Ore variability management at Mamatwan manganese mine for an improved sinter product

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ABSTRACT

Geological and more specifically mineralogical inconsistencies in solid solution variations within natural ore deposits creates processing difficulties in metal production chains. Mamatwan mine established in 1963 produced manganese ore without addressing such geological variability, as the focus at that stage was on the ore production only, with the resulting Mamatwan mine facilities developed to reflect this production requirement. Changes in product requirements led to expansion of the Mamatwan mine product range by adding a Sinter Plant. A Dense Medium Separation (DMS) facility was also added to manage the mentioned variability of Mamatwan ore and subsequent sinter product. DMS facilities require significant resources to operate, while also generating significant waste (+/-40% of ore treated via the DMS is discarded - ore containing 36% and less manganese becomes tailings). The operational costs of DMS and the waste it generates initiated a search for an alternative variability management methods other than the DMS, that would be less resource and waste-generation intensive.

This dissertation presents a literature review of possible ore variability management strategies, as well as methods and procedures that would generate the necessary data to determine whether a possible alternative to the DMS could be used at Mamatwan mine.

A modified Stuart Pugh method was used for the development and Selection of the best Blend Mining trial option. Pugh method modifications comprised dedicated taxonomy to decrease ambiguity during Option Selection. Trial Concept Option Development and Selection methods were implemented to provide an optimum solution for the validation and verification of Blend Mining as an ore variability management method.

More than 18000 tons of sinter was produced as part of the trial. Chemical compositions were determined for samples extracted at pre-selected sampling points at Mamatwan Mine.

From the trial study, it was proven that Blend Mining has the ability to manage ore variability present in Mamatwan manganese ore. If implemented at Mamatwan mine, Blend Mining could render annual operational savings of roughly R40m. The lack of waste-generation is an additional benefit brought about by Blend Mining, which further contributes to total efficiency of the mine, as well as the implied maximisation of the Mamatwan ore resource. Differently stated, Blend Mining would facilitate an increased life-of-mine (LOM) for Mamatwan mine.

Keywords: Ore Variability, Option Development and Selection, Variability Management, Sinter Product

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- S7 construction contractor (Dassen Industries)

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ABBREVIATIONS

$(Mn, Fe)_2)_3$	–	Bixbyite
µm	–	Micrometre
AISA	–	American Iron and Steel Institute
Al	–	Aluminium
ASM	–	American Society for Metals
BHP	–	The Broken Hill Proprietary Company Limited
BHP Billiton	–	Broken Hill Proprietary Company Limited & Billiton Merger
BOF	–	Basic Oxygen Furnace
BOP	–	Basic Oxygen Process
C	–	Carbon
$CaCO_3$	–	Calcium carbonate (limestone)
$CaMn(CO_3)_2$	–	Kutnahorite
CaO	–	Calcium Oxide
CO	–	Carbon Dioxide
CO_2	–	Carbon dioxide
DMS	–	Dense Medium Separation
EAF	–	Electric Arc Furnace
EF	–	Eva Fines
EL	–	Eva lumpy ore
ES	–	Eva sinter ore
EVA	–	Erts-verwerkingsaanleg
Fe	–	Iron
Fe:Mn	–	Iron Manganese Ratio
Fe_2O_3	–	Iron (II) oxide (Magnetite)
Fe_3O_4	–	Iron (III) oxide (Hematite)
FeO	–	Iron oxide

g	–	Gram
G	–	Gravitational force
g/cm^3	–	Grams per cubic centimetre
HCFeMn	–	High-carbon ferromanganese
HMM	–	Hotazel Manganese Mines
Kg/m^3	–	Kilogram per square meter
LCSiMn	–	Low-carbon silicomanganese
Li	–	Lithium
LOI	–	Loss on ignition
LOM	–	Life of Mine
M1L	–	Mamatwan Lumpy ore (same as EL)
MCDM	–	Multi-criteria decision making
MCFeMn	–	Refined ferromanganese
Mg	–	Magnesium
mm	–	Millimetre
MMT	–	Mamatwan
Mn	–	Manganese
Mn_3O_4	–	Hausmannite
Mn_7SiO_{12}	–	Braunite
MnO	–	Manganese Oxide
MnS	–	Manganese Sulphite
mΩ	–	Milliohm
N	–	Nitrogen
NOx	–	Nitrogen oxides
^oC	–	Degrees Celsius
OPP	–	Ore processing plant
P	–	Phosphorous
P&ID	–	Process and instrumentation diagram

PFD	–	Process flow diagram
PDS	–	Product Design Specification
pH	–	Scale used to indicate acidity or basicity on a scale of 1 -14
ppm	–	Parts per million
PSD	–	Particle size distribution
PuCC	–	Stuart Pugh Controller Convergence
R.D.	–	Relative Density
ROM	–	Run-of-mine
SAE	–	Society of Automotive Engineers
SAMANCOR	–	SA Manganese Ltd and Amcor Ltd
SF	–	Sinter feed
Si	–	Silicon
SiMn	–	Silicomanganese
SO_2	–	Sulphur dioxide
SP	–	Sinter Product
WBS	-	Work-Breakdown-Structure
wt %	–	Weight percentage

LIST OF SYMBOLS

%	–	Percentage
Ω	–	Ohm (electrical resistance)

CHAPTER 1: INTRODUCTION

1.1 Background

Manganese is an important element used in the production of most plain carbon and alloy steels, and is of significant importance in the refining of such steels. In other words, the quality of steel is highly dependent in the addition of manganese as a refining aid. Furthermore, manganese is sometimes deliberately added to steel alloys to manipulate the hardenability and mechanical properties thereof. For example; high percentage volume additions of manganese (11-15%), results in propriety steels that exhibit extreme wear resistance through work hardening, without the usual decrease in toughness. These propriety steels are known as Hadfield or Mangalloy, sometimes simply referred to as manganese steel grades. Another material type highly dependent on manganese additions, are the very important SAE 201, 202 and 205 series stainless steels, which uses manganese as a partial nickel replacement, leading to more affordable stainless steel grades.

Manganese is present in the earth in the form of several manganese mineral types, which are essentially stable, but complex manganese oxides and carbides. South Africa has large manganese mineral deposits, which are regarded for its considerable quality (manganese content), compared to manganese resources in other parts of the world. Extraction and liberation of such manganese deposits are only the beginning of an extensive manganese alloy (ferromanganese-alloy) production chain, consisting of ore preparation (liberation from mineral deposits), metallurgical extraction through smelter reduction, and finally used in steel refinement or as a specific alloying agent.

The South African manganese mine Mamatwan, situated in the Northern Cape at the Kuruman district, forms part of such a ferromanganese-alloy production chain, and is a typical open-cast manganese mine. The Mamatwan-works (operation) originally only provided lumpy ore (6 to 75mm particles) to downstream processing facilities. Smaller particles also referred to as fines, especially particles smaller than 6mm were not originally considered saleable product, due to the fact that such sized particles, inadvertently leads to catastrophic furnace explosions or eruptions during the ferromanganese-alloy smelting process.

Smelter furnaces are charged continuously with raw materials. Raw materials are deposited from materials bins containing manganese ore, iron ore, lime, and coke, onto a furnace raw materials conveyor belt which then delivers the predetermined or measured mixture of raw-furnace-charge material into the furnace through a charging chute. The conditions inside a typical alternating-current-submerged-arc furnace in terms of temperatures is such, that the mentioned fines (<6mm) within the charge can from a crust that is none-permeable to the smelter reaction gasses evolved during the reduction reactions taking place while smelting. Build-up of these gasses which is mainly CO_2, eventually leads to catastrophic furnace eruptions, causing destruction of capital equipment and potential loss of life.

Because of the hazard posed by the use of Mamatwan fines in smelters, Mamatwan mine constructed and commissioned a sinter plant in 1987 to agglomerate fines produced at the mine to ensure safe application in smelters. The Mamatwan sinter plant also beneficiated Mamatwan ore from its normal 37% Mn content to above 44% Mn, lowering smelter submerged-arc-furnace raw burden resistivity.

Smelter feedstock variability can have a vast influence on smelter slag resistivity, slag volumes and therefore production rates as well as electrical power consumption. In other words, economic and therefore affordable ferromanganese production is related to smelter feedstock and the management thereof. Smelter facilities control or manages variability through the use of blending yards.

1.2 Problem Statement

The standard open-cast mining practices, which includes, screening and crushing is not sufficient to control fines variability to within acceptable limits to provide manageable sinter feedstock which in turn allows for efficient smelting facility operation. At Mamatwan mine and ore processing facilities, space limitations allows for only limited options in terms of ore variability management. Due to greater pressure on manganese prices, the variability management needed to be directed at lowering operational costs.

1.3 Aim of this Study

The aim of this study is to review the processing or mineral liberation approaches that are applied through mining practises to address significant compositional variability, occurring in ore resources.

The aim of this study is also extended towards the application and evaluation of a viable variability management method at the Mamatwan Manganese mine, in order to improve operational and production efficiencies, by circumventing the DMS plant, thereby increasing the life-of-mine (LOM).

CHAPTER 2: LITERATURE REVIEW

2.1 Manganese and the Industry

The element, manganese, finds significant industrial application. Even though it is used in the general chemical applications, it is consumed in great volumes in the manufacturing industry. The following sections will elucidate on some of the more pronounced or significant applications of manganese.

2.1.1 Iron Ore Beneficiation (Pig-iron production)

The economic growth of countries is related to many interdependent factors. This interdependency can make the identification of one particular factor that can be used as a gauge for economic growth difficult (Radetski *et al.*, 1990). However, based on many economic studies a definite indication of economic expansion of a country is related to the increase in the production and consumption of steel (Warell, 2009) and subsequently iron ore (Tilton, 1990). Typically, economic growth is founded on the construction of industrial machinery, industrial processing facilities such as power plants, etc. Economic growth is also supported by the production of consumer goods, such as automobiles. These mentioned industries, require to a very large extent, the import or local production of steel.

The mentioned steel production requires the processing of iron-ore, which are essentially iron-oxides of different oxidation numbers; Fe_2O_3 and Fe_3O_4 (Fe^{2+} or iron(II) and Fe^{3+} or iron(III)), otherwise known as haematite and magnetite respectively. Therefore, reduction of the iron-oxides to iron is required before suitable iron or steel can be produced. Three stages of reduction is required to extract iron from iron ore; $Fe_2O_3 \rightarrow Fe_3O_4 \rightarrow FeO \rightarrow Fe$ (Higgins, 1993).

The mentioned reduction of iron ore is achieved by mixing and charging iron ore, coke and limestone together into a blast furnace through a double bell-and-cone gas-trap system. Simultaneously, air which is pre-heated to improve coke consumption economy is forced into the bottom of the furnace through the tuyeres. The coke is ignited through the use of gas burners at the bottom of the furnace. The reduction processes that arise from these reactions are generally referred to as smelting, and can be expressed in the by stating the chemical reaction occurring during this reduction process.

Most importantly a reducing atmosphere must be produced and can be achieved by the following reaction:

1. C (coke) + O_2 (air) => CO_2 + heat (exothermic)

2. CO_2 + C (excess coke) => 2CO - heat (endothermic)

The carbon monoxide produced in (2.) rises through the granular furnace charge and reduces iron(III):

3. Fe_3O_4 + 3CO => 2Fe + $3CO_2$

The reduction as indicated in (3.) continues progressively in three stages (Fe_3O_4=> Fe_2 O_3 => FeO => Fe), until all iron oxide is reduced to iron. The reduction starts at the top the top of the blast furnace and continues as the charge moves to the bottom of the furnace where liquid pig iron is extracted. Simultaneously to the reduction reactions, additional reactions takes place which are such that gangue material or unwanted material that is naturally part of iron bearing ore is removed from the reduced iron and accumulated as fluid slag that is lighter than the fluid pig iron and therefore floats on top of the pig iron. The following reactions takes place and it is important to also note that additional carbon dioxide is produced by the decomposition of limestone ($CaCO_3$) (4.), while the lime combines with gangue silica (5.):

4. $CaCO_3$ => CaO + CO_2 (Limestone => Lime + Carbon-dioxide)

5. 2CaO + SiO_2 => 2CaO.SiO_2 (Lime + Silica => Calcium-silicate or slag)

The Figure 2.1-1, (a) and (b) below are diagrammatic depictions of blast furnace configurations with indications of the temperatures and main reactions that take place.

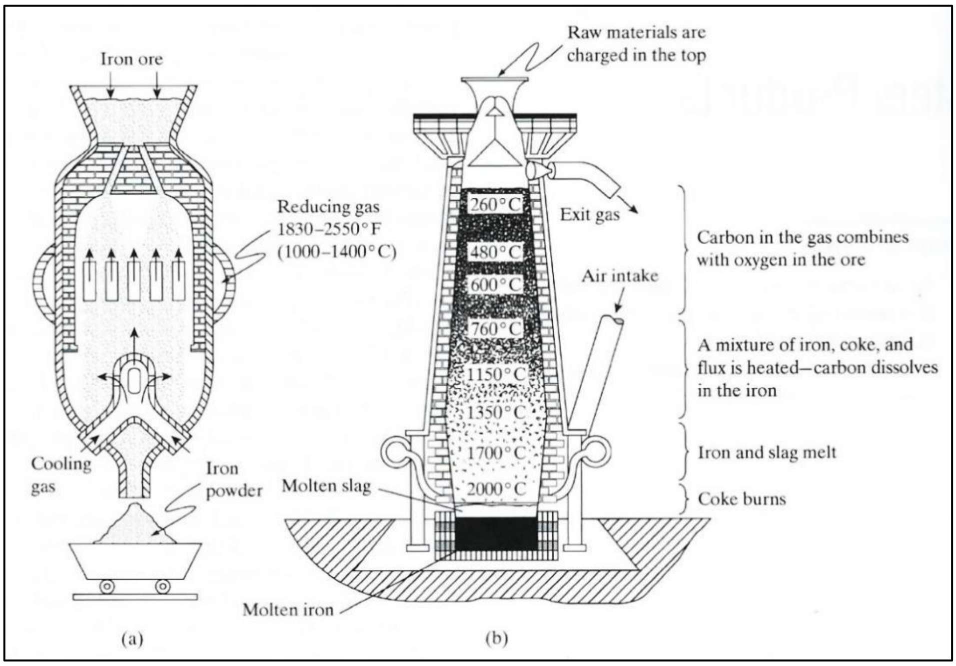


Figure 2.1-1 Typical blast furnace configurations (after Budinksi *et al.*, 2002)

2.1.2 Steel Refining Processes

After accomplishing the reduction of iron ore to pig-iron through reactions as presented in section 2.1.1, the molten pig-iron is further refined significantly through several refining steps. This is required due to the fact that pig-iron contains significant level of other elements; up to 10% (Higgins, 1993). Particularly, detrimental is the presence of sulphur, phosphorous and carbon. The inclusion of these elements leads to pig-iron being a crude stepping stone in the process of producing ferrous metals for engineering application. In other words, pig-iron is mechanically inferior. It is therefore required to further refine pig-iron through the application of additional processing steps.

Currently the most prominent methods of refining pig-iron are the basic oxygen furnace (BOF), or basic oxygen process (BOP), and the electric arc furnace (EAF) (Smith *et al.*, 2011). Because of the effectiveness of these to processes, most steel refining operations worldwide make exclusive use of these methods, and the Bessemer and open hearth processes are not widely used anymore due to being less efficient, than the BOF and EAF.

The BOF or BOP process entails transferring liquid pig-iron from the blast furnace to a basic oxygen furnace. These furnaces are referred to a basic due to the high pH levels of the slag and refractory liner. This is based on work by Henry Bessemer in 1856 which lead to the Bessemer process previously mentioned, whereby air is forced through liquid pig-iron to oxidise unwanted elements (Higgins, 1993). However, in the BOF process high purity oxygen is forced into the surface of the molten pig-iron to react with sulphur, phosphorous, carbon and other unwanted elements to form oxides. As this oxidation reaction takes place, flourospar (a mineral) is added to form additional slag where these oxides can be accumulated. In other words, detrimental elements can be separated from the steel and trapped in the slag. This treatment can sometimes also include other gasses such as nitrogen to impart specific steel product properties, by being absorbed by the molten steel. Figure 2.1-2 below presents a simplified diagrammatic representation of the BOF process.

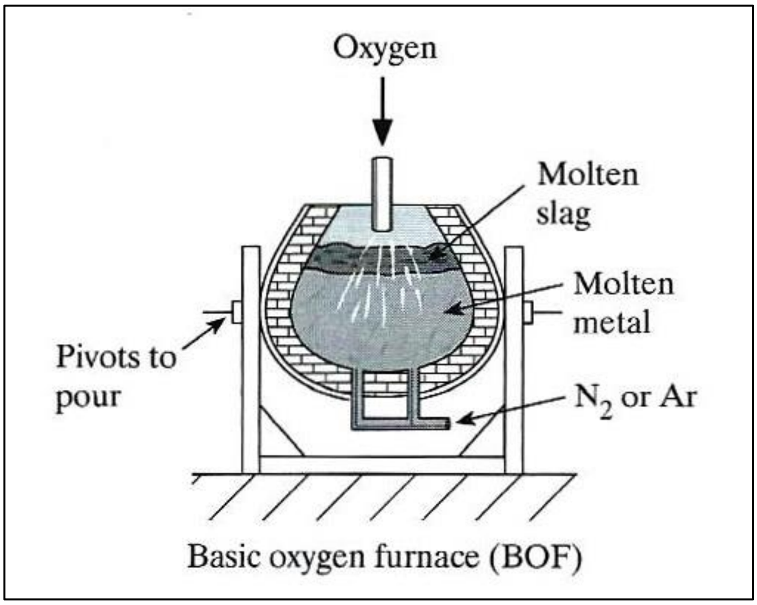


Figure 2.1-2 Simplified illustrations of typical BOF process (after Budinksi *et al.*, 2002)

Because if this refining step, carbon contents of the pig-iron can be reduced from a typical 4 to 5% (Budinksi *et al.*, 2002) to below 0.5% (Smith *et al.*, 2011), with lower carbon content levels achievable by strictly controlling oxygen flow rates and oxygen lance to metal bath distances, which improves the oxidation potential of the carbon in the molten metal (Barron *et al.*, 2014). The amount of time at which oxygen blasting commenced further enhances the refinement of steel by allowing more time to remove carbon and other constituents (Jalkanen *et al.*, 2004). Figure 2.1-3 below is presentation of how typical unwanted or detrimental constituent contents in a BOF processing cycles is lowered with increasing time.

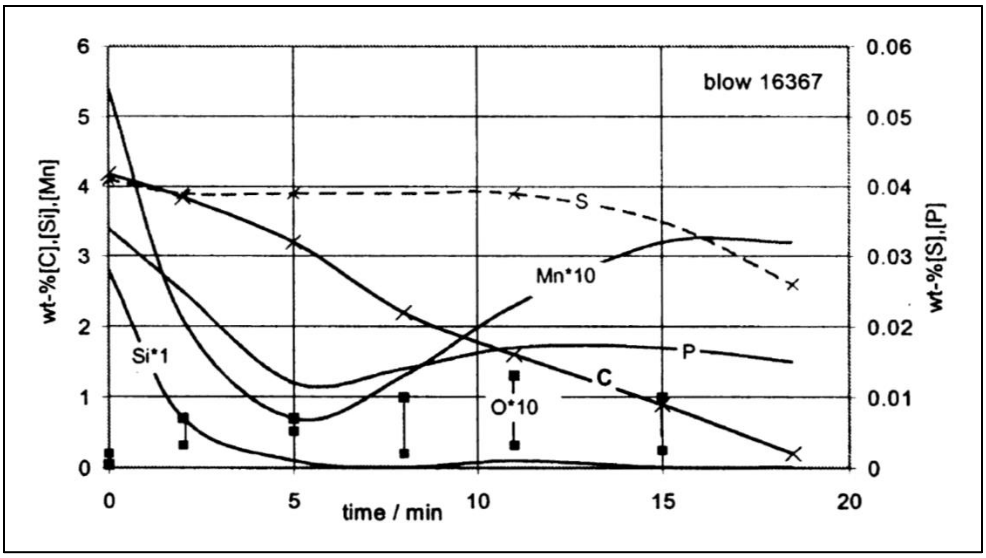


Figure 2.1-3 Graphical presentation of chemical constituent decreases with time during a BOF cycle (after Jalkanen *et al.*, 2004)

While the BOF processing step controls the level of unwanted elements in the steel products, it is also used as a point where other alloy additions are made on purpose to facilitate steel decarburisation, de-oxidation and aiding the removal of for example sulphur. Typically such additions are made through the addition of ferroalloys (ferromanganese and silicon-manganese) (Ho Jappa *et al.*, 2013).

