6. METALLURGICAL PROCESSING

6.1 Introduction

This section includes discussion and comment on the metallurgical processing aspects associated with the Material Properties. Specifically, detail is given on the process metallurgy and process engineering aspects relating to plant capacity, availability and metallurgical performance as incorporated in the LoM Plans.

6.2 Iron Ore

6.2.1 Sishen Mine and SEP

Sishen Mine: The original ore handling plant at Sishen Mine was a dry crushing and screening plant commissioned in 1953. In 1963, the first dense medium separation ("DMS") plant, South Plant, was commissioned. The North Plant, with a design capacity of 18Mtpa, was commissioned in 1976. In 1984, it was decided to rationalise production and the South plant was shut down.

Four types of hard high-grade ore are presently mined from the Sishen pit, namely massive, laminated, conglomerated and brecciated iron ore. The supply to the primary crusher of a suitable mixture of RoM ore largely controls the chemical quality of the final products. Presently only material with an iron content of greater than 60%Fe is fed to the plant.

Open pit ore is crushed via a primary gyratory crusher and two secondary cone crushers ahead of primary stockpiling. A smaller in-pit gyratory crusher is also available as required. Ore is withdrawn from the primary stockpile and sized into various fractions by washing and screening. Ore in the size range –90+25mm is beneficiated in the coarse dense medium ("DM") drum plant. This circuit also includes a Larcodem dense medium vessel. Ore in the size range –25+8mm is beneficiated in the medium DM drum plant. Fine ore is split into two size fractions, –8+5mm and –5+2mm ahead of beneficiation in the coarse and fine DM cyclone plants, respectively. The –2+0.2mm fraction is forwarded to a new up-current classifier circuit for beneficiation. Product from the coarse drum plant undergoes quaternary crushing and screening to meet product size specifications, whilst the other circuits are correctly sized ahead of beneficiation. In total five products are produced:

- Sishen 66%Fe 27mm Direct Reduction Ore;
- Sishen 66%Fe 25mm Lumpy Ore;
- Sishen 66%Fe 20mm Lumpy Ore;
- Sishen 65%Fe 8mm Coarse Sinter Ore; and
- Sishen 65%Fe 5mm Fine Ore.

Dense medium rejects are stored on waste dumps whilst slimes are stored in tailings dams.

Capacity at the North Plant has been steadily increased to the current capacity of 28 to 29Mtpa.

Considering its age, the plant appears to be in a fair condition, both mechanically and structurally. With normal preventative maintenance and continuation of the refurbishment programmes already initiated, the plant can be expected to operate for the period scheduled in the LoM Plan.

Key historical processing statistics for the Sishen Mine Process Facility are summarised in Table 6.1.

Table 6.1 Sishen Mine: Main Plant Operating Statistics

Description	Units	2001(F)	2002(F)	2003(H2)	2004(C)	2005(C)	2006(C)
Headfeed	**(Mt)**	30.9	32.0	16.3	32.8	31.8	33.5
Product	**(Mt)**	26.3	26.8	13.5	27.5	28.8	29.0
Proportion Fine	**(%)**	33	32	31	30	31	33
Plant Yield	**(%)**	84	82	83	85	89	82

(F) Financial Year ended 30 June.

(H2) Six months ended 31 December due to the change of Financial Year.

(C) Calendar Year ended 31 December.

The LoM Plan assumes an average RoM throughput of approximately 32.4Mtpa. The LoM yield is projected at an average of approximately 88% to yield total product of 29Mtpa. This is in line with current performance but somewhat higher than that achieved in recent years. The current performance has principally been ascribed to improved ore definition and the implementation of selective mining.

Average product quality achieved in recent years is summarised in Table 6.2.

Table 6.2 Sishen Mine: Main Plant Product Qualities

Description	Units	Fe	SiO_2	Al_2O_3	K_2O	P	Oversize Max	Undersize Max
27mm DR Ore								
2004 – 2005 Average	(%)	66.35%	2.92%	1.20%	0.12%	0.054%	10.6%+27mm	4.3%–13mm
Current Specification	(%)	66.00%	3.70%	1.50%	0.16%	0.057%	15.0%+27mm	5.0%–13mm
25mm Lump Ore								
2004 – 2005 Average	(%)	66.30%	2.90%	1.22%	0.13%	0.055%	6.4%+25mm	4.1%–8mm
Current Specification	(%)	66.00%	3.70%	1.50%	0.16%	0.057%	7.5%+25mm	5.3%–8mm
20mm Lump Ore								
2004 – 2005 Average	(%)	66.33%	2.88%	1.23%	0.13%	0.055%	8.2%+20mm	6.2%–8mm
Current Specification	(%)	66.00%	3.70%			0.057%	20.0%+20mm	9.0%–8mm
8mm CS Ore								
2004 – 2005 Average	(%)	65.85%	3.14%	1.40%	0.16%	0.057%	18.6%+8mm	8.5%–5mm
Current Specification	(%)	65.00%	4.20%	2.00%	0.24%	0.066%	22.0%+8mm	16.0%–5mm
5mm Fine Ore								
2004 – 2005 Average	(%)	65.49%	3.28%	2.08%	0.19%	0.061%	6.5%+5mm	7.0%–0.2mm
Current Specification	(%)	65.00%	4.20%	2.00%	0.24%	0.066%	8.4%+5mm	12.0%–0.2mm

It is evident that all other specifications have generally been met in recent years.

Sishen Expansion Project: Feed to the existing beneficiation plant is restricted to material with a grade of greater than or equal to 60% beneficiated Fe in order to meet the required product specifications. Included in Sishen's growth strategy is the implementation of the brown field Sishen Expansion Project ("SEP") aimed at beneficiating lower grade material in the range of 45% in situ Fe to + 60% in situ Fe to saleable product quality. This results in a significant increase in resource base and utilisation thereof.

Due to high separation densities required to beneficiate such material, DMS as currently employed at Sishen is not a suitable technology. Jigging however, is a viable option and a feasibility study incorporating this technology was completed in January 2005. Project start up is planned for July 2007, with capacity of 10Mtpa saleable product being realised by June 2008 and a further 3Mtpa by 2015.

A commensurate increase in Sishen Iron Ore's allocation on the iron ore export channel capacity from 23.5Mtpa to 35Mtpa is planned.

Extensive laboratory and pilot plant testwork was undertaken through the various phases of investigation:

- Pre-feasibility characterisation of stockpile material and mine samples;
- Feasibility characterisation of ten mine samples with confirmatory pilot plant tests; and
- Feasibility optimisation on forty-eight mine samples.

In the interest of sample representivity, significantly large primary samples of up to 3,000t were taken from stockpiles and pit faces for pre-feasibility and feasibility investigations, with 80t primary samples being taken for feasibility optimisation studies. These in turn were crushed before secondary samples of approximately 3t each were split out for laboratory testwork.

The testwork programme focused on selection of the best relative cut density, the generation of beneficiation curves for various ore types and stockpiles that could be included in the geological model, prescription of the metallurgical flowsheet and determination of design parameters for engineering design.

A Mineral Density Separator ("MDS") which is essentially a batch jig was used for laboratory characterisation of the various ore types and stockpiled material. MDS results were modified via a standard procedure to allow for process imperfection. Whilst each material type has a unique beneficiation curve, the modified results confirmed that at separation densities between 4.0g/cm^3 and 4.2 g/cm^3, lump and fine product at 64%Fe and 63.5%Fe, respectively, can be produced from feed between 50%Fe and 60%Fe at yields in excess of 60%.

The pilot plant tests were run in two campaigns as certain shortcomings were identified in the initial campaign. The second campaign confirmed the MDS beneficiation algorithms for the coarse and medium jigs but not the fine jig. Medium and fine jig capacity was also shown to be lower than originally anticipated. This was evaluated ahead of detailed design and subsequently a fourth jigging module was included. Pyrometallurgical testwork undertaken on the lump and fine products generally found both to compare well with current Sishen ore.

The SEP product specifications as summarised in Table 6.3 were determined through an iterative process between the resource beneficiation characteristics and market requirements. Laboratory and pilot test results confirmed that these specifications will be met in practice.

Table 6.3 SEP: Plant Product Qualities

Description	Units	Fe	SiO_2	Al_2O_3	K_2O	P	Oversize Max	Undersize Max
Lump Ore								
Sishen Specification	**(%)**	66.00%	3.70%	1.50%	0.16%	0.057%	6.5%+25mm	10%–8mm
SEP Specification	**(%)**	64.00%	5.90%	1.50%	0.16%	0.065%	6.5%+25mm	12%–8mm
Sishen Typical	**(%)**	66.27%	2.93%	1.25%	0.15%	0.055%	6.4%+25mm	6.5%–8mm
SEP Expected	**(%)**	64.35%	5.50%	1.22%	0.16%	0.063%		
Fine Ore								
Sishen Specification	**(%)**	65.00%	4.20%	2.00%	0.24%	0.066%	7.5%+5mm	18%–0.2mm
SEP Specification	**(%)**	63.50%	6.30%	2.00%	0.24%	0.074%	10%+8mm	18%–0.2mm
Sishen Typical	**(%)**	65.52%	3.26%	1.59%	0.19%	0.066%	5.1%+5mm	8.5%–0.2mm
SEP Expected	**(%)**	64.37%	5.20%	1.70%	0.24%	0.067%		

Letters of intent from existing Kumba clients support the demand for product of such chemical, physical and pyrometallurgical quality.

The proposed SEP flowsheet and process design criteria largely recognise the testwork findings. RoM ore will be fed to the primary gyratory crusher directly from the mine or from RoM stockpiles. Primary crusher product drops into a rock box ahead of a scalping screen and the secondary gyratory crusher. Scalping screen underflow and secondary crusher product drop into a rock box ahead of conveying to an intermediary stockpile. Material withdrawn from the stockpile is conveyed overland to the closed circuit tertiary crushing and screening plant. Screen underflow is conveyed to two pre-beneficiation blending beds. These principally serve to blend and homogenise the feed ahead of beneficiation. They also decouple the crushing and downstream beneficiation plant which in turn provides a maintenance buffer, improves mining equipment utilisation and allows for continuous feed to downstream beneficiation.

Material reclaimed from the pre-beneficiation blending beds is conveyed to three identical beneficiation modules comprising screening into a coarse (–25+8mm), a medium (–8+3mm) and a fine (–3+0.8mm) fraction. The three fractions report separately to coarse, medium and fine jigging. Product (sinks) from the coarse jigs is extracted via vibrating feeder ahead of screen dewatering and deposition on the lump product bed. Product (sinks) from the medium and fine jigs is extracted via vibrating feeders ahead of two stage screen and bunker de-watering before deposition on the fine product bed. Ore reclaimed from the product beds is conveyed to three existing load out stations for rail despatch to clients.

Waste (floats) from all jigs is dewatered before being conveyed to the discard dump. Each module has a degrit system comprising cyclones and dewatering screens. The –0.8+0.2mm fraction is combined with the plant discard and the –0.2mm fraction is thickened and pumped to tailings.

Presently the design excludes the processing of the –0.8mm fraction through up current classifiers. Such units will be tested and are likely to be included at a later stage.

Certain key aspects did require ratification before finalisation of the design. Firstly, a new generation screen with typical design specifications to those proposed for the SEP was tested in the Sishen washing and screening plant. The top deck did not meet efficiency claims under dry screening conditions. It is believed however, that sufficient screening capacity has been installed in the SEP tertiary crusher plant. This has been confirmed before finalisation of SEP screen selection. Secondly, utilisation of a bucket elevator to extract product from below the jig is more conventional than the proposed utilisation of a screen. Screen extraction has however, been successfully utilised in Australia at industrial scale. Screen extraction was selected on the basis of a visit to such an installation and has been ratified during final design.

Risk mitigation activities confirmed the jig throughput capacities and indicated that an additional jig beneficiation flow line was required to achieve the required product specifications and yields. These changes have been incorporated into the capital and operating cost estimates of the plant.

Annual plant throughput is planned at 15.6Mtpa RoM, comprising 13.3Mtpa low grade material and 2.3Mtpa high grade material at a planned yield of 64% to produce 10Mtpa of product (output relates to the first 10Mtpa expansion). The plant has however, been designed to process 16.7Mtpa RoM at an average design yield of 60% to produce 10Mtpa of product.

6.2.2 Sishen South Project

Metallurgical testwork was undertaken on drill core samples of the various ore types that will be encountered at the Sishen South deposits. This included the following:

- Crushing testwork;
- Chemical analysis of screen fractions;
- Densimetric analysis of coarse screen fractions;
- Pyrometallurgical behaviour of coarse screen fractions;
- De-watering characteristics of fine screen fractions; and
- Settling characteristics of slime fraction.

Due to the decision to exclude beneficiation in Phase I of the project, the crushing and pyrometallurgical characteristics are the most relevant to the project. In general the crushing and pyrometallurgical behaviour of the Sishen South products were found to be comparable with those from Sishen Mine.

