak Copper Project — Russian In Situ Quantities (various Cu % cog)

Area			Middle District			West District			Total		
Ore Type	Russian Classification	JORC Equivalent	Tonne (Mt)	Cu% grade	Cu Metal Tonnes (000's)	Tonne (Mt)	Cu% grade	Cu Metal Tonnes (000's)	Tonne (Mt)	Cu% grade	Cu Metal Tonnes (000's)
Oxide											
(cog >0.5% Cu)	C_2	**Inferred**	—	—	—	2.5	1.22	29.1	2.5	1.22	29.1
	Sub Total Oxide		—	—	—	**2.5**	**1.22**	**29.1**	**2.5**	**1.22**	**29.1**
Mixed											
(cog >0.7% Cu)	C_1	**Indicated**	7	0.54	178.8	—	—	—	7	0.54	178.8
	C_2	**Inferred**	0.2	2.43	5.8	6.7	1.2	80.7	6.9	1.24	86.5
	Sub Total Mixed		**7.3**	**2.53**	**184.6**	**6.7**	**1.2**	**80.7**	**14**	**1.89**	**265.3**
Sulphide (cog >0.4% Cu)	**B**		36.3	2.77	1,006.5				36.3	2.77	1,006.5
	C_1	**Indicated**	112.8	2.33	2,628.8	87.4	1.61	1,406.9	200.2	2.02	4,035.7
	C_2	**Inferred**	28.6	1.96	561.1	201.7	1.51	3,041.2	230.3	1.57	3,602.3
	Sub Total Sulphide		**177.8**	**2.36**	**4,196.4**	**289.2**	**1.54**	**4,448.1**	**466.9**	**1.85**	**8,644.5**
Total	**B**		**36.3**	**2.77**	**1,006.5**				**36.3**	**2.77**	**1,006.5**
	C_1	**Indicated**	**119.9**	**2.34**	**2,807.6**	**87.4**	**1.61**	**1,406.9**	**207.3**	**2.03**	**4,214.5**
	C_2	**Inferred**	**28.9**	**1.96**	**566.9**	**210.9**	**1.49**	**3,151**	**239.8**	**1.55**	**3,717.9**
		Total	**185.1**	**2.37**	**4,381**	**298.3**	**1.53**	**4,557.9**	**483.4**	**1.85**	**8,938.9**

Source: 2008 ENFI Feasibility Study

Notes: Middle District only includes economic In Situ Quantities within the pit area. Sub economic In Situ Quantities have not been reported as In Situ Quantities, as they do not meet the basic requirement for eventual economic extraction under JORC.

MCC through China ENFI Engineering Corp carried out a detailed due diligence leading to the compilation of the 2008 Feasibility study. As part of this due diligence ENFI attempted to create digital databases for all drill holes contained within the Russian resources. Where the original data was unavailable ENFI digitised the information from the geological sections. This will result in inaccuracy in the drill hole location and subsequent lode location. Based on this data, ENFI estimated resources for both the Middle and West Districts using Datamine software.

M-MC makes the following general comments regarding the ENFI estimates.

- The geological interpretation is consistent with the logging and understanding of the deposit.
- Resource wireframes have been constructed in accordance with a >0.4% Cu Cut Off grade and honour the geology. Wireframes not being snapped to drill holes has lead to slight misappropriation of grades inside the composite files.
- Resource model was estimated using IDW^3 methodology and orientated ellipsoidal searches, using 2m drillhole composite data from 4 main mineralisation domains in the Middle District and 1 in the West District. Some of the ellipses in the West District appear to be poorly orientated and cut across the mineralisation. Resource grade of the Middle District is higher than the underlying assay data by approximately 15%. Whilst this is due in some parts to the variable drill data spacing it should be checked further, prior to these models being employed for any optimisation.
- Bulk density was applied based on oxidation with 2.55t/m^3 used for Oxide and Secondary ores and 2.8t/m^3 used for Fresh ore. The underlying density information was not reviewed by M-MC and cannot be verified. Based on the mineralisation style and known geology M-MC considers the bulk density applied to the resource to be appropriate.

- Resource Classification has been applied by ENFI based on drillhole spacing. Areas drilled to less than 100m being assigned Indicated Mineral Resource Category. Areas with greater than 100m but less than 200m drill spacing have being assigned Inferred Mineral Resource Category. Remaining areas are coded as Mineral Potential. M-MC considers this classification method appropriate for the style of mineralisation however due to the poor quality of 2D digitised drill data the estimates do not meet the recommendations of the JORC code.
- The West District estimate provided to M-MC at depth has been truncated above the depth of the interpreted mineralisation. This appears to be an exporting problem in datamine and resulted in M-MC reporting a slightly lower tonnage than reported in the ENFI 2008 Feasibility Study.

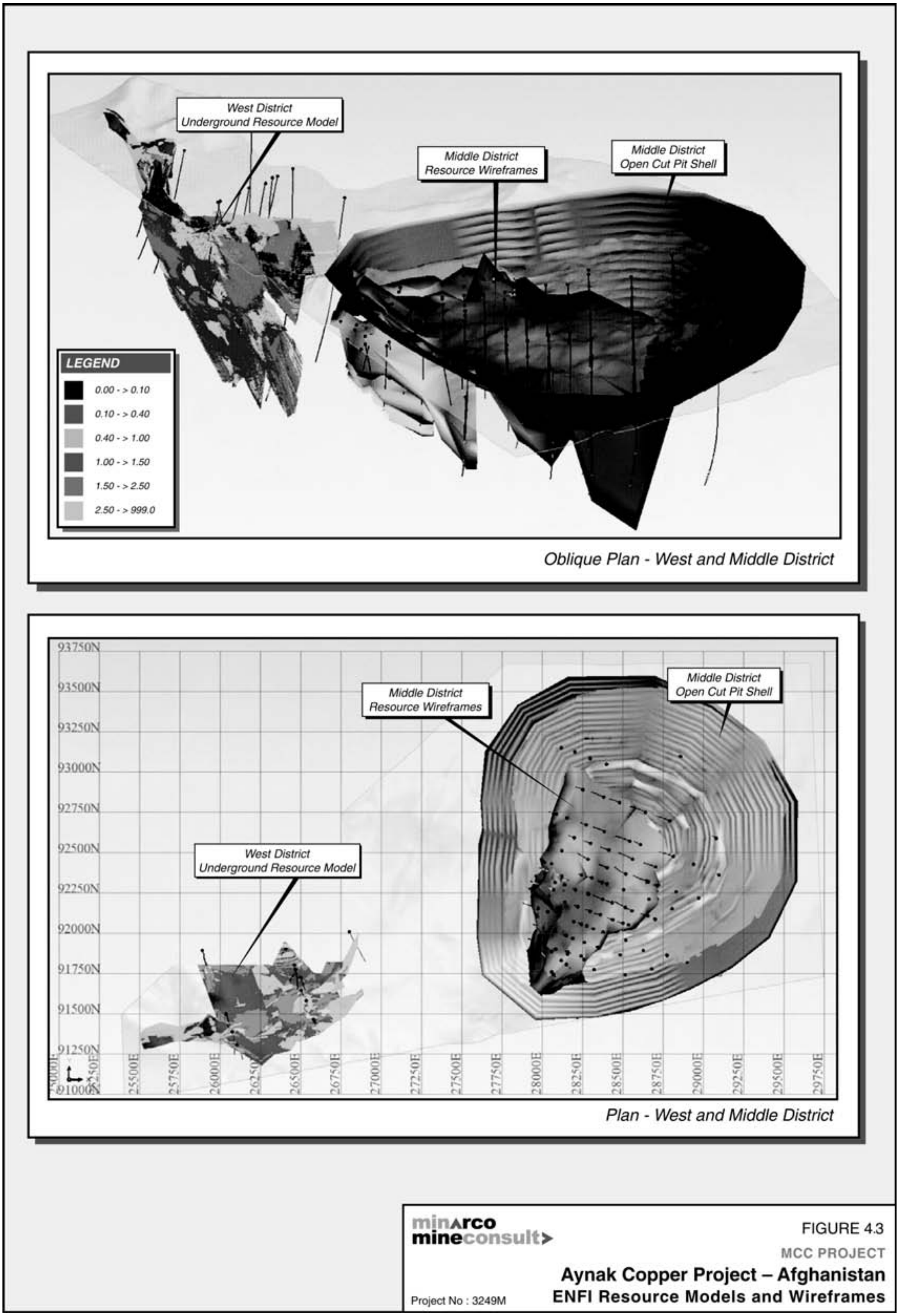
A summary of the ENFI estimates using similar Cu cutoff grades as in the Russian estimates is shown in ***Table 4.5***. The table for the Middle District only shows the In Situ Quantities within the currently designed pit shell. A further 140.9Mt at 1.28% Cu for 1,810kt of Cu Metal occurs outside of the current pit shell. This forms the basis for the planned Middle District Underground Mine which is planned to be exploited at the completion of the open cut.

Table 4.5 — Aynak Copper Project — ENFI 2008 In Situ Quantities (various Cu % cog)

Area		Middle District			West District			Total		
Ore Type	JORC Equivalent	Tonne (Mt)	Cu% grade	Cu Metal Tonnes (000's)	Tonne (Mt)	Cu% grade	Cu Metal Tonnes (000's)	Tonne (Mt)	Cu% grade	Cu Metal Tonnes (000's)
Oxide	**Indicated**	16.4	1.72	282.5	5.3	1.27	68.1	21.8	1.61	350.6
	Inferred	2.3	1.34	30.7	4.6	1.31	60.6	6.9	1.32	91.3
Sub Total Oxide		**18.7**	**1.67**	**313.4**	**10.0**	**1.29**	**128.6**	**28.7**	**1.54**	**442.0**
Mixed	**Indicated**	17.1	2.12	362.4				17.1	2.12	362.4
	Inferred	1.3	1.74	23.1				1.3	1.74	23.1
Sub Total Mixed		**18.5**	**2.09**	**385.3**				**18.5**	**2.09**	**385.3**
Sulphide	**Measured**	36.8	2.81	1,035.5				36.8	2.81	1,035.5
	Indicated	93.0	2.04	1,899.7	123.7	1.59	1,966.1	216.7	1.78	3,865.9
(cog >0.4% Cu)	**Inferred**	24.4	1.58	386.7	129.3	1.44	1,858.6	153.7	1.46	2,245.4
Sub Total Sulphide		**154.3**	**2.15**	**3,321.9**	**252.9**	**1.51**	**3,824.8**	**407.2**	**1.76**	**7,146.7**
Total	**Measured**	**36.8**	**2.81**	**1,035.5**				**36.8**	**2.81**	**1,035.5**
	Indicated	**126.6**	**2.01**	**2,544.7**	**129.0**	**1.58**	**2,034.2**	**255.6**	**1.79**	**4,578.9**
(various cog)	**Inferred**	**28.1**	**1.57**	**440.6**	**133.9**	**1.43**	**1,919.2**	**161.9**	**1.46**	**2,359.8**
Grand Total		**191.5**	**2.10**	**4,020.6**	**262.9**	**1.50**	**3,953.4**	**454.4**	**1.75**	**7,974.0**

Source: 2008 ENFI Resource Models Reported by M-MC

Figure 4.3 — Aynak Copper Project — ENFI Resource Model and Wireframes West and Middle Districts



4.6.2 Reserves — Mineable Quantities

ENFI in its 2008 Feasibility Study estimated reserves for both the Middle and Western Districts. The parameters used in these estimates are outlined in ***Table 4.6*** and on the whole appear reasonable. M-MC suggests that detailed mine design is required to assist in determining ore loss and dilution rates especially in the proposed underground mines. 100% recovery is rarely achieved in open cut mining and a 3-5% ore loss factor would be more reasonable.

The ENFI estimated Mineable Quantities are summarised in ***Table 4.7***.

Due to the lack of detailed mine design and planning combined with the resource issues discussed above M-MC considers that these reserves do not meet the recommendations of the JORC Code and hence refers to these as Mineable Quantities.

Table 4.6 — Aynak Copper Project — ENFI 2008 Mineable Quantities Parameters

	Middle District	West District
Ore Type	Sulphide and Mixed In Situ Quantities Only	
	In Situ Quantities Conversion	
Measured	100%	100%
Indicated	100%	100%
Inferred	50%	50%
Mining Method	Open Cut	Underground Sub Level Caving
Dilution	15%	15%
Ore Recovery	100%	85%

Source: ENFI 2008 Feasibility Study Report

Table 4.7 — Aynak Copper Project — ENFI 2008 Estimated Mineable Quantities

	Middle District	West District	Total
Mineable Quantities (Mt)	155.4	194.1	**349.5**
Grade Cu%	1.13	1.30	**1.22**
Cu Metal (kt)	1,751.4	2,525.8	**4,277.2**

Source: ENFI 2008 Feasibility Study Report

The oxide portion of the Middle District open cut is planned to be processed through a heap leach operation. M-MC using the same In Situ Quantities conversion parameters as used by ENFI in 2008 and a 15% dilution and 95% recovery factor estimates that the recoverable oxide In Situ Quantities is in the order of 16.7Mt at 1.44%Cu for 240kt of Cu metal.

It is also planned to mine the deeper portions of the Middle District resource below the currently designed open cut, once open cut operations are complete. Due to the preliminary nature of the study on this option no Mineable Quantities have been determined to date.

4.7 MINING

4.7.1 General Description

Mining of the Aynak Cu deposit will be through the use of a combination of both open cut and underground mining methods. The current plan for the Middle District is to develop a large open cut to the 2,070mRL followed by an underground sub level caving operation to the 1,680mRL. The West District Underground Mine will be developed from surface down to approximately the 1,600mRL. The lower extents of the underground mines are not definitive limits at this early stage of planning due to the current base of geological information.

The Middle District is planned to be initially mined using open cut methods, due to the relatively shallow dip (35-40 degrees) and the large width (~210m) of the ore body. The open cut will be mined using a typical truck and shovel operation utilising 6 Komatsu PC-5500 electric shovels and 28 Hitachi EH4000, 218t electric trucks. The ore and waste will be blasted separately using emulsion explosives. Blast holes will be drilled using one of 6 Ingersoll-Rand rotary drills to a depth of 17.5m, with a hole diameter of 250mm on a 10m by 8m spacing and burden respectively. The ore and waste will be mined on 15m benches using face shovels with a 26m^3 bucket capacity. It is apparent from the 2008 Feasibility Report that the large equipment is adequately supported by the planned numbers of dozers, graders, water trucks and smaller excavators.

Waste will be placed in waste dumps adjacent to the pit while ore will be transported to a crushing facility adjacent to the pit. The ore will be crushed using an in-pit crusher and transported via a 2.5km long conveyor belt to the processing facility located at the base of the mountain. Ore recoveries of 100% and an ore dilution of 15% were reported in the 2008 Feasibility Report for mining of the open pit. M-MC considers the dilution reasonable based on the dip of the orebody and mining bench height. Ore recoveries in these types of operations are usually in the order of 95-97%.

Based on the parameters in ***Table 4.8***, it is M-MC's opinion that there is sufficient excavator capacity to achieve the target production.

Table 4.8 — Aynak Copper Project — M-MC Estimated Shovel Production Rates Per Annum

Excavator Type	PC5500 - Waste	PC5500 - Ore
Bucket Size (cu. m)	26.00	26.00
In Situ Density (t/cu.m0	2.60	2.80
Swell Factor	1.50	1.50
Fill Factor	0.80	0.80
Number of Passes (actual)	6.00	5.00
Load time per Pass (Sec)	40	40
Total Working Days	354	354
Mechanical Availability	87%	87%
Operating Availability	86%	86%
Utilisation	77%	77%
Overall Utilisation	57%	57%
Effective Annual Work Hours	4,870	4,870
Production (tpa)	**12,537,460**	**13,370,886**

Source: M-MC Estimated

According to the 2008 Feasibility Study, a maximum number of 28 Hitachi EH4000 trucks will be required for waste and ore. According to M-MC's calculation based on the assumed parameters presented in ***Table 4.9***, approximately 38 EH4000 trucks will be required. This assumes a weighted average mining level of 2100mRL for all waste and ore and a progressive dump height of 64m.

Table 4.9 — Aynak Copper Project — Typical Haulage Profile

Parameter	Value
Assumed Weighted Average mRL	2,100
Pit Crest mRL	2,400
Ore Crusher mRL	2,420
Assumed Dump mRL	2,364
Pit Crest to Ore Crusher	500m
Crest to Dump Toe and Traversing	800m
Ramp Grade (Average)	8%
Waste Trucks Required	31
Ore Trucks Required	7

Source: M-MC Estimated

The underground mining operations have been planned to use a combination of natural block caving and sub-level caving mining methods. The Middle District has greater geology definition than that of the West District and for this reason the exact mining method of the underground mine planned for the West District has not been clarified. Assumptions have been made, by the client that the geology will be similar and as such a similar mining method could be employed.

Mining methods planned have been based on the ore body characteristics at different elevations. It has been determined by the client that mining above the 1,800mRL will utilise the natural block caving mining method, whilst mining below the 1,800mRL will use the sub-level caving mining method, due to the narrowing width of the ore body and higher rock strengths. Drilling of 76mm diameter blast holes in the sub-level caving zone will be done with either a Solo 7-7V or Simba H1354 longhole drill rig.

A combination of electric and diesel Toro load haul devices (scrapers) will be used to transport the ore from the caving draw point to a system or ore chutes (ore passes). The ore passes connect the upper levels of the mine to crushing stations located at the 1,680mRL and the 1,740mRL in the west and middle underground mines respectively. From the crushing stations, the ore will transported to the surface via the respective main shaft hoisting systems.

The Middle District Underground Mine will be accessed via a ramp from the planned open cut, a vertical shaft (central main shaft) from the 2,372mRL to the 1,620mRL and a vertical auxiliary shaft. The main shaft is planned to be 8.2m in diameter and will be used for hoisting of ore and waste using four 23m^3 skips with a capacity of 27,500tpd (25,000tpd ore and 2,500tpd waste). Two 5.5m diameter vertical ventilation shafts, between 206m and 360m in depth, will be sunk at either end of the ore body to provide adequate ventilation to the underground workings.

The West District Underground Mine will be accessed via a ramp from the 2,280mRL to the 2,000mRL, a vertical shaft (western main shaft) from the 2,378mRL to the 1,560mRL and a vertical auxiliary shaft. The main shaft is planned to be 8.7m in diameter and will be used for the hoisting of ore and waste using four 30m^3 skips with a capacity of 33,000tpd (30,000tpd ore and 3,000tpd waste). Two 7.8m diameter vertical ventilation shafts, between 360m and 430m in depth, will be sunk at either end of the ore body to provide adequate ventilation to the underground workings.

The auxiliary shaft, which services both underground mines, will be used as the main method of transporting materials and equipment to the underground workings. The shaft is planned to be 8.2m in diameter, with elevation ranging from the 2,382mRL to the 1,565mRL. Various levels will be developed to connect the two underground mines with each other and the auxiliary shaft.

Mining operations in both the open pit and the underground operations have been planned around three 8 hour shifts per day over 330 days per year.

Figure 4.4 — Aynak Copper Project — General Site Layout Plan

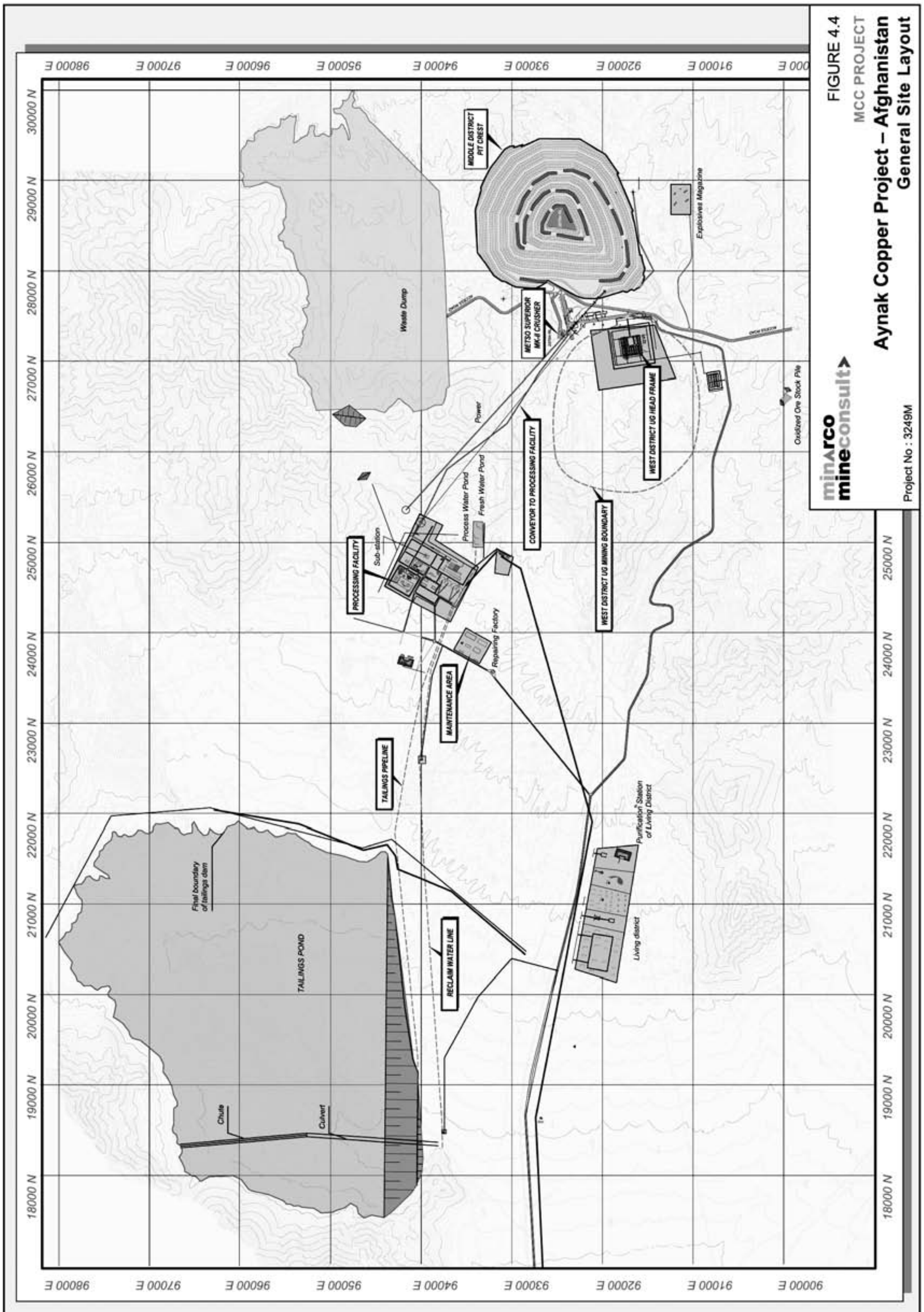
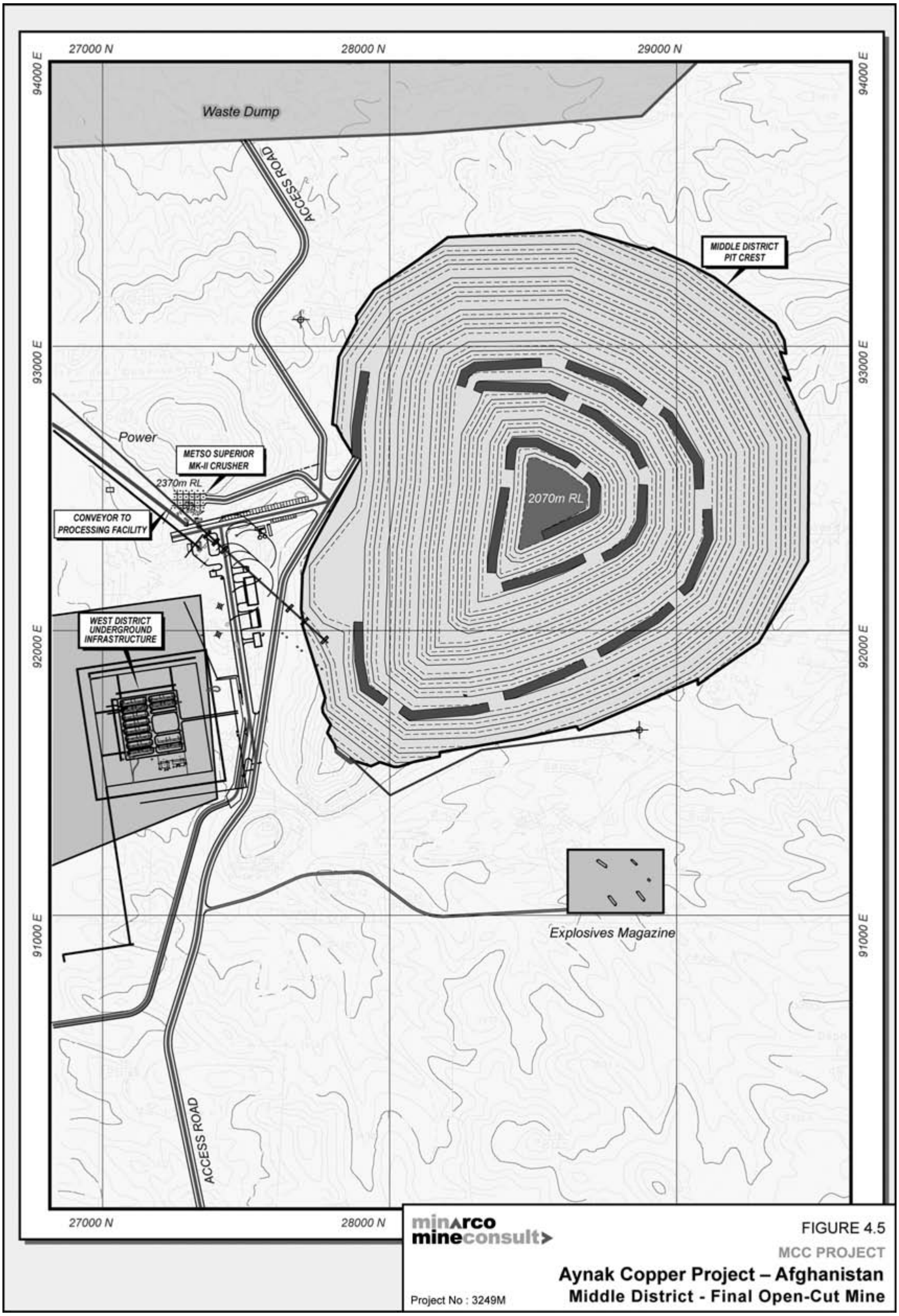


Figure 4.5 — Aynak Copper Project — Middle Open Cut Design



4.7.2 Forecast Production

M-MC held a meeting with ENFI in Beijing on the 16th March 2009 in order to gather more detailed information about the project. It was indicated during the meeting, by ENFI personnel, that civil construction work was due to commence in early 2010. Civil construction work is planned to be 2.5 years in duration, with production from the Middle District open pit commencing in mid 2012. At the planned production rate of 30,000tpd from the open pit, the estimated mine life of the open pit is 16 years according to the Revised 2008 Feasibility Report.

The planned annual production and intended ramp up period for both the open pit and underground mines are shown ***Table 4.10***. As described in the 2008 Feasibility Report, target production from the open pit is not reached until the second year of production and the target production from the Western Mining Area underground mine is not reached until the fifth year of production. At the time of this review, M-MC was not provided with any detailed underground mine designs. Only a review of the data contained within the 2008 Feasibility Report was possible.

Based on the information reviewed by M-MC the production forecast estimates provided in the 2008 Feasibility Report, appear to be reasonable based on the capital expenditure planned, equipment quantity and size and ore body characteristics.

Table 4.10 — Aynak Copper Project — Forecast Mining Production

Production	Type	2012	2013	2014	2015	2016	2017	2018
Middle Open Cut (Mtpa)	Ore	6.93	9.90	9.90	9.90	9.90	9.90	9.90
	Waste	29.7	42.5	61.8	61.8	61.8	61.8	61.8
West Underground (Mtpa)	Ore	—	—	1.70	5.26	6.81	8.35	9.90
Total Ore	**Ore**	**6.93**	**9.90**	**11.60**	**15.16**	**16.71**	**18.25**	**19.80**

Source: 2008 ENFI Feasibility Study.

As can be seen in ***Table 4.11***, the mining production rate contained in the latest data supplied by ENFI does not match the data presented in ***Table 4.10*** and appears optimistic. It is M-MC's opinion that the production rates presented in ***Table 4.10*** are more achievable. Moreover, the process data is not based on any testwork and has been assumed using the performance of an operation based in Zambia.

The use of a constant copper feed grade being supplied to the processing facility lacks credibility, since the underground copper grades are less than 2% and typically 1.6% Cu. Moreover, the mineralogy changes significantly (refer to the Processing Section) with the introduction of underground ores for processing. Consequently, after year 3, M-MC would expect the feed grade to fall to around 1.8% with a concentrate grade of 28-36% and a recovery of 84-86% copper (refer to ***Table 4.12***). Additionally, based on these figures, the forecast copper production rates will contain 320kt of copper rather than the forecast 395kt.

Table 4.11 — Aynak Copper Project — Forecast Production

Parameter	Unit	2011	2012	2013	2014	2015	2016
ROM production	Mt	8.05	9.90	9.90	19.80	19.80	19.80
Copper feed grade	(%)	2.19	2.19	2.19	2.19	2.19	2.19
Concentrate production	t	401,254	493,468	493,468	986,936	986,936	986,936
Copper concentrate grade	(%)	40.0	40.0	40.0	40.0	40.0	40.0
Copper recovery	(%)	91.0	91.0	91.0	91.0	91.0	91.0
Concentrate price	USD/t	1,267	1,267	1,267	1,267	1,267	1,267

Source: Data provided by company as of March 2009.

4.8 MINERAL PROCESSING

The processing operation at Aynak is expected to consist of four operations developed in three phases. The first phase consists of a mineral processing stage where a copper concentrate is produced as well as a heap leaching operation to treat the oxidised ore. After year 5, a smelting facility will treat the concentrate on site and the blister copper will be refined to cathode copper in an electrowinning operation. This stage will require the construction of a sulphuric acid plant as well as a large base load power station. In later years, it is proposed to increase the smelting and electrowinning capacity to around 500ktpa.

The ore has few impurities (manganese is the major one at typically 0.2%) and limited quantity of credits (silver is around 5ppm while only 'sample 3-3' has a gold content (0.27g/t)).

The project is at a conceptual stage, with the mineral processing facility based on a Zambian operation. Representative samples are currently being sourced for comprehensive testing which will form the basis of the final design. Other studies will be conducted on the flotation concentrate products to develop the smelting and electrowinning process flowsheets.

General Overview of Processing

Previous testwork conducted by the Russians found that while reasonable copper recoveries could be obtained, they were at rather low copper concentrate grades. M-MC believes that considerably better concentrate grades are possible however at lower copper recoveries.

Details of the ore samples 3-1, 3-2 and 3-3 have been lost and the representative nature of these samples with regard to three ore bodies is consequently not known.

MCC believe that better metallurgy is possible and, until the testwork has been conducted, have modelled the mineral processing operation on the Zambian Chambishi operation. The characteristics of this operation are as follows: 2.43% Cu feed grade, 60% bornite and 25% chalcopyrite to produce a 40% copper concentrate grade at 95.5% copper recovery. The Zambian ore possibly has some similarities to some of the ores at Aynak, however, based on the Russian testwork, has a better process response.

Table 4.12 — Aynak Copper Project — Process Metallurgy

Parameter	Ore Sample 3-1	Ore Sample 3-2	Ore Sample 3-3
Feed Grade (% Cu)			
Reported	2.29	0.96	1.52
Sample	2.37	1.00	1.68
Chalcopyrite (%)	12.3	84.8	39.8
Bornite (%)	79.7	13.9	58.8
Concentrate Grade (% Cu)			
Theoretical Maximum	59	39	52
Probable*	40	28	36
Testwork (Russian)	32	16	22
Copper Recovery (% Cu)			
Testwork (Russian)	93	86	89
Probable*	90	84	86
Grind Size (% passing 74 microns)	70	70	70-80

Source: ENFI 2008 Feasibility Study
* *M-MC estimate*

While a theoretical study was conducted in order to compare milling circuit options, the performance of existing mills in Chinese operations has been used as the basis in the economic study.

The Aynak Mineral Processing Flowsheet is presented in ***Figure 4.6***. Ore is crushed at the mine and transported by conveyor belt to a stockpile. Ore is withdrawn from the stockpile (1,538tph) and feed to a secondary crusher in closed circuit with a screen. The screen undersize (1,250tph) is conveyed to the Fine Ore Storage bins which feed the milling circuit. The hydrocyclone overflow from the milling circuit reports to a Rougher-Scavenger flotation bank. The Rougher concentrate is upgraded in a three staged cleaning circuit to produce the final copper concentrate. The Scavenger concentrate is re-ground in a ball mill circuit, with the hydrocyclone overflow reporting to a Middlings Rougher flotation circuit. Both Middling Rougher concentrates are upgraded in one stage of cleaning and the cleaned product joins the Rougher concentrate as feed to the three stage cleaning circuit. The final concentrate represents 4.99% of the feed by mass (60.4tph) and contains 91% of the copper. The final tailings is the Rougher-Scavenger tailings (1,187.7tph) containing 9% of the copper that was present in the feed.

The final concentrate is dewatered in a 30m diameter high rate thickener followed by a filter press.

Final tailings consist of tailings from the primary scavenger and cleaner scavenger as well as the scavenger after re-grinding. The final tailings is dewatered in a thickener and transported to a tailings dam. Water from both the thickener and tailings dam would be recycled in the process.

The operation will require 14,850tpa of lime to depress the pyrite during flotation, 495tpa of butyl xanthate to recover the copper minerals, 594tpa of sodium sulphide to activate oxidised copper minerals for flotation recovery and 198tpa of frother and flocculant for thickening.

Heap Leach

The oxidised material from the open cut operation will be heap leached in an area located 2.5 km from the mine site. The heaps will be 30m high and cover an area of 51.6ha and consist of 9.3 million m^3 of material. Details of the operation were not available. Presumably, the heap leach would not be operated until the acid plant and electrowinning facilities have been constructed in year 5. The planned production is 20,000tpa of copper over 10 years.

Smelting and Electrowinning

A conventional smelting operation is envisaged employing a smelting stage (200,000tpa) to produce matte which is then blown through to blister copper in a converting stage. The blister copper is cast as anodes and refined into high purity cathode copper using electrowinning methods (220,000tpa). Smelter slag will be cleaned in an electric furnace while the convertor slag would be recycled to the smelting stage. An acid plant will be constructed to produce sulphuric acid for the electrowinning operation. Additional facilities include an oxygen plant, waste heat boiler and an ElectroStatic Precipitator (ESP) to capture dust from the stack emissions.

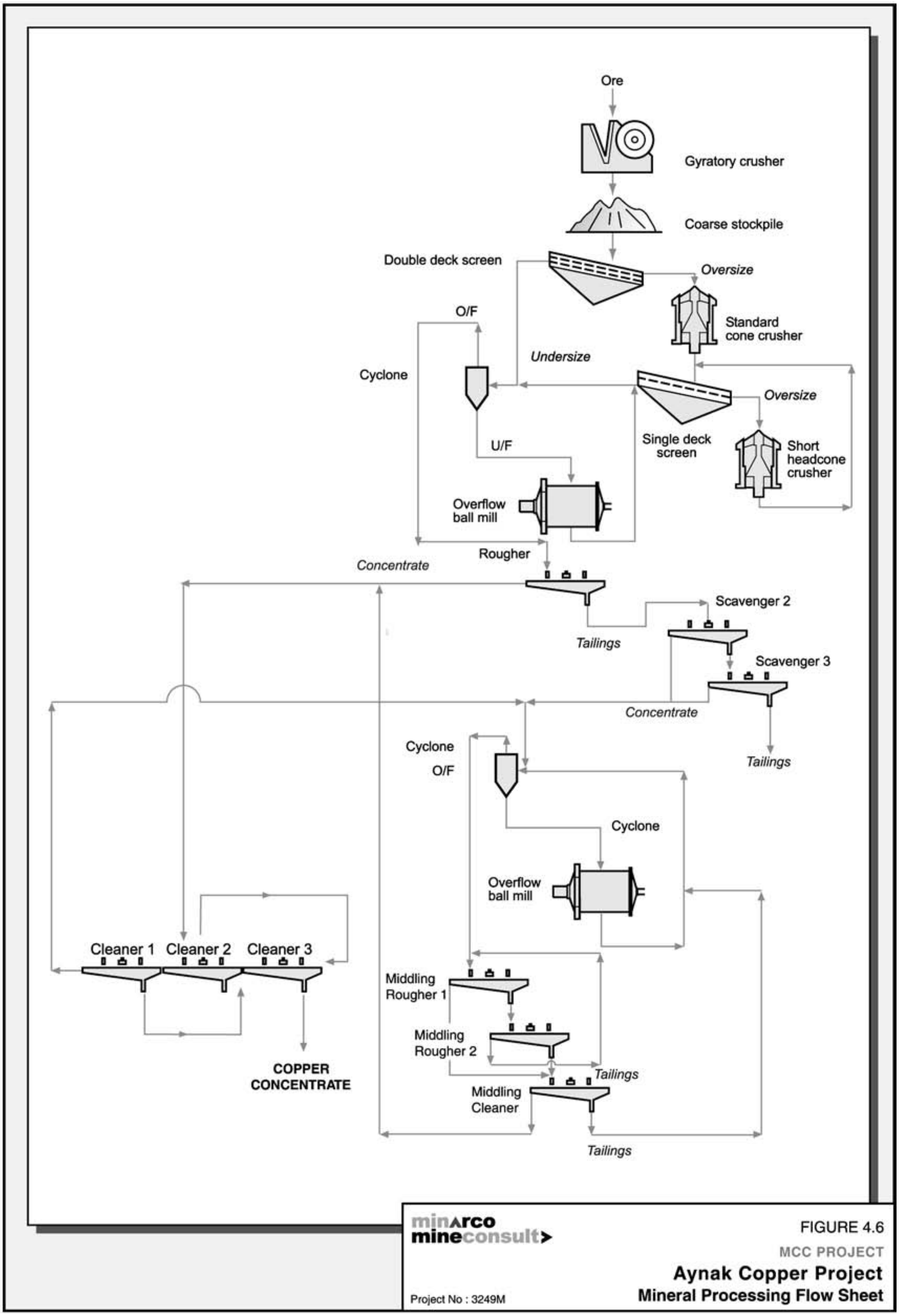
In the first ten years, 205,000tpa of sulphuric acid would be produced, rising to 305,000tpa in year 11.

M-MC recommends that the following items be addressed:

- The metallurgy of the ore types and various blends is unknown
- The selection of the milling circuit needs to based on Aynak ore types
- The selection of the flotation circuit needs to be based on Aynak ore types
- Heap leach testwork has not been undertaken
- Smelting testwork has not been conducted on Aynak copper concentrates
- Electrowinning testwork has not been conducted

Without an exact understanding of all the process requirements, it is difficult to correctly select the appropriate flowsheets and associated equipment and therefore accurate capital and operating costs cannot be developed.

Figure 4.6 — Aynak Copper Project — Processing Flowsheet



4.9 INFRASTRUCTURE AND SERVICES

Power will be supplied from diesel generators during the first five years of operation. The project power load upon completion of all the processing facilities is estimated to be 150MW. A joint venture coal fired power station (400MW capacity) near Tala is proposed using coal from northern Afghanistan. Electricity would be provided to Kabul and then to the processing facility via a 280km 220kV transmission line.

Water would be sourced from underground water reserves near the Luger River, where some 173ML/day is apparently available. This would appear to be more than sufficient to meet the processing, smelting and electrowinning needs. The water would be pumped 17 km by a three stage pumping station to a water treatment facility before being pumped 5km to the processing facilities. A small reverse osmosis plant will produce potable water (120kL/day). A separate reverse osmosis plant (480kL/day) will treat a part of the tailings return water for use in air conditioning and slag granulation.

Water consumption in the mining phases is estimated to be 1.6ML/day in the open cut and 6ML/day increasing to 11ML/day after year 10 for underground mining. The processing plant will need 39.3ML/day of fresh water, while power generation, smelting and electrowinning will need a total of 10.41ML/day.

It has been estimated that a total tailings dam capacity of 393 million m^3 will be required over the life of the operation. It is planned to use two locations, one located in a ravine in the northwest of the property and at a later date, a second tailings dam located in the south west. A rock fill construction method would be probably used.

While some basic climatic data has been reported, more detailed information will need to be gathered from the proposed site. It is noted that the area has a high seismic activity (earthquakes of up to 8 on the Richter scale). This will need to be addressed in the design requirements for the project and will increase the capital costs.

4.10 CAPITAL AND OPERATING COSTS

The Aynak Copper project is being developed in a region with limited infrastructure and without a significant mining tradition. Subsequently, considerable investment is required in infrastructure such as power, water and administration as well as the establishment of mining and processing facilities. The mining plan calls for the development of an open cut in the Middle District, followed by an underground development after year 5 in the West District and subsequently the development of another underground mining operation in later years in the Middle District.

On the processing side, after year 5 it is planned to develop a smelting and refining complex capable of producing 220,000tpa of cathode copper. At this stage, a coal fired power station will be constructed to service the requirements of the electrowinning plant as well as the processing plant.

This review only examines the first 6 years of operating and capital costs. ***Table 4.13*** presents the planned capital expenditure. In the Feasibility Study, it is stated that USD4.39 billion will be required to complete the project. The Stage 1 CAPEX will focus on the establishment of the infrastructure, open cut and processing plant. Stage 2 will concentrate on the development of the underground mine, concentrator, smelter and ancillary facilities.

Table 4.13 — Aynak Copper Project — Actual and Forecast Mining and Processing Capital Expenditure

Cost Centre	Unit	2008	2009	2010	2011	2012	2013	2014	2015	2016
Stage 1 Construction	M USD	117.6	500.8	723.5	323.5	—	—	—	—	—
Stage 1 Other	M USD	—	—	61.3	61.3	—	—	—	—	—
Stage 1 Sub total	M USD		**1,788**			—	—	—	—	—
Stage 2 Construction	M USD	—	—	—	—	27.0	360.0	1,137.0	405.0	55.0
Stage 2 Other	M USD	—	—	—	—	124.6	124.6	124.6	124.2	121.0
Stage 2 Sub total	M USD		**—**					**2,603**		

Source: Costs provided by company as of March 2009.

Forecast mining operating costs are shown in ***Table 4.14*** for the first six years. The dominant cost centre during the first four years is depreciation, varying initially from 69% of the total mining costs to 48% in years 4 and 5. Thereafter, it drops to 4% of costs. The mining costs are relatively high for a moderately high tonnage open cut operation. By year 6, even with the introduction of underground mining, the operating costs have decreased to a reasonable level. There is no escalation for inflation or allowance for potential increases in consumable costs such as fuel, tyres and capital spares as well as increases in salaries.

Table 4.14 — Aynak Copper Project — Forecast Mining Operating Costs

Cost Centre	Unit	2011	2012	2013	2014	2015	2016
Auxiliary Material	USD(000's)/a	31,395	38,759	38,759	85,270	85,270	85,270
Water & Power	USD(000's)/a	3,988	4,923	4,923	10,831	10,831	10,831
Labour	USD(000's)/a	2,151	2,655	2,655	5,841	5,841	5,841
Repair & Maintenance	USD(000's)/a	8,074	9,968	9,968	21,930	21,930	21,930
Depreciation	USD(000's)/a	124,866	124,866	124,866	124,866	124,866	5,364
Other	USD(000's)/a	9,938	12,269	12,269	12,269	12,269	12,269
Total	**USD/t ROM**	**22.41**	**19.54**	**19.54**	**13.18**	**13.18**	**7.15**

Source: Costs provided by company as of March 2009.

The forecast processing operating costs are shown in ***Table 4.15*** for the first six years. Like the mining operating costs, the dominant cost centre during the first four years is depreciation, varying initially from 48% of the total processing costs to 28% in years 4 and 5. Thereafter, it drops to 2% of costs. The operating costs are relatively high for a moderately high tonnage processing operation, which is mainly due to the depreciation charges. By year 6, the operating costs have decreased to a reasonable level. There is no escalation for inflation or allowance for potential increases in consumable costs such as fuel, reagents and capital spares as well as increases in salaries.

Table 4.15 — Aynak Copper Project — Forecast Processing Operating Costs

Cost Centre	Unit	2011	2012	2013	2014	2015	2016
Auxiliary Material	USD(000's)/a	14,650	18,086	18,086	36,172	36,172	36,172
Water & Power	USD(000's)/a	24,797	30,614	30,614	61,228	61,228	61,228
Labour	USD(000's)/a	1,425	1,759	1,759	3,518	3,518	3,518
Repair & Maintenance	USD(000's)/a	4,329	5,345	5,345	10,690	10,690	10,690
Depreciation	USD(000's)/a	44,471	44,471	44,471	44,471	44,471	1,910
Other	USD(000's)/a	3,588	4,430	4,430	4,430	4,430	4,430
Total	**USD/t ROM**	**11.59**	**10.58**	**10.58**	**8.11**	**8.11**	**5.96**

Source: Costs provided by company as of March 2009.

Projected operating costs for the smelter and electrowinning (or refinery) facilities are presented in ***Table 4.16***. Again, depreciation is the dominant operating cost, representing 63% of total operating costs in the first few years. This operating cost includes both smelting and electrowinning and may be reasonable at USD556/t of cathode product (or USD0.25/lb of copper). Current Treatment and Refining Costs (TRC) for copper concentrates are around USD70/t/USD0.70/lb, which, for this ore, equates to USD792/t of cathode copper or USD0.36/lb of copper. However, M-MC regards the actual operating costs, that is the costs before depreciation charges, to be too low at USD208/t of cathode copper or USD0.09/lb of copper.

Table 4.16 — Aynak Copper Project — Forecast Smelting and Refining Operating Costs

Cost Centre	Unit	2014	2015	2016
Auxiliary Material	USD(000's)/a	23,056	23,056	23,056
Water & Power	USD(000's)/a	29,448	29,448	29,448
Labour	USD(000's)/a	5,346	5,346	5,346
Repair & Maintenance	USD(000's)/a	21,670	21,670	21,670
Depreciation	USD(000's)/a	137,319	137,319	137,319
Others	USD(000's)/a	2,613	2,613	2,613
Total	**USD/t ROM**	**11.08**	**11.08**	**11.08**
	USD/lb Cu	**0.25**	**0.25**	**0.25**

Source: Costs provided by company as of March 2009.

Total projected operating costs for the whole operation are summarised in ***Table 4.17***. While the total operating costs are reasonable for the first few years, the major cost will be due to depreciation. M-MC considers that the underlying cost of production is too low, particularly when additional stages of processing are introduced in 2014, namely smelting and electrowinning. Although the ROM output is doubled, ROM material will be produced from predominately underground sources and the electrowinning operating costs are significant. M-MC would expect that the cash cost of producing copper to be around USD1.10/lb, depending upon the cost of fuel, coal and consumables. The concentrate costs would be associated with mainly transport and are estimate to vary between USD88 to 175/t of concentrate, which is reasonable.

Table 4.17 — Aynak Copper Project — Forecast Total Operating Costs

Cost Centre	Unit	2011	2012	2013	2014	2015	2016
Mining	USD(000's)/a	180,412	193,440	193,440	261,007	261,007	141,505
	USD/t ROM	**22.41**	**19.54**	**19.54**	**13.18**	**13.18**	**7.15**
Processing	USD(000's)/a	93,260	104,705	104,705	160,509	160,509	117,948
	USD/t ROM	**11.59**	**10.58**	**10.58**	**8.11**	**8.11**	**5.96**
Administration	USD(000's)/a	119,999	136,500	136,500	118,789	100,277	86,851
	USD/t ROM	**14.91**	**13.79**	**13.79**	**6.00**	**5.06**	**4.39**
Other Costs	USD(000's)/a	62,550	62,550	62,550	79,000	79,000	79,000
	USD/t ROM	**7.77**	**6.32**	**6.32**	**3.99**	**3.99**	**3.99**
Smelting and Refining	USD(000's)/a	—	—	—	219,452	219,452	219,452
	USD/t ROM	—	—	—	**11.08**	**11.08**	**11.08**

Cost Centre	Unit	2011	2012	2013	2014	2015	2016
Unidentified	USD(000's)/a	—	—	—	82,132	82,132	82,132
	USD/t ROM	—	—	—	**4.15**	**4.15**	**4.15**
Total Operating Cost	**USD(000's)/a**	**456,221**	**497,195**	**497,195**	**920,889**	**902,377**	**726,888**
Total	**USD/t ROM**	**56.67**	**50.22**	**50.22**	**46.51**	**45.57**	**36.71**
	USD/lb Cu	**1.29**	**1.14**	**1.14**	**1.06**	**1.04**	**0.84**
Taxation	USD(000's)/a	54,696	67,526	67,526	67,526	67,526	67,526
	USD/t ROM	**6.79**	**6.82**	**6.82**	**3.41**	**3.41**	**3.41**
Concentrate costs	USD(000's)/a	70,349	86,850	86,850	86,850	86,850	86,850
	USD/t ROM	**8.74**	**8.77**	**8.77**	**4.39**	**4.39**	**4.39**

Source: Costs provided by company as of March 2009.

4.11 SAFETY AND ENVIRONMENT

A Safety Programme is outlined in the Feasibility Study that addresses a number of issues, such as fire, site traffic, plant safety, hygiene matters, ventilation, noise and vibration as well as electromagnetic radiation. The programme details design, training and monitoring practices that would apply to each aspect of the operation. The programme is based safety requirements detailed under the "Mineral laws of Afghanistan (2005)".

In addition, the Feasibility Study outlines a Social Development plan for the local community. This encompasses education (USD1.2 million to be spent on building four schools), health (USD3.8 million to be used constructing a number of hospitals and clinics), infrastructure as well as sustainable development activities. These activities would support small business development such as agriculture. The operation would recruit local people, particularly experienced technical people. The programme also recognises the need for resettlement and compensation for affected communities.

The Feasibility Study outlines a detailed Environmental Policy that would address all the sources of potential pollution for the proposed operation. World Bank guide lines are used as the source of best practice. The sources of pollution include emissions from the open cut mining (dust), smelting operations and pulverised coal boiler (particulates, sulphur dioxide, nitric oxides and carbon dioxide), as well as waste solids and water streams and noise. Total expenditure for the protection of the environment is estimated to be USD59 million.

All furnace and stack emissions will be monitored by automated measuring systems (CEMS) and desulphurisation (CFB) technology will be employed to remove most of the sulphur dioxide (85% removal) not captured in the production of sulphuric acid from the convertor flue gases as well as other smelting fumes. Low nitrogen combustion technologies will be used to minimise the production of NOx gases in the pulverised coal boiler. Treated gas streams will be discharged into the atmosphere via a 180m high concrete chimney. The cost of environmental monitoring equipment is estimated at USD550,000.

The various water waste streams will be monitored and recycled where possible, otherwise treated where necessary before discharge into the environment. Granulated slag and ash would be buried in a waste dump site.

Noise control will be practiced within the processing facilities as well as the other operations and closely monitored.

The operation plans to employ a number of energy and water saving technologies to minimise the impact it has on the local environment.

Additionally, 'greening' would be practised whereby plants and trees would be planted around the site.

5 DUDDAR LEAD ZINC PROJECT

M-MC did not make a site inspection of this property due to perceived political instability in Pakistan. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "2005 Basic Design for Duddar Lead Zinc Mine, Pakistan" — including underlying resource models and drill data — Prepared by China Central Engineering Institute for Non-ferrous Metallurgical Industries (ENFI)
- "1997 Pasminco Exploration Report — Vol 1 - 8"

The Duddar Lead — Zinc (Pb-Zn) Mine is currently under construction. Planned ore production capacity from underground operations is 660ktpa.

MCC's equity stake in the Duddar Project is 40.8% (MCC holds 80% of the MCC Duddar Minerals Development Company (Pvt) Ltd., which holds 51% of the mine).

5.1 BACKGROUND

The Duddar Pb-Zn deposit (26° 05′ 32″ latitude, 66° 50′ 30″ longitude) is located in the northern end of Kanraj Valley, in the Balochistan Province of Pakistan approximately 135km to the NNW of Karachi, ***Figure 5.1.*** The Kanraj Valley is flanked by the Mor Range to the west and by the Pab Range to the east. Duddar is situated on the margin of the Mor Range.

The Duddar Deposit is a stratiform sulphide with high grade Pb and Zn. Mineralisation is hosted in a succession of Jurassic carbonate and clastic sediments.

5.2 ASSETS

The assets and status include;

- Mine development project, ready for production commencement.
- JORC Compliant Mineral Resources of 14.48Mt at 9.9% Zn, 3.4% Pb, 19.1g/t Ag, 7.8% Ba and 11.9% Fe (cog >7% Pb+Zn).
- Mineable Quantities of 9.13Mt at 9.3% Zn, 3.0% Pb (cog >7% Pb + Zn).
- A preliminary mining study "2005 Basic Design for Duddar Lead Zinc"
- Some site infrastructure is complete but currently on care and maintenance:
  - Completed construction of the processing plant at the end of 2008
  - Main shaft 50% completed as at April 2009 along with 2 underground development drifts

5.3 LAND TENURE AND MINERAL RIGHTS

Details of the "Mining Lease" for Pb and Zn are shown in ***Tables 5.1***. These are valid to 2024.