Where pig-iron solidified, or where only scrap-metal is used, EAF processing is employed in order to re-melt and solidified material and/or scrap (reclaimed) metal. Graphite electrodes and electric arcing is used to re-melt the solid material, while oxygen lancing and slag additions are used to refine the re-melted material to produce specific grade steels. Figure 2.1-4 present a simplified sectional illustration of typical EAF steel refinement. The use of ferroalloys in the EAF can significantly improve electrical energy consumption during processing

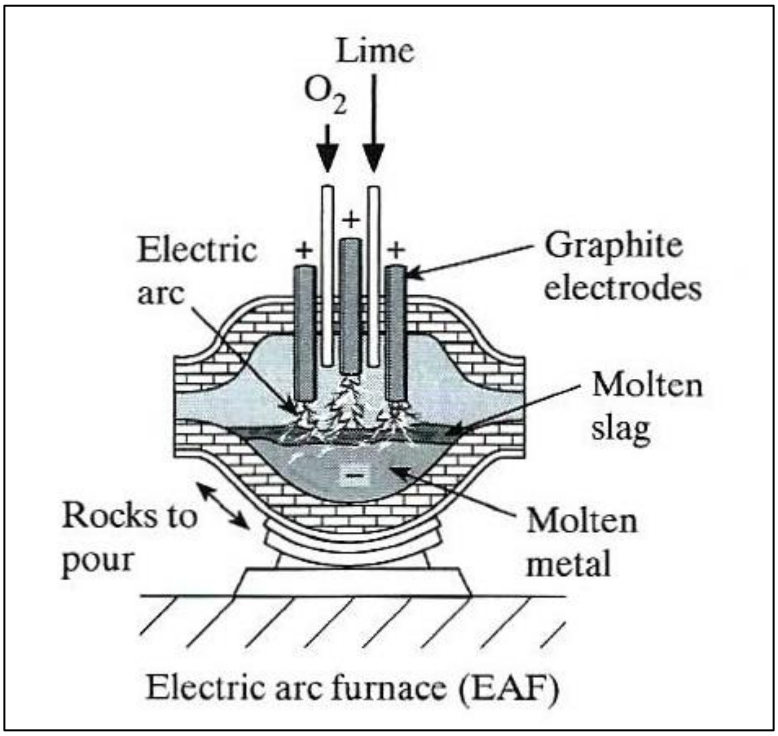


Figure 2.1-4 Simplified illustrations of typical EAF process (after Budinksi *et al.*, 2002)

The use of the BOF or EAF as the only refining step in steel making is limited to the production of mainly standard and structural steel grades. Where, special properties and lower levels of certain detrimental elements such as sulphur, phosphorous, and very low levels of carbon is required, secondary or even tertiary steel refining processes are used. Typically, austenitic stainless steels require carbon contents as low as 0.02% to prohibit sensitisations, which is the appearance of carbide precipitate at the grain boundaries of affect or sensitised materials. For typical stainless steel (Type 304), the carbides that do precipitate at the grain boundaries are chromium carbides and usually occur between 425 and 870^{o}C (ASM International, 2005). Figure 2.1-5 gives an indication of the effects of time temperature and carbon content in terms of the propensity Type 304 stainless steel to sensitise. This propensity to sensitise can adversely affect the application spectrum of such common stainless steels because arc welding used for modern construction can lead to sensitisation if the material with high carbon content (> 0.02%) is welded. Numerous case studies have been recorded where sensitisation caused premature failure of materials in corrosive environments. Such failures are sometime referred to as intergranular corrosion or weld decay where heat input during welding was the main cause of sensitisation (Fontana, 1986).

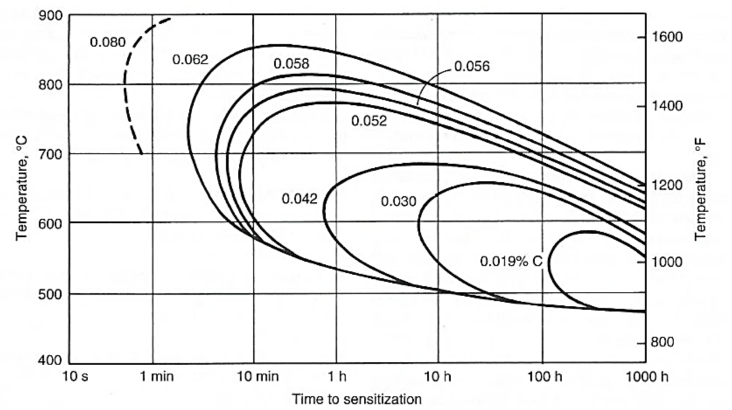


Figure 2.1-5 Time-temperature-sensitisation curves for Type 304 stainless steel in a mixture of $CuSO_4$ and H_2SO_4. The curves is indicative of the times required for carbide precipitation to start occurring for different carbon contents (ASM International, 2005)

Secondary and tertiary refinement can be performed for the production of speciality steel, such as tool steels, and primarily uses the application of vacuum furnaces to de-gas steels and also to remove undesirable elements and contaminants (Budinksi *et al.*, 2002).

2.1.3 Manganese and its Influence as Ferromanganese addition in Steel Refinement

The use of manganese stems the from its influences not only steel properties during purposeful addition thereof for the production of Hatfield steel and certain types of low cost stainless steel (further discussed in Section 2.1.4 of this dissertation), but also steel refinement as indicated in Section 2.1.2 of this dissertation. For steel refining purposes manganese is not added to molten steel in its elemental form, but as an alloy consisting of iron and manganese (ferromanganese) or silicon and manganese (silicomanganese). The use of manganese can therefore be categorised as follows:

- As steel making or steel refining aid with the addition of ferromanganese and silicon-manganese.
- Steel alloying addition, which can also be divided into several categories:
    - Manipulation of carbon steel properties
    - Manipulation of crystallographic phases

In terms of steel making, manganese contributes to the control of several aspects important to the final quality of steel products. These positive contributions can be summarised as follows:

- Manganese readily combines with oxygen to form manganese oxide (MnO), and therefore plays an important role in the deoxidisation of steel (Steenkamp *et al.*, 2013). In other words, it aids the removal of oxygen from the steel. It however, is important to state that manganese are not as strong and deoxidiser and silicon or aluminium, but stabilises these deoxidisers by forming stable manganese- silicates and aluminates which significantly improves the deoxidisation of the steel during secondary and tertiary refining.

- Manganese also has a significant tendency to combine with sulphur to form manganese sulphate (MnS). The formation of manganese sulphate removes free sulphur from the steel before the formation of iron sulphate can occur, which is extremely detrimental to mechanical properties of steel products (Olsen *et al.*, 2007).

Manganese is introduced during the steel refining process as manganese ferroalloys. These alloys are generally available in several variations (Olsen *et al.*, 2007):

- High-carbon ferromanganese (HCFeMn)
- Refined ferromanganese(MCFeMn)
- Silicomanganese (SiMn)
- Low-carbon silicomanganese (LCSiMn)

The chemical composition of these alloys is presented in Table 2.1-1. Of importance is the percentage manganese present in each variation.

Table 2.1-1 Ferromanganese and silicomanganese alloy chemical compositions (adapted from Olsen et al., 2007)

	Chemical Composition (wt%)					
	Mn	C	Si	P	S	B (ppm)
HC FeMn (high carbon)	78	7,5	0,3	0,2		
MC FeMn (medium carbon)	80-83	1,5	0,6	0,2		
LC FeMn (low carbon)	80-83	0,5	0,6	0,2		
SiMn	67	1,7	17,5	0,1	0,02	200
LC SiMn (low carbon)	60-63	0,5	25-27	0,1	0,01	100
ULC SiMn (ultra-low carbon)	58-62	0,05	27-31	0,05	0,01	100

It is also noteworthy to mention that effectively around 10kg of manganese is required for every ton of refined steel produced (Higgins, 1993).

2.1.4 Manganese as an alloying element

The information in the previous section gives an indication of the use of manganese in the production and refinement of steel. However, manganese is also extremely important in terms of its steel alloy properties, or differently stated, its ability to influence the properties of steels it is added to.

In terms of carbon steel, the addition of manganese reduces the eutectoid temperature of the iron-carbon phase diagram, effectively increasing the hardenability of low alloy carbon steels. The concentrations of manganese for this purpose are rarely higher than 2% (ASM International, 1990).

Sir Hadfield performed a significant number of experiments, by increasing the manganese content of steels and in 1882 he invented what is known today as Hadfield steel (Smith *et al.*, 2001). This class of steel alloys contained higher concentrations of manganese (10 to 14%), which changed the phase chemistry to such an extent (lowered the eutectoid temperature) that the resulting alloy displays an austenitic crystal structure at room temperature. Therefore, it is ductile and able to endure significant elongation during tensile testing, or tensile loading. It also has a significant propensity to work-harden, and finds application in railroad, earthmoving and certain crushing applications (Mahlami *et al.*, 2014).

Because manganese is an austenite stabilising alloying element, it has found application in certain grades of austenitic stainless steels, where a portion of the nickel (also an austenite stabiliser) is replaced by manganese to render a less expensive but still corrosion resistant austenitic stainless steel. These stainless steel grades are referred to as AISI/SAE Types 201, 202 and 205 (ASM International, 1990).

2.1.5 Other Industries and Manganese

2.1.5.1 Pollution control

Polymers are finding ever increasing uses in world in almost all aspects of consumer demands, such as packaging, automotive, clothing etc. (Devold, 2013). In 2003 it was estimated that the world consumed 200 million tons of plastic yearly (Vlachopoulos *et al.*, 2003). More recent estimated puts world polymer consumption at 299 million tons per year (Gourmelon, 2015). Polymers are almost exclusively made from derivatives produced from the refinement of crude oil, natural gas, synthetic organic gas (coal gasification) (Budinksi *et al.*, 2002). Any refinement of crude oil, natural gas, synthetic organic gas produces appreciable amounts of air pollution (Sage Environmental Consulting, LLP, 2015). The Republic of South Africa has also instituted regulatory legislation to control a wide spectrum of harmful emissions, which include sulphur dioxide (SO_2), nitrogen oxides (NOx) (National Environment Management: Air Quality Act, 2005). This statement can also be extended to the production of lubricant, waxes and fuels, as these are all primary products as well as by-products of the refinement process.

These regulatory requirements are not only imposed on plants but also on automotive vehicles (Department of Minerals and Energy, 2006). This mentioned regulation states <500ppm sulphur for standard grade diesel and <50ppm sulphur for low-sulphur grade diesel. Therefore the specifications for diesel fuel are meant to drive a decrease in sulphur content of the fuels to limit the levels of SO_2 emission released not only by plants but also internal combustion engine diesel vehicles. Because of the above mentioned, more effective means of removing sulphur from crude oil and resulting fuels is being explored. Therefore, manganese and more specifically manganese dioxide have been found to be very efficient and cost effective in terms of the desulphurization of crude oil (Adekanmi *et al*., 2012). Lower levels of sulphur compounds in crude oil will yield lower sulphur contents in end-product fuels such as diesel.

2.1.5.2 Batteries

A battery is a device that is capable of converting potential chemical energy directly into electrical energy through reduction-oxidation reactions (Linden *et al.*, 2002). Batteries are used in numerous applications where direct current electrical power is needed.

The use of specific combinations of material have enabled the purposefully design of batteries for very specific applications. Table 2.1-2 presents typical electrochemical and physical characteristics of battery cell materials.

Table 2.1-2 Characteristics and Properties of Battery Materials (after Linden *et al.*, 2002)

Material	Atomic or molecular weight, g	Standard reduction potential at 25°C, V	Valence change	Melting point, °C	Density, g/cm³	Electrochemical equivalents		
						Ah/g	g/Ah	Ah/cm³‡
Anode materials								
H_2	2.01	0 −0.83†	2	—	—	26.59	0.037	
Li	6.94	−3.01	1	180	0.54	3.86	0.259	2.06
Na	23.0	−2.71	1	98	0.97	1.16	0.858	1.14
Mg	24.3	−2.38 −2.69†	2	650	1.74	2.20	0.454	3.8
Al	26.9	−1.66	3	659	2.69	2.98	0.335	8.1
Ca	40.1	−2.84 −2.35†	2	851	1.54	1.34	0.748	2.06
Fe	55.8	−0.44 −0.88†	2	1528	7.85	0.96	1.04	7.5
Zn	65.4	−0.76 −1.25†	2	419	7.14	0.82	1.22	5.8
Cd	112.4	−0.40 −0.81†	2	321	8.65	0.48	2.10	4.1
Pb	207.2	−0.13	2	327	11.34	0.26	3.87	2.9
$(Li)C_6$ (1)	72.06	∼−2.8	1	—	2.25	0.37	2.68	0.84
MH (2)	116.2	−0.83†	2	—	—	0.45	2.21	—
CH_3OH	32.04	—	6	—	—	5.02	0.20	—
Cathode materials								
O_2	32.0	1.23 0.40†	4	—	—	3.35	0.30	
Cl_2	71.0	1.36	2	—	—	0.756	1.32	
SO_2	64.0	—	1	—	—	0.419	2.38	
MnO_2	86.9	1.28‡	1	—	5.0	0.308	3.24	1.54
NiOOH	91.7	0.49†	1	—	7.4	0.292	3.42	2.16
CuCl	99.0	0.14	1	—	3.5	0.270	3.69	0.95
FeS_2	119.9	—	4	—	—	0.89	1.12	4.35
AgO	123.8	0.57†	2	—	7.4	0.432	2.31	3.20
Br_2	159.8	1.07	2	—	—	0.335	2.98	
HgO	216.6	0.10†	2	—	11.1	0.247	4.05	2.74
Ag_2O	231.7	0.35†	2	—	7.1	0.231	4.33	1.64
PbO_2	239.2	1.69	2	—	9.4	0.224	4.45	2.11
Li_xCoO_2 (3)	98	~2.7	0.5	—	—	0.137	7.29	—
I_2	253.8	0.54	2	—	4.94	0.211	4.73	1.04

* See also Appendixes B and C.
† Basic electrolyte: all others, aqueous acid electrolyte.
‡ Based on density values shown.
(1) Calculations based only on weight of carbon.
(2) Based on 1.7% H_2 storage by weight.
(3) Based on x = 0.5; higher valves may be obtained in practice.

Table 2.1-3 Tabulated list of Primary, Reserve and Secondary Batteries and their respective electrochemical properties (after Linden *et al.*, 2002)

				Theoretical values†				Practical battery‡		
Battery type	Anode	Cathode	Reaction mechanism	V	g/Ah	Ah/kg	Specific energy Wh/kg	Nominal voltage V	Specific energy Wh/kg	Energy density Wh/L
Primary batteries										
Leclanché	Zn	MnO_2	$Zn + 2MnO_2 \rightarrow ZnO \cdot Mn_2O_3$	1.6	4.46	224	358	1.5	85[4]	165[4]
Magnesium	Mg	MnO_2	$Mg + 2MnO_2 + H_2O \rightarrow Mn_2O_3 + Mg(OH)_2$	2.8	3.69	271	759	1.7	100[4]	195[4]
Alkaline MnO_2	Zn	MnO_2	$Zn + 2MnO_2 \rightarrow ZnO + Mn_2O_3$	1.5	4.46	224	358	1.5	145[4]	400[4]
Mercury	Zn	HgO	$Zn + HgO \rightarrow ZnO + Hg$	1.34	5.27	190	255	1.35	100[6]	470[6]
Mercad	Cd	HgO	$Cd + HgO + H_2O \rightarrow Cd(OH)_2 + Hg$	0.91	6.15	163	148	0.9	55[6]	230[6]
Silver oxide	Zn	Ag_2O	$Zn + Ag_2O + H_2O \rightarrow Zn(OH)_2 + 2Ag$	1.6	5.55	180	288	1.6	135[6]	525[6]
Zinc/O_2	Zn	O_2	$Zn + \frac{1}{2}O_2 \rightarrow ZnO$	1.65	1.52	658	1085	—	—	—
Zinc/air	Zn	Ambient air	$Zn + (\frac{1}{2}O_2) \rightarrow ZnO$	1.65	1.22	820	1353	1.5	370[6]	1300[6]
Li/$SOCl_2$	Li	$SOCl_2$	$4Li + 2SOCl_2 \rightarrow 4LiCl + S + SO_2$	3.65	3.25	403	1471	3.6	590[4]	1100[4]
Li/SO_2	Li	SO_2	$2Li + 2SO_2 \rightarrow Li_2S_2O_4$	3.1	2.64	379	1175	3.0	260[5]	415[5]
$LiMnO_2$	Li	MnO_2	$Li + Mn^{IV}O_2 \rightarrow Mn^{IV}O_2(Li^+)$	3.5	3.50	286	1001	3.0	230[5]	535[5]
Li/FeS_2	Li	FeS_2	$4Li + FeS_2 \rightarrow 2Li_2S + Fe$	1.8	1.38	726	1307	1.5	260[5]	500[5]
Li/$(CF)_n$	Li	$(CF)_n$	$nLi + (CF)_n \rightarrow nLiF + nC$	3.1	1.42	706	2189	3.0	250[5]	635[5]
Li/I_2[3]	Li	I_2(P2VP)	$Li + \frac{1}{2}I_2 \rightarrow LiI$	2.8	4.99	200	560	2.8	245	900
Reserve batteries										
Cuprous chloride	Mg	CuCl	$Mg + Cu_2Cl_2 \rightarrow MgCl_2 + 2Cu$	1.6	4.14	241	386	1.3	60[7]	80[7]
Zinc/silver oxide	Zn	AgO	$Zn + AgO + H_2O \rightarrow Zn(OH)_2 + Hg$	1.81	3.53	283	512	1.5	30[8]	75[8]
Thermal	Li	FeS_2	See Section 21.3.1	2.1–1.6	1.38	726	1307	2.1–1.6	40[9]	100[9]
Secondary batteries										
Lead-acid	Pb	PbO_2	$Pb + PbO_2 + 2H_2SO_4 \rightarrow 2PbSO_4 + 2H_2O$	2.1	8.32	120	252	2.0	35	70[10]
Edison	Fe	Ni oxide	$Fe + 2NiOOH + 2H_2O \rightarrow 2Ni(OH)_2 + Fe(OH)_2$	1.4	4.46	224	314	1.2	30	55[10]
Nickel-cadmium	Cd	Ni oxide	$Cd + 2NiOOH + 2H_2O \rightarrow 2Ni(OH)_2 + Cd(OH)_2$	1.35	5.52	181	244	1.2	35	100[5]
Nickel-zinc	Zn	Ni oxide	$Zn + 2NiOOH + 2H_2O \rightarrow 2Ni(OH)_2 + Zn(OH)_2$	1.73	4.64	215	372	1.6	60	120
Nickel-hydrogen	H_2	Ni oxide	$H_2 + 2NiOOH \rightarrow 2Ni(OH)_2$	1.5	3.46	289	434	1.2	55	60
Nickel-metal hydride	MH[1]	Ni oxide	$MH + NiOOH \rightarrow M + Ni(OH)_2$	1.35	5.63	178	240	1.2	75	240[5]
Silver-zinc	Zn	AgO	$Zn + AgO + H_2O \rightarrow Zn(OH)_2 + Ag$	1.85	3.53	283	524	1.5	105	180[10]
Silver-cadmium	Cd	AgO	$Cd + AgO + H_2O \rightarrow Cd(OH)_2 + Ag$	1.4	4.41	227	318	1.1	70	120[10]
Zinc/chlorine	Zn	Cl_2	$Zn + Cl_2 \rightarrow ZnCl_2$	2.12	2.54	394	835	—	—	—
Zinc/bromine	Zn	Br_2	$Zn + Br_2 \rightarrow ZnBr_2$	1.85	4.17	309	572	1.6	70	60
Lithium-ion	Li_xC_6	$Li_{(i-x)}CoO_2$	$Li_xC_6 + Li_{(i-x)}CoO_2 \rightarrow LiCoO_2 + C_6$	4.1	9.98	100	410	4.1	150	400[5]
Lithium/manganese dioxide	Li	MnO_2	$Li + Mn^{IV}O_2 \rightarrow Mn^{IV}O_2(Li^+)$	3.5	3.50	286	1001	3.0	120	265
Lithium/iron disulfide[2]	Li(Al)	FeS_2	$2Li(Al) + FeS_2 \rightarrow Li_2FeS_2 + 2Al$	1.73	3.50	285	493	1.7	180[11]	350[11]
Lithium/iron monosulfide[2]	Li(Al)	FeS	$2Li(Al) + FeS \rightarrow Li_2S + Fe + 2Al$	1.33	2.90	345	459	1.3	130[11]	220[11]
Sodium/sulfur[2]	Na	S	$2Na + 3S \rightarrow Na_2S_3$	2.1	2.65	377	792	2.0	170[11]	345[11]
Sodium/nickel chloride[2]	Na	$NiCl_2$	$2Na + NiCl_2 \rightarrow 2NaCl + Ni$	2.58	3.28	305	787	2.6	115[11]	190[11]

