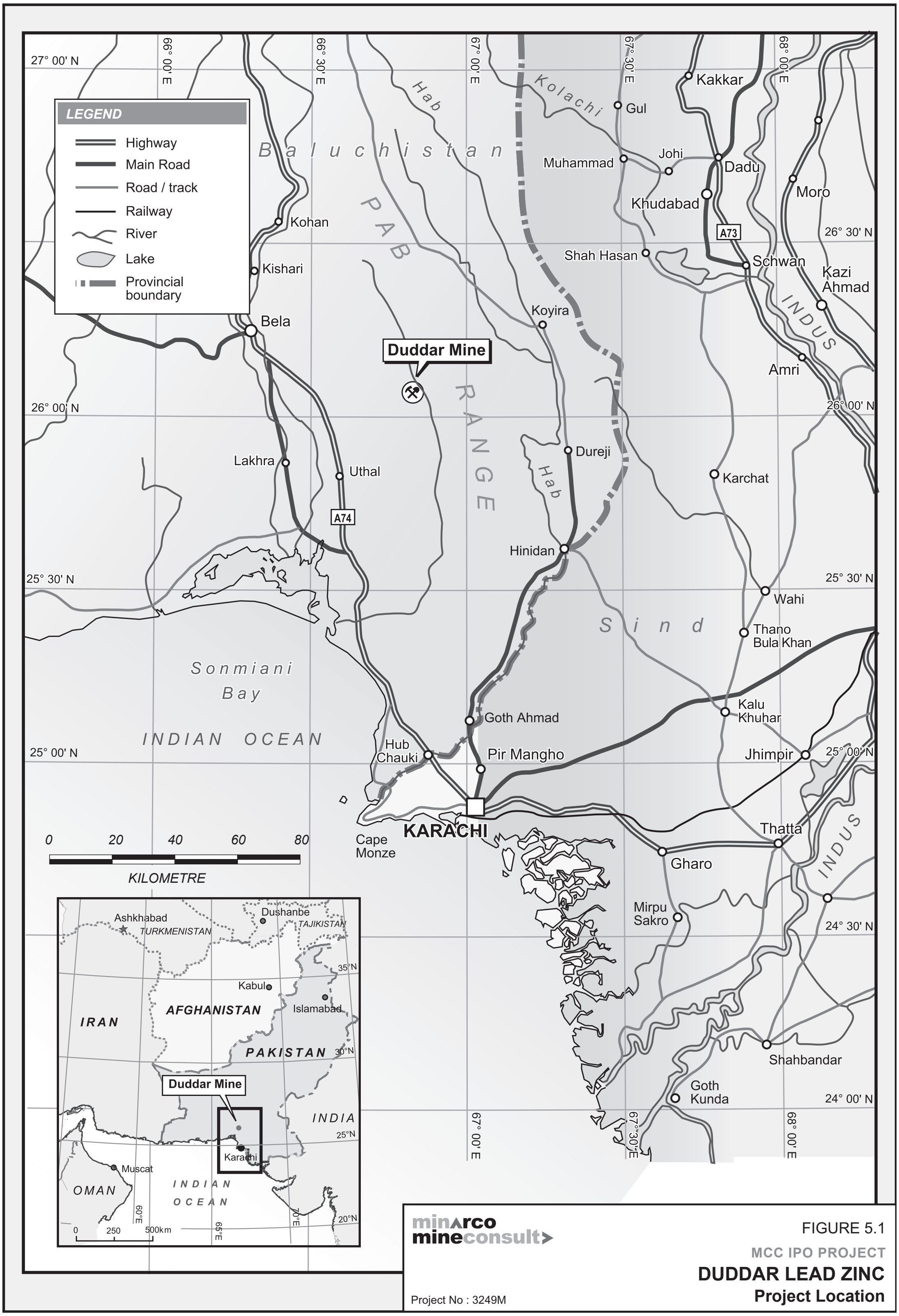
Table 5.1 — Duddar Lead Zinc — Mining Lease ML-100(132)