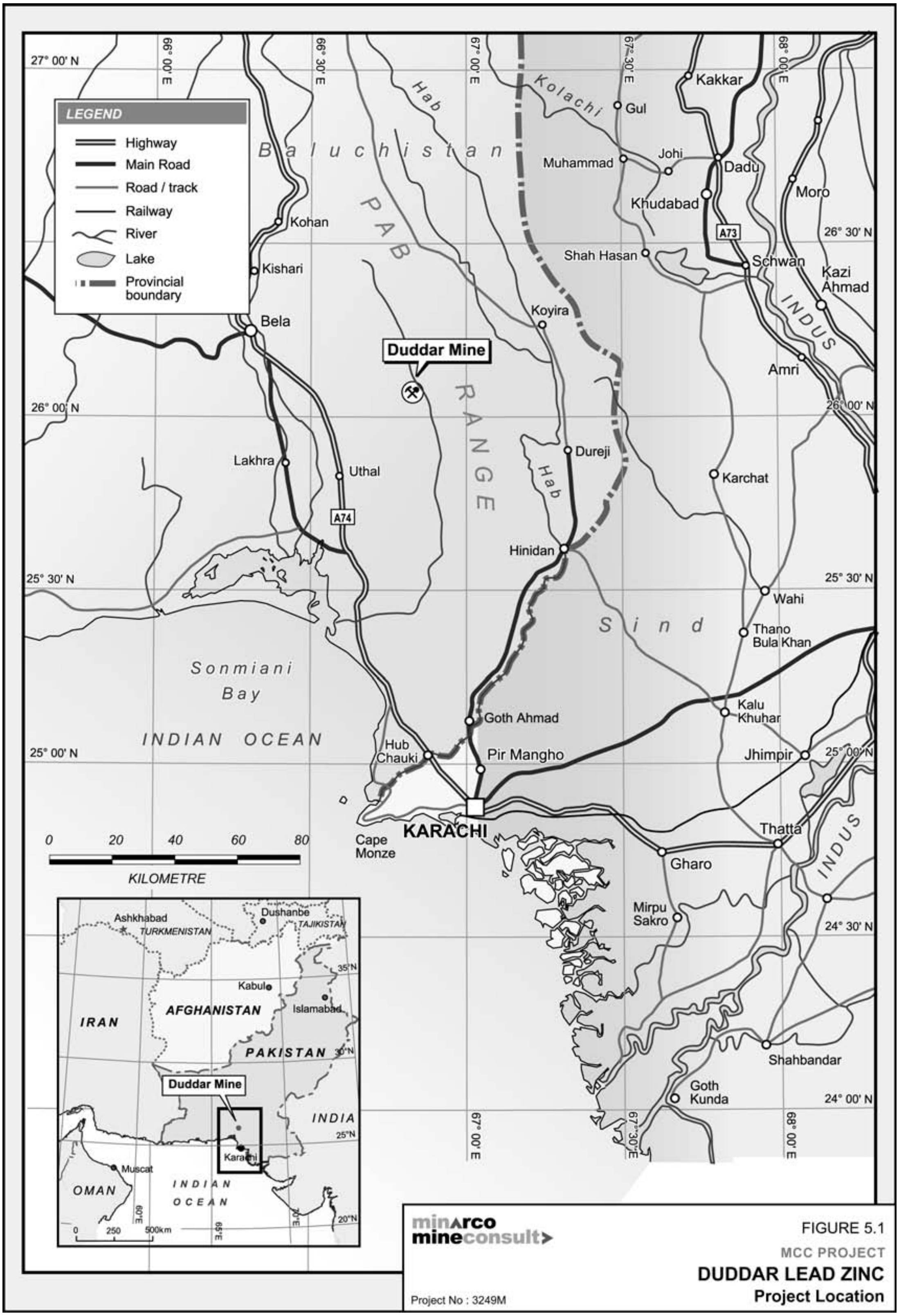
Table 5.1 — Duddar Lead Zinc — Mining Lease ML-100(132)

Mine/Project	Duddar
Title	Mining Lease
No	ML-100 (132)
Owner	MCC Duddar Minerals Development Company (Pvt) Ltd. Mining Lease for Lead/ Zinc over an Area of 1,500 acres Near Kanrach Valley Lasbela
Mine/Project Name	District Balochistan
Mine Method	n/a
Permit Capacity	n/a
Permit Area	1,500 acres
Permit Depth	n/a
Valid Date	December, 5th 2004 — December, 5th 2024
Issue Date	June, 25th 2005
Issuer	Directorate General Mines and Minerals Balochistan

Source: Formal documentation

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

Figure 5.1 — Duddar Lead Zinc — Project Location



5.4 EXPLORATION AND MINING HISTORY

Mineralisation in the area has been recognised since 3000BC. Ancient civilizations were producing Pb and Cu slag. Mining activity since the 1970's reports sporadic surface mining activities (diggings) mostly for barite ($BaSO_4$).

During the 1960s, the Geological Survey of Pakistan (GSP) started preliminary exploration in the Duddar area. A regional scale aeromagnetic survey was conducted during 1975. The GSP conducted regional scale mapping (1:50,000 scale) and also electromagnetic surveys (EM) in 1985. The deposit was discovered by a joint exploration program by the United Nations Development Program (UNDP) and the Geological Survey of Pakistan (GSP) in 1988.

The discovery hole, D001, intersected massive sulphide mineralisation with a corrected thickness of 6.5m grading 16.4% Zn and 3.9% Pb. More drilling was conducted jointly by UNDP and the Pakistan Mineral Development Corporation (PMDC) during 1992-94. Pasminco Pakistan (Private) Limited (PPL) and the Balochistan Development Authority (BDA) conducted detailed exploration (aerial photography, drilling and sampling) during 1995 -1997. Pasminco continued drilling in 1998 and MCC carried out its own metallurgical drilling in 2004. Exploration activities are summarised in ***Table 5.2***.

Table 5.2 — Duddar Lead Zinc — Exploration and Mining History

Year	Activity	Comment
1960 – 71	GSP-investigation on barite-galena	
1974 – 75	Regional aeromagnetic	No anomaly
	Gravity survey	Anomalous gravity
1977 – 80	Mapping of Duddar north and south	Anomalous Pb-Zn
1985	Regional (1:50,000 scale mapping) by GSP	Based on positive results of the geophysical surveys conducted in 1975
1987 – 89	Reconnaissance survey by GSP	Regional stream-sediment program completed
1988	Geophysics survey : IP, EM	Positive results on occurrence of mineralisation lead to drilling
1988 – 91	Eight diamond drill holes (DDH)	1577m drilling, Sedex type Zn-Pb-Fe mineralisation was discovered
1992 – 94	54 DDH completed (17,900m), Geophysical surveys (IP and magnetic), preliminary metallurgical tests and economic appraisals were completed.	A stratiform sulphide deposit with high grade Pb-Zn was delineated
1995 – 97	32 DDH drilled (18,733m), geophysical surveys (EM and IP) were completed.	
1998	11 DDH drilled by Pasminco	Resource confirmation Holes Holes used for bulk metallurgical samples.
2004	10 DDH drilled by MCC	Information for these holes was also used in the Resource

Source: 2005 Basic Design Report and ENFI communication

Exploration methods and reporting was completed in collaboration with international consultants.

5.5 GEOLOGY

In M-MC's opinion, the geological interpretation and mineralisation of the Duddar Pb-Zn deposit is well understood.

5.5.1 Regional Geology

The mineralisation at Duddar is associated with the Jurassic carbonate and clastic sediments in the Paleocene-Eocene collision zone between the Indian and the Iranian/Afghani plates. The Duddar area is of Loralai to Sember age and displays evidence of an extensional basin, likely developed during rifting and break up of Gondwanaland. Variable facies and debris flows indicate the onset of extensional rifting in the Duddar Member coinciding with the syn-diagenetic development of mineralisation. The stratigraphic sequences at Duddar are typified by rapid facies changes, with few internal consistent marker beds.

5.5.2 Local Geology

The lithology at Duddar consists primarily of carbonate — shale sequences with minor amounts of fine to coarse silici-clastics. Rapid facies changes typify the stratigraphic sequences at Duddar. The mineralisation is hosted in a clastic sediment sequence. The mineralisation is composed of pyrite/marcasite, sphalerite (Zn) and lesser galena (Pb) within de-calcified mudstones and silicified limestones. A veined and disseminated sphalerite-galena-chalcopyrite stockwork mineralisation zone underlies and cross cuts the stratiform assemblage.

The local geology is shown in ***Figure 5.2*** along with the location of the drill-holes.

The deposit demonstrates a compositional variation: from barium-rich mineralisation in the south (barite pit), to iron-rich mineralisation in the upper levels of the marcasite and pyrite dominant areas of the Pyrite and Zinc Zones (***Figure 5.2 and 5.3***).

The mineralisation at Duddar is associated with numerous faults, which indicate syn-depositional controls of mineralisation. Some faults limit the extent of mineralisation and hanging wall rocks, as shown in the cross-sectional view in ***Figure 5.3***. In the southerly up-plunge area, the mineralisation is hosted within the hinge of a syncline. To the north, the deposit is hosted within an easterly dipping monocline. Current interpretation has delineated 11 geological boundaries within the mineralised zone.

Figure 5.2 — Duddar Lead Zinc — Site Geology and Borehole Locations

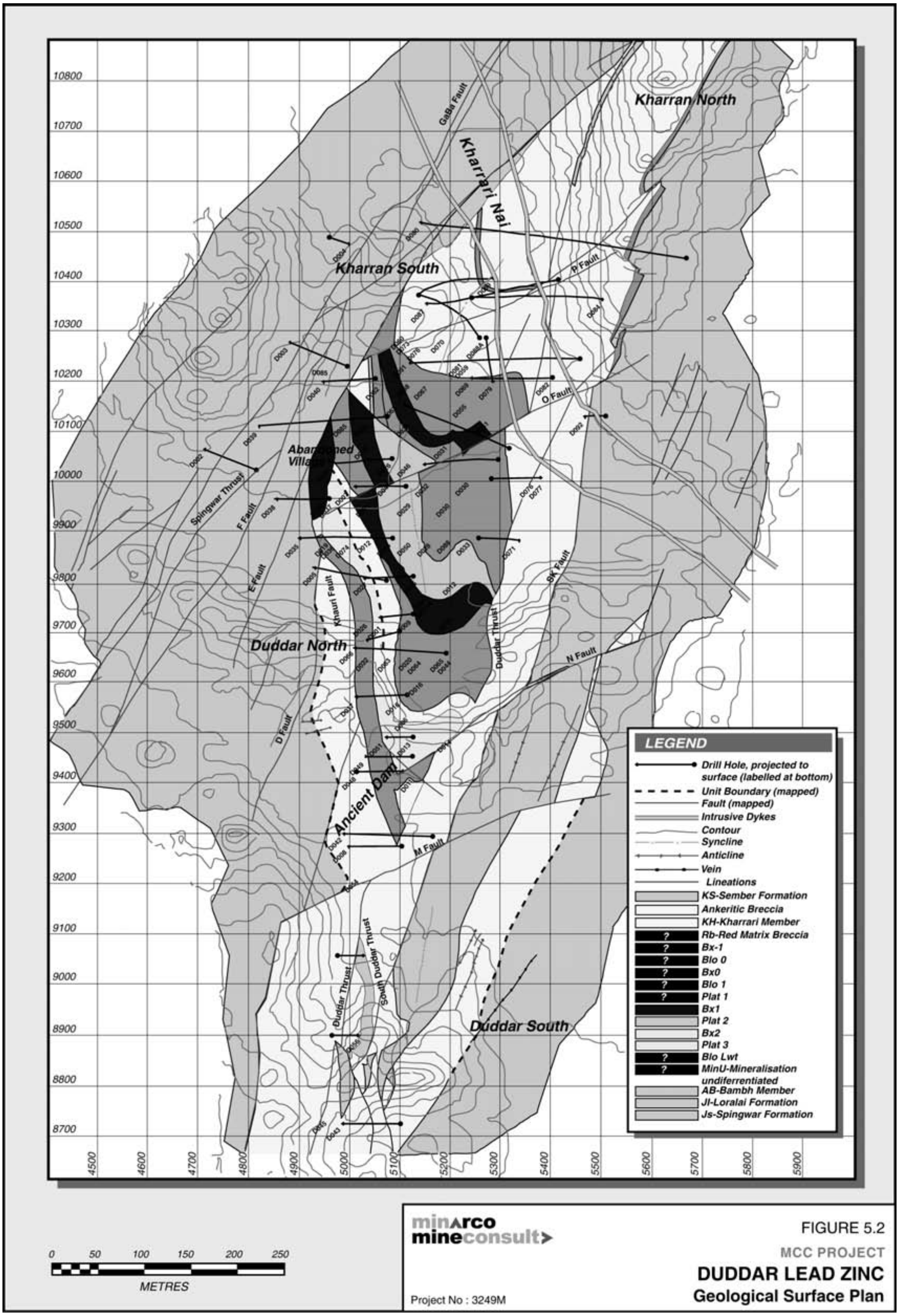
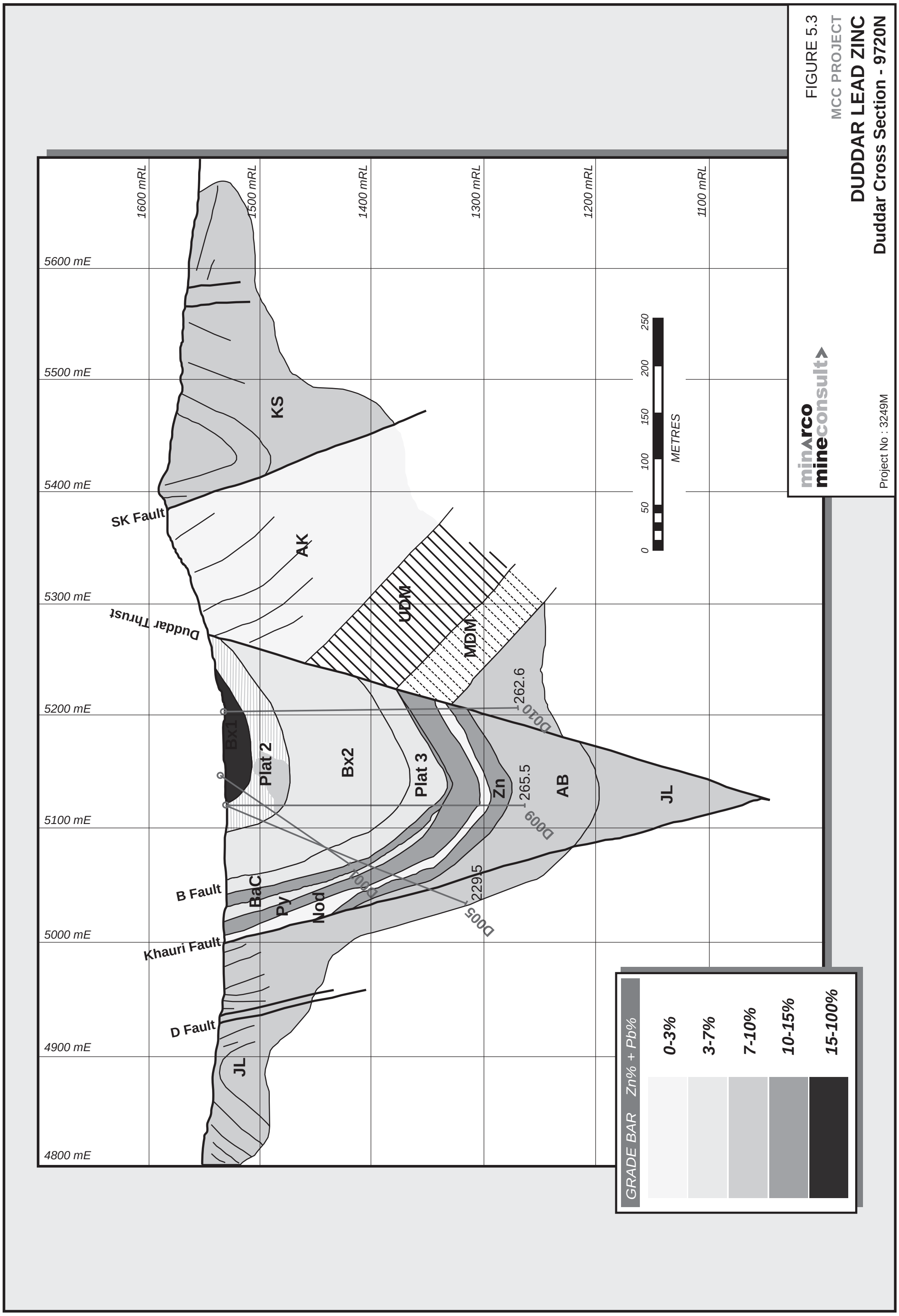


Figure 5.3 — Duddar Lead Zinc — Geological Cross Section



5.6 RESOURCES AND RESERVES

M-MC has reviewed and validated the 2005 Datamine resource model compiled by the China Central Engineering Institute for Non-ferrous Metallurgical Industries (ENFI). The methodology applied to the resource estimates by ENFI in their 2005 model is appropriate and correct for this style of mineralisation. The resource classification applied has been reviewed against the recommendations of the JORC code and appears reasonable. M-MC was unable to carry out a site visit due to perceived political instability and therefore has not validated the underlying data or its quality.

A total of 119 holes have been completed in three exploration programs totaling 46,426m. Some of the holes were angled holes. Most of the drilling was carried out at less than 100m spacing. The location of drill-holes is shown in ***Figure 5.2.***

Exploration methods are summarised in ***Table 5.3.***

Table 5.3 — Duddar Lead Zinc — Exploration Methods — Summary

Exploration Methods	Purpose	Comments
Aerial Photography	Structural mapping	1:44,000 scale was later enlarged to 1:10,000 scale
Geological Mapping	Geology	
Geophysical mapping	Physical properties of the mineralisation quantified	Loop E.M, gradient array IP,
Soil Sampling		62 MMI samples, No obvious anomaly in the prospect, but comparatively higher MMI response for multiple metals indicating faults controlling mineralisation
Mobile Metal Ion (MMI)	To test possible MMI response	
Elemental concentration	Zn, Pb and Cu analyzed	62 samples analyzed; strong correlation with MMI data
Diamond Drilling	Resource definition	46,426m drilling, poorly designed grids
Sampling	Geological logging and assaying	Good core recovery >98%, poor spatial mapping of some drill holes Down-hole surveys carried out at 50m Increments Database in good order, some issues with un-sampled intervals. 0.5-1m sample length based on geology.
Geochemical analysis	Cu, Pb, Zn, Ag, Ba and Fe analyzed	AAS technique was used
Duplicates and assay checks	Cu, Pb, Zn, Ag, Ba and Fe analyzed	AAS, XRF, ICP-OES techniques used, satisfactory correlation was found
Specific Gravity analysis	Ore Density	Diamond core samples. No bulk sampling reported
Geotechnical	Material characteristics	Poor sampling due to lack of training in the first phase, later revised.

Source: 2005 Basic Design Report and ENFI communication

The drill holes were surveyed (down hole) to locate the drill-hole data in three-dimensional space. The drill-hole cores were logged and recorded digitally for lithology and structure using a consistent lithological code and format. The data was stored for manipulation using industrial Techbase® software. An industry standard core-logging procedure was followed to log the geotechnical information of the diamond-drill cores.

In M-MC's opinion, exploration methods were appropriate and sampling and assay techniques are reliable.

5.6.1 Mineral Resources — In Situ Quantities

The Duddar deposit is a stratiform sulphide deposit with high grade Pb and Zn. Mineral Resource estimates for this project have been carried out in 5 main phases as drilling and understanding progressed from 1994 to 2005. The latest estimate was carried out in 2005 by ENFI using all available drilling information. This resource estimated Zn and Pb grades and formed the basis for the ENFI 2005 Basic Design Report. M-MC has reviewed this estimate and makes the following comments.

- The geological interpretation is consistent with the logging and understanding of the deposit.
- Geological wireframes were constructed for faults and geological stratigraphic units within the deposit.
- Resource wireframes have been constructed in accordance with a >7% (Zn+Pb) Cut-off grade and honour the fault and geological wireframes. Wireframes not being snapped to drill holes has lead to slight misappropriation of grades inside the composite files.
- Resource model was estimated using an unfolded IDW^3 methodology and orientated ellipsoidal searches, using 2m drillhole composite data from 4 main mineralisation domains. M-MC has reviewed and validated the block model and estimation parameters and considers them reasonable.
- Bulk density information was not reviewed by M-MC and cannot be verified. Based on the mineralisation style and known geology M-MC considers the bulk density applied to the resource to be appropriate.
- Resource Classification has been applied by ENFI based on drillhole spacing. Areas drilled to less than 40m being assigned Indicated Mineral Resource Category. Areas with greater than 40m but less than 80m have being assigned Inferred Mineral Resource Category. M-MC considers this classification appropriate for the style of mineralisation.

M-MC has reported the ENFI Resources in compliance with the recommendations in the Australasian Code for Reporting of Mineral Resources and Ore Reserves (2004) by the Joint Ore Reserves Committee (JORC). The ENFI estimate of Mineral Resources remaining as at December 2008 for Zn and Pb is summarised in **Table 5.4** using a 7% combined (Pb + Zn) Cut-off grade.

Table 5.4 — Duddar Lead Zinc — Mineral Resources, as at December 2008 at >7% (Zn + Pb) Cut-off grade

JORC Category	Indicated				Inferred				Total				Metal Tonnes	
Geological Domains	**Mt**	**Pb %**	**Zn %**	**Bulk Density**	**Mt**	**Pb %**	**Zn %**	**Bulk Density**	**Mt**	**Pb %**	**Zn %**	**Bulk Density**	**Pb (kt)**	**Zn (kt)**
Minbxhg	0.54	2.1	7.2	3.6	0.15	1.9	6.4	3.6	**0.69**	**2.1**	**7**	**3.6**	**14.4**	**48.6**
Py zone	1.35	1.8	11.5	3.6	0.53	1.7	11.6	3.6	**1.88**	**1.7**	**11.5**	**3.6**	**32.5**	**217.4**
Zn zone	4.52	3.1	11.9	3.6	2.38	2.5	10.6	3.6	**6.9**	**2.9**	**11.5**	**3.6**	**200.4**	**792.3**
Sw zone	2.87	5	7.8	3.3	2.14	4.9	7.2	3.3	**5.01**	**4.9**	**7.5**	**3.3**	**246.4**	**377.3**
Total	**9.28**	**3.4**	**10.3**	**3.5**	**5.2**	**3.4**	**9.2**	**3.5**	**14.48**	**3.4**	**9.9**	**3.5**	**493.7**	**1,435.7**

Source: Datamine 2005 Resource Model by ENFI.(modip55.mdl)
Notes: Mineral Resources are inclusive of Ore Reserves.

The 2005 ENFI resource model did not estimate the significant associated components Ag, Ba and Fe which are known to occur within the resource area. These have previously been estimated by Pasminco in their JORC reported 1997 model. For reference the results from this estimate are summarised in ***Table 5.5.***

Table 5.5 — Duddar Lead Zinc — Mineral Resource Summary Pasminco 1997 at >7% (Zn + Pb) Cut-off grade

JORC Category	Tonnes Mt	Ore Density t/bcm	Average Grades (Pb+Zn > 7%) Zn (%)	Pb (%)	Ag (g/t)	Ba (%)	Fe (%)
Indicated	6.49	3.58	11.0	2.7	17.8	3.0	18.7
Inferred	9.35	3.33	6.8	3.4	21.1	5.4	12.7
Totals	**15.84**	**3.43**	**8.5**	**3.1**	**19.7**	**4.4**	**15.2**

Source: 1997 Pasminco JORC Resource Report
Notes: Rounded totals to indicate accuracy of estimate
59% Indicated and 41% Inferred
Mineral Resources are inclusive of Ore Reserves

Previous Mineral Resource estimates were prepared by Jones (1994), O' Flaherty (1994) and Hudson, (1996). A comparative validation of these estimates with the 1997 estimate is summarised in **Table 5.6**.

Table 5.6 — Duddar Lead Zinc — Comparative Estimates

JORC Category	Inferred (Mt)	Indicated (Mt)	Measured (Mt)	Total (Mt)	Ore Density t/bcm	Zn (%)	Pb (%)	Comments
1997	6.5	9.4	nil	15.8	3.4	8.5	3.1	*cog Pb+Zn >7%*
Polygonal 1996	5.6	13.1	1	18.7	2	8.7	3.3	*cog Pb+Zn >7%*
Jones 1994	3.4	6.9	3	10.3	4	11.4	2.1	*Indicated Grades*
O' Flaherty 1994	2.4	6.9	5	9.3	6	8.5	1.7	

Source: 1997 Pasminco JORC Resource Report

Geological risks include:

- Unsnapped wireframes have lead to miscoding of the database which will result in poor mine designs and slight inaccuracies in estimated grades. (globally these will not be significant)
- Associated components such as Ag, Ba, Fe as well as penalty elements such as Hg and As should be estimated for future reference
- Bulk density information should be estimated in future resource estimates

Geological opportunities include:

- Significant potential for ore zone extensions (additional resources) down dip (at greater depths), this includes a significant drillhole intercept in drillhole D102 of 66m at 2.96% Pb and 11.3% Zn, 550m down dip of the current resource model.

5.6.2 Reserves — Mineable Quantities

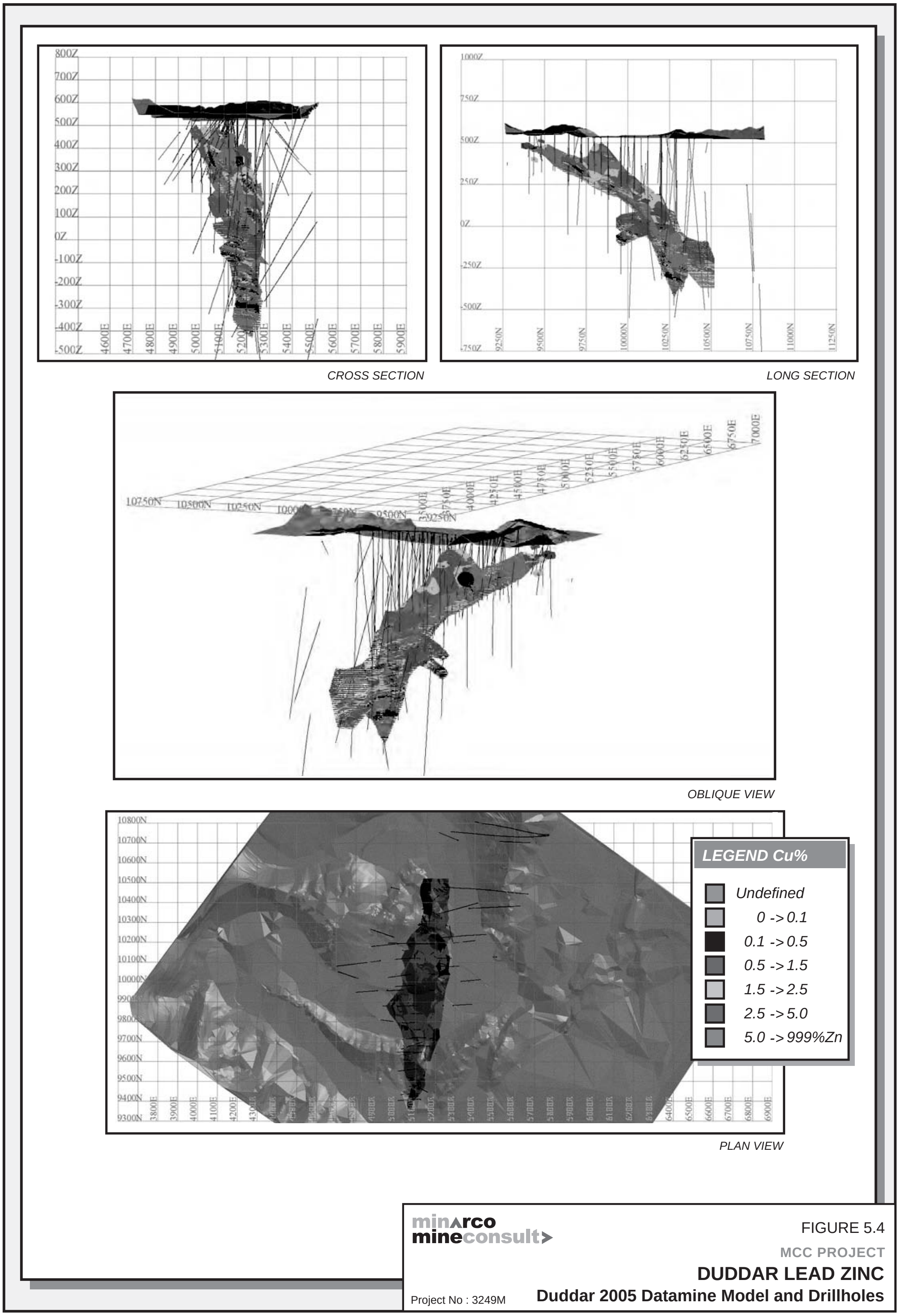
As part of the "2005 Basic Design Report" by ENFI an estimate of Mineable Quantities was carried out based on mine designs applied to the ENFI 2005 Datamine model. The resource is planned to be extracted using sub level open stopping with backfilling of voids where required. The currently designed mine plans will deplete the resource by 9.9Mt at 10.03% Zn and 3.29% Pb. The average ore loss is 14.76% and dilution 7.66%. Due to lack of reporting of the original resource/reserves into either Chinese or JORC code classification and a lack of digital mine design, M-MC could not report the reserves in compliance with the recommendations of the JORC code. For this reason M-MC has used Mineable Quantities when referring to the reserves as shown in ***Table 5.7***.

Table 5.7 — Duddar Lead Zinc — Mineable Quantities Estimates at >7% (Zn + Pb) Cut-off grade

	Total Mineable Quantities			Metal Tonnes	
Zones	Tonnes (Mt)	Pb %	Zn %	Pb (kt)	Zn (kt)
Total	9.13	3.0	9.3	273	849

Source: Datamine 2005 Resource Model by ENFI.(modip55.mdl)
Notes: Due to lack of backing documentation and break down of reserves into category M-MC has reported the reserves as Mineable Quantities.

Figure 5.4 — Duddar Lead Zinc — Datamine Block Model



5.7 MINING

5.7.1 General Description

M-MC's review was based on a document "2005 Basic Design Report" by ENFI.

The orebody has been divided into three different ore types which have then been divided into an upper and lower zone. The ore types are the Stratiform Zone, Stockwork Zone and the Mixed Stratiform and Stockwork Zone. The upper zone has been classified as all levels above the 100 Level and the lower zone has been classified as all levels below the 100 Level. These zones are separated into main levels, which are 100m apart and sublevels, which are 20m apart.

Considering orebody characteristics, mining conditions, rock stability and flexibility, three mining methods have been considered appropriate. Proposed mining methods and the proportion of production include:

- Point pillar overhand slicing and filling (13%)
- Sublevel filling (25%)
- Sublevel open stoping and subsequent filling (62%)

Due to the poor continuity and flat dip angles in the Duddar orebody, sublevels above 100m are mainly mined using the sublevel filling and point pillar slice filling methods. Sublevels below 100m are mainly mined using the sublevel open stoping and subsequent filling method as the orebody is relatively concentrated and steeper.

5.7.2 Forecast Production

Recoverable Resources for the Duddar Pb-Zn Underground Mine have been estimated at approximately 9.9Mt. The "2005 Basic Design Report" itemised 'Mineable Quantities' by 50m vertical increments in the underground mine. This allowed an accurate estimation of tonnes and grade. Each mining method had varying dilution rates and recovery rates, which were used along with the percentage of production for each mining method to estimate recoverable Pb and Zn metal quantities.

The upper stopes would be put into production ahead of schedule when the second year of infrastructure is completed, while the whole mine would be put into full production when the third year of infrastructure is completed.

Based on Mineable Quantities estimates of 9.13Mt, the Duddar Pb-Zn Mine has an approximate mine life of 13.8 years. Full production is not obtained immediately though so the actual mine life is approximately 15 years, based on current reserves. The orebody is not yet defined at the northern extent and there is over 6Mt of Inferred Mineral Resources, which may help to prolong the life of the mine.

The "2005 Basic Design Report" used a production rate of 2kt of ore per day. Based on the continuous working roster of 330 days/ year, 3 shifts/day and 8hrs per shift, annual production is estimated to be 660ktpa. There was no information detailing the date that full production would be reached, but it has been assumed by M-MC that full production is reached in 2012. ***Table 5.8*** illustrates the forecast production figures supplied by MCC in January 2009.

Table 5.8 — Duddar Lead Zinc — Forecast Production Figures

Stream	Unit	2,008	2,009	2,010	2,011	2,012	2,013	2,014
ROM Feed	kt	—	30	100	400	500	600	660
Lead grade	%	—	1.20	2.10	2.50	2.80	2.90	2.95
Zinc grade	%	—	7.80	9.10	9.20	9.25	9.30	9.35
Lead Concentrate	t	—	432	2,730	14,000	20,160	22,778	25,488
Grade (Pb)	%	—	50	50	50	50	55	55
Recovery (Pb)	%	—	60	65	70	72	72	72
Zinc Concentrate	t	—	3,744	15,288	63,296	74,000	81,840	83,546
Grade (Zn)	%	—	50	50	50	55	60	65
Recovery (Zn)	%	—	80	84	86	88	88	88

Source: MCC provided Capex and Production figures February 09

Based on a review of production estimates, production rates used in the "2005 Basic Design Report" are considered reasonable for the Duddar Pb-Zn Underground Mine.

5.8 MINERAL PROCESSING

The ore types at Duddar consist of massive and disseminated structures and it appears that a reasonable quantity of the ore types contain the economic minerals in fine associations with each other and the marcasite/ pyrite. A feature of the ores is the in situ interactions between the marcasite/pyrite and both the galena and sphalerite.

Two ore types are distinguished as either 'layer' or 'vein' types. The chemical and mineral compositions of the 'layer' ore are presented in ***Table 5.9*** and ***Table 5.10***. The tables show that the dominant economic mineral is sphalerite and the dominant gangue mineral is quartz followed by feldspar. The mined ore has a significant marcasite/pyrite content of 50% and an organic carbon content of 0.5%.

Table 5.9 — Duddar Lead Zinc — Layer Ore Feed Grade

Species	Zinc	Lead	Sulphur	Iron	Copper	Silver (g/t)	Gold (g/t)	Silica
Assay (%)	10.45	2.11	31.92	23.38	0.04	11.14	0.06	21.05

Source: 2005 Basic Design Report

Table 5.10 — Duddar Lead Zinc — Layer Ore Mineralisation

Mineral	Sphalerite	Galena	Chalcopyrite	Tetrahedrite	Iron Sulphide	Quartz	Feldspar	Organic Carbon
Proportion (%)	14.98	2.35	0.1	few	49.96	19.1	4.5	0.5

Source: 2005 Basic Design Report

For the 'layer' ore, the galena is typically fairly coarse, with more than 52% coarser than 74 microns. It has a close relationship with marcasite and the gangue minerals, particularly as composites. While sphalerite is quite coarse (45% coarser than 74 microns), it generally forms a very fine relationship with the gangue minerals, most notably quartz, which makes this separation difficult. Sphalerite can also have an intimate relationship with marcasite, similarly causing difficulties during separation. Iron sulphide is present predominately as marcasite, with very little pyrite present. While the marcasite is fairly coarse (82% coarser than 74 microns), it generally forms very fine structures with the sphalerite and the galena, which affects the separation process.

The chemical and mineral compositions of the 'layer' ore are presented in ***Tables 5.11*** and ***5.12***. The tables show that the composition of the vein ore is very similar to that of the layer ores, except that more silica is present while the amount of marcasite has decreased significantly to 37%. The structure of this ore is mainly zonal followed by massive.

Table 5.11 — Duddar Lead Zinc — Vein Ore Feed Grade

Species	Zinc	Lead	Sulphur	Iron	Copper	Silver (g/t)	Gold (g/t)	Silica
Assay (%)	10.72	3.07	25.19	18.18	0.03	21.44	0.05	26.07

Source: 2005 Basic Design Report

Table 5.12 — Duddar Lead Zinc — Vein Ore Mineralisation

Mineral	Sphalerite	Galena	Chalcopyrite	Tetrahedrite	Iron Sulphide	Quartz	Feldspar	Organic Carbon
Proportion (%)	14.67	3.2	0.08	few	36.87	23.5	7.76	0.8

Source: 2005 Basic Design Report

In this ore, the galena is typically fairly coarse, with more than 51% coarser than 74 microns. While it has some close relationships with sphalerite and the gangue minerals, it is generally easily separated from these minerals. As with the layer ore, the sphalerite is quite coarse (45% coarser than 74 microns), and generally forms a very fine relationship with the gangue minerals, most notably quartz, which makes this separation difficult.

Iron sulphide is present predominately as marcasite, with very little pyrite present. The marcasite is fairly coarse (68% coarser than 74 microns) and can form fine structures with the sphalerite and the galena, which may affect the separation process.

On a general analysis basis, between 6.35% and 9.60% of the lead is present as lead oxides while 4.88% and 8.15% of the Zn occurs as zinc oxides. In the separation process adopted at Duddar, these oxides minerals are not recovered, meaning that the maximum recoveries of these metals is limited to 90.4% to 93.65% for Pb and 91.85% to 95.12% for Zn.

A reasonable quantity of testing has been conducted on ore types from Duddar and the general process behaviour of the ore types is understood. This consisted mainly of flotation testing as well as mineralogy, measuring important parameters such as the Bond Indices and the settling and filtration rates of tailings and concentrates. In the 1990s, pilot plant testing was been conducted at Lakefield in Canada and Amdel in Australia, with Beijing General Institute of Mining and Metallurgy (BGRIMM) completing the testing in 2004 based on samples provided by the China Metallurgical Group Corporation Resources Development Company. Pasminco Pty. Ltd. (Australia) conducted a Pre-feasibility Study in 1997 which was complimented by both a Preliminary Study (2000) and a Feasibility Study (2004) undertaken by ENFI. BGRIMM also conducted testing to produce a 'sulphur' concentrate containing most of the marcasite. This concentrate has potential as feed for a sulphuric acid facility.

The processing method commences with mined ore being crushed underground in a primary crusher 600mm x 900mm with a 75kW motor. The crusher treats 114tph of ore up to 500mm in size and crushes it to below 200mm. Ore is then hoisted to the surface and stored in a stockpile.

Ore is recovered from the stockpile and crushed in a three stage crushing circuit ***Figure 5.5*** until it is below 12mm and then stored in a 2,000t live capacity fine ore bin. The crushed ore is withdrawn from the bin and fed to a two stage milling circuit. The first milling stage consists of 3.2m diameter by 5.4m long ball mill in closed circuit with a nest of three 550mm diameter hydrocyclones. The hydrocyclone overflow, 55% to 60% passing 74 microns, reports to a similar sized ball mill in closed circuit with a nest of five 500mm diameter hydrocyclones. This produces an overflow 80% passing (P_{80}) 74 microns at 34% solids. The Bond Ball Mill work index is quite high at 15.5kWh/t and the milling circuit is expected to draw on a continuous basis around 1.46MW of power.

The hydrocyclone overflow reports to a 22m^3 conditioning tank (7 minutes residence time) where lime is added to depress the marcasite and allow the economic minerals to float. The flotation feed rate is 83tph and Pb is then recovered in a rougher-scavenger flotation operation consisting of fourteen 20m^3 roughing flotation cells (residence time is 20 minutes) followed by two banks of ten 20m^3 scavenging flotation cells. Scavenger concentrate is returned to the head of the roughing circuit. The Pb rougher concentrate is upgraded in a cleaning circuit consisting of three stages: nine 8m^3 first cleaning flotation cells, four 8m^3 second cleaning flotation cells and two 8m^3 third cleaning flotation cells. This cleaning circuit produces a final Pb concentrate assaying 67% lead at an overall recovery of 72%. This is around 77% to 80% of the available Pb, which is a little low.

The tailings from the lead circuit report to two stages of conditioning (each 22m^3 and 5.7 minutes residence time) where the pH is adjusted with lime and copper sulphate added to make the Zn mineral float. The Zn is then recovered from the conditioned slurry in seven 20m^3 roughing flotation cells and eight 20m^3 scavenging flotation cells. The rougher flotation concentrate, the ‘fast floating Zn’, is cleaned in a two stage cleaning circuit consisting of three 20m^3 flotation cells followed by two 20m^3 flotation cells. The scavenger concentrate is re-ground in a 2.5m diameter by 3.6m long ball mill in closed circuit with four 250mm diameter hydrocyclones until the material has a P95 of 43 micron. The overflow from this circuit reports to a conditioning tank (11m^3 and 5 minutes residence time) where more copper sulphate is added in preparation for three stages cleaning: three 20m^3 first cleaner flotation cells, three 8m^3 second cleaner flotation cells and a final cleaning circuit of two 8m^3 flotation cells. The cleaned concentrates from both the roughing and reground scavenging flotation circuits are combined as the final Zn concentrate. Overall, a good quality concentrate is made at 55% Zn at 88% recovery, which is 92% to 95% of the available Zn.

Both concentrates are then dewatered in thickeners followed by filtration to produce a final product with 10% moisture. The Pb concentrate reports to a 18m diameter thickener (0.32t/m^2-d) to produce a 60% to 70% solids product for filtration in 15m^2 ceramic filter (0.26t/m^2-d). The Zn concentrate feeds a 38m diameter thickener (0.36t/m^2-d) and the underflow is filtered with two 30m^2 ceramic filters (filtration rate is 0.32t/m^2-h), see ***Figure 5.6***.

Both filtered concentrates are bagged for export to the ZhuZhou smelter. Silver is recovered during smelting from both the concentrates. The zinc concentrate amounts to nearly 105ktpa containing 57,615t of Zn metal. Some 17.9ktpa of lead concentrate is produced containing 11,975t of Pb metal. Overall, only 50% of the silver is recovered, mainly to Pb concentrate (4.52t of silver metal).

The plant is operated for 330 days a year with equipment availabilities of 62%, 90% and 80% respectively for crushing, milling and flotation and dewatering. The plant is manned by three 8 shifts a day.

Marcasite is a highly reactive form of pyrite and significant quantities of lime are required to depress this mineral (12.2kg/t). Butyl xanthate, a reasonably strong collector, is used to recovery the galena. Moderate quantities of copper sulphate (0.84 kg/t) are used to activate the sphalerite during Zn flotation while reasonable quantities of zinc sulphate (1.1 kg/t) are used during Pb flotation to prevent any activation of the Zn.

The mining operation employs cemented sand fill which uses some 994tpd of material from the flotation tailings. The remaining 638tpd is sent to a 15m diameter thickener and the underflow (50% solids) to a tailings dam (4 million m^3).

There appear to be a number of opportunities to improve the metallurgy in terms of Pb and Zn recovery. In addition, a number of approaches could be employed to lower the reagent consumption rates and make the flotation even more selective.

Figure 5.5 — Duddar Lead Zinc — Crushing and Mining Processing Flowsheet

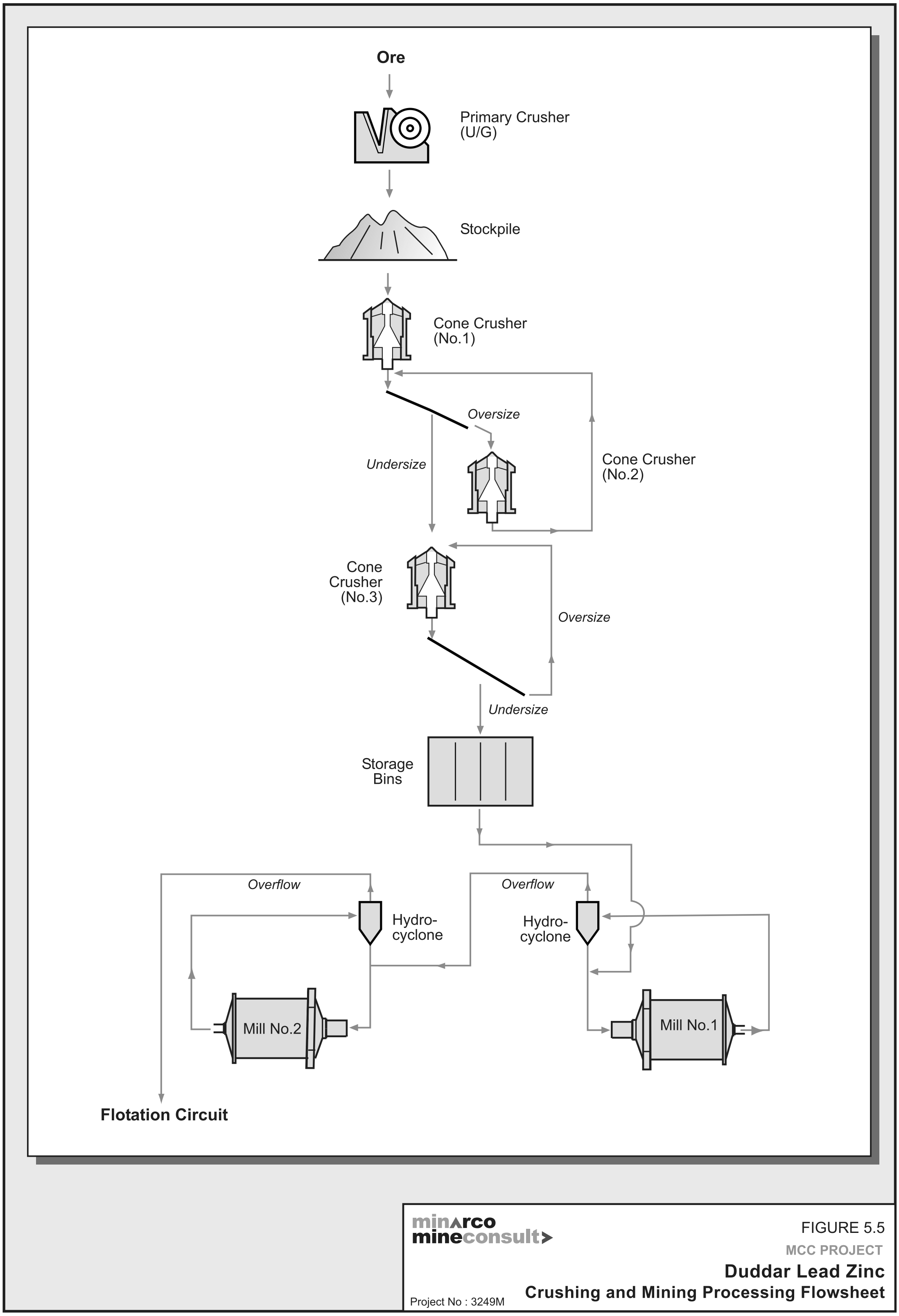
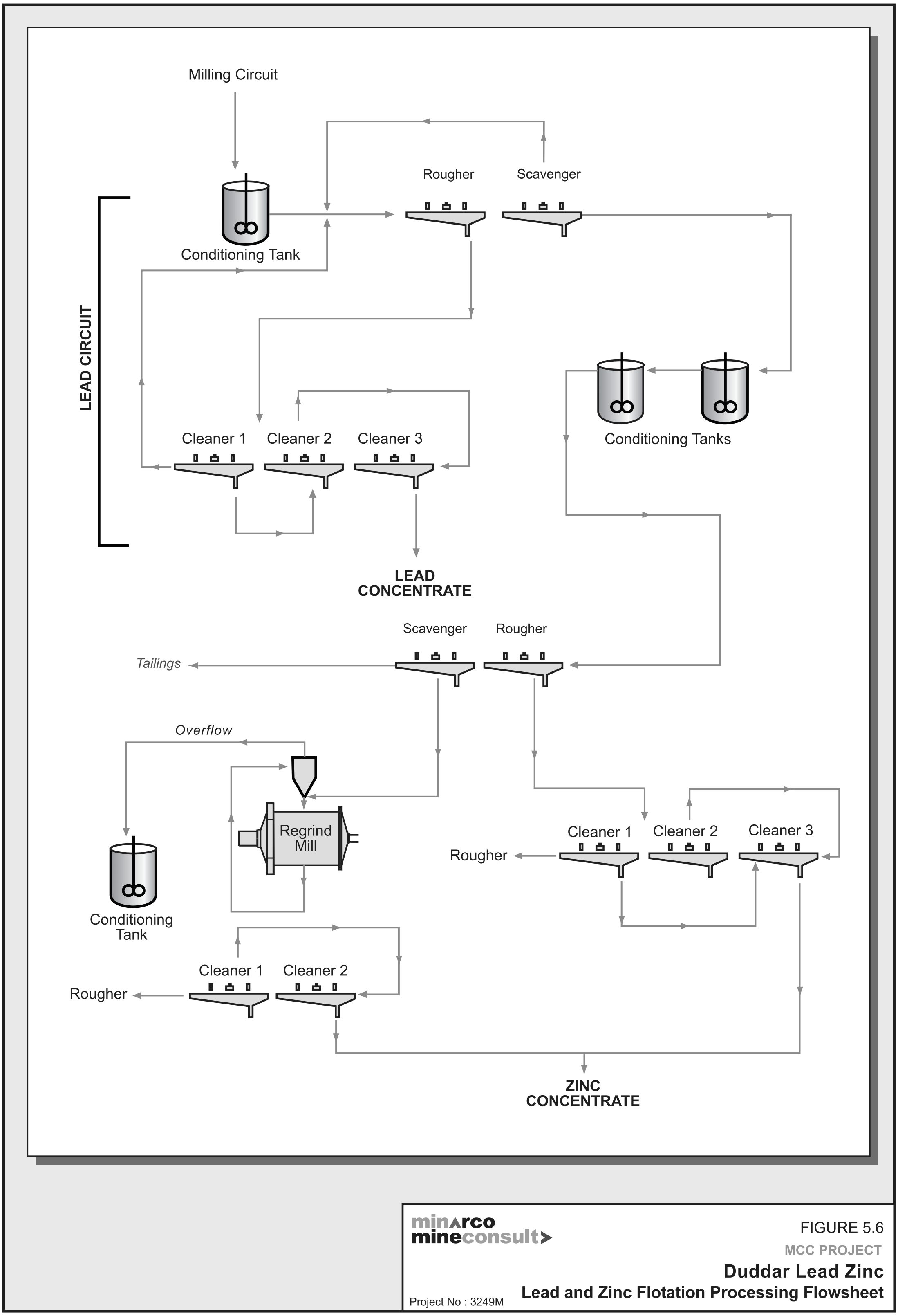


Figure 5.6 — Duddar Lead Zinc — Flotation Processing Flowsheet



5.9 INFRASTRUCTURE AND SERVICES

The level of services and infrastructure is typical of small mining and processing operations, which only requires modest quantities of power, water and consumables as well as manpower.

Electricity will be provided by four 2.2MW heavy oil generators.

The project consumes 26,503m^3 of water, of which 4,507m^3/d is required as fresh water, presumably gathered from the Kharrai River. Discharge of water to the Kharrari River is expected to be 372m^3/d, which is treated in a two stage biochemical process prior to discharge.

The underground mine experiences a maximum of 6,000m^3/d of water inflow, which may be converted to fresh water after treatment.

The design of the new tailings dam appears to have addressed all the local hydrological issues and should be able to resist a 30 year flood in the earlier stages and a 100 year flood thereafter.

5.10 CAPITAL AND OPERATING COSTS

Information regarding operating costs were provided for the period from 2009 to 2011. Historical production and cost data is not available, so comments on the reasonableness of the forecast figures cannot be made.

The costs originally provided by MCC in ***Table 5.13*** appeared to be extremely high compared to other similar operations. The total mining and processing cost estimates ranged from approximately 597RMB/ROM t to 1,731RMB/ROM t over the 6 year forecast. In the opinion of M-MC, a combined mining and processing cost of between 200RMB/ROM t and 500RMB/ROM t would be achievable, once production reaches full capacity.

Table 5.13 — Duddar Lead Zinc — Client Supplied Original Costs

Description	Unit	2008	2009	2010	2011	2012	2013	2014
ROM								
ROM Tonnes	kt	—	30	100	400	500	600	660
ROM Lead Grade	%	—	1.2	2.1	2.5	2.8	2.9	2.95
ROM Zinc Grade	%	—	7.8	9.1	9.2	9.25	9.3	9.35
Price								
Lead Metal	RMB/t	—	6,493	8,500	10,500	12,500	13,000	13,000
Zinc Metal	RMB/t	—	6,288	9,000	11,000	13,000	13,500	13,500
Total Mining Cost	**RMB/ROM t**	—	**733**	**590**	**455**	**410**	**385**	**375**
Processing Cost	RMB/ROM t	—	555	280	190	180	175	172
Administration Cost	RMB/ROM t	—	443	150	60	55	50	50
Total Operating Cost (Mining & Processing & Admin)	**RMB/ROM t**	—	**1,731**	**1,020**	**705**	**645**	**610**	**597**

Source: MCC provided Capex and Opex figures February 09

Verbal communication with site personnel confirmed that the actual mining cost was approximately 300RMB/ROM t and the processing cost was approximately 100RMB/ROM t. M-MC also believes that the concentrate prices supplied by MCC reflect the current world metal prices.

Table 5.14 tabulates the operating costs verbally provided by MCC and in M-MC's opinion, these costs appear reasonable in comparison to other similar operations.

Table 5.14 — Duddar Lead Zinc — Client Supplied Adjusted Costs

Description	Mining	Processing	Total
Unit	RMB/ ROM t	RMB/ ROM t	RMB/ ROM t
Cost	**300**	**100**	**400**

Source: MCC staff advice March 09

Total capital investment for this project was estimated at USD113.0 million in 2005. As of July 2009 MCC have invested 92.4M USD and have forecasted that an additional 20.6M USD will be required to bring the project into production. A detailed breakdown of the actual and forecasted Capex costs is not known. However, even allowing for a 75% escalation in costs, this capital cost is still relatively cost-effective for a project this type and size. Since details of the capital cost breakdown were not provided, M-MC is unable to comment on the reasonableness of the capital allocation.

5.11 SAFETY AND ENVIRONMENT

Sufficient attention appears to have been paid to both safety and environmental matters. The basis of the Safety Plan is based on the appropriate Pakistani regulations as well as the World Bank guidelines. Apparently these were referenced to the related regulations of China. These regulations cover Safety Regulations in metal underground mines, Blasting safety regulations, Hygienic Design Standards of industrial design, Noise Control Design Standards and Hygienic Standards of drinking water.

Attention has been paid to noise suppression around the mine site, particularly the processing plant. Good dust collection and suppression systems have been designed for the processing plant while air conditioning has been considered for most office and living areas.

The basis of the Environmental Protection is the Pakistan Environmental Protection Act (1997) as well as the National Environmental Quality Standards for Effluents, Gaseous Emissions and Motor Vehicle Exhaust and Noise. The project would start with these standards in place, but how well these standards are monitored by local authorities is not known.

So far, USD3.8 million has been spent on minimising the generation of pollutants. Notably, the major pollutant was dust. It is planned to set an environmental department to monitor and control the environmental commitments of the operation.

Greening has been actively undertaken, where trees have been planted on about 20% of the mine site, mainly along roads and around buildings.

Consultations with the local tribes has found strong support for the operation as long as a few issues were considered such as compensation for loss of lands, the opportunity for work, provision of potable water, provision of healthcare facilities and respect of local customs.

6 SAINDAK COPPER GOLD MINE

M-MC did not make a site inspection of this property due to perceived political instability in Pakistan. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "1991 Basic Design of Saindak Copper and Gold Project in Balochistan Province of Pakistan" (1991 Basic Design Report) — including underlying resource models and drill data — prepared by China Central Engineering Institute for Non-ferrous Metallurgical Industries (ENFI)
- "Pre-Investment Feasibility Study for The United Nations — The Saindak Copper Deposits of the Resource Development Corporation, Pakistan" prepared by Seltrust Engineering Limited, and
- "Detailed Project Report" by SIG: a consortium of PEC — COFRAL, OUTOKUMPU — RTB BOR.

MCC's exploits the Saindak Mine through a 10 year rental agreement with the Balochistan Government expiring in 2012.

6.1 BACKGROUND

The Saindak Cu-Au deposit is located in the "Sulphide Valley" at about 950mRL in the Chagai district, Balochistan Province of north-western Pakistan ***(Figure. 6.1)***. The deposit is about 1,540km from Karachi by road or railroad via Quetta. The topography of the surrounding area is rugged. There are three major mineralisation areas hosted by three stocks of tonalite (a felsic plutonic) porphyry namely: South, East and North. The South Orebody is considered the best economic opportunity.