In the absence of beneficiation, RoM ore will have to be mined at product quality grade and the plant will also have to be fed at product quality grade. Material that does not meet product specification will be stockpiled for blending at a suitable stage, or for processing at such time as a beneficiation plant is introduced to the project. Current understanding of the geology indicates that the Sishen South deposit comprises high quality, clastic-textured (28.8% of total), laminated (52.9% of total), collapsed breccia (9.8% of total) and conglomeratic (8.6% of total) ores. The laminated and clastic-textured ores are of uniform quality and constitute the high-grade ores (Fe > 65%) at Sishen South, although the laminated ores tend to have a variable P content. The collapsed breccia and conglomeratic ores generally have a lower Fe content and significantly higher SiO_2, Al_2O_3, K_2O and P contents than the laminated and clastic-textures varieties due to higher clay content. Furthermore, the clay content of the Ploegfontein orebody was found to high at greater than 15% compared to levels below 3% in the Leeufontein/Welgevonden/Kapstevel orebodies. Based on this understanding of the ore types, a number of actions are proposed to ensure that the final products meet the required specifications.

- Inclusion of Leeuwfontein and Welgevonden/Kapstevel resources into Phase I of the project;
- Exclusion of Ploegfontein from Phase I of the project due to an inferior in situ quality in the collapsed breccia ores which are mostly in abundance;
- Exclusion of high clay-bearing ores that would otherwise impact negatively on the levels of contaminants and fines in the products;
- Implementation of effective grade control practices utilising blast hole sampling; and
- Reduction of the interdependence between the pit and the plant and minimisation of variation in plant feed grade by stockpiling and rehandling 100% of RoM ore. Opportunities to minimise the proportion of ore rehandled are however, currently being investigated.

Successful implementation of the abovementioned actions will be important in minimising any risk of not achieving product specifications.

Sishen South product specifications are more in line with SEP product specifications than those of Sishen Mine, as shown in Table 6.4.

Table 6.4 Sishen South Project: Plant Product Qualities

Description	Units	Fe	SiO_2	Al_2O_3	K_2O	P	Oversize Max	Undersize Max
Lump Ore								
Sishen Specification	(%)	66.00%	3.70%	1.50%	0.16%	0.057%	6.55%+25mm	10%–8mm
SS Specification	(%)	64.00%	5.90%	1.50%	0.15%	0.057%	5%+25mm	12%–8mm
Sishen Typical	(%)	66.27%	2.93%	1.25%	0.15%	0.055%	6.4%+25mm	6.5%–8mm
SS Expected	(%)	64.89%	4.41%	1.45%	0.10%	0.037%		
Fine Ore								
Sishen Specification	(%)	65.00%	4.20%	2.00%	0.24%	0.066%	7.5%+5mm	18%–0.2mm
SS Specification	(%)	63.50%	6.30%	2.00%	0.24%	0.066%	10%+8mm	18%–0.2mm
Sishen Typical	(%)	65.52%	3.26%	1.59%	0.19%	0.066%	5.1%+5mm	8.5%–0.2mm
SS Expected	(%)	64.19%	4.65%	1.86%	0.14%	0.043%		

Letters of intent from existing Kumba clients support the demand for product of such chemical, physical and pyrometallurgical quality. Aluminium levels are seen as disadvantageous for a stand alone project but should be acceptable when Sishen South product is blended with SEP product.

The proposed grade split between lump and fine ore of 64.0%Fe and 63.5%Fe, respectively, is considered to be conservative. In the event of this not materialising however, the following compensating steps may be necessary:

- Adjust the blend with SEP product at Saldanha to achieve the required combined specification;
- Decrease the specification of the fine ore; and
- Increase feed grade with associated impact on resource/reserve.

A review of the LoM mined grade shows an average of 64.3%Fe with occasional spikes above 64.5%Fe on an annual basis. This represents an opportunity to better utilise the resource through alternative scheduling and an improved stockpiling strategy.

The proposed Sishen South flowsheet and process design criteria largely recognise the testwork findings. The Sishen South plant will consist of five integral processing steps, namely primary crushing, secondary crushing, stockpiling, screening and tertiary crushing, product handling stockyard and load out.

RoM ore is dumped via trucks into the primary gyratory crusher. The crushed product reports to a rock box from which it is withdrawn using an apron feeder. Primary crushed product is screened, with screen oversize being fed to the secondary cone crusher. Screen undersize is conveyed directly to the secondary screening plant. Secondary crusher product is conveyed to the intermediate buffer stockpile, from which it is withdrawn for tertiary cone crushing. Tertiary crushing operates in closed circuit with the secondary screening plant. The +25mm fraction is recycled to tertiary crushing whilst the –25+8mm fraction and the –8mm fraction report to the lump and fine product stockpiles, respectively. The products are reclaimed with a single bucket wheel reclaim system and conveyed to a single load out station.

It is accepted that the plant which has been designed for the direct shipping ore operation will not be able to process wet or high clay containing ore. The following steps have been taken to lessen the impact of such occurrences:

- Low design utilisation of 64% (5,637 production hours per annum); and
- Over design of screen size by 25%.

Annual plant throughput is planned at 3.0Mtpa RoM at a planned yield of 99.8% to produce 3.0Mtpa of product.

6.2.3 Thabazimbi Mine

The Thabazimbi process plant originally consisted of a washing and screening operation which was constructed in 1948. Since then, the plant has periodically been refurbished and upgraded. The dense medium drum section was added in 1954 and the cyclone section in 1970.

Generally, only material with an iron content of greater than 60% Fe is fed to the plant. When product quality allows however, a small proportion of lower grade ore at levels as low as 55% Fe is introduced but by-passed around the beneficiation plant. This has significant benefits in the better utilisation of the resource.

Open pit ore is crushed in one of two primary gyratory crushers ahead of stockpiling and conveying to the plant. Ore is sized into various fractions by further crushing, washing and screening. Ore in the size ranges –32+18mm and –18+8mm are beneficiated in static bath DM drums to yield a 62.5%Fe lumpy product. Ore in the size range –8+1mm is beneficiated in DM cyclones to yield a 63.0%Fe fine product. The –1mm fraction is de-watered and added to the fine product. Dense medium rejects are stored on waste dumps whilst slimes are stored in tailings dams.

Plant throughput is largely a function of plant utilisation but also of the feed size distribution. This latter aspect is important, as outside certain limits of the ratio of fine ore to lump ore, one or other of the plant sections will constrain overall throughput. With the present plant, the optimal proportion of fines is estimated at approximately 50%, under which circumstances the plant has a capacity of approximately 3Mtpa RoM feed.

Considering its age, the plant appears to be in a fair condition, both mechanically and structurally. A programme of structural refurbishment is in place in poorer areas of the plant. Thabazimbi operate a computerised maintenance management system although this has not yet been fully implemented on the plant. It is considered however, that with ongoing preventative maintenance the plant can be expected to operate for the period scheduled in the LoM Plan without the need for major refurbishment, other than that already identified.

Key historical processing statistics for the Thabazimbi Mine Process Facility are summarised in Table 6.5.

Table 6.5 Thabazimbi Mine: Plant Operating Statistics

Description	Units	2001(F)	2002(F)	2003(H2)	2004(C)	2005(C)	2006(C)
Headfeed	**(Mt)**	2.7	2.8	1.5	3.1	3.1	3.0
Product	**(Mt)**	2.4	2.4	1.3	2.6	2.5	2.5
Proportion Fine	**(%)**	46	46	46	48	53	56
Plant Yield	**(%)**	89	87	85	81	83	85

(F) Financial Year ended 30 June.

(H2) Six months ended 31 December due to the change of Financial Year.

(C) Calendar Year ended 31 December.

The LoM Plan assumes an average RoM throughput of approximately 3Mtpa. It is seen that this is accompanied by a sharp increase in the proportion of fines to 57% on average, which exceeds the indicated optimum of 50%. This is largely due to an increase in the proportion of Kwaggashoek ore, which is a lot finer than the other Thabazimbi orebodies. Fortuitously, the fine fraction of the Kwaggashoek ore is relatively clean and can in part be accepted on product beds without beneficiation. Currently approximately half of the Kwaggashoek ore is being screened at the pit. Screen undersize is transported directly to the fine blending beds, whilst screen oversize is combined with the balance of the ore and beneficiated as normal. It is proposed that this practice continue for the remaining LoM and that the plant not be upgraded to handle an increased proportion of fines. There may however, be the need to upgrade the slimes handling capacity to cater for future ores, for which the installation of a high rate thickener is proposed.

The LoM yield is projected at an average of approximately 84%, which is in line with recent achievements and considered to be sustainable on the projected feed ore.

Average product quality achieved in recent years is summarised in Table 6.6.

Table 6.6 Thabazimbi Mine: Plant Product Qualities

Description	Units	Fe	SiO_2	Al_2O_3	K_2O	P
Lump DR Ore						
2004 – 2005 Average	**(%)**	62.97%	6.45%	0.66%	0.09%	0.028%
Current Specification	**(%)**	62.50%	6.50%	1.40%	0.16%	0.035%
Fine Ore						
2004 – 2005 Average	**(%)**	63.71%	5.05%	0.92%	0.14%	0.030%
Current Specification	**(%)**	63.00%	6.00%	1.40%	0.16%	0.040%

It is seen that product quality has generally been well within specification.

6.3 Coal

6.3.1 Grootegeluk Mine

Grootegeluk Mine commenced operation in 1980 to produce a blend coking coal for use in the steelworks coke ovens. Thermal coal was produced in the washing plant (“GG1”) as a co-product and stored for use at Matimba Power Station.

A second washing plant (“GG2”) was constructed and commissioned in 1986 to produce power station thermal coal to augment the throughput of the first plant so as to produce the full tonnage of thermal coal for the Matimba Power Station.

The design of the large capacity plants (GG1 and GG2) is conventional with a primary wash of 150 +0.5mm coal to remove a discard fraction followed by a secondary wash on GG1 plant of –25mm coal, the large coal (–150 +25mm) being crushed to –25mm before the secondary wash. The newer washing plant GG2 is scheduled for a rebuild to increase the RoM feed and to include the secondary wash of destoned –150mm coal which is crushed to –25mm before rewashing. The fine coal is recovered on de-watering screens for the –0.5 +0.1mm fraction and on horizontal vacuum filters (0.1 – 0mm). The coal reports to the power station thermal coal fraction.

Grootegeluk plans to enlarge the second wash plant (GG2), to include two stage washing to produce increased tonnages of blend coking coal. Increased feed tonnage to the plant (GG2 and GG6) will maintain the tonnage of thermal coal, together with increased tonnage of blend coking coal and the associated discard material.

Small plants were constructed later including a crushing plant to produce raw coal for the Power Station ("GG3") and washing plants which produce low ash metallurgical and steam coal ("GG4 and GG5") for sale to local and export markets. The small washing plants (GG4 and GG5) produce 10% and 15% ash coal which has low phosphorous content making it suitable for either steam raising of specialised metallurgical applications such as pulverized coal injection of coal in blast furnaces or in the iron reduction plants. The plants produce a +0.5mm co-product of power station smalls and fine coal that reports to the power station smalls.

The plant operations are very largely influenced by the demand for power station thermal coal. The fraction of this coal within the RoM coal is set by the coal characteristics and the production of higher grade co products. This will be maximised by the construction of the GG6 project which extends the second large coal washing plant (GG2) to produce blend coking coal and power station coal, giving higher overall revenue/t RoM produced.

Further expansion is under investigation together with Eskom on the extension of the Power Station by approximately 60%, which will increase the production of the associated high-grade co-products by a similar amount. The operational costs of R3.20/t RoM for coal treatment are very low showing an efficient coal washing and handling operation.

The fundamental requirements of a coal washing plant are to recover the products in the raw coal and produce a discard that may be safely dumped with no risk of long-term pollution or combustion.

Ideally the product yields should correspond with the geological yields but this is approached rather than attained. The geological yield is measured by crushing a 102mm core from a drill rig to –25mm and then separating the different products by heavy liquids to give the geological yield of products and a discard having minimum energy and minimal tendency to combust after dumping.

The geological factor shows a loss of blend coking coal to power station thermal coal, resulting in lower mine revenue. This result gives a target for the plant designer and operator rather than a measure of operating performance. It can be expected that this factor may be improved over time from incremental plant improvement. Any changes that are proposed will consider improving this factor as part of any design work.

A large plant such as Grootegeluk is subject to continuous review and update in order to minimise operating cost and to maximise product revenue from the mine as a whole. New technologies improve recovery of different size fractions and developments within the mine change requirements for each part of the plant operation.

Continuous review is required on the plant feed top size so as to maximise revenue. The present top size of 150mm is large in the context of typical plant design and the coal characteristics where the coal in the RoM adheres well with shales present in narrow plates within the coal Seams. Crushing to a smaller size may increase yield but at the cost of producing fines which results in the loss of high quality coal to the unwashed fines which report to the power station feed product. Part of this review should encompass examination of new washing techniques for the smaller sizes of coal of less than 2mm and the primary wash recrush using a ring roller crusher, known to produce excessive fines.

The plant discard contains no products <1mm as these are included unwashed in the power station feed. The discard is of low calorific value of 5MJ/kg, much lower than most mine discards, but it has a tendency for spontaneous combustion in the dump. This results in a requirement for soil or other fine inert material to seal the discard in the dump.

The discard CV is a very good measure of plant performance. The above value shows little loss of energy in the discard.

A basic audit of the mechanical condition of the coal handling infrastructure and coal washing plants was undertaken in order to determine whether the maintenance carried out by the plant is sufficient such that no unexpected costs arise due to neglect or under funding of plant maintenance and upkeep.

The plant maintenance system was examined so as to understand the present and predicted maintenance costs for surface equipment. The audit was validated by inspecting plant mechanical availability records to establish that the plants are in a fully ready condition to run to the operating schedules.