†Based on active anode and cathode materials only, including O_2 but not air (electrolyte not included).
*These values are for single cell batteries based on identified design and at discharge rates optimized for energy density, using midpoint voltage. More specific values are given in chapters on each battery system.
(1) MH = metal hydride, data based on 1.7% hydrogen storage (by weight).
(2) High temperature batteries.
(3) Solid electrolyte battery (Li/I_2 (P2VP)).
(4) Cylindrical bobbin-type batteries.
(5) Cylindrical spiral-wound batteries.
(6) Button type batteries.
(7) Water-activated.
(8) Automatically activated 2- to 10-min rate.
(9) With lithium anodes.
(10) Prismatic batteries.
(11) Value based on cell performance, see appropriate chapter for details.

Apparent from Table 2.1-3, are the significant benefits arising from the use of Lithium-Manganese-dioxide batteries. From the presented table is it clear that this combination yields a relatively favourable specific energy compared to most other battery-material-combinations. As a comparison, a lead-acid battery delivers a practical specific energy of 35Wh/kg, whereas the Li/MnO_2 battery delivers a practical specific energy of 265Wh/kg. From this comparison the power to weight benefit of using Li/MnO_2 batteries compared to conventional batteries are apparent.

The above information is only meant to briefly indicate some benefits of Mn and MnO_2 combinations. However, a significant amount of experience and knowledge is available on battery design and application. However, it is not the aim of this study to review the application of manganese in batteries is great detail, but rather to highlight some beneficial aspects of it application in batteries. Table 2.1-4 give some indication of typical commercial electrochemical cell or battery systems, as well as characteristics and application. It is noteworthy that MnO_2 containing battery systems finds use in common consumer application, such as flashlights, radios and instruments, but also specialist civilian applications, such as portable battery operated equipment. MnO_2 also finds highly specialised application in military and aerospace radio systems (Linden *et al.*, 2002).

Table 2.1-4 Characteristics and Applications of Primary Batteries (after Linden *et al.*, 2002)

System	Characteristics	Applications
Zinc-carbon (Leclanché) Zinc/MnO_2	Common, low-cost primary battery; available in a variety of sizes	Flashlight, portable radios, toys, novelties, instruments
Magnesium (Mg/MnO_2)	High-capacity primary battery; long shelf life	Military receiver-transmitters, aircraft emergency transmitters
Mercury (Zn/HgO)	Highest capacity (by volume) of conventional types; flat discharge; good shelf life	Hearing aids, medical devices (pacemakers), photography, detectors, military equipment but in limited use due to environmental hazard of mercury
Mercad (Cd/HgO)	Long shelf life; good low- and high-temperature performance; low energy density	Special applications requiring operation under extreme temperature conditions and long life; in limited use
Alkaline (Zn/alkaline/MnO_2)	Most popular general-purpose premium battery; good low-temperature and high-rate performance; moderate cost	Most popular primary-battery: used in a variety of portable battery operated equipments
Silver/zinc (Zn/Ag_2O)	Highest capacity (by weight) of conventional types; flat discharge; good shelf life, costly	Hearing aids, photography, electric watches, missiles, underwater and space application (larger sizes)
Zinc/air (Zn/O_2)	Highest energy density, low cost; not independent of environmental conditions	Special applications, hearing aids, pagers, medical devices, portable electronics
Lithium/soluble cathode	High energy density; long shelf life; good performance over wide temperature range	Wide range of applications (capacity from 1 to 10,000 Ah) requiring high energy density, long shelf life, e.g., from utility meters to military power applications
Lithium/solid cathode	High energy density; good rate capability and low-temperature performance; long shelf life; competitive cost	Replacement for conventional button and cylindrical cell applications
Lithium/solid electrolyte	Extremely long shelf life; low-power battery	Medical electronics, memory circuits, fusing

For a more extensive comparison of battery system characteristics and application data please refer to Table 5.1-2 and Table 5.1-3 presented in Appendix A.

From the information presented in this Appendix, it apparent that the demand for MnO_2 containing batteries is on the increase and will increase in the future (Linden *et al.*, 2002).

2.2 Ferro-manganese Alloy Production Chain

As indicated in previous sections of this dissertation, manganese is of great importance to steel production as a steel refining aid. However, manganese ore (manganese containing minerals) is not added to the steel refining process directly, but processed in such a way that the resulting ferromanganese alloy enhances the steel refining process in terms of efficiency of electrical use (ferromanganese reacts exothermally during the steel refining process adding process heat and ensure compositional consistency. Because of this function, an entire ferromanganese production chain exists. Figure 2.2-1 gives a process flow diagram from the ore resources through ferromanganese production and finally through to the consumer products (Schutte, 2011) .

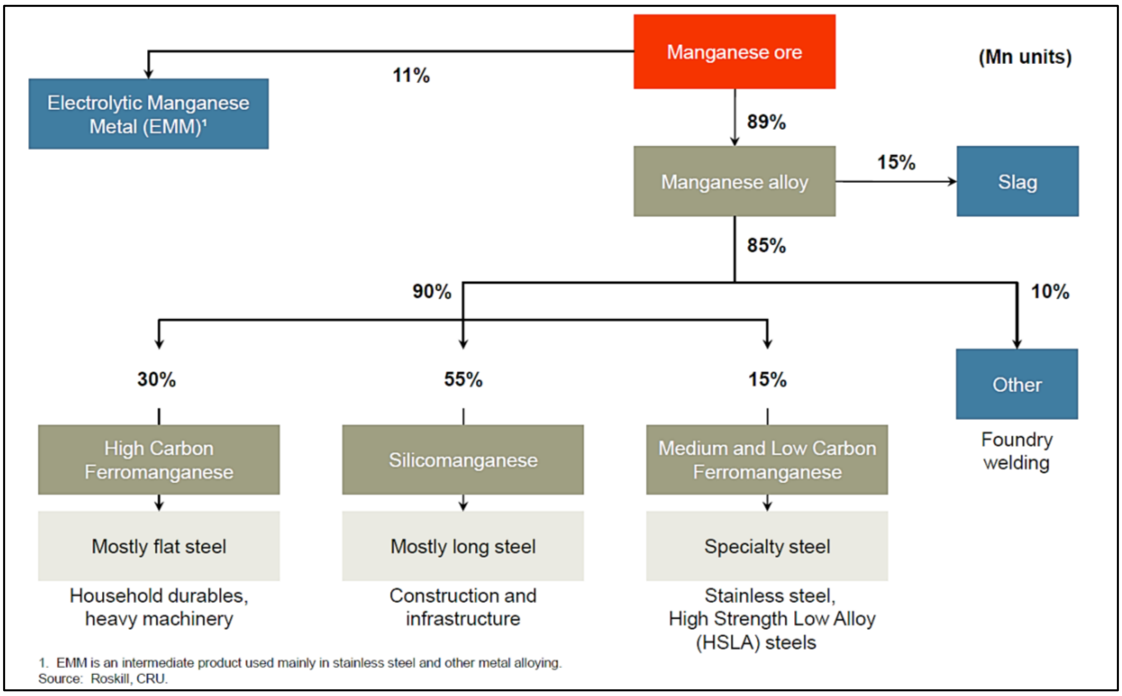


Figure 2.2-1 Schematic presentation of the production chain for steel making (after Schutte, 2011)

The following sections will attempt to explain the typical ferromanganese production chain in more detail, by first reviewing geological aspects of the South African manganese geology of importance for mining. Since this study is focused on ore variability management at Mamatwan mine, the review will contain almost exclusively Mamatwan mine geology literature.

2.2.1 South African Manganese Resources – Geology

For more than six decades, South African Manganese have been extracted mainly from to the Postmasburg and the so-called Kalahari manganese fields (Preston, 2001). Since this study and dissertation is focused on the management of Mamatwan manganese ore variability, and Mamatwan mine is situated in the Kalahari manganese fields, only the Kalahari Manganese field will be further discussed.

2.2.1.1 The Kalahari Manganese Field and Mamatwan

In the Kalahari Manganese Field (Cairncross *et al.*, 1997), mainly two ore types are recognised; low-grade Mamatwan-type ore, rich in carbonates with a 30 – 39% manganese content, and a high-grade Wessels-type ore, rich in oxides with a manganese content greater than 42% (mostly up to 44% and sometime even higher).

However, some geological researchers differentiates between Mamatwan-type ore, Wessels-type ore and Blackrock-type ore based on marked differences in chemical composition, variations in different ore's mineralogy, as well as on differences in the textures of the ores (Kleyenstuber, 1993).

Even so, on-going geological studies are indicative of a fourth ore type that can be added to the classification; a supergene enriched ore found in close association with the sub-outcrop on the eastern side of the Kalahari basin underlain by Mamatwan-type ore. The term supergene-ore can be used to describe this enriched sub-outcrop ore, since all indications are that this ore has formed due to oxidised enrichment from the surface down (Kleyenstuber, 1993). Figure 2.2-2 below presents the position of the Kalahari manganese field with respect to the greater Southern Africa, and also indicates the thrust direction at Black Rock and Wessels Mine, as well as faults associated with several mines in the area Kalahari Manganese Field.

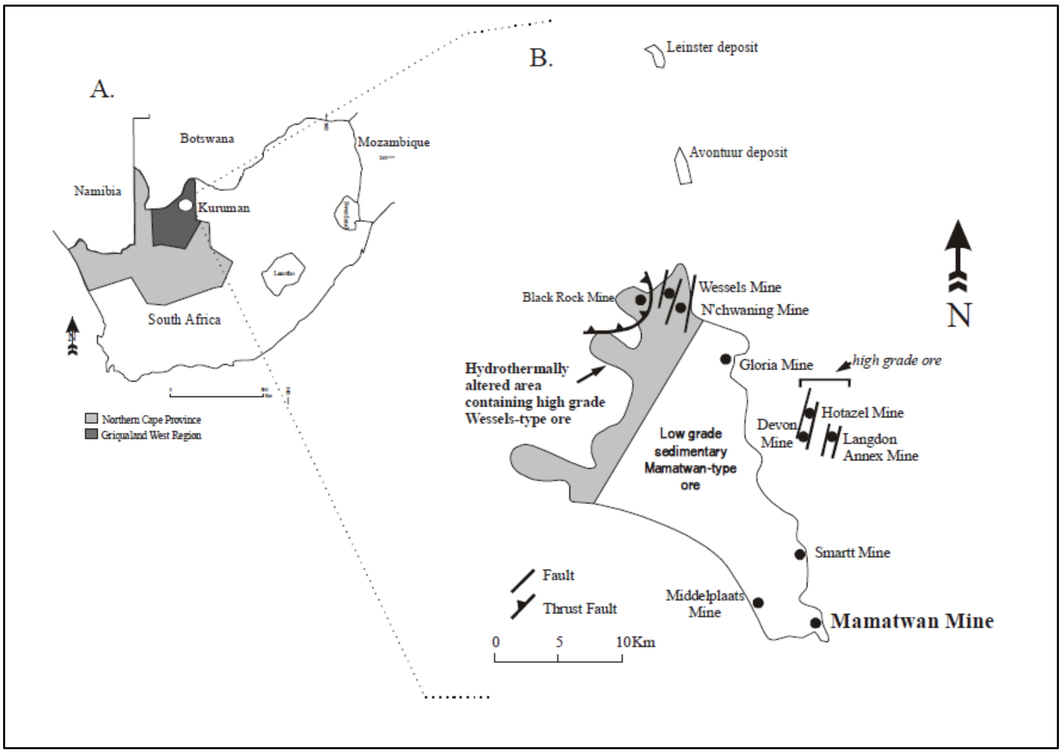


Figure 2.2-2 (A) General map of South Africa indicating the primary Manganese deposits, and (B), and more detailed diagram of the deposit and the position of some major manganese mines enduring over the last 6 decades (after Gutzmer *et al.*, 1996).

As mentioned two main ore types are prevalent in the area indicated in Figure 2.2-2; low-grade carbonate-rich Mamatwan-type ore (30 to 39 wt % Mn), and high-grade oxide-rich Wessels-type ore (>42 wt % Mn), with possible further ore type sub-division is possible. However, due to the fact that this study is related specifically to Mamatwan ore, Wessels-type ore and further subdivisions will not be discussed in significant detail in this dissertation.

Low-grade Mamatwan-type ore comprises 97% of the total ore reserves known as the Khalahari Manganese Field (Kleyenstuber, 1984), and consists of several microcrystalline constituents with particle sizes of smaller than 10 ɥm. These micro-constituents are typically a combination of kutnahorite ($CaMn(CO_3)_2$), braunite (Mn_7SiO_{12}), and hematite. Furthermore, a significant amount of larger than millimetre-sized ovoids of kutnahorite, recrystallized hematite, and braunite occur in this so-called microcrystalline structure, considered to be of sedimentary or early diagenetic origin (Kleyenstuber, 1984). This typical Mamatwan-type ore indicates that the proto-ore was subjected to a series of metamorphic, hydrothermal and supergene alteration events (Gutzmer *et al.*, 1996).

Even though there were several regional alteration events, two of these events are of importance to the Kalahari ore field; the event related to the formation of Mamatwan-type ore and the event related to the formation of Wessels-type ore.

The alteration event responsible for the formation of Mamatwan-type ore involved late diagenetic (process of changes that occurs within sedimentary deposits due to the influence of high temperature en pressure) replacement of kutnahorite by a first generation hausmannite (Mn_3O_4). Some recrystallized hematite(II), bixbyite ($(Mn, Fe)_2)_3$), and Mn-calcite were also formed during this event. Such event driven replacements or alteration are associated along specific strata in the mentioned Mamatwan-type ore in terms of bedding-parallel veinlets. These veinlets are evidenced by replacement products of carbonate laminae and ovoids (Nel *et al.*, 1986). Furthermore, Hausmannite(I) occurs as a coarse subhedral (displaying on the partial formation of respective crystal faces) to eahedral (displaying well defined respective crystal faces) crystals, which contains less than 3 wt % Fe_2O_3.

The second alteration event, is commonly referred to as the Wessels alteration event, and was the result of structurally controlled hydrothermal activity, caused by east-verging thrust duplication and a north to northeast trending fault (figure 1). This second alteration is considered responsible for the existence of high-grade ore (>42 wt % Mn). Wessels-type ore therefore formed due to metasomatical transformation of Mamatwan-type ore to oxiderich and coarse crystalline Wessels-type ore (Kleyenstuber, 1979) However, since only Mamatwan-type ore is under consideration in this study, events leading to the formation of Wessels-type ore will not be elucidated or dwelled upon in great detail. It can be stated that the alterations are generally associated with formation joints, veins or erosional irregularities (Evans, 2001)

2.2.1.2 Mamatwan Ore Mineralogy and Chemistry

Over the course of the previous century significant geological studies were performed on the Kalahari manganese field to establish a better understanding thereof, in order to enable more effective mining strategies and activities. Such studies also allowed for the necessary ore resource knowledge to design ore processing plants. Figure 2.2-3 below present a diagrammatic overview of Mamatwan mine pit as well as the general mineralogical stratigraphy important to mining economy.

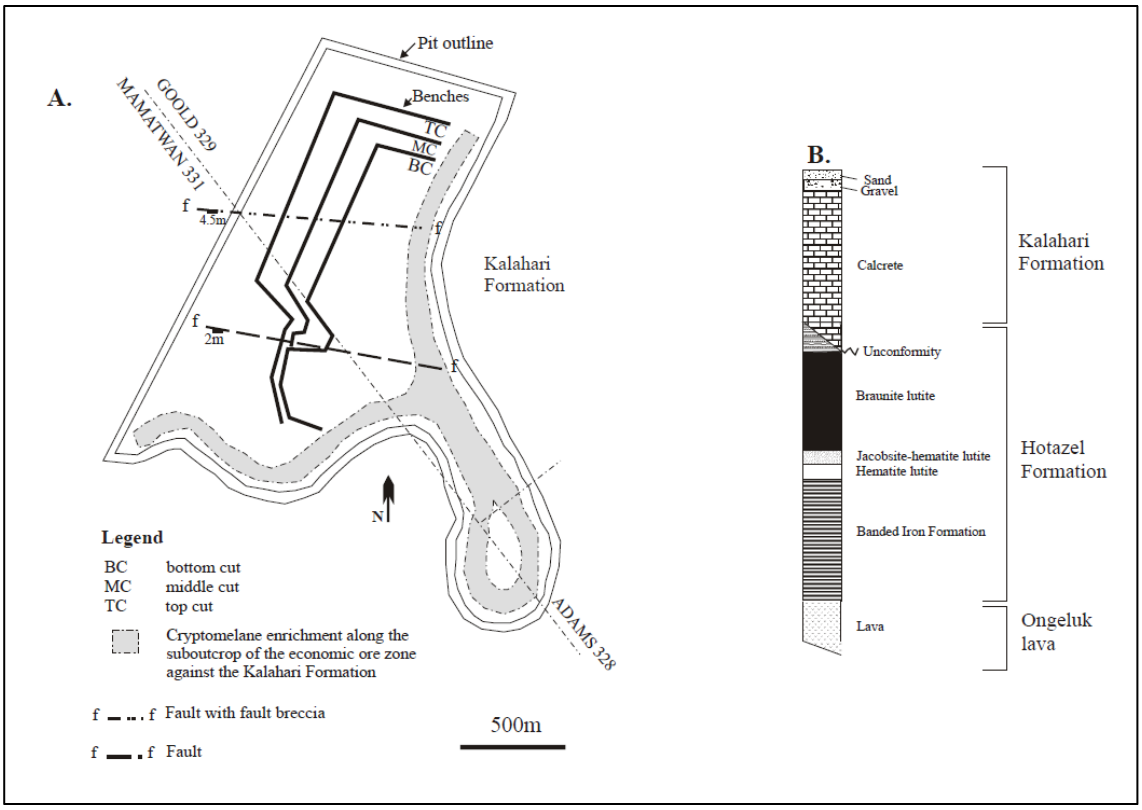


Figure 2.2-3 (A) Open-cast pit map of Mamatwan mine and (B) Simple stratigraphic succession of it ore deposits (after Gutzmer *et al.*, 1996)

Several in depth geological studies were conducted to allow detailed mine planning as well as process design for the Mamatwan mine and ore processing and beneficiation plants to be performed. Mamatwan mine ore is representative of low-grade ore mentioned in Section 2.2.1.1. Again, as mentioned in previous sections the term low-grade refers to manganese content which is typically less than 39 wt %. This typical Mamatan-type ore is mainly dark brown in colour, but also contains intermittent white and light brown spots. These spots represent ellipsoidal carbonate ovoids (oval shape structures). The fine grained matrix material between the ovoids is braunite with an average grain size of 20µm. Braunite is considered the main manganese bearing mineral presents in Mamatwan ore. Kutnahorite (manganese-dolomite) is die main mineral that comprises the carbonate ovoids, but can sometimes be found as fine grain material dispersed in the braunite matrix material. Minerals considered as minor manganese bearing minerals, are hausmanite and jacobsite. Typical weight present value for the major elements and constituents representative of grade and mineralogy as determined during geological studies (Kleyenstuber, 1979) as Table 2.2-1.