6.2 ASSETS

The assets and status include;

- An operating open cut mine since 2003, including concentrator and furnace smelter.
- M-MC reported JORC Compliant Mineral Resources of 50.9Mt at 0.47% Cu and 0.46g/t Au, remaining as at December 2008. (cog 0.25% Cu)
- Open cut Mineable Quantities of 49.7Mt at 0.45% Cu and 0.47g/t Au, as at December 2008 based on 1991 Basic Design Report. (cog 0.25% Cu)

6.3 LAND TENURE AND MINERAL RIGHTS

Details of Exploration Rights or "Prospecting Licence" for Cu are shown in ***Tables 6.1*** *and* ***6.2***. These are valid to 2009. Other mining licences are also held for quartz and limestone.

Table 6.1 — Saindak Copper Gold — Prospecting Licence 30K — 30L

Mine/Project	**Saindak**
Title	Prospecting Licence
No	30K/8.12 – 30L/5.9
Owner	Saindak Metals Limited
Mine/Project Name	Prospecting for Copper over an Area of 3801.65 acres Near Durbanchah District Chagai
Mine Method	n/a
Permit Capacity	n/a
Permit Area	3,801.65 acres
Permit Depth	n/a
Valid Date	May, 31th 2007 – May, 31th 2009
Issue Date	May, 31th 2007
Issuer	Directorate General Mines and Minerals Balochistan

Source: Formal documentation

Table 6.2 — Saindak Copper Gold — Prospecting Licence 34-C

Mine/Project	**Saindak**
Title	Prospecting Licence
No	34 — C
Owner	Saindak Metals Limited
Mine/Project Name	Prospecting for Copper over an Area of 46487.60 Acres Tehsil District Chagai
Mine Method	n/a
Permit Capacity	n/a
Permit Area	46,487.60 acres
Permit Depth	n/a
Valid Date	Jan, 1st 2007 – Jan, 1st 2009
Issue Date	Jan, 1st 2007
Issuer	Directorate General Mines and Mineral Balochistan

Source: Formal Documentation

M-MC has been informed that the current exploitation of the Saindak mine is through a 10 year rental agreement between the Balochistan government and MCC, signed on the 28th of April 2005. This agreement covers access, development and the royalty and rental payments due to the government as well as terms of the lease with respect to rehabilitation and environmental protection. Rental costs have been reported by the Company as USD 500,000 per annum.

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

6.4 EXPLORATION AND MINING HISTORY

Comprehensive regional exploration began at Saindak in the early 1960's by the Geological Survey of Pakistan (GSP). The first geological mapping at 1:50,000 scale was prepared in 1962 by Ahmed and co-workers.

Detailed geological mapping of the South Orebody at 1:5,000 and later at 1:2,000 scale was prepared to assess economic importance of the mineralised stocks.

Geochemical and geophysical surveys were completed to assess the economic importance of the deposit, as the basis of a diamond drilling programme. This drilling program was planned to define the extensions of mineralisation of the South Orebody.

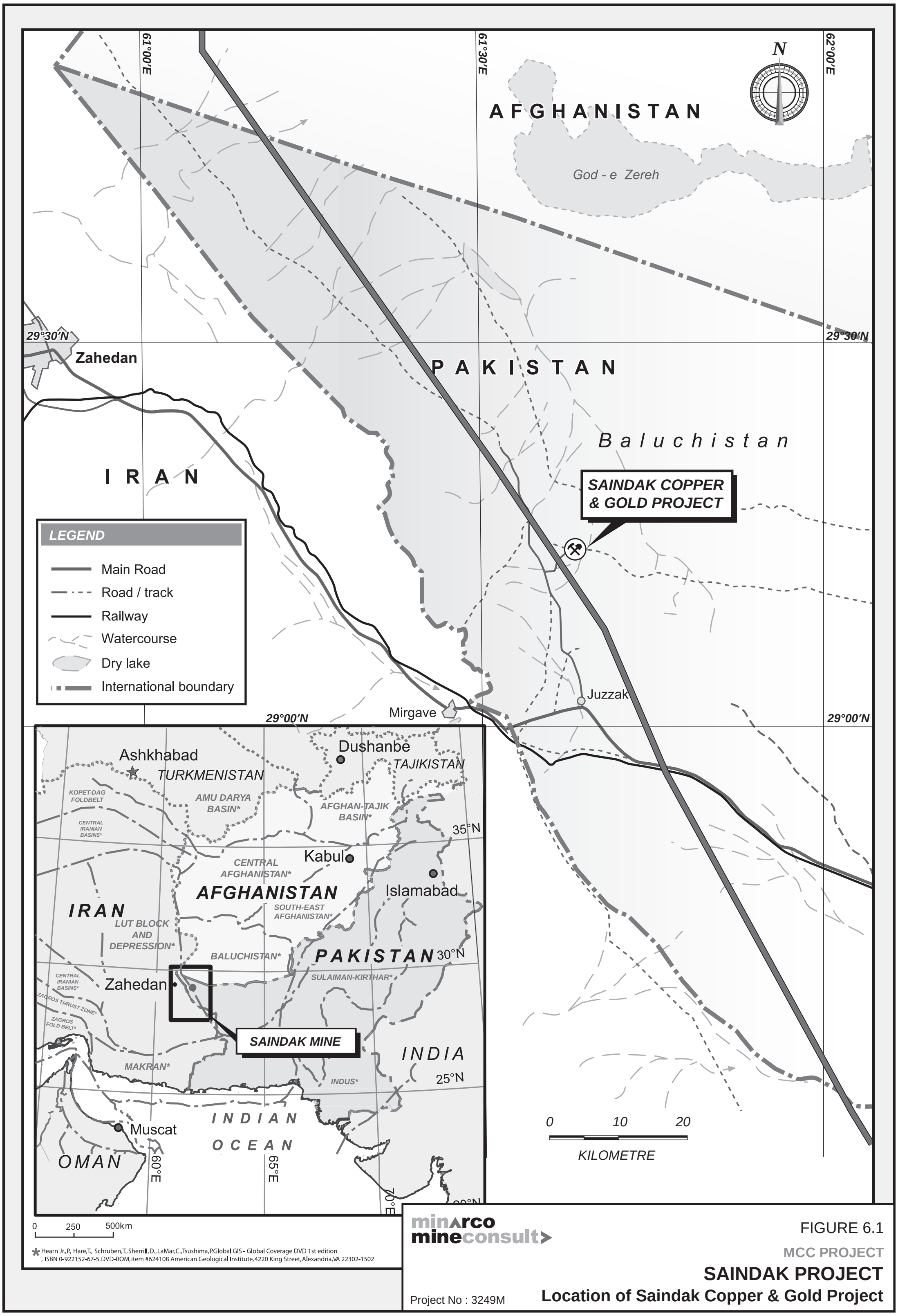
The exploration history is summarised in **Table 6.3.**

Table 6.3 — Saindak Copper Gold — Exploration Summary

Year	Activity	Agency	Comments
1898	Geological survey of the Chagai Belt	Not known	
1959	Systematic geological survey	Colombo plan	
1962	First report of the mineralisation at Saindak, Ahmad and Co. prepared a 1:50,000 geological map of Saindak area	Geological Survey of Pakistan (GSP),	Map included lithological, structural & tectonic elements, crosscuts of quartz-diorite porphyries and the zones of hydrothermaIly altered rocks were superimposed.
1971 – 74	Ground reconnaissance : geological mapping, rock geochemistry, exploratory drilling, geophysical survey (magnetic)	Geological Survey of Pakistan (GSP),	Magnetic survey by Farah and Nazirullah (1974) I.P. survey by Nicholas (1974)
1974 – 76	Geological reconnaissance	Pakistan Resource Development Company (RDC).	Initial grid spacing: approximately 200m E-W and 120m N-S. Later close spaced drilling at 130m x 60m grid along E-W and N-S.
1977	Total 74 vertical holes were drilled in South Orebody; 5,967 samples (3m long) were collected for Cu, Au and Mo analysis.	RDC	38 drill holes were drilled for preliminary exploration of the South Orebody, 36 drill holes were drilled to collect samples for metallurgical tests. Core recovery factor was above 90%.
1994 – 97	Geological survey work on South Orebody	RDC	Basic geology of the deposit, physical/ metallurgical properties of the ore are satisfactorily established.

Source: Basic Design Report 1991

Figure 6.1 — Saindak Copper Gold — Location Plan



6.5 GEOLOGY

In M-MC's opinion, the geological interpretation and mineralisation of the Saindak Cu-Au deposit are well understood.

6.5.1 Regional Geology

The Saindak Cu-Au deposit is located in the Chagai District, Balochistan Province in the north-western corner of Pakistan. The Chagai calc-alkaline series Magmatic Belt is a well known, economically important geological belt hosting porphyry Cu deposits. The Chagai Belt runs about 480km along and inside the Pakistan border with Iran.

The rock types of the Chagai Belt comprises mainly plutonic rocks; granodiorite, greisen diorite and quartz monzonite (granite→diorite series). The mineralisation at Saindak is hosted by the Amalaf and Saindak sedimentary formations of Upper Cretaceous to Oligocene age. The Amalaf Formation is predominantly made up of siltstones, whilst the Saindak Formation is comprised of marls, volcanic agglomerates and andesitic tuffs. The tonalite stocks (quartz diorite porphyry) intrude the Amalaf sedimentary rocks.

6.5.2 Local Geology

Due to prolonged weathering and erosion of the upper portions of the intrusives, a blanket of intensely yellow and red oxidation colours were formed, which is responsible for the name "Sulphide Valley". The local lithology and three mineralised areas are shown in ***Figure 6.2*** and summarised in ***Table 6.4***. Three mineralised stocks are identified as the South, East and North orebodies.

Table 6.4 — Saindak Copper Gold — Orebody Characteristics

	Dimensions of the orebody			
Orebody	Vertical	Length	Width	Comments
South	350m	500m N-S	400m E-W	Simple form of mineralisation, even distribution of grades
North	20-80m	350m	30 – 60m	About 2km away from the South Orebody, extends northwestwards 50^o ~ 60^o, structurally disturbed.
East	500m	1,300m	600m	About 700m away from the South orebody; extends from NW – SE

Source: Basic Design Report 1991

Locally, the host-rocks are cross-cut by quartz-tourmaline, gypsum and anhydrite veins. Two major geological structures are identified in the deposit: the Amalaf Syncline and Saindak Fracture (generally termed as ruptures in the documents reviewed by M-MC). The Amalaf Syncline centres roughly on the East Orebody and extends towards the northwest. The Saindak Fracture extends in an approximate east-west direction, dividing the Saindak area into the South and North Zones. The three mineralised diorite-porphyry stocks occur to the south of Saindak Fracture. Other structural elements in the area include secondary fractures and dykes. There seems to be no apparent structural control of mineralisation.

The sub-vertical stocks are hydrothermally altered. The hydrothermal alteration zones are characterised by distinct mineralogical assemblages. Major alteration zones form roughly concentric patterns, with potassic zone in the centre gradually transiting outward into silicic, sericitic and propylitic zones. The pyrite mineralisation in the potassic alteration zone is uniform with variable Au contents. The Cu content varies in these alteration zones. The metal content in Saindak deposit is primarily controlled by the alteration zones as summarised in ***Table 6.5***.

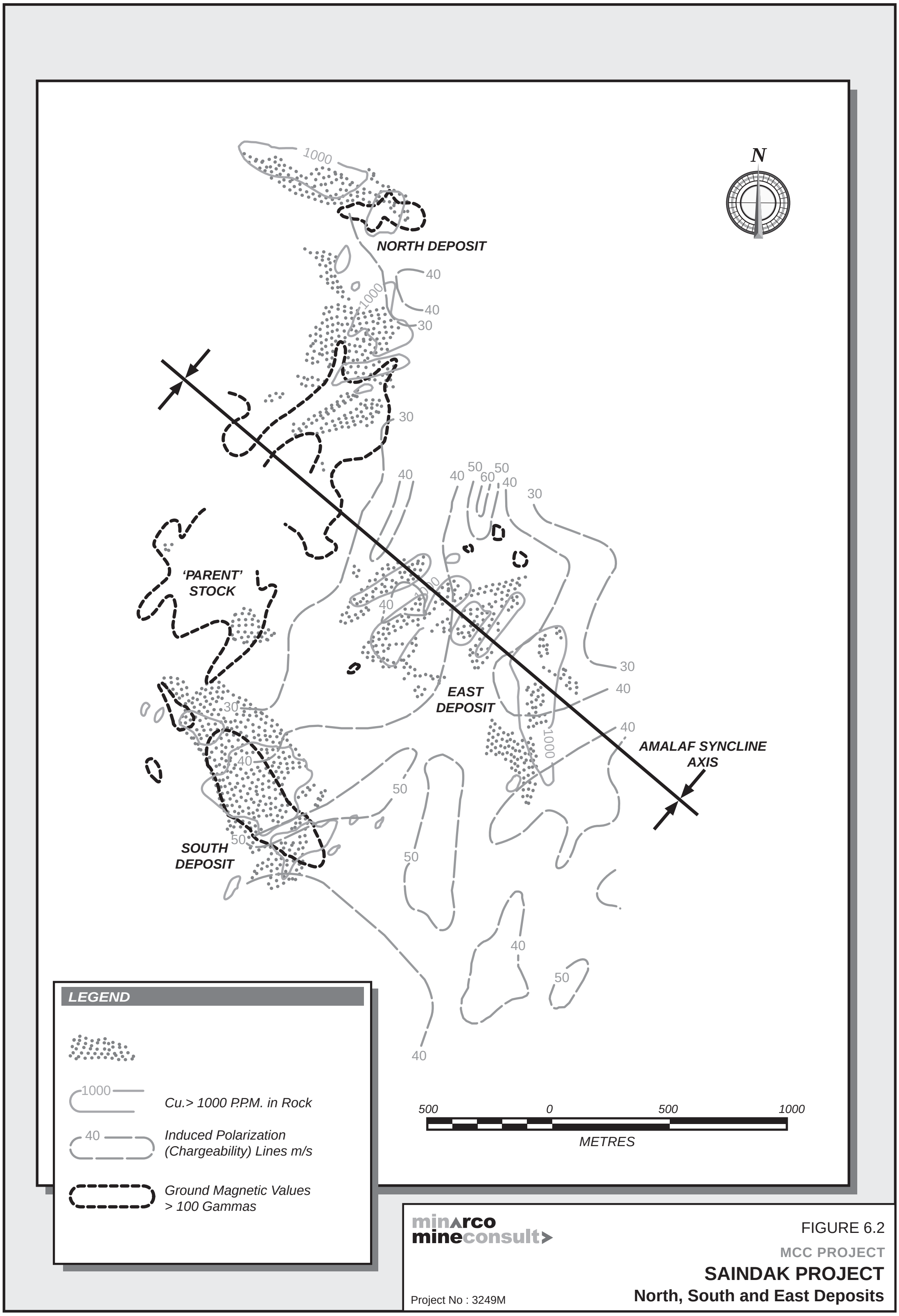
Table 6.5 — Saindak Copper Gold — Grade Distribution in Alteration Zones

	Average grade		
Alteration Zone	**Cu (%)**	**Au (g/t)**	**Comments**
Strong potassic	0.49	0.56	*Based on data from 74 drill holes*
Weak potassic	0.24	0.31	
Sericitic	0.20	0.24	
Propylitic	0.10	0.22	

Source: Basic Design Report 1991

Three types of mineralisation are reported in the South Orebody: (i) primary mineralisation - dominated by sulphides such as pyrite (FeS_2), chalcopyrite ($CuFeS_2$) and minor molybdenite (MoS_2), (ii) oxidation zone — extending up to ~10m in depth, characterised by Cu carbonate and oxide minerals such as malachite and azurite, and (iii) secondary enrichment zones (cementation zone) — weakly expressed, characterised by Cu sulphide minerals such as chalcocite (Cu_2S), bornite (Cu_5FeS_4) and covellite (CuS).

Figure 6.2 — Saindak Copper Gold — Local Geology and Mineralisation



6.6 RESOURCES AND RESERVES

M-MC has reviewed and validated the 2004 Datamine resource model compiled by ENFI. The methodology applied to the resource estimates by ENFI in their 1991 Basic Design Report and 2004 update model is appropriate and correct for this style of mineralisation. The resource classification applied has been reviewed against the recommendations of the JORC code and appears reasonable. M-MC was unable to carry out a site visit due to perceived political instability and therefore has not validated the underlying data or its quality.

A diamond drilling program based on the geochemical and geophysics surveys was prepared to define the extensions of mineralisation of the South Orebody. The GSP started the diamond-drilling program with two holes drilled (inclined at 43^{o}) in the South Orebody. Between 1974 and 1976, Pakistan Resource Development Company (RDC) drilled more diamond holes on a grid pattern of 200m × 120m (along E-W and N-S directions respectively). Detailed surveying of the drillhole collar locations and deviations along the drill holes was completed.

A total of 74 resource drill holes (18,079m) were completed in two drilling programmes. The second drilling programme included samples for metallurgical testing. A third program of shallow angled holes was undertaken for metallurgical purposes targeting the shallower portions of the resource, this drilling was not used in the resource estimate. The drilling summary is shown in ***Table 6.6.***

Table 6.6 — Saindak Copper Gold — Drilling Summary

Stage	Holes	Metres	Spacing
Stage #1	38	10,993	200m × 120m
Stage #2	36	7,086	50m × 60m
Stage #3	19	1,150	Metallurgical
Total	**93**	**19,229**	

Source: Basic Design Report 1991

Exploration methods and data is summarised in ***Table 6.7.***

Table 6.7 — Saindak Copper Gold — Exploration Methods

Exploration Methods	Details of activities	Comments
Litho-geochemical survey Geophysical mapping	500 samples analysed	Three areas were identified with Cu and Mo anomaly. These areas overlap with potash-silicate alterations.
Magnetic Survey	Magnetic survey by Farah and Nazirullah in 1974 and I.P survey by Nicholas in 1974	Magnetic anomalies are conformable with the effects of intrusive structures and sedimentary formations in the area
Diamond Drilling	38 exploration drill holes and 36 metallurgical drill holes	Total 18,078.91m drilling
Sampling	Core samples were collected for geochemical analysis and metallurgical testing	Approx. 10,000 samples were prepared for geochemical analysis. Samples of 3m length for metallurgical analyses were prepared at 5m or 10m for inclined drill holes and at 6m for vertical drill-holes.
Geochemical analysis Duplicates and assay checks	Elemental analyses were conducted for Cu (total, sulfide and oxide), Au and Mo Duplicates of every tenth sample were verified by other Laboratories.	All work originally carried out included extensive QAQC and indicated no problems with precision or accuracy.
Specific Gravity analysis	More than 200 diamond drill core samples tested by RDC.	Average specific gravity (Ore Density) was determined as **2.68 t/m^3**

Source: Basic Design Report 1991

6.6.1 Mineral Resources — In Situ Quantities

The Saindak Cu-Au deposit is a hydrothermally altered porphyry type deposit with medium to low grade Cu and Au with minor Silver (Ag). A resource estimate for this project has been carried out in 2 main phases and only includes the South Orebody. A geological model was prepared in proprietary 3D mining software (Orpheus, ENFI-DEPOSMODEL v.4.0 in 1991). This model included geological rock types, alteration zones with grade estimated for Cu and Au using Kriging. It formed the basis for the 1991 Basic Design Report by ENFI. ENFI subsequently updated this resource using Datamine software in 2004, see ***Figure 6.3***. This Kriged model used the same drill data as in 1991 but accounted for mining depletion as at December 2004. M-MC has validated this estimate for estimation errors and against the underlying drill data and considers the estimate to be a good representation of the mineralisation and grade within the South Orebody.

The total identified In Situ Quantities reported in 1991 was 96Mt at 0.41% Cu and 0.44g/t Au (cog 0.25% Cu). M-MC has updated this estimate using the 2004 Datamine model provided by MCC to reflect the current mine depletion. M-MC has reported the Resource in compliance with the recommendations in the Australasian Code for Reporting of Mineral Resources and Ore Reserves (2004) by the Joint Ore Reserves Committee (JORC). The M-MC estimate of Mineral Resources remaining as at December 2008 is summarised in ***Table 6.8.*** Resource classifications applied to the 2004 Datamine model by ENFI were applied to the blocks based on kriging variance and distance from the drillhole. Due to the good distribution of drilling and continuity of mineralisation M-MC considers this classification appropriate.

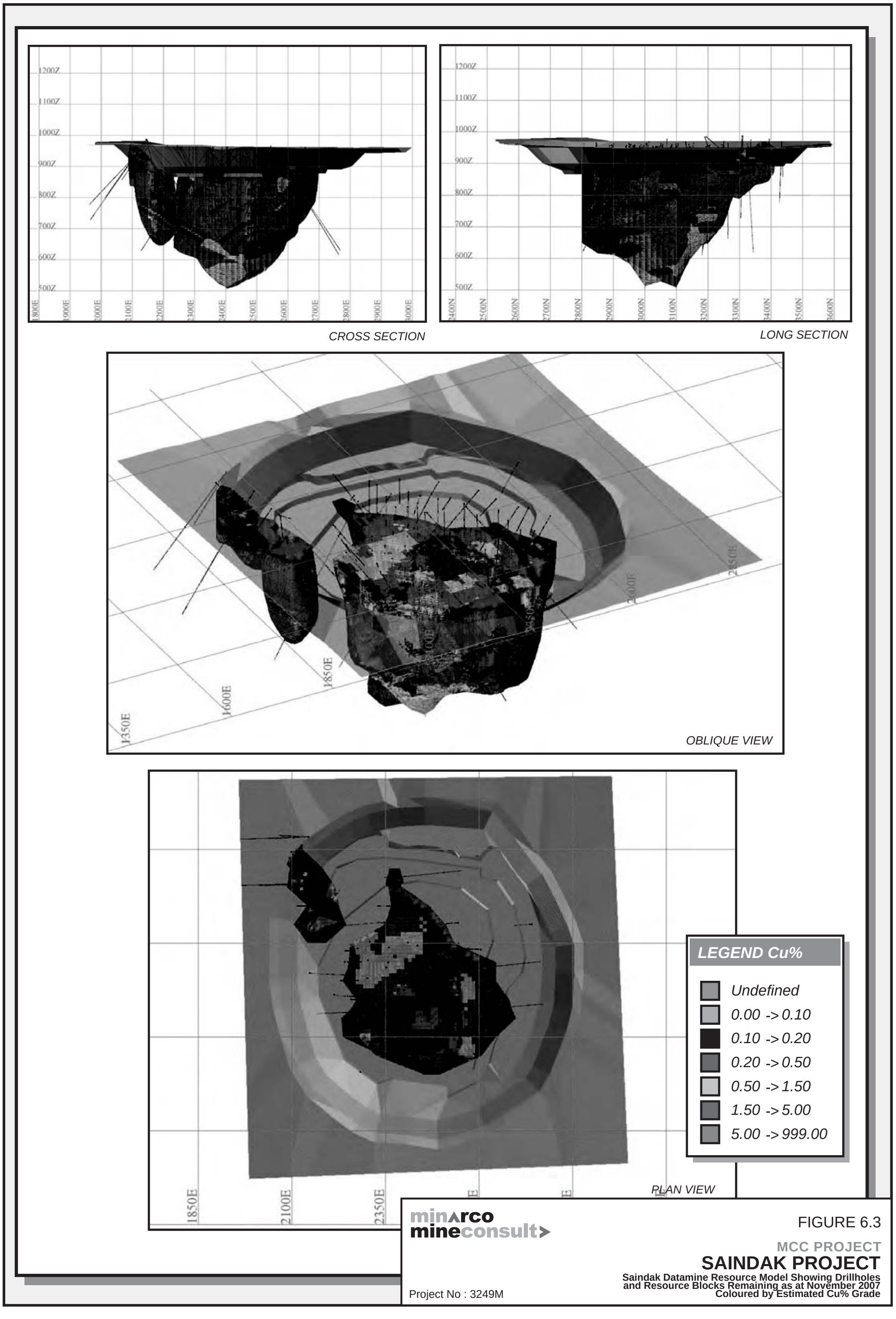
Table 6.8 — Saindak Copper Gold — (South Orebody) M-MC Estimated Mineral Resources, as at December 2008

JORC Category	Tonnes Mt	Average Grade (cog >0.25% Cu) Cu %	Au g/t
Measured	21.6	0.51	0.49
Indicated	14.8	0.46	0.45
Inferred	14.6	0.44	0.44
Totals	**50.9**	**0.47**	**0.46**

Source: M-MC estimated based on the, Datamine 2004 Resource Model by ENFI.
Notes: M-MC has estimated the Resources remaining as at December 2008 based on site surveys and production information. Mineral Resources are inclusive of Ore Reserves.

Ag is known to occur in this deposit in close association with Au. Ag grades in the order of 2.2-2.6g/t are expected to occur throughout, based on previous production. These however cannot be reported under Resources due to lack of supporting drill information.

Figure 6.3 — Saindak Copper Gold — Resource Model and Current Pit Level



6.6.2 Reserves — Mineable Quantities

Ore Reserves have not been reported by category, and do not include parameters to reflect mining loss and dilution. The estimates of "ore reserves" have therefore been referred to as "Mineable Quantities". M-MC has estimated the Mineable Quantities remaining as at December 2008 based on information contained within the 1991 Basic Design Report and recent site surveys and production information for the South Orebody only.

The South Orebody open cut design is based on various optimisations carried out by ENFI in 1991 using USD2,680/t Cu and USD12,868/kg Au prices. This work indentified the 626mRL as the base of economic extraction by open cut with an overall slope angle of 45^{o} above the 758mRL and 42^{o} below. The reserves were reported within this pit shell with no dilution or ore loss factor applied and a variable Cu cut-off grade, based on depth, between 0.275% and 0.25% Cu.

A summary of M-MC's estimated Mineable Quantities remaining as at December 2008 for the South Orebody is shown in **Table 6.9.** M-MC reported these quantities as "Mineable Quantities" without application of recoveries or dilution, as quoted in the 1991 Basic Design Report.

Table 6.9 — Saindak Copper Gold — (South Orebody) — M-MC Estimated Mineable Quantities Summary, as at December 2008

Elevation	Mining	Cut-off Grade % Cu	Total (Mt)	Ore Density	Cu(%)	Au(g/t)
854-626 mRL.	Open cut	0.25%	49.7	2.68	0.45	0.47
Total			**49.7**	**2.68**	**0.45**	**0.47**

Source: Basic Design on Geology of Saindak Mine 1991
Notes: Estimate updated by M-MC based on Site Surveys and Production Information
Mining as at December 2008 had reached parts of the 842mRL.
These estimates do not include mining loss or dilution.

Silver grades in the order of 2.2-2.6g/t are also expected to be recovered during mining, based on previous production. These however cannot be reported under Mineable Quantities due to lack of supporting drill information.

6.7 MINING

The Saindak Cu-Au Mine uses open cut truck and shovel mining methods. Waste is mined in successive cutbacks and dumped in three adjacent dumps. The ore is mined and transported approximately 1km south of the pit. The layout of the mine is illustrated in ***Figure 6.4***.

The equipment used includes five electric face shovels with a bucket size of 10m^3 and a smaller face shovel with a 4m^3 bucket. The fleets comprise of three Terex TR100 trucks and 32 smaller Chinese LN392 68t trucks. Support equipment includes small dozers, graders and water trucks. In M-MC's opinion, based on the specifications and number of equipment, there is more than required capacity to achieve the forecast production targets.

Delineation of the ore and waste during mining is carried out through the use of infill drilling on 25m by 25m spacing to a depth of 50m and followed up with blast hole sampling on a 7m by 7m drill spacing, with Cu and Au assayed at this stage. Based on the amount of data collected it appears that MCC has a well established procedure for controlling dilution and ore loss during mining.

A reconciliation of the block model between the December 2004 surface and the December 2007 surface against actual production revealed larger quantities of ore had been mined at a lower grade. This according to MCC is the result of the lowering of the mining cut off grade to take advantage of the high commodity prices of the past few years.

6.7.1 Historical and Forecast Production

The tables below detail the historical and forecast production. The final pit shell which is illustrated in ***Figure 6.5*** has an average stripping ratio of 1.91. In the previous six years, the operation had stripped large quantities of waste at an average stripping ratio (S/R) of 3.38 t waste/t ore. The next four years has an average S/R of 3.32. As of December 2008, the remaining quantities of waste in the pit have a S/R of 0.82. There is an opportunity to defer this waste therefore increase the value of the project.

Table 6.10 — Saindak Copper Gold — Historical Production

Item	unit	2003	2004	2005	2006	2007	2008	Total
ROM	**Mt**	2.2	4.3	5.0	5.3	5.4	5.3	27.4
Waste	**Mt**	9.4	13.4	16.2	17.6	18	18	92.6
S/R	**t/t**	4.38	3.11	3.24	3.34	3.34	3.43	3.38

Source: Basic Design on Geology of Saindak Mine 1991

Through direct communication with the mine site, the following forecast production quantities were provided.

Table 6.11 — Saindak Copper Gold — Forecast Production

Item	unit	2009	2010	2011	2012	Total
ROM	**Mt**	5.0	4.3	4.3	4.3	17.8
Waste	**Mt**	16.0	15.0	14.0	14.0	59.0
S/R	**t/t**	3.20	3.53	3.29	3.29	3.32

Source: Provided by on-site personnel

M-MC have calculated the remaining mineable in situ quantities from the block model using the December 2007 topography then subtracting the 2008 production data. Based on M-MC's calculation, the forecast waste targets seem to be overestimated.

Table 6.12 — Saindak Copper Gold — M-MC Estimated Remaining Mineable Quantities (December 2008)

Item	unit	Quantity
Ore	**Mt**	49.7
Waste	**Mt**	40.7
S/R		0.82

Source: M-MC Estimated based on the ENFI Resource Model
Note: Final pit design from Basic Design Report 1991

The historical and forecast metallurgical production for Saindak is shown in ***Table 6.13***. Production peaked in 2007 with 18,277 tonnes of blister copper containing 102,316 ounces of silver and 58,619 ounces of gold. With the ROM tonnage dropping slowly until 2010, the subsequent production of blister copper will be 15,500tpa containing 35,525 ounces of silver and 30,871 ounces of gold. The silver and gold appears to harder to recover from ores mined after 2010.

Table 6.13 — Saindak Copper Gold — Historical and Forecast Metallurgical Production

Stream	Unit	2006	2007	2008	2009	2010	2011	2012
ROM Feed	kt	5,265.4	5,384.5	5,250.5	5,000.0	4,250.0	4,250.0	4,250.0
Copper grade	%	0.18	0.39	0.39	0.39	0.39	0.39	0.39
Silver grade	g/t	2.63	2.72	2.63	2.30	2.20	2.20	2.20
Gold grade	g/t	0.51	0.51	0.51	0.42	0.49	0.49	0.49
Concentrate	t	81,501	82,187	81,547	78,000	71,180	71,180	71,180
Copper grade	%	22.4	22.7	22.4	22.3	22.0	22.0	22.0
Copper recovery	%	90.4	89.7	89.0	88.8	89.0	89.0	89.0
Smelter (Blister Copper)	t	18,266	18,277	17,861	17,800	15,500	15,500	15,500
Copper grade	%	99.44	99.43	99.45	99.30	98.5	98.5	98.5
Silver grade	g/t	170.4	174.1	166.5	142.9	71.3	71.3	71.3
Gold grade	g/t	100.7	99.7	99.5	74.3	80.0	80.0	80.0

Source: MCC provided Capex and Opex figures February 09

Opportunities

- Revise the current mine plan to defer waste mining.
- Operating costs could be reduced by decreasing equipment numbers
- Undertake a mineralogical and processing plant audit to reveal the potential for improving the recovery of copper, silver and gold into a higher grade copper concentrate
- Producing blister copper as anode ingots to expand the marketing opportunities

Risks

- Geotechnical risk with any open cut mine greater than 300m deep

Figure 6.4 — Saindak Copper Gold — Mine Site Layout

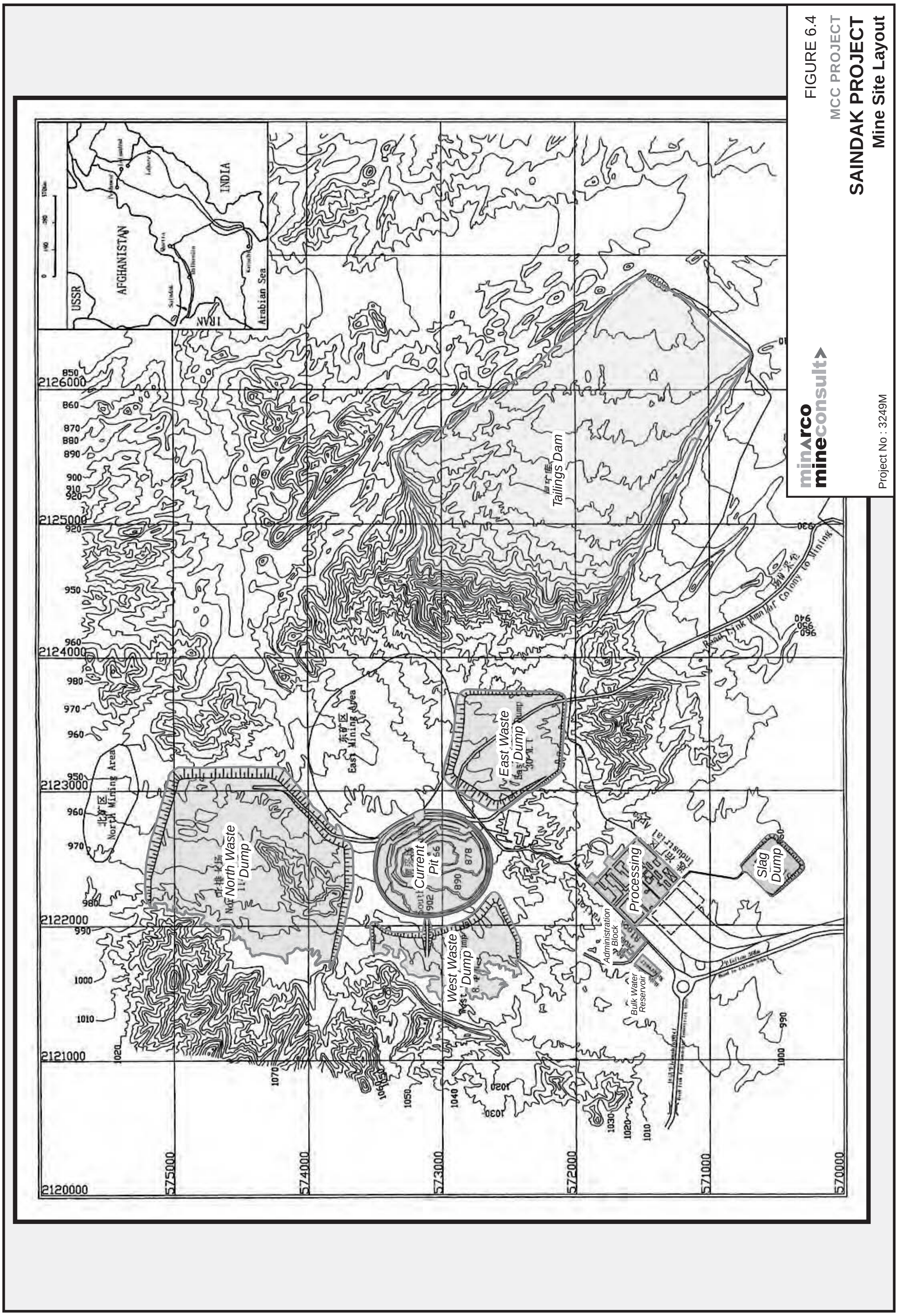
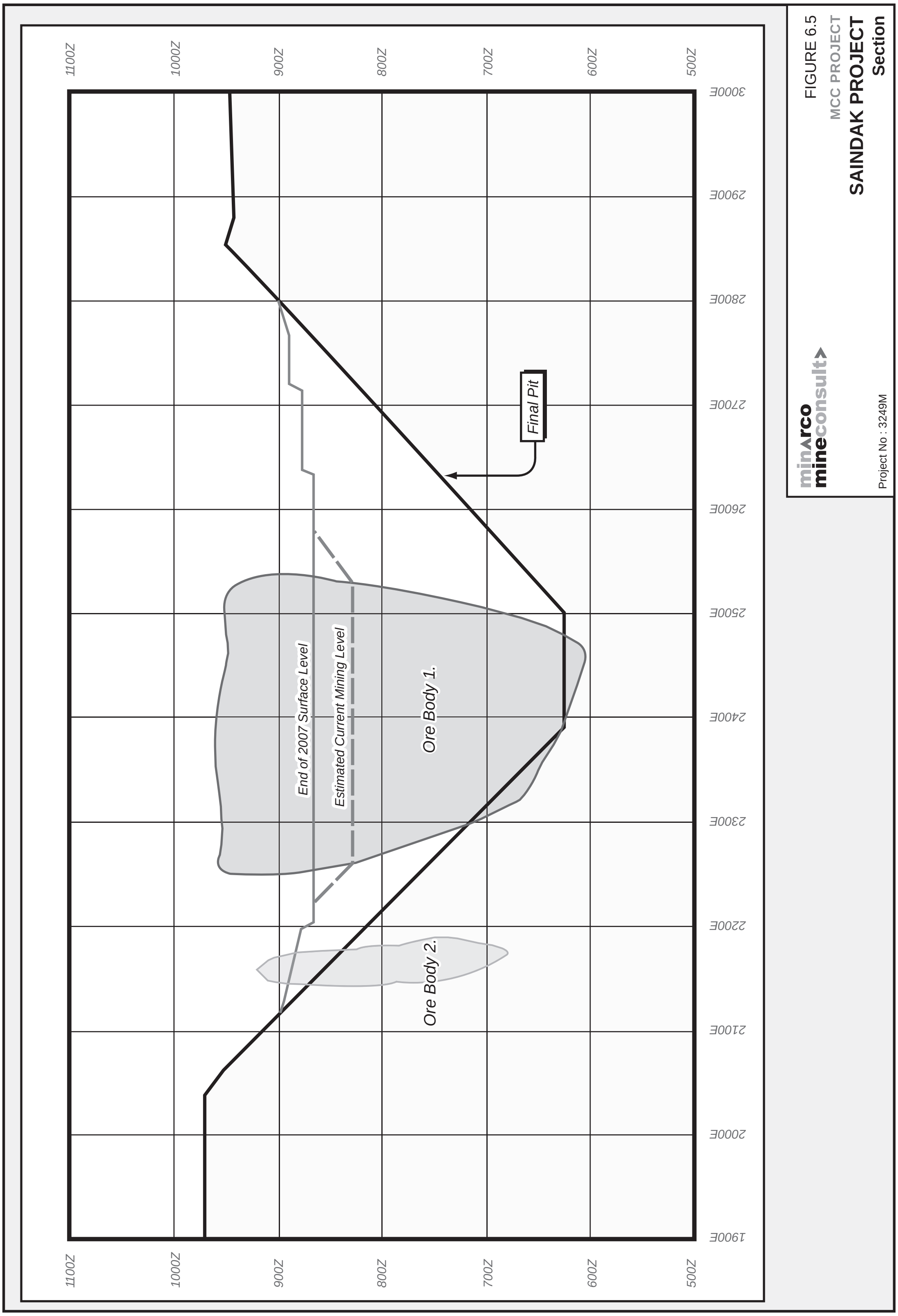


Figure 6.5 — Saindak Copper Gold — Estimated Current Mining Level



6.8 MINERAL PROCESSING

The Saindak processing circuit is conventional and employs a three stage crushing circuit, two stages of milling followed by a flotation circuit which includes a regrinding mill.

Crushing and floatation

The crushing circuit consists of a gyratory crusher (1,250tph with a Closed Side Setting (CSS) of 150-170mm), standard cone crusher (740tph with a CSS 22-25mm) and short head cone crusher (422 tph with a CSS 7-8mm) with a double screen (40mm/14mm apertures) after the primary crusher and a single screen (12mm apertures) after the secondary crusher. The final product is minus 10.5mm and is stored in a fine ore storage bin.

Ore from the fine ore storage bin is fed to a series of three parallel overflow ball mills (5.03mØ x 6.4m each with a capacity of 174-200tph) in closed circuit with a nest of 650mmØ hydrocyclones. The underflow is returned to the ball mill feed for further regrinding while the overflow (P_{68}=74 microns) reports to the flotation circuit.

The flotation circuit is a typical circuit with a rougher and two scavengers (39m^3 flotation cells with a slurry pH of 8-9 with lime) where the concentrate from all stages are upgraded in a cleaning circuit consisting of two stages of cleaning (8m^3 flotation cells with a slurry pH of 9-12) and a cleaner scavenger bank. The cleaner scavenger concentrate reports back to the rougher feed while the second cleaner concentrate re-circulates to the first cleaner feed. The first cleaner concentrate is further processed to produce a final copper concentrate. Scavenger tailings is the final tailings.

This consists of a re-grinding to P_{95}=74 microns in two ball mills in parallel (2.7mØ x 4m [45tph] and 2.1mØ x 3m [15tph]). The cyclone overflow is floated in a secondary rougher/scavenger circuit with the rougher concentrate undergoing two stages of cleaning to produce the final Cu concentrate. The scavenger tailings are further treated in a second scavenger bank and the tailings is directed to the final tailings. The second scavenger concentrate reports back to the feed of the first scavenger while the first scavenger concentrate re-circulates to the secondary rougher feed. The first cleaner tailings also reports to the secondary rougher feed while the second cleaner tailings is fed back to the first cleaner feed. The opportunity exists for the concentrate from the first few cells of both the rougher and first cleaner to report to the final Cu concentrate.

There is the possibility of producing a separate pyrite concentrate by fine grinding the Cu concentrate however no local market exists for this product.

The flowsheet for the crushing, roughing and cleaning flotation circuits are presented in ***Figure 6.6***.

The final Cu concentrate is dewatered in a 30mØ thickener and filtered to 12% moisture with a press filter. The final tailings are thickened in a 27.4mØ high rate thickener and the underflow is pumped to the tailings dam. A total of 301 people are employed in the processing plant, 65 being Chinese.

Typical metallurgy shows that nearly 90% of the Cu is recovered to a 22% grade Cu concentrate — refer to ***Table 6.14***.

Table 6.14 — Saindak Copper Gold — Historical and Forecast Copper Concentrate Production

Measure	unit	2006	2007	2008	2009	2010	2011	2012
Quantity	**t**	81,501	82,187	81,547	78,000	71,180	71,180	71,180
Grade	**% Cu**	22.39	22.63	22.41	22.30	22.00	22.00	22.00
Recovery	**% Cu**	90.43	89.72	89.04	88.77	89.0	89.0	89.0

Source: MCC provided Capex and Opex figures February 09

The crushing, milling and flotation separation appears reasonable based on the limited information provided however no comment can be made on the capacity or sizing of the various pieces of equipment without more detail being made available. This includes the potential for increased throughput and improved metallurgy.

The rather rapid fall in the amount of silver and gold produced after 2010 suggests there are mineralogical problems that are not resolved with the current processing facility. It would indicate either a change in precious mineral associations and/or in grain size. As noted earlier, there would appear to be an opportunity to improve the recovery of copper, silver and gold into a higher grade copper concentrate by undertaking a mineralogical and processing plant audit.

smelter

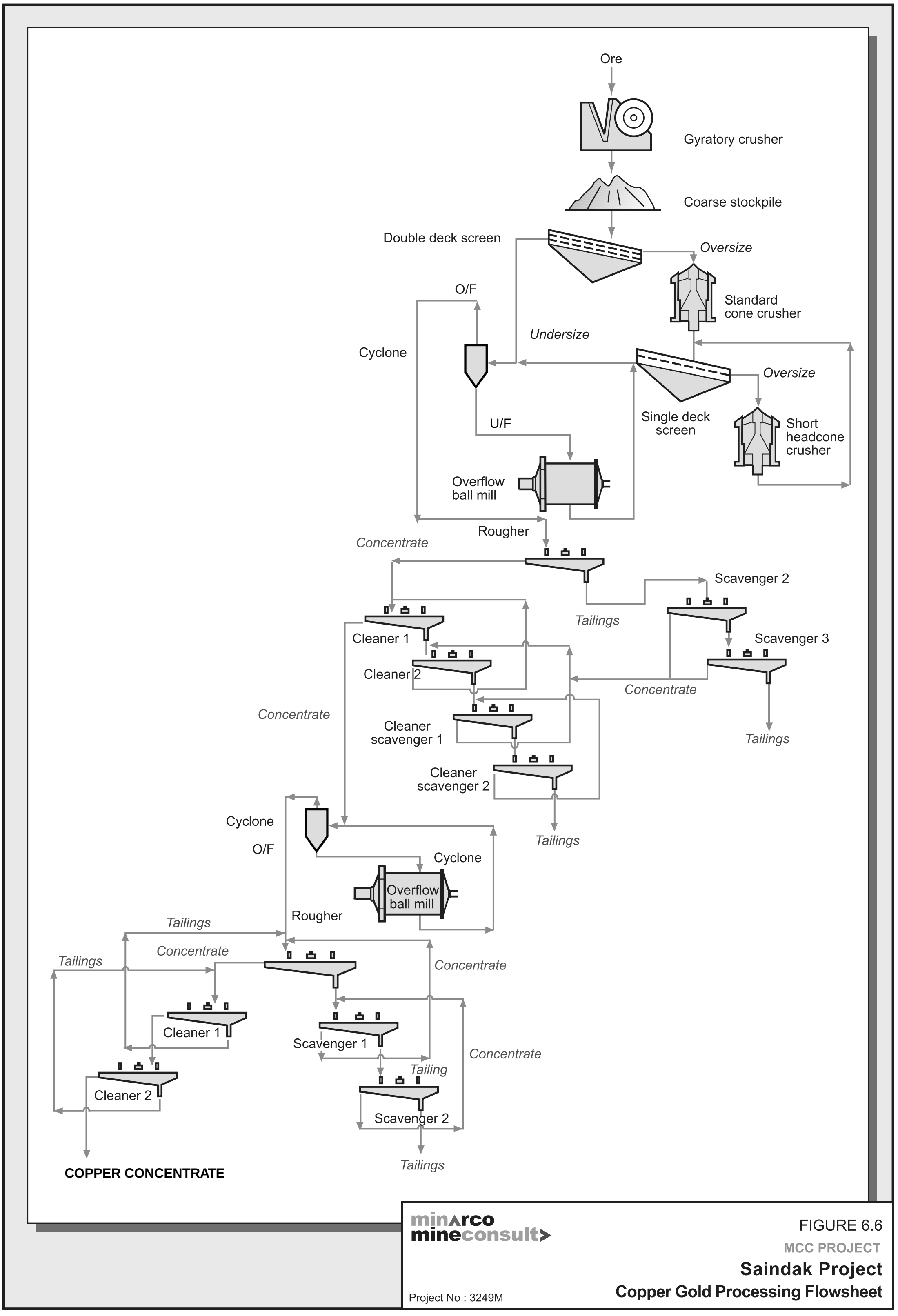
The copper concentrate is dewatered and smelted in a 140m^2 reverberatory furnace with a specific smelting, capacity of 2.47t/m^2 d. Two P-S40 rotary converters (3.2mØ x 6.6m long) are employed to produce blister copper at a rate of 70tpd and a recovery of 97.5% Cu. The blister copper is then cast into 800kg ingots, cooled with water sprays and stored until transport to market. A total of 252 people work at the smelting operation, 63being Chinese.

The smelting operation is capable of handling 90,000tpa of Cu concentrate and 18,500t of Cu concentrate (26.8% Cu) are purchased and blended as feed to the smelting operation. Moisture content of the smelter feed is less than 8% and an overall copper content of 23%. Nearly 21,000 tonnes of silica grading 90% Si is used during the first stage of smelting to remove the iron as a slag contained in the copper minerals.

The flue gases from the smelting and converting operations are treated in an Electrostatic Precipitator (ESP) to remove dust. There is no sulphuric acid plant to capture the sulphur dioxide in the flue gases. These gases are diluted so that exhaust specifications are meet and exhausted to the atmosphere via a stack.

The smelting operation also has a jaw crusher and small ball mill where spent refractory linings are prepared for treatment to recover the entrained copper. It is not clear what this treatment consists of, but presumably the finely ground refractory material is returned to the flotation plant for upgrading. The convertor slag is presumably returned to the reverberatory furnace capture of the entrained copper and the reverberatory slag is possibly discarded with a very low copper content.

Figure 6.6 — Saindak Copper Gold — Copper Gold Processing Flowsheet



6.9 INFRASTRUCTURE AND SERVICES

The level of services and infrastructure is typical of a relatively large mining and processing operation, which requires significant quantities of power, water and consumables as well as manpower.

Electricity is provided by five 11.52MW heavy oil Sulzer engines coupled to 12.25kVA Siemens generating sets, four operating and one on standby. Total generating capacity is 50MW at 6.3kV and this provides power for the mine and processing as well as the township.

Water is supplied from an underground source located some 37km from the operation. Six pumps draw water from eight wells to a 2,000m^3 storage dam, before being pumped to a 6,000m^3 storage dam on site. Water is also recovered from the concentrate and tailings dewatering operations including the tailings dam, for re-use in the process.

6.10 CAPITAL AND OPERATING COSTS

The capital costs associated with this project were not made available to M-MC for comment. MCC site personnel however have indicated that no capital expenditure is planned until the end of the current rental agreement. M-MC considers that if the rental agreement is extended beyond the current cut off date some capital expenditure will be required to upgrade some of the mining equipment which has been in service since 1993 and other fixed plant infrastructure.

Table 6.15 summarises the total historical and forecast operating costs for the Saindak Cu-Au operation, including concentrate sale costs and management fees.

Table 6.15 : Saindak Copper Gold : Historical and Forecast Operating Costs

Mining Cost	Unit	2006	2007	2008	2009	2010	2011	2012
Auxiliary Material	USD (000's)	10,023	11,719	14,824	13,303	12,638	11,000	11,000
Water & Power	USD (000's)	1,292	1,312	2,016	1,386	1,240	1,200	1,200
Labour	USD (000's)	4,475	4,931	6,119	5,848	5,560	5,000	5,000
Repair & Maintenance	USD (000's)	7,285	10,152	9,398	6,200	6,000	5,500	5,500
Mine Development	USD (000's)							
Others	USD (000's)	1,296	2,339	2,796	3,069	2,915	2,600	2,600
Sub-total	**USD (000's)**	**24,371**	**30,453**	**35,153**	**29,806**	**28,353**	**25,300**	**25,300**
Processing Cost	USD (000's)	29,315	31,104	41,884	34,234	32,523	31,300	31,300
Smelting cost (concentrate)	USD (000's)	12,511	13,814	20,487	15,762	14,974	14,500	14,500
Other costs	USD (000's)	228	88	108	190	200	200	200
Concentrate sale	USD (000's)	1,599	1,559	1,647	1,574	1,550	1,463	968
Management fee	USD (000's)	6,616	9,119	10,249	8,226	7,500	7,500	8,000
Sub-total	**USD (000's)**	**50,269**	**55,684**	**74,375**	**59,986**	**56,747**	**54,963**	**54,968**
Total	**USD (000's)**	**74,640**	**86,137**	**109,528**	**89,792**	**85,100**	**80,263**	**80,268**

Source: MCC provided Capex and Opex figures February 09

*Notes: * Includes smelting, concentrate, other and management costs*

The Saindak Cu-Au Mine unit operating costs vary from USD18.80/t to USD 27.56/t (refer to ***Table 6.16***). While the forecast operating costs appear low, they are reasonable for a large tonnage, high throughput operation.

Table 6.16 — Saindak Copper Gold : Historical and Forecast Unit Operating Costs

Operating Cost	Unit	2006	2007	2008	2009	2010	2011	2012
Mining	**USD/ROM t**	4.63	5.66	6.70	5.96	6.67	5.95	5.95
Processing	**USD/ROM t**	5.57	5.78	7.98	6.85	7.65	7.36	7.36
Smelting	**USD/ROM t**	2.80	2.80	3.35	2.70	2.69	2.90	2.90
Other Costs	**USD/ROM t**	5.80	7.41	9.53	8.41	9.70	8.63	8.63
Total*	**USD/ROM t**	**18.80**	**21.65**	**27.56**	**23.92**	**26.69**	**24.84**	**24.84**

Source: MCC provided Capex and Opex figures February 09

6.11 SAFETY AND ENVIRONMENT

Sufficient attention appears to have been paid to both safety and environmental matters. The basis of the Safety Plan is based on the appropriate Pakistani regulations as well as the related regulations of China, including Noise Control Standards of Drinking Water standards. Emissions such as smoke, gaseous and dust emissions are rated against World Bank standards.

Environmental audits are conducted annually by an independent environmental protection agency on behalf of the Pakistan government. Labour and management practices are also checked regularly. The basis of the Environmental Protection is the Pakistan Environmental Protection Act (1997) as well as the National Environmental Quality Standards for Effluents, Gaseous Emissions and Motor Vehicle Exhaust and Noise.

As noted earlier, the operation does not employ a sulphuric acid plant to capture the sulphur dioxide from the flue gases produced during smelting, particularly converting. Under these circumstances, the expected practice would be to dilute the flue gases after dust removal with air and disperse into the atmosphere via a tall stack. The practices that MCC does employ on this site appear to meet the required Gaseous Emissions Standards of Pakistan.

Attention has been paid to noise suppression around the mine site, particularly the power generating facility ($<$85 decibels) and the living quarters ($<$55 decibels). Good dust collection and suppression systems have been employed for the processing and smelting plants while air conditioning has been considered for most office and living areas.

7 CAPE LAMBERT MAGNETITE PROJECT

MCC Australia Holdings Pty Ltd (MCCAH) is completing technical and financial studies to determine the feasibility of developing a magnetite iron ore mine at the Cape Lambert Project. MCC's equity stake in the project is 100%.

M-MC carried out a site visit to the Cape Lambert Project in July 2009. The site visit confirmed the layout of the project, the location of drill holes and site and regional infrastructure.

Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "Resource Model Update, Cape Lambert Iron Ore Deposit, Western Australia" (2009 resource report) — prepared by Golder Associates for MCCAH.
- "Cape Lambert Magnetite Iron Ore Project Pre-feasibility Study Report" prepared by Northern Engineering and Technology Corporation, MCC for MCCAH.

7.1 BACKGROUND

The Cape Lambert Iron Ore (CLIO) deposit is located in the Pilbara region of Western Australia. It lies less than 10km from the coast, some 5km south-west of Wickham and 25km east of the regional centre of Karratha. The project area consists of flat lying coastal plain and ironstone ridges with elevation ranging from 20m to 100m above sea level. Much of the area in the east of the tenement that is underlain by the Cleaverville Formation rocks is moderately rugged consisting of low but moderate to steep sided ridges with intervening rolling country in the valleys. Similar country although with even steeper slopes is present in the Cleaverville Beach area in the north of the property.

The CLIO deposit is a magnetite bearing Banded Iron Formation (BIF) deposit which outcrops to form prominent ridges over a strike length of some 7km. The stratigraphy and mineralisation dips to the east at moderate to shallow angles and has been drill tested to a depth of approximately 400m.