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

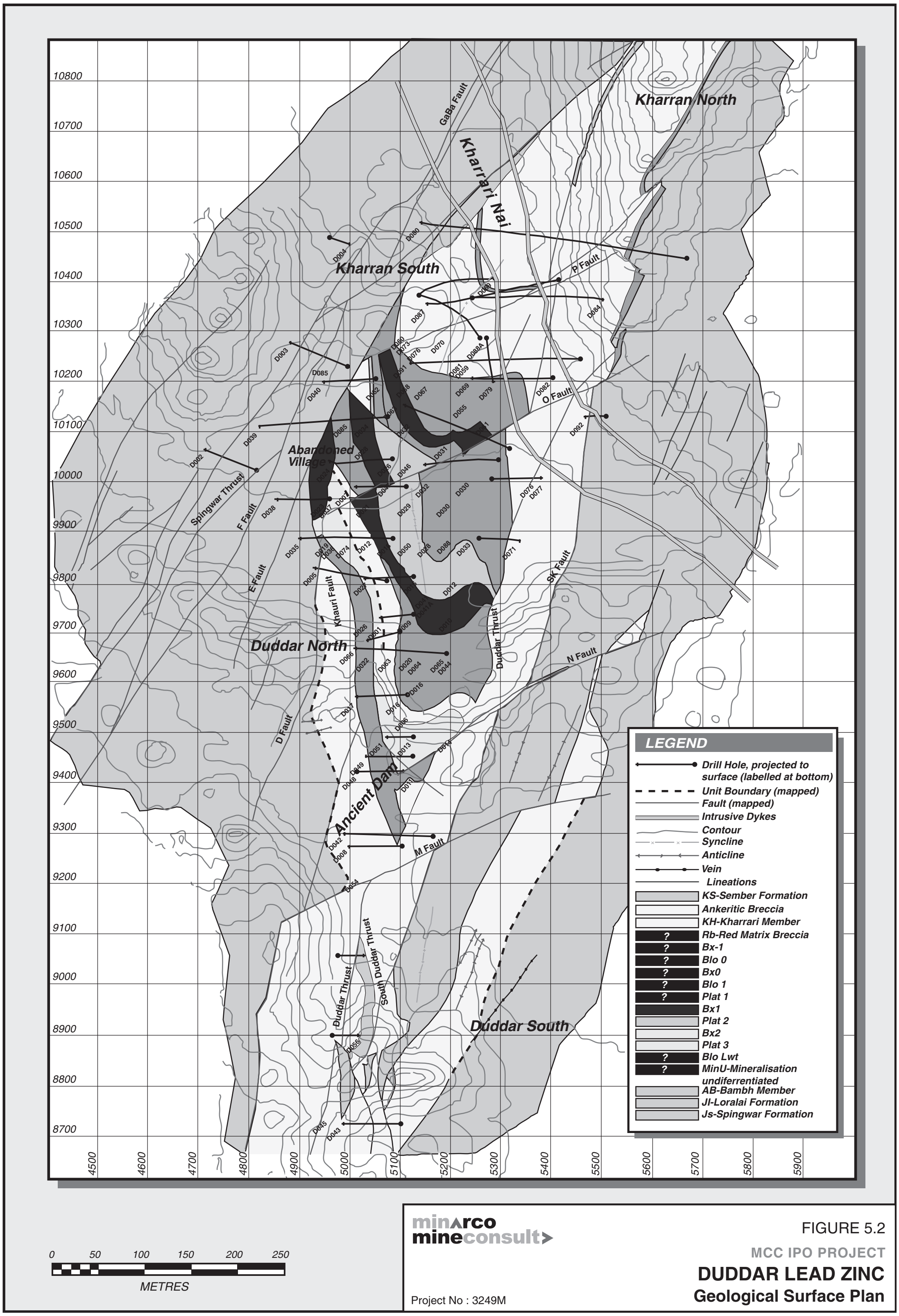
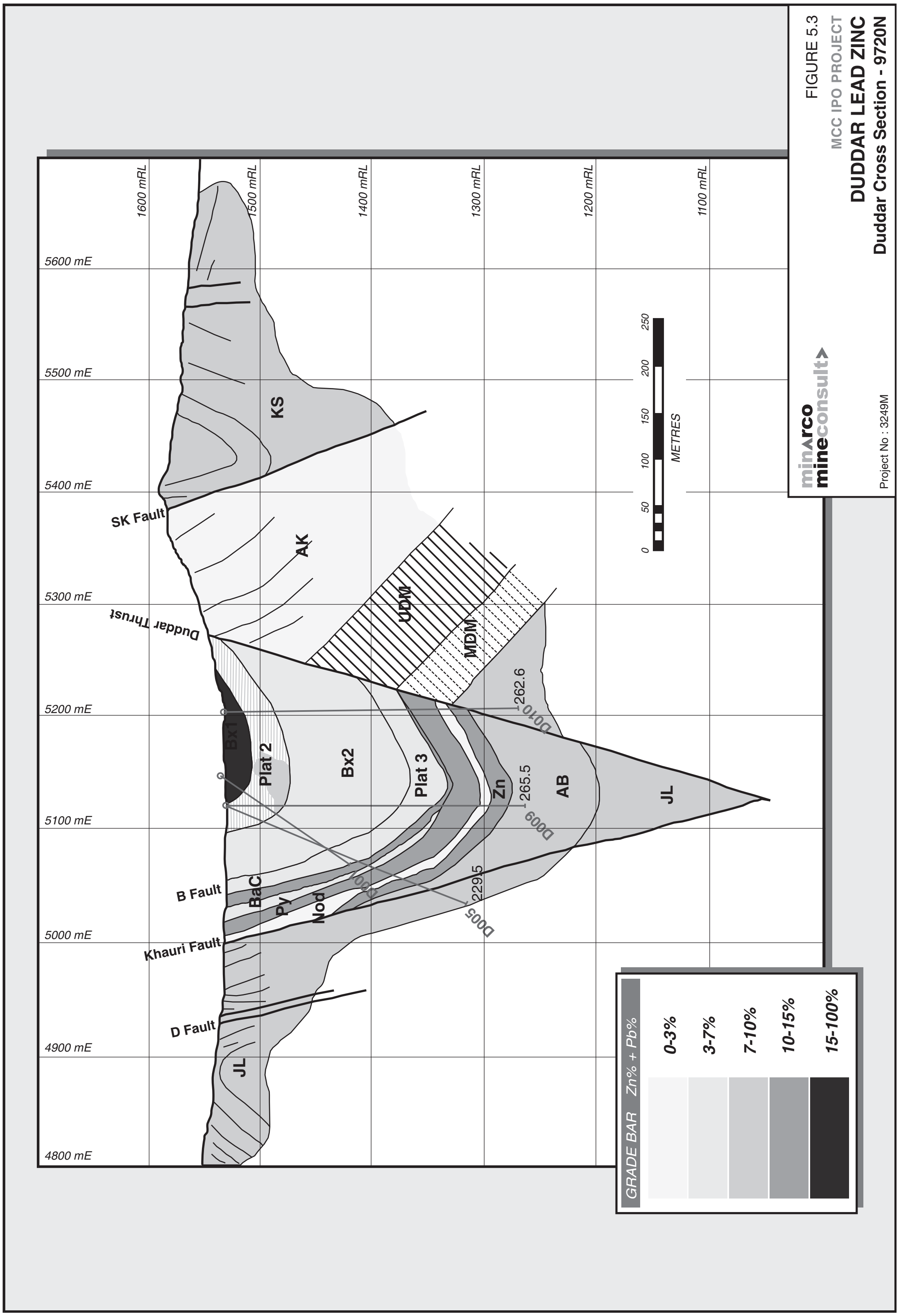