Table 2.2-1 Mamatwan ore major elements and constituents as per studies and Samancor product specifications (adapted from Kleyenstuber, 1979)

Elements	Mass %	
	Kleyenstuber, 1979	Samancor 1979
Manganese (Mn)	38.09	37 – 38
Iron (Fe)	2.15	4 – 6
Calcium Oxide (CaO)	12.81	12 – 16
Magnesium Oxide (MgO)	2.52	3 – 5
Carbon Dioxide (CO_2)	16.08	15 – 17
Silica (SiO_2)	3.62	4 - 6

The information in Table 2.2-1 is representative of the average chemical and constituent compositions for Mamatwan as determined during geological studies(Kleyenstuber, 1979). The Mamatwan mine ore specification at that time is reflected in this table. Note that at that time Mamatwan mine was part of Samancor.

In Table 2.2-1 Mamatwan ore's major elements and constituents as per studies and Samancor product specifications even though acceptable from a mine product management viewpoint is not a true reflection of the mineralogical composition of Mamatwan ore. Data divulged in this table should be viewed as summary of the major elements and constituents.

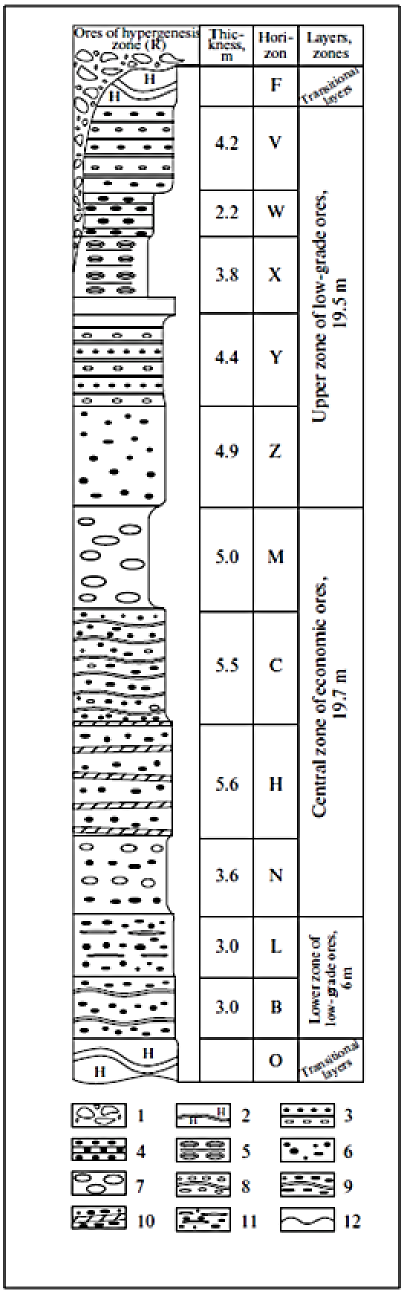
A more comprehensive understanding of the mineralogical composition is presented in Table 2.2-2, which is a description of Mamatwan ore in terms of the chemical composition of the respective minerals as well as their respective physical properties of which density was extensively studied (Overbeek, 1986).

Table 2.2-2 Chemical composition and densities of the main minerals in Mamatwan ore (adapted from Overbeek, 1986)

Mineral	Chemical Formula	Mineral Mn content %	Density g/cm^3	Mineral Mass % in Ore
Baurnite	$Mn^{2+}Mn^{3+}{}_6SiO_{12}$	53.1	4.7	30 – 50
Hausmannite	Mn_3O_3	72.0	4.7	2 – 20
Cryptomelane	KMN_8O_{16}	59.8	4.3	2 – 8
Manganite	MnOOH	62.5	4.3 – 4.4	Trace
Jacobsite	$MnFe_2O_4$	23.8	4.7	2 – 3
Hematite	Fe_2O_3	0	5.2	7 – 10
Kutnahorite	$Ca(Mn, Mg)(CO_3)_2$	23.0	3.8	18 – 25
Calsite	$CaCO_3$	0	2.8	2 - 15

In Table 2.2-2 the percentage elemental manganese present in the respective minerals are varied. Further variation is evident from the mineral mass percentage range in ore. It is therefore clear that elemental manganese content as indicating in Table 2.2-1 will vary as the mineral composition of the ore varies within the range as indicated in the last column of Table 2.2-2. This can also be described as the general Mamatwan mine ore variability.

A simplified lithostratigraphic subdivision of the Mamatwan ore body is presented in Figure 2.2-4 (Nel *et al.*, 1986).



Simplified lithostratigraphic sub-division of the Mamatwan ore body

Legends 1 to 12 is described in more detail below:

(1) Manganese oxide ores (hypergenesis zone);
(2) Purple red hematite lutites with carbonate laminas;
(3)Light grey massive carbonate manganolutites with small ovoids of white carbonate and fine (white and pink) carbonate laminas;
(4) Dark and light grey banded braunite lutites with fine inclusions of white and brown carbonate ovoids;
(5) Dark grey braunite lutites with interlayers containing abundant white carbonate ovoids;
(6) Massive dark grey hematite bearing braunite lutites (with abundant inclusions of brown carbonate ovoids);
(7) Dark grey massive braunite lutites (with large white carbonate ovoids and hausmannite inclusions);
(8) Dark and light grey banded braunite lutites with white carbonate laminas and numerous small (white and brown) carbonate ovoids;
(9) Jacobsite and hematite lutites with laminas and ovoids (brown and pink);
(10) Light grey braunite lutites (with small brown carbonate ovoids) alternating with reddish brown hausmannite interlayers;
(11) Purple red braunite lutites with numerous brown carbonate ovoids;
(12) Braunite lutites with numerous brown to white carbonate nodules.

Figure 2.2-4 Simplified lithostratigraphic subdivision of the Mamatwan ore body (after Nel et al., 1986). Legends 1 to 12 is described more detail under points 1 to 12.

Even though the typical lithostratigraphic sub-division of the Mamatwan ore is presented in Figure 2.2-4 above, it is important to consider the continuation thereof to the north and west of the open cast mining area of Mamatwan mine, as this is the future direction mining expansion. Figure 2.2-5 below presents the mine pit design as it was in 2001 during which time an extensive geological drill core analyses was performed along line A – A’ (East to

West) and B – B' (South to North), revealing the stratigraphic consistency in each of the direction mentioned. Figure 2.2-6 present diagrams A and B that give an indication of stratigraphic variation towards the West and North of the Mamatwan mining pit along lines A and B in Figure 2.2-5. It is evident from these diagrams that the ore deposits as already described in terms of mineralogy and chemistry, angles downward from the eastern perimeter of the pit. In other words, manganese ore deposits dips deeper below the surface towards the west and north.

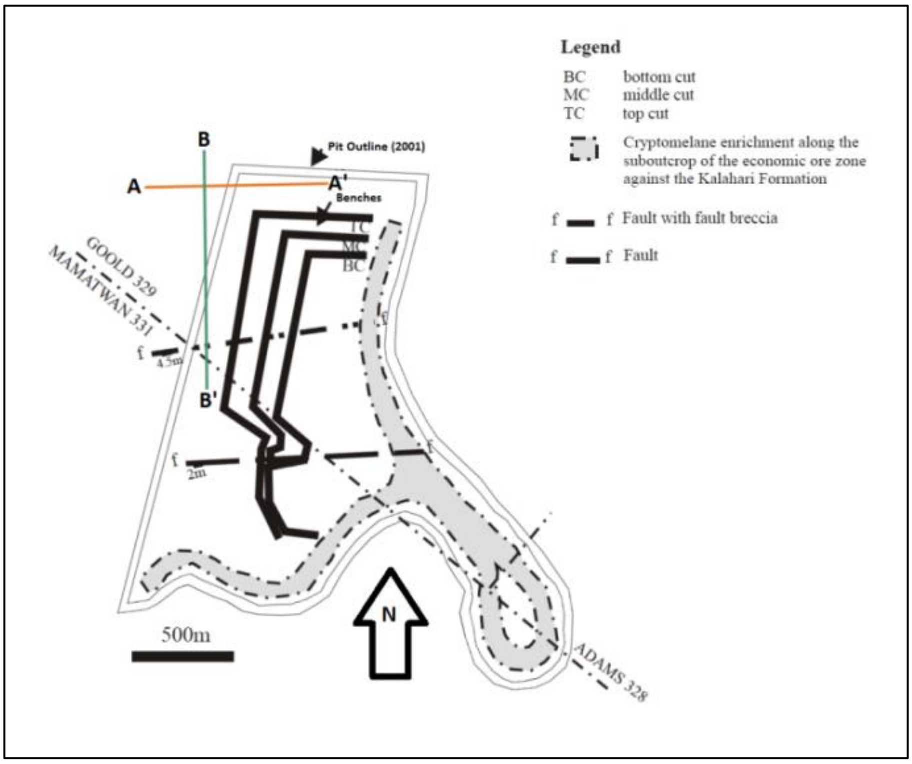


Figure 2.2-5 Mamatwan mine pit design (adapted from Preston, 2001)

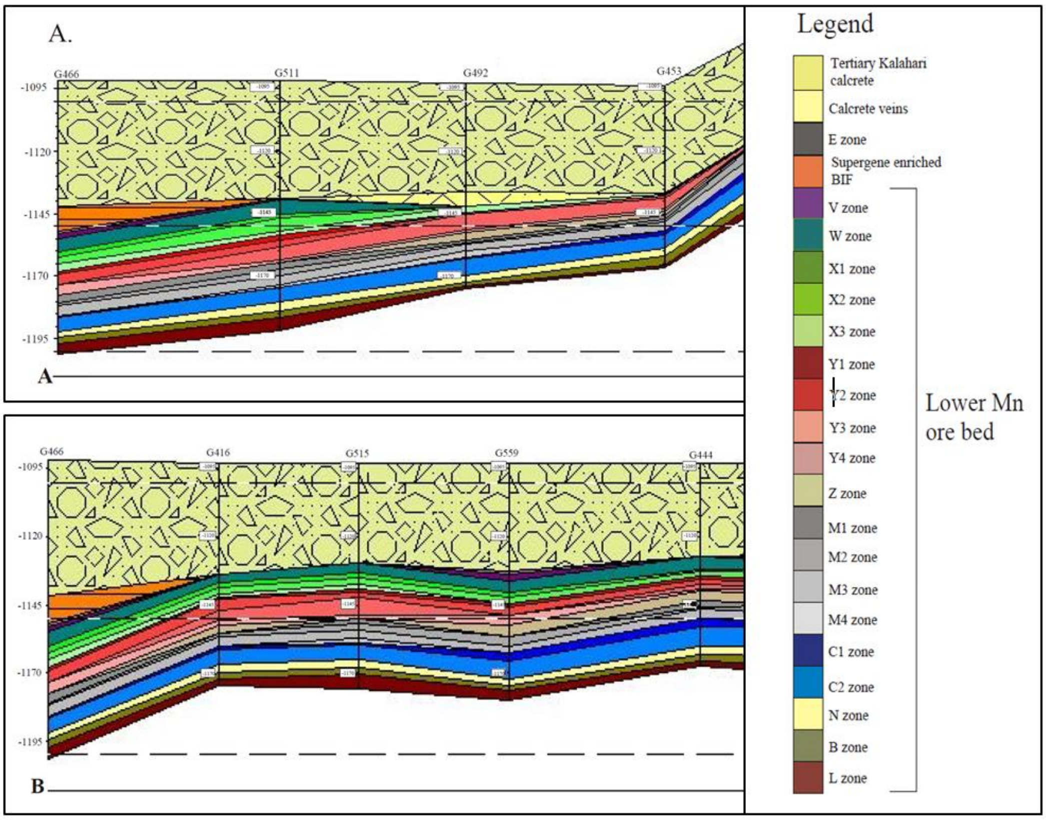


Figure 2.2-6 Diagram A and B presents the stratigraphic variation towards the West and North of the Mamatwan mining pit along lines A and B in Figure 2.2-5 (after Preston, 2001)

2.2.1.3 Mamatwan Ore Manganese Variability

In previous sections, some evidence of Mamatwan ore variability was hinted upon. However, when comparing two drill core samples considered representative of the diversity of the ore to be found in the Mamatwan resource (drill cores G552 and 558), in terms of the most important constituents as far as mining and processing of the ore is concerned, significant variability is revealed against the stratigraphic composition of the two drill cores (Preston, 2001). The most economical deposits (thickest), with the best manganese content, lowest CaO and best Manganese to iron ratio is found in zones M, C and N.

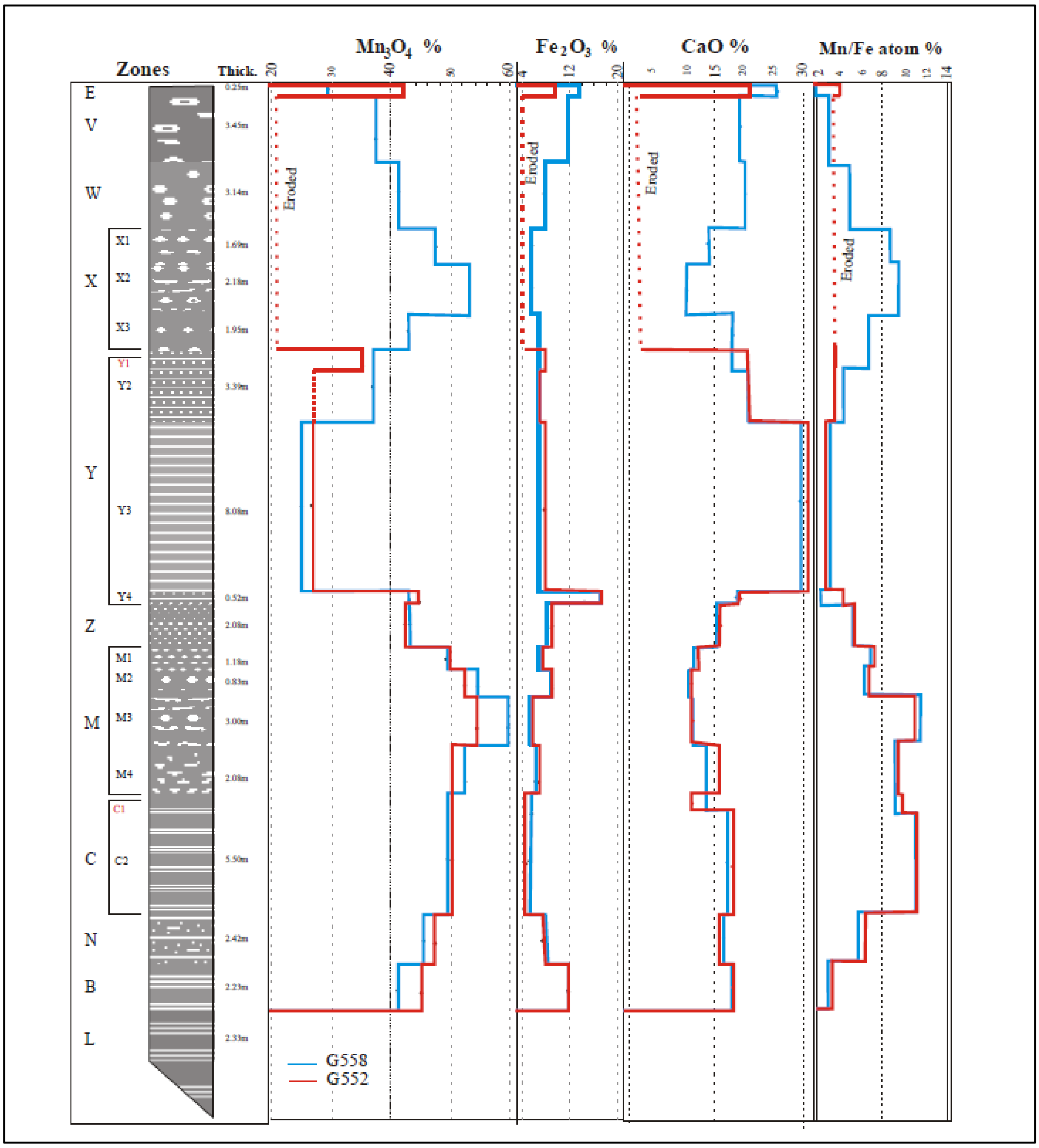


Figure 2.2-7 Mn_3O_4, Fe_2O_3 and CaO concentrations across ore drill cores G558 and G552. The Manganese-Iron ratios for the two drill cores are also indicated (after Preston, 2001).

2.2.2 Manganese Mining and Ore Preparation

The start of the ferromanganese production chain is at the point where the manganese containing minerals in the earth's crust is accessed through mining. Mining can be either open-cast or surface mining, where mineral deposits are not buried at a significant depth beneath the ground surface. Sub-surface mining is employed where mineral deposits are found at a significant depth below the surface. The quality of the deposit will determine the type of mining approach most appropriate, or even if mining is at all feasible. In the future as high quality mineral deposits are depleted and ferromanganese alloy demands rises, previously less feasible mineral resources will become more attractive for mining. This will most probably not occur in our life time due to the volumes of high quality manganese deposits currently available in world, of which South Africa possesses some of the largest deposits in the world, which is estimated to be 78% of the world manganese containing mineral resources (U.S. Geological Survey, 2017).