The deposit was explored initially by Robe River Mining Company Pty Ltd (Robe) in the early 1990's. Robe completed a number of drilling programs between 1994 and 1996. The project was subsequently acquired by Cape Lambert Iron Ore Limited (Cape Lambert) which completed drilling programs in 2006 and 2007.

The project was acquired by MCC Australia Holding Pty Ltd (MCCAH) in 2008 for AUD $320M. MCCAH completed further drilling, resource estimation and a pre-feasibility study (PFS) in 2008. The company is about to commence a bankable feasibility study (BFS) on the project, with anticipated completion in first half 2010.

The development scenario being proposed is large scale open pit mining with ore production of 48Mtpa at a grade of 29.5% Fe. Processing will involve crushing, grinding and magnetic separation to produce a high grade concentrate of 15Mtpa at 65% Fe. The concentrate will be transported in slurry form to the coast via a pipeline then loaded onto ships for export.

An underground gas pipeline runs through the central and eastern portions of the tenement to the north of the CLIO deposit and the Robe River Railway that terminates at the iron ore port of Cape Lambert lies across the southern portion of the CLIO deposit. Two power transmission lines are also present in the southern part of the tenement area. This infrastructure will need to be relocated to allow mining to proceed on much of the CLIO deposit.

7.2 ASSETS

The assets and status include;

- A substantial magnetite iron ore deposit delineated by a large number of quality drill holes.
- JORC Compliant Mineral Resources of 1.9Bt at 30.7% Fe (20% Fe cut-off)
- Potential open cut Mineable Quantities of 1.31Bt at 29.5% Fe (20% Fe cut-off)
- Potential annual concentrate production of 15Mtpa at 65% Fe based on 2008 PFS

7.3 LAND TENURE AND MINERAL RIGHTS

Details of the tenements held by MCC are shown in ***Tables 7.1*** *to* ***7.4***. All of the tenements are Exploration Licences and are in good standing.

Exploration licences (EL's) provide a right to explore the area. Prior to any development of the project, Mining Leases (ML's) need to be granted. This process can take several years and requires negotiation with government departments and environmental authorities. The region is also subject to the Aboriginal Heritage Act and requires a Native Title agreement to be negotiated with the traditional Aboriginal owners of the land.

Within the tenement areas are existing small mining leases held by other parties involved in the production of river sand, gravel and aggregate. Miscellaneous Licences also surround the rail, power and gas infrastructure that crosses the CLIO tenements. Agreements need to be reached with the owners of these various licences prior to the conversion of the EL's to ML's.

Table 7.1 — Cape Lambert Iron Ore Project — Exploration Licence 47/1233

Mine/Project	Cape Lambert
Title	Exploration Licence
No	E47/1233
Owner	Cape Lambert Iron Ore Ltd
Mine/Project Name	n/a
Mine Method	n/a
Permit Capacity	n/a
Permit Area	25 blocks
Permit Depth	n/a
Valid Date	November, 17th 2005 - November 16th 2010
Issue Date	November, 28th 2002
Issuer	Department of Industry and Resources, Western Australian Government

Source: Formal documentation

Table 7.2 — Cape Lambert Iron Ore Project — Exploration Licence 47/1248

Mine/Project	Cape Lambert
Title	Exploration Licence
No	E47/1248
Owner	Cape Lambert Iron Ore Ltd
Mine/Project Name	n/a
Mine Method	n/a
Permit Capacity	n/a
Permit Area	4 blocks
Permit Depth	n/a
Valid Date	January, 23rd 2006 - January 22nd 2011
Issue Date	January, 23rd 2003
Issuer	Department of Industry and Resources, Western Australian Government

Source: Formal documentation

Table 7.3 — Cape Lambert Iron Ore Project — Exploration Licence 47/1271-I

Mine/Project	Cape Lambert
Title	Exploration Licence
No	E47/1271-I
Owner	Cape Lambert Iron Ore Ltd
Mine/Project Name	n/a
Mine Method	n/a
Permit Capacity	n/a
Permit Area	20 blocks
Permit Depth	n/a
Valid Date	September, 6th 2006 - September 5th 2011
Issue Date	July, 11th 2003
Issuer	Department of Industry and Resources, Western Australian Government

Source: Formal Documentation

Table 7.4 — Cape Lambert Iron Ore Project — Exploration Licence 47/1462

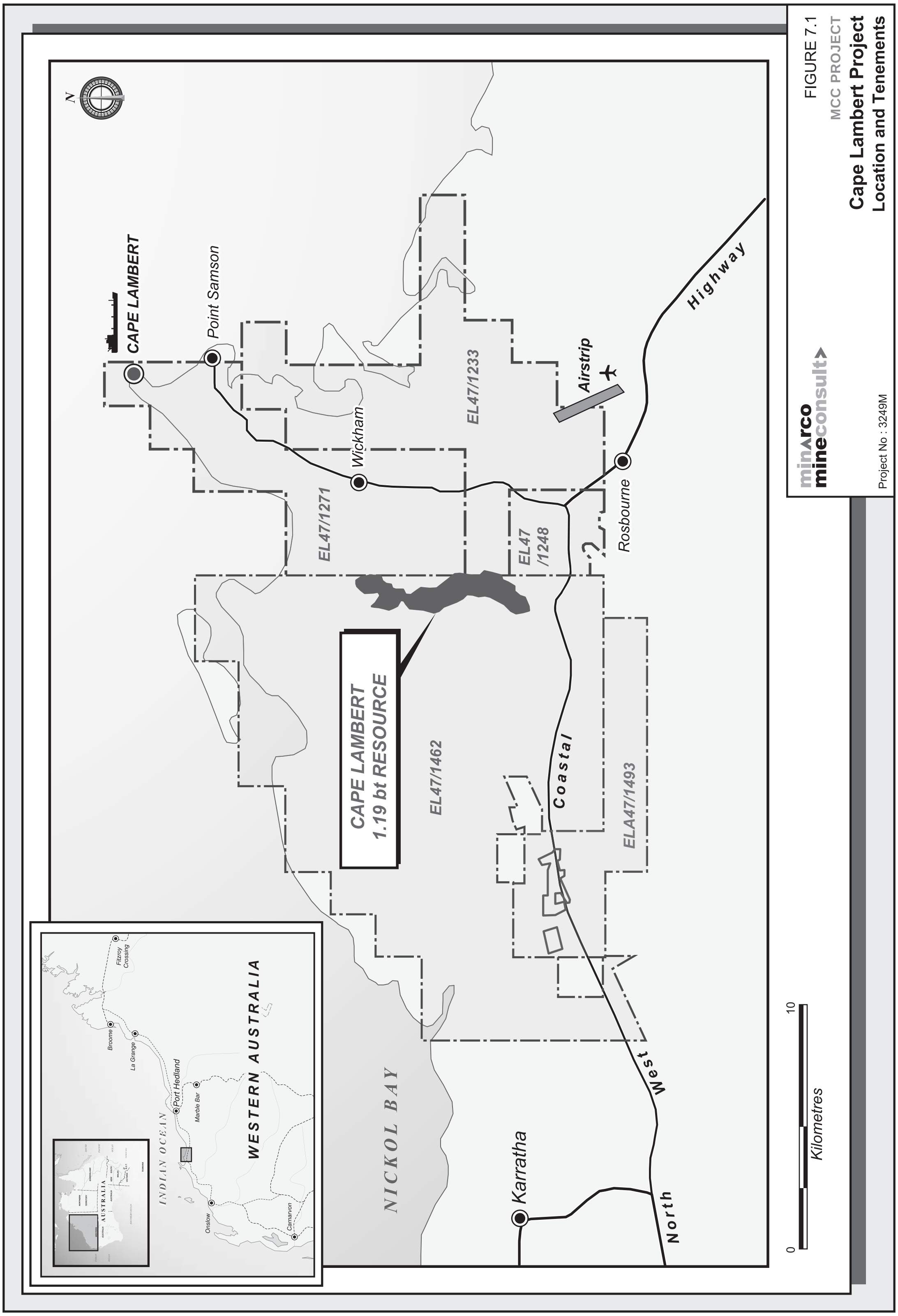
Mine/Project	Cape Lambert
Title	Exploration Licence
No	E47/1462
Owner	Cape Lambert Iron Ore Ltd
Mine/Project Name	n/a
Mine Method	n/a
Permit Capacity	n/a
Permit Area	70 blocks
Permit Depth	n/a
Valid Date	March, 24th 2006 - March 23rd 2011
Issue Date	October, 28th 2004
Issuer	Department of Industry and Resources, Western Australian Government

Source: Formal documentation

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

The project location and tenement outlines are shown in ***Figure 7.1.***

Figure 7.1 — Cape Lambert Iron Ore Project — Location Plan



7.4 EXPLORATION AND MINING HISTORY

Detailed exploration began at the CLIO deposit in the early 1960's when the project was held by Robe. Robe completed 186 drill holes (183 RC and 3 RCDD). Preliminary evaluation by Robe suggested that the deposit was not viable and the tenement was relinquished (Met-Chem 2007).

Cape Lambert acquired the project in 2006. The company completed a number of drilling programs and resource estimates between 2006 and 2008.

In 2008 the CLIO project was sold to MCC Corporation and is now operated by its Australian subsidiary MCCAH. MCCAH has completed further drilling, updated the resource estimate and carried out a pre-feasibility study (PFS). In 2009, MCCAH called for tenders from suitably qualified companies to prepare a bankable feasibility study (BFS) for the project with an expected completion in the first half of 2010.

Geochemical and geophysical surveys were completed to assess the economic importance of the deposit, as the basis of a diamond drilling programme. This drilling program was planned to define the extensions of mineralisation of the South Orebody.

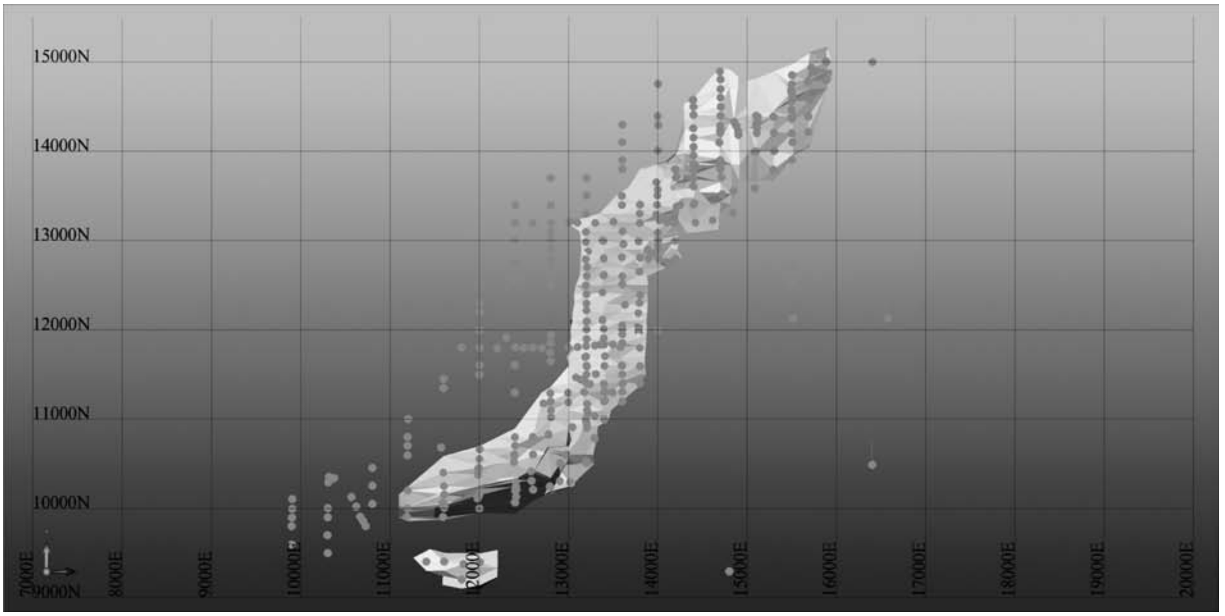
The exploration history is summarised in **Table 7.5.**

Table 7.5 — Cape Lambert Iron Ore Project — Exploration Summary

Year	Activity	Agency	Comments
1994-1996	Drilling and resource estimation	Robe River Mining Company Pty Ltd	Project surrendered
2006-2007	Drilling and resource estimation. Scoping study completed	Cape Lambert Iron Ore Ltd	Positive outcome from scoping study.
2008	Project sold to MCC Corporation	MCCAH	
2008	Drilling and resource estimation. PFScompleted	MCCAH	Positive outcome from PFS, decision to proceed with BFS
2009	Further drilling and resource estimate.	MCCAH	
2009	Tenders called for completion of BFS	MCCAH	

The extent of the deposit and the location of drill holes as at March 2008 are shown in ***Figure 7.2***.

Figure 7.2 — Cape Lambert Iron Ore Project — Drill hole Location Plan



7.5 GEOLOGY

In M-MC's opinion, the overall geological interpretation and mineralisation of the CLIO deposit are well understood.

7.5.1 Regional Geology

The following information was sourced from Golder, 2009.

The Cape Lambert property is located within an Archaean sequence of basic volcanics, felsic volcanic, ultramafics, sediments and cherts and banded iron formation (BIF) rocks. Volumetrically the BIF sequence, known as the Cleaverville Formation, is a subordinate part of the sequence but nevertheless has an estimated thickness ranging from 800m to 1,400m (***Figure 7.3***)

The depositional environment of the Cleaverville Formation is generally considered to be shallow water. The various chert units have been attributed to a number of origins including primary deposition, silicification and weathering. The iron bearing BIF units are however generally considered to be primary.

The Cleaverville Formation is part of the Gorge Creek Group of BIF and clastic sedimentary rocks. The unit is underlain by the predominantly volcanic rock sequence of the Whundo Group with the contact being a possible low angle unconformity. Overlying the Cleaverville Formation are rocks of the Fortescue Group, the basal unit of which is the Mount Roe Basalt that is present in the north-eastern and eastern parts of the Cape Lambert property. From the evidence of the drilling, the area of basalt is more extensive than shown on the published geological maps, as it is present within the eastern part of the tenement. The Mount Roe Basalt unconformably overlies the Cleaverville Formation.

The Archaean sequence in the Cape Lambert property area is intruded by a number of generally small Archaean granitoids. Much of the area, especially towards the coast, has Cainozoic surficial deposits overlying the bedrock sequence.

7.5.2 Local Geology

The following information was sourced from Golder, 2009 and NETC, 2008.

The large-scale structure of the Cape Lambert area is a series of north-east trending synclines and anticlines. The axis of such an anticline is located in the central portion of the Cape Lambert tenement area. The Cleaverville iron deposit is located on the north-western limb of the anticline located near the coast with the main area of drilling by Robe being on the south-eastern limb of the syncline and the north-western limb of the adjacent anticline.

The area is typically overlain by Cainozoic surficial cover up to 6m thick. The local stratigraphy has been gently folded resulting in three defineable zones within the main deposit. These are referred to as Northern, Central and Southern Regions. The local lithology and three mineralised areas are shown in ***Figure 7.3*** and summarised in ***Table 7.6***. The overall dip of the sequence in the drilled Cape Lambert deposit area is easterly at moderate angles.

At a small scale, interpretation of the results from the drilling has demonstrated the presence of a series of synclines and anticlines that have separations between fold axes of around 100m. It is possible to discern these in densely drilled locations where they can be traced in areas of distinct lithological changes but in more monotonous sequences, their presence cannot be proved. It is in general not possible, therefore, to trace such folds over more than short distances although their presence is suspected in much of the sequence with the overall easterly dip.

Geological mapping also resulted in the recognition of small scale folding although few fold axes were shown on the maps. Folding styles were considered to be isoclinal with an overall dome and saddle pattern.

Faulting is known both from the published geology and the more detailed mapping undertaken during the middle 1990's. A dominant fault direction is north-easterly but other directions are northerly and easterly. In the course of interpreting the geology of the drilled area, the presence of faults was suspected in some areas but there is insufficient evidence for the orientation and continuity to be assessed.

The project area comprises felsic volcanic, sedimentary, ultramafic assemblages as well as the iron bearing chert and BIF lithologies.

Table 7.6 Cape Lambert Iron Ore Project — Orebody Characteristics

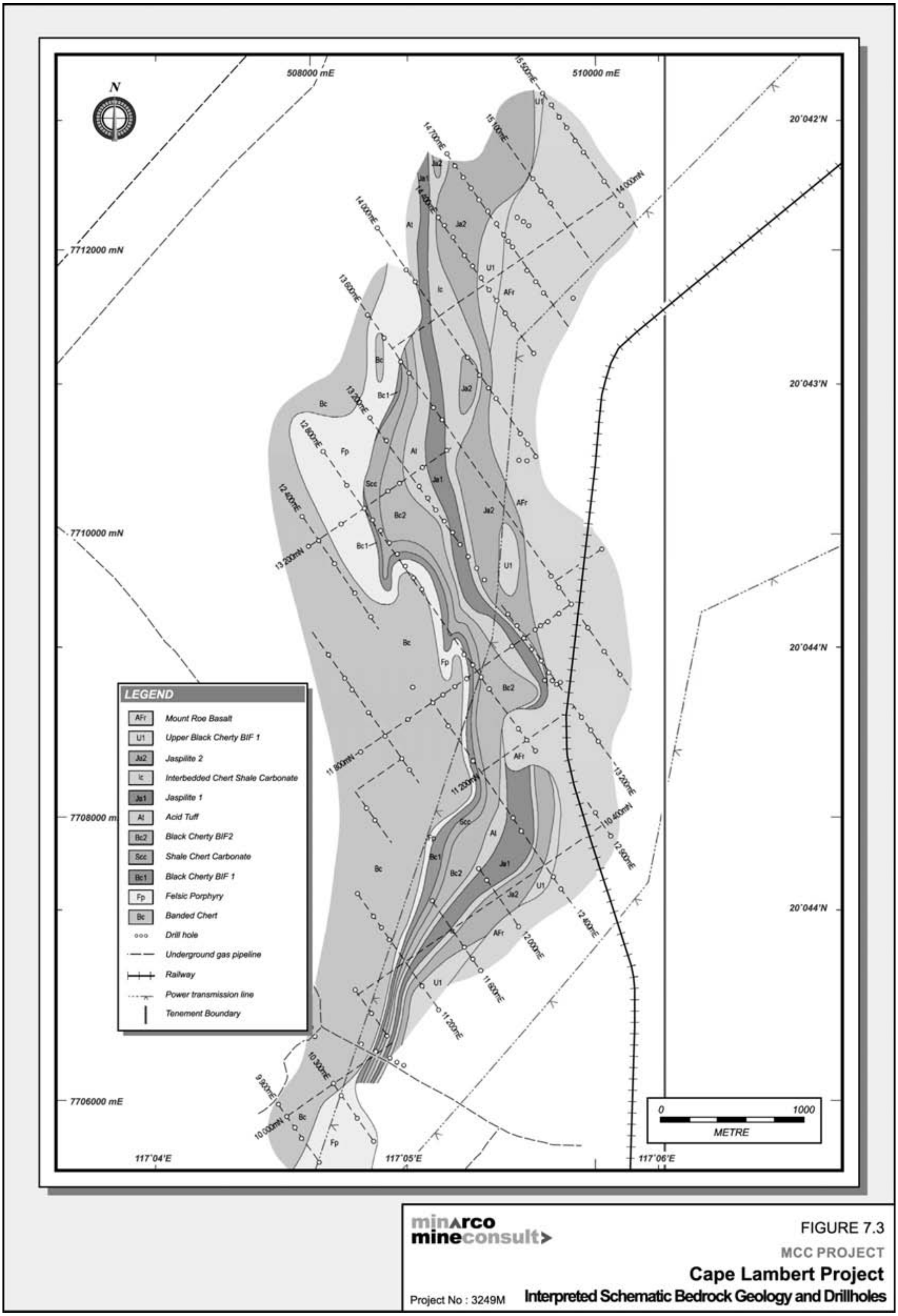
	Dimensions of the Orebody			
Region	**Vertical**	**Length**	**Width**	**Comments**
Northern	100m	1,200m N-S	2,000m E-W	NE strike, SE dipping at 20^o
Central	100m	3,000m N-S	1,400m E-W	NS strike, E dipping at 15-20^o
Southern	150m	2,500m NE-SW	1,500m NW-SE	NE strike, SE dipping at 30^o

Source: NETC 2008

In the deposit area, the BIF and chert horizons are typically 80-100m in thickness. Magnetite is the main iron bearing mineral, with hematite, limonite and goethite also present. Gangue minerals include quartz, carbonate (ankerite) and grunerite.

The grades of Fe and accessory and deleterious elements are relatively uniform throughout the deposit.

Figure 7.3 — Cape Lambert Iron Ore Project — Local Geology and Mineralisation



7.6 RESOURCES AND RESERVES

M-MC has reviewed the resource estimate prepared in 2009 by Golder. The methodology applied to the resource estimate by Golder is generally appropriate and correct for this style of mineralisation. The resource classification applied has been reviewed against the recommendations of the JORC code and appears reasonable. The overall project layout, drill hole location and mineralisation were confirmed by M-MC during a site visit on 7 July 2009.

A total of 377 resource drill holes (83,957m) were completed between 1994 and 2008. The majority of holes were used the reverse circulation (RC) drilling method. A total of 31 holes were core drilled, with RC pre-collars. The drilling summary is shown in ***Table 7.7.***

Table 7.7 — Cape Lambert Iron Ore Project — Drilling Summary

Stage	Holes	Metres	Spacing
1994-1995 Robe Drilling	186	22,505	200m × 120m
2006-2007 Cape Lambert	166	52,849	Variable
2008 MCCAH	25	8,608	Variable
Total	**377**	**83,960**	

Source: Golder 2009

Drilling samples were combined into either 2m or 4m composite samples and analysed for a range of elements using XRF. In addition, samples determined to be within the resource envelope were also analysed using the Davis Tube Recovery (DTR) method. This is a laboratory scale method for determining the grade and recovery volume of magnetic separation products. This was expected to provide an indication of beneficiation performance throughout the resource. The DTR grades and recovery were also estimated by Golder in the2009 Mineral Resource estimate. CLIO and MCCAH used the ALS laboratory in Perth for head and DTR analyses.

Exploration methods and data is summarised in ***Table 7.8.***

Table 7.8 — Cape Lambert Iron Ore Project — Exploration Methods

Exploration Methods	Details of activities	Comments
Drilling	346 RC and 31 precollared core holes	Total 83,960m drilling
Sampling	Samples were taken at 2m or 4m in RC drilling and from 1m to 12m in core drilling.	Approx. 23,600 samples were prepared for geochemical analysis.
Geochemical analysis	Elemental analyses were conducted for Fe, Fe++, Al_2O_3, MgO, TiO_2, SiO_2, S, P, CaO, K_2O, Na_2O, LOI	XRF technique by ALS Laboratories in Perth, WA.
Beneficiation analysis (DTR)	Concentrate grades and LTR were prepared Fe, Fe++, Al_2O_3, MgO, TiO_2, SiO_2, S, P, CaO, K_2O, Na_2O, LOI	Samples with Fe>10% were routinely analysed using DTR
Duplicates and assay checks	Duplicates of every tenth sample were verified by other Laboratories.	All work originally carried out included extensive QAQC and indicated no problems with precision or accuracy.
Specific Gravity analysis	In mineralized zones, 132 density determinations carried out on drill core.	Average specific gravity (Ore Density) was determined as 3.35 **t/m^3**

Source: Basic Design Report 1991

Due to the early stage nature of the project, Ore Reserves cannot be quoted. The results of preliminary mining studies are shown in Section 1.6.2 of this report. All mining studies to date have included Inferred Mineral Resources which will be excluded from any future stated Ore Reserves.

7.6.1 Mineral Resources — In Situ Quantities

The CLIO magnetite deposit is a BIF and chert hosted deposit with medium to low grade Fe. An independent resource estimate for this project has been carried out in March 2009 by the international consulting group Golder Associates in its Perth office.

Drill hole data and resource interpretations were provided to Golder by MCCAH. Golder validated the drill hole database and analysed the QAQC results. Golder determined that the QAQC results were very good, although a number of sample swaps were noted.

The resource interpretations were checked and adjusted by Golder, then used to prepare wireframes within which the mineralised zones were defined. Vulcan software was used for geological modelling. Golder then composited the samples within the wireframes to even 4m intervals then carried out exploratory data analysis and spatial data analysis to identify characteristics and grade trends within the mineralised zones. Golder used its own proprietary statistical software for spatial analysis of the data (variography) which showed relatively low nugget variance and long ranges of continuity for Fe.

Golder separated the deposit in to three main domains (north, central, south) for variography and analysis. The domains were based solely on Fe grade using a 20% Fe threshold to define resource outlines. The characteristics of each domain were found to be similar, with orientation and geometry being the main differences. The different orientations were incorporated into the grade estimation utilising Ordinary Kriging. M-MC has reviewed this estimate and compared it against the underlying drill data and considers the estimate to be a good representation of the mineralisation and grade within the Southern and Northern Zones. Interpolation directions for the Central Zone are not optimal and will have caused incorrect grade assignment on a local basis. The global estimate is unlikely to be materially affected.

Density determinations were carried out on drill core. A total of 132 values were available from the mineralised zones. An average value of 3.35t/m^3 was derived from the data. M-MC considers this reasonable at scoping level, however the data is inadequate for a BFS level evaluation.

The total identified Mineral Resource reported in the 2009 estimate was 1.91Bt at 30.7% Fe (20% Fe cut-off). M-MC has reported the Resource in compliance with the recommendations in the Australasian Code for Reporting of Mineral Resources and Ore Reserves (2004) by the Joint Ore Reserves Committee (JORC). The M-MC estimate of Mineral Resources at the project at June 2009 is summarised in ***Table 7.9.*** Resource classifications defined by Golder have been retained. Due to the good distribution of drilling and very regular Fe grade distribution, M-MC considers this classification appropriate.

Table 7.9 — Cape Lambert Iron Ore Project — Golder Associates Estimated Mineral Resources, as at March 2009

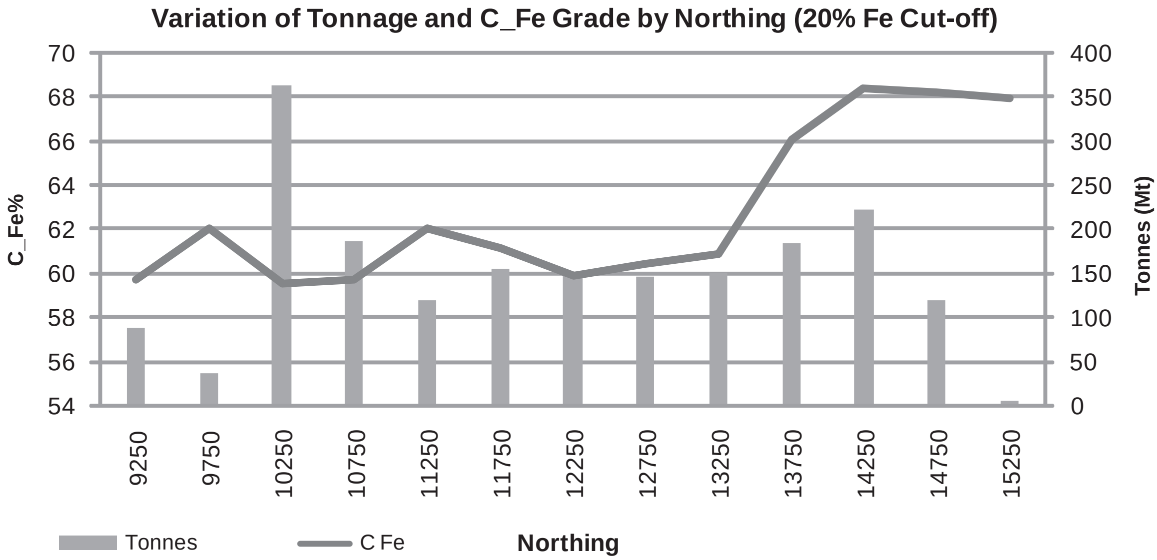
	Head Grade Estimate (20% Fe Cut off Grade)											
JORC Classification	**Tonnes**	**Fe**	**Fe++**	**SiO_2**	**Al_2O_3**	**P_2O_5**	**LOI**	**CaO**	**K_2O**	**MgO**	**S**	**TiO_2**
	Mt	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**
Measured												
Indicated	1,434	30.7	16.0	40.4	2.32	0.03	7.22	2.66	0.19	2.61	0.14	0.17
Inferred	481	30.5	16.0	41.1	2.81	0.03	5.44	3.09	0.28	2.67	0.19	0.20
Total	**1,915**	**30.7**	**16.0**	**40.6**	**2.44**	**0.03**	**6.78**	**2.77**	**0.21**	**2.63**	**0.15**	**0.17**

Source: Golder 2009 Resource Report.

	Concentrate Grade Estimate (20% Fe Cut-off Grade)											
JORC Classification	**DTR**	**Fe**	**Fe++**	**SiO_2**	**Al_2O_3**	**P_2O_5**	**LOI**	**CaO**	**K_2O**	**MgO**	**S**	**TiO_2**
	Rec%	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**
Measured												
Indicated	31.7	61.7	22.0	10.2	0.62	0.01	−.77	0.72	0.05	1.00	0.11	0.08
Inferred	32.2	62.0	22.7	10.4	0.63	0.01	−1.3	0.67	0.05	0.89	0.26	0.09
Total	**31.8**	**61.8**	**22.1**	**10.3**	**0.62**	**0.01**	−0.9	**0.71**	**0.05**	**0.97**	**0.15**	**0.08**

The tonnage and C_Fe grade varies throughout the deposit. M-MC has reported the resource in northing intervals to show this variation. Results are shown in ***Figure 7.4.***

Figure 7.4 — Cape Lambert Iron Ore Project — Variation of Tonnage and C_Fe grades



The resource model coloured by Fe% is shown in ***Figure 7.5***, and the model coloured by C_Fe% is shown in ***Figure 7.6.***

Figure 7.5 — Cape Lambert Iron Ore Project — Resource Model Coloured by Fe% (Looking Northwest)

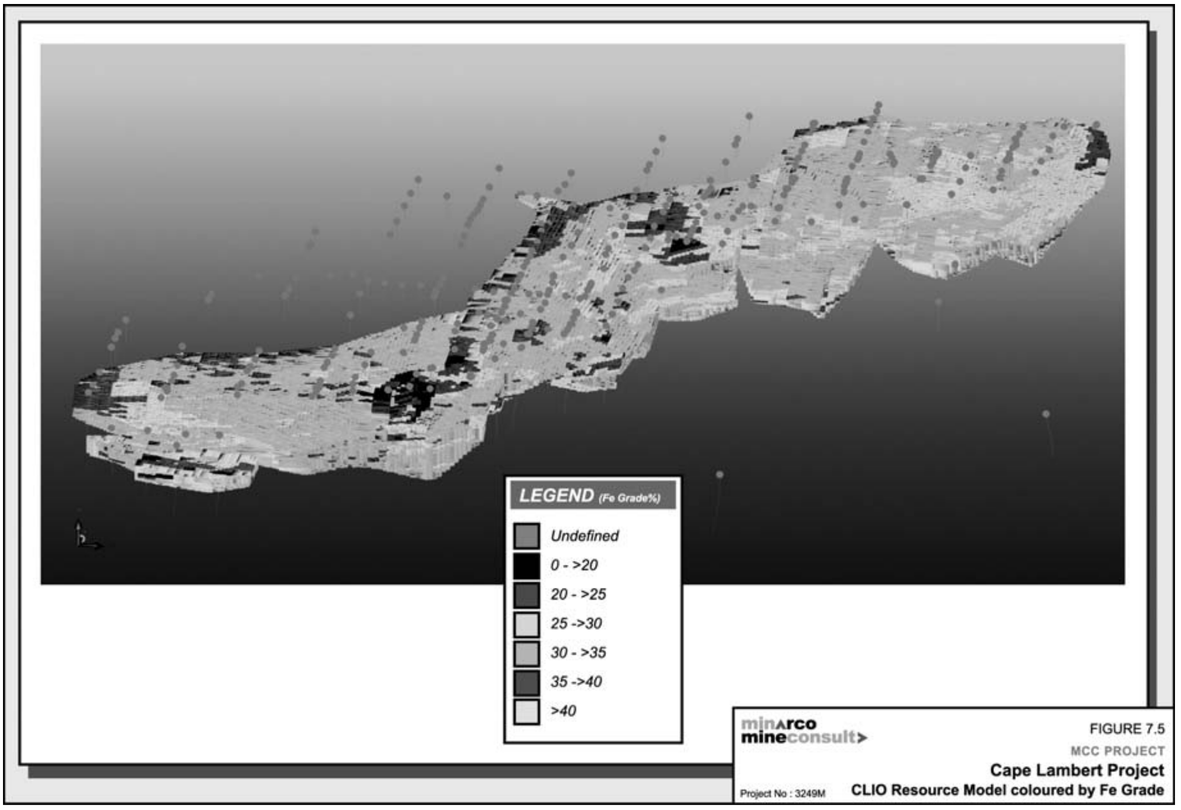
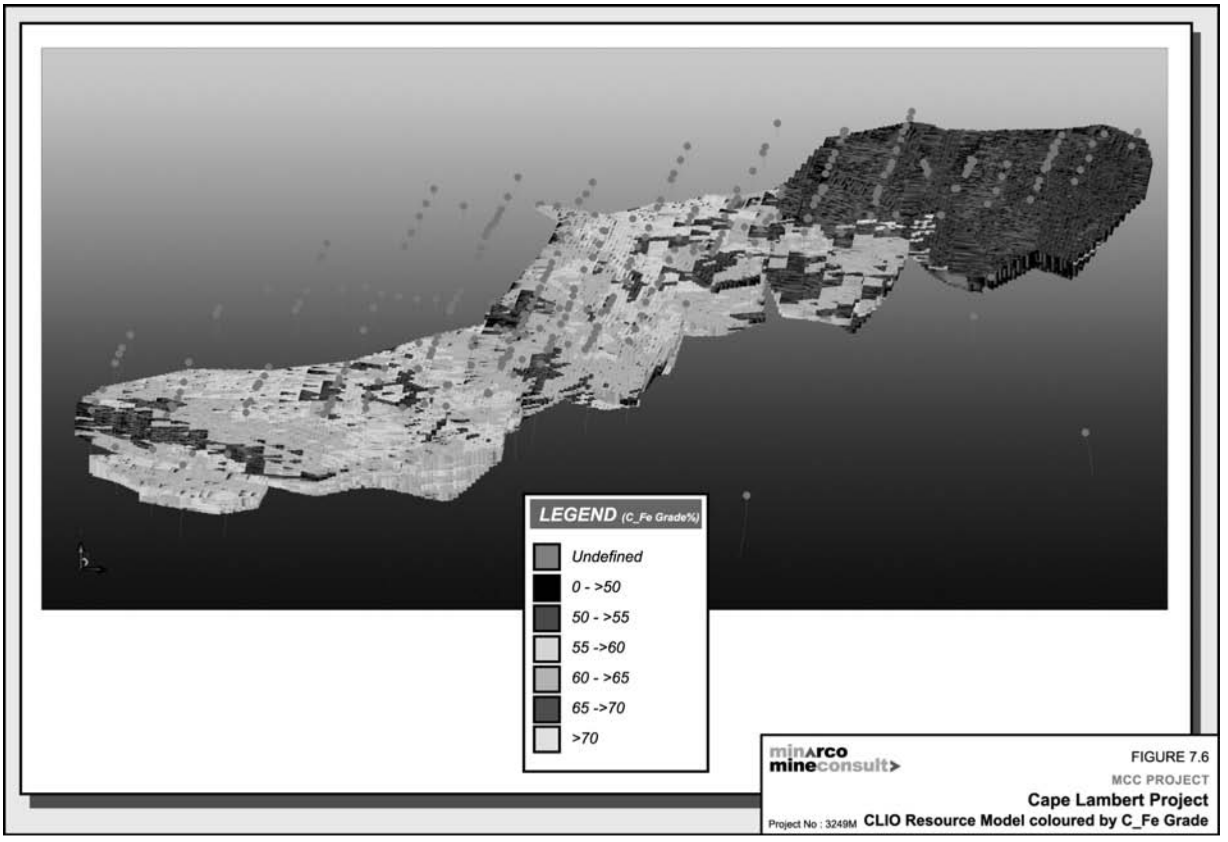


Figure 7.6 — Cape Lambert Iron Ore Project — Resource Model Coloured by C_Fe% (Looking Northwest)



7.6.2 Reserves — Mineable Quantities

At the current stage of the evaluation, Ore Reserves cannot be defined. However mining studies were carried out as part of the 2008 PFS, and these resulted in the preparation of a pit design from within which mineral inventories can be reported. Note that the mineral inventories and all production forecasts include Indicated and Inferred Mineral Resources. Under JORC Inferred Resources cannot be included in any future Ore Reserves stated for the project. For this reason MMC has referred to the Resources within the pit as Mineable Quantities.

The pit design was prepared by NETC as part of the 2008 PFS and utilised the 2008 Golder Mineral Resource model. It was based on pit optimisation carried out by NETC, using the parameters shown in ***Table 7.10***. Lower values for the parameters were used in the financial evaluation analysis regarding a mining cost of AUD$5.09/ROM t, a processing cost of US$8.51 and a concentrate value of AUD$100/t at US$0.80: AUD$. The total operating cost, however, is closer to AUD$22/ROM tonne. A longer term concentrate price (US$60/t) should be considered in all analyses.

Table 7.10 — Cape Lambert Iron Ore Proj ect — 2008 PFS Optimisation Inputs

Item	unit	Quantity
Fe Concentrate Grade	Fe%	65
Fe Concentrate Selling Price	AUD$/t	113.22
Mining Cost	AUD$/t	5.81
Processing Cost	AUD$/t	9.72
Management Cost	AUD$/t	0.60
Shipping Cost of Concentrate	AUD$/t	1.25
Other Financial Cost	AUD$/t	0.18
Overall Pit Slopes	Footwall/North & South/Hangingwall	Max 45°/46°/52°
Dilution/Mining Loss	—	—

Source: NETC 2008 PFS Report

The costs and revenues used in the optimisation are considered by M-MC to be appropriate for an iron ore project of the proposed size operated in the Pilbara region of Western Australia.

The mining inventory is a conceptual estimate of the likely tonnage and grade that could be captured within an ultimate pit design. The design was generated as part of the PFS using assumed design parameters without any geotechnical investigation. Furthermore, no sensitivity analysis, scheduling, blending or other options were considered.

No geotechnical drilling has been carried out in the project area to date. In the absence of actual geotechnical data, the slope parameters used for the optimisation have been calculated from a desktop study based on estimated rock strength properties of the various geological domains. In MM-C's opinion the 52° overall wall angle used for the hangingwall is at the upper end of the range of likely final wall angles, particularly given the uncertainty surrounding the depth of weathering of the rock profile.

No statement is made in the 2008 PFS that any sensitivity studies have been carried out on inputs likely to impact the optimised pit shell, such as selling price, exchange rate or wall angles.

The mineral inventory for the design produced from the floating cone study is shown in Table 7.11.

Table 7.11 — Cape Lambert Iron Ore Project — 2008 PFS Pit Design Mineable Quantities

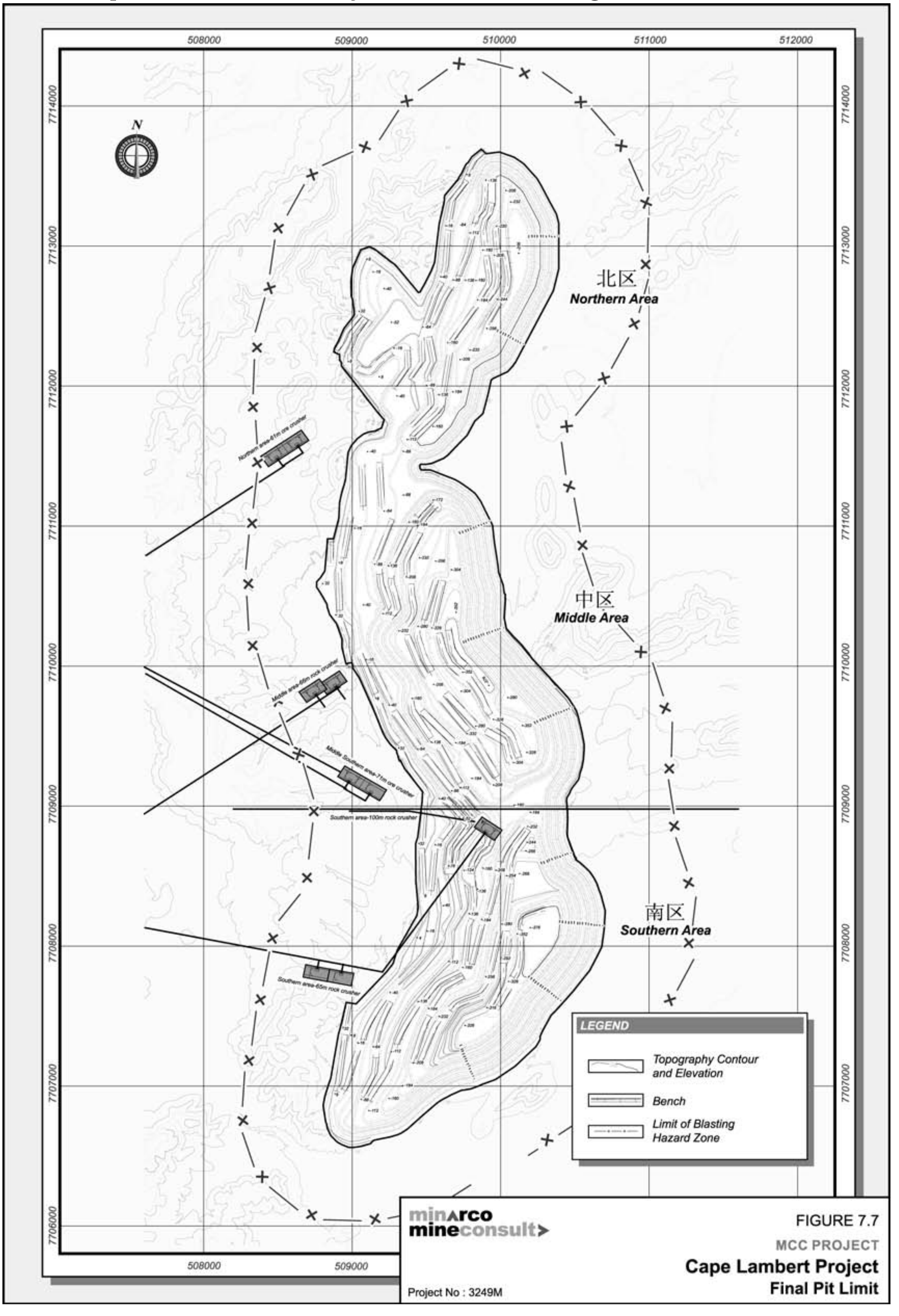
Region	Ore (Mt)	Waste (Mt)	Strip Ratio
Southern	606	989	1.63
Middle	486	1,146	2.36
Northern	221	569	2.58
Total	**1,313**	**2,704**	**2.06**

Source: NETC 2008 PFS Report
Note: Mineable Quantities composed of Inferred and Indicated JORC Resources. These are not JORC Reserves.

The design has been split into three regions which are planned to be mined sequentially, starting in the south and progressing towards the north on the basis that resource confidence is highest and strip ratio is lowest in the southern region of the deposit.

The pit design is shown in ***Figure 7.7***.

Figure 7.7 — Cape Lambert Iron Ore Project — 2008 PFS Pit Design Plan



The parameters used to design the pit are shown in ***Table 7.12***.

Table 7.12 — Cape Lambert Iron Ore Project — 2008 PFS Pit Design Parameters

Parameter	Value	Units
Bench Height	12	Metres
Berm Interval	24/36	Metres
Berm Width	5/14	Metres
Operating Face Angle	65	Degrees
Final Face Angle	70	Degrees
Haul Road Width	33	Metres
Ramp Gradient	8	Percent

Source: NETC 2008 PFS Report

The bench height, berm width and berm intervals result in an overall slope angle of approximately 52^o as used for the optimisation input. The eastern wall of the pit is designed to this configuration, whilst the western wall follows the flatter dip slope of the orebody. The road and ramp design parameters are appropriate for the size of the proposed truck fleet.

The ramp system incorporates a large number of switchbacks down the western wall of the pit to access the lowest benches of the pit. Given the flat wall angle used on the western side of the pit, there is potential to use a straighter ramp system with likely little increase in stripping requirements. A ramp system utilising less switch backs will be safer, will reduce wear on the truck fleet and will increase truck productivity due to higher average speeds. Potentially a redesigned ramp system could also reduce the number of crusher moves required by the current development strategy.

7.7 MINING

Cape Lambert will be a conventional open cut mine utilising a combination of truck/shovel and mobile crushing and conveying methods. Mining is to be carried out utilising contractors.

The overall development strategy is to commence mining in the southern area of the pit and then progress successively to the middle and northern areas. Initially waste will be placed in an ex-pit dump, however as development allows, waste will be backfilled into mining voids. Given this strategy, consideration needs to be given to ensuring that in-pit backfill is not sterilising potential future resource.

Ore will be hauled to semi-mobile crushing plants located either in-pit or immediately adjacent to the pit. Once crushed, the ore will be conveyed to the beneficiation plant for further processing. As mining progresses north, the crushers will be progressively relocated further north adjacent to active mining areas.

Waste will be hauled to semi-mobile crushing plants located near to those used to crush ore. The crushed waste will be conveyed to either external pits or in pit voids as the development schedule allows.

The decision to utilise a combination of hauling and crushing/conveying for waste movement is based on an economic comparison carried out in the 2008 PFS. Hauling waste directly to dumps using trucks was compared with hauling waste to nearby crushers and then conveying the crushed waste to dumps. This comparison found the latter option to be the most cost effective. Insufficient detail exists within the 2008 PFS document to determine the validity of analysis.

The load and haul fleet is proposed to consist of six 34m^3 hydraulic face shovels and twenty one 365t haul trucks. In MM-C's opinion this fleet combination will provide sufficient capacity to achieve the required total movement initially. However it is MM-C's belief that a larger truck fleet than anticipated will be required to maintain total movement targets once deeper sections of the pit are being mined. Further investigation is recommended.

Drill and blast is to be carried out utilising 6 drill rigs each drilling 100,000m/yr. Ore patterns will be drilled and blasted on a 7m by 8m pattern, whilst waste will be drilled on an 8m by 10m pattern. All patterns will use a 2m sub-drill. Sufficient drilling capacity has been allowed for using these pattern sizes to achieve the planned total movement requirements.

An appropriate amount of ancillary equipment including graders, water trucks, refuelling trucks, dozers, explosives trucks and other minor items has been allowed for to support mining operations.

No mention of planned grade control practices is made in the 2008 PFS document.

7.7.1 Forecast Production

Approximately two years of pre-strip is planned to take place at Cape Lambert. A proportion of the waste mined during this period will be used for site construction projects with the remainder being sent to an external waste dump located to west of the southern mining area. Ore will be primarily sourced from the southern mining area up until Year 16. From Year 16 through to Year 26 the main ore source will be the middle mining area, with waste being sent ex-pit until mining is completed in the south at which point the southern mining void will be backfilled. After Year 26, production will be derived from the northern mining area. Waste will be dumped ex-pit until final pit limits are reached in the middle mining area allowing in-pit backfill to commence.

The forecast mining schedule is shown in ***Table 7.13***.

Table 7.13 — Cape Lambert Iron Ore Project — 2008 PFS Mining Schedule

Item	unit	Year 0	Year 1	Year 2	Year 3-15	Year 16-22	Year 23-26	Year 27	Year 28	Year 29	Year 30	Total
Ore	**Mt**	0	0	16	48	48	48	46	46	37	16	1,313
Waste	**Mt**	45	127	114	82	101	115	88	74	21	7	2,703
Total	**Mt**	45	127	130	130	149	163	134	120	58	23	4,016
S/R	**t/t**	n/a	n/a	7.1	1.7	2.1	2.4	1.9	1.6	0.6	0.4	2.06

Source: NETC 2008 PFS

At full production, 15Mt of iron concentrate grading 65% Fe is planned to be produced. The concentrate grade has been assumed to remain constant throughout the life of the project, however the DTR results used to estimate concentrate recovery and grade in the resource model shows a total deposit average C_Fe grade of 61.8%. The C_Fe% grade varies from around 60% in the southern portion of the deposit to around 68% in the northern portion. The mass recovery of 31.8% demonstrated by the DTR results supports the project life of mine assumption of 31.25%.

7.8 MINERAL PROCESSING

Overview

The processing flowsheet is at an advanced conceptual stage, with possibly at least two more iteration's before it is frozen. This would mainly involve the silica rejection circuit and possibly the comminution circuit. Sufficient testing of an adequate standard has been undertaken for this stage of the project using reasonably representative ore samples of the proposed mining area. The measured ore characteristics as well as process testing results support the suitability of the proposed flowsheet as well as the interpreted metallurgical performance.

A Pre-Scoping Study (Met-Chem), focussed on the development of the flowsheet and a Scoping Study (MetPlant Engineering Services), developing capital and operating costs, have been conducted. This work was re-examined and updated in the NETC 2008 PFS.

The processing operation at Cape Lambert would be located near the mining operation within company owned leases. The process used to produce a marketable magnetite concentrate consists of two stages where the first stage would recover magnetite-silica composites from unwanted material. A standard magnetic separation process is proposed to produce a magnetic rich concentrate. This concentrate would be upgraded in the second stage by removing the silica as a concentrate using a standard flotation process. Further testing will identify any need for additional cleaning technologies. The tailings from this operation would be the final product and would be dewatered for transport. Both the magnetic separation tailings and the silica flotation concentrate would be pumped for storage in a tailings dam. The proposed 2008 PFS flowsheet is presented in ***Figure 7.8***.

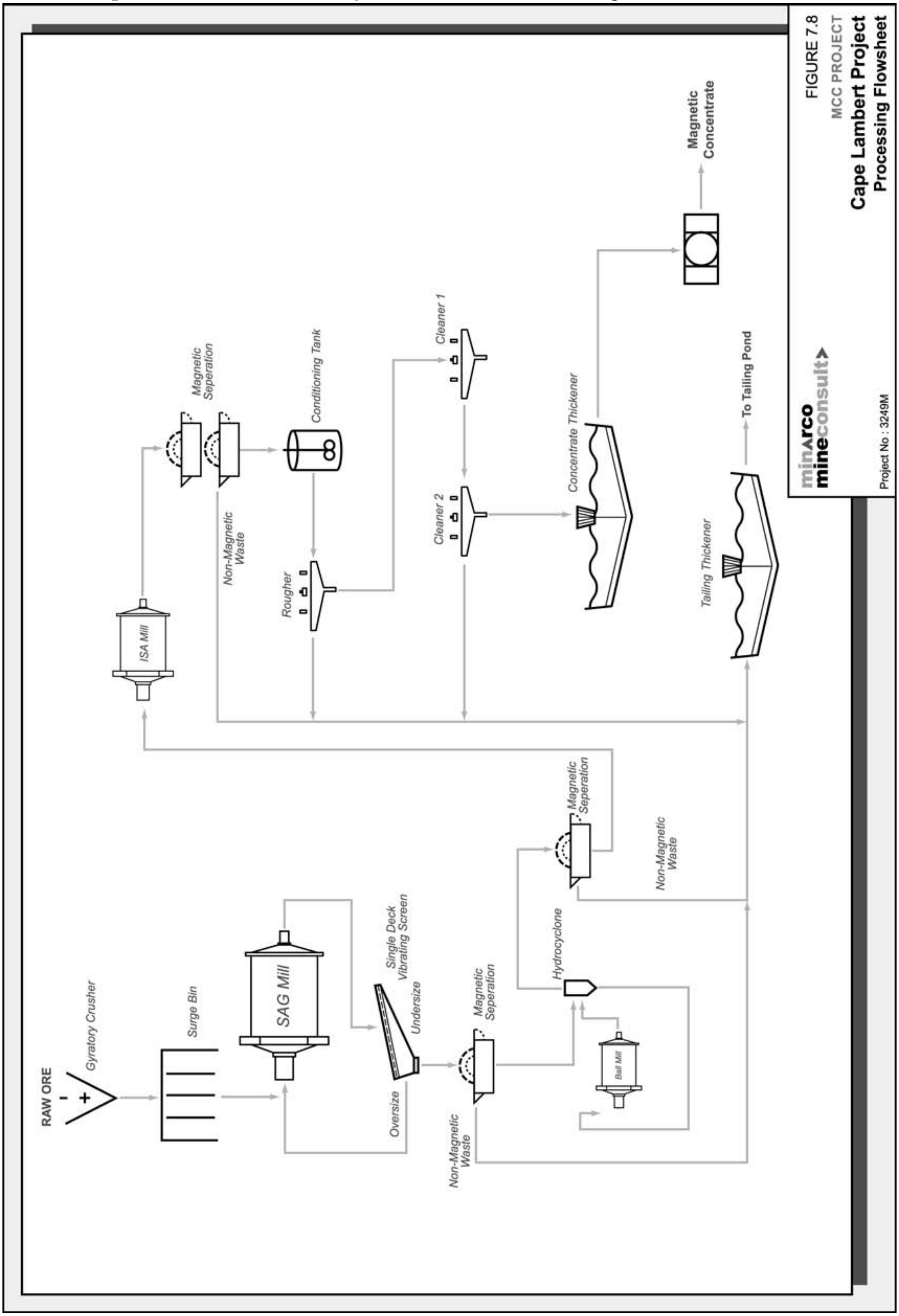
The proposed Cape Lambert processing facility will be a very large operation when in full production; processing 48 million tonnes of ore annually to yield 15 million tonnes of magnetite concentrate (refer to ***Table 7.14***). Overall, an iron recovery of 68.8% is projected into a magnetite concentrate containing 65% iron and weighing 31.25% of the original feed. The 2008 PFS anticipates that the received magnetite concentrate value would be USD$80/tonne over the first three of production (2012 to 2014), indicating a nominal annual revenue of USD$120 million (AUD$150 million).

Table 7.14 — Cape Lambert Iron Ore Project — Forecast Production

Material	Unit	2012	2013	2014
Run-Of-Mine (ROM)				
Production	t	16,000,000	32,000,000	48,000,000
Grade	% Fe	29.54	29.54	29.54
Magnetite Concentrate				
Production	t	5,000,000	10,000,000	15,000,000
Grade	% Fe	65.00	65.00	65.00
Total Iron Recovery	%	68.80	68.80	68.80
Mass Recovery	%	31.25	31.25	31.25
Forecast Price (exclude. Taxes)	US$/t conc	80	80	80

Source: Provided by CLIO, June, 2009.

Figure 7.8 — Cape Lambert Iron Ore Project — 2008 PFS Processing Flowsheet



Mineralogy

The Cleaverville Formation, which constitutes the deposit, consists of 15 stratigraphic units of which four main units host iron contents greater than 25% Fe. These units are identified by the waste mineral type, such as Jaspilite 1 and 2 and Black Cherty Banded Iron Formation (BIF) 1 and 2. Magnetite is the dominant mineral present and typically several microns in size and intergrown with mainly silica minerals. Ore near the surface also contains iron as hematite, siderite, goethite and limonite, which range in magnetic properties from moderate to weak and are not directly recovered during the recovery of magnetite.