Figure 5.3 — Duddar Lead Zinc — Geological Cross Section



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

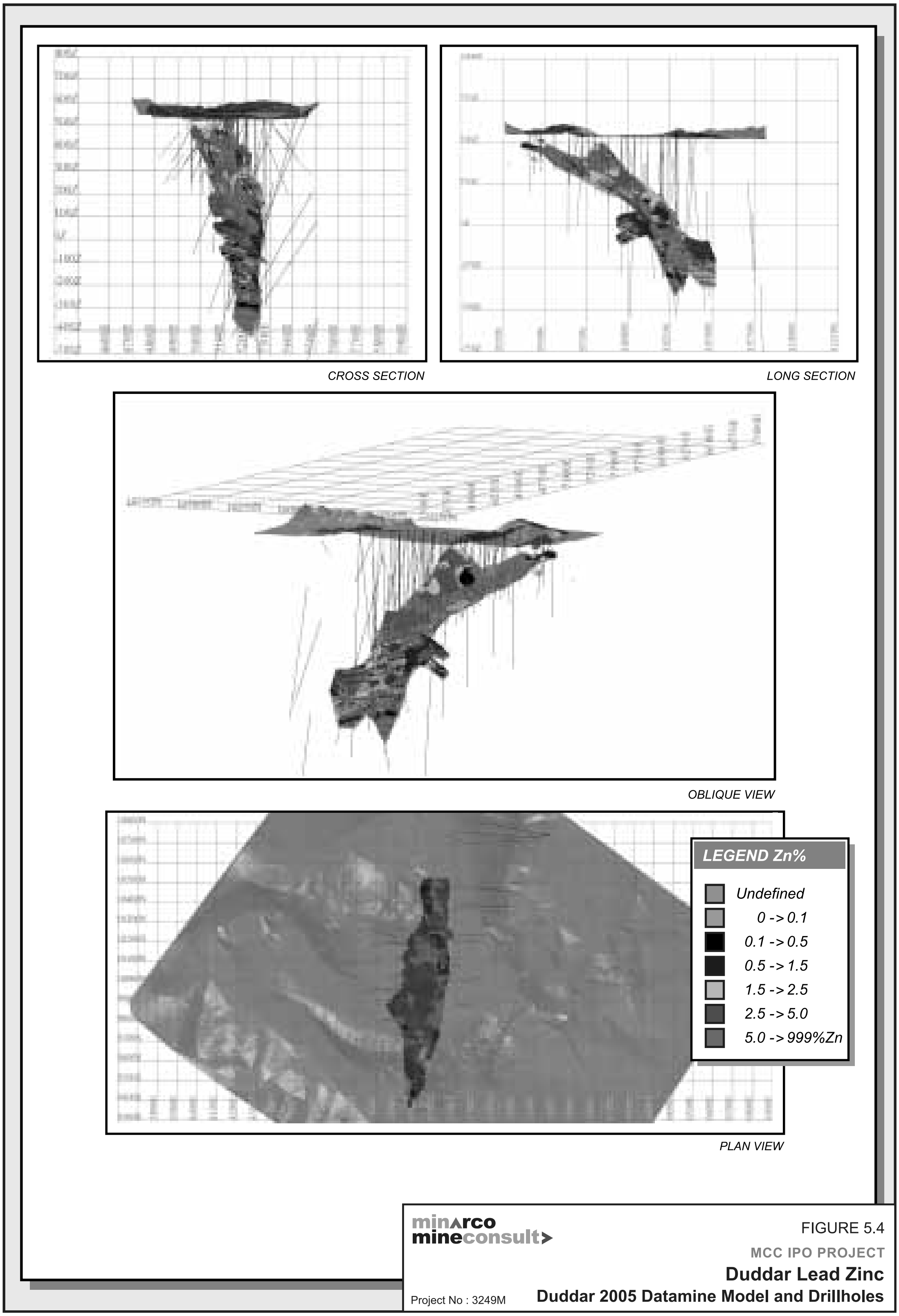
	Total Mineable Quantities			Metal Tonnes	
Zones	Tonnes (Mt)	Pb %	Zn %	Pb (kt)	Zn (kt)
Total	9.13	3.0	9.3	273	849

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.

Whilst no JORC compliant reserves have been estimated, the likely result would be of a similar order of magnitude to the current estimates of 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

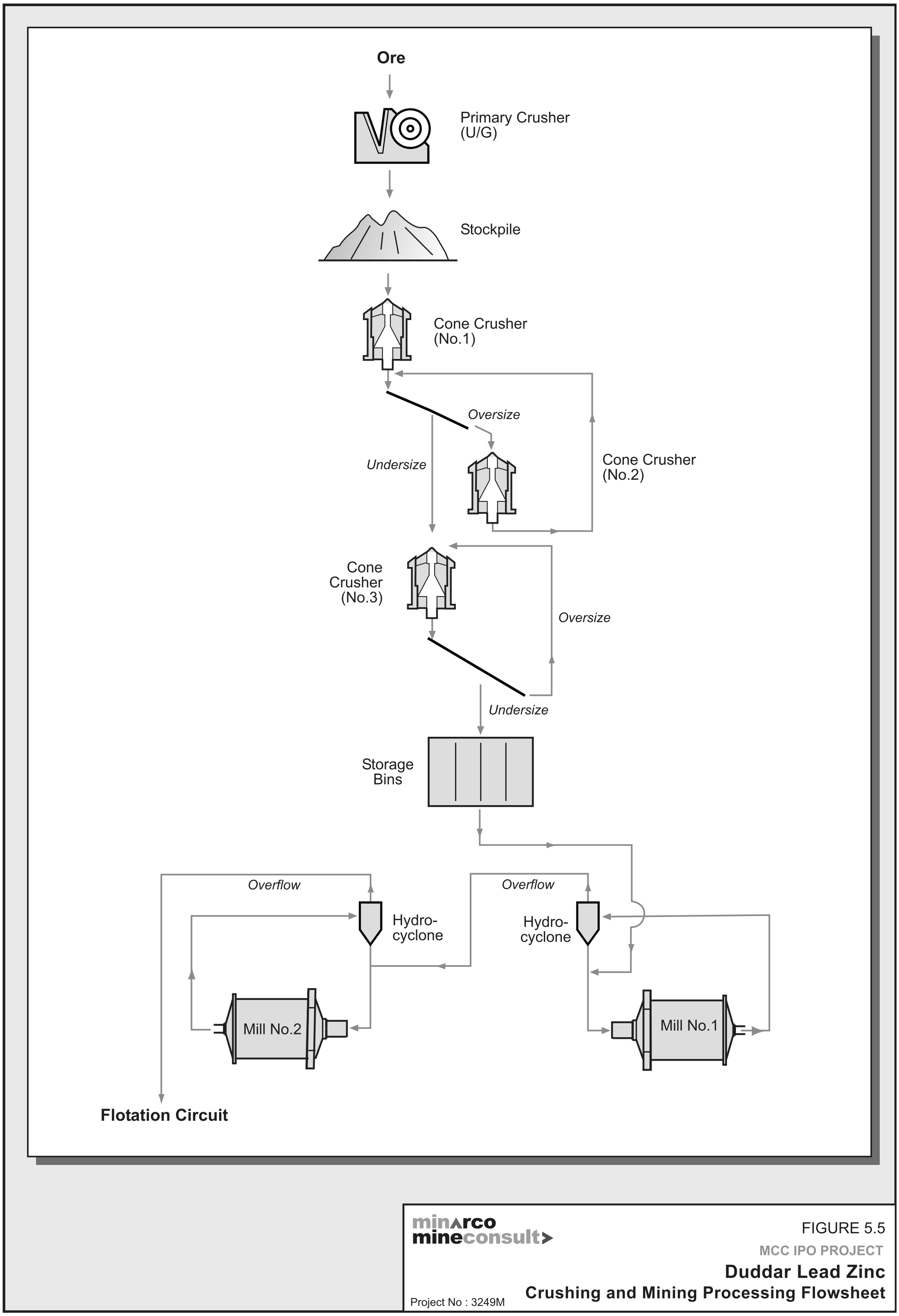
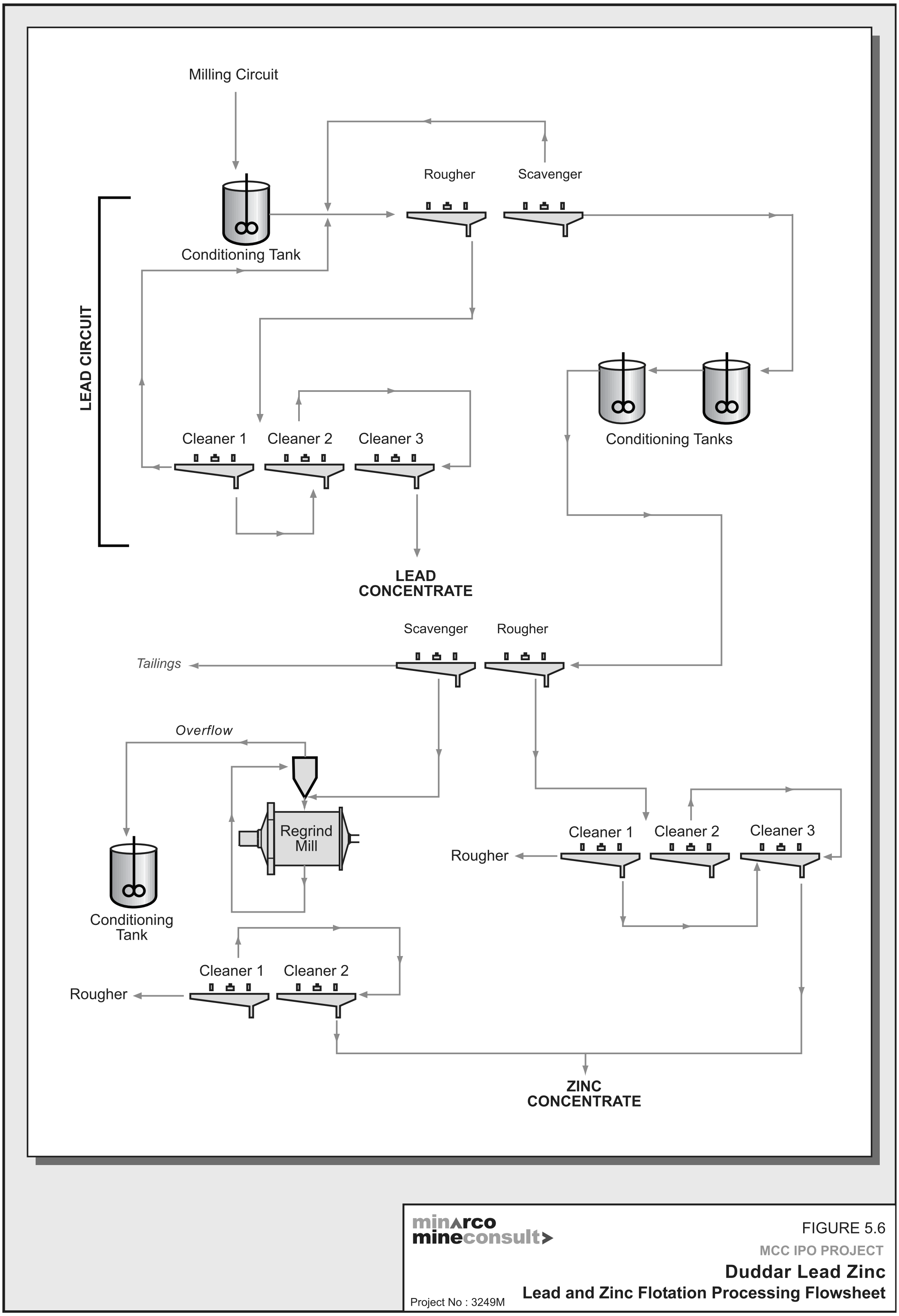