Mamatwan mine is the starting point of such a ferromanganese production chain, as indicated in Section 2.2 and is a typical opencast or surface mine due to the fact that manganese mineral deposits are found at a relatively shallow depth. Standard slope and berm bench methodology is used to expose Mamatwan manganese ore. Figure 2.2-8 is a diagrammatic presentation of a typical section through the Mamatwan bench, and gives a simplified indication of the different layers of materials and minerals that needs to be processed in order to expose the Manganese ore (Mkhatshwa, 2009). Sand, top lime, middle lime, banded iron, as well as low grade manganese containing zones (zones V through Z) is considered overburden, that needs to be removed to expose the higher grade manganese containing zones (zones M, C and N). Each of the indicated zones displays variation in thickness and mineral (chemical composition), and will be further discussed in Section 2.2.1 of this dissertation. While mining progresses at the front of the pit, as indicated in the schematically diagram, removed overburden is reintroduced to the pit as backfilling. Overburden removal and backfilling is planned in such a way that enough space is left between the front of the pit and the back of the pit to allow ore hauling equipment to operate safely. Mamatwan ore deposits slopes downwards in a westerly direction, which subsequently means that as mining continues the pit will be become deeper. The Mamatwan ore resource will be discussed further in Section 2.2.1.

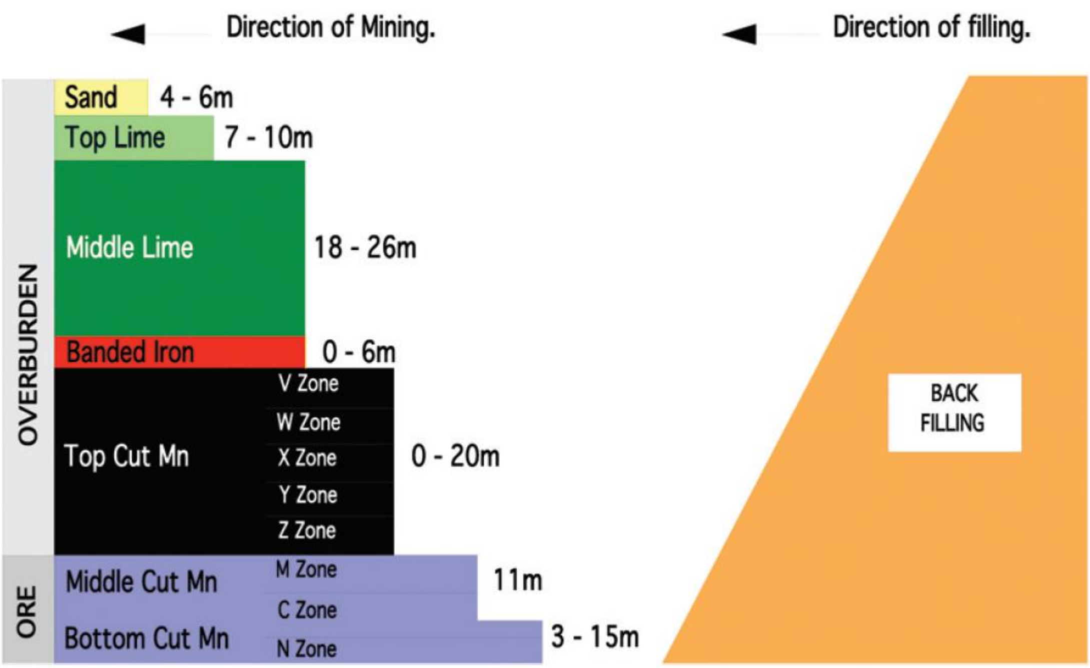


Figure 2.2-8 Schematic diagram of opencast mining practiced at Mamatwan Mine (after Mkhatshwa, 2009)

Once ore is exposed (zones M, C and N), several ore processing steps is required to enable industrial use of Mamatwan ore. Manganese mineral deposits are accessed by drilling and blasting the overlaying none-manganese containing material or overburden, which consists mostly of sand and calcretes (Preston, 2001). After the removal of the overburden, manganese containing mineral deposits is drilled and blasted using mining explosives. Based on the explosives and blasting design, blasted material falls onto the pit floor, which is known as blast-hole material or ore. This material is then recovered and transported to the first stage of ore preparation.

Ore preparation is a very necessary step in minerals processing, as ore removed through mining operations (blasting) consists of both valuable and less valuable minerals in unison. In other words, in the case of Mamatwan mine ore, in one piece of blasted rock, both high manganese content minerals as well as low manganese content minerals may be present. Blast-hole material is transported 80 ton rear-tipping trucks or haulers primary crusher (jaw-crusher) for initial size reduction or crushing. This primary crushing step reduces the size of the ore from a maximum rock size of 700mm to 150mm. After primary crushing, the ore is transported via 2 kilometre conveyor system and deposited on the "run-of-mine" (ROM) stockpile. Figure 2.2-9 presents a picture of Mamatwan mine ore recovered from the ROM after the primary crushing stage. The dark areas are manganese rich (todorokite and hausmannite) whereas the white and pinkish coloured areas consists of gangue minerals (kutnohorite, silicates, etc) can be present in blast-hole material. Figure 2.2-10 is a picture of another sample of Mamatwan mine ore recovered from the same ROM after the primary crushing stage. In this case almost no gangue material is evident. The different minerals present in Mamatwan mine ore is discussed in greater detail in Sections 2.2.1.2 and 2.2.1.3.

Figure 2.2-9 Sample from the Mamatwan ROM stockpile with high mineral composition variability (with permission: Gerry Ray)

Figure 2.2-10 Sample from the Mamatwan ROM stockpile with low mineral composition variability (with permission: Gerry Ray)

From the ROM, material or ore is removed through the use of vibrating feeders, and yet another conveyor system is used to transfers material to a secondary stage of ore preparation which is referred to by Mamatwan mine as the ore preparation plant (OPP). A parallel screening and crushing system now classifies ROM material in terms of particle size. Particles with a diameter greater 75mm is removed by scalping screens and allowed it to be crushed by the secondary crushers (gyrating cone crushers – Nordburg crushers).

Particles with diameters between 25 and 75mm are removed by what is referred to as the lumpy screen and conveyed to the lumpy stockpile for rail transport to ports and smelters. Material that has passed through the crushers are deposited onto a feedback conveyor and delivered to the main incoming feed to the OPP from the ROM stockpile. This allows this freshly crushed material to also be size-classified through the scalping screens until it has a maximum particle size of 75mm.

Directly beneath the scalping screens, material falls onto a double deck screen. The first of the two decks has screening apertures of 40 mm, which means that it only allows particle smaller than 40mm to pass to the next screening deck, which has screening apertures of 6mm, which will allow only particle smaller than 6mm to pass through into the under-pan of the screens. Material between 6 and 40mm is conveyed to the Area 6 stockpile, while material with particle sizes of 25 – 75mm is conveyed to the Area 7 stockpile (Lumpy – M1L). Material collected in the under-pan of the screens then flows to the grid sump which is pumped via a centrifugal pump through a small hydro-cyclone to recover ore washing water as cyclone overflow, which is pumped to the thickener dam for water recovery. The underflow of the cyclone is conveyed and dumped on the OPP fines stockpile. Generally these fines are not sold but discarded. However, if the grade is acceptable may be used as downstream sinter feed feedstock material. It should be noted that amount of fines produced during ore preparation is relatively small; only about 10% fines are produced. But, considering the fact that Mamatwan mine produces 3 million tons of manganese ore per year, this seemingly small percentage of fines material produced over the course of a production year can lead to a significant volume of material (300k tons per annum).

The 6 to 40mm material (Area 6) is then further crushed and screened through a tertiary screening and crushing facility (Kawasaki Crushers), similar to the OPP, accept for the fact that only one product stockpile is produced (DMS feedstock – Area 4). This stockpile is beneficiated through the DMS, which will be discussed in the next section.

Please note that this liberation processes as described in the previous paragraphs, are the current situation at Mamatwan mine and plant. Originally there were no tertiary crushing, screening and washing or any ore beneficiation facilities at Mamatwan, and crushing was only performed at the OPP to produce lumpy ore between 10 and 63mm (Panel in Mn Supply & Its Industrial Implications, 1981). According to this report only Middelplats Mine; a sister Samancor mine of Mamatwan made use of ore washing in addition to crushing and screening to produce similar sized lumpy ore product. The fact that Mamatwan mine's OPP was originally a dry crushing and screening plant is evidenced by a sunken screen under-pan sump indicating that the plant was originally constructed without washing. Its current configuration came about due to plant alterations as industry demands changed over several decades. This will be discussed in the following sections.

The process described above is related to the preparation of ore to ensure suitability for application in downstream processing methods. However, during preparation of the ore, liberation of manganese bearing minerals or differently stated, the removal of gangue minerals from the major manganese bearing minerals are an added benefit, related to the processing steps followed to decrease particle size and subsequent washing and screening steps (Gupta *et al.*, 2006).

All processing where ore is reduced in size, screened and washing can be considered ore, subjected to liberation. Some authors even refer this liberation step as beneficiation due to the fact that this processing method removes materials of low value to some extent, in other words, gangue material from the valuable ore (Panel in Mn Supply & Its Industrial Implications, 1981).

2.2.3 Manganese Ore Beneficiation and Economy

As mentioned in the previous paragraph, crushing, screening and washing processes facilitate the liberation of high value material from low value material. However, liberated ore may still be a mixture of high-value and low-value material. Beneficiation or upgrading in terms of manganese or chemical content by the additional specialised treatment thereof may be required by processing downstream in the Manganese Production Chain. This is related to the requirements of submerged arc furnaces used for the manufacture of ferromanganese alloys which is less efficient when only Mamatwan ore is used directly, and will be explained in more detail in proceeding paragraphs of this section of the dissertation. Mamatwan ore and Mamatwan type ore, which is considered low grade, and has always been available in such vast quantities compared to high grade Wessels type ore (Panel in Mn Supply & Its Industrial Implications, 1981), as indicated in the two Tables below. Table 2.2-3 compares the composition of ore from Wessels, Mamatwan and Middleplaats mines.

Table 2.2-3 Typical comparison of ore from three sister Samacor mines in the Hotazel complex (after Panel in Mn Supply & Its Industrial Implications 1981)

	Wessels	Mamatwan	Middelplaats		
	Lumpy	Lumpy	M1 Lump	M2 Lump + Fines	Ferruginous Lumpy
Mn	+47	+38	38–40	34–36	33
Fe	12–14	4–6	4.0	4.5	5.0
SiO_2	3–5	4–6	5.0	5.0	5.0
CaO	4–6	9–13	14.0	16.0	18.0
MgO	0.4	3.5	-	2.6	3.5
Al_2O_3	0.5	0.5	0.5	0.5	0.5
P	0.05	0.05	0.03	0.03	0.03
S	0.12	0.03	0.04	0.05	0.05

Normal Sizing-Lump -63 mm +10 mm   -75 mm + 6 mm
-Fines -10 mm   -6 mm + 2 mm

SOURCES: Weiss, 1977; Middelplaats Manganese Ltd., 1978; S. A. Manganese Amcor Ltd., 1978

Table 2.2-4 Samancor ore reserve estimates for Wessels and Mamatwan mines (million m.t.) (after Panel in Mn Supply & Its Industrial Implications 1981)

Wessels Grades (+44% manganese)	184.8 proven
	6.0 inferred
	190.8
Mamatwan Grades (38 to 40% manganese)	328.0 proven
	105.0 inferred
	433.0
RESERVE TOTAL	623.8

SOURCE: Rheeder, 1978.

Because of this vast amount of available Mamatwan ore reserves compared to Wessels ore reserves, a trial was performed with a small quantity of Mamatwan ore to upgrade its manganese content via the use of carbo-thermal reduction (sintering) on a trial basis to be used as ferromanganese smelter feedstock. It was thought that upgrading Mamatwan ore through sintering would allow a greater effective portion of Mamatwan ore to be utilised for direct smelter application. Even though the ore grade was improved significantly its mechanical properties was such that normal rail transport and handling at the Meyerton ferromanganese smelter would have caused significant particle-size reduction (Featherstone, 1974), which lead to more fines materials entering smelters and increasing the risk of smelter eruptions. The most recent of such eruptions was at a Katoridge smelter furnace in 2008 (Assmang Limited, 2008).

Again, due to the amount of available ore at Mamatwan mine, more intensive research into the refining of Mamatwan ore as well as several methods of beneficiation. The major methods researched for Mamatwan ore beneficiation were hydrometallurgical and pyro-metallurgical beneficiation (carbon-thermal reduction). The methods eventually implemented in terms of hydrometallurgy, was the DMS process. In terms of pyro-metallurgy a more refined sintering process was implemented than what was previously tested in the 1974 Featherstone trial.

DMS treatment of ore entails the manipulation of the density of liquids, which is related to increasing the density of for example water. This is achieved through the addition of very fine insoluble particle to water. Because of the size of the added particles (40-100µm) a relatively stable suspension is formed. Some refer to DMS treatment as gravity separation (Rosenqvist, 1974).

Two of the main particulate materials used to form a dense medium suspension with water are magnetite, with a relative density (R.D.) of 5.1, and ferrosilicon with a relative density (R.D.) of 6.8. By adding a particulate material in certain ratios to water, suspension densities of between 1250 and 3500kg/m^3 can be obtained (Hayes, 1993).

DMS methods are generally divided into two categories, namely DMS bath- and DMS cyclone-processing.

DMS baths are used for ore with a particle size of between 15 - 100mm, and comprises a relatively static vessel filled with a densified medium or suspension. Material is introduced from one side of the vessel allowing denser material to sink through the dense medium (sinks) while lighter material floats on the surface of the dense medium (floats). Both sinks and floats subsequently separated by the dense medium are removed or recovered by means of mechanical intervention (paddles for floats, elevators for sink – there are other methods for removing floats and sinks).

DMS cyclones are used for ore with a particle size of between 0.5 - 50mm. It is considered more efficient and can be expected to deliver greater material throughput than DMS baths. This improvement in efficiency is achievable by generating a centrifugal force on material particles that flow through a hydro-cyclone. Up to a hundred times the gravitational force of the earth can be generated (100 G) (Hayes, 1993). Figure 2.2-11 presents an exploded view of the hydro-cyclones used at Mamatwan DMS plant. Figure 2.2-12 is a schematic representation of the material flows through a hydro-cyclone.

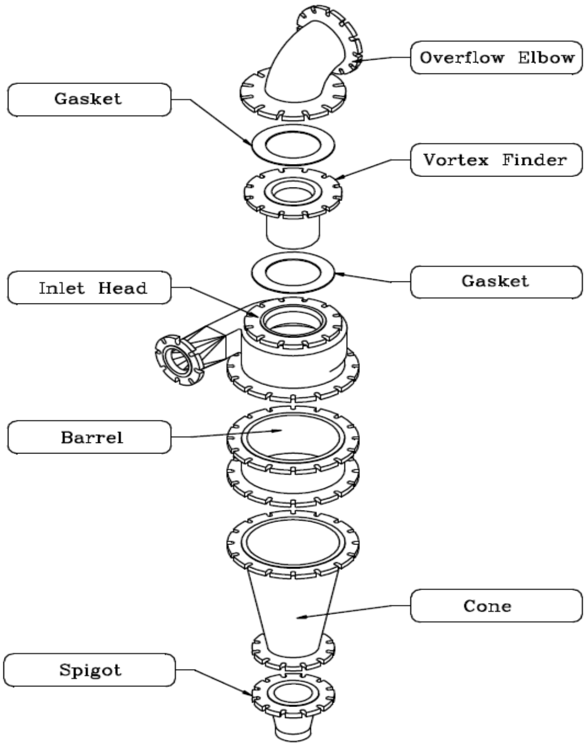


Figure 2.2-11 Typical Mamatwan Hydro-cyclone configuration (Multotec, 2010)

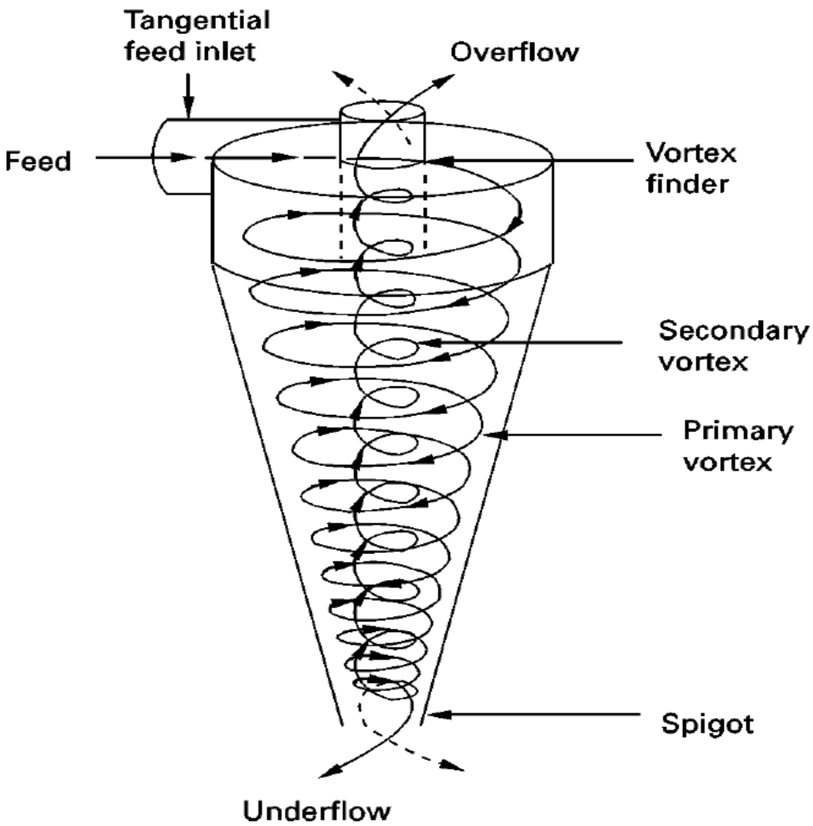


Figure 2.2-12 Schematic presentation of a Hydro-cyclone (after Cilliers, 2000)

To explain the operational aspects of a hydro-cyclone, a cyclone fed with only water and solids will be discussed. A mixture of material (ore) and water is fed through the inlet of the cyclone, either via a slurry pump or gravity fed from an elevated contain or vessel. The inlet nozzle of the cyclone enters tangentially into the head of the cyclone. This type of tangential inlet generates a vortex in the cyclone inlet head, which in turn generates centrifugal forces on particles in this vortex. High density particles will be forced to the outer perimeter of the cyclone inlet head, while low density particles will remain in the central perimeter of the cyclone inlet head. The direction of the primary vortex will be downwards, entering first the cyclone barrel, then the cyclone cone, and lastly exiting through the cyclone spigot. High density particles will remain trapped in the primary vortex and leave the cyclone spigot as underflow. A secondary vortex is generated in the opposite direction to the primary vortex. Mostly water and small low density particles reside in this secondary vortex. Water and material trapped in the secondary vortex flows to the top of the cyclone and through the vortex finder, and exits the cyclone as overflow.

Using only water, particles with a density of 2700kg/m^3 can be easily separated from water (Cilliers, 2000). However, when using a dense medium suspension in conjunction with a cyclone, a much wider range of separation densities for particles can be sufficiently induced by the cyclone system to separate particles with relative densities of 3200 – 4200kg/m^3 (Grobler *et al.*, 2002), making it very applicable to manganese ore beneficiation (upgrading), as manganese ore has a general density of between 2800 - 4200kg/m^3 (Kleyenstuber, 1993).

Due to the variability as discussed in Section 2.2.1.3, cyclone assisted DMS is employed at Mamatwan mine to ensure the required level of Manganese is maintained for the portion of the ROM that is fed into the sinter plant. Mintek performed contract research for Samancor with regards to heavy medium separation for the upgrading of Mamatwan ore and to concurrently manage ore variability. The initial work indicated that ore fines (<6mm) could be beneficiated via hydro-cyclone assisted dense medium separation, but that only low recoveries was possible (40%) (Overbeek, 1984). As part of additional contract work by Mintek for Samancor is was found that higher recoveries was possible (60%) when particles of 10mm were used in the dense medium (Overbeek, 1986). Subsequently the Mamatwan DMS plant was established in 1989 after numerous process refinements specific to Mamatwan ore (Pienaar *et al.*, 1992).