The plant inspection further established the general condition of surface equipment including washing plant, mechanical platework, steel and control relative to a well maintained plant with a scheduled life of at least 10 years. The rotating machinery on the plant is well-guarded. Refurbishment of grating and sheeting around the plant includes a total major refurbishment of the original plant (GG1) this year.

Spillage from plant conveyors is well controlled and any accommodation of spillage collected and returned to the product stockpiles. Conveyor walkways are in good condition.

HDPE piping has been installed in the older as well as the new sections of the plant and spare key sections are available on site as replacements should these pipes fail.

The steel piping around the plant is in good condition indicating that the clarified water is not corrosive, as is sometimes the case on other plants. The product stacker/reclaimers were operating during the visit. Scheduled overhauls are included in the preventative maintenance plan. The plant maintenance system in use is based on preventative maintenance to ensure maximum availability.

Each unit selected as a significant piece of equipment is individually identified and a scheduled preventative maintenance job card generated to ensure maintenance is carried out to manufacturers recommendation, or based on operating experience. The maintenance is scheduled during maintenance periods and a condition report generated on the condition of the equipment and wear resistant liners that are inspected as part of the maintenance procedure.

The cost of plant maintenance, part of the plant operating cost, is favourable because of the large size of the equipment in the modules, and the module throughput.

The Grootegeluk Plant is well maintained operating at high availabilities and at rated feed tonnages to the plant. The availability, measured as time on coal, is 90% of the available time after taking off the Sundays and scheduled maintenance periods.

The plant yields are satisfactory, as shown by the small energy losses to discard. There are losses of coking coal to lower value power station coal, as shown by the geological factors for the plant. These however are complex, but should be subject to continuous design review so as to improve the mine revenue. The first design opportunity occurs during the extension to the GG3 plant (GG6 Project) to produce coking coal. There are no potential increases in maintenance costs during the life of the plant on a cost/t RoM basis.

The requirement for covering of soil or inert sand on the dump to control the oxidation reactions of the discard, which result in spontaneous combustion is an ongoing matter. It is matter which requires careful control, as has occurred in the past. The costs associated with this, should not be significantly more than those incurred at present. Crushing of discard, using a crusher, which generates fines, may be an alternative to reduce air flows in the discard dump. This crusher may be at some stage be followed by a jig to remove liberated coal and further reduce the tendency for the dumps to burn. Other alternatives include different techniques in dumping waste, as is happening at present in dumping waste in the pit. The occasion where it is possible to use these technologies may occur for reasons outside the plant itself, such as improvement in waste dumping when the mine overall size is increased to meet future plans, which raises the coal handling throughput.

6.3.2 Leeuwpan Mine

The plant consists of a conventional dense medium drum and dense medium cyclone plant with spirals to produce saleable products. A new jig has been installed to destone the coal from the upper part of the Seam to produce a power station feed coal or a feed to the existing plant.

The plant produces a large number of products in different size ranges and grades resulting in a large medium section relative to the plant throughput.

The plant is extensively lined with ceramics and utilises HDPE medium piping throughout in order to minimise maintenance and the RoM coal throughput demonstrates a high availability.

The jig plant was not fully commissioned or integrated into the operation at the time of the site visit.

The coal Seam consists of an upper and lower section. The upper Seam is low grade with 60% discard when producing the 15% ash products sold at present. A power station coal with 25% ash may be produced directly in the jig, which is used to remove waste carbonaceous shales. The lower part of the Seam is washed to produce a 15% ash product with little discard.

The coal is crushed to –80mm in open circuit before washing and to –50mm as product coal to the power station or to low ash products, there being no demand for +50mm coal from the present market. Roll crushers are used to minimise fines production. The geological yield factor of 86.5% product to feed coal ratio indicates that the losses to discard is significant even allowing for the slimes that are filtered and dumped in the pit. The new supply facility for power station coal includes the crushing of feed coal to 50mm from the jig product coal that may be rewashed in the drum/cyclone plant, or delivered directly to stockpile for railage to the power station. This results in the drum product crusher having no feed when the jig is part of the coal treatment circuit because the plant feed coal is –50mm.

A new crusher or relocating the drum product crusher to crush the main plant feed coal to –50mm coal may improve the geological factor.

The RoM coal is trucked to a loading point and dumped over a fixed grizzley into a receiving bin. The coal is discharged on an apron feeder and crushed in a double roll feeder to –180mm and conveyed to a double deck screen, the oversize coal is crushed to –80mm nominal size (–100mm maximum). The sized coal is conveyed to the plant feed silo. A magnet removes any tramp material.

The coal is discharged from the silo onto each of two plant feed belts and the tonnage controlled by a belt weigher on the plant feed conveyors.

The silo is emptied between batches of coal as the design is not mass flow with no cut-off possible between batches of different grade.

The feed coal is discharged into a head chute where it is pulped with water and screened on a 2.4m wide double deck inclined screen. The top deck has a 25mm to 30mm cut point and the oversize reports by belt conveyor to the drum washer. A single drum washes oversize coal from the two modules.

The coal is mixed with magnetite medium in the launder feeding the drum and the coal is separated into product (floats) and discard (sinks). The product and discard are drained and rinsed on dedicated screens and report to specific conveyors. The product is crushed to –50mm as there is no market for larger coal and then screened at 25mm with the –25mm reporting to the peas product from the cyclone module. The large coal medium circuit and dilute medium circuit are conventional with headboxes to set the flows and a single magnetic separator recovers magnetite from the dilute medium slurry. The separating density of 1.5 – 1.6 g/ml results in low loading of the magnetic separator.

The sizing screen coal undersize coal reports to the lower deck of the plant feed sizing screen.

Sized small coal of –25+6mm is rinsed and dewatered on the lower deck of the sizing screen and discharges into a launder where it is mixed with magnetite medium. The slurry is pumped to an 800mm diameter cyclone where the floats and sinks are produced, and the medium removed on a 3mm wide split drain panel and screen to produce a peas product and discard. The dilute medium circuit is similar to the drum dilute medium circuit.

The –6mm undersized coal from the plant feed sizing screen is collected as slurry and the –1mm coal and water removed through a fixed drain panel and a 2.4m wide screen. The coal is rinsed on the screen and discharged into a launder where it is mixed with magnetite medium. The screen was overloaded with water with no discernable de-watering section of the screen before discharge.

The medium/coal slurry is pumped to a 710mm diameter cyclone and a product/discard panel and 3m wide split screen drains medium from the coal product and discard streams. The product reports to a specific conveyor and sampled while discard reports to the general discard conveyor, together with discard from the drum plant and the peas plant.

The fine coal slurry collected from the cyclone feed screen and panel is pumped to a desliming cyclone and the –1mm +100 micron material spiraled in two stages to produce a product and discard.

These are cycloned to produce a de-watering screen feed and the product de-watered and conveyed to a product stockpile. The de-watered discard reports to the discard conveyor.

Discard is conveyed to the discard bin and then trucked to the pit where it is dumped as part of the fill.

Water from the fine coal plant reports to three 22m diameter thickeners where it is clarified and water recirculated. The thickener underflow is pumped out at high solids content and filtered on plate and frame filters.

The water is recovered and the filters periodically discharged onto a stockpile from where it is dumped into the pit as fill so that there is no slimes lagoon with the associated water collection problems.

The filter cake could be mixed with the power station feed coal as at Grootegeluk when the jig is commissioned and the coal grade allows this addition.

The coal yields are 40% for top coal and 90% for bottom coal. The plant produces 145ktpm from 220ktpm RoM coal giving a nominal yield of 66%. The product coal specification is 15% but there is a penalty in producing a 14% ash small coal (–25mm), especially from the lower Seam. Any discard reports predominantly to the drum as large discard.

The coal is washed as –100mm coal but all coal is sold as –50mm coal. Any coal produced in the new jig is crushed to –50mm in the jig product crusher.

Quality control is achieved through accurate sampling of the coal products using belt samplers. The different sections of the coal Seam are washed separately to optimise yield. The laboratory is contracted out to produce sample results to schedule. There have been no recorded customer complaints on the coal product specification.

The plant operating costs and throughput will change with the addition of the new jig. The existing plant and infrastructure is in good condition with no build-up of maintenance tasks so no change is expected in this area of the plant. The jig maintenance and operating costs are recovered from revenue generated from sales of power station coal and increased throughput of coal to the drum/cyclone plant when operating on the low yield upper Seam because of the removal of excessive discard in the jig.

A fine coal separation plant is under consideration due to the high value of low ash coal from this mine. The mining, laboratory and the product loading and plant cleaning is outsourced. The supervisory staff and control room operators are directly employed.

The laboratory is contracted out to CMT. The equipment was adequate for analysis and sales quality control. There is some dust issuing from the laboratory mill when the coal was crushed before splitting down and sampling.

The coal washing plant was running at the time of the visit through from the coal receiving bunker to outloading of products. The rotating machinery on the plant is well-guarded. Walkways, grating and handrailing were in acceptable condition and all areas were accessible with no buildup of spillage. The plant staff use adequate personal protective equipment.

A basic audit of the operation and physical condition of the coal washing plant and associated conveyors was undertaken in order to determine whether the condition and maintenance carried out is sufficient that no unexpected costs arise due to neglect or under funding of plant maintenance and upkeep.

A visual inspection of plant and equipment was conducted to:

- Examine operational efficiencies;
- Audit production;
- Determine quality standards.

The steelwork is in good condition with no visible corrosion on steel and no evidence of changes to steelwork since the plant was built.

No undue vibration in the structure was noticed. The screens are well supported and there is no evidence of stress fractures of steel members. There is no evidence of corrosion caused by the clarified water in the plant and to wash the floors.

The small coal platework is ceramic lined and in good condition. There is no evidence of patching of chutes and underpans. All conveyors are in good condition with no visible damage to belts or damaged conveyor idlers. The belt scrapers operate with no tell-tale spillage at return idlers. There is no damage to conveyor trestles from mobile equipment indicating good plant design and layout.

HDPE piping is used extensively throughout the plant. This is good practice, carried low maintenance risk and maintenance friendly. There was one leak on the plant which was under repair during the visit.

Electrical equipment appears to be in good condition and no electrical motors were seen under spillage which reduces cooling efficiency.

The control is by PLC and Scada with all required information on well-designed screens.

The planned preventative maintenance is similar to the other Kumba operations and based on examining major equipment. Weekly maintenance schedule allows 24h/wk for maintenance using mine employed tradesman (helpers are contractors).

The equipment selected such as screens, ceramic lined cyclones and Warman pumps assists maintenance through proved reliability.

6.3.3 Tshikondeni Mine

A technical audit of the coal preparation plant was carried out on 11 July 2005 and 12 July 2005 to determine if the plant was operating at the required tonnage and efficiencies.

During the visits discussions were held with the Plant Manager, Plant Controller and operating personnel. Inspections were made of the primary crushing screening and stockpile areas, the secondary crushing and screening plant, the dense medium cyclone plants, the froth flotation plants spiral plants and thickener, product stockpiles out-loading, waste dump and slimes dams and the laboratory. Operating procedures, reporting systems and accounting systems were discussed.

A previous audit of the plant was carried out by DMP Consulting in November 2001 and the information from that audit was used for comparison purposes.

For the period April, May, June 2005 the plant treated an average of 65,352tpm and produced an average of 34,273tpm of sales versus a budget of 32,500tpm at a 49.7% yield.

Table 6.7 Tshikondeni Mine: Plant Product Specification

Quality	Values	
Ash content (air dried)	Average 12%	Range 11.7 – 12.3%
Total moisture content	Maximum 10%	
Size	13mm x 0	
Roga	+ 80	

The plant is capable of producing the budgeted sales per month at the current plant yield which varies from 48% – 54%.

RoM coal is delivered by lorry to the RoM tip bin. Coal is conveyed from the tipping bin to a raw coal scalping screen where the 200mm oversize is discharged to ground for disposal by front end loader and lorry. The –200mm raw coal is delivered to a 3,000t capacity raw coal bin then conveyed to a screening and crushing plant where the coal is reduced to a –13mm nominal top size, then conveyed to the plant feed bins.

Raw coal from the plant feed bins is conveyed by two plant feed conveyors to the coal washing plant. The coal preparation plant at Tshikondeni was extended in 1997 and comprises of a twin module dense medium cyclone plant, a twin module froth flotation plant, a single module thickening plant and a product storage and out-loading plant.

Desliming screens in the plant fitted with 1.6mm slot aperture decks size the coal at 1.4mm. The +1.4mm raw coal is separated in 2 x 710mm diameter dense medium cyclones and the –1.4mm raw coal in 2 banks of 6 x 6m^3 froth flotation cells. Double stage spiral concentrators were added later to treat the flotation tailings and recover any misplaced fine coal. In order to meet the require product specification the individual plant products are controlled as shown in Table 6.8.

Table 6.8 Thshikondeni Mine: Plant Products

DM Cyclone Product	> 13.5% ash
Froth Flotation Product	> 10.0% ash
Spiral Product	> 12.0% ash (currently not working)
Final Product Blend	> 12.0% ash ±0.5%

The cyclone product quality is normally adjusted to control final product quality.

The plant is currently fed at 150t/h which is well below its design capacity. The operating philosophy is to maximise yield at the expense of tonnage through-put. Batch washing and operating on extended hours at a lower feed rate is reported to have increased the yield by 4% – 6%.