Figure 5.6 — Duddar Lead Zinc — Flotation Processing Flowsheet



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.

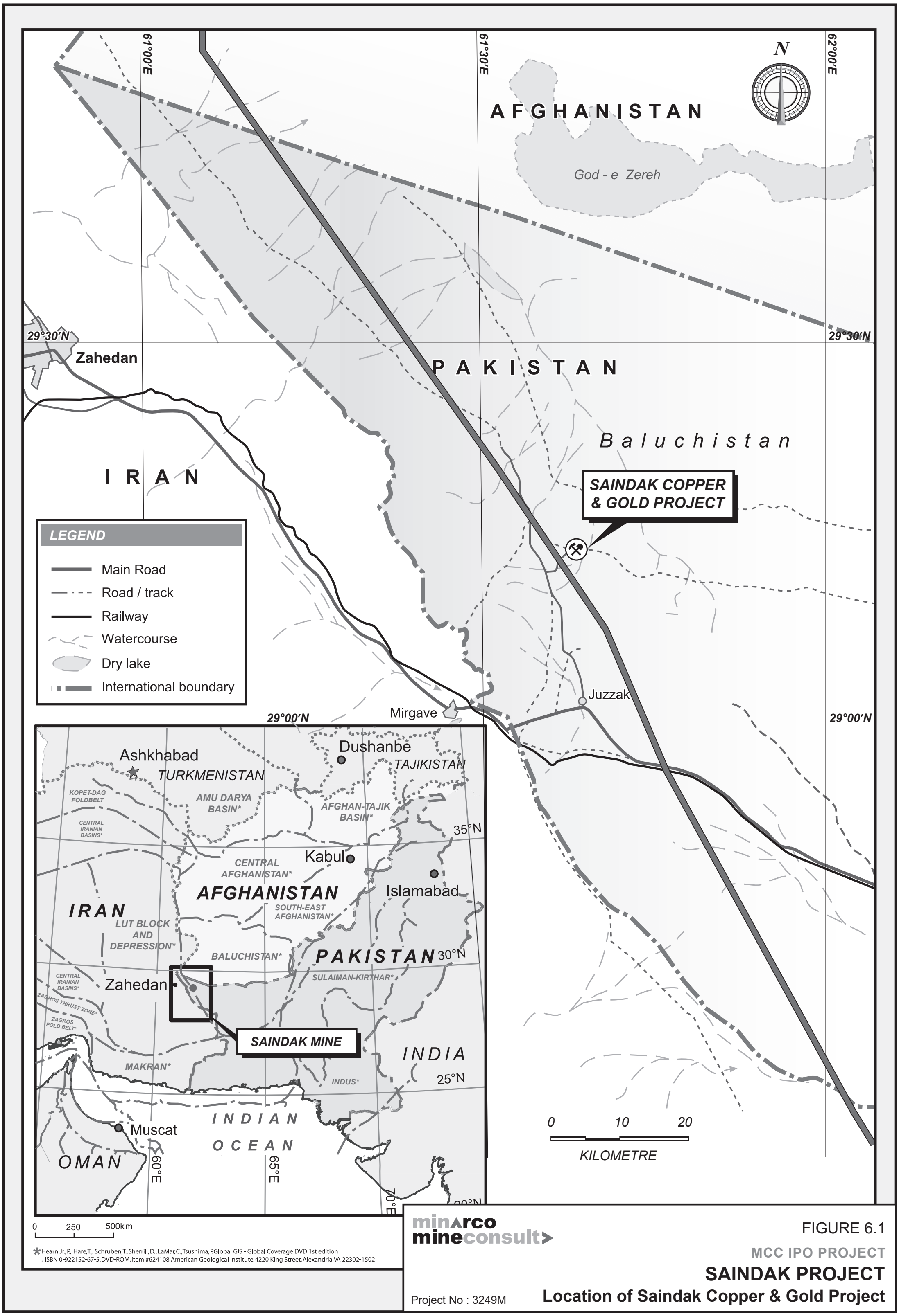
The exploration history is summarised in **Table 6.3.**

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

Orebody	Dimensions of the orebody			Comments
	Vertical	Length	Width	
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***.