The predominate gangue or unwanted mineral is quartz followed by carbonates, ankerite and grunerite. A feature of the ore types at Cape Lambert are the low phosphorus, sulphur, alumina and magnesia contents. Both drill cores and composite samples were used in testing, typically assaying 32.6% Fe, 38.9% Si and 26.3% magnetite. An interesting feature of the ore types was the Loss On Ignition (LOI), which can be relatively high due to the presence of iron carbonate (siderite). It was noted that the northern area ore types displayed higher LOI's and thus contained a higher quantity of siderite.

The iron and silica content of the drill core was found to increase with depth and makes the ore harder. At greater depths, it is likely that quartz would be the dominant form of silica. Harder ores would mean a small increase in grinding and media costs however higher grade magnetite concentrates.

Testing

The testing is of a suitable standard and scope for the current level of study. Both drill core and composite samples were used in testing and appear to be representative of the potential mined ores. Basic parameters such as mineralogical and physical properties have been identified as well as magnetic and flotation process options. Similar ore process responses and process flowsheet requirements were reported by all of the testing facilities.

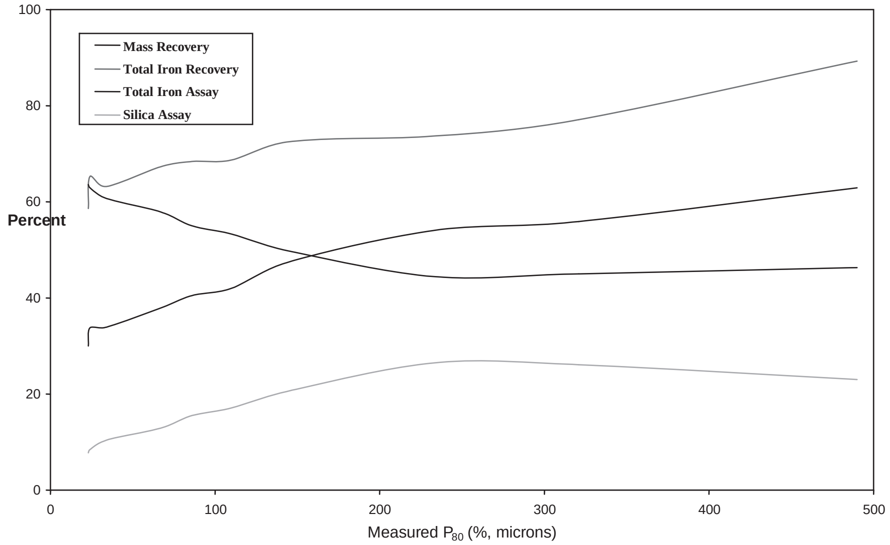
More testing would be required, particularly the determination of the dewatering properties of the tailings and magnetite concentrate streams. Fine tuning of the final milling circuit configuration in terms of equipment types (e.g. HPGR, SABC, etc.) and milling parameters through testing and milling simulations would be required in the next stages of the project development.

Ore type characterisation and the subsequent variation in process response deserves further investigation, particularly with respect to the non-quartz silica content. Confirmation of the capacities of various pieces of equipment, such as that of the third stage of magnetic separation, would be determined.

Previous testing on ores from this deposit was conducted by the Robe River Mining Company between 1994 and 1997. This work consisted of Davis Tube testing followed by flotation with RC and diamond drill core samples. A fine grind (20 microns) was needed to achieve a reasonable magnetite-silica separation, while the silica content could be decreased to below 5% by flotation of the magnetite concentrate.

More recent testing determined the mineralogical properties of a number drill cores as well as determining the physical properties (METS, March 2009). CLIO ore types were found to be hard and abrasive. Davies Tube testing was conducted where the iron and silica content as function of size and after treatment in a Davis tube to produce a magnetic concentrate (refer to ***Figure 7.9***). It was noted that to achieve any increase in the iron content, grinding to below 200 microns is required and that to reach a marketable concentrate grade at a relatively high recovery is only possible below 25 microns.

Figure 7.9 — Cape Lambert Iron Ore Project — Davis Tube Testing



Pneumatic flotation testing was recently conducted by Maelgwyn Mineral Services, examining the chemical and flotation environment for the production of a marketable magnetite concentrate. Initial testing indicated that a relatively low silica magnetite concentrate at reasonable iron recoveries was possible. More flotation testing was recommended. As an equipment supplier, this type of testing provides a basis for both equipment and process performance guarantees.

Flowsheet

The flowsheet is depicted in ***Figure 7.8***. It depicts a classical and well established processing approach to the production of a magnetite rich concentrate from a fine grained admixture of magnetite and silica. The selected processing approach is reasonable based on the ores tested and practice elsewhere in the world.

To achieve high iron concentrate grades, these ore types need to be finely ground to liberate the individual magnetite grains, necessitating a large quantity of milling capacity. Sequential size reduction would be employed, where after each stage of grinding, magnetite separation is used to recover magnetite and discard unwanted material. This approach allows the use of smaller equipment with each subsequent processing stage by rejecting unwanted material early as possible in the processing flowsheet. This results in lower capital and operating costs.

ROM ore (F_{100}=1,200mm) would be reduced to 80% passing 175mm by two 60" by 89" mobile gyratory crushers and stored on a stockpile with 410kt live capacity (2.8 days of production). Ore from the stockpile would be fed to five parallel processing lines each consisting of a 36' Ø by 22' SAG mill (16MW motor and P_{80}=850microns) and four primary magnetic drum separators (7.5kW) to recover magnetite from the undersize stream. The oversize stream as well as the primary magnetic separator tailings would report to a 7.2m Ø ball mill (14.5MW motor and P_{80}=850microns) where the overflow would be upgraded in a secondary magnetic drum separator and the concentrate deslimed in 16m Ø vessel (65% solids underflow). The underflow would be further reduced in size using an ISA fine grinding mill (2.6 MW motor and P_{80}=20microns) and processed with four tertiary magnetic double drum separators (7.5kW).

The tertiary magnetic concentrates would report to a pneumatic flotation separation stage consisting of three parallel circuits with a roughing and two cleaning stages where the silica is removed as a concentrate. The final concentrate would be dewatered to 65% solids in two 70m Ø thickeners and stored in two agitated 19m Ø x 21m tanks before pumping 8 km to the filtration facility located near (0.7 km) the port. Sixteen filter presses would reduce the moisture content of the magnetite concentrate to 10% for storage on the concentrate stockpile (436.2kt and 9 days production). Concentrate would be recovered by a reclaimer (5,000tph) and transported by conveyor for to the load-out facility where barges would deliver the product to ships.

The final tailings, consisting of all the primary magnetic separator tailings, deslimer overflow and the second cleaner flotation concentrate would be dewatered in three 80m Ø thickeners to 57% solids and pumped for storage in tailings dam.

The technical challenges may be summarised as follows:

- A very large scale operation where the size of some of the separation circuits may prove challenging (e.g. fine grinding, filtration,...)
- The silica removal flotation circuit may prove a challenge when quartz makes up less of the silica content

Achieving design metallurgy would be the first task upon commissioning the operating plant and ramping up to full production over the first three years. Assuming no significant design or construction fault is encountered during this time, a better understanding of the equipment performance as well as the processing flowsheet would lead to improvements in throughput and concentrate grade. Ideally over this time, the ore blend produced by mining would exhibit reasonably constant physical and chemical properties, limiting the tasks to mainly fine tuning, such as training, procedures, small equipment upgrades or re-directing piping.

Equipment Selection

The 2008 PFS has selected predominantly western equipment, which trades proven reliability against capital costs. Based on M-MC's experience with projects where MCC is actively involved, further studies would be focused on both finalising technological issues and significantly driving down both capital and operating costs. It is likely that many of the technologies can be replaced by Chinese manufactured equipment with proven performance records, which would significantly lower capital costs without sacrificing too much productivity or process performance.

In the 2008 PFS, established comminution technologies, such as crushers, screens, hydrocyclones and particularly SAG and Ball mills, are employed which are well understood and can be correctly sized. The potential ISA mills also have a demonstrated operational track record, proving to be an effective technology. It is, however, expensive technology and is bundled with Intellectual Property matters. There a number of competitive fine grinding technologies, such as tower mills, that may offer a more satisfactory and lower capital cost solution.

In later studies, it would be expected that the SAG mills would be replaced by ball milling, which is a comminution technology that is more familiar with Chinese operators and manufacturers. This may require the introduction of another crushing stage however overall may lower both capital and operating costs.

The use of pneumatic flotation is a good process equipment choice for the flotation separation of the silica from magnetite. While some mixing is required during the flotation separation, excessive agitation (e.g. mechanical) can affect the quality of this type of separation. The proposed equipment is proprietary German technology and offers a number of features. A competitive and better known technology is column flotation, which may offer similar performance at lower costs.

Process Processing Opportunities

Three processing opportunities are proposed, with the ability to improve the concentrate grade being the most attractive during the first three years of production; thereafter, recovering the non-magnetic iron oxides may become attractive:

- The ability to increase the concentrate grade by 1 or 2% iron may improve both the marketability and value of the magnetite concentrate. This would appear achievable, however probably at the expense of iron recovery.
- Conversely, improving iron recovery would most likely decrease concentrate grade.
- After the primary goal of magnetite production has been achieved; consideration of recovering the non-magnetic iron oxides into a saleable concentrate. Two standard processing options may be suitable and would require confirmation by testing. Only moderate capital expenditure would be required due to the mineral liberation that occurred during the recovery of the magnetite. This process opportunity would enhance both revenues and project economics.

Potential Processing Risks

Besides the standard processing risks, such as the representivity of ore samples, variability of potential ore types as well as the potential comminution and dewatering circuit performances, a major risk is associated with the silica rejection stage. If insufficient silica is removed, the quality of the final magnetite concentrate would be decreased in terms of a lower iron content and an increased quantity of silica. This would not only lower revenue in some ore types but could also result in an unsalable concentrate being produced.

The success of this flotation separation stage is based on the presence of the silica as quartz, since other forms of silica do not respond satisfactorily. It would appear that silica occurs as a number of non-floatable minerals in a number of ore types, thus limiting the amount of silica that can be rejected by this separation technique. Further research would be focused on characterising these ore types in terms of the number and mineable quantities as well as physical properties, variability and process response. Low mineable quantities would mean that blending may well solve this problem. Larger mineable quantities of a number of troublesome ore types would require the investigation of other separation techniques, such a flotation, de-sliming and magnetic separation.

7.9 INFRASTRUCTURE AND SERVICES

The Cape Lambert Iron Ore Project would require the installation of a significant amount of infrastructure, namely a power plant, a desalination plant, port facilities, a camp site as well as the buildings, offices and warehouses. Another aspect of the project development is the relocation of existing infrastructure.

The estimated drawn power load of 255.5MW would be supplied by a 369MW (300MW nominal capacity) gas fired power plant consisting of five CCGT and 2 OCGT units. The breakdown of the required power distribution is shown in ***Table 7.15***. This is a sensible approach to the provision of power since massive gas resources as well as pipelines are located very close to the project. Natural gas provides the best opportunity to supply significant quantities of low cost energy with a low carbon footprint. The estimated power cost of US$120/MW is a little high however probably realistic if the recovery of capital is included.

Table 7.15 — Cape Lambert Iron Ore Project — Estimated Power Requirements

Project Area	Unit	Installed Power	Drawn Power
Mining Operation	MW	55.19	39.26
Processing Operation	MW	254.45	205.66
Filtration Operation	MW	14.47	10.59
Total	**MW**	**324.11**	**255.51**

Source: CLIO Preliminary Pre-Feasibility Study, June 2008.

It is proposed to build a desalination plant located in the vicinity of the concentrate filtration facility to supply water of a suitable quality to the operation. The project water requirements including water quality by area was not disclosed in the 2008 PFS. It may be possible to reduce the size of the desalination plant since the primary demand for high quality water would be for flotation and personnel needs.

A number of options have been considered with regard to the port and transport options. It is proposed to build a temporary port near the concentrate filtration and storage area (4.5km north of the mine) and barge the concentrates to an off-shore ship for delivery to the market.

A residential area is proposed 5.5km to the southwest of the processing facility for 800 people. Houses will be provided for permanent staff while portable housing would be utilised for peak labour periods, such as during maintenance and project closure. A high quality residential environment and facilities is planned to attract and retain staff.

A number of roads allowing site and internal access are proposed. Permanent access to the project facilities would be constructed from the nearby Dampier-Roebourne highway. The 15km road to the concentrate filtration facility would require water culverts, particularly when crossing the tidal flats.

Three pieces of existing infrastructure are required to be re-located, namely a railway line, two power lines and a gas pipeline. Discussions with the infrastructure owners have been initiated and an associated scope of work outlined. Confirmation drilling is planned to confirm the new railway line route while the pathway for the power lines has been determined. With Horizon Power's approval, 20km of 220kV and 12km of 132kV powers lines would need to be relocated. More work is required concerning the re-location of the buried gas pipelines.

The cost of these infrastructure re-locations does not appear to have been included in the capital cost estimate supplied by MCC.

The assumptions for the tailings dam requirements appear to be satisfactory for this level of study; a number of geological and hydraulic studies would be conducted before the final location, design and capital costs can be forecast more accurately.

7.10 CAPITAL AND OPERATING COSTS

The estimated capital costs based on the use of Western manufactured equipment under remotely located Western Australian construction and installation costs to an appropriate standard for an operation producing and processing 48Mtpa of ore are shown in ***Table 7.16***. The total project capital cost is estimated to be AUD$3,773 million. The cost is dominated by the processing plant costs, representing 38.44% of the total investment cost. Another 10.97% would be required to dewater, store, reclaim and ship the magnetite concentrate for export. The proposed infrastructure also would be a significant cost centre, with 25.54% needed for the power station and desalination plant.

Table 7.16 — Cape Lambert Iron Ore Project — Estimated Capital Costs

Cost Centre	AUD (million)	Percent
Auxiliary Mining Facilities	175.45	4.66
Processing Plant	1,450.50	38.44
Tailings Dam	61.10	1.62
Concentrate Handling	210.17	5.58
General Infrastructure	59.44	1.56
Harbour	203.62	5.39
Power Station	587.59	15.57
Desalination Plant	376.01	9.97
Campsite	144.57	3.81
Pre-Construction	504.66	13.38
Total	**3,773.10**	**100**

Source: CLIO Pre-Feasibility Study, June 2009.

The breakdown of this proposed capital expenditure in terms of project activities is shown in ***Table 7.17***. The dominant area of capital expenditure requiring 40.5% of the total investment is the purchase of equipment, followed by construction and installation costs at 34.49%. This figure also includes a rather small contingency of 17.28%, which is low for an engineering study at this early stage (typically 30 to 50%). Additionally, the design management cost, which presumably includes construction, installation, commissioning and training costs, is reasonable at 7.5% of the total project costs.

Table 7.17 — Cape Lambert Iron Ore Project — Activity Capital Cost Breakdown

Cost Centre	AUD (million)	Percent
Construction	1,053.22	27.91
Installation	300.90	7.95
Equipment	1,364.56	36.17
Design Management	212.20	5.62
Other	445.71	11.81
Contingency	396.51	10.51
Total	**3,773.10**	**100**

Source: CLIO Pre-Feasibility Study, June 2009.

There is only a modest small capital allocation (AUD $175.45 million) for mining, indicating that the mining would be contracted out and be expensed as an operating cost. This philosophy may change during the final stages of the project development and if ***'owner operated'*** mining is preferred, then it would be expected that substantial capital (circa AUD$750 million) would be required to acquire the appropriate mining fleet.

As noted earlier, the application of the Chinese project approach would be expected to significantly lower capital costs and possibly operating costs. This would consist of sourcing Chinese equipment with proven performance records and is some cases, changing the equipment selection. For example, it may be expected that the SAG mills would be replaced by Ball mills, which is a comminution technology that is more familiar with Chinese operators and manufacturers. This may require the introduction of another crushing stage however overall may lower both capital and operating costs.

The proposed expenditure for purchase of the equipment and subsequent construction, installation and management costs is spread over four years, with an initial outlay of AUD$500 million during 2010 (refer to

Table 7.18: AUD$1 = USD$0.80). The estimate is based on construction starting sometime in 2009 and this is unlikely to be the case. It is possible that the project timetable may be delayed by one or two years, depending on the speed with which necessary licenses and approvals are processed by regulatory agencies and other interest groups.

Table 7.18 — Cape Lambert Iron Ore Project — Capital Cost Expenditure Timetable

Cost (AUD million)	2010	2011	2012	2013
Expenditure	500	1,500	1,500	277
Total	**500**	**2,000**	**3,500**	**3,777**

Source: Provided by CLIO, July 2009.

The estimated operating costs at full production would be AUD$18.42/ROM tonne as presented in ***Table 7.19***. The processing costs dominate operating costs (45.30%) followed by mining costs (30.30%). It appears that the power costs for the processing have not been included, some, AUD $3.50 /ROM tonne, based on AUD $120MWh. This suggests that the overall operating cost would be around AUD $21-22/ROM tonne, which is reasonable.

Table 7.19 — Cape Lambert Iron Ore Project — Estimated Operating Cost

Cost Centre	2012	AUD/ROM t 2013	2014	Percent
Mining	**5.565**	**5.565**	**5.565**	**30.30**
Auxiliary Material	N/A	N/A	N/A	
Water & Power	N/A	N/A	N/A	
Labour	N/A	N/A	N/A	
Maintenance	N/A	N/A	N/A	
Other	N/A	N/A	N/A	
Processing	**8.32**	**8.32**	**8.32**	**45.30**
Consumables	3.888	3.888	3.888	
Water & Power	3.004	2.60	2.60	
Labour	0.389	0.35	0.35	
Maintenance	1.04	0.53	0.53	
Depreciation	N/A	N/A	N/A	
Other	0.1	0.1	0.1	
Resource tax (6% of income)	**1.88**	**1.88**	**1.88**	**24.40**
Administration	**0.6**	**0.6**	**0.6**	(at full
Other	**1.64**	**1.64**	**1.64**	production)
Sale Expense (exclude. Transport**)**	**0.48**	**0.48**	**0.53**	
Total	**18.37**	**18.37**	**18.42**	**100**

Source: CLIO Pre-Feasibility Study, June 2009.

7.11 SAFETY AND ENVIRONMENT

The mining site is located approximately 20km east of Karratha in the northwest of West Australia. This area is an established iron ore mining and shipping region. The environmental and social aspects of the Cape Lambert Iron Ore project are based on a raft of Australian and Western Australian legislations. These regulations provide guidelines in terms of acceptable standards with respect to indigenous, health, safety, water, conservation, contamination, soil and wildlife matters. In addition, due cognisance have been paid to climatic (e.g. cyclones) and fire safety factors in the design and operation of all facilities.

The primary ecological impact sand pollution sources have been identified as:

- Those arising from the construction land use
- Surface water
- Air pollution (dust from ore crushing, storage and hauling and drying slopes of tailings dam) and waste gas (power station)
- Environmental impact of waste rock and tailings dam material
- Sound pollution (crusher, fans and trucks)

Measures for the control and management of water around the site have been proposed while traditional solutions to the dust, waste gas and sound pollution issues would be applied.

The proposed investment for environmental protection is 9% of the total investment or AUD$248.73 million. The breakdown of this proposed expenditure is shown in ***Table 7.20***.

Table 7.20 — Cape Lambert Iron Ore Project — Environmental Protection Costs

Cost Centre	AUD (million)	Percent
Land & Ecology Restoration	115.20	46.3
Process Water System	29.30	11.8
Tailings Dam	53.89	21.7
Dust Suppression	3.74	1.5
Sewerage Disposal	4.74	1.9
Noise Control	6.50	2.6
Environmental Monitoring	0.85	0.3
Landscaping	34.51	13.9
Total	**248.73**	**100**

Source: CLIO Preliminary Pre-Feasibility Study, June 2008.

No reference was made to the Environmental Impact Statement (EIS) which underpins all site development. This document would describe all the environmental management practices and potential remedies, including remediation of the site after completion of the project. Additionally, a number of agreements and permits, such as the gas supply agreement, the port facility and the location of the desalination plant, would need to be obtained.

7.12 REFERENCES

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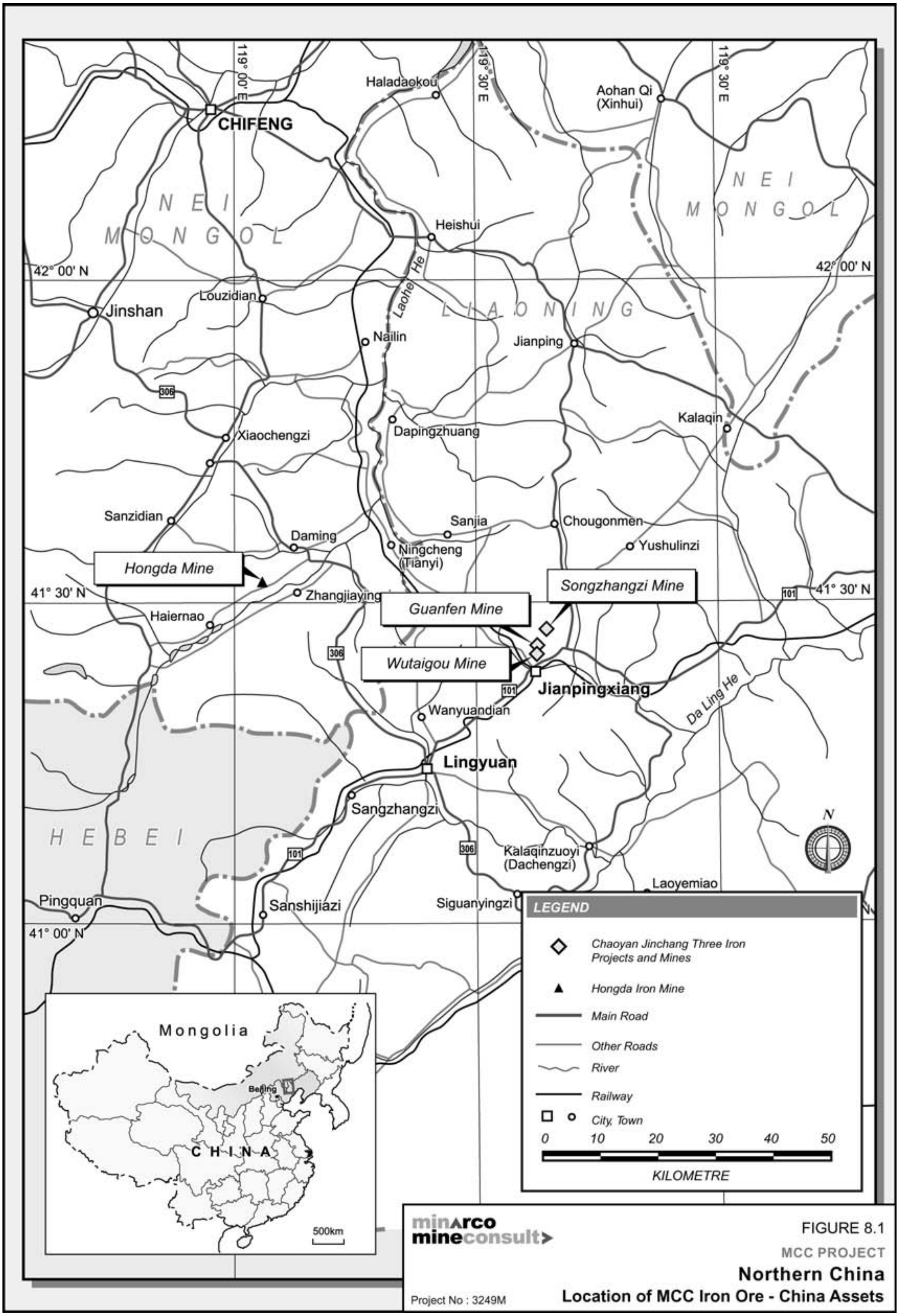
Golder, 2009: Resource Model Update, Cape Lambert Iron Ore Deposit, Western Australia. Golder Associates report to MCCAH dated March 2009.

METS, 2009: Preliminary Metallurgical Testwork Summary Report. Cape Lambert Magnetite Project. Mineral Engineering Technical Services, dated March 2009.

Figure 8.1 — Chinese Domestic Iron Ore Mines Locations



8 JINCHANG MINING ASSETS

M-MC carried out site inspections of these properties in February 2008 to review the resources, processing and mining. In March 2009 M-MC carried out a final site visits to review recent changes to resources, as well as update the current operational status. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "2007, Feasibility Study on Renovation and Extension Projects for Ore Mining and Separating of JianPing Northern Mining Industry Co. Ltd., NETC".
- "July 2008, Songzhangzi Iron Ore Resources Review Reports" prepared by Liaoning 3th Geological Team 2008 Resource Reserve Report).
- "Dec 2008, Songzhangzi Iron Ore Excavation Project Feasibility Study" prepared by MCC North Engineering Technology Co. Ltd.

MCC's effective equity stake in the Jinchang Assets is 85.1%. (85.1% of a subsidiary owning 100% stake in the project)

8.1 BACKGROUND

Various iron ore mines owned by Chaoyang Jinchang Mining Group Co. Ltd (CJMG) are located in Jianping County of Liaoning Province, Northern China, ***Figure 8.1.*** These assets collectively referred to as the "Jinchang Mining Assets" are both disseminated and lode type magnetite (Fe_3O_4) deposits, and include the following iron ore projects:

- Guanfen Iron Ore Mine (Open cut);
- Wutaigou Iron Ore Mine (Open cut);
- Songzhangzi Iron Ore Mine (Underground);

The mining area is located within the sub-drought area of the northern temperate zone with continental monsoon climate. Temperature ranges from -36.9°C to 40.0°C with sub zero temperatures occuring from November to March. The annual average precipitation is 443.4mm, with 70% occurring from June to August. The mine area has undulating topography with elevations generally ranging from 465m to 500m above sea level. The vegetation cover is sparse with considerable exposure of rock.

These projects are located within a small area of 5km^2. Guanfen and Wutaigou Mines are located to the SW and the Songzhangzi Mining Area is located 5km to the NE. A central processing plant and dispatch facility is located between the open cut and underground mining areas as shown in ***Figure 8.2.***

Ore processing for soft ores (now not operating) comprised two stages; a dry magnetic pre-concentration process (Stage 1), located at the individual open cut pits and a wet separation process (Stage 2) in the central processing area. The processing aspects for the Jinchang Assets are described in ***Section 8.9.***

Operating and capital costs for the Jinchang Assets are discussed together in ***Section 8.11.***

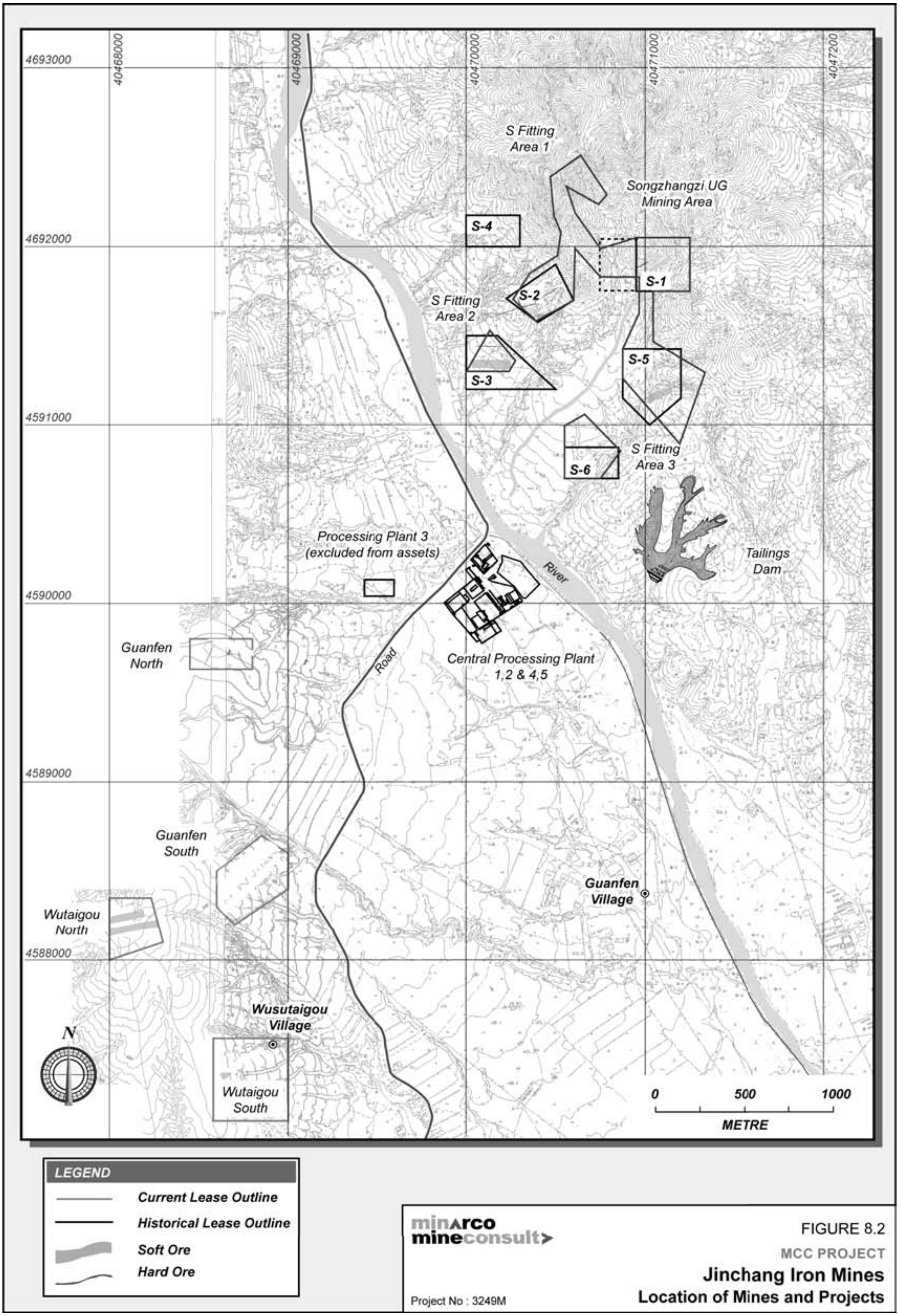
The Chinese institutes report the term "ore" as economically extractable magnetite iron ore. The latest reporting of resources by the Third Geological Team of Liaoning Province (The Institute) has removed all 'soft ore' due to the current metal prices and the low grades, however 'hard ore' resources still remain for the Wutaigo and Songzhangzi mining areas. Current total overall production costs of greater than 500RMB/t product for hard ore types may be uneconomic, and this was supported by concentrate stockpiles (approximately 7,000t) located at the processing plant which remains unsold.

8.2 ASSETS

During the site visit conducted in March 2009, the assets included:

- 1 operating underground shaft;
- 1 newly developed shaft (not operating);
- 1 inclined drift (abandoned);
- 4 open cut pits (not operating);
- Multiple Stage 1 magnetic separation dry processing plants (not operating);
- 1 operating wet processing plant (Stage 2 — crushing, grinding and magnetic separation); and
- 3 "dormant" smaller wet processing plants.

Figure 8.2 — Jinchang Mining Assets — Location Map



8.3 LAND TENURE AND MINERAL RIGHTS

The Liaoning Province Department of Land & Resources granted CJMG the mining rights to the various projects (Guanfen, Wutaigou and Songzhangzi) on November 20, 2006. An updated mining lease was granted in January 2009 covering all three projects under one license. Details are shown in ***Table 8.1***.

Table 8.1 — Jinchang Mining Assets — License Details

Mine/Project	Jinchang Mining Assets
Title	Mining License
No	2100000910006
Owner	CJMG
Mine/Project Name	CJMG of MCC, Songzhangzi Mine
Mine Method	Open cut/Underground
Permit Capacity	80 ktpa
Permit Area	1.072 km^2
Permit Depth	550mRL — 410mRL (UG planning 450mRL to 420mRL)
Valid Date	Jan,2009 — Jan,2010
Issue Date	Jan,2009
Issuer	Liaoning Province Land and Resource Bureau

Source: Formal documentation

Note: One License includes all mining areas.

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

8.4 REGIONAL GEOLOGY

The regional structural feature is a deep, crustal fault which separates medium and high-grade metamorphic rocks of Archaean age to the north from Mesozoic age carbonate, clastic and volcanic rocks to the south.

The host for magnetite enrichment and emplacement of discrete magnetite-quartzite lodes is amphibolite gneiss, within the Archaean metamorphic complex adjacent to major fault structures. Metamorphic texture (gneiss foliation) is generally east-west, varying to north-east and north-west. Dips are generally steep southerly at approximately 70 - 80°.

Magnetite (Fe_3O_4) is an oxide of Fe and is the most magnetic of all minerals. The common term is "lodestone". Magnetite reacts with oxygen (oxidises) to form hematite (Fe_2O_3) which does not report to magnetic product. Mineralisation in the Jinchang Iron Ore deposits is characterised by magnetite enrichment within the host gneiss which has background grades of magnetite (estimated at $<2\%$). Mineralisation is controlled by faulting forming possible pathways or tectonic zones for magnetite enrichment.

At the Songzhangzi Underground Mine (Area S2), high grade ore lenses (termed "lodes") up to 4m wide within the enriched structure are being mined by conventional underground methods. The underground mines are mining extensions of the orebodies down dip and down plunge from the abandoned surface (open cut) mines. In the old open cuts, there is evidence that softer wall zones were mined as low-grade magnetite ore.

The area is covered by a regolith (soil and weathered rock) up to 5m thick. The regolith consists of light clay soil overlying bleached clay / saprolite layer grading to thick gravely soils overlying saprolitic bedrock. Weathering extends to a depth of some 30m. The gravels are poorly sorted and consist predominantly of blocks, cobbles and pebbles of gneiss and magnetite-quartzite.

At the dormant Guanfen and Wutaigou Open Cut Mines, the regolith was previously mined and dry-treated through several small scale plants to produce a concentrate carrying in the order of 20%mFe, magnetite (site advice). This beneficiation process was achieved by coarse screening and two-stage magnetic roll / belt separation. Remnant pods of hard ore report to lump and this material was stockpiled for delivery to the central processing plant. Reject from magnetic separation was discarded.

The magnetite-quartzite mineralisation is termed by local operators as "soft ores" and "hard ores", based on physical mineability, i.e. soft ores are open cut resources and hard ores are underground resources. There are remnant pods of hard ore within the soft ore which report to lump in the Stage 1 process. For the soft ores, Fe grade is reported as mFe (magnetic iron content), as magnetite (the predominant material) is easily separated in processing. For the hard ore, Fe grade is reported as TFe (total iron content) which includes some paramagnetic material. The concentrate grade represents total magnetic iron or TFe. This includes up to 15% paramagnetic material.

8.5 GUANFEN IRON ORE MINE

8.5.1 Background

The Guanfen Open Cut Mine operation was commenced in 2006 by a private company then acquired by Northern Engineering & Technological Corporation in late 2006. CJMG was granted tenure in 2006. Open cut mining ceased in late 2008 due to falling commodity prices, low ore grades and lack of 'soft' ore resources.

Small scale surface mining had taken place by local residents which now, has mostly been incorporated into the current open cut pits. Mining operations were from two separate pits using truck and shovel methods.

The magnetite deposit consists of two separate Mining Areas (rights). The main South Pit (Mining Area 1) consists of 2 'ultra lean' (low grade) Fe ore zones named mining stope CK1. CK1 dips to the northwest at between 76° and 78°. The two ore horizons are approximately 50m and 100m wide.

The North Pit (Mining Area 2) consists of 1 'ultra lean' (low grade) Fe orebody and it is named CK2. CK2 dips to the south-southwest at between 78° and 80° and has an average width of 36m.

Open cut reserves (soft ore) are currently depleted and long term production is dependent on the mineability of the harder ore and the metal price. Further exploration could increase the availability of soft ore sources, but at present this has not been completed. During the site visit, there was evidence of historical underground mining in the Guanfen South Pit, but information obtained from site personnel, indicated that there are no future plans for any further underground activity. Open cut mining has been completed to a depth of approximately 30m below the surface.

8.5.2 Geology

Orebody characteristics are summarised in ***Table 8.2***.

Table 8.2 — Guanfen Iron Mine — Orebody Descriptions

Mining Area	Location	Mining Stope Name	Orebody Type	No. of Orebodies	Max. Thickness (m)	Dip Angle (°)	Mining Method & Status
1	South	CK1	Zone	2	105	76-78 NW	Open cut — abandoned
2	North	CK2	Zone	1	44	78-80 SSW	Open cut — abandoned

Notes: Sourced from site information

Two separate irregular open cut pits (approximately 1.5km apart) have been developed to extract weathered or "soft ore" in three identified bands of magnetite-quartzite mineralisation which represent the ore zones at

Guanfen. The pits initially stripped the regolith and extended about 25m into saprolitic gneiss. The open cut "reserves" have been depleted. Small piles of hard, blue / grey magnetite gneiss and quartz magnetite rock were observed on the floor of the pits. These piles may represent fresh, unweathered "hard ore".

The three identified ore zones consist of magnetite bearing amphibolite gneiss lenses with approximately 350m strike length ranging in width from 30m to 105m and dipping steeply. These ore zones occasionally include irregular, lenses of higher grade quartzite-magnetite. The Guanfen South Pit was based on two soft ore type magnetite gneiss bodies with grades of 7-14%mFe which are approximately 50m and 100m wide respectively and strike NE. Guanfen North Pit was based on a single magnetite-gneiss ore zone striking NNW with a width of 50m.

Faulting with displacement of 20m in OB2 (South Pit) is evident on plans.

Geological Risks include:

- Some faulting, and
- Lack of definition of extensions of ore at depth

Geological Opportunities include:

- Additional open cut In Situ Quantities may exist as extensions along strike ore zones outside mining areas,
- Thinner lodes may be recovered by opportunity if located near mining development, and
- Exploration may identify additional ore zones, thicker intersections and deeper ore.

Figure 8.3 — Guanfen Iron Mine — Topography and Geology

8.5.3 Mining

The Guanfen Mine comprises of two small leases referred to as the South Pit (Mining Area 1) and the North Pit (Mining Area 2). Both these areas are currently dormant. The South Pit is the oldest and largest of the two Guanfen pits. The mining method used was a conventional open cut mining technique utilising 30t hydraulic backhoe excavators and 25t rear-dump trucks. The soft ore type is a free digging material and requires no blasting.

Guanfen South Pit

The main Guanfen South Pit is a small, narrow open cut that is approximately 200m long and has been developed to a depth of approximately 30m. The crest elevation is 500mRL and the pit floor is 470mRL. The Pit floor is at the base of weathering (soft ore) and any deeper open cut mining in Hard Ore will require drilling and blasting to fragment the rock before excavation.

The pit includes two main ore bodies (CK1 and CK2). The Guanfen South Pit open cut has been depleted of all soft ore sources.

No geotechnical issues were noted with only minor superficial cracking near the pit crest. In M-MC's opinion, there are no identified geotechnical or groundwater risks.

Guanfen North Pit

The Guanfen North Pit, which commenced in 2007, is located approximately 1.5km north of the South Pit. It is a small narrow open cut that is approximately 100m long and has been developed to a depth of approximately 30m. The crest elevation is 500mRL and the pit floor is 470mRL.

The pit incorporates one main orebody (CK4). The Guanfen North Pit open cut has been depleted of all soft ore sources.

In M-MC's opinion, there are no identified geotechnical or groundwater risks to prevent the open cut operation from extending to the limits of the soft ore.

Historic production is given in ***Table 8.3***.

Table 8.3 — Guanfen Iron Mine — Historic Production

Ore Production	Units	2007	2008
Opencut	ROM kt	246.78	278.52
Strip Ratio (OC Only)	Waste t : Ore t	24:1	24:1

Source: Client provided information.

There is no forecast production in the above table due to current commodity prices, however the maximum production level for the Guanfen Mine is 300ktpa based on site information.

Mining risks include:

- Lack of drill defined Mineral Resources (or In Situ Quantities).

Mining opportunities include:

- Additional Mineable Quantities defined by further exploration (drilling) and economic mining methods
- Potential to mine higher grade "hard ore" from open cut development which could achieve lower mining costs and higher recovery than underground mining.

8.6 WUTAIGOU IRON ORE MINE

8.6.1 Background

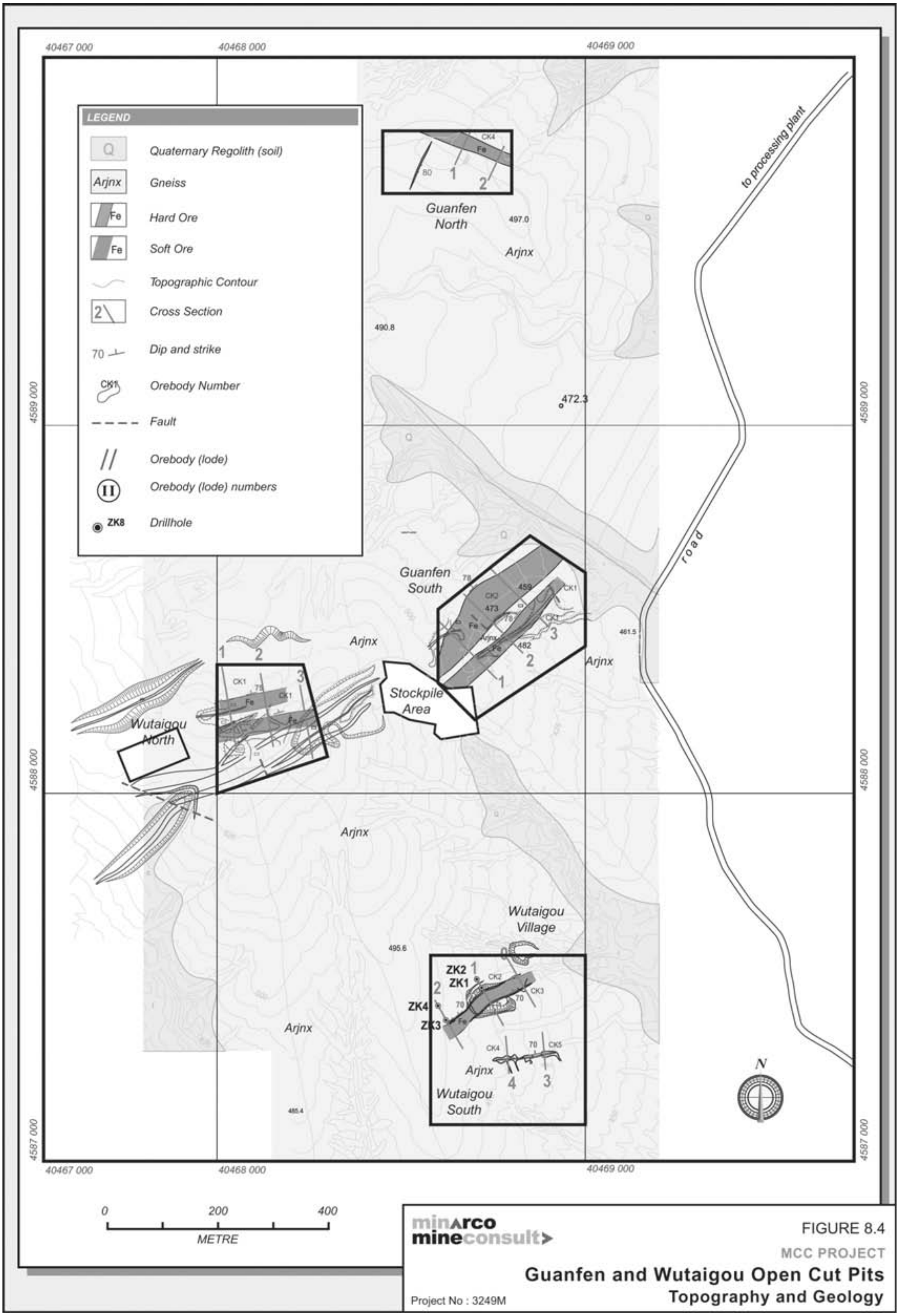
The Wutaigou Iron Ore Mine comprises two separate Mining Areas within separate leases adjacent to the Guanfen South Pit. The North Area is located 500m to the west and the South Area 800m to the south of the Guanfen South Pit ***(Figure 8.4)***.

In 2004 and 2005, the geological team conducted detailed mapping at 1:2,000 and completed four diamond drill holes in the South Pit totalling 495m. This data was the basis of a (332, 333) resource estimate (In Situ Quantities) of 136.4kt with average grade in two hard ore zones of 29.51%TFe. The soft In Situ Quantities was also reported at this time as 1,458.5kt at an average grade of 10.03%mFe.

Before development of the mine in 2006, local residents had conducted small-scale diggings. Evidence remains with some small-scale mining tunnels into orebodies defined as Line 1, Line 2 and Line 3. Since commencement there have been seven small-scale open cut pits in the southern and northern Mining Areas.

Recent operations were based on two larger pits; the South Pit (Mining Area 1) and the North Pit (Mining Area 2). Open cut mining methods used truck and shovel equipment with drill and blast as required. Open cut "reserves" (soft ore) have been depleted.

Figure 8.4 — Guanfen and Wutaigou Iron Mines — Topography and Geology



8.6.2 Geology

The North Area comprised several pits based on narrow lodes (5m to 20m wide) which strike NE and dip steeply (approximately 75° NW). Orebody characteristics for the lodes are shown in ***Table 8.4***. Plans reviewed by M-MC show thin orebodies which are the high grade portions of the lodes. M-MC assume the wider orebodies outlined in ***Table 8.4*** include the higher grade lodes and the surrounding lower grade material.

Grades of 6%mFe to 13.5%mFe were recorded in 3 sections of channel samples in the North ore zones. Minor faulting is indicated on maps with horizontal displacements of approximately 10m.

Table 8.4 — Wutaigou Iron Mine — Orebody Descriptions

Location	Mining Area No.	Ore Body No.	Pit Location	Length	Real Thickness	Dip Angle	Average Grade	Ore Type
					(m)	(°)	TFe%	
South	1	2	North	85	7.94-9.28	73	27.99	Hard
		1	Centre	100	6.89-8.32	73	28.52	Hard
		3	South	170	2.04-2.23	76	26.99	Hard
North	2	7	North	110	3.73-6.54	69	14.62	Soft
		5	Centre	310	9.61-17.30	74	11.01	Soft
		4	South	320	17.30-21.15	74	13.21	Soft

Source: Site information

The South Area comprised two pits based on two separate ore zones which strike NE and ENE and dip steeply at approximately 70°NW. The weathered zone is approximately 20m to 30m deep.

The North Pit was the main operating pit and based on an ore zone defined by four (4) diamond drill holes. Ore zone width (soft ore) was approximately 80m with boundary zones of less than 5m. The main part of the ore zone (soft ore) had grades ranging from 8%mFe to 13.6%mFe. The boundary zones (hard ore) have grades ranging from 27%TFe to 30%TFe.

M-MC did not observe any significant water in-flow in the Wutaigou pits.

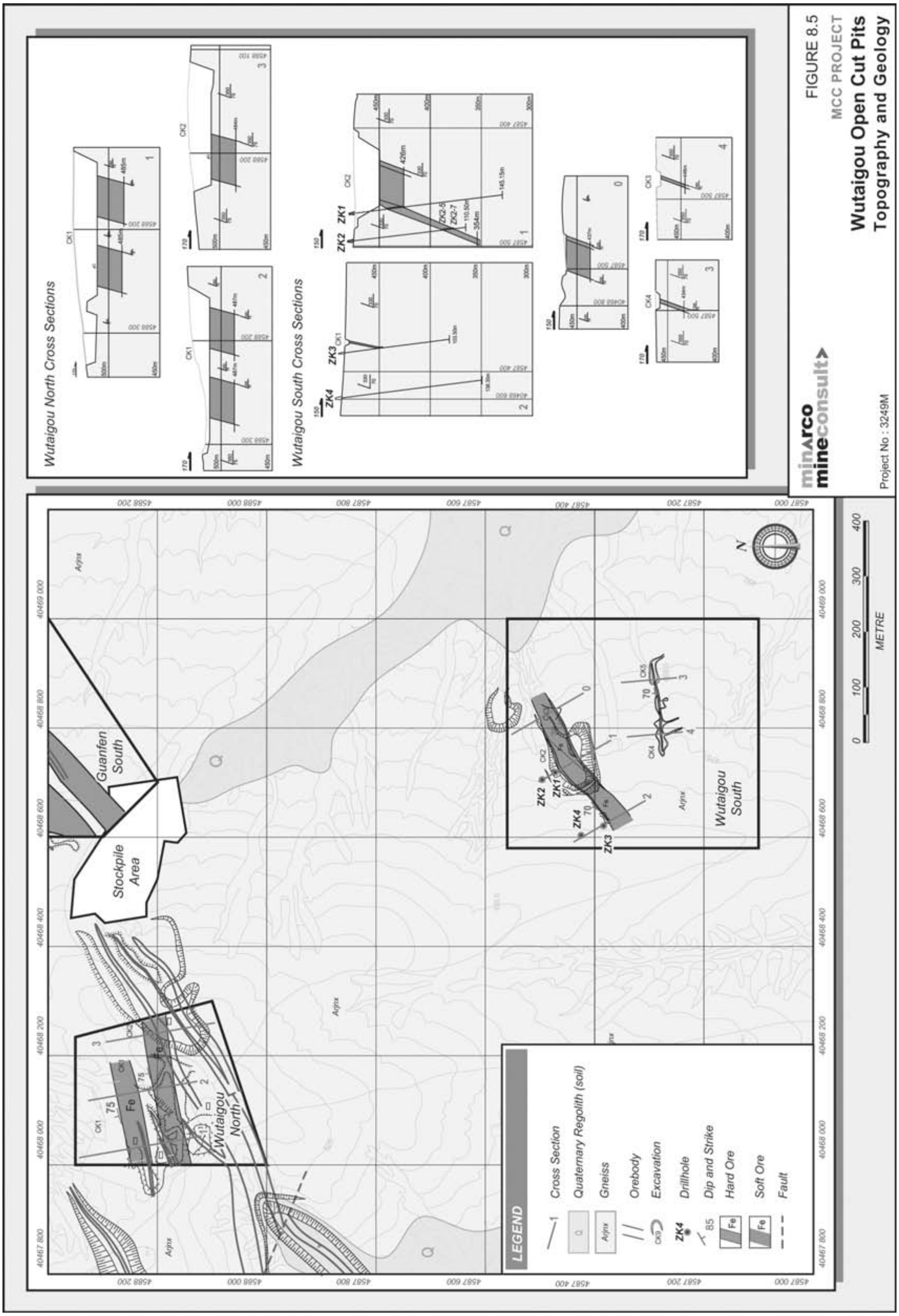
Geological risks include:

- Some faulting, and
- Lack of definition of extensions of ore both along strike and at depth (down dip).

Geological opportunities include:

- Additional open cut In Situ Quantities (low grade) may exist as extensions along strike ore zones outside mining areas,
- Thinner lodes may be recovered by opportunity if located near mining development, and
- Exploration may identify additional ore zones, thicker intersections and deeper ore.

Figure 8.5 — Wutaigou Iron Mine — Topography and Geology



8.6.3 Mining

The Wutaigou Mine comprises two small leases; the Southern and Northern Areas. Both these areas are currently dormant. The mining method used was a conventional open cut mining technique utilising 30t hydraulic backhoe excavators and 25t rear-dump trucks. The soft ore type is a free digging material and requires no blasting.

Wutaigou South Area-South Pit

The South Pit is a very small pit approximately 140m long by 60m wide with an ore zone width of approximately 8-15m. Current open cut development is at a depth of 32m with a crest at 476mRL and pit floor at 444mRL

M-MC did not observe any significant geotechnical or groundwater risks.

Wutaigou South Area- North Pit

Most of the soil and lateritic material that has previously been mined was being dry treated through a two stage belt magnetic separator to recover magnetic cobbles and fines within the weathered ore zones. The underlying weathered saprolitic bedrock, (gneiss and / or granite) comprises weathered magnetite-quartzite lodes plus minor narrow stringer lodes which could be recovered by selective mining.

The Main Pit was predominantly mining lower grade soft ore, however, during the March 2009 site visit all soft ore had been mined and only hard ore remained in the bottom of the pit.

Pit dimensions are approximately 300m by 200m with development to a depth of 35m.

M-MC did not observe any significant geotechnical or groundwater risks.

Historic production is given in ***Table 8.5***.

Table 8.5 — Wutaigou Iron Mine — Historic Production

Ore Production	Units	2007	2008
Opencut	kt Ore	186.35	29.254

There is no forecast production in the above table due to current commodity prices, however the maximum production for the Wutaigou Mine is 300ktpa based on site information.

Open cut reserves are currently depleted and long term production is dependent on the mineability of the harder ore and the metal price. Further exploration could increase the availability of soft ore sources, but at present this has not been completed.

Mining risks include:

- There are currently no mining risks given the lack of defined reserves and associated mine planning.

Mining opportunities include:

- Additional Mineable Quantities defined by further exploration (drilling) and economic mining methods, and
- Potential to continue mining higher grade "hard ore" from open cut development which could achieve a lower mining costs and higher recovery than underground mining.

8.7 SONGZHANGZI IRON ORE MINE

8.7.1 Background

The Songzhangzi Mine incorporates Mining Areas 1 to 3 based on numerous shallow open cut workings and narrow underground hard ore lodes.

Previous mining activity at the Songzhangzi Mine was initially by small open cut pits, however all open cut operations have now ceased. Mining activity is currently restricted to a single underground operation exploiting Orebody III-2 located within Mining Area 1. The underground mining method is short-hole shrinkage stoping. The vertical shaft located on the south western side of the orebody is currently being used for extraction of the ore, whilst a newly developed shaft has been constructed in the north east. This shaft was not being used at the time of the site visit, but it was indicated by site personnel that only the electrical wiring was required before operation could begin. There was evidence of historical underground workings, but no infrastructure was present on site.

The inclined drift located north east of Orebody III-2, which was previously the main route used for hoisting $3.2m^3$ skips to the surface has recently been decommissioned. All future production for this orebody will use a combination of the current active shaft (SJ1) and the newly constructed shaft (SJ2), see ***Figure 8.6***.

Local prospectors are still active in the area and recover small amounts (pickings) of hard ore which are "processed" with a hand held magnet. M-MC assume these "pickings" are onsold to the mine owners.

Figure 8.6 — Songzhangzi Iron Mine — Topography and Geology

8.7.2 Geology

Orebodies are represented by a few wide zones within gneiss foliations and are mostly thinner lode type. The general strike is E to NE and dip steeply south to SE. Ore bodies within Mining Areas reported at Songzhangzi are shown in ***Table 8.6***.