For the Tshikondeni plant configuration, a plant discount factor of 0.925 can be applied to the theoretical yields for predicting expected plant yields. In practice a plant does not always operate at maximum efficiency due to breakdowns, stops and starts, control problems, etc. and a further factor of 0.975 should be considered giving an overall plant discount factor on yield of 0.902. These factors vary with washing yield and plant configuration but are a suitable guide for the current operation.

Last year the plant yield averaged 54.0% compared to the typical average of 56.1%. The difference can be attributed to the fact that the typical washabilty data is based on a specific area.

Typical washability data from each of the three shafts compare very favourably in terms of relative density and yield when producing a 12% ash product. However the variability in the Seams has led the mine to operate on a batch wash system as this has proven over time to give a higher yield. A homogenising stockpile could be installed to control the variability of the feed but this is deemed impractical on cost with respect to the short life of mine.

Run of mine coal from the incline shaft is delivered by conveyor to a bi-furcated chute. From the bifurcated chute it is conveyed to either the “A” frame RoM ground stockpile where it is recovered by vibrating feeders and front end loader and returned to the circuit or conveyed to the RoM tip bin.

The RoM tip bin receives coal by bottom discharge lorry from the operating shafts. Coal is withdrawn from the tip bin by belt feeder and conveyed to a raw coal inclined scalping screen fitted with a 200mm spacing bar deck. A belt magnet removes tramp iron and discharges to ground for later removal. The scalping screen feed conveyor is fitted with a single idler weightometer and belt speed indicator.

Oversize from the scalping screen, which is mainly waste, is discharged by chute to ground level where it is later removed by front end loader into lorries. The oversize currently averages 2.0%. Undersize from the scalping screen is conveyed to a twin outlet concrete raw coal silo of 3,000t capacity.

6.3.4 Arnot Colliery

The plant supplies Arnot Power Station at a nominal rate of 5,000ktpa. There is a large stockpile between the plant and power station such that there is no production link between the plant and power station. There is little storage between the mining operation and the stockpile resulting in a close link between the mining operation and the plant. This link is carefully monitored to ensure that the mining operation is never affected by the plant. The overall conveyor capacity is over 1,200t/h for all overland conveyors to handle the product from a continuous mining machine.

The washing plant capacity is large due to the size of the plant and the large coal size –300 + 25mm coal handled. The screen capacity allows operation at up to 500t/h, equivalent to 1,200t/h RoM coal. The washing plant is operated only to control the abrasion index of the coal, due to rock reporting to the coal underground. The abrasion index is a progressive index over a month so the feedback from coal samples allows the tonnage that is washed to be pre-planned and scheduled. The typical operating time is 10% – 20% of the conveyor operating time. The raw coal crusher sizing screens may be checked and maintained when the washing plant is operating.

RoM coal is crushed underground at each shaft to –300mm the feed size to the ring roller crusher and washer. The coal is collected underground in a surge bin and is conveyed to the shaft bin of total storage capacity of 400t. The coal is conveyed to the primary screening plant where it is fed to three screens which screen out the oversized coal which is conveyed to a ring roller crusher with a 40mm aperture screen. The crushed coal and the screened fines are collected in the fines bin and then conveyed to the power station stockpile. The primary screening plant has three further screens which screen out oversize coal which is conveyed to the secondary bin. The coal is screened at 25mm. The fine coal is conveyed to the power station stockpile feed conveyor or to a variable angle screen where the –25 +15mm coal is screened out and may be conveyed to the washing plant feed silo together with the –300 +25mm coal.

The crushed power station coal from either the screening plant or the DMS plant is conveyed to the power station.

The dense medium separation plant is operated intermittently to remove large discard so that the abrasive index of the coal is to specification. There is a large inertia in the supply chain due to the power station stockpile and the sample collection specification so that the cumulative index for each month is the key factor in determining the operational time for the plant. The large coal either +25mm or +15mm is fed to the wet feed sizing screen where –12mm coal is removed to a de-watering screen. The screen underflow slurry is pumped to a cyclone and the cyclone spigot solids recycled to the de-watering screen and to the power station coal.

The large coal is fed to the Drewboy washer where the product floats in a magnetic medium of 1,7RD. It is removed from the bath by paddles and reports to the product drain and rinse screen where medium is drained from the coal and the coal rinsed with water to produce a product coal and a dilute medium. The coal is returned to the product conveyor and reports to the power station colliery stockpile.

The discard sinks in the Drewboy where it is removed and drained in the inclined discard wheel and discharged onto the discard conveyor and trucked from the discard bin to the Arnot Colliery opencast operation as fill. The fines from the washing plant are collected on the dewatering screen with the –12mm undersize coal from the feed screen and a –12mm +100 micron de-watered coal is discharged onto the plant product conveyor. The slimes from the plant are collected in the thickener a 10m, diameter high rate unit and periodically filtered on a horizontal belt filter to produce a filter cake that is added to the product coal.

The operating costs are low for conveying coal at R2/t coal handled on surface and R6/t of RoM coal for washing because of the low operating time for the plant.

6.3.5 Matla Colliery

The Matla Colliery surface coal handling operation is a conveying, screening and crushing operation to produce a –40mm crushed coal which is the feed coal to Matla Power Station. The only beneficiation and control is achieved by a waste picking operation at the crushing plant and at the shafts before the overland conveyor. The No. 5 Seam coal, if mined, will be washed in an off-site plant. No plans were presented.

The sizing and crushing plant consists of two streams each operated at up to 1,800t/h RoM coal. The coal is sized on 40mm mesh screens and the undersized coal bypasses the crushers. The oversized coal passes over a picking belt with two stations where large sandstone waste is removed at 0,2 – 0,4% of the feed rate. The rock is removed in order to control the abrasive index of the coal (approximately 150 – 200t/day removed).

The coal is crushed in one ring roller crusher/stream with a standby crusher and the crushed coal reports to the small screened coal. The coal is conveyed to the stockpile where it is stacked by trucks. The capacity of the conveyors that feed the power station is 1,800t/h per stream. The power station capacity is rated at 15Mtpa requiring an operating time of 347h/month for each crushing plant at 1800t/h. This is 63% of the time on coal out of a total available time of 580h/month. There is a 500kt capacity stockpile between the crushing plant and the power station making the operation of the crushing plant independent of the power station. Raw coal silos at the feed to the crushing plant from the shaft overhead conveyors ensure that the availability of the crushing plant is easily achieved.

The RoM coal is delivered to a shaft silo at each shaft. It is crushed to –300mm and waste picked at the shafts before the coal is conveyed by overland conveyors to the crushing plant. The coal is dumped into silos to provide storage between the shafts and the crushing plant.

The coal is discharged from the silo by four vibrating feeders feeding a single conveyor belt of 1200mm width. Some –40mm coal is screened out on selected feeders and reports to a fines belt below the screen feed belt.

The coal reports to two sets of two 2,2m x 6m Dabmar double deck sizing screens in series where the –40mm coal is removed by the lower deck. The oversize coal reports to the picking belt where 150 – 200t/day stone is removed from 35,000 t/day of coal on a typical day.

The oversize coal is then charged into a ring roller crusher to reduce it to –40mm nominal size and the crusher product is discharged onto the product belt. Dust from the crusher is collected in a scrubber and the dust is dumped as a dense slurry on the coal conveyors.

The crushed coal from each plant is conveyed to the stockpile area and stacked using mobile equipment. The coal can be conveyed directly to the power station or recovered from the stockpile. The coal is then screened over a guard screen, one for each of two streams and then conveyed to the power station. The oversize coal is recycled to the feed to the sizing and screening plant. A new crusher is under construction to crush oversized coal.

The coal reports without any treatment to the power station. Some rock from the roof reports to the product during changes of the long and short wall operations. This is removed by waste picking at the shafts and at the screening and crushing plant. This is done to control the abrasive index of the coal as delivered to the power station. The coal is at present within specification.

The coal is crushed to –40mm and a guard screen ensures that no oversize coal reports to the power station.

The coal is sold on contained energy so is weighed on calibrated conveyor scales and automatically sampled. Because of the large stockpile there is a large momentum on quality change so that short-term quality control is not required.

The change from short wall mining to the use of continuous miners is expected to reduce contamination at the coalface.

The plant throughput is planned to increase from 11 – 14Mtpa. This will require upgraded control to increase plant availability to the rated capacity of 63% on line, based on a three-shift operation.

The operating costs will not change from the present levels and the enhanced control system can readily be financed by the increased coal tonnage.

6.3.6 New Clydesdale Colliery

The total Coal Preparation plant configuration is capable of treating 600tph of RoM and is rated at a maximum capacity of 240kt RoM per month or 2,880kt RoM per year.

The washing plant can be seen as comprising four distinct modules or sections.

The first of these is the screening plant, which crushes and screens RoM coal down to –40mm in size from the received 150mm raw product. At present No. 4 Seam lower seam coal is received from the Diesel Power opencast operation (65ktpm). The coal is reduced at the Diesel Power site to –150mm before being trucked approx 6,5km to the plant tip area.

Size reduction in the screening plant is done via 2 Jeffrey double roll crushers of which the primary crusher set at 75mm is in “open circuit” and the secondary crusher set at 40mm is in “closed circuit.” This screening plant can handle 300tph and is fed via a vibrating feeder under the stockpile, which is fed by the trucks bringing across this No. 4 Seam coal. The second of the modules is the HMS section or inland plant, which treats 100tph of the screened and crushed product. The washing is done in a 600mm diameter Multotec mild steel DMS cyclone at a density of 1,680 producing a product and a discard. Currently this product is sized for domestic customers; three separate products are made:

- Small nuts 40mm x 28mm A grade (27,85MJ/kg) 4,000tpm.
- Peas 28mm x 10mm A grade (27,85MJ/kg) 11,500tpm.
- Duff 10mm x 0,5mm A grade (27,85MJ/kg) on demand.

Currently all the duff is reporting to the Export product line.

The budget for this plant is 180kt of sales (15.5ktpm). Yield in this plant is currently at 63% of the coal fed into it. Sufficient area is available to stockpile large quantities of washed product.

The third of the modules is the PCI or export plant (Module A), which treats the other 200tph of the screened and crushed product. This plant was erected by Bateman in 1997/1998 and was part of the double stage LAC plant. The washing is done in a 810mm diameter Multotec ceramic lined DMS cyclone at a density of 1,680 producing a product and a discard. About 60% of this product is returned to the HMS product for screening into peas, small nuts and duff. The remaining 40% (sizing 40mm x 40mm) is placed on the Export coal stockpile. Yield in this plant is currently at 63% of the coal fed into it. There are two export stockpiles, each capable of holding 8kt each. Should FEL's be utilised stockpiling capacity can be increased to at least 150kt.

The final module is a stand alone Module B component which produces both domestic and inland washed coal. At present feed for this plant comprises 75ktpm of No. 2 Seam coal mined underground. This coal is sent to the module B plant via a system of surface conveyor belts. Some spare Mafube coal that is trucked across is also being fed into this plant Expected tonnages of this RoM until the end of 2005 is expected to be around 20ktpm. This plant was commissioned in early 2003 after it was extended and modified from the stagnant LAC portion of the existing Bateman plant. A crusher, thickener and a feed in section were added to make it operational. This plant has a feed capacity of 300tph of RoM feed. Feed is as mentioned from the overland conveyor system. The crushing is done in a Hazemag impact crusher (single stage from 150mm to 40mm – open circuit). The washing is done in two 810mm diameter Multotec ceramic lined DMS cyclone at a density of 1,680m^3 producing a product and a discard. The washed product is fed over a sizing Dabmar screen to produce:

- Small nuts 40mm x 28mm A grade (27,85MJ/kg).
- Peas 28mm x 10mm A grade (27,85MJ/kg).

This tonnage assists the budget of Inland sales to be made. The remaining tonnage reports to the Export stockpiles. Yield in this plant is currently at 62% of the coal fed into it.

All export trains are loaded on the Colliery via FEL's onto a conveyor belt transecting the two export stockpiles. Currently 3 Liebherr FEL machines are used for these trains and the average time taken to load one of these trains is 6.8 hours. This conveyor feeds a silo situated directly over the rail line and weighbridge. Wagons are loaded (58t each) in 100 truck blocktrains (5.8kt). The Colliery locomotives take these wagons to the Bezuidenhoutsrus Siding approx 5,5km away, which is linked to the main RBCT rail line.

Current export budget is 830ktpa of entitlement coal plus an additional 240kt BEE component totalling 1,070kt. This equates to some 16 blocktrains per calendar month.

RBCT is 480 kilometres from the Colliery. The weighbridge is a Klerkscale accredited (SABS) module that is frequently checked. Current costs in the coal preparation plant are ZAR13.78/t of RoM washed coal, of which ZAR2.20/t is for water and electricity.

Plant budget for 2005 staff wise is 74 persons; 3 x 9-hour shifts are worked during the week (45hr week) and each of the shifts comprise 15 employees. The rest of the number is made up of drivers, weighbridge clerks and laboratory assistants.

Magnetite used is medium grade type from Martin and Robson and is received in bulk from their depot at Broodsynersplaas, which is close to the Colliery (about 15km away). Magnetite consumption is fairly high at 2kg/t although a figure of 1kg/t is being worked towards. This has been affected by dirty water. Flocculant consumption is 18g to 20g per RoM feed ton. Flocculant is the granular type received in 25kg bags from Sudchemie. Water usage is 1,5m^3 per RoM feed ton.