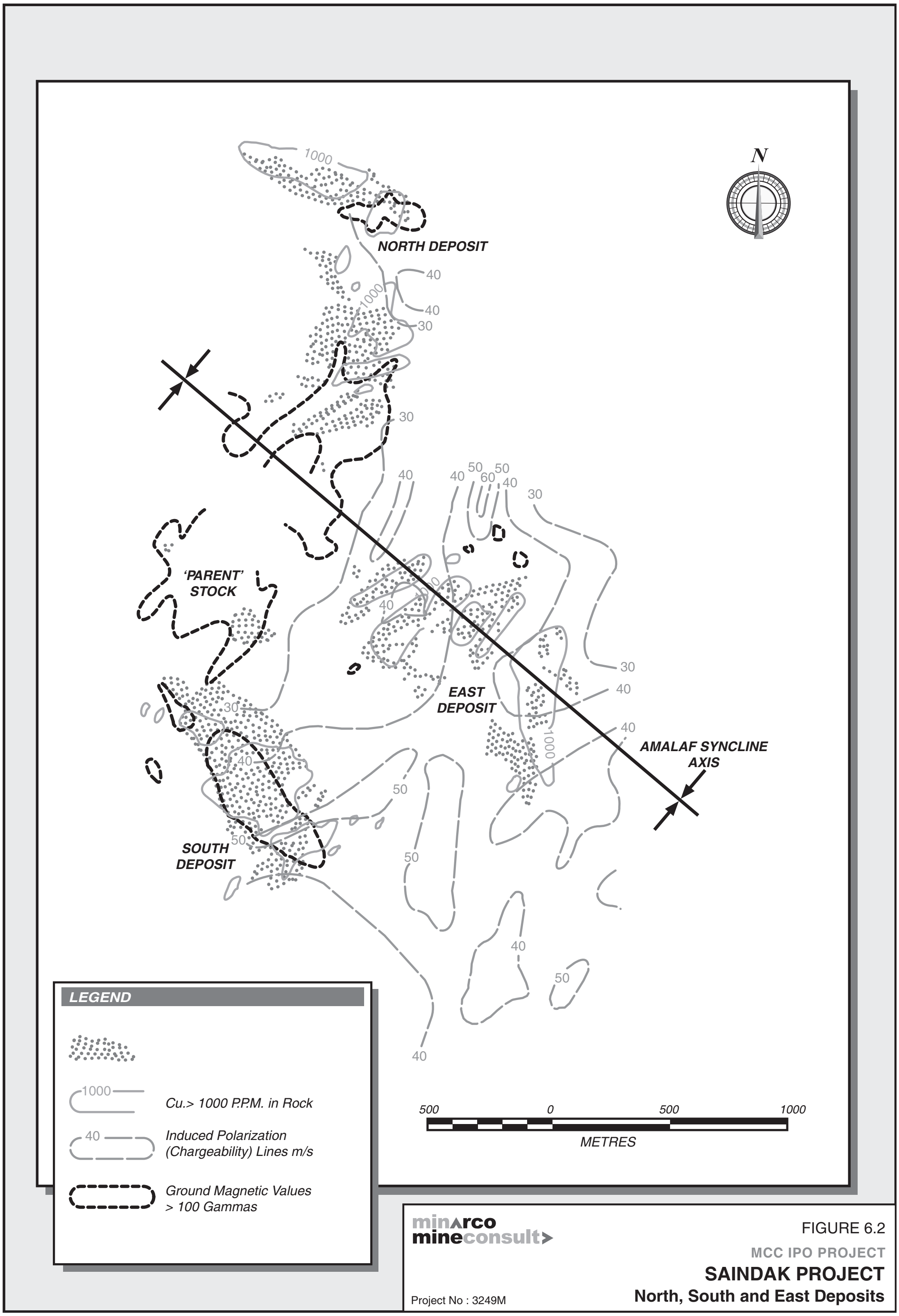
Table 6.5 — Saindak Copper Gold — Grade Distribution in Alteration Zones

Alteration Zone	Average grade		Comments
	Cu (%)	Au (g/t)	
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.

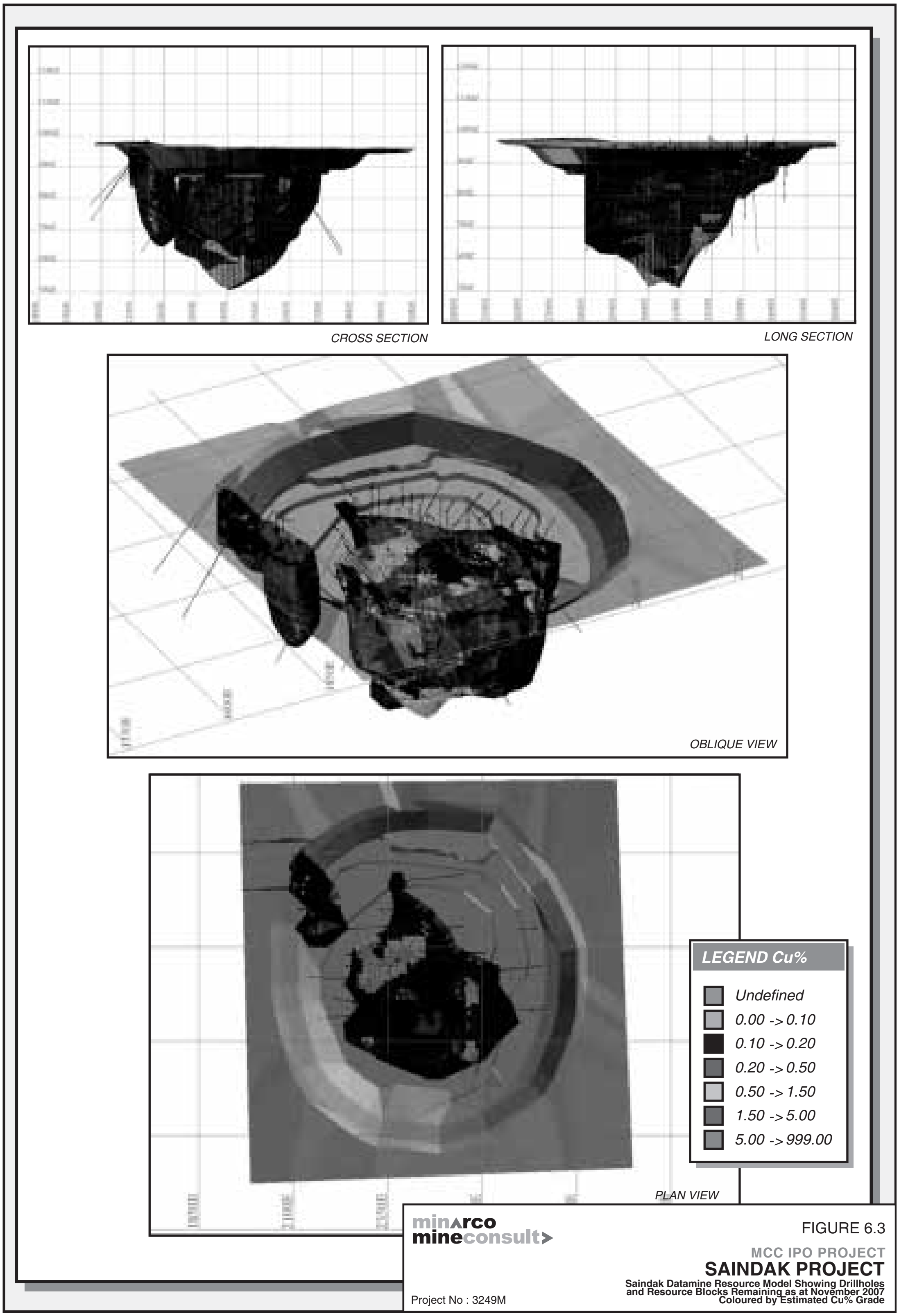
Table 6.8 — Saindak Copper Gold — (South Orebody) M-MC Estimated Mineral Resources, as at December 2008