Table 8.6 — Songzhangzi Iron Mine — Orebody Descriptions

Mining Area	Orebody No.	Orebody Type	Length (m)	Real Thickness (m)	Dip Angle (°)	Average Grade TFe (%)	UG Status
	I	Lode	320	3.40-6.80	80	28.55	
	I-1	Lode	180	2.06-2.36	80	28.10	
	I-2	Lode	170	4.83-6.00	80	29.07	
1	III	Lode	230	4.33-10.44	80	30.00	2 shafts+drift
	IX	Lode	280	1.97-3.25	83	28.08	
	VIII	Lode	310	1.69-5.08	85	29.42	
	XI	Lode	136	1.29-1.44	83	Unknown	
2	IV	Lode	210	2.09-3.98	85	29.32	
	V	Lode	60	1.49	85	27.22	
	V-1	Lode	60	1.49-1.59	85	26.32	
3	X	Lode	290	1.88-3.75	81	27.43	

Source: Site information

Unlike the Guanfen and Wutaigou Mines to the west, the Songzhangzi Mines are situated on the flank of a relatively steep hill. There are five magnetite-quartzite lodes within the underground workings.

From observations, within the old open cut pits an oxidised high-grade lode (or lodes) with a disseminated halo up to 20m wide was extracted. The gneiss host rock is very friable and unstable walls have partially collapsed into the pit floor. During the site visit in February 2008, on a wall of the open cut near the new shaft / incline drift, a prospector was observed hand mining a near vertical, magnetite-quartzite lode approximately 40cm wide. This indicated that mineralisation may consist of several lodes concentrated within the foliations. There may be some mineralisation control along fault or fracture pathways.

Remnants of underground ore stockpiles near shafts consist of hard, fresh magnetite-quartzite and gneiss with occasional pieces of dolerite and granite.

Ground conditions for the underground operation are reasonable (site advice), however care is taken and the ground is supported by props as necessary. Maps indicate intrusive granodiorite within fault zones dislocating the lodes. M-MC observed many small faults and fractures dislocating lodes in the old pits. These geological risks are expected to continue at depth and may impact on underground operations. Faulting is recorded in an underground mining level plan which indicates more dislocation of lodes than reported.

From observations in the pits, M-MC confirmed that lodes dip at 74^{o} to 80^{o} the south and south-east. In M-MC's opinion, the steep dips are well suited for underground mining methods.

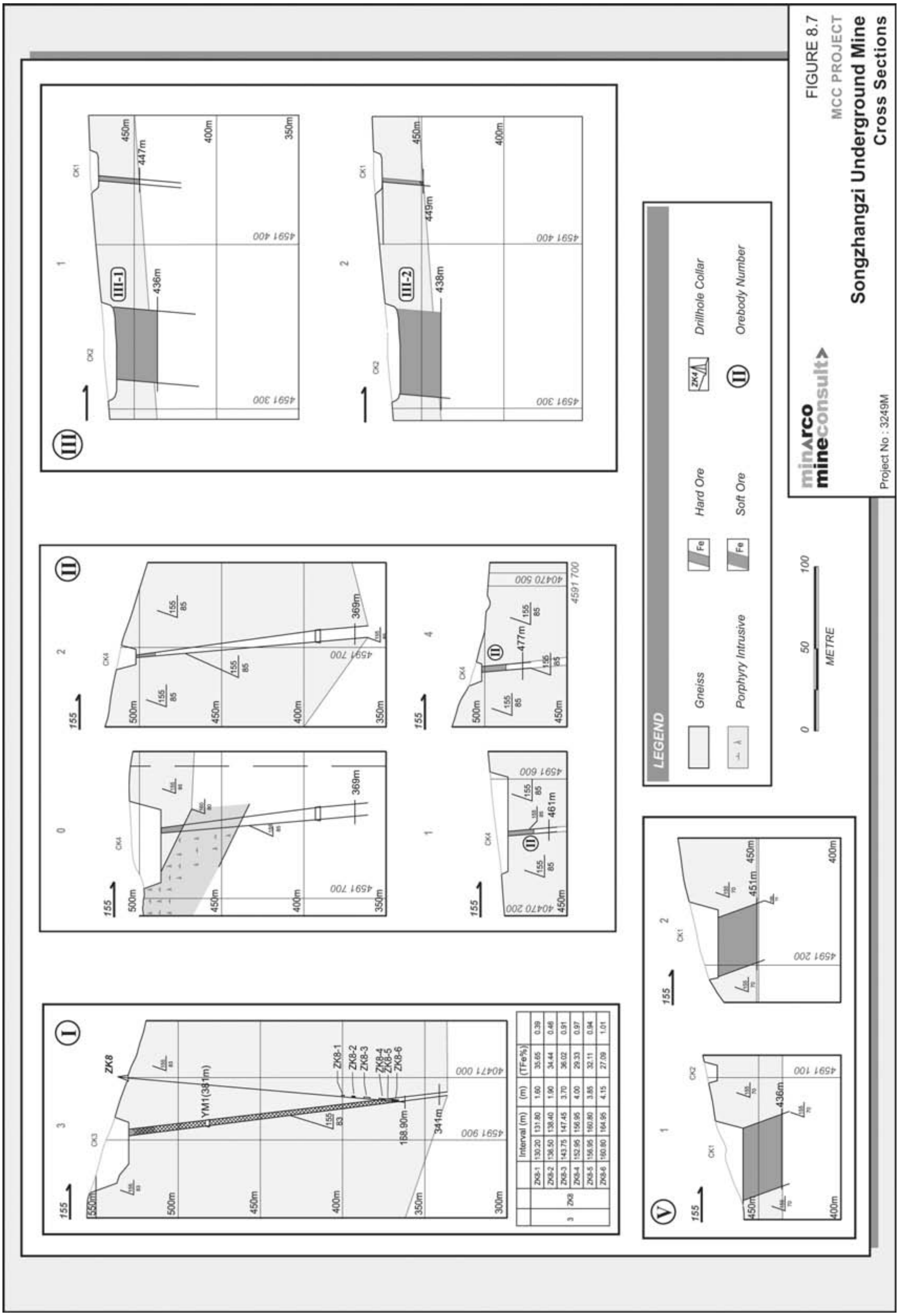
Geological risks include:

- Limited width ore bodies,
- Variable thickness and extent (strike and dip) of lode type ore,
- Thin lodes will impact on mining recovery and dilution, and
- Faulting and fractures will impact on continuity of mine development.

Geological opportunities include:

- Grade within lodes is relatively consistent,
- Additional open cut In Situ Quantities exist as extensions along strike of lodes and ore zones outside Mining Districts,
- Thinner lodes may be recovered by opportunity if located near mining development, and
- Exploration may identify additional lodes, thicker ore zones and deeper ore.

Figure 8.7 — Songzhangzi Iron Mine — Cross Sections



8.7.3 Mining

Iron ore in zones and lodes outcrop on relatively mountainous topography. Mining is located at elevations ranging from 550mRL to 460mRL.

Underground — Hard Ore Operations

Lodes are mined underground to a width of 4m (site advice). It is likely that narrower lodes have been extracted by opportunity.

The underground deposits use a short-hole shrinkage stoping mining method. This method utilises handheld drills and small scrapers. It requires the operator to develop horizontal drives along the strike of the orebody. Lifts are taken on top of each other and work is conducted on top of the broken ore produced from the lift below. This method continues for 50 vertical metres until the panel has been completed. Ore is then removed from the stope with the scrapers and transported to the surface. Small 5t capacity trucks are loaded directly from the shaft and haul the ore to the central processing plant for crushing and processing.

During the site visit to Jinchang in March 2009, it was observed that due to the metal price, the only operating mine was at orebody III-2, which is located in Mining Area 1. The mine was accessed via a vertical shaft located at the south western corner of the orebody and was the only means of ore extraction at the time of the visit. The inclined shaft, which was previously used, has been decommissioned and a new vertical shaft has been constructed 50m south of the inclined shaft. The new shaft head frame has been constructed to a high standard and the shaft measures 3.5m in diameter. The new shaft provides access to the 340mRL at the north eastern end of the orebody. The shaft was not being utilised at the time of the March 2009 site visit, but it was very close to being commissioned.

In M-MC's opinion, equipment currently available at the mine is sufficient to produce the targeted production level.

Major development levels are located at elevations 440mRL, 390mRL and 340mRL.

Historic and forecast production figures are given in ***Table 8.7***. Forecast production figures are totally dependent on the current metal price, although maximum capacity of the current operations is 100ktpa.

Table 8.7 — Songzhangzi Iron Mine — Historic and Forecast Production

Ore Production	Ore Type	Units	2007	2008	2009	2010	2011	2012
Opencut	Soft	kt Ore	20					
Underground	Hard	kt Ore	80	50.63	100	100	100	100
Total		**kt Ore**	**100.46**	**50.63**	**100**	**100**	**100**	**100**
OC Strip Ratio		Waste(t):Ore(t)	8.1:1					

Source: Client Information

Mining Risks include:

- Geological risks (faulting) impacting on underground mining development, lower ore recovery and higher dilution, and
- There is no consideration for pillars in the current underground design as a depth limit has not been defined. This factor may impact on ore recovery.

Mining Opportunities include:

- Additional Mineable Quantities defined by further exploration (drilling) and economic mining methods.

8.8 RESOURCES AND RESERVES

The total reported In Situ Quantities for Jinchang are 669,000t at 31.69% TFe. M-MC have not validated these estimates. However in M-MC's opinion, based on the site visit and a review of the Liaoning Institute's estimation methods, the order of magnitude of both tonnes and grades are reasonable.

Whilst the Chinese Code estimates appear reasonable, M-MC has referred to the resources and reserves for Jinchang as In Situ Quantities accordingly, and compared them against JORC equivalent terms. These resources are not reported in compliance with the JORC Code due to the lack of Quality Control and Quality Assurance (QAQC) information and drilling data.

8.8.1 Mineral Resources — In Situ Quantities

Historical resource estimates for the Guanfen and Songzhangzi projects were reported by the Liaoning Institute in October 2007. An historical estimate for Wutaigou was provided by MCC in the 2005 Resource Review report. A summary of the historical estimates are shown in ***Table 8.8.*** The low average grade <10%mFe indicates a soft ore type, whilst the hard ore type generally contains average grades greater than 25%TFe.

Table 8.8 — Jinchang Mining Assets — Summary of Historical In Situ Quantities

Mine	Ore Type	Chinese Code	In situ Quantities (kt)	Average Grade
Guanfen[#1]	Soft	333	3,167.8	8.76% - mFe
	Hard	333	136.4	29.51% - TFe
Wutaigou[#2]	Soft	333	1,458.5	10.03% - mFe
	Hard	332	391.1	32.4% - TFe
Songzhangzi[#1]	Hard	333	242.8	32.4% - TFe
	Soft	333	794.7	9.90% - mFe

Source: #1 Estimated and reported by the Liaoning Institute in October 2007 using the Chinese Code.
#2 Sourced from Resource Review Report 2005, confirmed by Client email
Notes: Estimates are not reported in accordance with the JORC Code.

Current resource estimates are based on an updated report in July 2008 by the Liaoning Non-Ferrous Geological Exploration Institute personnel, assisted by mine engineering personnel (2008 Resource Reserve Report).

The latest resources were calculated according to the Chinese Code and were reported using a cut-off grade of 20%TFe. All 'soft' ore was excluded from the reported In Situ Quantities due to the low Fe grades. Resources were based on limited diamond core drilling and current pit and underground excavations. A detailed tabulation is shown in ***Table 8.9.***

Table 8.9 — Jinchang Mining Assets — In Situ Quantities

Mine	Area	Orebody	In Situ Quantities (kt) Chinese Code JORC Equivalent	122b Measured	332 Indicated	333 Inferred	Average Grade (% TFe)
Songzhangzi	1	I	I	8.3	40.0	103.6	29.15
		II	II	31.4	236.5	29.4	34.52
Songzhangzi	2	III-2	III-2			26.5	28.88
Sub Total				**39.7**	**276.5**	**159.5**	**32.49**
Wutaigou	1	I	I	23.9	22.3	20	29.07
		II	II	24.3		59.7	31.7
		III	III	7.7		35.4	26.78
Sub Total				**55.9**	**22.3**	**115.1**	**29.70**
Total				**95.6**	**298.8**	**274.6**	**31.69**

Source: 2008 Resource Reserve Report

Notes: Figures shown excluding resources outside the Mining License as reported by the Institute Estimates are not reported in accordance with the JORC Code.

8.8.2 Reserves — Mineable Quantities

Due to the current commodity price for iron ore and the costs associated with production of iron concentrate at the Jinchang Mining Assets, there are currently no mining reserves as defined by either Chinese or International Mining Codes.

Mining is however ongoing at Songzhangzi, although at reduced capacity targeting the underground Hard Ore resources. Mineable Quantities are reported in the "Feasibility Study on Renovation and Extension Projects for Ore Mining and Separating of JianPing Northern Mining Industry Co". This study using standard Chinese recovery calculations estimated Mineable Quantities in the order of 300-350kt at 28.2%TFe.

M-MC has reviewed these estimates and considers them reasonable. However due to the high mining cost and low iron ore concentrate price these estimates in March 2009 are considered sub economic and cannot be reported as either reserves or Mineable Quantities.

8.9 MINERAL PROCESSING

The Jinchang operation has four processing plants (Concentrator No's. 1 and 2 and Plant No's.1 and.2) of which only one was in operation during the recent visit to the site by M-MC. The mineral processing consists of two stages: a dry pre-concentration stage located at the mine-site and a wet processing stage. The dry processing stage uses two stages of dry magnetic separation to upgrade the iron content of the ROM ore from 8% to approximately 15%mFe. This product is magnetic iron, primarily magnetite with some minor paramagnetic material.

ROM feed from adjacent pits are fed to a scalping screen with 150 mm aperatures. The oversize material is hard ore with higher iron grades and is stockpiled for transport to the Stage 2 wet processing operation. Reported yields are 36% (Stage 1 product t /ROM t). In M-MC's opinion, the reported yields are reasonable based on visual estimates of approximately 30%.

- Stage 1 dry processing for pre-concentration — upgrade to approximately 15% to 16.8%TFe
- Stage 2 wet processing plant. — upgrade to approximately 65%TFe

Any ROM production from the Guanfen and Wutaigou Open Cut Mines was soft ore and was pre-concentrated to approximately 20%TFe. Remnant hard ore (higher grade) that was present in the soft ore reported as lump scalping screen oversize (+150mm) during pre-concentration in Stage 1 processing. This material was stockpiled and transported to the No. 2 processing plant.

ROM production from Songzhangzi is hard ore and is processed through the No. 2 wet processing plant.

Stage 2 — Wet Processing

The technology in use is conventional crushing and grinding followed by magnetic recovery through wet, high intensity magnetic separators in both primary and secondary (scavenging) duties.

There was no evidence of any liberation studies to determine the optimum grinding size for any process plant feed.

A summary of the Stage 2 wet processing plant facilities located in the central processing area is provided in ***Table 8.10.*** The No. 3 processing plant is located in a separate work area near the central processing area; however it is not included in the Relevant Assets.

Table 8.10 — Jinchang Mining Assets — Wet Processing Facilities

Plant Name	Units	Concentrator No. 1	Concentrator No. 2	No. 1 Plant	No. 2 Plant
Status		Not Operating	Not Operating	Not Operating	Operating
Capacity (Feed)	kt/year	70	396	240	400
Circuits / Stages	No.	4/10	4/10	4/10	4/10
ROM Grade	TFe%	14-17	14-17	20-46	15-20
Product Grade	TFe%	65.4	65.8	65	65
Tailings Grade	TFe%	3.8	3.8	4.5	3.5-4.0

Source: client, report & site advice

Some typical process grades and recoveries for soft and hard ores are presented in ***Table 8.11***. Recoveries at the Jinchang wet processing plants varies from 66-71% for soft ores to 80% for hard ores, depending upon the feed grade. In M-MC's opinion, it should be possible to improve magnetite recovery, which may require finer grinding to liberate more magnetite.

Table 8.11 — Jinchang Mining Assets — Stage Process Grades and Recoveries

Description	Units	Guanfen Soft Ore	Wutaigou Soft Ore	Songzhangzi Hard Ore
Grade in situ	%TFe	11.25	12	21
Grade Stage 1	%TFe	16.8	15	21
Grade Conc	%TFe	65	65	65
Recovery	%	66	71	80

Source: client, report & site advice

Reports state that ore is low in impurities such as phosphorous and sulphur, however this has not been confirmed by any data. While the deportment of these impurities is not clear, the operational staff stated that the final iron concentrate was very low in these materials and that they fell below the impurity limit (<0.03% P). In fact, neither the operation nor the customers bother to assay for these impurities.

The current designed production capacity is 200ktpa iron concentrate at a grade of 65%TFe. It was suggested that there were plans to increase concentrate production to 500ktpa, however no plans or details were sighted by M-MC to substantiate this. Currently, the No. 2 Processing plant has stockpiled 7,000t of concentrates.

8.9.1 Processing Circuits

The flowsheet for the No. 1 and No. 2 wet processing plants, which process hard ores, is given in ***Figure 8.8***. The Concentrator No. 1 and No. 2 processing plants, which processed soft ores, have similar milling and magnetic separation flowsheets, however without the crushing, screening and dry magnetic separation sections. In the case of the No. 1 and No. 2 plants, there is a standby crusher.

The No. 1 and No. 2 processing plant flowsheet consists of a jaw crusher followed by a cone crusher. The cone crusher product is then screened at 25mm and the undersized further upgraded by dry magnetic separation (approximately 15% rejection by mass). The upgraded material feeds a ball mill in closed circuit with a spiral classifier. The magnetite in the fine product stream (P_{50}=74 microns) from the spiral classifier is recovered by a wet magnetic separator. The non-magnetic material is discharged to final tailings while the magnetite rich stream undergoes further upgrading.

This consists of regrinding in a rod mill in closed circuit with a high frequency screen. A final grade concentrate is prepared from the screen underflow (P_{100}=74 microns) by two stages of wet magnetic separator and dewatered with a vacuum drum filter before being discharged onto the product stockpile. The non-magnetic material is discharged to final trailings.

Processing risks include:

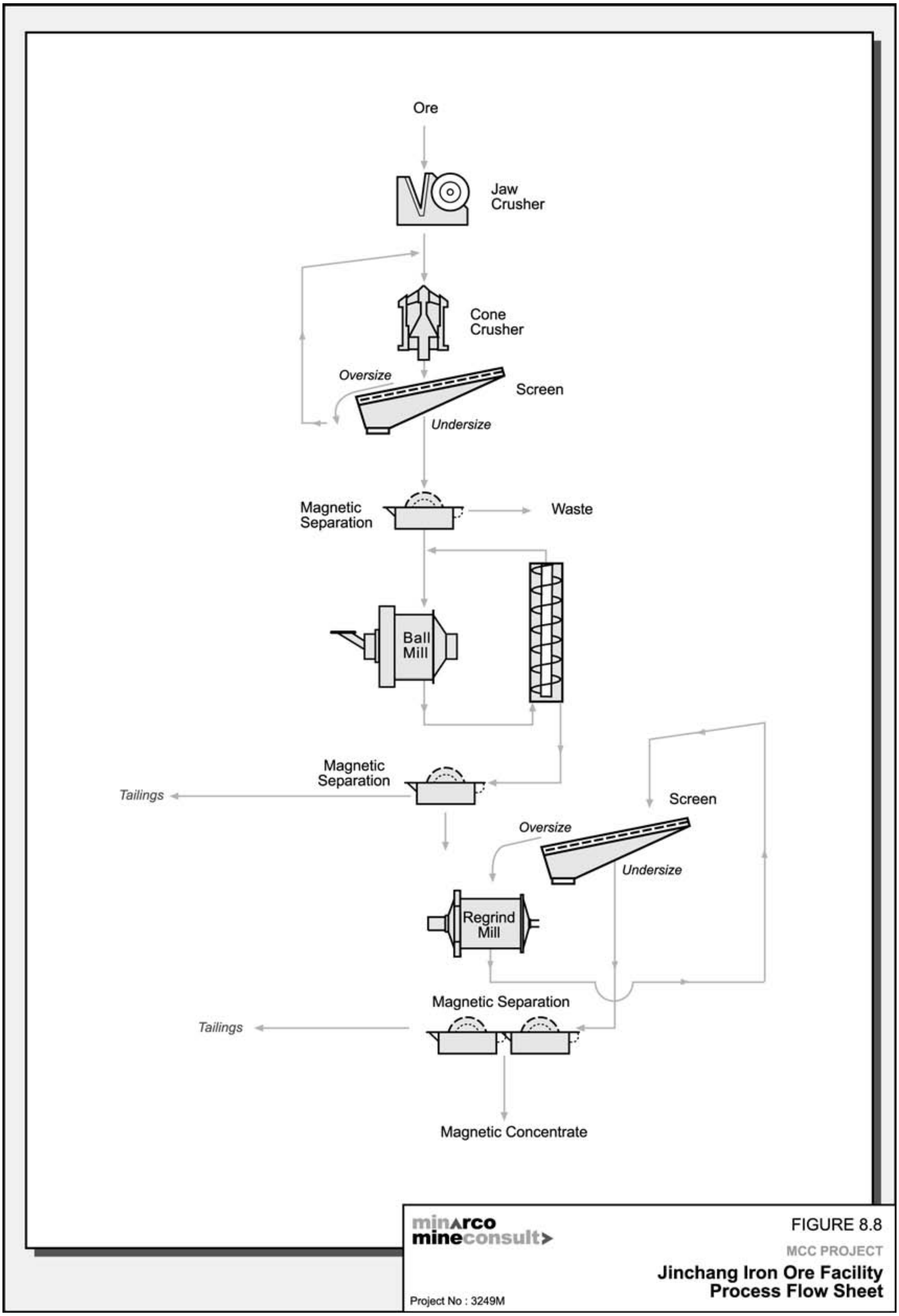
- Poor plant efficiency due to old technology (low intensity magnetic separators), and
- Poor process control evidenced by magnetic material reporting to tailings and other plant spills and rejects.

Processing opportunities include:

- Improved management of plant efficiency (metallurgical efficiency) by monitoring magnetite (mFe) vs total iron (TFe) as well as tailings grade,
- Improved process control including level control,
- Improved plant efficiency using high intensity magnetic separators.
- Improved magnetite recovery through finer grinding to liberate more magnetite

M-MC recommends improving magnetite recovery by using a multi-stage process which includes a high strength / high gradient scavenger.

Figure 8.8 — Jinchang Mining Assets — Nº 1 and 2 Processing Plant Flowsheet



8.10 INFRASTRUCUTRE AND SERVICES

The Jinchang operation is located in an active mining region with established infrastructure, including grid power, sufficient underground water resources and adequate roads.

Water is re-cycled from the concentrate and tailings dewatering operation for re-use in the process.

8.11 CAPITAL AND OPERATING COSTS

Based on the scale and type of mining, operational staff indicated that mining costs were in the range of 28-35RMB/ROM t for open cut and 52RMB/ROM t for underground mining. For some mining areas, the underground mining costs can be cheaper than the open cut mining costs. This is due to the cost of land acquisition, high open cut mining strip ratios and poor yields during pre-concentration. The cut-off price is around 500RMB/ concentrate tonne.

The forecast overall mining costs are presented in ***Table 8.12***. Going forward, it is hard to predict the actual mining costs since it will be dependent upon the iron ore prices. Higher iron ore prices mean that soft ores may be mined again, thereby potentially reducing mining costs. On the other hand, if the current low iron ores are maintained for some years, then the mining costs should be above 50RMB/ROM tonne.

Table 8.12 — Jinchang Mining Assets — Actual and Forecast Mining Costs

Mining Cost	Unit	2007	2008	2009	2010	2011	2012	2013	2014
Auxiliary Material	RMB/ROM t	2.42	2.81	1.85	1.85	1.85	1.85	1.85	1.85
Water & Power	RMB/ROM t	0.99	1.75	0.90	0.90	0.90	0.90	0.90	0.90
Labour	RMB/ROM t	18.23	23.04	18.5	18.5	18.5	18.5	18.5	18.5
Repair & Maintenance	RMB/ROM t	0.21	0.35	0.17	0.17	0.17	0.17	0.17	0.17
Depreciation	RMB/ROM t	—	9.06	9.00	9.00	9.00	9.00	9.00	9.00
Others	RMB/ROM t	26.76	28.47	9.58	9.58	9.58	9.58	9.58	9.58
Total	**RMB/ROM t**	**48.61**	**65.48**	**40.00**	**40.00**	**40.00**	**40.00**	**40.00**	**40.00**

Source: MCC provided Capex and Opex costs February 09

The Jinchang processing costs are around 20-25RMB/ROM t according to operational staff for all processing plants. Typical milling media consumption is 0.7kg/t while about 20kWh/t of power is consumed during processing. The historical and forecast processing cost breakdown is presented in ***Table 8.13***.

Table 8.13 — Jinchang Mining Assets — Actual and Forecast Processing Costs

Processing Cost	Unit	2007	2008	2009	2010	2011	2012	2013	2014
Auxiliary Material	RMB/Con t	59.93	54.22	25.00	25.00	25.00	25.00	25.00	25.00
Water & Power	RMB/Con t	64.53	62.65	60.00	60.00	60.00	60.00	60.00	60.00
Labour	RMB/Con t	20.68	13.19	13.00	13.00	13.00	13.00	13.00	13.00
Repair & Maintenance	RMB/Con t	2.07	2.31	2.50	2.50	2.50	2.50	2.50	2.50
Depreciation	RMB/Con t	54.69	32.57	24.00	24.00	24.00	24.00	24.00	24.00
Other	RMB/Con t	118.66	67.53	25.50	25.50	25.50	25.50	25.50	25.50
Total	**RMB/Con t**	**320.56**	**232.47**	**150.00**	**150.00**	**150.00**	**150.00**	**150.00**	**150.00**

Source: MCC provided Capex and Opex costs February 09

Using an upgrade ratio of 5t of ore per t concentrate (i.e. underground ore), the predicted operating costs appear reasonable based on current processing costs (20-25RMB/t). However, the actual cost is highly dependent upon the ratio of ore types to be treated i.e. soft to hard ores and the subsequent upgrade ratio, which is much greater for soft ores.

8.12 SAFETY AND ENVIRONMENT

An extensive set of safety documents were sighted, including the safety licences of the responsible Managers of Safety (two in the Mining Department and one in the Processing Department). Other safety documents included a list of the safety team members, safety qualifications for mining, development and construction as well as the storage of explosives and inspection standard for underground mining.

Personnel safety equipment was worn by operators in the processing plant and generally it appeared that a safety culture and program was being established at the Jinchang mine sites.

It is understood that an Environmental policy and subsequent monitoring program has been developed for Jinchang operations, however these aspects could not be fully confirmed on this visit. The tailings dam would appear to require more attention due to the proximity of stockpiled tailings to the river.

9 HONGDA IRON ORE MINE

M-MC made a site inspection of this property in February 2008 to review the resources, processing and mining. In March 2009 M-MC carried out a final site visit to review the resource base and geological controls on mineralisation, as well as update the current operational status. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "Wuguan Yinzi Iron Mining Design Report" by the Chengde Huatai Engineering Design Co., Ltd. March 2006.
- "Project of 20,000 tons/day Mining and Ore Dressing Feasibility Study" for Hongda Mining Industry Group Co., Ltd, Ningcheng County, Inner Mongolia Autonomous Region by Chengde Huatai Engineering Design Co., Ltd September, 2006.
- "Preliminary design of iron ore mining, Wuguangyingzi mine for Hongda Mining Ltd, by Chengde Huatai Engineering Design Co., Ltd. 2007 July
- MCC's effective equity stake in the Hongda Mine is 48.6% (90% of a subsidiary owning a 54% stake in the project).

9.1 BACKGROUND

The Hongda Iron Ore Mine is located 25km southwest of Chifeng City in Ningcheng County, Inner Mongolia Autonomous Region. The mine is owned and operated by Ningcheng Hongda Mining Co., Ltd, of which MCC Jingtang Construction Co., Ltd (MCC) holds 54% of shares while Chengde Iron & Steel Group Co., Ltd. holds 46% of company shares.

In 2004, a government team completed several surveys of the area including geological mapping at a scale of 1:5,000, ground magnetometer and IP surveys over the magnetite resource, prospecting trenches and 15 holes (1,477m) of drilling. An additional 5 holes have been drilled since 2004.

Tenure over the Hongda Mine area has been held since 2003. Mining commenced in 2005 with the establishment of a trial open cut and relatively small treatment plant. Following the successful mining trial and recovery of magnetite from the hard rock resource, a much larger plant was constructed between October 2006 and November 2007.

Open cut mining commenced January 2006, delivering ore to the plant with an installed capacity of 300ktpa of concentrate.

In 2007, a further expansion of the process plant was undertaken to increase the concentrate production to 600ktpa. ROM production was increased in September 2007 to meet the upgraded feed rate of the process plant.

The 2008 ROM ore production rate was 9.32Mt at a grade of 12.02%TFe and an open cut mining strip ratio of approximately 0.25 (t waste/ t ore). Concentrate production for 2008 was 410,048 tonnes at a grade of 60.74%TFe. In late 2008, the operation was put on a care and maintenance basis due to low commodity prices, with 3 months of stockpiled concentrate. MMC was notified by the Company that operations recommenced at the end of July 2009 after recovery at the Local Commodity Prices.

The Chinese Institutes report the term "ore" as economically extractable magnetite iron ore. The low ore grades are compensated by relatively simple mining methods and processing methods resulting in relatively low overall costs. Total overall costs of greater than 500RMB/t product for the hard ore type may be marginal or uneconomic.

9.2 ASSETS

The assets and status include;

- An open cut mining operation (care and maintenance)
- Tailings dam
- Two 3 stage crushing plants
- Two processing plants (12 Mtpa of capacity)
- Associated workshops, office and accommodation

9.3 LAND TENURE AND MINERAL RIGHTS

The Licence conditions are summarised in ***Table 9.1***.

Table 9.1 — Hongda Iron Mine — Mining License Details

Mine/Project	Hongda
Title	Mining License
No.	150000510461
Owner	Ningcheng County Hongda Mine Co,. Ltd
Mine/Project Name	Ningcheng Hongda mine Co,.Ltd Wuguanyingzi Iron Mine
Mine Method	Opencut
Permit Capacity	1,000kt/a
Permit Area	1.3101km2
Permit Depth	685-396mRL
Valid Date	June, 2005 — June, 2010
Issue Date	June, 2005
Issuer	Cifeng City Land and Resource Bureau

Source: Formal documentation

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

9.4 GEOLOGY

Magnetite mineralisation is hosted in an intrusive gabbro-pyroxenite which is medium to coarse grained. Geological age is late Archaean or early Proterozoic. The gabbro intrusive dimensions are oval shaped, approximately 1,500m long north-south and approximately 800m wide, surrounded by gneissic and granitic rocks. The gabbro itself is intruded by minor felsic and mafic dykes.

A regolith of sandy soils covers the deposit to a maximum depth of 15m (generally 2-5m). Drilling indicates a weathered zone ranging from 10 to 20m depth.

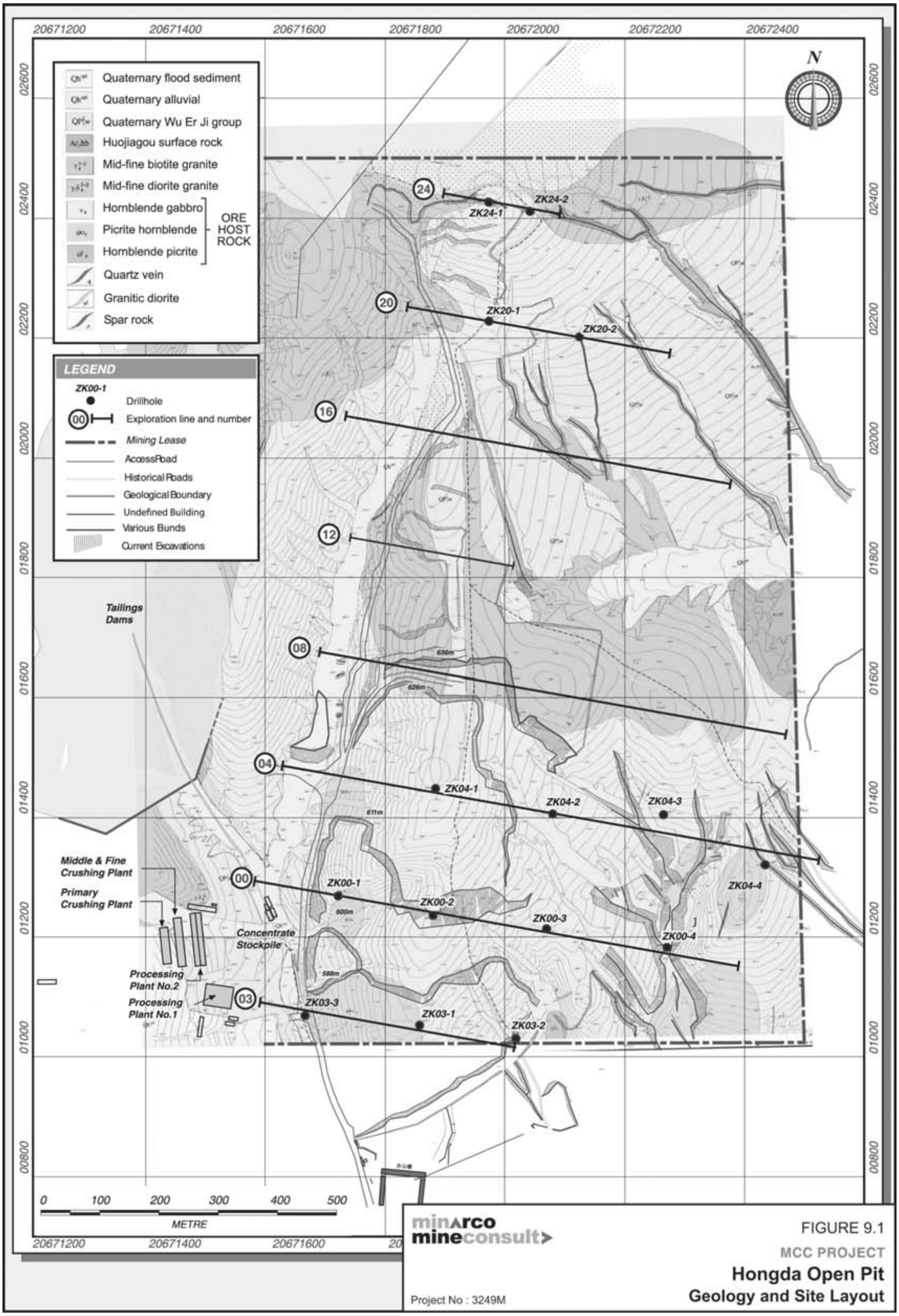
The geological map indicated that the gabbro-pyroxenite host body had granite mapped (surface mapping) in the centre of the mining area. This granite was interpreted (site geology) as flat-lying and approximately 20m thick. M-MC was able to verify the presence of a significant Quaternary alluvial palaeochannel in the area of the previously mapped granite (through mined exposures developed during 2008). The palaeochannel is at least 20m thick in the central portion of the southern mining area and trends in an approximate WNW-ESE direction. Sub-horizontal zones of calcrete are clearly visible and sharp contacts with the underlying gabbro-pyroxenite are present in some of the pit walls. However, there was an intrusive granitic body at the northern end of the mining area.

Magnetite occurs as discrete grains within the host rock. Mineral grain size or distribution is not reported. M-MC observed that grain size was approximately 2mm in size and appears to constitute about 45% of the rock mass based on theoretical plant recovery data. Total Fe content of approximately 12% TFe is reported. This includes the magnetic Fe (magnetite) plus additional Fe contained in ferro-magnesian minerals which comprise the majority of the host rock.

The ore naturally is vanadium-magnetite born in hornblende-pyroxene. Total Fe grade (TFe) is low and the mFe/TFe ratio is less than 85%. The ore is classified as extremely-low-grade, weak-magnetic iron ore.

Reports state that drilling indicates slightly increasing grades with depth. This is not supported by data as the drilling indicates consistent grades with depth consistent with a large low grade disseminated orebody.

Figure 9.1 — Hongda Iron Mine — Geology and Site Layout



9.5 RESOURCES AND RESERVES

9.5.1 Mineral Resources — In Situ Quantities

Mineral Resources were last estimated and reported by Chengde Huatai Engineering Design Co., Ltd in September 2006 (2006 Feasibility Study) as given in ***Table 9.2.*** A prior estimate carried out in March 2006 and quoted in the "Wuguan Yinzi Iron Mining Design Report" by the Huatai Engineering Design Co., Ltd reported a total In Situ Quantities of 175,000kt. This number was later reduced to 87,066kt in September 2006 report to full fill the local licence requirements.

Table 9.2 — Hongda Iron Mine — In Situ Quantities

Area	Chinese Codes 333 (kt)	2M22 (kt)	122b (kt)	Total (kt)	Average Grade %TFe	%mFe	Ratio mFe/TFe
North	2,615	2,177	7,194	11,986	12.20	5.77	47%
South	8,297	36,631	30,152	75,080	12.71	5.01	39%
Total	**10,911**	**38,808**	**37,347**	**87,066**	**12.64**	**5.11**	**40%**

Source: Design Reserve Statement September 2006

Note: 2006 estimate is similar to the 2004 exploration report with 80% factor applied to the 333 resources.
>8% TFe cut off grade
4m minimum mining height

M-MC has calculated an estimate of the depleted In Situ Quantities based on the 2006 Reserve Statement (***Table 9.2***) and the production figures for 2007 and 2008 as provided by site staff (***Table 9.4***). The M-MC estimate gives 73,455kt at 5.11%mFe. This is shown in ***Table 9.3***.

Table 9.3 — Hongda Iron Mine — M-MC Estimated Depleted Resource

Area	Chinese Codes 333 (kt)	2M22 (kt)	122b (kt)	Total (kt)	Average Grade %TFe	%mFe	Ratio mFe/TFe
2006 Total	10,911	38,808	37,347	87,066	12.64	5.11	40%
M-MC Total	**10,911**	**38,808**	**23,736**	**73,455**	**12.64**	**5.11**	**40%**

Source: M-MC estimate based on 13,611kt of ore mined in 2007 & 2008

Geological risks include:

- Lack of drilling results on Sections 8, 12 and 16 in the centre of the deposit (and mining area), and
- Lower ore grades indicated in the base of the pit may limit the depth of economic ore.

Geological opportunities include:

- Exploration drilling may identify higher grade zones within the host rock, and
- Exploration drilling may identify replacement reserves outside the Mining Lease.

9.5.2 Reserves — Mineable Quantities

Due to the current commodity prices and the costs associated with production of iron concentrate at the Hongda operation, there are currently no defined Ore Reserves or Mineable Quantities.

The life of mine (LOM) plan is based on a final pit with dimensions of approximately 1.4km by 700m and an average depth of 70m. The maximum high-wall height is 140m in the north. The pit shell indicates that the southern and northern pits will develop and converge. Estimated quantities within this pit shell are approximately 60Mt (site advice).

There is no geotechnical report, however in M-MC's opinion there are no significant geotechnical risks and that the designed pit batter of 45^o is reasonable.

Mine planning is prepared on site using manual methods with assistance provided by the Chinese Mining Institute using CAD software.

Estimation parameters:

Recovery	+95%
Dilution	2%

9.6 MINING

9.6.1 General Description

The Hongda Iron Ore Mine was based on a high production, bulk mining (quarrying) operation to achieve economy of scale and low ROM ore costs.

The mine was a conventional open cut mining operation. There were two separate operating pits at the southern and northern end of the deposit. Estimated ore losses are negligible and dilution is 2% (site advice) which is reasonable given the disseminated nature of the mineralisation and the low strip ratio. At the cessation of mining in November 2008, the Southern Pit adjacent to the process plant was developed with 5 active benches of 12m height. The northern area was still at a relatively early stage of development with most of the waste used to construct the tailings dam.

The 2008 ROM production rate was 9,316kt at 12.02%TFe grade and a strip ratio of approximately 0.25 (t waste/t ore). The minimum ore grade was 6%TFe with maximum grades up to 30%TFe.

Concentrate product for 2008 was 410kt at a grade of 60.76%TFe.

The mine was operated on a 3 x 8 hour shift roster and worked approximately 300 days per year.

Conventional mining methods were employed using 35t excavators and 40t trucks. Drill and blast (D&B) was required due to the hard material characteristics of ore and waste. The drill pattern for ore and waste was 7m × 5m and blasted using ANFO explosives. Previously, M-MC observed reasonable primary fragmentation with some occasional secondary blasting required.

Pit grade control was primarily managed by visual determination with some planning from geological maps. Based on the first visit, M-MC felt that grade control management was poor and may have resulted in the excavation of excessive waste in the northern pit resulting in a slightly higher strip ratio in this area.

9.6.2 Historic and Forecast Production

Historic production is shown in ***Table 9.4***. Since the mine has been placed on care and maintenance, the nature of any future production is unclear. However, it should be noted that the mine is capable of producing 12Mtpa and when iron ore prices exceed 500RMB/t, it would be expected that the mine could operate at this rate.

Table 9.4 — Hongda Iron Mine — Historic Production

Ore Production	Units	2,006	2,007	2,008
Open cut	kt Ore	1,370	4,295	9,316
Strip Ratio (Open cut Only)	Waste(t):Ore(t)	____	0.25:1	~0.25:1

Source: Site Advice

Mining risks include:

- Poor geological and grade control management impacting on higher strip ratio, lower ore recovery and higher dilution,
- Lack of site based planning could lead to production and development issues, and
- Lower grades (high upgrade ratios) indicated in the South Pit suggest economic limits at current depth.

Mining opportunities include:

- Exploration drilling may identify higher grade zones for selective mining,
- Improved geological and grade control management to improve strip ratios, ore recovery and dilution, and
- Improved efficiencies or additional equipment to increase mine production.

9.7 MINERAL PROCESSING

There are two processing facilities, namely the No. 1 and the recently constructed No. 2 processing plants, with each facility having four parallel lines. The equipment sizing in the older No. 1 plant is smaller than that in the No. 2 plant, however the flow sheet is the same and is similar to that shown in ***Figure 8.8*** for the Jinchang Assets. The major differences lie in the use of an extra stage of crushing as well as an extra stage of milling and magnetic separation to produce a final magnetite concentrate.

At full processing capacity rate of 12Mtpa the combined plant output should be in the order of 672kt of magnetite powder concentrate.

The crushing circuit consists of a primary jaw crusher followed by a secondary cone crusher. The cone crusher product is fed to a screen in closed circuit with a tertiary cone crusher. The screen undersize (-20mm) is upgraded by dry magnetic separator and 15% of the material is rejected as non-magnetic waste. The crushed ore is stored in concrete fine ore storage bins before being withdrawn to feed one of four primary ball mills (1.425MW motor) in closed circuit with a spiral separator. The spiral separator undersize feeds a wet magnetic separator to produce a magnetite concentrate, which is further upgraded. This upgrading consists of two further stages of milling, classification and magnetic separation to produce the final concentrate, which is then dewatered by vacuum disc filters (No. 2 plant) and vacuum drum filters (No. 1 plant), before being discharged onto the product stockpile. There is about 3 months of magnetite production stockpiled.

In the earlier visit, M-MC observed that magnetic separators were operating without level control, with some machines operating at too low a level and others overflowing. This would lead to losses both in the overflowing streams and in the failure to capture the magnetite.

The gabbro-pyroxenite iron mineralisation includes magnetite with minor proportions of pyrrhotite, pyrite and platinum group minerals. Site advice was that "impurities", such as sulphur and phosphorous, all report to tailings. This was not supported by any reports. No product analysis showing phosphorus and sulphur levels were available to confirm final product impurity levels. The magnetite also contains vanadium and titanium.

M-MC's brief examination of tailings and testing with a hand magnet indicated that a proportion of the contained magnetite is being lost. In M-MC's opinion, this indicates that processing efficiencies could be improved.

Processing risks include:

- Poor process control evidenced by magnetic material reporting to tailings and other plant spills and reject, and
- A proportion of the contained magnetite appears to be lost to tailings.

Processing opportunities include:

- Improved management of plant efficiency (metallurgical efficiency) by monitoring magnetite (mFe) vs total iron (TFe),
- Improved management of magnetic separators to operating with level control to improve magnetite recovery, and
- Improved plant efficiency by monitoring tailing grade to assess potential loss of concentrate.

9.8 INFRASTRUCTURE AND SERVICES

The Hongda operation is located in an active mining district and is well serviced by infrastructure and other services. The mine is located 9km from the town of Daming, 24 km from the provincial seat of government in Tianyi as well as 25km from Chifeng, a major industrial city in the region. There are sealed roads and highways connecting these towns and cities as well as the operation.

The operation consumes 250kWh/t concentrate (11.00kWh/ROM t) and this is sourced locally from a nearby (10km) government owned power generating facility. The facility was constructed during 2008 and power costs are 0.52RMB/kWh.

The operation requires 3 tonne of water per ROM tonne of feed, mainly for the production of the magnetite concentrate. About eighty percent of the water is recycled from the tailings dam and the additional water is reliably sourced from a nearby river.

The tailings dam is located close to the plant and has a capacity of 26 million m^3. It only has 18 months of capacity remaining and it is planned to construct a second dam with a capacity of 25 million m^3 during 2009.

Accommodation is provided on-site for the staff.

Major overhauls and repairs are conducted through local suppliers and maintenance contractors in nearby Chifeng.

9.9 CAPITAL AND OPERATING COSTS

Table 9.5 summarises the capital expenditure undertaken at the Hongda operation. Most of the expenditure over the last two years has been applied to increasing the processing capacity to 12mtpa and the new No. 2 processing plant is the most obvious result of this investment. Some capital expenditure was required to ensure that the mining rate would match the new processing capacity. There are no plans to increase the processing plant capacity any further.

Table 9.5 — Hongda Iron Mine — Historic Capital Expenditure

Cost Centre	Unit	2007	2008
CAPEX for mining expansion	RMB	2,060,000	1,145,000
CAPEX for processing expansion	RMB	130,000,000	50,000,000

Source: MCC provided Capex and Opex figures February 09

The Hongda operation exhibits low mining costs and it may be expected that similar mining costs would be experienced in future high volume mining activities (refer to ***Table 9.6***). Processing costs are acceptable for a reasonably energy intensive process (USD3.32/ROM t) where a relatively large quantity of ore is required to produce one tonne of saleable concentrate (22.72 ROM t for each tonne of concentrate).

Table 9.6 — Hongda Iron Mine — Historic Operating Costs

Cost Centre	Unit	2006	2007	2008
Mining Cost				
Auxiliary Material	RMB/ROM t	2.07	2.38	2.11
Labour	RMB/ROM t	0.28	0.11	0.06
Repair & Maintenance	RMB/ROM t	1.35	1.16	0.22
Other	RMB/ROM t	4.92	9.31	10.01
Total Mining Cost	**RMB/ROM t**	**8.62**	**12.96**	**12.4**
Processing Cost	**RMB/Con t**	**282.81**	**455.83**	**515.92**
Concentrate Sales	**RMB (10k)**	**508.8**	**1,199.1**	**1,472.9**
Management Fees	**RMB (10k)**	**420.9**	**1,898.2**	**3,661.3**
Administration Cost	**RMB (10k)**	**421**	**1,892**	**3,661**
Other	**RMB (10k)**	**1,826**	**—**	**—**

Source: MCC provided Capex and Opex figures February 09

The concentrate is sold to the Chengde Steel Group, who own 46% of the operation. Apparently a premium of 10-20RMB/tonne of concentrate is paid for the vanadium content. ***Table 9.7*** summarises the recent concentrate production and the prices received for the concentrate product.

Table 9.7 — Hongda Iron Mine — Historic Concentrate Production and Prices Received

Details	Unit	2006	2007	2008
Magnetite Concentrate	tpa	15,199	217,369	410,048
Concentrate Grade	% Fe	59.34	59.56	60.74
Price	RMB/t	390.22	649.70	956.59

Source: MCC provided Capex and Opex figures February 09

9.10 SAFETY AND ENVIRONMENT

The Hongda operation has a safety policy that covers both construction and operational activities. The policy uses the best safety equipment and methodologies available and actively trains employees to work in a safe manner. Due to the current status of the operation, M-MC were unable to confirm the practical implementation of such policies. Nonetheless, the No. 2 plant was well designed and had clearly adopted many features that would encourage safe practices and minimise the potential for accidents to occur.

The operation also has an active environmental program, with a strong focus on dust, noise and water. Dust suppression is employed in the mine while dust capture is used during the dry processing stages of crushing and magnetic separation. Water quality is actively monitored based on Chinese Government standards, even though little water is released into the environment. Further attention may be required to minimise the dust losses associated with the strong winds blowing across the tailings dam.

Rehabilitation is undertaken through the planting of trees, particularly on the tailings dam.

10 XIANGXI CARBON SHALE VANADIUM PROJECT

M-MC made a site inspection of this property in February 2008 to review the resources, processing and mining options and the general site layout. In March 2009 M-MC carried out a final site visit to review the resources and geology, as well as update the current project status. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "The Geological Survey Report for Carbonaceous Shale (Stone Coal) of Xinglongchang Mine, Luxi County, Hunan Province" by No. 405 Team, Hunan Bureau of Exploration & Development of Geology & Mineral Resources in August, 2007 (2007 Geology Report).
- "Comprehensive Utilization & Survey of Carbonaceous Shale (Stone Coal) of Xinglongchang Mine, Luxi County, Hunan Province" (2007 Feasibility Report).

The project is currently awaiting final government approvals before proceeding to final plant and site design.

MCC's effective equity stake in the Xiangxi Project is 80%.

10.1 BACKGROUND

The Xiangxi Carbon Shale Project is located near Xinglongchang Township in Luxi County, Hunan Province (***Figure 10.1***). China Metallurgical Xiangxi Mining Co Ltd's (CMXM — MCC 80%) initial plan was to build a power plant capable of burning carbonaceous shale (stone coal) from the Xinglongchang Carbonaceous Shale deposit. The carbonaceous shale is a high ash, low energy fuel type.

The deposit also hosts a vanadium rich horizon in the lower section of the carbonaceous shale seam. This vanadium oxide mineralisation (V_2O_5) is being considered for separate extraction. Technical grade V_2O_5 is produced as a concentrate (black powder) used for the production of vanadium metal and ferrovanadium (strengthened steel). Vanadium is also used as a catalyst for producing sulphuric acid.

Extraction of the vanadium ore is now the primary focus for ongoing production studies and planning. Power station feed shale is not currently being considered.

Initial studies included a report "The Geological Survey Report for Carbonaceous Shale (Stone Coal) of Xinglongchang Mine, Luxi County, Hunan Province" (submitted by No. 405 Team, Hunan Bureau of Exploration & Development of Geology & Mineral Resources in August, 2007 (2007 Geology Report). A subsequent report "Comprehensive Utilization & Survey of Carbonaceous Shale (Stone Coal) of Xinglongchang Mine, Luxi County, Hunan Province" (2007 Feasibility Report) considered the mining of both the upper and lower mining horizons. The products would be carbonaceous shale for power generation and vanadium ore for production of a vanadium concentrate.

Initial mining studies considered open cut mining methods. The current mining schedule includes only vanadium ore quantities from the lower mining horizon. The co-product is waste heat from the Vanadium calcining process which may also be used for power generation. The mining schedule does not include any volumes of carbonaceous shale from the upper mining section. Extraction of carbonaceous shale from the upper mining section for power station feed represents an additional opportunity for the Xiangxi Project.

The current mining schedules, combined with objectives of the 2007 Feasibility Report include:

- Principal extraction of vanadium oxide (V_2O_5) from stone coal (carbonaceous shale)
- Comprehensive utilisation of waste heat from carbonaceous shale as a co-product of vanadium ore processing
- Generating capacity of 173 million kwh and yearly power supply capacity 159 million kwh
- To produce 500ktpa glued stone cement from slag after V extraction.

Site advice indicated that options exist for the power plant location to be close to the mine site and the vanadium processing plant to be located in the SW corner of the lease.

The mine site lease area contains a water storage reservoir (Loutouchong) with a storage capacity of between 5M and 7.3M cubic metres. The dam overflow level is assumed to be the 324mRL contour.

10.2 ASSETS

- Comprehensive utilisation feasibility report compiled in 2007.
- MCC reported In Situ Quantities of 71.1Mt at 3,507J/g of carbonaceous shale an 17.147Mt at 0.79% V_2O_5 vanadium ore.

10.3 LAND TENURE AND MINERAL RIGHTS

The Xinglongchang Mining area is approximately 70km southwest of Luxi and 90km from Jishou via county roads and a national highway towards the western edge of Hunan Province. The mining rights belong to the China Metallurgical Xiangxi Mining Co Ltd (CMXM) valid until April 2014. Licence Details are shown in ***Table 10.1.***

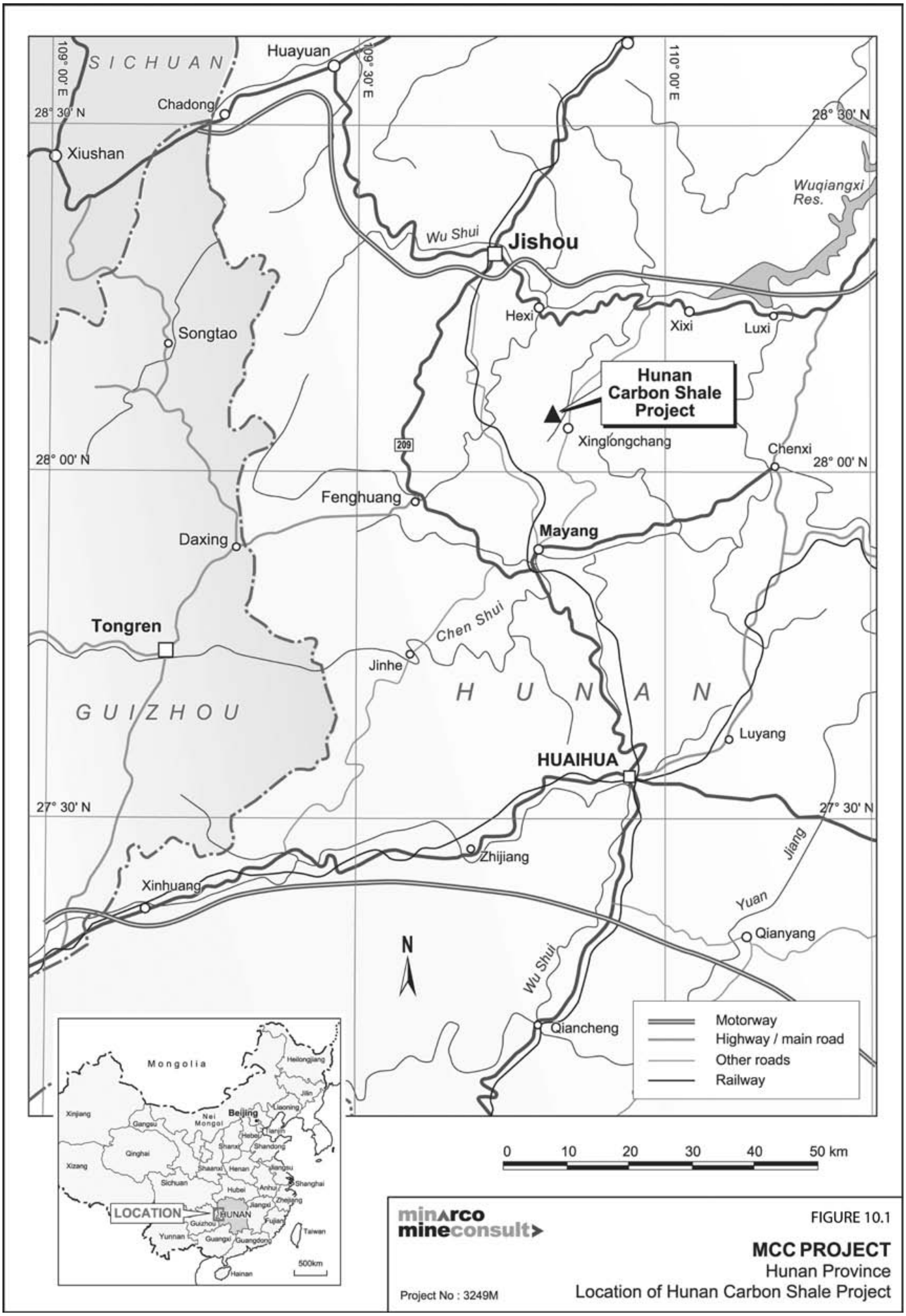
Table 10.1 — Xiangxi V Shale Project — Mining Licence Details

Mine/Project	Xiangxi Carbon Shale
Title	Mining Licence
No.	4331220510288
Owner	MCC Xiangxi Mine Co., Ltd
Mine/Project Name	Luxi County, Xinglongchang Carbon Shale Ore
Mine Method	Opencut
Permit Capacity	1,000ktpa
Permit Area	2.94km^2
Permit Depth	elevation 370mRL to 270mRL
Valid Date	April,2005 to April,2015
Issue Date	April,2005
Issuer	Luxi County Land and Resource Bureau

M-MC validated these coordinates of the Mining Lease area with lease plans used to report the carbonaceous shale and vanadium In Situ Quantities.