6.3.7 North Block Complex

Glisa Plants: The Eskom and export coal crushing plants were operating well, producing crushed coal in the correct size grades. The Eskom coal and high grade coal screening and crushing plants each have a capacity of over 300tph. The condition of the equipment is good through regular scheduled maintenance and it has a life which is much longer than the scheduled operation at Glisa. The crushers are suitable for the proposed new plant at Belfast and the screens are in good condition and will operate for the life of plant with normal maintenance and replacement of components.

- ***Eskom Plant*****:** The plant consists of a three stage crushing circuit in open circuit to produce an Eskom Spec Coal. The primary crusher is a feeder breaker set at 250mm followed by a primary double roll crusher which reduces the coal to 80mm nominal (100mm max). The coal is then screened at 50mm and the +50mm is crushed in a MMD double roll crusher to –50mm in open circuit. The plant produced a coal with significant oversize resulting in a rejection of coal at the power station. An examination of other plants shows that the crusher, usually a rolling ring crusher has a discharge aperture between 40 and 45mm diameter so as to meet specification.

  Models of a crusher circuit showed that a double roll crusher must be set with the gap between rolls at least 5mm less than required. The capacity of the crusher was 836t/h for a 900mm wide crusher set at 50mm between rolls so there is no capacity restraint in closing the crusher gap between the rolls. The required capacity of the plant of around 300t/h is much less than the maximum operating capacity.

- ***High Grade Coal Plant*****:** This plant capacity is 150ktpm making the required availability less than the plant capacity. At present only a day shift operation is required to screen and load the coal. Coal of D grade (24MJ/kg) is selectively mined from No. 2 Lower Seam. It is sized and sold to both the export and inland market. The feed capacity of the equipment is larger than that required for the plant feed by trucks. The expected feed capacity determined by the crushers and screens is in excess of the usual trucking capacity and is expected to be 300 – 400t/h RoM. The coal is dumped through a fixed grizzley into a bin and discharged by a vibrating feeder into 2 double roll crushers mounted one above the other to crush the coal to –80mm. The coal is conveyed to an inclined double deck screen 1.8m x 4.8m where cobbles and large nuts are screened out. The cobbles are stockpiled and loaded onto trucks or recycled to a tertiary double roll crusher and conveyor. The undersized coal is conveyed to a second double deck inclined screen of 1,8 x 4,8m where small nuts, peas and duff is screened out.

  Based on the screen and conveyor sizes the capacity of the screening plant of approximately 300t/h is in excess of the mine requirements. At the time of the visit the recycle coal crushing stream was not used due to the demand for large coal. The condition of the plant was acceptable for the operation with access and safety well addressed and all major equipment and conveyors serviced to schedule, based on the preventative maintenance plan.

- ***Strathrae Plant*****:** The Strathrae Plant is a complete washing plant with a capacity of 130ktpm. The present operating cost of R20/t will not be reduced immediately as the coal throughput is increased from 50ktpm to the scheduled 130ktpm because further refurbishment of the plant screens is required to improve plant availability.

  This is temporary as the mechanical work is limited and the steelwork, platework, crushers and conveyors are in adequate condition requiring a continuation of the present scheduled maintenance.

  The dump reclamation at Strathrae is a temporary operation to be completed in 12 months. It is operated with mobile equipment which builds a layered stockpile of crushed RoM coal and screen waste from the existing dump. This is then loaded onto trucks and transported to Eskom power stations.

  The Strathrae coal washing plant is located on the Carolina road some 23km from Carolina itself. It is utilised to wash coal from local mini pit mines and other deposits including possibly washing Matla Colliery No. 5 Seam coal.

  The RoM coal is dumped into a receiving bin and discharged using a feeder breaker set to give –300mm coal. It is conveyed to a secondary crusher plant consisting of two double roll crushers and conveyed to a tertiary plant consisting of a 1,8m inclined double deck screen with the oversize fed to a Osborne double roll crusher to produce –100mm feed coal to the plant.

  The coal is conveyed to the plant feed stockpile. The coal is discharged from the stockpile and fed to the plant at a controlled rate where it is washed in large coal (–100 +25mm) and small coal (–25 +1mm) plants. The fines (–1 mm) are deslimed and the –1mm +0,1mm coal is washed on spirals. The –0.1mm slimes from the thickener is pumped underground.

  The plant capacity is determined by the product drain and rinse screens, together with the separation equipment and the feed screens. A typical size distribution is taken, based on opencast and underground coal to size the equipment so that any coal feed may be washed in the plant.

  The coal yields are set for approximately 70% based on the product screen split. This will become important if larger cyclones are installed. The coal is washed as –100mm coal (–80mm nominal) and there are sales for all size fractions. The –100 micron coal is pumped to underground voids in worked out sections of the mine.

6.3.8 Sintel Char Plant

The plant was designed by B Morgan and Associates and is based on a well-tried and tested concept of a continuous vertical retort used extensively in the past in the UK and elsewhere (Johannesburg Gasworks) to produce towns gas and non-metallurgical coke for domestic use. The process mass flow diagram given in the report shows a conventional flow diagram based on well-established technology. No calculations were available to check on the mass flow figures given.

The heart of the process is the retort itself. B Morgan and Associates designed and built a plant based on their technology for United Carbon Producers, known as the Prolon Plant situated near Ogies on the Witbank Coalfield. This plant produced char, but was subsequently closed for business reasons. The engineering and operating experience gained from this plant together with technical input from Kumba was used as a base to design a new char retort for the Sintel Project, designated a Modern Char Retort. The concept of this new retort is well-described in the feasibility study report and the reasons for the change in design configuration are discussed in principle. In effect the new retort is based on engineering out the perceived deficiencies in the Proton Plant to produce a modular design with two smaller retorts to give a similar throughput. The changes in design are well-reasoned and there is every reason to conclude that the Modern Char Retort will have an improved performance when compared to the original design. However there are some aspects of the plant that require further consideration particularly as no detailed drawings were available, only basic schematics.

It is most important that the mass flow through both the feed bunker, the body of retort itself and the char discharge section is as even as possible. Channels, “rat holes” or blockages in any part of the system with adversely affect plant performance. Kumba are well aware of this and stated that the well-known principles of mass flow developed by Jenike and Johanson will be applied to the design. This can only be checked once detailed drawings are available. These questions will need to be answered once design drawings are available.

Char strength is an important parameter, particularly important if briquettes are to be manufactured. No tumbler or drop shatter test results on char or briquettes were available.

Char and formed coke plants constructed in the past incorporated a “soakng pit” where the product was allowed to cool and as a result char strength increased. In the Sintel Process the char is cooled by gas before quenching in water. How this affects char strength, if at all, is not clear. It would be prudent for Kumba to consider an alternative to the quenching system proposed. This should result in a decrease in the free moisture content of the final product and could improve char strength. It would be advisable to incorporate sufficient space in the plant layout to install such a system should the final product be adversely affected by water quenching.

Tramp iron in the coal feed is a serious problem for a char plant as it could result in jamming the coal feeder mechanism. Tramp iron should not be present in beneficiated coal as it should be removed in the beneficiating process, but this is not always the case. In addition to the tramp iron magnet shown in the schematic it would seem advisable to include a metal detector which stops the feed belt when tramp iron is detected.

Correct coal sizing is stated to be an important factor in the efficient operation of the retort. No details of the screening plant were given in the schematic drawing. With no stockpile or mixing (“blending”) facility for the plant feedstock, feedstock quality control is in the hands of the Grootegeluk beneficiation plant. The screening plant therefore requires careful design and should be sited as far as possible in such a way that degradation after screening is kept to a minimum.

The plant, although forming part of the Grootegeluk area, will be operated and run as a separate company.

SRK’s understanding is that it will have its own management, operation and maintenance teams. We believe that this approach is correct as a char plant is substantially different to a standard coal washing plant operation and requires different skills, operating and maintenance strategies. Special attention needs to be taken during the staff selection process.

The decision for major maintenance to be co-ordinated with Grootegeluk and to use Grootegeluk’s facilities and major equipment is correct. Otherwise, the new plant will need to incur additional, and in this case avoidable, costs. The manning of the site operation, with the eight operators and four multi-skill millwrights, appears correct. This is in addition to the management, lab and office personnel.

Plant availability is planned at 85% average and is acceptable being within industry norms. Maintenance on this type of plants is heavy and time consuming. Ideally an allowance of 10% for planned maintenance and a further 5% to 10% for unscheduled maintenance breakdown/maintenance work should be made, therefore the 85% availability will comply with this requirement.

6.4 Heavy Minerals

6.4.1 Ticor SA

The metallurgical process at Hillendale Mine comprises three distinct sections:

- The Primary Wet Plant ("PWP") where RoM material is initially treated to produce Heavy Mineral Concentrate ("HMC") which is then fed to the Mineral Separation Plant ("MSP");
- The MSP where HMC is separated into ilmenite and non-magnetic minerals, such as zircon, rutile and leucoxene, both being further processed prior to smelting or sale; and
- The smelter where ilmenite concentrate undergoes smelting to produce titanium dioxide slag and low manganese pig iron.

The Hillendale Mine PWP, with a design throughput rate of 1,200tph has been retrofitted with a desliming circuit, in order to improve the spiral recovery efficiency. The PWP flow essentially comprises de-sliming, screening, spiral gravity separation, low intensity wet magnetic separation and slimes disposal. Finally, HMC is de-watered by hydrocyclones ahead of road transport to the MSP and the smelter at the Central Processing Complec ("CPC") at Empangeni.

The MSP operation comprises several different processes in a very complex, although fairly typical application of appropriate technology. The HMC is initially screened and separated into a magnetic fraction (ilmenite) and a non-magnetic fraction (zircon, rutile and leucoxene) using Wet High Intensity Magnetic Separation ("WHIMS") techniques. Ilmenite is further upgraded through two parallel process streams in order to remove chromite. The non-magnetic fraction separated in the WHIMS section is further processed by wet gravity and electrostatic/magnetic separation techniques to separate zircon from rutile and leucoxene.

Smelting uses electrical power to convert the ilmenite feed material into titanium dioxide slag and LMPI. The slag, after cooling, undergoes crushing, drying, grinding and screening to produce two size fractions. The recovery factors relating to the feed preparation section, the ilmenite upgrading sections, the roasting plant and the zircon/rutile dry mill as assumed by Ticor SA are considered to be reasonable. In an operation of this nature, the process represents a significant risk. Ticor SA has attempted to minimise such risk by undertaking extensive test work on a variety of ores. Primary concentration, mineral separation and smelting have been tested at pilot scale.

The No. 1 furnace at the CPC in Empangeni was commissioned during 2003. During September 2004 the No. 2 furnace was shut down in order to make improvements to it identified during the commissioning process. These improvements will in the near future also be applied to the No. 1 furnace.

***Hillendale Mine and PWP*:** RoM (around 660 to 750ktpm) is supplied to the PWP at the correct density and tonnage. The RoM is screened to discard oversized material. The undersized material is sent through a de-sliming circuit. The overflow of the de-sliming circuit reports to the thickeners. The underflow is sent through banks of spirals. The spirals split the slurry into three main streams:

– The HMC (around 50ktpm) is sent through a magnetic separator to remove magnetite. The HMC is stacked on a stockpile at the Hillendale Mine for transport to the MSP.

– The sand tails are pumped to dewatering cyclones in the mining void and deposited as backfill in the mined out areas or dune. The slimes (particle size <45µm) collected from the spirals are sent to the thickeners where flocculent is added.

– The underflow is pumped to the sub-aerial deposition dam where it is dried over a 21-day cycle.

MSP: The HMC is tipped into receiving bunkers at the MSP. From these bunkers the HMC is screened of oversize. The screened HMC is then fed into the WHIMS. Here the ilmenite is separated from the non-magnetic material. The non-magnetic material (rutile, leucoxene and zircon) contains some silica and the silica is removed from the non-magnetic material in a gravity circuit.

The crude ilmenite from the feed-preparation circuit is refined to a low chrome content ilmenite. This is done by the use of specialised drum magnets. The high chrome material is more magnetic and can therefore be separated easily from the lower magnetic material. The ilmenite is sold as final product and used as smelter feed.

The primary dry circuit is an electrostatic circuit where conductive and non-conductive materials are separated from each other rutile (conductive) and leucoxene (conductive) is separated from zircon (non-conductive)). This separation is effected by the use of high-tension roll separators and electrostatic plate separator machines. The zircon from the primary dry circuit is iron stained and is sent to the hot acid leach circuit for leaching. The rutile and leucoxene is sent to the rutile dry circuit for further separation between the rutile and leucoxene. The requirement of world markets is that the iron content in the zircon must be less than 0.06%. The zircon that enters the hot acid leach circuit contains 0.18% iron. The reason for this high percentage iron is due to a thin layer of iron oxide around the zircon particles. This layer is removed by leaching the zircon with sulphuric acid. The acid solution is neutralised using gypsum. The water is re-used in the circuit.

The leached material from the hot acid leach circuit is fed to the wet zircon circuit. Here the trash minerals still contained in the wet zircon circuit feed, namely kyanite, hornblendes and silica are removed from the zircon concentrate. The separation is done with a gravity circuit as the tails in this circuit have a lower particle density than zircon. The concentrate from the wet zircon circuit is fed to the final zircon beneficiation stage the dry zircon circuit.