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.
We use the geological reserves standard published by the PRC Government in reporting the Mineable Quantities for the Saindak copper-gold mine. Whilst no JORC compliant reserves have been estimated, the likely result would be of a similar order of magnitude to the current estimates of Mineable Quantities.

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

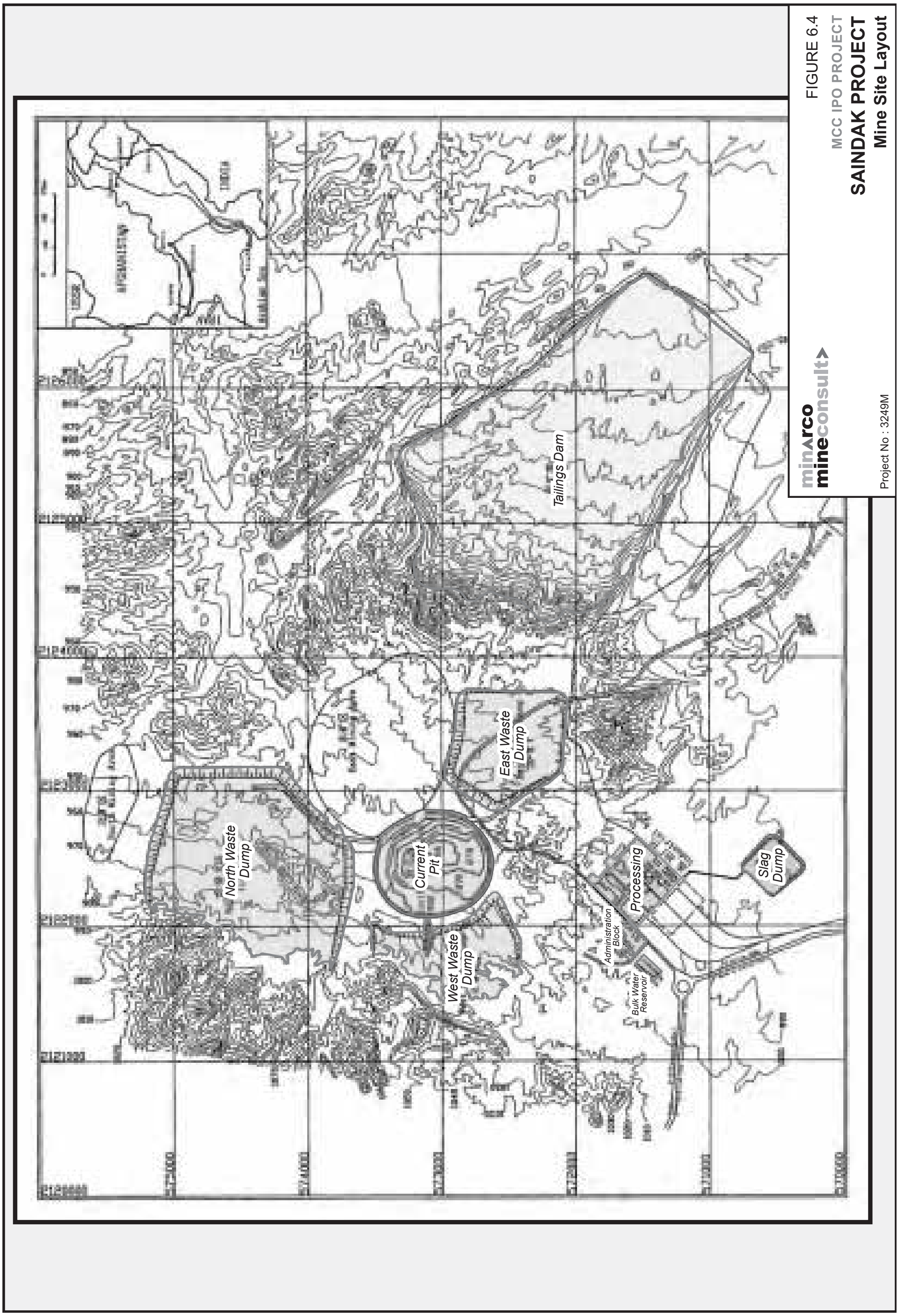
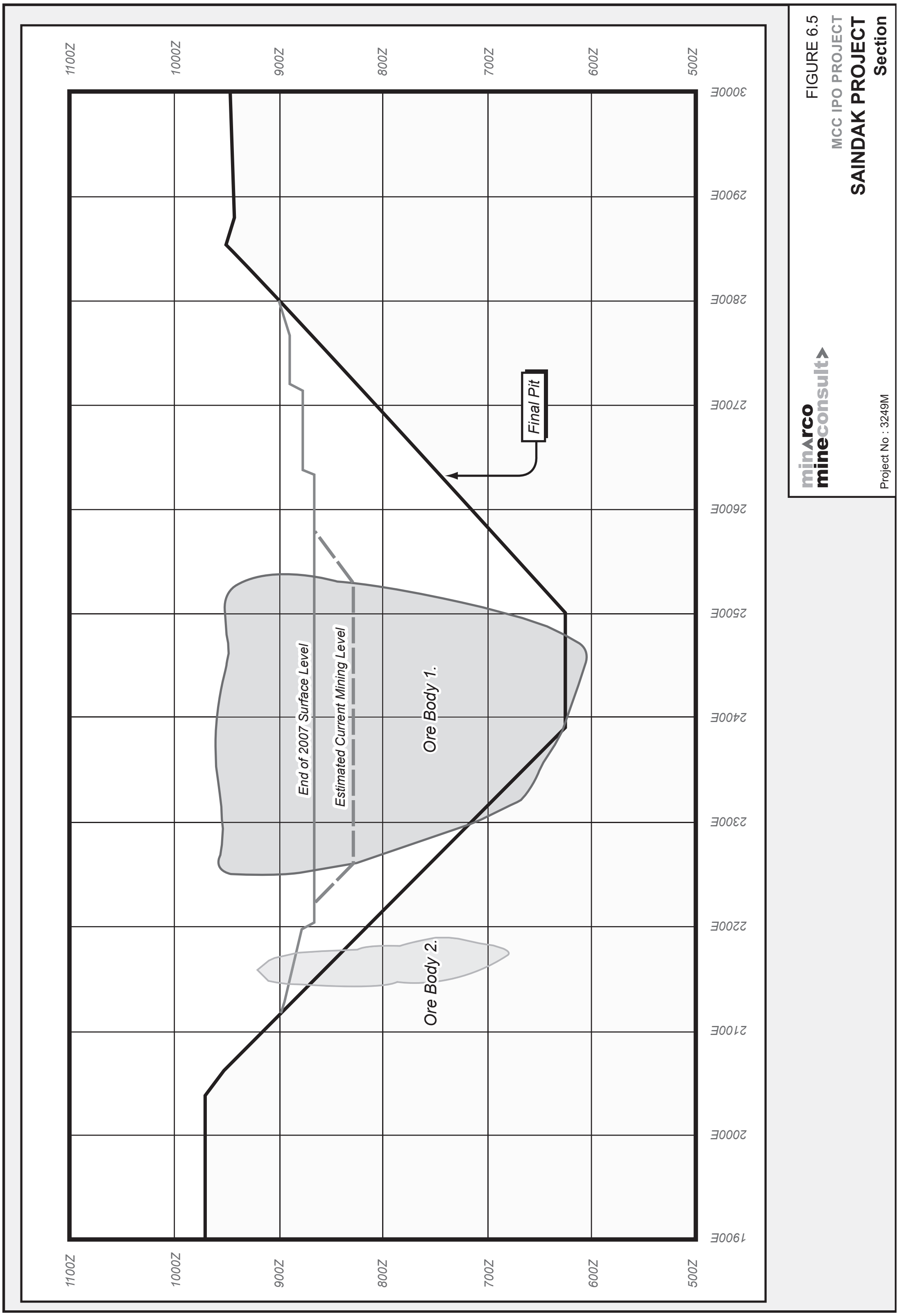