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

Figure 10.1 — Xiangxi V Shale Project — Project Location



10.4 GEOLOGY

10.4.1 Background and Previous Work

Previous geological exploration comprised regional mapping targeting mineral deposits in the Cambrian strata. In the 1950’s, the aeromagnetic team of the Ministry of Geology conducted 1:500,000 regional gravity and aeromagnetic surveys. Between 1969 to 1975, 405 Team of Hunan Provincial Geology and Mineral Resources Bureau in Jishou, Guzhang, Luxi and Yongshun area conducted geological mineral prospecting and exploration work for nickel (Ni), molybdenum (Mo), vanadium (V), multi-metal and phosphorus (P) minerals. Between 1987 and 1991, a regional stream sediment geochemical survey identified metal anomalies of Ni, Mo and V within Cambrian Age black shales.

Site advice reports no previous mining of carbonaceous shale in the area. However, there has been some mining activity in adjacent areas extracting approximately 100kt of zinc (Zn) ore.

In April to July 2006, the 405 Team of the Hunan Provincial Metallurgical and Geological Prospecting Bureau (405 Team) completed field mapping of the carbonaceous shale strata. The 405 Team completed detailed mapping at 1:10,000 over a 1 sq km area and included trench excavations and testing of 100 rock samples.

In January 2007, the 405 Team concentrated exploration work at 1:5,000 scale. Filed work included mapping of the carbonaceous shale, 25 diamond core drill holes totalling 2,321 metres (core viewed on site), 38 trenches and taking nearly 1,000 rock samples. The exploration was completed by May 2007.

10.4.2 Local Geology and Mineralisation

The carbonaceous strata are Lower Cambrian age, formed in an environment of shallow marine sediments which produced black shales in the Niutitang Formation. Quality characteristics are a very low energy and high ash, suitable as a product for a mine mouth power station.

The strata strikes north-south and the carbonaceous shale resource is separated into a northern block and southern block by the water storage reservoir. The geology of the deposit is reasonably simple, with a consistent dip to the east at approximately 10° to 20°. Drilling in the northern block indicates a fault that trends northeast-southwest, that has a displacement in the order of 20m downthrown to the west. Structure contour levels to the base of the carbonaceous shale indicate no significant structural features. The depth of weathering is approximately 10 to 20 metres.

The carbonaceous shale resource is overlain by shale and underlain by silty sandstone. The average thickness of the carbonaceous shale horizon is 15 to 40 metres thick. The carbonaceous shale has an upper carbonaceous section and a lower vanadium oxide (V_2O_5) rich section approximately 5 to 10m thick. A summary of the stratigraphy and mining section characteristics is shown in ***Table 10.2.***

Table 10.2 — Xiangxi V Shale Project — Stratigraphy and Mining Sections

Formation	Section	Lithology	Mining Section	Thickness (m)	CV (Joules/gm)	Vanadium (% V_2O_5)	Comments
Balang		Black Shale IB with siltstone					
Niutitang	Upper	Slate		60			
	Middle	Carb Shale	CS	15-40	3,350-4,000		Higher energy towards base
	Lower	Carb Shale	V4		3,350-3,800		Increasing IB siltstone towards base
		IB Carb Shale & Silt	V3	2-3		2.21%	
		IB Silt & Carb Shale	V2	2-3		0.30%	
Niutitang	Lower	Silt IB Carb Shale	V1		2,500-3,500		
Dengying		Siliceous Shale					

Notes: IB = Interbedded
Carb Shale = Carbonaceous Shale

The main mining section for carbonaceous shale is CS. The main mining sections for vanadium are V2 and V3. The V1 and V4 sections are secondary (lower grade) V sections. The 2007 Exploration Report states that energy value is lower (approximately 2,500 to 3,500J/g) in the lower V mining section.

The carbonaceous shale quality with regard to thermal coal is low quality with very high ash, high sulphur and very low energy contents. Quality of the carbonaceous shale section CS is shown in ***Table 10.3***. The reported units of energy are Joules per gram (J/g). The conversion to kcal/kg is 4.19. The highest energy content sampled is 4,449J/g. The average energy of 3,500J/g is equivalent to approximately 850kcal/kg. While this energy content is very low, it compares favourably with the adjacent 925 Power Plant in Yiyang City which utilises carbonaceous shale with a thermal value of 3,350 J/g.

Table 10.3 — Xiangxi V Shale Project — Carbonaceous Shale Quality

Sample No.	#1 Moisture % IM	Ash % Ash	Fixed Carbon % FC	Volatile Matter % VM	Energy Joule/g	Sulfur %S
ZK3-1	0.12	87.96	5.59	6.33	3403	3.18
ZK9-1	0.13	86.68	7.49	5.70	3567	2.97
Average	**0.13**	**87.32**	**6.54**	**6.02**	**3485**	**3.08**

Source: 2007 Feasibility Study and 2007 Exploration Report
Notes: #1 M-MC assume quality is reported on an, as dried basis (adb) indicated by low moisture values

The vanadium rich lower section has 4 units as shown in ***Table 10.2***. The lower mining section lithology is interbedded carbonaceous and siliceous shale. The average grade across the vanadium-rich section is 0.80% to 1.2% V_2O_5. However, the main host lithology for vanadium is the carbonaceous black shale with grades of 2.2% V_2O_5. The siliceous shale has much lower grades of 0.23% V_2O_5.

There is some evidence that the grade of vanadium increases in the oxidised (weathered) zone.

Figure 10.2 — Xiangxi V Shale Project — Geology Map

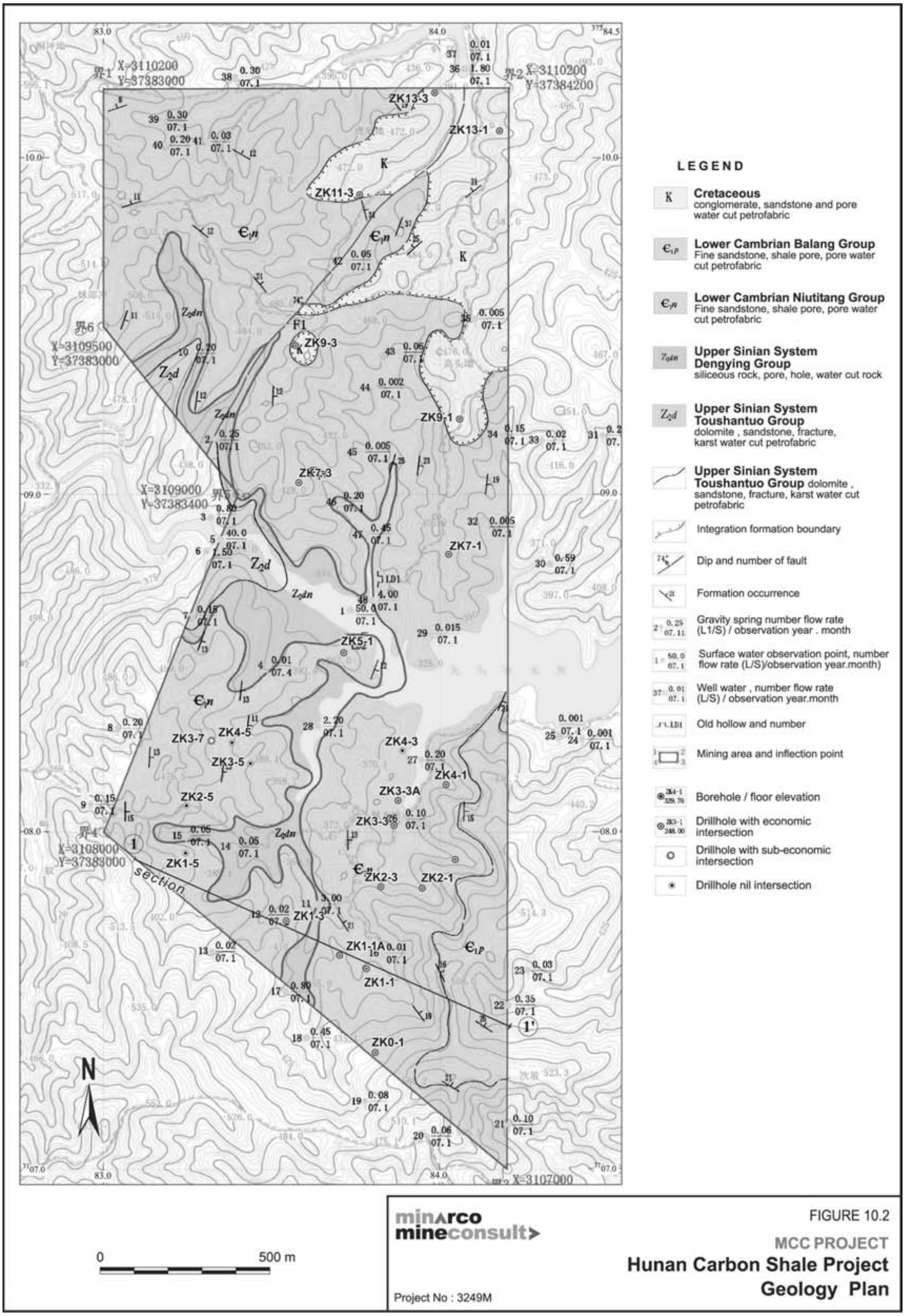
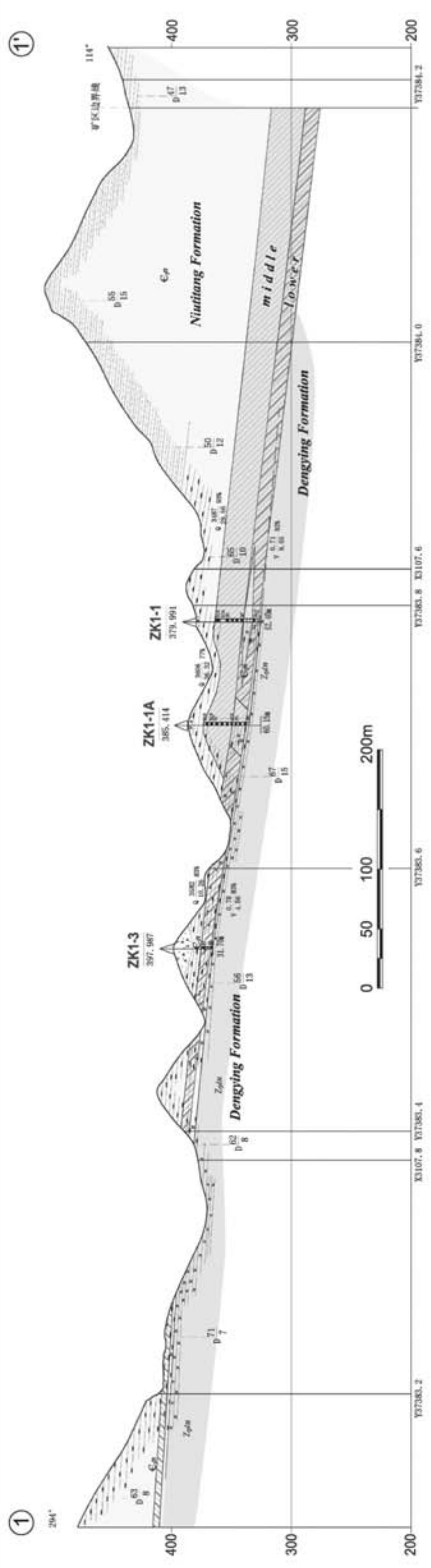


Figure 10.3 — Xiangxi V Shale Project — Geological Type Cross Section



SEAM

- Carbonaceous Shale
- Vanadium rich zone
- ZK1-1 ◉ Drillhole collar and number

ZK1-3

		Depth (m)				Analysis	
		From	- To	Length	Depth	J.g⁻¹	V_2O_5(%)
ZK1-3-H6		19.40	23.20	3.80	3.61	3399	
H1	ZH1		24.40	1.20	1.14	3627	0.67
H2			26.20	1.80	1.71		0.70
H3	ZH2		28.00	1.80	1.71	3722	0.93
H4			29.20	1.20	1.14		0.40
H5			30.20	1.00	0.95		0.40

ZK1-1A

		Depth (m)				Analysis	
		From	- To	Length	Depth	J.g⁻¹	V_2O_5(%)
ZK1-1A-H16		8.53	11.95	3.42	3.10	3658	
H17			15.20	3.25	2.95	3523	
H18			18.30	3.10	2.81	4449	
H19			21.90	3.60	3.26	4213	
H7			23.80	1.90	1.79	3810	
H8			26.20	2.40	2.26	3039	
H9			28.70	2.50	2.35	3523	
H10			31.00	2.30	2.16	1978	
H11			33.20	2.20	2.07	3283	
H12			34.80	1.60	1.50	3386	
H13			36.30	1.50	1.41	3810	
H14			38.40	2.10	1.97	3879	
H15			39.70	1.30	1.22	4017	
ZK1-1A-H1	ZH1		41.83	2.13	2.00	3790	0.27
H2			43.95	2.12	1.99		0.22
H3	ZH2		45.45	1.50	1.41	3413	0.08
H4			47.65	2.20	2.07		0.17
H5	ZH3		49.45	1.80	1.69	2119	0.33
H6			50.85	1.40	1.32		0.37

ZK1-1

		Depth (m)				Analysis	
		From	- To	Length	Depth	J.g⁻¹	V_2O_5(%)
ZK1-1-H14		15.20	20.20	5.00	4.53	3063	
H15			25.60	5.40	5.07	3397	
H1	ZH1		27.30	1.70	1.54		0.27
H2			29.50	2.20	1.99	3929	0.22
H3			31.10	1.60	1.51		0.08
H4	ZH2		33.30	2.20	2.08	3551	0.17
H5			35.11	1.80	1.70		0.33
H6	ZH3		37.81	2.70	2.55	3323	0.37
H7			39.76	1.95	1.84		0.33
H8	ZH4		42.31	2.55	2.40	2886	0.69
H9			44.21	1.90	1.79		1.23
H10	ZH5		46.51	2.30	2.16	3718	0.46
H11			48.01	1.50	1.40		0.53
H12	ZH6		49.51	1.50	1.40	3422	0.56
H13			51.70	2.19	2.04		0.52

STRATIGRAPHIC COLUMN

Formation	Member	Lithology	Mining Section	Thickness (Avg m)
Balang		Black shale IB with siltstone		
Niutitang	Upper	Slate		60
	Middle	Carb Shale	CS	20
	Lower	Carb Shale	V4	<2
		IB Carb Shale & Silt	V3	2 to 3
		IB Carb Shale & Silt	V2	2 to 3
		Silt IB Carb Shale	V1	<2
Dengying		Siliceous Shale		

(IB = interbedded)

minarco mineconsult

FIGURE 10.3

MCC PROJECT

Hunan Carbon Shale Cross Section 1-1'

Project No : 3249M

10.5 RESOURCES AND RESERVES

10.5.1 Mineral Resources — In Situ Quantities

In Situ Quantities (Chinese Code) for both the carbonaceous shale and V_2O_5 deposits were estimated and reported by the 405 Team using manual polygonal methods. M-MC was provided with the original exploration report and subsequent "2007 Development and Utilisation Report". Only the "Exploration Report" contains a breakdown of the carbonaceous shale by blocks and Chinese Code. For this reason M-MC has included the data from the Exploration Report in it's resource table.

Estimates of the upper mining section, carbonaceous shale are shown in ***Table 10.4.***

Table 10.4 — Xiangxi V Shale Project — Carbonaceous Shale In Situ Quantities

	Upper Mining Section — Carbonaceous Shale				In Situ Quantities Chinese Codes			
	Dimensions							
Block	Thickness (Avg m)	Area (m^2)	Volume (m^3)	Estimated Density (t/bcm)	334 (Mt)	333 (Mt)	Total (Mt)	Quality Energy SE (J/g)
North				#1				
1	18	472,962	8,513,316	2.25		19.141		3,547
2	21	290,995	6,110,895	2.26	13.786			3,357
3	21.65	78,382	1,696,970	2.22	3.767			3,774
4	22.94	41,877	960,658	2.22	2.133			3,411
5	20.97	206,204	4,324,098	2.22	9.600			3,504
South								
1	25	30,624	765,600	2.24		1.718		3,465
2	20	315,003	6,300,060	2.23	14.070			3,490
3	15.5	199,877	3,098,094	2.23	6.900			3,393
Total	**20.67**				**50.256**	**20.859**	**71.114**	**3,507**

Source: Exploration Report 2007, Resource plans.

Notes: Density default 2.22 t/bcm.

#1 Density for blocks back-calculated by M-MC.

* *Development and Utilisation Report In Situ Quantities did not provide a break down of the resource into separate blocks and therefore could not be reported by M-MC.*

M-MC notes that the estimates of In Situ Quantities do not include potential resources to the western side of the lease, up dip. This may be due to oxidation (weathering) of the relatively shallow carbonaceous shale in this area. These areas represent potential additional carbonaceous shale resources. The carbonaceous shale in the western area sub-crops around truncated topography. There is an opportunity for contour mining these shallow and potential open cut resources.

The area of the reservoir within the lease is approximately 0.25sq.km. The dam area has sterilised potential carbonaceous shale resources in the order of 10Mt.

Mineral Resource estimates of the lower mining section, V_2O_5 ore are shown in ***Table 10.5.*** This table from the "2007 Feasibility Report" shows a change in Chinese Code classification based on design work carried out by the Institute. M-MC considers the estimated amounts and grades reasonable.

Table 10.5 — Xiangxi V Shale Project — Vanadium In Situ Quantities

Lower Mining Section — Vanadium				In situ Quantities Chinese Code		Average Grade%	Contained Metal V_2O_5 Chinese Code	
Orebody No	Thickness (Avg. m)	Area sqm	Density t/bcm	334kt	333kt	V_2O_5	334kt	333kt
I-1	3.84	54,001	2.22		460.4	0.81		3.73
I-2	6.18	21,378	2.22		293.3	0.77		2.26
I-3	7.38	68,904	2.22		1,128.9	0.77		8.69
I-4	5.34	54,469	2.22	645.7		0.79	5.10	
I-5	2.13	103,451	2.22		489.2	0.82		4.01
I-6	4.68	55,207	2.22	573.6		0.82	4.70	
I-7	7.03	48,657	2.22	759.4		0.77	5.85	
I-8	6.79	138,686	2.22	2,090.5		0.78	16.31	
I-9	5.98	109,718	2.22	1,456.6		0.77	11.22	
Sutotal South				**7,897.6**		**0.78**	**43.17**	**18.69**
II-1*	4.67	47,881	2.22		513.9	0.73		3.75
II-2	3.67	48,665	2.22	396.5		0.8	1.60	
II-3	4.64	654,277	2.22	6,739.6		0.81	54.59	
II-4	2.86	139,828	2.22	887.8		0.77	6.84	
III	4.95	64,789	2.22	712.0		0.9	6.41	
Subtotal North				**9,249.8**		**0.81**	**69.43**	**3.75**
Total	**5.01**	**>0.7 cog**		**17,147.39**		**0.79**	**135.05**	

Source: 2007 Feasibility Report

Notes: cog; cut off grade

** Computation error should have 496.4kt of In Situ Quantities and not 513.9kt*

The same resource has been reported at a 0.9% V_2O_5 cut off grade yielding 4,244.99kt of In Situ Quantities at 0.98% V_2O_5.

Estimates of In Situ Quantities for V_2O_5 ore include the western area where carbonaceous shale was excluded from estimates of In Situ Quantities.

The area of the reservoir within the lease is approximately 0.25sq.km. The dam area has sterilised potential V_2O_5 ore resources in the order of 2.7 Mt

10.5.2 Reserves — Mineable Quantities

Mineable quantities have been estimated by MCC in previously reported open cut life of mine schedules, produced for the project. No details of the parameters used in these estimates are available and M-MC assumes these have been calculated using simple recovery factors. The reported recoverable Mineable Quantities are outlined in ***Table 10.6***.

Table 10.6 — Xiangxi V Shale Project — Vanadium Mineable Quantities

Tonnes (kt)	Grade (V_2O_5 %)	Vanadium Concentrate (kt)
13,000	0.79	104

Source: MCC 2008 — 12 year life of mine open cut plan

Note: Grade reported is undiluted

Due to the lack of supporting evidence such as mine plans and design as well as the uncertainty regarding the potential approval of an open cut mine in the area, the Mineable Quantities reported are considered by M-MC to be preliminary in nature. Significant amounts of work will be required to finalise the mine plans once approval is granted which could result in marked changes to the reported Mineable Quantities. M-MC also does not consider the use of Chinese Code 334 or 333 In Situ Quantities as reasonable for the estimation of Mineable Quantities when compared to JORC equivalent terms.

10.6 MINING

10.6.1 Mine Planning

The initial planning for the Xiangxi Project was an open cut mining method to include the carbonaceous shale and vanadium mining sections. The initial mining strategy was primarily for power station supply with co-processing of vanadium ore.

In M-MC's opinion, the relatively undisturbed stratiform geology, shallow dips with consistent thickness and grades provide an opportunity for large scale, high production mining. The geological conditions would provide only reasonable mining conditions for underground mining methods, since the friable nature of the dominant matrix material i.e. shale indicate that Room and Pillar mining techniques would be required.

Site discussions suggested that it is unlikely that open cut mining would be permitted in this water catchment area. An underground mining method sequence was described on site that included extraction of each mining section from separate underground entries. This mining strategy was not supported by detailed mine plans or a schedule that included Mineable Quantities of both the carbonaceous shale mining section and the vanadium mining section. However, the strategies presented appeared conceptually reasonable.

The current mining schedule (***Table 10.7)*** is conceptual and includes mining of both vanadium ore quantities from the lower mining horizon and carbonaceous shale from the upper mining horizon utilising an open cut approach. The co-product is a slag which site staff indicated could be sold for cement manufacturing.

Final details of mining production rates will only be determined once mining approvals have been granted by the government. This had yet to happen in March 2009 when M-MC carried out its latest site visit.

M-MC assume:

- The schedule as confirmed by MCC is for open cut mining of both the upper and lower mining sections.
- Site advice indicates schedule calculated using: In Situ Quantities recovery of 70%
  Processing recovery of 62%

Inconsistencies in the vanadium ore schedule include:

- ROM ore grade appears to be In Situ (i.e. no mining dilution applied)

Table 10.7 — Xiangxi V Shale Project — Mining Schedule

Description	Unit	2008	2009	2010	2011	2013	2014
ROM							
ROM Tonnes	kt	—	570	570	570	570	570
ROM V_2O_5 Grade	%	—	0.797	0.797	0.797	0.797	0.797
Carbon Shale	kt	—	43	43	43	43	43
Product							
V_2O_5	t	—	2,850	2,850	2,850	2,850	2,850
Power	M kWh	—	173	173	173	173	173
Recovery as V_2O_5 (V ore)	%	—	62.7	62.7	62.7	62.7	62.7

Source: MCC provided Capex and Opex figures February 09

Mining risks include:

- The proposed mining strategy both open cut and underground is not supported by detailed plans, consistent schedules or a subsidence study,
- Governmental approval for open cut may be hard to receive within the water catchment area
- Poor ground conditions linked to friable shales will require careful management in the underground,
- The underground barrier on the dam may not include an angle of draw,
- Lack of detailed mine planning and extraction methods for separate mining sections (carbonaceous shale and vanadium ore).

Mining opportunities include:

- Consistent orebody within regular stratiform horizons,
- Prepare separate mine plans and schedules for separate mining sections (carbonaceous shale and V).

Figure 10.4 — Xiangxi V Shale Project — Structure Floor Resource Estimates

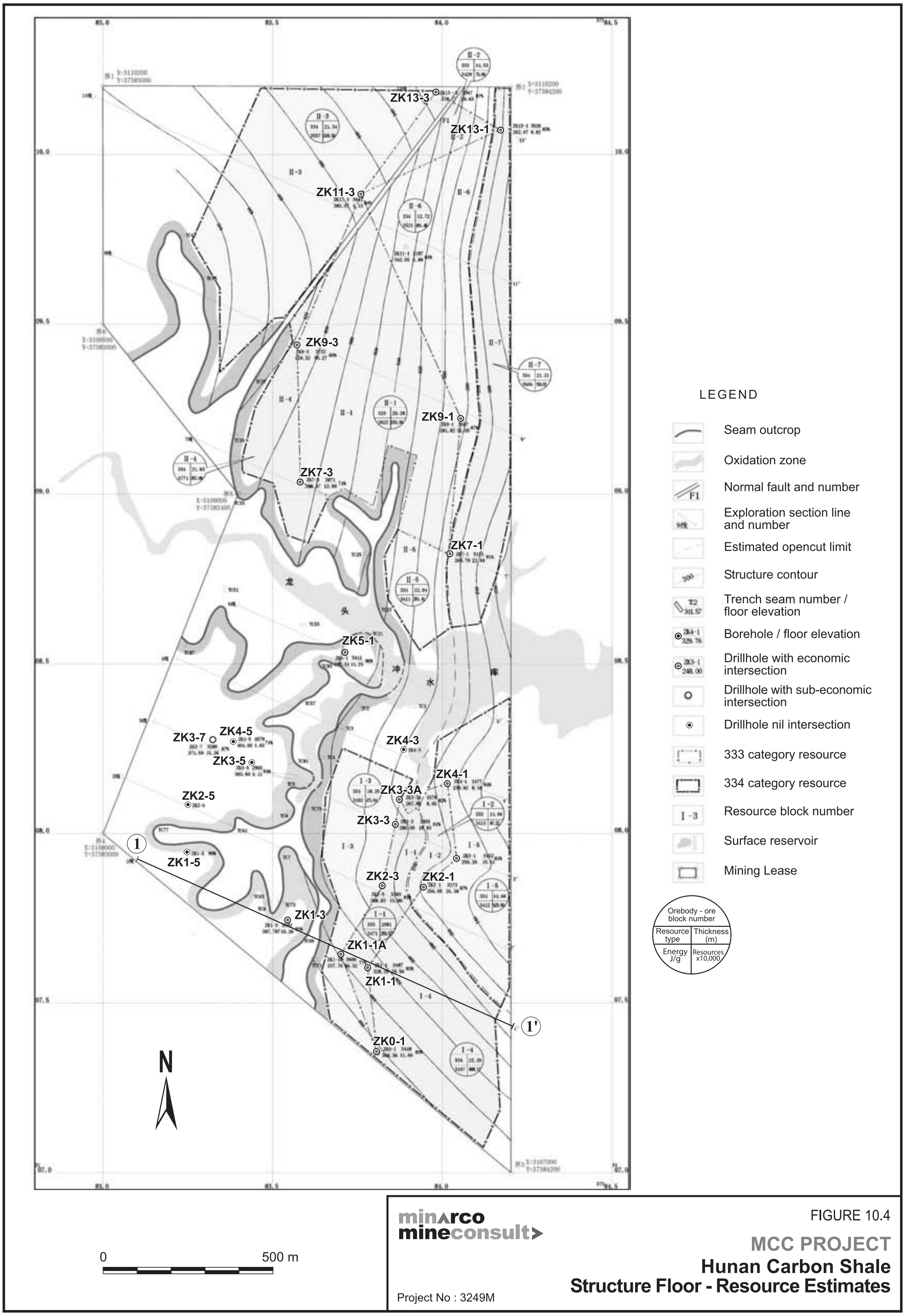
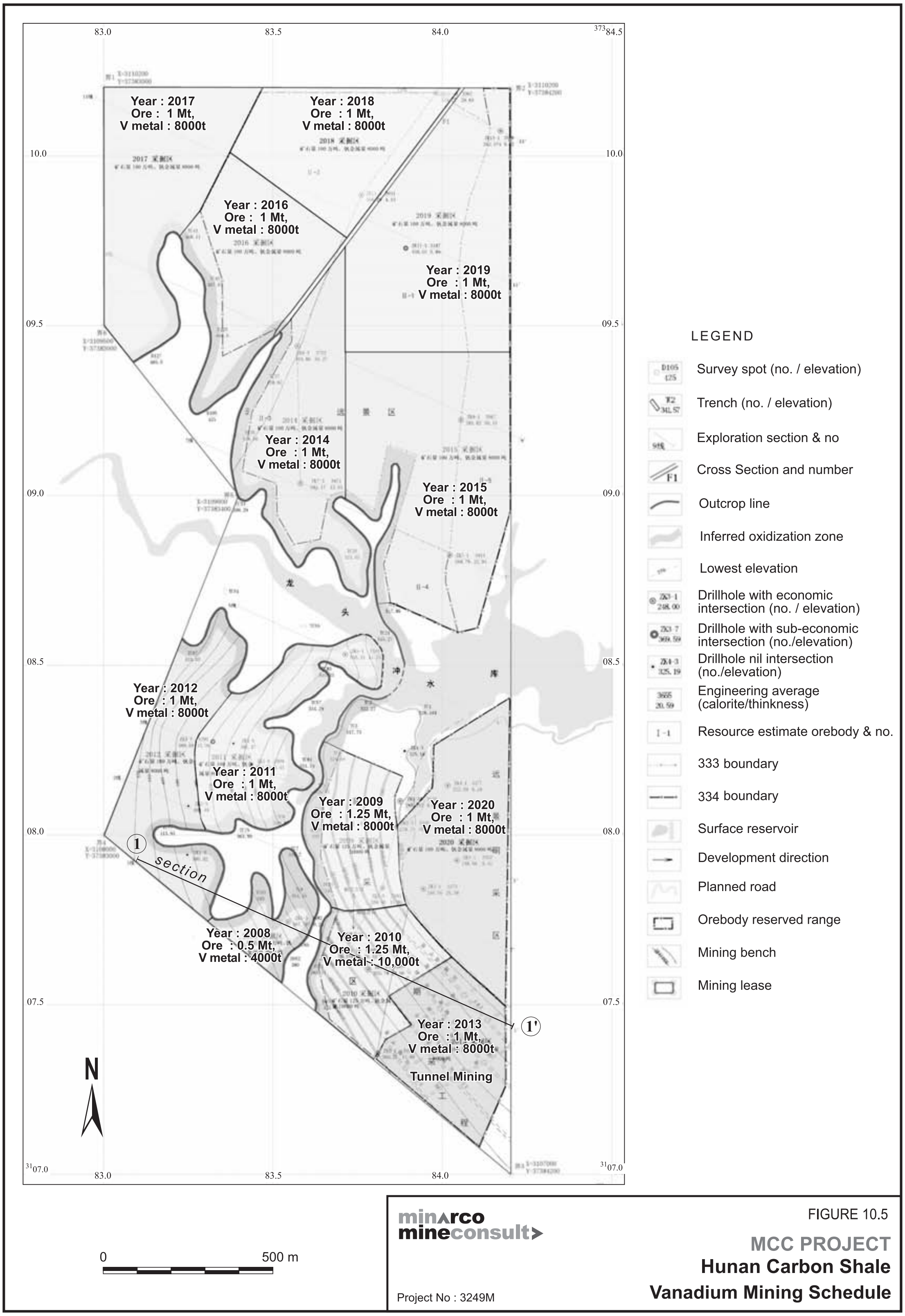


Figure 10.5 — Xiangxi V Shale Project — Mining Schedule



10.7 MINERAL PROCESSING

The objectives of the 2007 Feasibility Study include:

- Extraction of vanadium (V) from carbonaceous shale
- Comprehensive utilisation of waste heat from carbonaceous shale as a co-product of vanadium ore processing
- Power generating capacity of 173 million kWh and power supply capacity 159 million kWh per annum
- Production of a cement product at 500ktpa from slag after V extraction

A detailed study of the vanadium processing flowsheet as well as the power generation from waste heat was not available.

Vanadium Processing:

The conceptual flowsheet is based upon the extraction of vanadium by roasting with salt followed by leaching and ion exchange to produce a fused vanadium pentoxide (V2O5) product that would meet the national standard specification (GB32 83-1987).

Waste Heat Power Generation:

Proposed plans include the generation of power from waste heat during the extraction of vanadium from the carbonaceous shale using a steam turbine and a generator. Conceptual plans are to reticulate power to local areas and possibly into the provincial grid. Additional extraction of carbonaceous shale from the upper mining section provides an opportunity to increase power generation.

Tailings dam:

The mining and processing studies are not supported by details of a tailings dam. Site advice indicated that the dam may be located along southern boundary of the mining lease. Site advice indicated possible disposal of some tailings as underground fill. Since only a small part of the processing requires water, most of the waste products would be of a dry nature and probably involve simple dry placement techniques.

Co-Product Glued Stone Cement:

Glued stone product is part of an environmental and sustainable objective to fully utilise resources, co-products, waste heat and tailings. The conceptual plan is to fully utilize the slag after vanadium extraction to produce glued stone cement with annual production of 500kt/year.

Glued stone has the advantages of sand consolidation, soil-fixation, corrosion resistance, high strength, etc. The strength of the glued stone materials increases to some certain degree if glued stone is immersed in acid or alkali solutions. Glued stone has very good anti-freezing and thawing performance.

Processing risks include:

- Unconfirmed burn test of Xiangxi carbonaceous shale from vanadium mining section,
- Low energy of Xiangxi carbonaceous shale may require other fuel for ignition and sustainability of burning, and
- Lack of details of tailings disposal or tailing dam capacity.

Processing opportunities include:

- Additional and higher energy carbonaceous shale from upper mining section, and
- Additional feed (upper mining section) for potential expansion of power generation.

10.8 INFRASTRUCTURE AND SERVICES

The Xinglongchang Mining area is located in a rural area dominated by subsistence farming. The project is located approximately 70km southwest of Luxi and 90km from Jishou via sealed county roads and a national highway towards the western edge of Hunan Province. Site advice indicates that approximately 3km of roads are required to be constructed from the existing county road to the planned mine site.

Grid power lines transect the project area however additional power would be required either using waste heat from the vanadium production or possibly using hydro power (site advice).

The current mining study includes recycling of all site water with minimal need for additional water generation and presumably water would be sourced from the nearby lake to satisfy the initial processing requirements.

No detailed information regarding the tailings dam site or construction was made available to M-MC during the site visits.

10.9 CAPITAL AND OPERATING COSTS

There are no detailed reports of mining or processing costs. Indicative costs and project value were included in the 2007 Exploration Report and Feasibility Study for the previously planned open cut. These have subsequently been updated by MCC in February 2009 (***Table 10.8***).

The mining and processing costs supplied by MCC include CAPEX costs relating to repairs and maintenance but not construction. In M-MC's opinion the total operating costs appear reasonable when compared against other similar operations.

The 2009 Capex and Opex information provided by MCC included the following indicative estimates. Note that the operating costs are assigned to the vanadium-rich ore production rate (i.e. 570,000tpa) and not the total quantity of material mined:

Table 10.8 — Xiangxi V Shale Project — Indicative Capital and Operating Costs

Item	Units	2009	2010	2011	2012	2013	2014
Mining unit cost:	RMB/ ROM V ore t	–	35.9	36.3	37.0	37.5	38.0
Processing Cost	RMB/ ROM V ore t	–	260	263	267	271	275
Power generation cost	RMB/ ROM V ore t	–	63.7	64.6	65.6	66.7	67.7
Cement cost	RMB/ ROM V ore t	–	131.6	133.5	135.6	137.5	139.6
Administration fee	RMB/ ROM V ore t	–	12	43	43	43	43
Opex Total	RMB/ ROM V ore t	–	**503.2**	**540.4**	**548.2**	**555.7**	**563.3**
Capex Price	**'000 RMB**	–	**6,500**	**6,500**	**6,500**	**6,500**	**6,500**
V_2O_5	RMB/product t	–	120,000	120,000	120,000	120,000	120,000
Power	RMB/ KWh	–	0.49	0.49	0.49	0.49	0.49
Cement material (slag)	RMB/product t	–	90	90	90	90	90

Source: MCC provided Capex and Opex figures February 09

The mining rate provided by MCC in 2009 appears to reflect mining via open cut method only. The mining unit costs could increase significantly if underground mining was undertaken.

M-MC considers these cost estimates to be highly conceptual at this stage and they will be reviewed after government approvals for mining have been received and more detailed mine planning and feasibility assessment undertaken.

Prices used for V_2O_5 appear reasonable based on long term market averages. There is a market for the generated power in the region and the price of 0.49RMB/kWh is reasonable. With the amount of civil construction occurring in the region, the slag as cement would find a ready market and the adoption of 90RMB/t as the received value is also reasonable.

The construction capital expenditure requirements have been outlined during conversations with MCC staff in 2009 and are summarised in ***Table 10.9***. Based on similar plants in China, M-MC believes the construction costs appear reasonable.

Table 10.9 — Xiangxi V Shale Project — Construction Capital Expenditure Requirements

Item	Units	Value	Comments
Construction Period	years	1	
Mining	RMB	60	may be underestimated for UG
Vanadium processing plan	RMB	99M	
Power station	RMB	50M	
Cement factory	RMB	60M	
Capital Cost (Total)		269M	

Source: Client Information

10.10 SAFETY AND ENVIRONMENT

Although no safety plans or policies were sighted by M-MC, based on other Chinese MCC projects that M-MC has inspected, it is likely that this project would adopt acceptable safety policies and implement them during construction and operation.

The project is located in a relatively pristine part of China and the deposit extends under a major water catchment lake. It is expected that stringent environmental practices would be required in order to progress the development of the project. The only Environmental Plan that was sighted addressed issues such as storage of the waste rock from mining and minimising landslides. Any future environmental policies with regard to mining and the environment will be directed by government approvals and environmental design requirements.

11 NONGGESHAN LEAD ZINC PROJECT

M-MC was not able to visit this project site due to inclement weather and inaccessibility (snow). Also the lack of operational status (development stage) meant a site visit was not critical. The review progressed as a desktop study using data provided. Various reports were reviewed as technical background to this property, the most significant of which are as follows;

- "The Updated Utilisation and Development Scheme of Zn-Pb Mine at Nonggeshan", compiled by the Lanzhou Non-ferrous Metallurgy Design and Research Institute Company Ltd, June 2007. (2007 Feasibility Report)

Project construction is scheduled to re-commence in the summer of 2009.

MCC's equity stake in the Nonggeshan Project is 49.9% (97.83% of a subsidiary owning a 51% stake in the project).

11.1 BACKGROUND

The Nonggeshan Pb-Zn deposit was first discovered in 1958. Currently it is operated by the Sichuan Nonggeshan Multi-metal Mine Company Ltd, which is a joint venture between Yadide Mine Construction Company Ltd (Ganzi Prefecture, Sichuan Province) and Huaye Resource Development Company Ltd (a subsidiary of China Metallurgical Group Corporation).

The mine site is located in the village of Xiede in the Daofu County of the Sichuan Province, at an altitude of between 4,600m and 5,200m (***Figure 11.1***). The area is a high alpine terrain and has an average annual temperature of 14.2°C.

11.2 ASSETS

- In Situ Quantities of 20.4Mt at 1.18%Pb, 1.4% Zn and 16.6g/t Ag.
- Mineable Quantities of 5.7Mt at 2.5%Pb, 1.5% Zn, 17.4g/t Ag.
- "The Updated Utilisation and Development Scheme of Zn-Pb Mine at Nonggeshan", compiled by the Lanzhou Non-ferrous Metallurgy Design and Research Institute Company Ltd, June 2007. (2007 Feasibility Report)

11.3 LAND TENURE AND MINERAL RIGHTS

The 2.96Km^2 Exploration Licence incorporating the three orebodies one in the west ore zone (OB I) and the other two in the east ore zone (OB II and OB III) expired in September 2008. A Mining Lease was granted for orebody N^o I only by the Sichuan Province Land & Resources Bureau in March 2008.

The current mining Licence for orebody N^o I is shown in ***Table 11.1.***

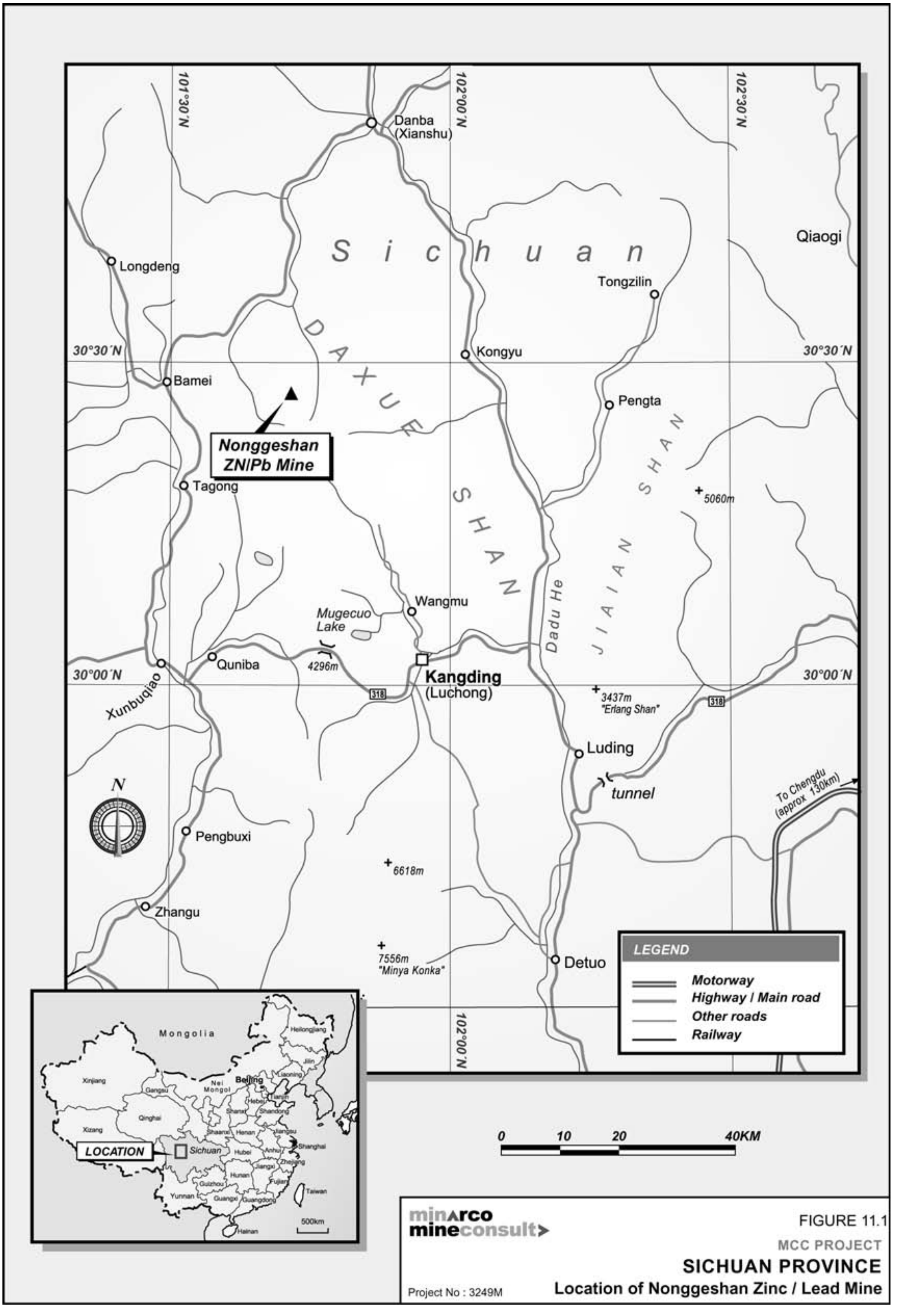
Table 11.1 — Nonggeshan Lead Zinc Project — Mining Licence Details

Mine/Project	Nonggeshan
Title	Mining License
No	5100000810159
Owner	Sichuan Nonggeshan Multi-Metal Mine Co., Ltd
Mine/Project Name	Daofu County Nonggeshan Lead-Zinc Mine, Nonggeshan multi-metal Mine Co., Ltd
Mine Method	UG
Permit Capacity	600ktpa
Permit Area	0.4213km^2
Permit Depth	4795mL - 4485mRL
Valid Date	Mar,2008- Mar 2028
Issue Date	Mar, 2008
Issuer	Sichuan Province Land and Resource Bureau

Note: MCC provided information

M-MC provides this information for reference only and recommends that land titles and ownership rights be reviewed by legal experts.

Figure 11.1 — Nonggeshan Lead Zinc Project — Location Plan



11.4 GEOLOGY

The Nonggeshan Project area is located in the Sanjiang Ore-Forming Belt.

Pb-Zn mineralisation is associated with contact margins between the rock mass and wall rocks and crevices between Triassic strata. The Nonggeshan mineralisation type is a hydrothermal liquid-filled or veinlet impregnated Pb-Zn deposit.

The Nonggeshan deposit has two main ore horizons (East and South) and comprises three (3) orebodies with the following characteristics:

No. I Orebody:	- This is the main mineralised zone (Central and Western Belts), - The host is the Zheduoshan alkali granite body, and - Mineralisation is controlled by the cata-clastic granite structural belt
No. II and III Orebodies:	- Located in the external contact belt to east of Orebody I, - The host is the in Zagunao petrofacies (garnet marble and sericite quartz schist).

Orebody I is the largest within the exploration and mining licences. The hanging wall is granitic mylonite, the host is cataclastic granite or cataclastic breccia, and the footwall is cataclastic granite and gneissic, cataclastic granite. The wall rocks of Orebody I are bounded by fault zones.

The trend of thickness and grade is reducing at depth (down dip).

Ore minerals are mainly galenite (PbS), zinc blende or sphalerite (Zn,Fe,S) and lesser quantities of iron pyrite (FeS_2), chalcopyrite ($CuFeS_2$) and cerussite ($PbCO_3$), smithsonite ($ZnCO_2$), hemimorphite (oxidised sphalerite), magnetite (Fe_3O_4), argentite or native silver (Ag). Minerals in outcrop include secondary ferrite and minor malachite.

There are four types of ore structure:

Disseminated:	sparse impregnated
Massive:	dense impregnated
Vein:	dictyogenous (veiny)
Breccia:	dense block (breccia) form.

Orebody 1 is also defined by a small proportion of oxidised ore ($<$10% proportion).

Figure 11.2 — Nonggeshan Lead Zinc Project — Geology

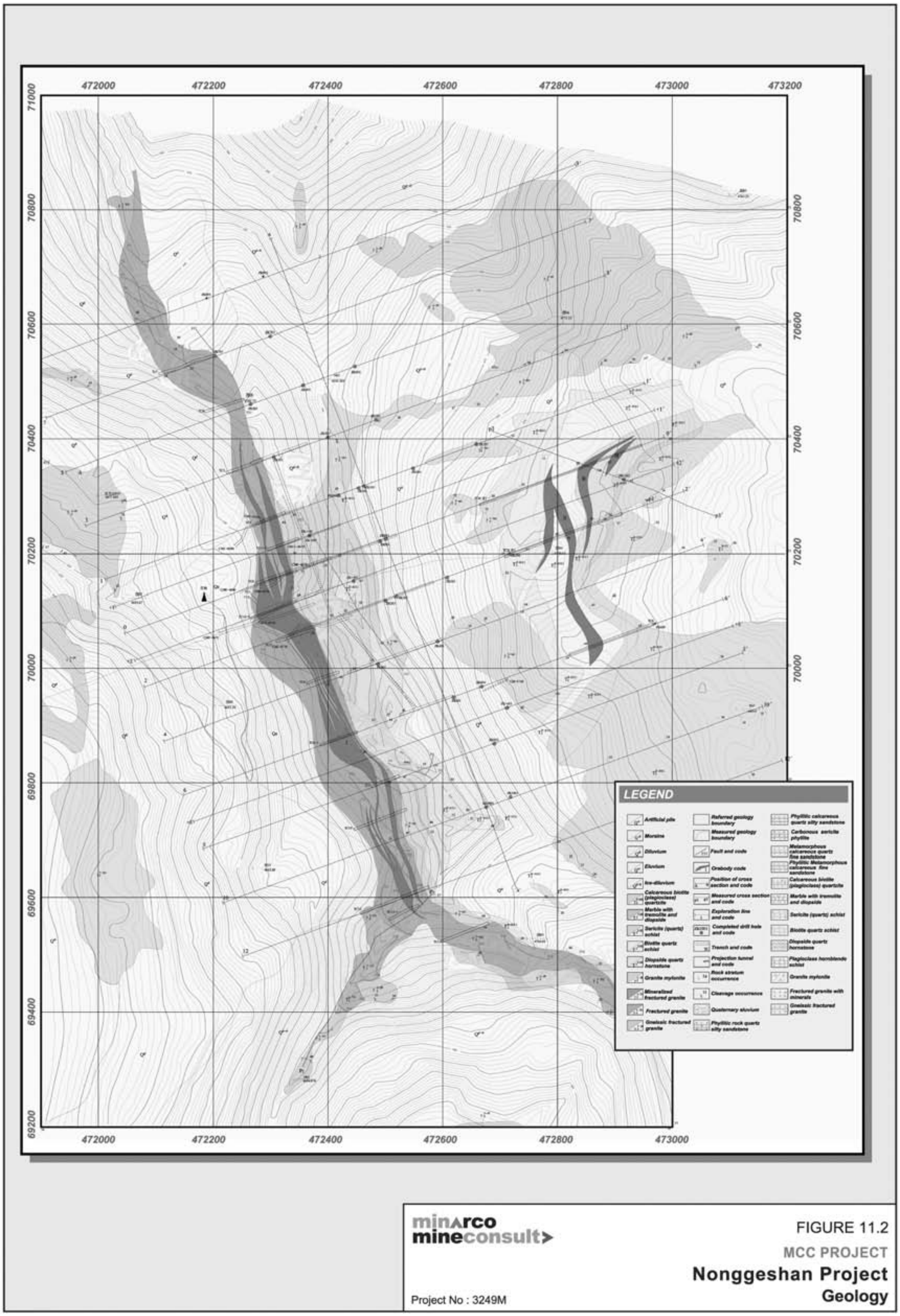
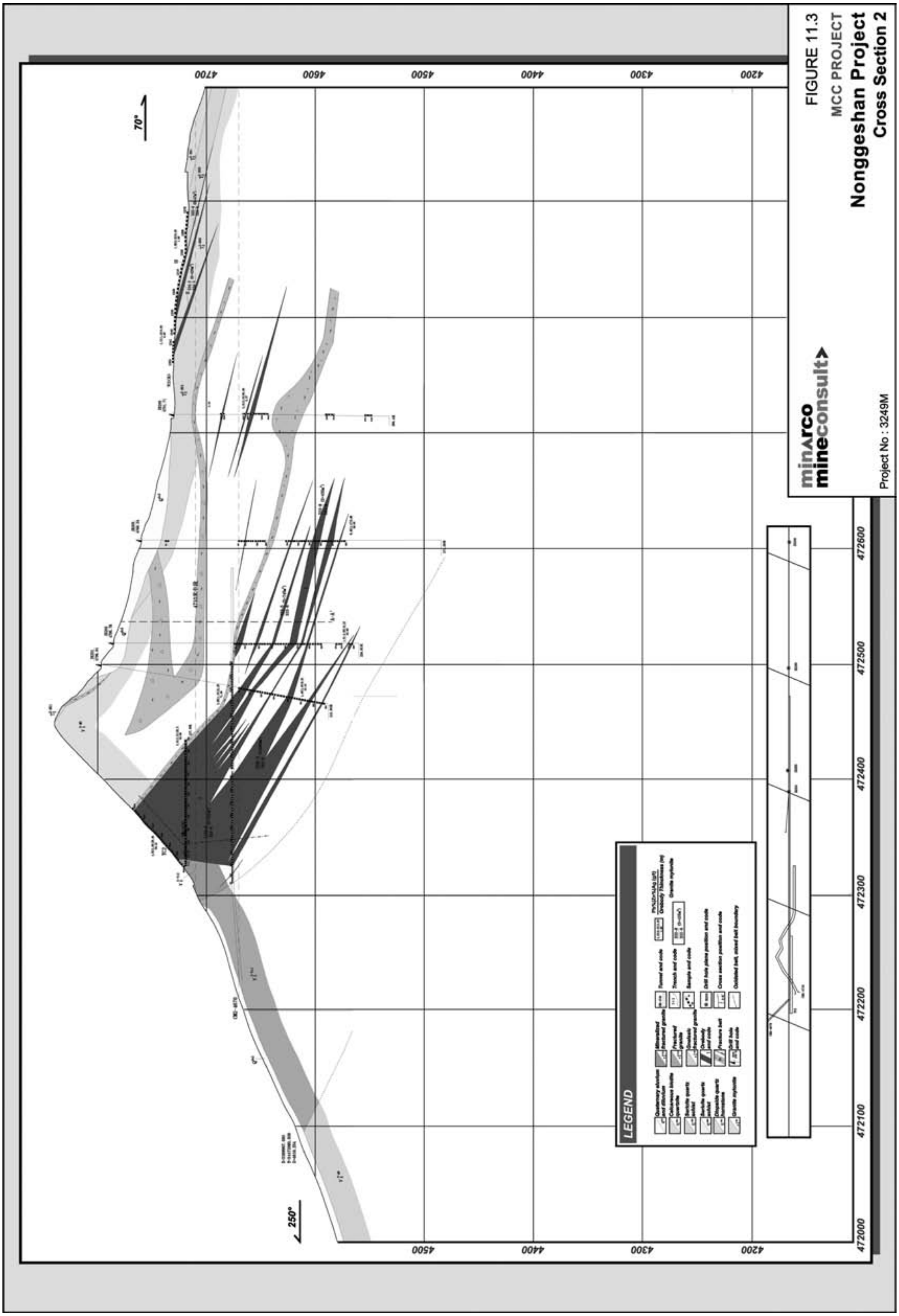


Figure 11.3 — Nonggeshan Lead Zinc Project — Cross Section



11.5 RESOURCES AND RESERVES

11.5.1 Mineral Resources — In Situ Quantities

In Situ Quantities of sulphide ore were estimated and reported by the Lanzhou Institute in 2007 ***(Table 11.2)*** for all 3 orebodies.