In the dry zircon circuit the final refining of the zircon is done. Here a small amount of rutile and leucoxene is removed and the monazite in the zircon is separated from the zircon with induced roller magnetic separator. The zircon concentrate from this circuit is the final zircon concentrate. The product is fed to three zircon product bins. From here the zircon is either taken to the bulk terminal or to a bagging plant with road tankers.

In the dry rutile circuit, the feed from the primary dry circuit, that contains leucoxene and rutile, is split into two products. The rutile and leucoxene are separated from one another using an induced roll magnet separator. Both these products are stored in product bins that are unloaded with road tankers that take the product to the bulk terminal.

Ticor Smelter: The smelter comprises two 50 MVA DC arc furnaces. The ilmenite from the MSP is fed into the furnace together with anthracite (reductant). The mixed feed is fed via a hollow electrode. The feed can be fed as a cold material or as a hot material. Hot material is achieved by feeding the ilmenite through a pre-heater that heats the material up to a range of between 800°C and 900°C. This allows the furnace to operate at a lower energy consumption rate than when feeding cold material. The ilmenite and anthracite is fed in the correct ratio in order to obtain the correct chemistry and thermal balance within the furnace.

The main chemical reaction that takes place is:

$$FeO.TiO_2 + C = Fe \text{ (metal)} + TiO_2 \text{ (slag)} + CO \text{ (gas)}.$$

The iron and slag are tapped from the furnace from different tap holes. The carbon monoxide gas is burnt off in a flame above the furnace building. The iron is processed further at the metal processing plant. At the metal processing plant the tapped iron is treated with calcium carbide to reduce the sulphur content of the iron. Ferro-silicon and carbon are added to achieve the final customer specification for the iron. The iron is finally cast into small ingots (pigs) and stored in bunkers. From the bunkers, the pigs (Low Manganese Pig Iron) are transported by road to the bulk terminal for export. All scrap that is generated from the metal processing plant is collected and sold locally.

The titanium dioxide slag that is tapped from the furnace is allowed to cool and then processed further at the slag processing plant. At the slag processing plant, the slag is crushed, dried, screened and then classified according to coarse and fine size fractions. The coarse material becomes the final product of chloride grade slag and the fine material becomes the final product of sulphate grade slag. Both final products are exported.

6.4.2 Tiwest JV

Tiwest JV mines and processing plants: The Tiwest JV mines and processing plants operate in an integrated manner such that ilmenite originating from the mines is processed through to the finished product of titanium dioxide pigment.

HMC is produced at the primary concentrating plants (“PCP”) at Cooljarloo then transported to Chandala. Mineral products are produced in the MSP at Chandala. Ilmenite from the MSP is taken directly into the adjacent synthetic rutile plant (“SRP”). Synthetic rutile either is transported to the pigment plant at Kwinana or exported. Waste products from the processing plants at Chandala and Kwinana are returned to Cooljarloo for final disposal.

Separate PCPs operate in the two mining areas at Cooljarloo. In the Cooljarloo South mine two dredges mine in parallel to feed the PCP at 2,200tph, which floats in the dredge pond. In the Cooljarloo North mine scrapers dump ore into a slurrying hopper to feed the land-based PCP at 650tph. This PCP can be relocated when the mine advances beyond an economic pumping distance. The PCPs have similar flowsheets.

Gravel, rock and clay lumps are removed from the slurried ore by a rotating trommel. The resulting sand slurry is deslimed by hydrocyclones before treatment in a conventional spiral circuit. HMC containing approximately 96% heavy minerals is stockpiled by hydrocyclone. Sand tailing is pumped back to the mined out area to reform the land surface. Slime tailing from the North mine, derived from higher slime containing ore, is thickened before pumping to shallow solar drying areas. Slime tailing from the South mine settles in the dredge pond from where it can also be pumped to solar drying areas when required. The combined output of concentrate from the two PCPs is up to 800,000tpa. The two concentrates are blended before trucking to Chandala.

The maximum treatment rate of HMC at the MSP is 90tph. The total consumption is 740ktpa leaving the possibility of stockpile accumulation at the mines. The flowsheet required to separate mineral products in the MSP is complex. The HMC is first attritioned to remove coatings from the mineral grains and screened to remove coarse oversize grains. The resulting concentrate is dried and heated before electrostatic separation to separate the conducting minerals ilmenite, rutile and leucoxene from the non-conducting mineral zircon and most of the trash minerals including quartz, monazite and kyanite. Ilmenite, rutile and leucoxene are separated into products by magnetic separators. The non-conducting minerals are classified into two size ranges and magnetically cleaned before reslurrying then concentrating on wet tables, spirals and Kelsey jigs.

The concentrates are dried then zircon is produced after many stages of electrostatic and magnetic separation. A small output of staurolite, derived from the coarse non-conductor magnetics, has recently been added to the products.

The Becher Process employed in the Chandala SRP is an elegant combination of pyro and hydrometallurgy which removes most of the iron oxide portion of ilmenite and about 50% of the manganese oxide impurity resulting in synthetic rutile containing up to 93% TiO_2.

Ilmenite and Collie sub-bituminous coal are mixed in a rotary kiln. The coal is combusted to provide carbon monoxide for reduction and heat up to 1,100ºC. Iron oxide is reduced to metallic iron within the ilmenite grains. Sulphur is also introduced to the kiln to sulphidise manganese oxide within the grains. The reduced ilmenite is cooled and separated from coal char, which is recycled to the kiln.

Excess char is cleaned for sale as activated carbon. The reduced ilmenite is then slurried in aeration tanks with a weak solution of the oxidising catalyst ammonium chloride. Each batch is sparged with air while being agitated until the metallic iron has been reoxidised. During this process iron migrates from the reduced ilmenite to form separate particles of iron oxide less than 5 microns in diameter. Iron oxide is separated from synthetic rutile by hydrocyclones. The synthetic rutile is then agitated with dilute sulphuric acid to remove manganese sulphide and residual iron.

Treatment and disposal of waste products is a major part of the process, which occurs in the waste management plant. Iron oxide slurry is thickened and filtered. Acidic effluent is neutralised and the precipitates thickened and filtered. Sulphurous liquor from the kiln waste gas scrubber is treated with a double alkali process and the precipitates thickened and filtered. All liquors are recycled to the SRP. All filter cakes are returned to the mine for disposal.

The capacity of the SRP has been gradually increased to 220ktpa. An increase to 240ktpa is currently underway and is forecast to produce at that rate in 2006. Ilmenite consumption will rise to a maximum of 400ktpa, which still allows excess ilmenite from the MSP to be stockpiled at Chandala. This stockpiled ilmenite of high quality has the potential to maximise kiln capacity in the future if blended with lower quality ilmenite from orebodies developed to replace Cooljarloo North.

Kwinana Pigment Plant: The plant is situated in Kwinana, Western Australia where the process employed is the Chloride Process. The technology is provided by the Tronox Inc of Oklahoma. The initial capacity of the plant was 54.6ktpa when the plant was commissioned in 1991. The plant has since been expanded, largely by de-bottlenecking, to the point where the capacity is now 108,000tpa.

The feedstock for the plant is synthetic rutile ("SR") which is obtained from the Tiwest JV synthetic rutile production plant located near Muchea about 70km north of Perth. The feedstock has a TiO_2 grade of around 93%. The TiO_2 part of the synthetic rutile comprises mainly iron oxide as well as other metal oxides. The impurity – TiO_2 metal oxides – must be separated from the TiO_2 because, in general, these TiO_2 metals impart colour to pigment other than the desired white colour. The separation of the $TiCl_4$ from the other metal chlorides is achieved as described below.

Firstly almost all the SR is converted to a metal chloride vapour. Titanium tetrachloride ("$TiCl_4$") is one of the most volatile of the metal chlorides and has the convenient property that $TiCl_4$ is a liquid at ambient temperature and pressure. The SR is reacted with chlorine and coke to produce a mixture of metal chloride vapour (mainly $TiCl_4$), carbon monoxide, carbon dioxide and a small amount of hydrochloric acid vapour. This reaction is carried out in a fluid bed reactor referred to as a chlorinator. On leaving the chlorinator the exit gas is cooled. As the exit gas cools most of the metal chlorides other than $TiCl_4$ freeze out as solids.These solids are separated in a waste solids cyclone and are quenched with water to form an acidic metal chloride solution which is then transferred to the Effluent Treatment Plant. The gas passing through the waste solids cyclone now contains $TiCl_4$ vapour, carbon monoxide, carbon dioxide and a small amount of hydrochloric acid vapour. The gas is further cooled to condense out the $TiCl_4$ as a liquid. This $TiCl_4$ liquid is then treated and distilled to produce a pure form of $TiCl_4$. This purified $TiCl_4$ is stored prior to being used as the feedstock for the Oxidation Plant. The non-condensable gases report to the Waste Gas Scrubbing train. Waste gas is scrubbed with water to remove the hydrochloric acid vapour and forms a saleable 28% Hydrochloric Acid product.

$TiCl_4$ can be burned with oxygen in a special burner (called an Oxidizer) to form rutile (one of the crystal forms of TiO_2) crystal particles in a size range where they exhibit maximum light scattering properties. Light scattering translates to hiding power, which is the chief commercial property of TiO_2 pigment. In order to stabilise the rutile crystal it is doped with a small amount of aluminium, which is achieved by dissolving a small amount of aluminium chloride in $TiCl_4$ prior to oxidation. Gaseous oxygen is obtained from the L'Air Liquide plant in Kwinana by pipeline.

During the oxidation process chlorine is liberated from the $TiCl_4$. This chlorine is separated from the TiO_2 product and is recycled to the Chlorination Plant. The TiO_2 product from the Oxidation Plant is referred to as Raw Pigment. Raw Pigment particles do not mix into paint resins very well. To facilitate the incorporation of Raw Pigment into resins an alumina coating is applied to the Raw Pigment particles. Some grades of pigment will also be coated with silica and zirconia to enhance the durability of the paint. These coating steps are carried out in the Pigment Finishing Area. The main reagents consumed in the process to carry out these coatings include Sodium Aluminate, Sodium Silicate, Zirconium Oxychloride, Caustic Soda and Sulphuric Acid.

Raw Pigment slurry is first subjected to Sand Milling to breakup agglomerates of pigment. The product from the Sand Milling section is then cycloned and screened to remove remaining oversize particles. The screened Raw Pigment slurry is then fed to the Treatment Tanks where the coatings are applied. The product slurry from the treatment section is washed on rotary vacuum filters to remove remaining soluble chemical from the treatment process. The washed pigment slurry is then fed to gas-fired Dryers which remove water from the pigment. The pigment is then micronised. Micronisation is a process whereby pigment agglomerates are broken into particles of only a few microns in size. This is accomplished using high pressure steam jets in a device known as a microniser. The micronised pigment is then cooled and bagged into 25kg paper bags or bulker bags. The major part of the pigment produced goes into paper bags. The paper bags are shipped on pallets which are shrink-wrapped for protection against moisture.

6.5 Base Metals

6.5.1 Rosh Pinah

Original plant feed capacity was 570ktpa. This was increased to 670ktpa by plant modifications and upgrading in 1981 and during the latter part of the 1990s. These included the installation of column and tank flotation cells, and a secondary and a regrind mill. While the upgrades were motivated on efficiency improvements, some additional capacity was achieved. More recently, modifications were made to mill liners and mill ball load was increased, which increased mill power draw and hence capacity. Present plant capacity is around 750 – 800ktpa.

Ore is mined from a number of orebodies and a number of different ore types are identified including siliceous and non-siliceous carbonates, carbonaceous shales, etc. The ore contains about 2.5% lead (range 2% – 4%) as galena, about 10% zinc as sphalerite and 2.5% – 8% pyrite. It also contains minor copper and silver. Mined ore is crushed in three stages of crushing, milled and fed to a two-stage flotation circuit to recover lead and zinc. The lead and zinc concentrates are de-watered and stockpiled for dispatch to customers. The tailing is thickened and pumped to a slimes dam.

Mined ore is crushed to –150mm in an underground primary jaw crusher. The ore is brought to surface on a conveyor belt which discharges to a primary double-deck screen (45mm and 25mm top and bottom decks, respectively). Total oversize (+25mm) is fed to a secondary (Nordberg) cone crusher, in closed circuit with the primary screen. The screen undersize (–25mm) is conveyed to the primary stockpile, which has a live capacity of 6kt.

Ore is withdrawn from the stockpile by means of three vibrating feeders and conveyed to a double-deck secondary screen (19mm and 11mm top and bottom decks). In the case of both screens, the purpose of the top deck is simply to protect and reduce the load on the bottom deck. Plus 9mm oversize is fed to a tertiary crusher, a 66-inch Nordberg gyradisc, in closed circuit with the screen.

Screen undersize is conveyed to twin stockpiles with a total live capacity of 12,000t.

Ore is withdrawn from the stockpiles via vibrating feeders (three under each of the stockpiles) and conveyed to a primary, 12ft diameter. x 12ft Osborne Marcy ball mill. The mill is rubber-lined, has a grate discharge and is fitted with a 1,000kW motor. Cyanide (to depress zinc and pyrite in the lead flotation circuit) and a xanthate promoter (to float the lead) are added to the mill feed. Mill discharge is classified in two stages of cyclones. Primary cyclone underflow is returned to the mill, while secondary underflow is directed to the mill discharge sump. Secondary cyclone overflow is de-watered in a cyclone cluster, the overflow of which is also returned to the mill sump.