Figure 6.5 — Saindak Copper Gold — Estimated Current Mining Level



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

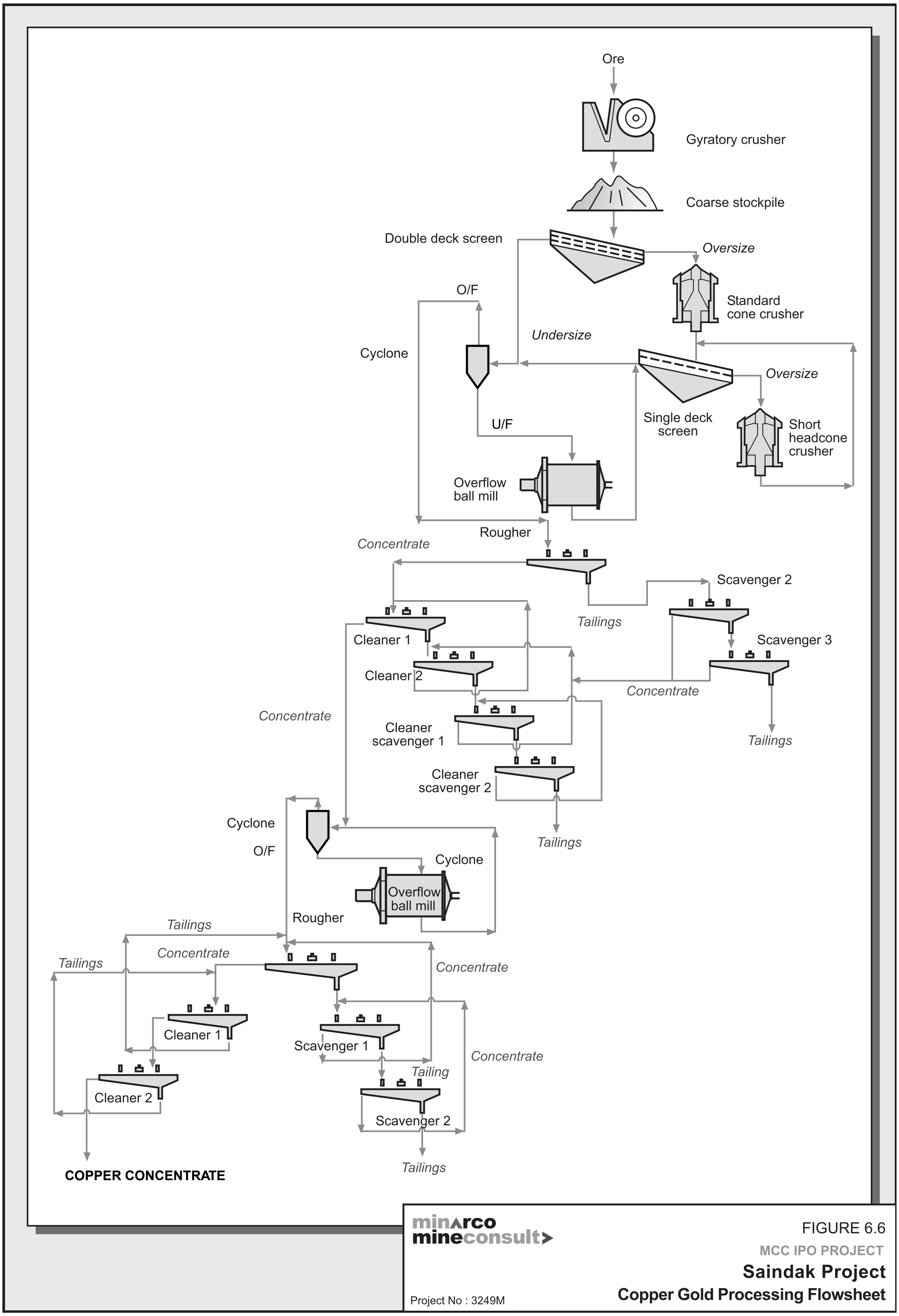
The copper concentrate is dewatered and smelted in a 140m^2 reverberatory furnace with a specific smelting, capacity of 2.47t/m^2 d. Two P-S40 rotary converters (3.2mØ x 6.6m long) are employed to produce blister copper at a rate of 70tpd and a recovery of 97.5% Cu. The blister copper is then cast into 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