The outcomes of the application of these methods yield an ore-product containing higher percentages of manganese than ore liberated by original mining, and conventional ore preparation activities (crushing and screening) only. Another outcome would be a decrease in the variability in terms of manganese content due to the removal of gangue-mineral-rich particles. A negative outcome of beneficiation as described via DMS, would be the generation of waste. In the case of Mamatwan mine, the DMS discards material containing a 36% and lower manganese content, while retaining material with a 37% and higher manganese content. This can in some cases lead to 40% or more of the ore to exist the DMS plant is discarded (Overbeek, 1986). This implies that 40% of the costs incurred during ore extraction and preparation is lost.

The next step in the ferromanganese production chain as indicated in Figure 2.2-1 would be the production of ferromanganese alloy through an additional reduction treatment via submerged-arc furnace (smelter). However, when considering the use of manganese ore for ferromanganese alloy production it must be stated that using Mamatwan manganese ore as the sole ore component in furnace burdens, it was found that extremely low electrical resistivity's (mΩ) exist for the respective mixtures of Mamatwan ore and Delmas coal, and Mamatwan ore and Iscor coke (Koursaris, 1980). It was found that the both coke and coal mixed with Mamatwan ore only, displayed extremely high resistivity values at temperatures lower than 1200°C. This implies that low furnace electrode resistivity will only be present at the electrode tips of the furnace where material is molten, while furnace burden at higher levels in the furnace will be at lower temperatures and high electrical resistivity which will lower furnace efficiency in terms of electricity consumption. Because of the mentioned efficiency reasons, even from the onset of submerged-arc furnace use where Mamatwan ore was used as part of a mixture of high-grade Hotazel mine ore (55% Mn content) (Selmer-Olsen, 1974), to increase smelter efficiencies by lowering electrical resistivity at higher levels in submerged-arc furnaces, and improving electrical power consumption. Even though these tests were performed with very high grade Hotazel mine ore, by 1981 only Mamatwan and Wessel ore were available as reported in world resource summaries (Panel in Mn Supply & Its Industrial Implications, 1981). It must be stated again that Mamatwan is an open cast mining operation, whereas Wessel mine is an underground mining operation. Significant differences in operational costs and products are inferred. As reported, Wessels ore are of a significantly better grade than Mamatwan which also made is more suitable for submerged-arc furnace raw material application (furnace burden component), and a larger submerged-arc furnace burdens comprised a larger amount of the more expensive Wessels ore than Mamatwan ore.

As already indicated, Featherstone considered the use of Mamatwan mine fines for sintering (carbo-thermal reduction) to increase its manganese content to closely approach that of Wessels ore and therefore allow more Mamatwan product to be used in submerged-arc furnace application and decreasing submerged-arc furnace operational costs. This would ensure the then Samancor owned entities could be more competitive in terms of operational effectiveness and diversification of its product range. Studies were initiated over the course of two decades that supported the establishment of a sintering facility at Mamatwan mine in 1987 (Pienaar *et al.*, 1992). However, the establishment of a sinter plant at Mamatwan mine was preceded by in-depth research into carbo-thermal reduction of Mamatwan fines.

One such an in-depth study was related to the mechanism and rate of carbo-thermal reduction of Mamatwan ore (Grimsley *et al.*, 1977). In this study the reactions possible during carbo-thermal reduction (sintering) was studied at temperatures ranging from 1000 to 1300°C. Graphite was used as a reduction agent, in mass percentage additions of between 5 to 20%, in increments of 5%. The influences of increase in reduction agent at the mentioned temperature ranges were plotted in terms of percentage reduction against time at certain temperatures, relative to mineralogical changes that was thus facilitated. Figure 2.2-14, Figure 2.2-14 and Figure 2.2-15 below present the reduction agent addition influences at 1000 and 1300°C. Finally, the reduction reactions predicted by pre-developed reaction models were compared with the experimental results of the study. Even though a good approximation existed between the calculated reduction reactions and empirically determined reduction reaction values, the deviations as presented in Figure 2.2-15 were attributed to variations in particle-size-distribution, which is again related to the surface area available for carbo-thermal reduction reactions to take place.

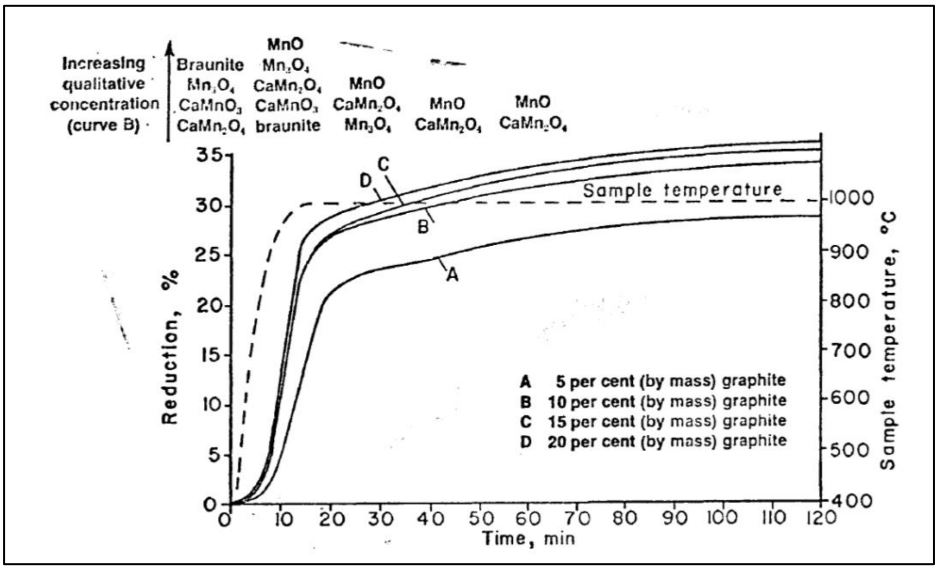


Figure 2.2-13 Relative reduction percentages for different reluctant (graphite) additions to Mamatwan ore. Argon atmosphere and temperature of 1000°C was maintained for the purpose of pre-heating the samples. Mamatwan ore samples of 16g were used during the reduction experiment (after Grimsley *et al.*, 1977).

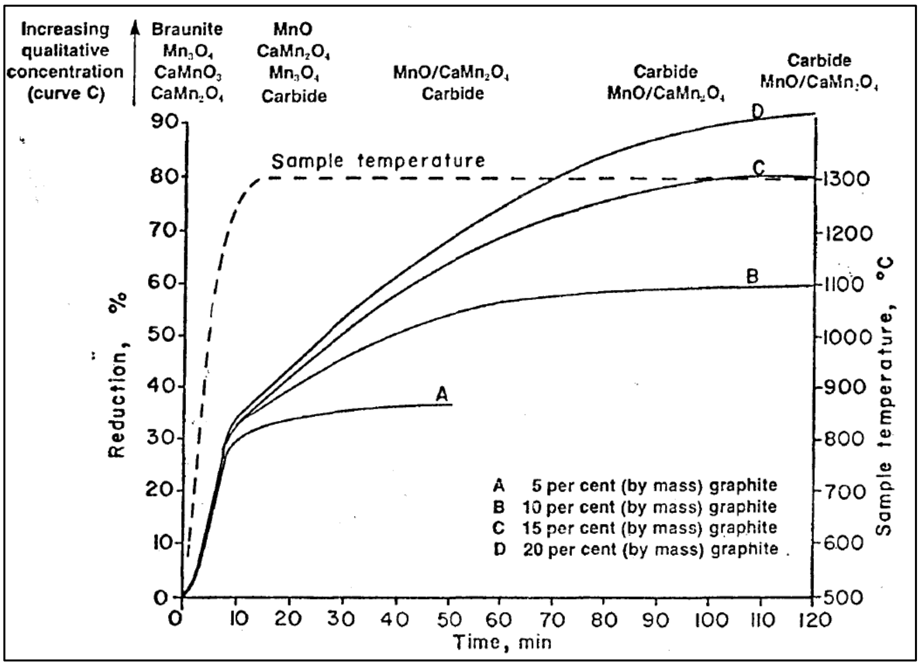


Figure 2.2-14 Relative reduction percentages for different reluctant (graphite) additions to Mamatwan ore. Argon atmosphere and temperature of 1300ºC was maintained for the purpose of pre-heating the samples. Mamatwan ore samples of 16g were used during the reduction experiment (after Grimsley *et al.*, 1977).

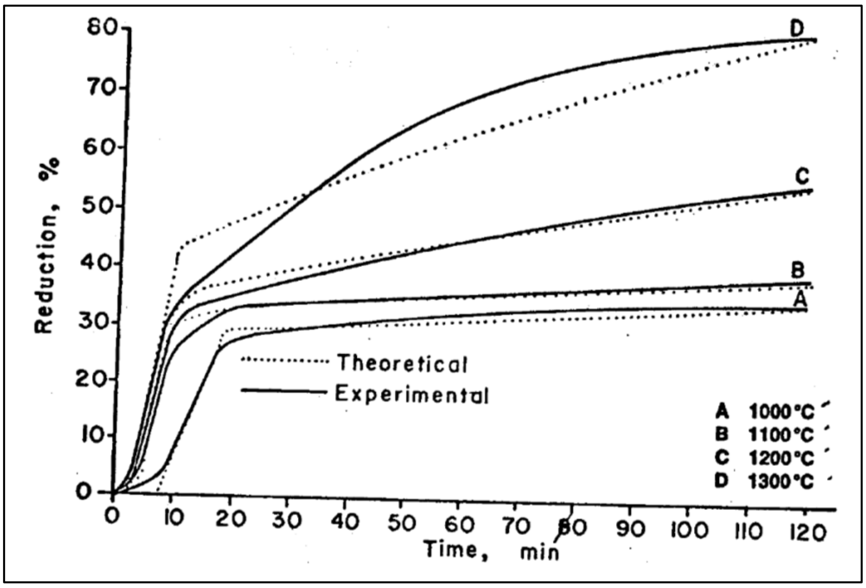


Figure 2.2-15 Comparative plots of percentage reduction of Mamatwan ore with the addition of 15% (by mass) graphite, at different preheating temperatures (A, B, C and D), (after Grimsley *et al.*, 1977).

Similarly, submerged arc furnace experimental studies were performed for mixtures of Mamatwan manganese ore and several reducing agents. The resistivity of these mixtures was measured to determine the feasibility of the use of Mamatwan ore for arc furnace smelters (Koursaris, 1980).

These studies indicated positive outcomes in terms of sintering and the use of Mamatwan ore in ferromanganese production chain.

It is important to again highlight that one of the purposes of sintering ore, is to enable use ore fines in smelter. It was mentioned before that fines materials in smelters can lead to catastrophic incidents. It was also mentioned that even the handling (commutation) of sintered ore fine may create unacceptable fines introduction into smelters. One typical facility that can be used as a case study when considering the installation of a sinter plant at Mamatwan mine was Tokushima works, Nippon Denko Co. Ltd. The significant driver or motivation for the sinter (carbo-thermal reduction) facility at Tokushima Works was the use of manganese ore fines (Naruse, 1974). It was found that lump ore combined with sintered ore fines improved the permeability of smelter loads.

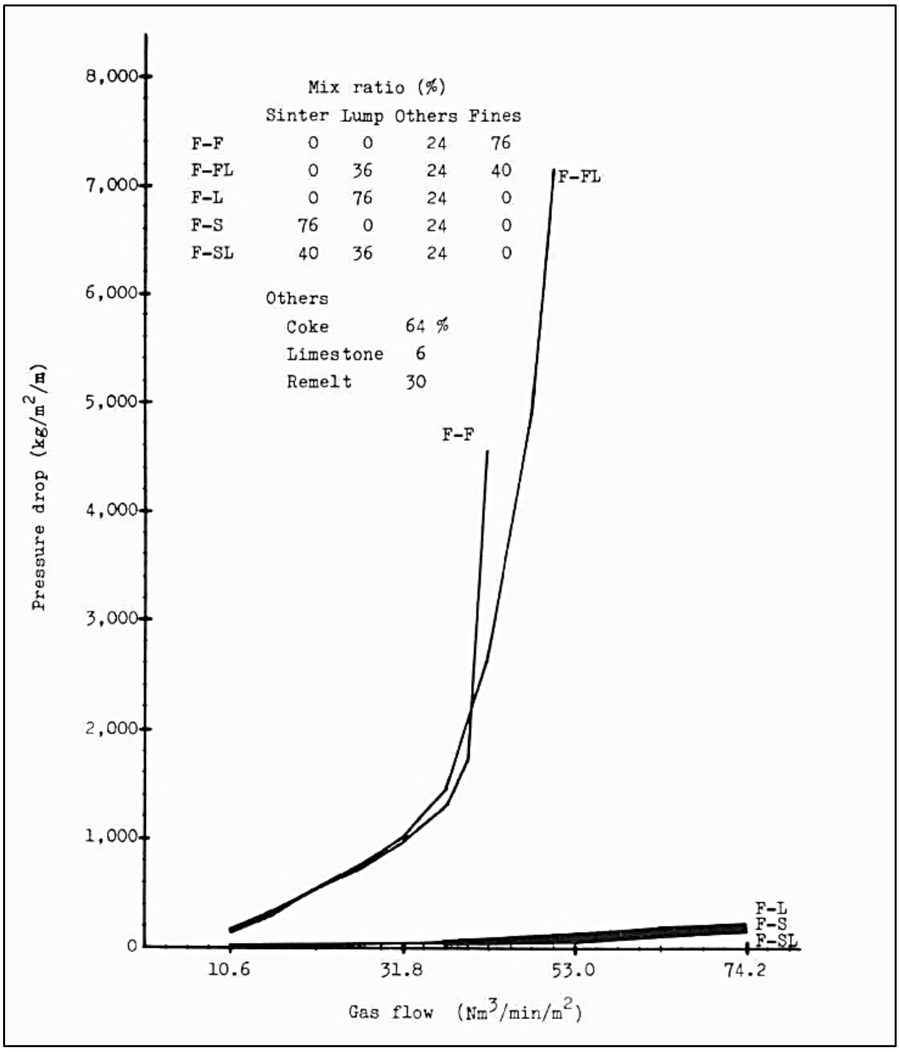


Figure 2.2-16 Plot of pressure drop against gas flow through the cap of a typical furnace loads (permeability of the furnace cap). Each graph was generated for a specific mixture of raw feed materials. All mixtures that included sintered ore fines indicated high permeability compared to mixtures containing ore fines and no sinter (after Naruse, 1974).

Other drivers considered as benefits, brought about by the use of sinter as smelter feedstock at Tokushima Works, can be listed as follows:

- Furnace load and quality of product produced

- Decreased dust emissions
- Smelting at lower temperatures (lower energy consumption)

In summary, the application of a sintering (carbo-thermal reduction) step in the manganese-alloy production chain, was therefore proven to add significant value not only in terms of efficiency, but also in terms of pollution and waste generation.

2.3 Mamatwan Mine Products, Facilities and Evolution

Mamatwan mine has over the past five decades experienced several additions in terms of the products made available through its mining and processing facilities. Even though export of none-beneficiated ore have never ceased, additional processing facilities added more product lines, which has allowed the Mamatwan mine and plant asset to be capable of complying to more diverse market conditions. Differently stated, a single ore resource can therefore supply several products through different processing methods and facilities.

2.4 Multiplicity of Mamatwan Ore Handling in Summary

In the previous sections it was indicated that ore variability exists at Mamatwan mine, and that ore variability can have an effect on the operational costs of ferromanganese production chains. This effect is mostly related to additional handling steps required to ensure a smelter charge produces more alloy verse slag. This equates to a higher alloy-to-slag-ratio, which yields greater production rates, and production stability.

It has also been established that preparation of ore, produces ore fines of sufficient quality, but also significant risk when used in ferromanganese smelters due to mentioned furnace eruptions.

At Mamatwan both these aspects were addressed via the application of a DMS facility reduces ore variability to minimum before such ore is treated via carbo-thermal reduction (sintering)

Another benefit of carbo-thermal reduction is the fact that the ore is beneficiated by this step, which increases the manganese content of from 37% in the ore to 45% in the resulting sinter. This increases the effective manganese ton units transported via rail to smelter facilities with obvious financial benefits.

Since it has already been established that variability in the feedstock of smelters does not lead to the most effective smelting process, the management of such variability has been stated.

There are several methods employed in not only the mining steps but also ore alteration steps of ferromanganese alloy production chains. The typical methods will be reviewed in the following paragraphs, and can be divided into two main categories, namely, mechanical methods and density separation methods.

2.4.1 Alternative Variability Management Methods

The methods employed by Mamatwan mine to manage ore variability have been discussed. However, there are other methods also used in the mining industry to manage ore variability. These methods are related to blending of higher and lower grade ore to obtain an optimum resulting ore grade. Blending can be sub-categorised into blending yards, blending silos and in-pit-blend-mining practices.

Blending yards rely upon the availability of expansive surface areas for establishing stockyards, as well as sufficient chemical or compositional analyses capabilities, which then enables a stockyard management system to catalogue all material in the stockyard. The availability of the stockyard compositional data allows the reclaiming of material from the stockyard to be such that ore variability is significantly limited. In other words, material in the stockyard is reclaimed such that blending of the material in the stockyard yields a stream of material with a consistent composition. However, a stockyard which allows blending whilst reclaiming, requires even larger surface areas to allow for enough stock to be available for continues reclaiming.

A typical blending yard as presented in Figure 2.4-1 would entail die following operational step (Zhu, March 2013):

(1) Ore of which the content is already known is conveyed and stacked to the one-time yard ore or Area 1
(2) Ore stacked in Area 1’s composition is verified via analyses
(3) According to verified data of step (2), an ore reclamation strategy is determined for stacked ore in Area 1 (red block)
(4) Ore is then reclaimed from Area 1 as per the pre-determined reclamation strategy mixed through the mixing tanks and staked onto Area 2 (green block)
(5) Blended ore is then reclaimed from blending stockyard for sintering.

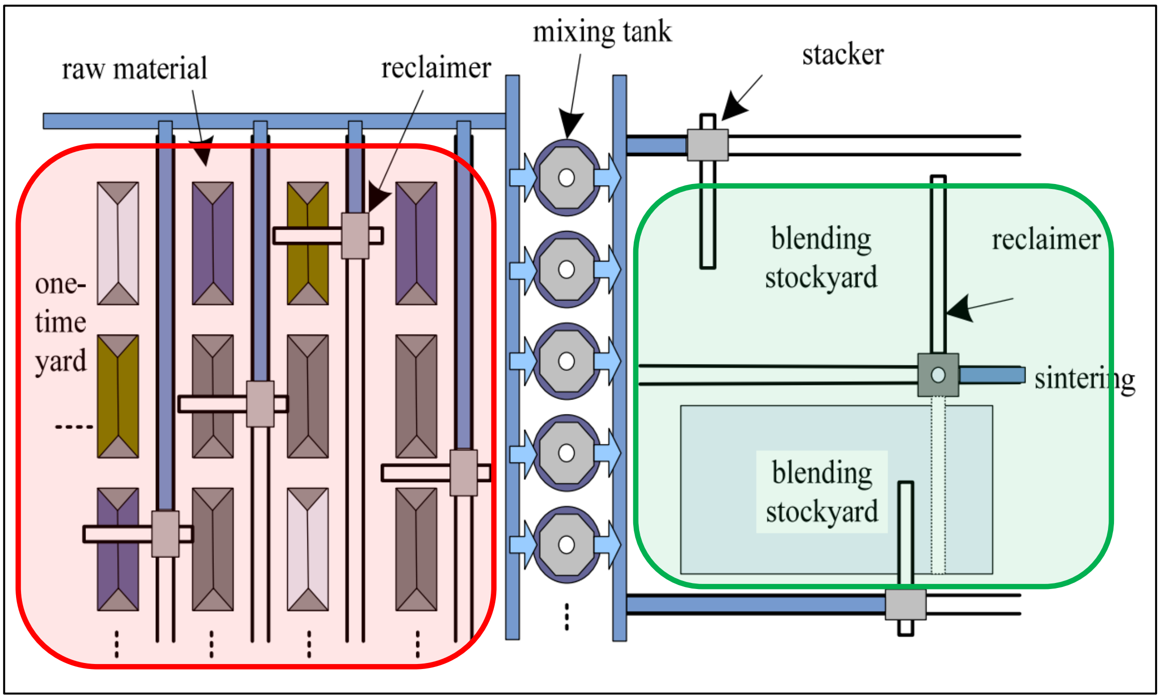


Figure 2.4-1 Diagrammatic presentation of a blending yard; Area 1 (red block) relates to ore of specific or known compositions that is stockpiled for controlled reclamation, blending via mixing tanks and a subsequent mixing field by stackers followed by a second reclamation before being conveyed to the sintering plant (Area 2 – green block) (adapted from Zhu, 2013).