The cluster cyclone overflow may either be pumped to the secondary mill for further grinding or to the lead flotation conditioners. Present grind is about 80% passing 106m.

The secondary mill is a 5ft diameter x 10ft variable speed drive mill with a 100kW motor. The primary mill product is classified in a cyclone. Underflow is the secondary mill feed; overflow is directed to the mill discharge sump. Mill discharge is pumped to the lead conditioners.

The milling and flotation circuits are controlled by a Proscon PLC which incorporates a Minovex expert system for mill control, supported by an on-line Courier XRD analyser. The primary and secondary mills are well-instrumented with a variety of mass, flow, density, pressure, power and level transmitters (as indicated on the flowsheet) which feed into the expert control system. The Courier analyser provides lead, zinc, iron, copper and silver analyses of the float feed, lead tail, lead and zinc concentrates and final tails. Samples for the Courier are taken by automatic samplers, which also provide samples for lab analysis. Operator inputs to the system include:

- High and low limits for lead and zinc in feed (typically 4.5 and 1.5tph for lead and 18 and 10tph for zinc);
- Plus 106m and –38m fractions in the milled product;
- New mill feed, between 100 and 85tph; and
- Cyclone feed pressures and mill water dilution.

The expert system then controls the milling circuit to maximise tonnage (and consequently lead and zinc production) within the set-point limits and rules of the system. Typically with a low grade ore, tonnage would be increased towards the high limit, while with high grade ores metal input tonnage would become controlling.

The expert system was only commissioned a few months ago, but appears to be functioning successfully.

Slurry from the lead conditioner, to which frother has been added, is fed to three banks of conventional rougher and scavenger flotation cells. Flotation pH is 8.5 – 9.0. Rougher concentrate is cleaned in a column cell. Scavenger concentrate and cleaner tail is returned to the conditioner. The first cleaner concentrate (second and third lead cleaner cells are not presently in operation) is thickened, de-watered on a belt filter and discharged to the lead drying pad. From where it is loaded for trucking to Aus and on by rail to port for export to Walvisbay.

Scavenger tails are thickened and pumped to the zinc float conditioners. Concentrate and tail thickener overflow water is recycled to the mills and the lead circuit for re-use.

The lead tail slurry is conditioned with a xanthate promoter, copper sulphate to reactivate the zinc, lime to a pH of 10.5 – 11.0 and frother. zinc rougher flotation is carried out in an Outokumpu tank cell. The rougher concentrate is cleaned in a first cleaner tank cell and recleaned in a second cleaner column cell. Both cleaner tails are recycled to the zinc conditioners. Rougher tails are pumped to a bank of conventional rougher-scavenger cells. Concentrate is presently returned to the rougher tank cell. The regrind mill is not presently in circuit but may be used to regrind either or both of the rougher-scavenger concentrates.

The second cleaner concentrate is thickened, filtered on a belt filter and discharged to the zinc drying pad for dispatch to Aus by road and on to Zincor by rail.

Scavenger tails are thickened in one thickener and pumped to the tailings dam. Flocculant is used occasionally to aid settling.

Zinc concentrate and tailings thickener overflow is recycled and used for dilution in the zinc circuit. No water is returned from the tailings dam. The tailings are discharged at a L : S ratio of about 1 : 1, so plant fresh water consumption is about 1t per ore tonne treated. The ore contains small quantities of gold (0.1 – 0.3g/t) and 30 – 80g/t of silver. A Knelson concentrator was installed to investigate gold recovery, but this is not in operation. About 50% of the silver reports to the lead concentrate, 30% to the zinc concentrate and the balance reports to tails.

6.5.2 Zincor

The plant is split into three operating sections, comprising roasting (concentrate receiving, roasting and acid plants), the zinc plant (neutral leaching, purification, electrowinning and smelting and casting) and the recovery section (residue treatment and effluent treatment). Each section falls under a section manager who has reporting to him a plant superintendent responsible for production and a section engineer responsible for maintenance.

There are a number of service departments providing support to the production units. These include:

- Engineering services, providing electrical and instrumentation services, workshop facilities and such functions as rigging, scaffolding and cranes;
- Metallurgical services, who are engaged in research and development and project implementation, metal accounting and production forecasting;
- Laboratory, which provides analytical services; and
- SHEQ, which provides safety, health, environmental and quality assurance support services.

Concentrate Receiving: Concentrate from various sources is received by rail (some lots by road), off-loaded and stored separately in bunkers. The various feedstocks are blended and stored in bins, from which the roasters are fed. The prime reason for blending is to control $Cu+Pb+SiO_2$ content of the feed to the roasters. Ideally this should be below 5% – 6%. Above this level, there is a danger of calcine sintering in the roaster bed. The 2003 and 2004 averages were 4.7% and 5.2%. In the early months of 2005, the average has been around 5.5%. A magnesium pre-leach plant for strong acid leaching of dolomite from concentrate ahead of roasting has been re-commissioned. Rosh Pinah concentrate is treated in this circuit because of its high dolomite (Ca-Mg carbonate) content. Magnesium increases electrolyte conductivity, adversely affecting electrowinning efficiency and capacity.

Roaster-Acid Plant: This plant has two streams, each comprising two fluid bed roasters and an acid plant. Line 1 has the smaller roasters (noted earlier) and the smaller acid plant. The relative capacities of the two lines are about 1/3 : 2/3. The concentrate, containing up to 10% moisture, is fed via conveyors, rotary feeders and slingers which feed the roasters. Roasting takes place at ~950°C. Oxygen is injected into the roasters to increase kinetics and therefore capacity. The SO_2 gas produced exits the roasters, is cleaned and cooled and then fed to the acid plants where it is converted to sulphuric acid. Acid production capacity is about 550tpd. The calcine is cooled, dry milled and transported to the calcine storage tanks.

Neutral Leach: The zinc oxide (contained in the calcine) is leached with spent electrolyte in the ‘neutral’ leach circuit (pH 4.5 – 5). Purchased oxide concentrate is also leached in this circuit, depending on its composition. If halide content is excessive, the oxide would enter via the roasters to remove the halides. The leached slurry is thickened and the thickener underflow is pumped to the residue treatment (recovery) plant for more intensive leaching and zinc recovery. The thickener overflow solution, containing the leached zinc, is purified ahead of electrowinning.

Solution Purification: The impure zinc sulphate solution is subjected to two stages of purification. In the first, copper, cobalt and nickel is removed by precipitation with zinc dust and As_2O_3. In the second stage, cadmium is removed, also by way of precipitation with zinc dust and copper sulphate. The hot, purified zinc sulphate solution (steam heated at various points in the process to keep gypsum in solution) is pumped to storage ahead of electrowinning. The Cu-Co-As precipitate and the cadmium precipitate after further upgrading, are stored for sale.

Electrowinning: The hot purified solution is passed through cooling towers in order to precipitate gypsum, which is removed in a thickener and dewatered on a sieve bend. The gypsum is disposed of via the effluent treatment plant. Gypsum removal is critical to prevent scaling of pipelines as the solution cools after purification. It is also important to maintain temperature in the impure hot solution, through purification, until it reaches the cooling towers, for a similar reason. The cold solution is fed to the electrowinning circuit via storage/surge tanks where zinc is plated onto aluminum cathodes. Lead alloy anodes are used. A strontium carbonate salt is added to the electrolyte to precipitate impurities, in particular manganese. Sludge removed from the cells, rich in MnO_2, is sent to neutral leach. The cell house contains 42 banks of cells, each bank comprising 12 cells. Zinc is manually stripped from the cathodes once per day. These are transported to the melt house. Spent electrolyte is circulated to the neutral leach as well as to the residue treatment plant. Excess solution is recirculated to electrowinning. Spent electrolyte may also be bled from the circuit to maintain the overall solution balance and for Mg control.

Smelting and Casting: Cathode zinc is melted in four induction furnaces with an ammonium chloride flux, to increase the dross layer fluidity, which improves furnace performance. One and 2t jumbo ingots and 25kg ingots are produced. In addition, aluminium and lead-zinc alloy ingots are produced to meet customer requirements. Dross skimmed off the furnaces is sold; excess is returned to the roasters. Metallics from the dross and scrap zinc is converted to zinc dust and used in the solution purification circuit.

Residue Treatment: The residue treatment plant retreats the neutral leach residue to recover residual zinc and to separate dissolved iron. The neutral leach residue is subjected to a series of successive leaches with spent electrolyte at higher temperatures and stronger acid conditions. The leached slurry from each stage is thickened. Thickener overflow from each stage is returned to the previous stage, while underflow advances to the following leach stage. Final thickener underflow is filtered and washed on a belt filter and the final Pb/Ag residue disposed of on the tailings dam. The first stage overflow is treated with ZnO (calcine) to precipitate iron (as an iron-hydroxy-sulphate). The resulting iron rich slurry is thickened. The zinc rich thickener overflow solution is returned to the neutral leach while the underflow is filtered and washed on a second belt filter. The diluted belt filter filtrate is recycled within the circuit, the iron residue is disposed of to the slimes dam.

Effluent Treatment and Disposal: A number of effluent streams are treated in the effluent section before disposal:

- Spent electrolyte bleed is treated with slaked lime (to a pH of 6.0 – 7.0) to precipitate zinc. The resulting slurry is filtered. Filtrate containing magnesium and halides is pumped to the effluent treatment plant. The solids are repulped with spent electrolyte (at a pH of ~4.5) to leach zinc and added to the residue treatment process stream. The neutral leach plant run-off water is collected in an emergency dam and treated in the effluent treatment plant as is weak acid bleed.
- In effluent treatment, the various waste streams are neutralised (at pH 9.0 – 10.0) with limestone and slaked lime to precipitate other heavy metals. The resulting slurry is thickened. The thickened solids are disposed of to a demarcated section on the tailings dam. Thickener overflow is pumped to an evaporation system on the tailings dam. The lead/silver and iron residues are also disposed of separately on the tailings dam. Dam decant runs into a "penstock return water dam" and is returned to the effluent treatment plant. Some penstock return water is also pumped to a spray evaporation area adjacent to the penstock return water dam.

6.5.3 Chifeng

The nominal design capacity of the Chifeng smelter originally was 21ktpa slab zinc with 36ktpa sulphuric acid. Since 1998 the smelter has consistently exceeded the design capacity for both zinc slab and sulphuric acid production. The capacity of Phase I is now stated to be 23.5ktpa zinc. This highlights the robustness of the chosen process, which is supported by an original conservative design.

Baiyinuoer Mine started with the construction of a roaster and sulphuric acid plant at Lindong. With the intervention of Kumba this facility was taken over to form part of Chifeng.

The Lindong design as proposed by the Yangzhou Chemical Engineering Company is conservatively based on producing the equivalent of 30ktpa of contained zinc in calcine and 47ktpa of sulphuric acid.

Roasting: Concentrate roasting is performed via fluidized bed roasters roasting according to the standard ENFI design, or variants of it, and the Lurgi fluidised bed roaster concept. The major difference in the above roasting concepts being in the hearth and tuyere design, where the ENFI concept makes use of a bubble cap design whilst the Lurgi design makes use of small annular perforations in the bed of the hearth. All the concepts however basically adhere to the fluidisation velocity criterion for zinc concentrates of 0.4 – 0.7m/s in the hearth area, and 0.25 – 0.4m/s in the freeboard zone. The above figures imply air rates per hearth area of between 500 and 550$Nm^3/m^2/hr$. Typical dry solids design throughputs for ENFI roasters are 5.5 dry $t/m^2/day$, which for the latter air fluidising rates equates 90 to 100 dry t/Nm^3 air.

Zinc concentrate roasting for Chifeng and Lindong is preferentially conducted at temperatures in the region of 870º – 920ºC. This is a precautionary step governed by the high silica levels of the concentrates treated. Temperature control is executed by cooling coils and feed control. As a backup plan and when problems occur, direct water injection into the bed is used. At the fluidising air rates and operating temperatures that the roasters operate at, the typical ratio of calcine to bed material discharge is 60% – 65% and calcine to fume dust carryover to waste heat boiler are between 40% – 35%.

Fume/Particulate removal from the gas stream exiting the roaster is executed via conventional methods. Coarser particulates are first recovered in the waste heat boiler, cyclones and electrostatic precipitators. Finer air-borne particulates are scrubbed from the gas in a humidifying tower or venturi-scrubber, followed by a gas cooling tower and electric demister (mist precipitator). In this gas scrubbing circuit a weak acid solution is continuously circulated across the humidifying/venturi-scrubber and gas cooling towers to prevent solid and impurity carry-over to the sulphuric acid. This circulating weak acid stream undergoes heating in the process and requires to be cooled with cold water through a plate heat exchanger. A constant weak acid bleed stream is removed to the wastewater treatment plant in order to control a build-up of impurities.

Calcine Handling: For both Chifeng and Lindong the roaster calcine has to be cooled before being transferred to a storage or loading facility.

The process typically consists of cooling of roaster bed calcine in a water-cooled cooling drum before it is transferred to a pneumatic transfer vessel, in the case of Chifeng, and to a storage shed in the case of Lindong. The calcine/fume recovered in the waste heat boiler, cyclone and electrostatic precipitators is collected and conveyed via redler/conveyors to a common collecting water-cooled redler conveyor. All conveyors are enclosed units in order to prevent material spillages and losses.