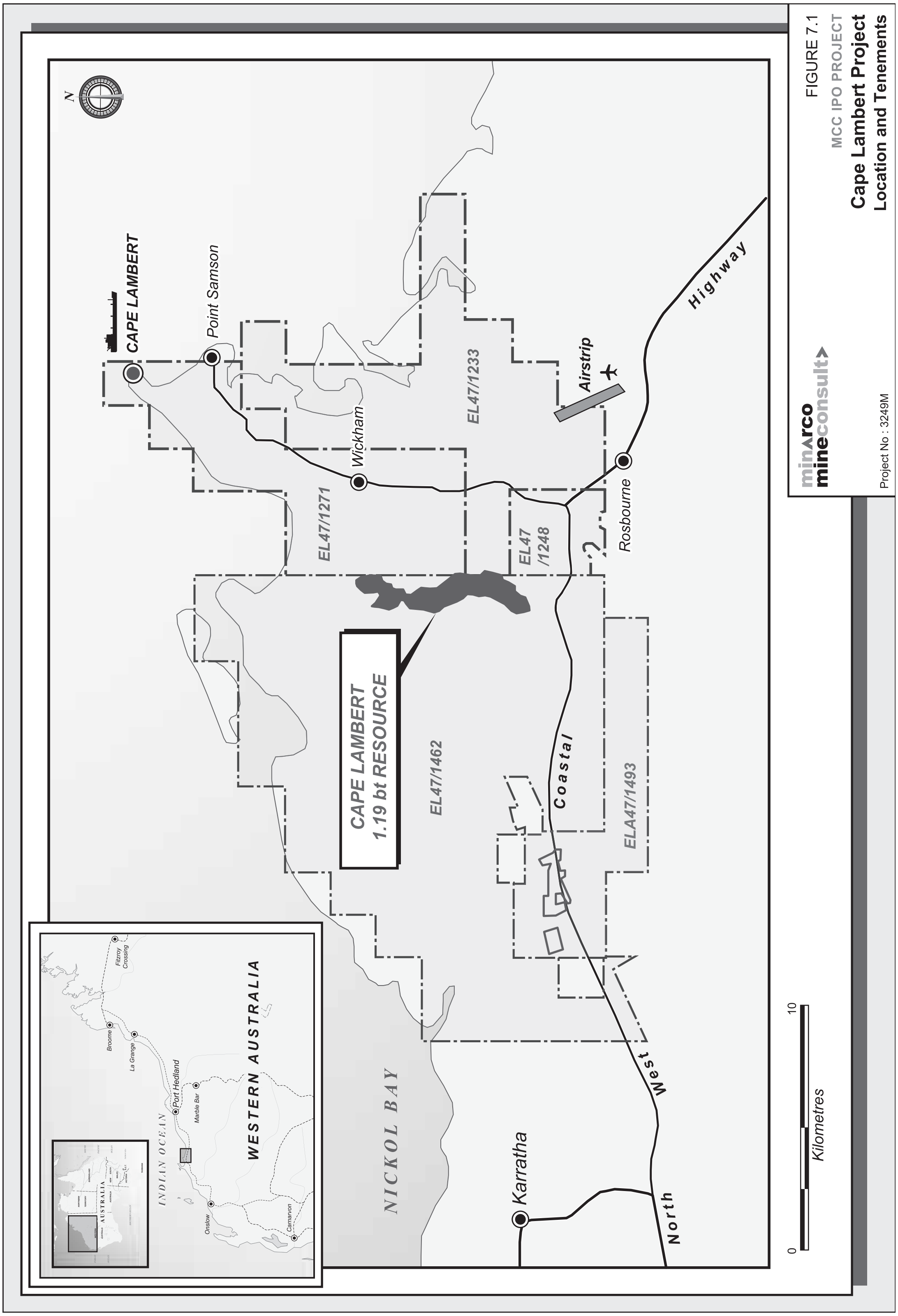
Mine/Project	Cape Lambert
Title	Exploration Licence
No	E47/1462
Owner	Cape Lambert Iron Ore Ltd
Mine/Project Name	n/a
Mine Method	n/a
Permit Capacity	n/a
Permit Area	70 blocks
Permit Depth	n/a
Valid Date	March, 24th 2006 - March 23rd 2011
Issue Date	October, 28th 2004
Issuer	Department of Industry and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

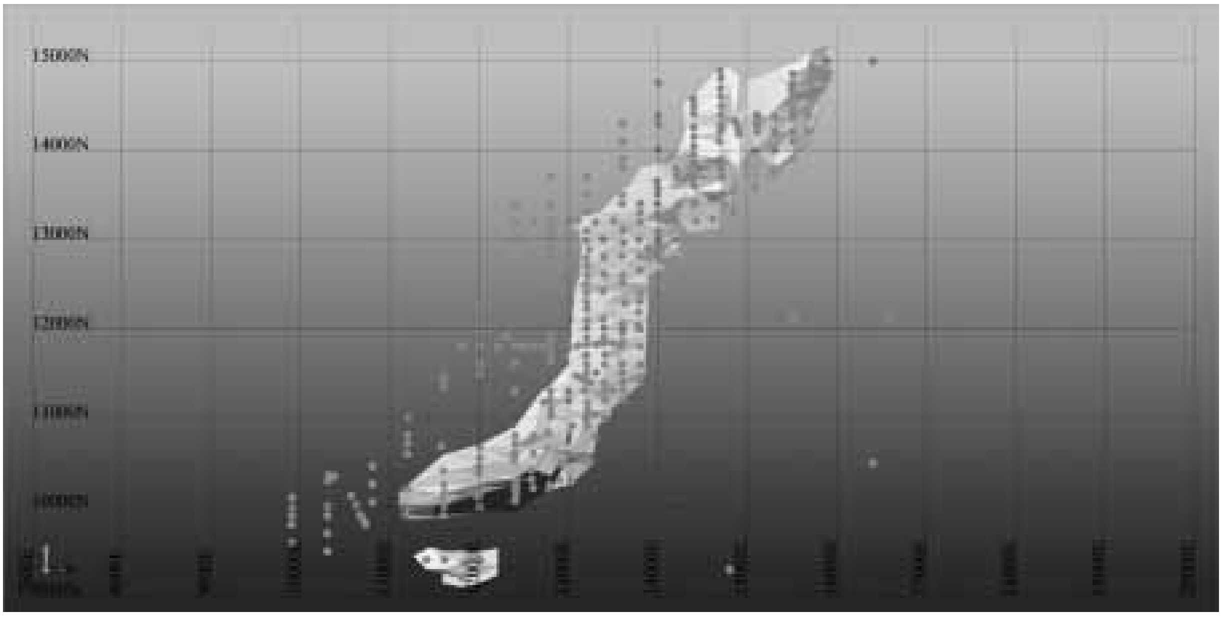
The exploration history is summarised in **Table 7.5.**

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