Blending silos are similar to blending yards but stores stock within numerous silos. The successive silos will each store ore with respectively increasing quality. When blending is required, ore from the respective silos are delivered on a single conveyor via variable speed feeders and relative feed ratio to ensure variability is managed. Such a trial was performed at Black Mountain, a division of Anglo Operations Limited and made use of five blending silos to blend 13 994 tons of ore containing a variety of copper, lead, zinc and silver (Williams *et al.*, 2001).

Yet another mechanical method of managing ore body variability is referred to as Blend Mining (Gnoinskig, July 2007). In this method, high resolution geological data is required. However, in Blend Mining, high quality ore is not targeted as such. Its premise is based on the sound understanding of the ore body, the blasting or liberation of large volumes of this ore body and thereafter the collection of blasted material in such a way that blending of lower and higher grade material is such that variability management is achieved. In other words, knowledge of the blasted material is of such a level that collection and hauling of the material from the mine pit can be directed in such a way that the higher and lower grade material can be blended, even before the first crushing and screening step takes place. This method does require substantial laboratory analyses capabilities, since blasted (blast-hole) material must be sampled to confirm the composition of thereof before the blending designs can be performed. The benefit of this method is the fact that the requirement for large stock- and blending yards are omitted. Therefore, the space requirements for this type of variability management method are not as great as in the case of blending yards.

2.4.2 Variability Management and Mamatwan

Several variability management methods were reviewed in the preceding Sections of this dissertation, and even though it was indicated that Mamatwan Mine has been using a density separation method to control ore variability since 1989 in Section 2.2.3, recent world markets are continuously putting pressure on ore and other commodity prices. Typically events such as the extremely high steel prices experienced in 2008 with a subsequent deterioration of prices in 2009 coupled with a very slow recovery until 2011 followed by decline in steel prices up until 2016. Volatility in steel prices seems to be continues in the near future (SteelBenchmarkerTM, 2018) . As manganese alloy prices are related to steel prices due to ferromanganese being used in steel refining and even scrap steel recovery, greater pressure is placed on manganese ore prices, which has prompted high level mine management to review operational costs and question the current methods of processing ore. This included the current methods of variability management at Mamatwan (DMS).

Based in the above mentioned, a variability management review was performed to identify a method other than the DMS for managing ore variability.

2.5 Mamatwan Constraints and Alternative Methods

Based on preceding discussions regarding methods of variability management, it is important to review certain constraints that exist at Mamatwan mine and plant. These constraints are important considerations when the development of possible solutions and the selection of the best possible alternative variability management method are performed.

The Mamatwan mine and plant was first put into service in 1963. At that stage with the knowledge at hand, it led to a certain plant layout and design philosophy. The plant was initially a dry crushing and screening facility with no DMS or sinter plant. Due to downstream production chain requirements, first a sinter plant (1987) and later a DMS (1989) was added to the Mamatwan processing facilities. The implementation of the DMS required the addition of wet screening and crushing at the Secondary ore preparation plant. This change in the processing method was required to decrease ferrosilicon losses and improve ferrosilicon stability during the DMS processing of ore by removing very fine particles from the processed ore (Grobler *et al.*, 2002). With the implementation of the DMS and wet crushing and screening the addition of a thickener as well as slimes dump was required for storage of these newly produces slimes or waste. This material was pumped to Adams pit (Figure 5.2-5). After the implementation of these facilities, no space was left for additional facilities.

Therefore, these constraints can be listed as follows within Mamatwan mine boundaries (please refer to Section 3.4.3 for detailed aerial pictures and diagrams:

- Space availability for additional facilities.
- Existing access routes and existing plant infrastructure will make the implementation of additional facilities problematic.
- Waste dumps for DMS discard and slimes storage prohibits any facility expansion that may interfere with waste routes.
- Overhead conveyor systems associated with existing facilities, as well as rail infrastructure also limits expansion.

The listed constrains are the most obvious considerations that needs to be taken into account when considering an alternative ore variability management method.

2.6 Alternative Method Approach

As already mentioned, an alternative approach to the existing DMS ore variability management at Mamatwan is required, and as mentioned in Section 2.2, and specifically Sections 2.2.1.3 and 2.2.3.

The implementation of an alternative approach will, due to the nature of a mining and ore preparation and beneficiation plant require the application of engineering methodologies and design to select and test and alternative to the mentioned DMS facility.

Traditionally engineering methodologies and design requires several minimum supporting activities to ensure the success. These minimum activities can be summarised as activities such as decision making, modelling, knowledge or data generation, prototyping, ideation, and evaluation or testing (Adams, 2015).

The following sections of this dissertation will elucidate on aspects of engineering methodologies and design as indicated above.

2.6.1 Option Selection and Decision-Making

Traditionally, when option or concept selection is performed, emphases are placed on “superior” knowledge in for instance engineering, and more specifically, for example, chemical-, mechanical-, and electrical engineering. This is also true for financial and business decision making. However, in terms of this project, such an approach is less applicable due to the fact that the entire manganese production chain under consideration already exists, and has existed for several decades. Therefore, the Option Selection need to take existing infrastructure, continued production, operational personnel and uncertainty into consideration. Three common elements are associated with making decisions under conditions of uncertainty, and can be summarised as (Albright *et al.*, 2009):

- A set of strategies (options) available to decision makers
- A set of possible outcomes and probabilities of outcomes
- A value model that prescribes values to various outcomes as foreseen for decisions made

To ensure that the best possible Option Selection approach is followed, several engineering design approaches for option or concept selection methodologies will be reviewed. Option or concept selection methods are important tools in ensuring not only the best concept or option, but also the most implementable option within the mentioned constraints, not only for engineering applications, but also for general projects, investment and business related options, where decision making is performed. To be able to decrease the risks associated with Option Selection, models for decision making is more and more frequently being used in engineering projects and allows the review and testing of concept options before full implementation (Blanchard, 2006).

2.6.2 Option Selection and Decision Making Support Methods

When attempting to develop a product or manage a project that has outcomes related to a product, service of improvement, a structured approach is required. This approach will be such that it supports Option Selection and decision making. Based on this, several approaches or feature that supports design and project management approaches should be integrated into the processes of design and project management, as presented in Table 2.6-1 (Adams, 2015), which considers stakeholder requirements, requirements analyses, design, implementation, integration, verification, transition and validation.

Table 2.6-1 Several features that supports design (after Adams, 2015)

Feature	Description
Exploratory	Design is a formal professional endeavor requiring specific knowledge, skills, and abilities
Rational	Design is rational involving logical reasoning, mathematical analysis, computer simulation, laboratory experiments and field trials, etc
Investigative	Design requires inquiry into the stakeholder's requirements and expectations, available design techniques, previous design solutions, past design failures and successes, etc
Creative	Design requires know-how, ingenuity, memory, pattern recognition abilities, informed solution scanning, lateral thinking, brainstorming, analogies, etc
Opportunistic	Both top-down and bottom-up approaches are used by the design team based upon the situation presented
Incremental	Improvements or refinements are proposed during the design process in order to achieve an improved design
Decision-making	Design requires value judgements. Courses of action and selection from competing solutions are based on experience and criteria provided by the system's stakeholders
Iterative	Design is iterative. Artifacts are analyzed with respect to functional and non-functional requirements, constraints, and cost. Revisions are based on experience and feedback mechanisms
Transdisciplinary	Design of engineering systems requires a transdisciplinary team
Interactive	Design is interactive. The design team is an integral part of the actual design

The approach used in project management effectively integrates these processes as part of a phased project management methodology, where each phase of the project gathers more information, decreasing uncertainty and allowing decision-making to take place (Steyn, 2015). Figure 2.6-1 presents a schematic presentation of a phased project approach that also integrates the features as structured in Table 2.1-1.

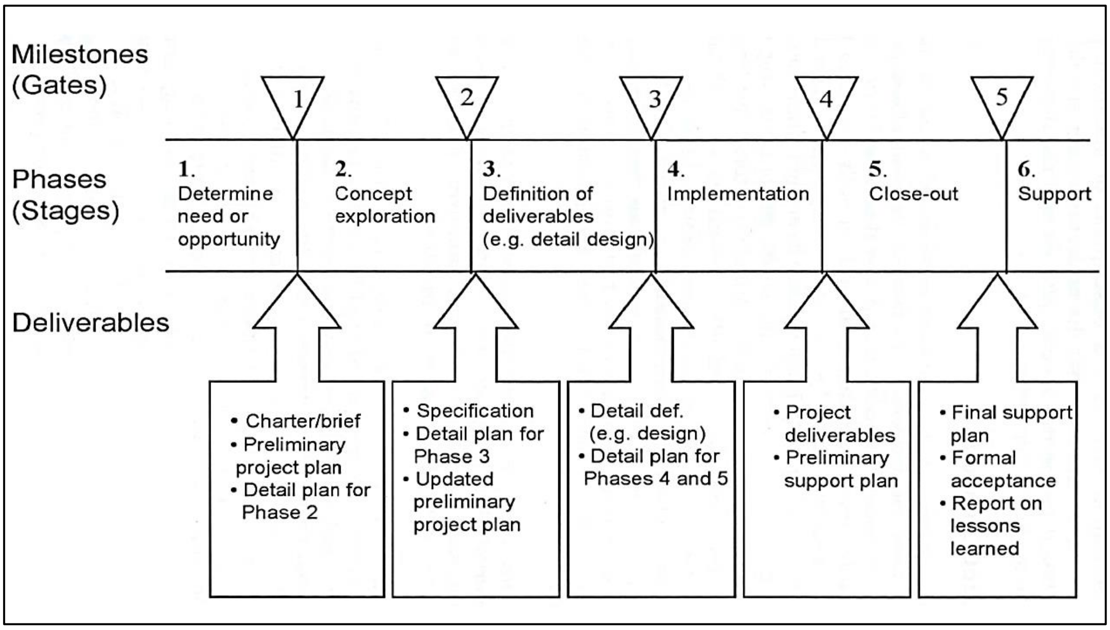


Figure 2.6-1 Example of typical phases of a project (after Steyn, 2015)

It is clear from the preceding table and figure that design and project teams need to follow a certain way of thinking or directing thoughts to allow the achievement of satisfying the original need or specification.

Analyses of design supporting features and project phases is indicative of a pertinent though process and has been termed the Double-diamond Model of Design, where divergence and convergence of thoughts, ideas and concept occur, which then effectively or decisively indicates problems or needs as well as solutions, options or concepts. In other words, initially thought diverge by being are exploratory, rational, investigative, etc. and eventually converges to the point of decision making. It is important to apply such a model to finding both correct problem and an associated solution.

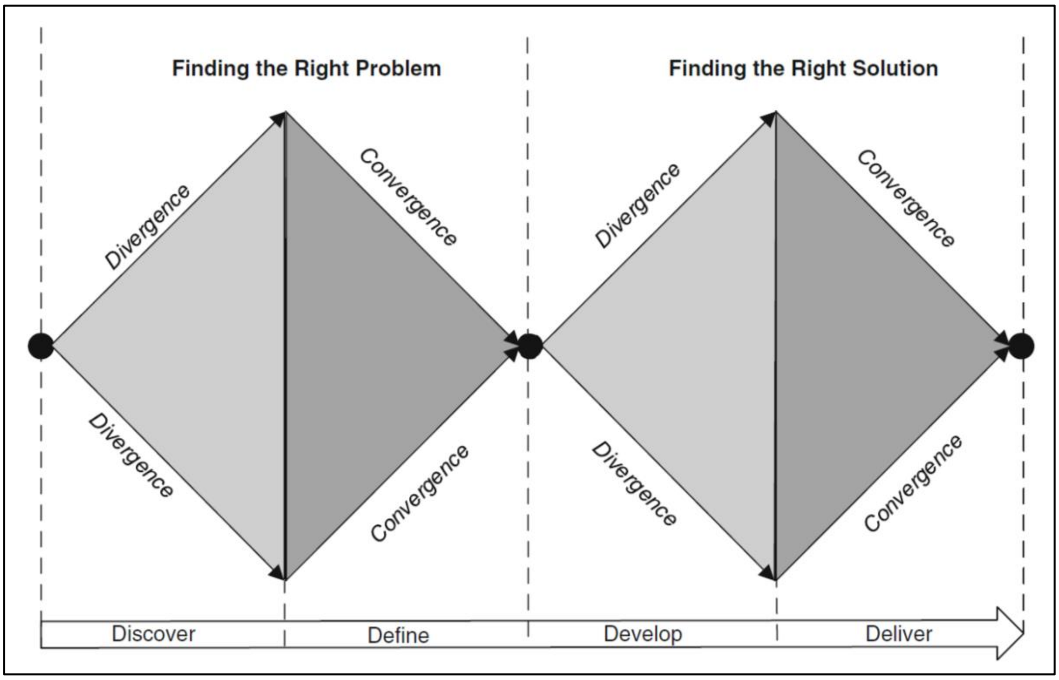


Figure 2.6-2 Schematic of the Double diamond model of design (after Adams, 2015)

Due to the analogies between design and project management, several design methodologies will be briefly reviewed. Since there are a significant number of design mythologies used in engineering and project disciplines, only some of the generally used methodologies will be included in this literature survey, and can be listed as follows (Adams, 2015):

- Methodology by Morris Asimov
- Methodology by Nigel Cross
- Methodology by Stuart Pugh

2.6.2.1 Morris Asimov Design Methodology

In 1962 Professor Morris Asimov published a structured methodology for application in design projects. This methodology is an approach consisting of a 7 linear phases that follows in a chronologically on one another. This approach is preceded by the formulation or identification of a need, following by a three design phases, which then feeds into four production and so-called consumption phases (Asimow, 1962). There are many similarities between this methodology and phased project management approaches. It would therefore be apt to refer to the production and consumption phases as execution and operational phases. Figure 2.6-3 is a schematic presentation adapted from the Morris Asimov design methodology.

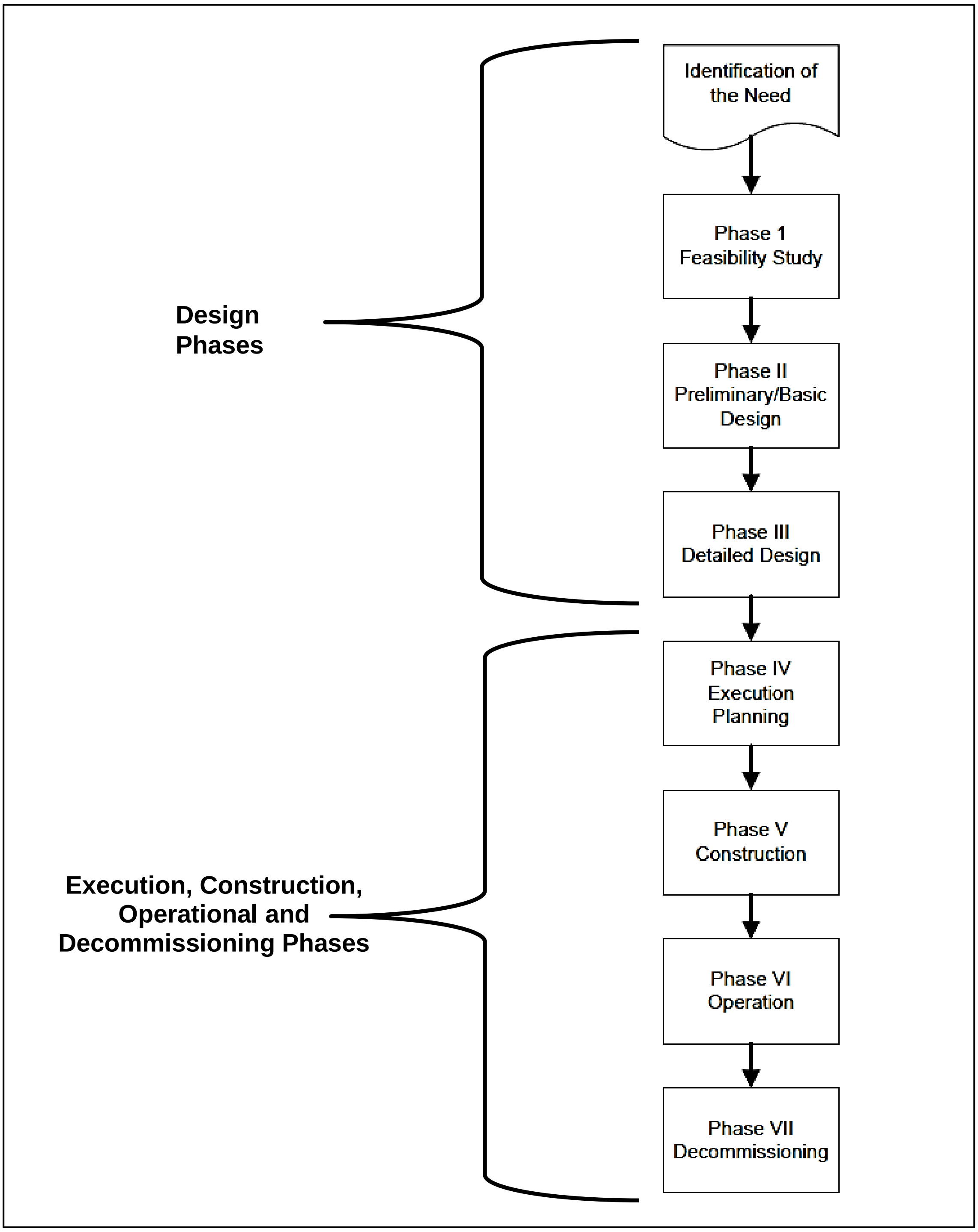


Figure 2.6-3 Phases of a design project [adapted from (after Asimow, 1962)

Of specific interest to the project presented in this dissertation are Phases II and III, namely preliminary design and detailed design. During preliminary design the emphasis is generating and selecting concepts, which the detailed design focuses on describing the concept in engineering terms and the testing thereof.

The remainder of phases in Figure 2.6-3 is related to the different phase of the execution of the project and is related to construction or production, operation and decommissioning of the design project's outcomes. Again, this is very similar to current standardised project management methodologies (PMBOK, 2013).