At the Lindong site calcine is conveyed to a covered storage facility where it is bagged and loaded onto trucks for transport to the Chifeng site. On arrival at the Chifeng site the trucks are offloaded via an overhead crane and the bagged calcine stored in a new storage shed ahead of the main calcine storage silos.

The calcine collected from the Chifeng roasting facility is pneumatically transferred to the main calcine storage silos at the head of the leaching section, whilst the Lindong calcine is first emptied and milled in a dedicated milling circuit before being pneumatically conveyed to the same common calcine storage silos.

From the main storage silos the Chifeng and Lindong calcine mix is transferred to intermediate calcine storage silos located within the leach building from where it is added normally only at neutral leach stage and sometimes to stabilise the process also at the pre-neutralisation stage.

Acid Production: Acid production typically follows the double contact – double absorption process across a converter with vanadium pentoxide catalyst beds. A variant of this process is a possibility where a triple contact – double absorption system may be adopted. In the latter, the gas passes through three beds of V_2O_5 catalyst (versus two for the former) before passing through the inter-pass absorption tower. From the inter pass absorption tower the gas then passes through two beds of catalyst before being contacted with 98% H_2SO_4 in the final absorption tower.

Gas temperatures into the converter are maintained at the required and optimum temperatures for SO_2 conversion by contacting hot and cold gas streams counter currently in insulated vertical shell and tube heat exchanger arrangements. For cold start-ups an in-line electrical primary pre-heater is found in the ducting prior to the first entry point of the converter. This pre-heater is assisted by a secondary pre-heater to aid falling gas temperatures prior to entry of the last two contacting stages. The secondary pre-heater is generally also utilised for cold start-ups but may also be required during normal operations to raise falling converter temperatures.

Air and gas transport across the roasting to contact sections of the acid plant may be accomplished by various blower/fan arrangements. Typically a "push-pull" concept is adopted across the gas-train with the roaster blower "pushing" and an intermediate fan located prior to the contacting section of the Acid plant "pulling" the gas through the waste heat boiler, electrostatic precipitator and gas scrubbing sections and then "pushing" it on to the contact plant. This operation may be conducted with a roaster blower working either in tandem with a hot gas fan (located after the electrostatic precipitator) and a main SO_2 blower located in front of the contacting circuit (drying tower), or only with a main SO_2 blower (without a hot gas fan). Only a main SO_2 blower without a hot gas fan in both current acid plants is used.

Leaching: Leaching of the calcine is conducted using the conventional neutral leach and hot acid leaching stages with an intermediate pre-neutralisation step located between the two for acidity control prior to iron precipitation.

The Chifeng process operates a neutral leach at typical pH levels encountered in the industry. Its single hot acid leach differs from the norm in that is a single step operating at moderate terminal acidities (60 – 80 g/l H_2SO_4) and temperatures (80º – 85ºC).

The main reason for this is to prevent unmanageable circulating loads of silica building up within the leach circuit due to the high silica input from the raw materials. Furthermore this operating regime requires very little to no calcine addition to the pre-neutralisation stage for acidity control. The lack of process control and instrumentation complicates the operation of this pre-neutralisation stage and periodically leads to troublesome physical liquid/solid separation and throughput problems.

A secondary reason the chosen flow sheet is pursued is that there is no need to upgrade the Pb/Ag residue for silver recovery, as the silver input in raw materials is low. A slight sacrifice in zinc extraction is made at the option of operating a more forgiving and operator friendly circuit. At this stage the option of expanding the hot acid leaching stages is not justified as the operating costs and process throughput problems will outweigh the potential extra zinc recovery.

Iron Removal: Iron removal is in the form of ammonium jarosite based on the low contaminant jarosite process proposed in the late 1970s by Electrolytic Zinc Australia (Pasminco). The low contaminant claim comes from the fact that no calcine is added to the stage during iron precipitation, hence the precipitate should be very low in contaminants such as zinc, cadmium, copper and lead. Trace levels of heavy metal contaminants would then be associated with co-precipitated species and occluded soluble species.

The Chifeng process makes use of ammonium hydro carbonate as the source of alkali for both acid neutralization as well as ammonium jarosite precipitation and operates the process as close to 95ºC as possible and within the prescribed acidity levels.

The process operates smoothly with the required amount of iron being removed easily within the prescribed residence time. An easily filterable product is produced at the end of the reaction period.

The major weakness the process suffers from is the precipitate contamination caused though solids carry-over in the preceding pre-neutralisation stage.

Purification: Purification technology adopts a batch purification technique for the removal of copper and cobalt with traces of nickel in a common precipitate, followed by cadmium removal in low-grade cement. For the removal of cobalt the process adopts the hot zinc dust/arsenic technique with the arsenic being added as sodium arsenate after dissolving arsenic trioxide with caustic soda solution. In the cadmium scavenging process, zinc dust additions are made together with copper and ferrous sulphate, and potassium permanganate solution. The emphasis in the batch processing approach is placed on the ability to produce a first-pass high quality solution with extremely low levels of cobalt and cadmium. This quality comes at a high price in terms of reagent consumptions – particularly zinc dust powder. The lack of proper process control together with the high zinc dust dosage renders low-grade primary cakes in terms of cobalt, copper and cadmium. These primary cakes also contain high levels of basic and metallic zinc salts. Both the primary purification cakes undergo a mild acid leach/wash to recover the bulk of the zinc as well as to upgrade the by-product metal contents in a subsequent secondary treatment outside of the main process flow for zinc production.

By-Products Production: The by-products production process entails the production of three upgraded products of which two can be sold (containing copper and cobalt), and the third (cadmium), which is stockpiled. In the treatment process the combined primary purification filter cakes are firstly exposed to a mildly acidic leach/wash, which renders most of the cadmium and cobalt soluble but leaves the bulk of the copper precipitate intact. The solids recovered from this first stage represent the upgraded copper residue. In subsequent processing the dissolved cadmium is first cemented out with zinc dust powder under controlled zinc dust additions to yield high-grade cadmium cement. This step is followed by the controlled precipitation of the dissolved cobalt with arsenic trioxide and zinc dust powder to produce an upgraded cobalt residue.

Electrowinning: In the process of electrolysis, standard industry operating techniques are generally used within a cell house operating with a 24-hour plating cycle with manual cathode stripping. A conservative design in terms of operational current density is applied upfront.

The cell house configuration adopts tight cell electrode packing with small inter-electrode gap, which enables the operation to take full advantage of low electrolyte resistances and resultant low cell voltages and power consumption. There are two tank houses with two banks each of 57 cells containing 46 cathodes and 47 anodes. The current efficiency is 86%.

Casting: Casting of the zinc cathode into zinc slab adopts standard induction furnace and continuous casting belt technology.

Zinc Metal Supply: There are currently 25 different mines supplying concentrate to Chifeng. No formal toll treatment contracts are in place. There is a format on which payment is made. The compensation to each mine is negotiated every month. Based on current market conditions, a base line zinc price equivalent to ±65% of the zinc price is determined. Concentrates with >50% zinc content are fully compensated for with a RMB20/t deduction for every 1% below 50%. For concentrates containing 45% and less zinc RMB50/t is subtracted for every 1% less than 45% zinc. Concentrates containing 40% and less zinc are generally not accepted but if accepted RMB100/t is subtracted for every 1% less than 40% zinc. A penalty is imposed for CaO content of more than 0.5% at a rate of RMB60/t.

6.6 Industrial Minerals

6.6.1 Glen Douglas

The treatment process is essentially one of crushing, washing and screening to produce the various grades (sizes) of dolomite. The plant presently produces 17 different products, and has considerable flexibility to switch production between grades and product sizes to meet market requirements. Clearly, only high or low silica rock should be processed at any one time. Before a switch is made, live tonnage on the plant stockpile is run down to avoid cross contamination. Obviously this is more important when switching from high to low silica rock to avoid contamination of metallurgical grade products. This is not an issue on the product side, since the various products are stored in separate stockpiles, which provide supply for one market when products for the other are being produced.

The existing plant configuration and capacity has been unchanged for a considerable period, and no major modifications are planned. Ore from the pit (either metallurgical or aggregate grade) is fed to the primary gyratory crusher, set at 150mm. The crushed ore is conveyed to a 3,000t live stockpile. Two minus 150mm products are produced through the primary crusher: minus 150mm dump rock (aggregate stone) (product 959) and "Sub Base" (product 602).

Ore is withdrawn from the stockpile by three vibratory feeders. Maximum feed rate to the main plant is about 600tph. At present production rates, target feed rate is 460 – 480tph. The ore is fed to a double-deck washing screen fitted with a 80mm opening top deck and a 10mm bottom deck. The +40mm rock is conveyed to one of two secondary gyratory crushers. The crushed product is returned to a second, similar, washing screen in closed circuit with the crusher. The –40+10mm fractions from both screens constitutes product: 40mm metallurgical dolomite, or 37.5mm stone, depending on whether <2.5% or >2.5% SiO_2 rock is being processed.

These products may also be fed to a 19mm screen, which produce further products and feed for further processing. Minus 40+19mm rock is conveyed to separate sections of G-heap (<2.5% or >2.5% SiO_2) for further processing. The live capacity of this stockpile is 12,000 – 15,000t. A –53+19mm aggregate product, 53mm ballast, may be produced. Also, a 53mm metallurgical dolomite may be produced. These require alterations to secondary crusher setting and screen sizes.

The 'lumpy' products (–40+10mm and –53+10mm) are stacked separately and reclaimed for dispatch by means of a stacker-reclaimer installation. The 19mm screen undersize is an aggregate product designated 19mm stone.

The –10mm slurry from the double deck screens is dewatered in a spiral classifier. The fines and water is pumped to a clay-lined earth settling dam. These fines, which constitute about 6% of the new feed, are subsequently reclaimed for agricultural lime production. The spiral underflow (coarse –10mm fraction), is screened on a 5mm vibrating screen; the +5mm fraction constitutes the –13.2mm flat stone while the –5mm fraction is designated washed crusher sand. These are both aggregates.

It is possible to send the entire +5-40mm fraction to the G-heap, for example when metallurgical grade rock is being processed. The water used in the plant, largely on the washing screens, is ground water pumped out of the pit, or from the nearby Bass Lake (previously the A-pit).

The sinter plant produces sinter sand and BHK limestone from metallurgical grade rock from the G-heap. The –40mm rock is fed to four Hazemag impact crushers at a rate of 155tph. The crushed material is screened on a double-deck vibrating screen fitted with a 5.5mm top deck and a 1.5mm bottom deck. Oversize is returned to the crusher, the intermediate product, about 160tph, is the sinter sand, and the –1.5mm fraction, >15tph is the BHK limestone.

Minus 40mm rock from the aggregate grade section of the G-heap is fed to two vertical impact crushers. The crusher discharge constitutes Crusher Run product. Crusher feed rate is around 150tph.

When super sand is produced, feed rate is reduced to 50tph and crusher discharge is fed to 5-deck Morgenzon sizers. Oversize from the screens is returned to the impactors. A fine (–4.75mm screen) fraction is the super sand product.

An intermediate –13mm+4.75mm fraction is fed to the survival plant. Here the material is fed to a double deck screen with a 10mm top deck and a 5mm bottom deck; the plus 10mm fraction is designated 13.2mm round stone and the intermediate product is 9.5mm round stone. The –5mm fraction is returned to the G-heap.

Two agricultural lime products are produced. The first is BHK, produced in the sinter plant. The second, designated silver. Lime, is a mixture of BHK and main plant classifier fines reclaimed from the settling ponds. The two fractions are mechanically mixed by front end loader and fed to a horizontal impact crusher acting as a de-clogger. Crusher discharge is screened on a 7.5mm screen. Oversize is returned to the de-clogger or discarded. The screen undersize fraction is conveyed to the lime store.

Plant monitoring and control is by way of PLC controllers in the main and super sand/sinter plants. Emergency stops are provided at machines. This control system is about six years' old.

Two wet scrubbers and a dry, Donaldson filter, are employed to extract dust. Chemically dosed water sprays are applied at conveyor discharge points to allay dust.

6.6.2 Kumba FerroAlloys

The raw materials are iron strips and crude FeSi lumps that are purchased from Silicon Technology (Proprietary) Limited. The composition of the crude FeSi lumps is: silicon 77.8%, aluminum 1%, carbon 0.06% and sulphur and phosphorus both < 0.01%. The steel strips and crude ferrosilicon lumps are weighed off in batches of 2t in the ratio 0.2 crude ferrosilicon lumps to 0.8 iron strips.

The batches are then charged into the 2 x 2.5t 750kW induction furnaces for smelting. When the bath temperature reaches 1,600°C the slag is scraped off and the molten FeSi is poured through a nitrogen atomizer to form the FeSi powder. The nitrogen is injected at a pressure of 14 to 16 bar.

The powder is now pneumatically blown through a cyclone. The dust overflow goes to a bag house. The nitrogen is vented to atmosphere and the dust collected and returned to the product bin. The underflow from the cyclone is cooled and transferred by means of a screw conveyor in the product bin.

The powder is screened on a vibrating screen into a +212µm and –212µm fraction. The –212µm fraction is the product and the +212µm fraction is returned for re-smelting. The product is sampled manually. A laboratory on site does the screening analysis. Depending on the screening analysis the product is then graded as coarse, fine or super fine.