Table 11.2 — Nonggeshan Lead Zinc Project — In Situ Quantities — Orebody I, II and III Sulphide Ore

Ore Body	Chinese Code	Sulphide Ore (kt)	Average Grade Pb(%)	Average Grade Zn(%)	Average Grade Ag(g/t)	Contained Metal Pb (kt)	Contained Metal Zn (kt)	Contained Metal Ag (t)
I	111b	3,878	1.76	1.39	17.09	68	39	49.4
	122b	3,252	1.76	1.39	17.09	57	40	43.6
	333	12,454	1.76	1.39	17.09	219	193	241.6
	Sub Total	**19,584**	**1.8**	**1.4**	**17.1**	**345**	**273**	**335**
II	333	559	1.66	1.21	5.55	93	68	3.11
	Sub Total	**559**	**1.7**	**1.2**	**5.5**	**93**	**68**	**3.1**
III	333	272	1.56	2.42	2.42	3	4	0.7
	Sub Total	**272**	**1.6**	**2.4**	**2.4**	**3**	**4**	**0.3**
I+II+III	111b	3,878	1.76	1.39	17.09	68	39	49
	122b	3,252	1.76	1.39	17.09	57	40	43.6
	333	13,285	1.75	1.40	16.30	315	265	245.3
Total		**20,416**	**1.8**	**1.4**	**16.6**	**440**	**345**	**338**

Source: 2007 Feasibility Report

Notes:
Ore Density: 2.89 t/bcm
cog: >3% Pb + Zn

Additional Exploration Targets (334) inside the exploration area are reported by the Lanzhou Institute of approximately 0.68 Mt.

The licence status of orebody II and III cannot be confirmed and the In Situ Quantities shown in ***Table 11.2*** may be reduced by 831kt if the licence is not renewed.

11.5.2 Reserves — Mineable Quantities

Mineable Quantities have been estimated by the Lanzhou Institute in the 2007 Feasibility Report for orebody No I within the proposed mining area. Based on a total Mineable Quantities of 17.9Mt the Lanzhou Institute estimates a life of mine of 30 years based on a 600ktpa mining rate.

In M-MC's opinion the Mineable Quantities estimated are conceptual and not based on detailed mine designs. No dilution or mining recovery factors appear to have been applied to the underlying In Situ Quantities used to estimate the Mineable Quantities. M-MC considers that only 111b and 122b classified In Situ Quantities should be considered of high enough confidence to be considered as Mineable Quantity and has shown this break down in ***Table 10.3***. Review of the grade distribution of the Mineable Quantities shows that grade decreases with depth; this will have a negative impact on mining and processing costs.

Table 11.3 — Nonggeshan Lead Zinc Project — Mineable Quantities — Orebody Sulphide Ore

Ore Body	Chinese Code	Sulphide Ore (kt)	Average Grade Pb(%)	Average Grade Zn(%)	Average Grade Ag(g/t)	Contained Metal Pb (kt)	Contained Metal Zn (kt)	Contained Metal Ag (t)
No I	111b	2,878	2.8	1.51	19.29	80	44	55.5
4710 – 4550 Level or below	122b	2,831	2.14	1.44	15.48	61	41	43.8
Sub Total 111b + 122b		**5,709**	**2.47**	**1.48**	**17.4**	**141**	**84**	**99**
Potential Mineable Quantities	333	12,263	1.64	1.36	18.66	201	167	228.9

Source: 2007 Feasibility Report, Orebody I Mineable Quantities are within the recently granted Mining Licence.

Note: 333 Chinese Code resource are not considered of high enough confidence to include as Mineable Quantities.

11.6 MINING

11.6.1 Mine Planning

The civil construction project is currently on hold due winter. Construction is due to recommence in the summer of 2009. As at the time of writing, construction had yet to recommence.

Boundaries between ore, wall rocks and included waste do not have sharp boundaries. Orebody N^{o} I is planned to be extracted utilising a combination of both underground room and pillar stoping and sub-level open stoping.

From the varying widths of the orebody, it is assumed by M-MC that the wider areas or the orebody will be mined using the room and pillar method of mining and the narrow sections will be mined using the sublevel open stoping method of mining.

11.6.2 Forecast Production

The first 3 years forecast ROM ore production is shown in ***Table 11.4***. The target ROM ore production of 0.6Mt is reached in Year 2.

Table 11.4 — Nonggeshan Lead Zinc Project — ROM Ore Production Plan

UG Production	Units	2012	2013	2014
ROM ore	ktpa	450	600	600
Grade	Ag g/t	21.38	21.38	21.38
	Pb%	2.86	2.86	2.86
	Zn%	1.55	1.55	1.55
Contained Metal	Ag t	9.62	12.83	12.83
	Pb t	12,870	17,160	17,160
	Zn t	6,975	9,300	9,300

Source: MCC provided Capex and Opex figures February 09

M-MC considers this plan to be highly conceptual at this stage, and subject to more detailed mine planning assessment. Based on information provided in the 2007 Development report the current plan is to mine orebody N^{o} I. M-MC notes that the average In Situ grades for Pb, Ag and Zn for ore-body I when reported by elevation are not as high as those quoted in ***Table 11.4***. M-MC considers the grades shown as optimistic considering dilution has not been factored into the production plan.

11.7 MINERAL PROCESSING

In many of the Nonnggeshan lead-zinc ore types, the zinc containing mineral (sphalerite) coats the lead bearing mineral (galena) while in other ore types the two minerals are disseminated and closely associated, making it difficult to produce high grade concentrates at high recoveries. The ore also contains some copper as chalcopyrite which will be recovered to the final lead concentrate, which vary between 0.5-1.3% Cu. Arsenopyrite is also present in the ore and is some areas up to 0.2% As. It is presumed that blending of the high arsenic ores with lower grade arsenic ores (<0.3% As) will be practised in order to minimise the arsenic content of the final concentrates.

The forecast concentrate production is summarised in ***Table 11.5,*** which shows that the lead would be recovered to a concentrate grading 60% Pb with 76% lead recovery and 68% of the silver in the feed. Sixty-six percent of the zinc would be recovered to a concentrate assaying 45% zinc.

Table 11.5 — Nonggeshan Lead Zinc Project — Forecast Production Summary

Product	Unit	2012	2013	2014
ROM Feed	ktpa	450	600	600
Lead Grade	%	2.86	2.86	2.86
Zinc Grade	%	1.55	1.55	1.55
Silver Grade	g/t	21.38	21.38	21.38
Lead Concentrate	ktpa	16.3	21.7	21.7
Lead Grade	%	60	60	60
Lead Recovery	%	76	76	76
Silver Grade	g/t	400	400	400
Silver Recovery	%	67.8	67.8	67.8
Zinc Concentrate	ktpa	10.2	13.6	13.6
Zinc Grade	%	45	45	45
Zinc Recovery	%	66	66	66

Source: MCC provided Capex and Opex figures February 09

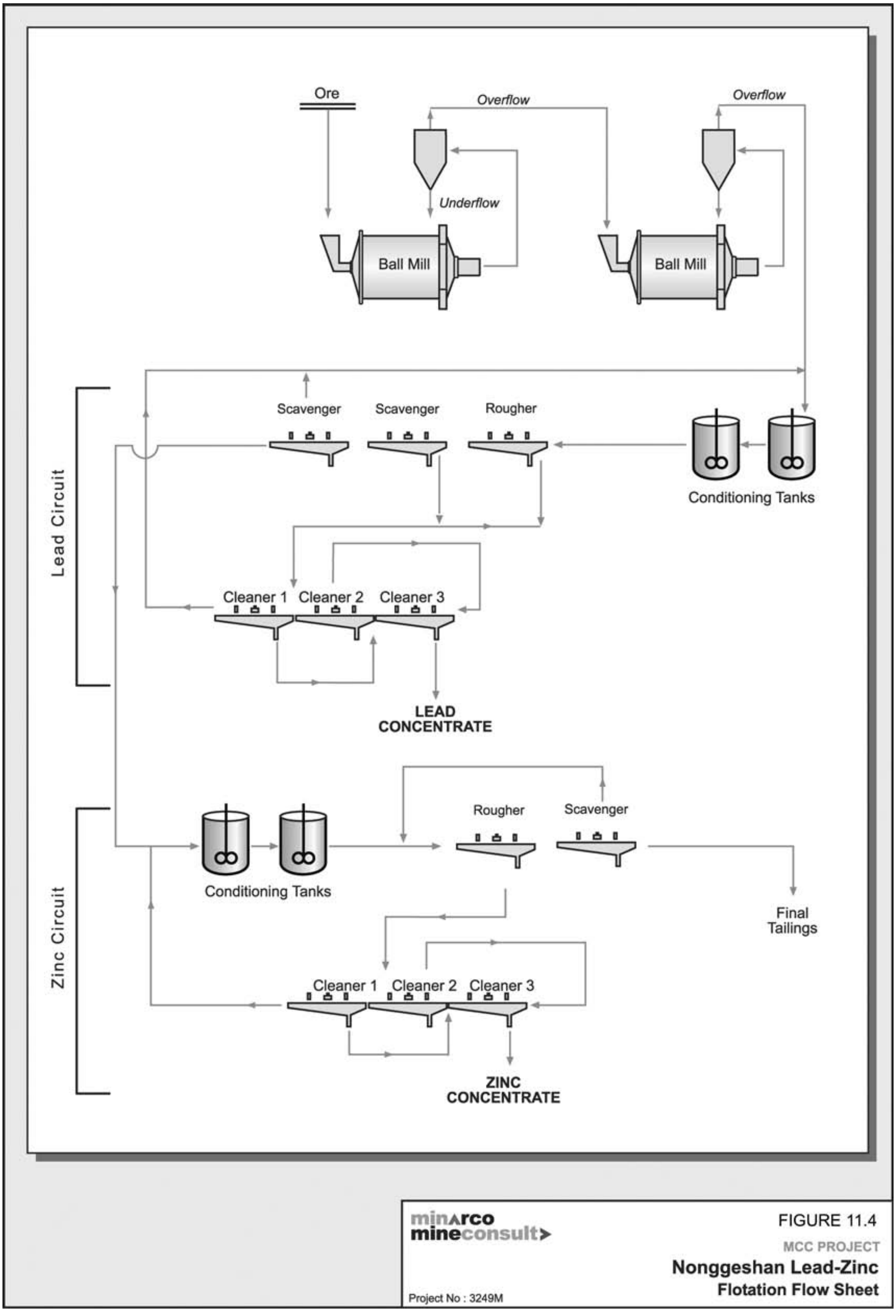
The Nonggeeshan processing flowsheet reflects a typical lead-zinc separation where lead is firstly recovered followed by the zinc. Ore from the mine will be crushed in a three stage crushing and screening circuit to reduce the ore to below 12mm and stored in fine ore storage bins. Ore would be withdrawn from the fine ore storage bins and fed to a two stage milling circuit where a ground slurry (P_{90} = 74 microns) will be produced for the flotation circuit. The flotation processing flowsheet is shown in ***Figure 11.4.***

The cyclone overflow from the milling circuit will be conditioned with reagents in two stages before flotation in the lead roughers followed by two stages of scavenger flotation. Both the rougher and the first stage scavenger concentrate would progress to a three stage cleaning circuit to produce the final lead concentrate. Both the first cleaner tailings and second scavenger concentrate will report back to the conditioning stage for re-treatment in the lead flotation circuit.

The zinc circuit flowsheet is very similar to that of the lead flowsheet. After two stages of conditioning, the tailings from the lead flotation circuit would be fed to the zinc rougher and scavenger flotation banks. The rougher concentrate will be upgraded to produce the final zinc concentrate in three stages of cleaning. The first cleaner tailings will be re-cycled to the conditioning stage while the scavenger concentrate would be re-cycled to the rougher feed. The tailings from the zinc rougher-scavenger flotation bank will be final tailings and would be pumped to the tailings dam.

Both concentrates will be dewatered to 8-10% and bagged for transport to the market.

Figure 11.4 — Nonggeshan Lead Zinc Project — Processing Flowsheet



11.8 INFRASTRUCTURE AND SERVICES

The Nonggeshan project is located in a remote, mountainous region at altitude (4,600m ASL). Moreover the region is prone to earthquakes and all facilities, including the tailings dam, are apparently designed to withstand earthquakes up to 8 on the Richter scale. Additionally, the mine site experiences very heavy snow falls (30m) which needs to be addressed in the plant design as well as operating procedures.

Access

The project is located near the convergence of two highways : Bamei Highway to the south and the Sichuan-Tibet Highway to the west. The major city in the region, Chengdu, can be reached by three highways; the shortest route is 377km by way of Danba, Xiaojin and Dujiangyan. A road to the mouth of Taizhangou valley appears to have been built.

Power

There will be 12.455MW of installed power on the Nonggeshan site of which 7.426MW will be in use. Overall, 40.516GWh of power will be consumed annually. Mining will consume 26.17kWh/t while Processing, including water supply and tailings, will require 35.13kWh/t. Power will be supplied from the grid by a 35kV overhead power line form the Bamei 110kV sub-station some 35km away. There is apparently sufficient power for all the projects' needs. This power line, including a 110kV/35kV transformer at Bamei sub-station, appears to have been constructed, possibly to the mouth of Taizhangou valley. On site, a 35kV/10kV transformer will transform the power for local transmission as well as for operating the milling circuit.

Communications

An optical fibre communications line has been laid to the mouth of Taizhangou valley, where the proposed processing site is to be located.

Water

Some 9,693m^3/d has been estimated as the water requirement for the project, consisting of 3,227m^3/d of raw water, 6,032m^3/d of recycled water from the tailings dam and 432m^3/d of water in re-circulation from dewatering of concentrates and tailings.

Water will be sourced from the nearby Jiemei Lake, some 3.6km away. The water is fairly hard (hardness of 30) with a pH of 7.2 and appears to be suitable for both processing and human consumption. All piping will be located beneath the perma-frost level ($>$2m), insulated and with heating as required.

Heating

Heating provided by two 2t coal boilers will be required in the working areas as well as in the accommodation facilities. Overall, 2,193kW of heating will be generated and reticulated as hot water (110^{o}C), with 1.872kW as heating and 321kW as ventilation.

Tailings Dam

A tailings dam will be located in the nearby Sangjidonggegou Valley, constructed from compacted rock filled wall, initially 32m high (and 4m wide) and with a full height of 82m. The volume of the dam is calculated at 7.5 million m^3. Two pumps will be installed to pump the recovered water, which will be around 76% of the water in the tailings.

Plant/Accommodation Location

A number of locations have been examined and the proposed site has been selected near the mine site (4,360mASL), identified as the Gounao Plant Location in the Sangjidonggegou Valley. Experiences in other countries where mining is conducted at altitudes above 4,000m (e.g. Chile), recommend that accommodation facilities are located at lower altitudes such as 3,000m for reasons of safety and productivity.

11.9 CAPITAL AND OPERATING COSTS

Forecast Mining Costs are shown in ***Table 11.6.*** The second year after production commences represents the first year at the full production rate of 600ktpa.

In M-MC's opinion the total operating costs appear reasonable when compared against other similar operations.

Table 11.6 — Nonggeshan Lead Zinc Project — Forecast Mining Costs

Description	Unit	2011	2012	2013	2014
ROM					
ROM Tonnes	ktpa	—	450	600	600
ROM Lead Grade	%	—	2.86	2.86	2.86
ROM Zinc Grade	%	—	1.55	1.55	1.55
ROM Silver Grade	g/t	—	21.38	21.38	21.38
Price (Exclude Tax)		—			
Lead Concentrate	RMB/t	—	10,500	10,500	10,500
Zinc Concentrate	RMB/t	—	15,000	15,000	15,099
Total Mining Cost	**RMB/ROM t**	—	**60.53**	**60.53**	**60.53**
Total Processing Cost	**RMB/ROM t**	—	**67.94**	**67.94**	**67.94**
Total Operating Cost (Mining, Operating & Other)	**RMB/ROM t**	—	**168.12**	**168.12**	**168.12**

Source: MCC provided Capex and Opex figures February 09

There was no detailed information regarding capital costs available for review, only a summary of the estimated costs as shown in ***Table 11.7***.

Table 11.7 — Nonggeshan Lead Zinc Project — Forecast Capital Costs

Description	Unit	2009	2010	2011	2012	2013	2014
CAPEX							
Mining Maintenance	10k RMB	—	—	—	548.10	730.80	730.80
Mining Expansion	10k RMB	9,282.67	9,282.67	14,282.67	171.90	229.20	229.20
Safety Production	10k RMB	—	—	—	360.01	480.02	480.02
Total	**10k RMB**	**9,282.67**	**9,282.67**	**14,282.67**	**1,080.01**	**1,440.02**	**1,440.02**

Source: MCC provided Capex and Opex figures February 09

M-MC considers these cost estimates to be highly conceptual at this stage and they will be reviewed at a later stage, following completion of a more detailed mine planning and Feasibility assessment.

11.10 SAFETY AND ENVIRONMENT

It would appear that an appropriate safety program will be adopted by the Nonggeshan Project, using proven procedures and reliable safety equipment. The proposed standards and methods appear to be similar to that required and found in Western operations.

The proposed environmental plan appears to be relatively comprehensive and addresses issues such noise, dust and waste gas pollution, water quality as well as soil conservation and land restoration.

Standard dust control methods are employed while the safety protocols and designs adopted for fire prevention and control are satisfactory. 2.5% of the total project capital expenditure (nearly 6 million RMB) will be spent on Safety and Industrial Sanitation.

In terms of prevention and control measures for soil conservation and land restoration:

- Grasses and trees to be planted to stabiles soil
- Engineering measures such as retaining walls to reduce water and soil losses
- Area for restoration is 24.3 Ha
- Water-soil conservation budget 3.3% (7.82 million RMB) of total project capital expenditure

Water treatment consists of sedimentation ponds for mine water and treatment plant for sanitary materials. Waste from the coal fired boilers and the accommodation facilities will be buried at a waste residue site.

12 ANNEXURE A — QUALIFICATIONS AND EXPERIENCE

Technical Experts Involved in Preparation of the ITRR

Andrew Ryan — Runge Asia Limited — General Manger North Asia — Bachelor of Engineering, Mining — University of New South Wales, Master of Applied Finance and Investment (Finsia) — Member of Australasian Institute of Mining and Metallurgy — Associate of Financial Services Institute of Australasia

Andrew has worked with M-MC over the past seven years and has been actively involved in all areas of mining consulting. Most recently, in 2009 has relocated to Hong Kong to establish Runge Asia Limited's new office. Prior to this Andrew was in Beijing as M-MC's Chinese Business Manager responsible for the establishment and growth of M-MC's China business. During this time Andrew was involved with and/or project managed numerous mining related assignments in North Asia. This work has included the project management of due diligence studies, valuation reports, opportunity assessments, conceptual development studies, and feasibility assessments for both domestic and international clients. The projects that these studies have focused on have covered a variety of minerals including coal, iron ore, copper, nickel, gold and molybdenum. Andrew has also had significant experience with due diligences for capital raisings and IPO related projects. Andrew has travelled to and worked on mining projects in Australia, China, Mongolia, Russia, the Ukraine, the Democratic Republic of the Congo, and Papua New Guinea.

Philippe Baudry — Geologist/Geostatistician, Bsc. Mineral Exploration and Mining Geology, Assoc Dip Geo science, Grad Cert Geostatistics, MAIG

Philippe is a geologist with over 10 years of experience. He has worked as a consultant geologist for over 4 years first with Resource Evaluations and subsequently with Runge after they acquired the ResEval group in 2008. During this time Philippe has worked extensively in Russia assisting with the development of 2 large scale copper porphyry projects from exploration to feasibility level, as well as carrying out due diligence studies on metalliferous projects throughout Russia. His work in Australia has included resource estimates for BHPB, St Barbara Mines and many other clients both in Australia and overseas on most styles of mineralisation and metals. Philippe furthered his modelling and geostatistic skills in 2008 by completing a Post Graduate Certificate in Geostatistics at Edith Cowan University.

Prior to working has a consultant Philippe spent 7 years working in the Western Australian Goldfields in various positions from mine geologist in a large scale open cut gold mine through to Senior Underground Geologist. Before this time Philippe worked as a contractor on early stage gold and metal exploration projects in central and northern Australia.

With relevant experience in a wide range of commodity and deposit types, Philippe meets the requirements for Qualified Person for 43-101 reporting, and Competent Person ("CP") for JORC reporting for most metalliferous Mineral Resources. Philippe is a member of the Australian Institute of Geoscientists (Membership N^{o} : 3721)

Andrew Newell — BE, MEngSc, University of Melbourne, PhD, University of Cape Town. Member of the SME, CIMM, AusIMM & IEA as well as a Chartered Professional Engineer, Australasia

Over 30 years of broad experience in the fields of minerals processing, hydrometallurgy, plant design, process engineering (including equipment selection and design) and metallurgical testwork. Andrew has worked on five iron ore projects, one involving flotation, and is knowledgeable about iron ore processing techniques such as magnetic separation. The experience includes operating and management experience in base-metal concentrators, precious metal leaching facilities as well as diamond processing and base-metal smelting in several countries, including Chile, Peru, South Africa, USA and Australia. Responsible for the design of flotation equipment, concentrators and commissioning of flotation and precious metals leach plants. In addition, Andrew has had experience in process and process plant evaluations, due diligence audits, feasibility studies and metallurgical testwork and program development.

Aaron Green — Senior Consulting Geologist, Bsc. Geology (Hons), Grad Dip Applied Finance and Investment, MAIG.

Aaron is a geologist with over 15 years of experience in the mining industry. With a strong background in exploration and mine geology, he has been responsible for the planning, implementation and supervision of various drill programs, underground production duties, detailed structural and geological mapping and logging, geological modelling, and resource estimation. Aaron's wide range of experience with various mining operations in Australia and overseas gives him an excellent practical and theoretical basis for resource estimation of various metalliferous deposits.

In his recent consulting work, Aaron has run or been involved with resource estimation, geological modelling, due diligence investigations, studies ranging from scoping level to Bankable Feasibility, resource drill-out planning and management, and exploration programs.

Commodity experience includes gold, copper, nickel sulphide, fluorite, lead-zinc, iron and industrial minerals. Country experience includes Australia, Zambia, Malawi, Finland, Kazakhstan and China.

With relevant experience in a wide range of commodity and deposit types, Aaron meets the requirements for Qualified Person for 43-101 reporting, and Competent Person ("CP") for JORC reporting for most metalliferous Mineral Resources. Aaron is a member of the Australian Institute of Geoscientists.

Brendan Parker — Mining Engineer , B.E (Mining)

Brendan has over 5 years of experience working in the mining industry. During this time he has been responsible for the planning, design and day to day operation of several underground mines. Brendan's wide range of experience within various mining operations in Australia and Canada gives him an excellent practical basis for the design and planning of underground metalliferous mines.

Brendan has gained extensive experience in the planning and design of underground mines. This includes design and operating experience with with both narrow vein and bulk long-hole stoping methods, as well as flatback — cut and fill mining. His knowledge obtained as a ventilation officer, production engineer, planning engineer and his exposure to managerial roles have provided him with a sound understanding of all aspects of underground mining.

Brendan has also gained experience using software packages specific to underground mine design, production stope ring design and mine scheduling. These include Surpac, RingKing, Vulcan, Datamine and Ventsim.

Ron Siwinski — BSc. M-MC — Minarco -MineConsult Senior Mining Engineer, Engineering (Massachusetts), Water Resources (MIT)

After a 6 year career as a civil engineer, he worked as Senior Mining Engineer, Operations at Cliffs Robe River Iron Ore in Western Australia in the 1970's. From 1980 to 2000 he was Mining Engineer, Project Management Engineer and General Manager, Engineering for Southern Pacific Petroleum NL and was heavily involved in their Queensland shale oil projects. Ron has considerable experience in mine planning, feasibility studies, economic analysis and mine development. His main area of expertise is project management. Ron has been a mining consultant since 2000.

Bill Knox — Minarco — MineConsult China Executive Consultant — Bachelor of Science Geology — Curtin University Western Australia, — Member of Australasian Institute of Mining and Metallurgy

Bill joined Minarco — MineConsult, now part of the Runge Group, in 1993 as an associate and is based in the Beijing Office. His experience includes:

- 10 years in Iron Ore operations
- 10 years in Coal operations and planning
- >10 years in Consulting, coal, metals and quarries.

Bill's areas of expertise include Mineral Resource and Ore Reserve assessment, geological modelling and mine design using Gemcom software, feasibility studies, operational management and grade/quality control. He has held site positions from Mine Geologist to Mine Superintendent and Head Office positions in business planning and development. Recent operations work has included alliance management and project management of two large opencut coal mines in NZ on behalf of Solid Energy and the mining contractor, which included developing a planning process and dispute resolution. Relevant technical experience in Coal projects has included Australia, New Zealand, Indonesia, Columbia, Bangladesh and Zimbabwe. Other work has concentrated on coal resource/reserve audits of procedures and reporting standards of major coal operations in Australia and Indonesia for presentations to corporate management and resource funding institutions. Metals experience has included 10 years in Iron Ore mining in the Pilbara, Western Australia and Mineral Resource modelling and reporting on poly-metallic and uranium projects in Australia and Mongolia. The range of commodities experience in operations and feasibilities studies includes iron ore, coal, poly-metallics, uranium, molybdenum, gold, oil shale, diamonds and quarries. Technical reviews include projects in China and Mongolia.

With reference to the Australasian Code for Reporting of Identified Mineral Resources and Reserves (JORC), he is considered a "Competent Person" to validate statements for Mineral Resources and Ore Reserves.

Peter Goodman — Process Engineer — Minarco — Mineconsult Associate — Bachelor of Applied Science — Graduate Diploma in Mineral Processing — Quarry Managers Certificate of Competency — Metallurgy Certificate — Member Australian Coal Preparation Society

Peter has managed, designed and constructed mineral processing plants in both Australia and South East Asia with over 30 years experience associated with the mining industry. During this period he has undertaken all levels of technical studies and audits of current and prospective operations in Queensland, NSW, China, New Zealand, India, South Africa and Indonesia. Peter has also been involved with numerous mineral processing plants that have been built or are currently under construction in China and elsewhere.

Rod Dale — Fellow of the Royal Melbourne Institute of Technology, 1960. Fellow of the Australasian Institute of Mining and Metallurgy

Experience includes exploration for iron ore, gold, base metals, uranium and industrial minerals in most states of Australia, in Indonesia, India, China, Brazil, Chile, Peru, Zimbabwe and Botswana. Rod has worked for several major and minor companies and has been an executive director of three ASX listed public companies. He has run his own "one man" geological consultancy since 1972 and has worked in and managed two small gold mines, both opencut and underground, as part owner. Rod has also reviewed numerous independent technical reviews of both Au projects operating Au mines in China.

Igor Bojanic BE (Mining, Hons), Minarco — MineConsult Senior Mining Engineer, M.Appl.Sc. (Env Mgmt), MAusIMM, CPMin, MMICA.

Igor is a mining engineer with extensive practical experience in all facets of opencut mining. His strengths lie in project mine planning and scheduling in opencut metalliferous, coal and quarries. Metalliferous projects undertaken include pit optimisations using both Whittle 4D and 4X, pit design, scheduling, equipment selection and mine costing. Igor has also worked on a number of quarry projects, developing quarry plans for both operations and to support environmental documents. He has obtained a Masters in Environmental Management and has a particular interest in incorporating environmental planning into the mine planning process and also has a very good working knowledge of Gemcom, MicroLynx, Datamine, Surpac and Whittle software. Recently Igor has had significant exposure to the development and running of detailed economic models as part of Due Diligence and Detailed Feasibility Studies.

Matthew Hoare — Minarco — MineConsult Mining Analyst, Bachelor of Information and Technology, MBA

Matthew's experience has mainly been in the form of opencut coal mines where he worked in a long term planning position at Kideco, a large Indonesian coal mine for over three years. He was also seconded to the Dawson Project in central Queensland where he assisted in short and medium term planning. Matthew has recently completed a Master of Business Administration majoring in finance. His undergraduate studies included Information Technology and Indonesian Language at the University of Southern Queensland. He is proficient in the use of Minescape.

Abani R Samal — Geologist/Geostatistician — Minarco — MineConsult — PhD in Environmental Resources and Policy (Focus: Energy and Mineral Resources), Certified Professional Geologist (CPG) — Registered Member Society of Mining, Metallurgy and Exploration, Member of the Australasian Institute of Mining and Metallurgy (AusIMM).

Abani has worked as a geologist and a consultant in India and USA. He has seven years of industrial experience and five years of research experience related to mineral exploration and mining industry. He has done technical studies: modelling, auditing various commodities (Au, Ag, Cu, Pb, Zn etc.) for projects in USA, Canada, South America and India.

Oscar Tesari — M.Sc., Economic Geology, University of British Columbia, Diploma in Geology, University of São Paulo.

Over 35 years experience in iron ore mines of the Minas Gerais State Iron Ore Quadrangle through the development of geological studies, drilling programs, sampling, databases, assay quality assurance and quality control, exploration property evaluations, research, design and management of exploration programs, geological modelling and resource estimation, in addition to mine designs and production planning. Through technical visits and development of geological works, he has become familiar with iron ore mines in Canada, Russia, Australia and South Africa, in addition to gold mines in Brazil and South Africa and silver-lead-zinc deposits in Canada.

Relevant iron ore Experience: Mineracoes Brasilerias Reunidas (MBR) — Manager of Development, Long Term Development Plan, Design and development of exploration programs covering the all of the iron ore mining claims of the company (Pico Complex, Vargem Grande Complex and Paraopeba Complex).

- Resource estimation.
- Orientation of technological studies aimed at finding the most suitable itabirite cut-off grade.
- Reserve estimation and mine sequencing.

Company's Relevant Experience

Minarco — MineConsult, part of the Runge Group, is a premier international consulting and engineering firm. We provide a full range of services from pure technical consulting through to strategic corporate advice. We undertake assignments on mining projects covering a range of commodities and countries, and we serve clients in most of the countries around the West Pacific Rim region.

Minarco — MineConsult maintains a full-time staff of qualified specialists in the fields of mining engineering, geology, process and metallurgical engineering, environmental and geotechnical engineering, and environmental economics.

Minarco — MineConsult typically completes over 200 assignments per year and has over 300 professionals (through Runge Group) available in disciplines including:

- Mining Engineering;
- Minerals Processing;
- Coal Handling and Preparation;
- Power Generation;
- Environmental Management;
- Geology;
- Contracts Management;
- Project Management;
- Finance;
- Commercial Negotiations.

The roots of Minarco — MineConsult were established in the Australian mining industry. Minarco — MineConsult is committed to compliance with the codes which regulate Australian corporations and consultants and has established an International business which has continued to give its clients and those that rely on its work the confidence that the relevant Australian codes invoke.

These codes include:

- The Australian Corporation Law;
- The Australian Institute of Company Directors Code of Conduct;
- The Securities Institute of Australia Code of Ethics;
- The Australasian Institute of Mining and Metallurgy Code of Ethics;
- The Australasian Code for Reporting of Exploration Results, Mined Resources and Ore Reserves (The JORC Code).

Minarco — MineConsult has conducted numerous mining technical due diligence programmes and reporting for IPO's and capital raisings over the past 6 years, with involvement in projects raising a total of over $US 10 billion of capital. This and other work is summarised in ***Table A1***.

Table A1 — Mining Related IPO and Capital Raising Due Diligence Experience

2008 China Blue Chemical Limited, Wangji and Dayukou Phosphate Mines: Independent Technical Review for inclusion in a Stock Exchange Circular to support a mining asset purchase by a listed Hong Kong Company.

2008 Kenfair International (Holdings) Limited, Shengping Coal Mine: Independent Technical Review for inclusion in a Stock Exchange Circular to support a mining asset purchase by a listed Hong Kong Company.

2007 China Railway Company Limited, African Copper/Cobalt Assets: Capital raising for mining assets on the Hong Kong Stock Exchange. Preparation of CPR for planned IPO on the HKSE.

2007 Ko Yo Ecological Agrotech (Group) Limited Sichuan Phosphate: Independent Technical Review for inclusion in a Stock Exchange Circular to support a mining asset purchase by a listed Hong Kong Company.

2007 Prosperity International Holdings Limited, Guilin Granite Project: Independent Technical Review for inclusion in a Stock Exchange Circular to support a mining asset purchase by a listed Hong Kong Company.

2007 China Primary Resources — Independent Technical Review for inclusion in a Stock Exchange Circular to support a mining asset purchase by China Primary Resources.

2008 Kenfair International (Holdings) Limited, Shengping Coal Mine: Independent Technical Review for inclusion in a Stock Exchange Circular to support a mining asset purchase by a listed Hong Kong Company.

2007 China Railway Company Limited, African Copper/Cobalt Assets: Capital raising for mining assets on the Hong Kong Stock Exchange. Preparation of CPR for planned IPO on the HKSE.

2007 Gloucester Coal Limited — Independent Technical Review for Australian Stock Exchange Scheme of Arrangement.

2007 Confidential Hong Kong Private Equity Partners — Independent Technical Review to support private equity capital raising to purchase lead/zinc mining assets in Tibet.

2007 Confidential International Investor — Independent Technical Review to support private equity capital raising to purchase iron ore assets in Hubei. Preparation of ITR.

2007 Whitehaven Coal Limited — Independent Technical Review for Australian Stock Exchange IPO.

2007 Confidential Privately Owned Coke Producer — Capital raising for purchase of Coal Mines and downstream coal washing, coke production and chemical production facilities. Preparation of CPR for planned IPO on the HKSE.

2007 China Molybdenum Group — Capital raising for large scale Molybdenum mine on the Hong Kong Stock Exchange. Preparation of CPR for IPO on the HKSE.

2007 Confidential International Investor — Independent Technical Review to support purchase of Gold Mine In Hubei Province.

2006 Excel Mining — Independent Technical Review for Australian Stock Exchange Scheme of Arrangement.

2006 Celadon Mining Investment Group (UK) — Capital raising for coal mine purchase in China and planned subsequent listing on AIM

2005 Yanzhou Coal Mining Company Limited — Independent Technical Review of coal projects to satisfy ongoing listing requirements of the HKSE and NYSE following IPO.

2004 Excel Mining — Independent Technical Review for Australian Stock Exchange IPO (current market capitalisation over $US1billion)

2004 Excel Mining — Independent Market Review for Australian Stock Exchange IPO

2003 New Hope — Independent Market Review for Australian Stock Exchange IPO

2003 Confidential — Independent Market Review on 50 Mtpa operation in Kazakhstan for LSE listing (has not proceeded)

2003 Xstrata plc — Competent Person's Report for London Stock Exchange Chapter 19 Report for Acquisition of MIM Assets including mines, rail and port review ($US 2.5 billion)

2002 Xstrata plc — Competent Person's Report for London Stock Exchange IPO ($US2.3 billion)

2002 Kaltim Prima, Indonesia — Independent Technical Review for advising project financiers to acquisition ($US445 million)

2001 Enex Resources — Independent Technical Review for Australian Stock Exchange IPO

2001 Macarthur Coal Limited — Independent Technical Report and Market Review for Australian Stock Exchange IPO

13 ANNEXURE B — GLOSSARY OF TERMS

The key terms used in this report include:

- **assets** — means the Title to Mineral Resources and associated mining and processing plant and equipment.
- **AUD** — Australian dollar
- **Bt** — stands for billion tonnes
- **cog/cut off grade** — the lowest grade, or quality, of mineralised material that qualifies as economically mineable and available in a given deposit. May be defined on the basis of economic evaluation, or on physical or chemical attributes that define an acceptable product specification
- **Company** — means Metallurgical Corporation of China Ltd
- **concentrate** — a powdery product containing higher concentrations of minerals resulting from initial processing of mined ore to remove some waste materials; a concentrate is a semi-finished product, which would still be subject to further processing, such as smelting, to effect recovery of metal
- **contained metal** — refers to the amount of pure metal equivalent estimated to be contained in the material based on the metal grade of the material.
- **element** — Chemical symbols used in this report
  Au — Gold
  Ag — Silver
  C Fe — Iron grade of concentrate (based on Davis Tube Recovery (DTR) analysis)
  Cu — Copper
  Co — Cobalt
  Fe — Iron (TFe Total Iron, MFe magnetite Iron)Magnetic Iron)
  Fe++ — Iron (TFe which occurs as ferrous iron (FeO)
  Ni — Nickel
  P — Phosphorous
  Pb — Lead
  V_2O_5 — Vanadium Pentoxide
- **ENFI** — stands for China Central Engineering Institute for Non-ferrous Metallurgical Industries
- **exploration** — activity to identify the location, volume and quality of a mineral occurrence
- **Exploration Target/Results** — includes data and information generated by exploration programmes that may be of use to investors. The reporting of such information is common in the **early** stages of exploration and is usually based on limited surface chip sampling, geochemical and geophysical surveys. Discussion of target size and type must be expressed so that it cannot be misrepresented as an estimate of Mineral Resources or Ore Reserves.
- **exploration right** — the licensed right to identify the location, volume and quality of a mineral occurrence
- **flotation** — is a separation method for to the recovery of minerals using reagents to create a froth that collects target minerals
- **gangue** — is a mining term for waste rock
- **grade** — any physical or chemical measurement of the concentration of the material of interest in samples or product. The units of measurement should be stated when figures are reported
- **grind** — means to crush, pulverize, or reduce to powder by friction, especially by rubbing between two hard surfaces
- **HKEX** — stands for Hong Kong Stock Exchange
- **In situ** — means rock or mineralisation in place in the ground

- **In Situ Quantities** Estimates of total in ground tonnes and grade which meet the requirements of the PRC Code or other international codes for reserves but do not meet either NI 43-101 or Joint Ore Reserves Committee's recommendations
- **Indicated Mineral Resource** is that part of a JORC compliant Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a **reasonable** level of confidence. It 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. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.
- **Inferred Mineral Resource** is that part of Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a **low** level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.
- **ITR** stands for Independent Technical Review
- **ITRR** stands for Independent Technical Review Report
- **JORC** stands for Joint Ore Reserves Committee
- **JORC Code** The Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves published in 2004 by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, the by the Australian Institute of Geoscientists and the Minerals Council of Australia. It set out minimum standards, recommendations and guidelines for Public Reporting in Australasia of Exploration Results, Mineral Resources and Ore Reserves.
- **kcal** stands for kilocalorie, or 1,000 calories
- **km** stands for kilometer
- **kt** stands for thousand tonnes
- **lb** stands for pound, a unit of weight equal to 453.592 grams
- **m** stands for metres
- **MCC** refers to Metallurgical Corporation of China Ltd.
- **Measured Mineral Resource** is that part of a JORC compliant Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a **high** level of confidence. It 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. The locations are spaced closely enough to confirm geological and grade continuity.
- **metallurgy** Physical and/or chemical separation of constituents of interest from a larger mass of material. Methods employed to prepare a final marketable product from material as mined. Examples include screening, flotation, magnetic separation, leaching, washing, roasting etc.
- **mine production** is the total raw production from any particular mine
- **Mineable Quantities** Estimates of in ground tonnes and grades which are recoverable by mining
- **mineral right** for purposes of this document, mineral right includes exploration right, mining right, and leasehold exploration or mining right
- **mineralisation** any single mineral or combination of minerals occurring in a mass, or deposit, of economic interest. The term is intended to cover all forms in which mineralisation might occur, whether by class of deposit, mode of occurrence, genesis or composition

- **mining rights** means the rights to mine mineral resources and obtain mineral products in areas where mining activities are licensed
- **M-MC** refers to Minarco-MineConsult
- **mRL** means meters above sea level
- **Mt** stands for million tonnes
- **Mtpa** means million tonnes per annum
- **OC** Means open cut mining which is mining from a pit open to surface and usually carried out by stripping of overburden materials
- **ore** is the portion of a reserve from which a metal or valuable mineral can be extracted profitably under current or immediately foreseeable economic conditions
- **ore processing** is the process through which physical or chemical properties, such as density, surface reactivity, magnetism and colour, are utilized to separate and capture the useful components of ore, which are then concentrated or purified by means of flotation, magnetic selection, electric selection, physical selection, chemical selection, reselection, and combined methods
- **ore selection** the process used during mining to separate valuable ore from waste material or barren rock residue
- **ore t** stands for ore tonne
- **ounce** refers to troy ounce which is equal to 31.1034768 grams
- **primary mineral deposits** are mineral deposits formed directly from magmas or hydrothermal processes
- **Probable Ore Reserve** is the economically mineable part of a JORC compliant Indicated, and in some circumstances, a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors These assessments demonstrate at the time of reporting that extraction could reasonably be justified. A Probable Ore Reserve has a lower level of confidence than a Proved Ore Reserve but has adequate reliability as the basis of mining studies.
- **project** means a deposit which is in the pre-operating phase of development and, subject to capital investment, feasibility investigations, statutory and management approvals and business considerations, may be commissioned as a mine
- **Proved Ore Reserve** is the economically mineable part of a JORC compliant Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified. A Proved Ore Reserve represents the highest confidence category of Ore Reserve estimates. This requires detailed exploration and high quality data to provide a high level of geological confidence.
- **raw ore** is ore that has been mined and crushed in an in-pit crusher, but has not been processed further
- **recovery** The percentage of material of initial interest that is extracted during mining and/or processing. A measure of mining or processing efficiency
- **regolith** is a geological term for a cover of soil and rock fragments overlying bedrock
- **reserves** the [economically] mineable part of a Measured and/or Indicated Mineral Resource, including diluting materials and allowances for losses which may occur when the material is mined

- **Reserves** reserves which have been estimated in accordance with the recommendations of the JORC Code
- **resources** a concentration or occurrence of a material of intrinsic economic interest in or on the earth’s crust in such form, quality and quantity such that there are reasonable prospects for eventual economic extraction
- **Resources** Resources which have been estimated in accordance with the recommendations of the JORC Code
- **RL** means Reduced Level, an elevation above sea level
- **RMB** stands for Chinese Renminbi Currency Unit; 103 RMB means 1,000 RMB
- **RMB/t** stands for Chinese Renminbi per material tonne
- **ROM** stands for run-of-mine, being material as mined before beneficiation
- **saprolite** is a geological term for weathered bedrock
- **secondary mineral deposits** are mineral deposits formed or modified as a result of weathering or erosion of primary mineral deposits
- **shaft** a vertical excavation from the surface to provide access to the underground mine workings
- **t** stands for tonne
- **t/bcm** stands for tonnes per bank cubic metre (i.e. tonnes in situ) a unit of density
- **tonnage** An expression of the amount of material of interest irrespective of the units of measurement (which should be stated when figures are reported)
- **tonne** refers to metric tonne
- **tpa** stands for tonnes per annum
- **tpd** stands for tonnes per day
- **UG** means underground mining which is an opening in the earth accessed via shafts, declines or adits below the land surface to extract minerals
- **upgrade ratio** is a processing factor meaning ROM Grade% / Product Grade %
- **USD** stands for United States dollars
- **VALMIN Code** refers to the code and guidelines for technical assessment and or valuation of mineral and petroleum assets and mineral and petroleum securities for independent expert reports
- **$** refers to United States dollar currency Unit
- **AUD $** refers to Australian dollar currency Unit
- **¥** is the symbol for the Chinese Renminbi Currency Unit

Note: *Where the terms Competent Person, Inferred Resources and Measured and Indicated Resources are used in this report, they have the same meaning as in the JORC Code.*

ANNEXURE C — RESOURCE REPORTING STANDARDS

Chinese Resource Reporting Standards

In 1999, with a view to creating a standard that was comparable with international resource reporting standards, The Chinese National Land and Resource Department introduced its own national standard for the Classification of Resources/Reserves for Solid Fuels and Mineral Commodities (GB/T 17766-1999).

This code was to replace the previous code (China GB 13908-1992 — General rules for Geological Exploration of Solid Ore Resources) and was based upon the United Nations international code (UN Economic and Society Committee, UN document ENERGY/WP.1/R.70). Some elements of the American resource reporting standards were included and modifications made to suit Chinese conditions. All new resource estimates are reported under this new code and old estimates either re-estimated or converted to the new system.

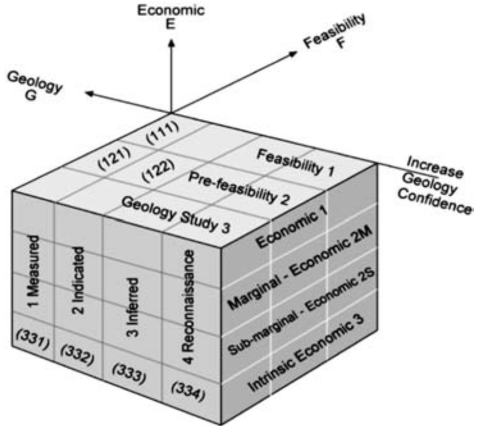
The previous Chinese standard (GB 13908-1992) divided resources into four categories (A, B, C and D) which were loosely comparable to the JORC — (December 2004) classifications of Measured Resource (A-B), Indicated Resource (B-C) and Inferred Resource (D). The old standard was more prescriptive than JORC in that it specified minimum borehole spacings (see ***Table C1***) for each category, along with implied levels of geological understanding.

Table C1 — Borehole Spacing Comparison (Chinese, UN and JORC Codes)

(Chinese Reserve Code)	Classification (Chinese Reserve Class)	UN Code	JORC(Dec 2004)	Minimum Borehole / Drill Line Spacing
A	111 - 121		Measured	<100 m
B	121 - 122	331	Measured	<=100 m x 100 m
C	122 - 2 m22	332	Indicated	<=200 m x 100 m
D	122	333	Inferred	>200 m

The old code was essentially a geological classification, taking little account of the deposits economics or the level of mining studies that had been carried out on it. The new code (see ***Figure C1***) attempts to address this by using a three component system (EFG) that considers the deposit economics (E), the level of mining feasibility studies that have been carried out (F) and the level of geological confidence (G) using a numerical ranking.

Figure C1 — New Chinese Resource/Reserve Classification Matrix (1999)



This system produces a three digit code for a deposit that reflects these three variables. For example a deposit classified as a 121 is economically viable (1), has had pre-feasibility studies carried out (2) and is well understood geologically (1). Various suffixes are used to distinguish Basic Reserves — essentially JORC Resources — (121b) from Extractable Reserves (121) and to identify the assumed economic viability (S or M). Certain categories are not allowed, for example pre-feasibility or feasibility level studies cannot be conducted on Inferred Resources, and so 123 and 113 are invalid classifications. Also Extractable Reserves are not estimated for marginally economic (or lesser) deposits so the (b) suffix is considered redundant. The term Intrinsically Economic indicates that while the deposit may be economic, insufficient studies have been carried out to clearly determine its status.

A tabulation of this concept is shown in ***Table C2***.

Table C2 — New Chinese Resource/Reserve Categories (1999)

	Geological Confidence			
	Identified Mineral Resource			
Economic Viability	Measured (1)	Indicated (2)	Inferred (3)	Undiscovered Resource Reconnaissance (4)
Economic (1)	Basic Reserve *[Resource]* - 111b			
	Proved Extractable Reserve - 111			
	Basic Reserve *[Resource]* 121b	Basic Reserve *[Resource]* - 122b		
	Probable Extractable Reserve - 121	Probable Extractable Reserve - 122		
Marginally Economic (2 m)	Resource 2 m11			
	Resource 2 m21	Resource 2 m22		
Sub-marginally Economic (2S)	Resource 2S11			
	Resource 2S21	Resource 2S22		
Intrinsically Economic (3)	Resource 331	Resource 332	Resource 333	Resource 334

Note: *First digit reflects Economic viability; 1= Economic; 2 m=Marginally Economic; 2S= Sub-marginally Economic; 3=Intrinsically Economic; 4=Economic interest undefined.*
Second digit reflects Feasibility assessment stage, 1=Feasibility; 2=Pre-feasibility; 3=Geological study.
Third digit reflects Geological assurance, 1=Measured, 2=Indicated, 3=Inferred, 4=Reconnaissance.
b=Basic Reserve (prior to recovery factors, mining losses and dilution) — [JORC Resource].

Unlike the old code, the new code does not specify required borehole spacings for each category. In the case of copper Cobalt and Gold (and other metals), there is an accompanying Chinese Professional Standard (DZ/T 0214-2002) that lays out rules for determining the level of geological confidence.

Russian Resource Reserve Reporting Standard

The Russian reporting code evolved from the requirements of a centrally — planned economy with every stage of a mining project documented according to statutory regulations administered through a set of Federal Agencies. The State Commission on Reserve (GKZ) is the main Federal body in charge of approving mining projects of significant size or importance. The Territorial Commission on Reserves (TKZ) are regionally based and deal with smaller mining projects.

Resources are Classified according to two controlling variable:

- Deposit Complexity on a scale of 1-4 with 1 being the simplest most continuous deposits (coal and laterite) and 4 being the most complex (hydrothermal gold, diamond)
- Level of knowledge on 7 levels:

• P3	regional reconnaissance	resources
• P2	target identification	
• P1	initial trenching and drilling	
• C2	scoping / pre-feasibility	reserves
• C1	feasibility study	
• B	blocking out ore panels	
• A	production	

Standards have been created by the GKZ stipulating all requirements to achieve the level of knowledge based on the specific deposit complexity. These standards outline approved exploration methods, drill spacing, sampling and assaying methodology, quality control checks as well as mining study requirements to achieve a specific level of knowledge. A basic level of mining economic analysis has to be carried out in order to report resources with a C2 classification. C1 category material is often called reserve as by this stage a reasonable level of economic analysis has been undertaken and the confidence in the estimate has increased. Some level of reserve categories under the Russian Code cannot be achieved if the complexity of the deposit is deemed too high. These can however be reported with a high classification under the recommendation of the JORC Code as long as appropriate studies have been undertaken (See table C3).

Mining companies can evaluate resources up to C2 category as part of their exploration activities without submission to the GKZ. From C1 category onwards the company must submit its feasibility studies to the GKZ for audit and approval. Once approved the reserves are placed "On the Balance" or "Out of Balance" based on a chosen cut off grade. This cut off grade is not always based on commercial cut off's, but is often a compromise between the company and the GKZ. Out of Balance reserves attract a much lower tax rate than On Balance reserves.

Table C3 outlines an approximate conversion guideline of the Russian Code to the JORC Code based on the controlling variables discussed above.

Table C3 — Russian Code to JORC Code conversion Guidelines

		Russian Resource/Reserve Code Categories						
		A	**B**	**C1**	**C2**	**P1**	**P2**	**P3**
	I	Measured (Proven)	Measured (Proven)	Indicated (Probable)	Inferred			
Deposit	II		Measured (Proven)	Indicated (Probable)	Inferred		Exploration	
Complexity	III			Indicated (Probable)	Inferred		results unless drilling	
	IV				Inferred		is undertaken	

Note: *greyed out areas indicate reserve categories which due to the deposit complexity cannot be reached under the Russian Code. These under JORC may still be reported as either resources or reserves base on the level of study.*

International Standards and the JORC Code for Resources

Two main styles of resource reporting codes exist internationally. These are the American style (USA and much of South America) and the JORC style (Australia, South Africa, Canada, and UK). This is further complicated by the listing and reporting requirements of different stock exchanges. It is generally true that a resource estimation that complies with the JORC code (or one of its sister codes) will meet the standards of most international investors.

The USGC 1976 Reporting Code as specifically quoted in this report for the Sierra Grande mine, uses reporting classification terminology which is in line with those outlined for the JORC code below. As reference a comparison of the JORC code and USGS 1976 code for reporting resources and reserves is shown in Table C4 and C5.

Table C4 USGS 1976 Resource Classification Scheme (Bulletin 1450 A) to JORC conversion guideline

USGS Identified resources		JORC
	Measured	Measured
Demonstrated	Indicated	Indicated
Inferred		Inferred (if supported by sampling or geological continuity

Table C5 USGS 1976 Reserve Classification Scheme (Bulletin 1450 A) to JORC conversion guideline

	USGS			JORC		
	Identified resources					
	Measured	Indicated	Inferred	Measured	Indicated	Inferred
Economic	reserves		inferred reserves	Proven	Probable	Not classified
Marginally Economic	marginal reserves		inferred marginal reserves	Reserves	Reserves	as reserves
Sub Economic	demonstrated sub economic resources		inferred sub economic resources	Not classified as reserves or resources		

The new Chinese code is a blend of the old Chinese Code and the codes in current use today, including JORC and the current United Nations (UN) standard, with some additional local components added.

JORC is a non-prescriptive code, in that it does not lay out specific limits for resource classification in terms of such things as borehole spacing. Instead it emphasises the principles of transparency, materiality and the role of the Competent Person. Whilst some guidelines do exist (e.g. the Australian Guidelines for the Estimation of Coal Resources and Reserves) they are not mandatory and classification is left in the hands of the Competent Person. When combined with its Professional Standards (which are effectively mandatory), the Chinese code is much more prescriptive but does not include the role of the Competent Person.

An examination of the details of the Chinese code suggests that in terms of broad categorisation, the levels of geological confidence ascribed to Measured and Indicated resources are quite similar in both the codes. The ranges of borehole spacings, thickness cut-offs and quality limitations that are enforced by the Chinese system would generally result in the same resource classification under the JORC Code.

The JORC Code uses the following definitions for Mineral Resources and Ore Reserves:

Measured Mineral Resource is that part of Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a **high** level of confidence. It 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. The locations are spaced closely enough to confirm geological and grade continuity.

Indicated Mineral Resource is that part of Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a **reasonable** level of confidence. It 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. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.

Inferred Mineral Resource is that part of Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a **low** level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.

Exploration Target/Results includes data and information generated by exploration programmes that may be of use to investors. The reporting of such information is common in the **early** stages of exploration and is usually based on limited surface chip sampling, geochemical and geophysical surveys. Discussion of target size and type must be expressed so that it cannot be misrepresented as an estimate of Mineral Resources or Ore Reserves.

A **'Proved Ore Reserve'** is the economically mineable part of a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. These assessments demonstrate at the time of reporting that extraction could reasonably be justified

A Proved Ore Reserve represents the highest confidence category of Ore Reserve estimates. This requires detailed exploration and quality data "points of observation" to provide high geological confidence.

A **'Probable Ore Reserve'** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. It includes diluting materials and allowances for losses which may occur when the material is mined. Appropriate assessments and studies have been carried out, and include consideration of and modification by realistically assumed mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors These assessments demonstrate at the time of reporting that extraction could reasonably be justified.

A Probable Ore Reserve has a lower level of confidence than a Proved Ore Reserve but has adequate reliability as the basis of mining studies.