2.6.2.2 Nigel Cross Design Methodology

Emeritus Professor Nigel Cross published a rather unique approach to engineering design projects, and stated an eight stage model for design. This approach differs from the Morris Asimov approach in that it is not linear in it progression, but circular. Therefore, it contains numerous direct and indirect feedback in terms of addressing primary- and secondary problems as well as primary- and secondary solutions while following eight stages of design which can be listed as follows (Cross, 2008):

- Objectives
- Functions
- Requirements
- Characteristics
- Alternatives
- Evaluation
- Improvements
- Opportunities

These stages of design can occur iteratively and could be applied to feasibility, basic and detailed design. This approach does require continues cognisance of primary- and secondary problems as well as primary- and secondary solutions which should ensure that problems and solutions are aligned.

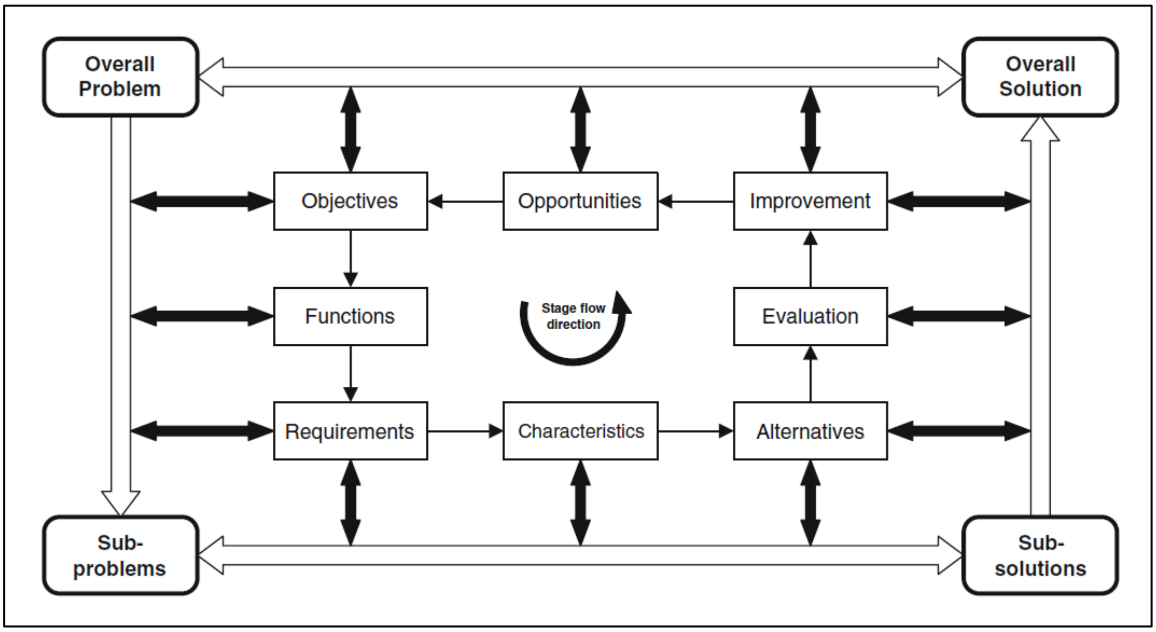


Figure 2.6-4 Stages of the design process as (adapted from (Adams, 2015) and (Cross, 2008))

2.6.2.3 Stuart Pugh Design Methodology

Stuart Pugh considered the design process that traditionally was the domain of engineering disciplines only, to be less successful in ensuring the achievement of outcomes of both technical and none-technical aspects of products or design projects.

Based in this consideration Stuart Pugh developed a methodology for design that is not only focused on engineering disciplines as such, but all other relevant disciplines with in project teams. His approach was thus highly trans-disciplinary in terms of its structure and facilitation of product of design project phases, which he presented in his publication; Total Design: Integrated Methods for Successful Product Engineering (Pugh, 1991).

Stuart Pugh did not relinquish the phased approach to design projects but rather incorporated additional aspects that could assist in making decisions taking technical and non-technical factors into consideration. These considerations forms part of the Option Selection criteria and is applied in a four part approach. The first part is breaks the design process into six general phases:

- User needs definition
- Product specification (this could also be construed as project outcomes and outputs)
- Conceptual design
- Detailed design
- Manufacturing, construction or generally referred to an execution
- Sales or operation, depending on the outcomes of the design project

The second part of Pugh's approach is the compilation of a product design specification (PDS). This so-called PDS encompasses primary outcomes of the design project and contains the elements required for satisfying the needs originally defined for the product of project. Important elements that should be contained in a typical design project PDS is presented in Table 2.6-2.

Table 2.6-2 Typical elements contained in a product design specification (PDS) (after Adams, 2015)

Customer	Processes	Size	Shipping	Performance
Disposal	Aesthetics	Politics	Installation	Weight
Maintenance	Competition	Packing	Reliability	Shelf life storage
Patents	Environment	Testing	Safety	Legal
Documentation	Quality	Product lifespan	Materials	Ergonomics
Standards specifications	Manufacturing facilities	Market constraints	Company constraints	Life in service
Product cost	Time scale			

Thirdly, the Stuart Pugh approach considers inputs from discipline-independent elements, in a divergent and convergent manner, similarly as presented in Figure 2.6-2. As the last step in the Stuart Pugh approach inputs from all discipline-dependent elements, in a divergent and convergent manner. Again, similarly as presented in Figure 2.6-2.

While this schematic do provide a means for understanding the Stuart Pugh methodology for design, its implementation requires the use of matrixes that combines PDS element as selection criteria, against which concept options are evaluated. The basic Stuart Pugh matrix of which one of the concept options is taken as a baseline or base-case. The concept options are then compared in terms of being similar (S), better (+) or worse (-) that the baseline concept and is indicated with the associated symbols, "S", "+", and "–" in the matrix. Table 2.6-3 where one of the concept options was indicated as the best option as compared to the baseline concept option. Such selection matrixes have been used in numerous design selection applications where Option Selection was performed through the use of Stuart Pugh selection matrixes for six design cases for deep-sea equipment (Muller, 2011).

The Stuart Pugh methodology is considered to be generally successful in design of component and was tested in automotive product development, and was used to define the design process into systematic activities that lead to client satisfaction (Villanueva, 2016).

Table 2.6-3 Simple Stuart Pugh Concept Selection Matrix

Selection Criteria \ Concept Options	Concept 1	Concept 2	Concpet 3	Concept 4
Criteria 1	S	S	+	Baseline
Criteria 2	S	+	+	
Criteria 3	S	+	+	
Criteria 4	S	+	-	
Criteria 5	S	-	-	
Total "+"	0	3	3	
Total "-"	0	1	2	
Total Score	0	2	1	
Option Ranking vs. Baseline	Similar to Baseline	Better than Baseline	Worse than Baseline	

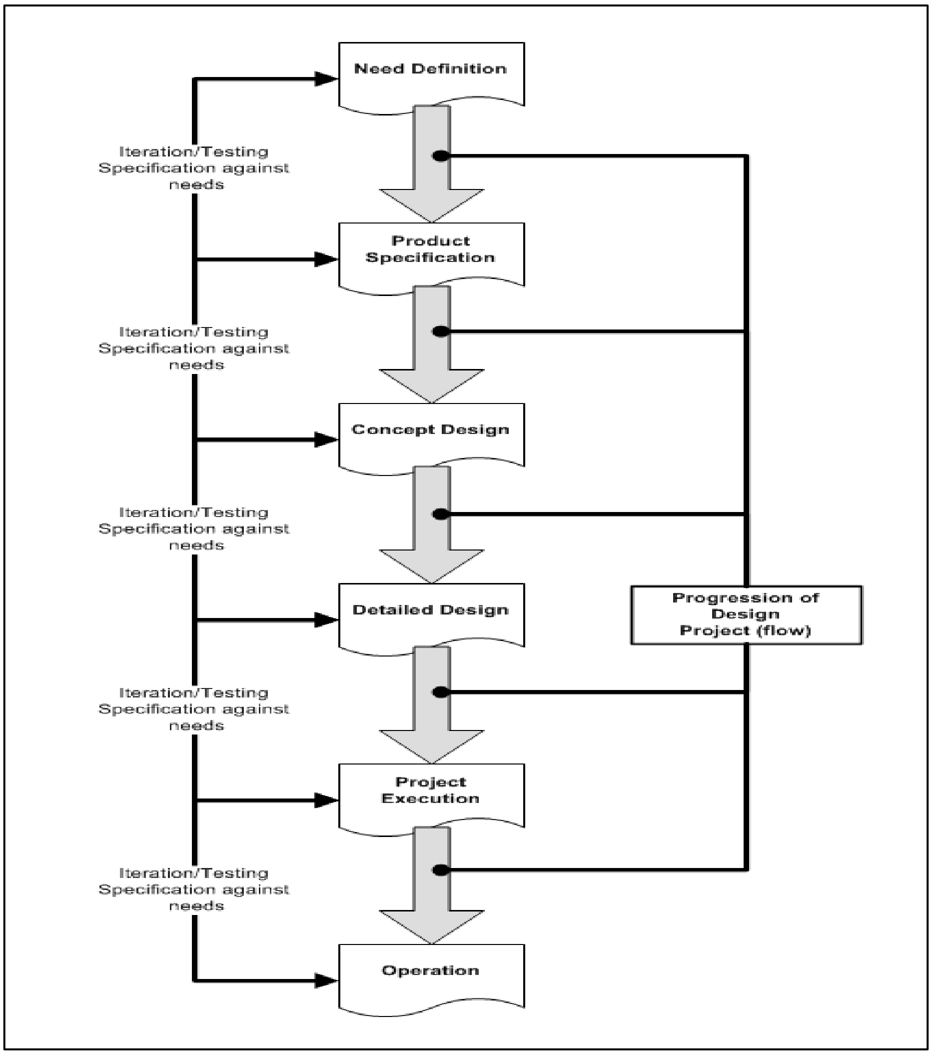


Figure 2.6-5 Schematic presentation of the Stuart Pugh design Option Selection methodology (adapted from Pugh, 1991)

Therefore, in the Pugh method, concept options are considered as being better, worse or similar to the baseline concept option, therefore, indicated with a "+", "-" or "S" respectively, and then ranked based in number of "better", "worse" or similar tick marks for each concept option.

Also important to note about the Stuart Pugh method, is that reaching a "+", "-" or "S" for a certain criteria is not left to a vote. Differently explained, if two members of a design team or even two team disciplines have different opinions when allocating rankings, the matter is resolved by the following (Frey *et al.*, 2009):

- Presentation of facts, experience or even concrete historical data
- Clarification regarding the Option Selection concept which may include another level of discussion:
  - Additional or refinement of criteria definition
  - If agreement ensues, the usual "+" or "-" allocation can be performed. If disagreement remains an "S" is entered into the matrix, denoting either agreement that the concept is similar to the baseline when viewed against the criteria, or disagreement. This dual meaning can lead to distortion of selection outcomes if not managed or just accepted. This is where the Pugh method might experience inadequacy. To address this shortfall other authors implemented the used of an "i" or "?" to indicate that more investigation, design or concept development be performed and selection be repeated (Pahl *et al.*, 1984).

Even though the abovementioned appears to indicate a flaw in the Pugh method is should also be stated that Pugh did not intend his method to be rigid but rather to be adaptable to the team and to allow divergence of concept ideas before facilitating convergence. This convergence is referred to as the Stuart Pugh Controller Convergence or PuCC (Frey *et al.*, 2009). Multiple case studies conducted by independent authors considering Option Selection for design projects (Muller, 2011) as well as Option Selection for organisational/operational change (Von-Sparr, 2014) and (Burgren *et al.*, 2015) indicated positive Option Selection results and also refers to Pugh method as a Multi-criteria decision making process of MCDM.

From the studies performed the main conclusions regarding the use of MCDM which includes the Pugh method, were as follows:

- The lowest ranked options can be eliminated
- The highest ranked option cannot be considered the most suitable go-forward option
- Several high ranking options should or could be considered for further evaluation

Several factors were identified by these authors as important considerations when using MCDM or Pugh selection methods, and are mostly related to human factors:

- Experience
- Knowledge
- Ability to handle information
- Ability to use the models correctly

With these human factor influences in mind it is recognised that the implementation of any decision method would require cognisance of such factors during facilitation of the decision making process.

2.6.2.4 Ambiguity Influences on Specification, Decision Making and Pugh

It has been stated in this dissertation that a design process is intended to be structured and purposed to control uncertainty. Even though the Pugh method breaks down the design process into six phases, it is generally accepted that any project of design phases consist of basically three main activities (Roman, 1985):

- Identification of the need
- Development of a solution
- Implementation of a solution

This is in essence what the previously discussed method of design and project management encompassed. What is important to know is that these activities and the eventual outcomes of the design or project is controlled by specifications aimed at defining the three points listed above (Roman, 1985), and the taxonomy used in the development of requirements and specification is increasingly recognised influential in ensuring the accuracy in requirements specifications .

The method as proposed by Stuart Pugh uses in addition, “+”, “-“ or “S” as a means of rating and ranking concept options. As mentioned this may bring human factors or ambiguity into the rating and ranking within selection matrixes. This ambiguity has been identified as having a significant impact of critical control systems design logic and the response of such systems to operational conditions (Kamsties *et al.*, 2001).

Modification of the Pugh method should therefore attempt to decrease ambiguity as far as possible, but not in such a way the option generation is hindered. In the design of computer controlled systems, reducing ambiguity is paramount to ensure the expected response from a system. However, this ambiguity starts as early as the development of design requirements specifications, which is a very early stage of any design project (Popescu *et al.*, 2008). These authors stated that ambiguity is related to the use of natural language by individuals responsible for defining requirements specifications. It is possible to decrease this ambiguity by limiting freedom in terms of rating and ranking concept options through identification and constraint of ambiguity stemming from the use of natural language to define requirements specifications (Popescu *et al.*, 2008). To understand ambiguity in terms of natural language, ambiguity taxonomy becomes very important and was considered problematic, even in the early eighties, as far as computer programing was concerned, as efforts by engineers to understand and limit ambiguity evidenced by the development of specific taxonomy of issues in requirements engineering (Roman, 1985). This taxonomy approach was more recently further developed by listing the type of ambiguities, their definitions, and applicable examples to allow case studies to be performed involving participant software engineers (Massey *et al.*, 2014). A total of 48 case study participants partook in this case study exercise. Participants were required to review selected text portions in terms of taxonomy as presented in Table 2.6-4 presents the taxonomy used during case studies while Figure 2.6-6 presents results of the case studies.

These ambiguity types can be summarised in terms of Taxonomy as follows:

- Lexical type – A word or sentence or portion of a sentence that may have multiple valid meanings
- Syntactic type – A sequence of words that may be lead to multiple valid interpretations regardless of context
- Semantic type – A sentence with more than one interpretation within the context of the provided text
- Vagueness type – A statements that is indicative of upper and lower extreme cases or leads to relative interpretations
- Incomplete type – A sentence that is considered correct grammatically, but provide little information as to the concrete meaning thereof
- Referential type – A sentence that is considered correct grammatically, but provide confusing references within its intended context

Table 2.6-4 Ambiguity Types in terms of Taxonomy, Definitions and Examples (after Massey *et al.*, 2014)

Ambiguity Type	Definition	Example
Lexical	A word or phrase with multiple valid meanings	Melissa walked to the bank.
Syntactic	A sequence of words with multiple valid grammatical interpretations regardless of context	Quickly read and discuss this tutorial.
Semantic	A sentence with more than one interpretation in its provided context	Fred and Ethel are married.
Vagueness	A statement that admits borderline cases or relative interpretation	Fred is tall.
Incompleteness	A grammatically correct sentence that provides too little detail to convey a specific or needed meaning	Combine flour, eggs, and salt to make fresh pasta.
Referential	A grammatically correct sentence with a reference that confuses the reader based on the context	The boy told his father about the damage. He was very upset.

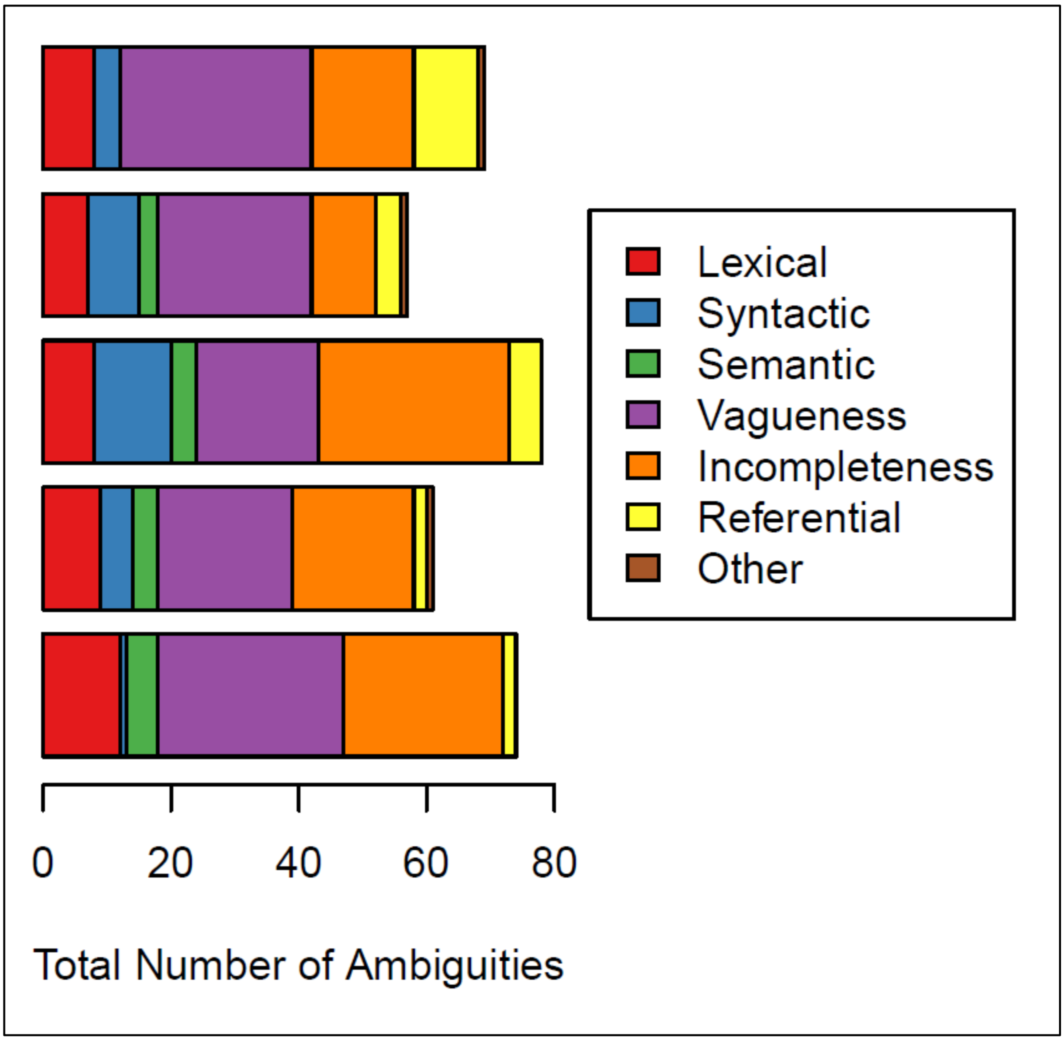


Figure 2.6-6 Bar chart of the ambiguities Identified per case study paragraph. It is notable that vagueness and incompleteness were the most prevalent ambiguity types identified by the study participants (after Massey *et al.*, 2014).

Based on the bar chart presented in Figure 2.6-6, vagueness and incompleteness was indicated as the largest contributors to ambiguity within the case study.

2.6.3 Decision Support Model Classification

With design methodologies as discussed in the preceding sections of this dissertation, decisions between different options still rely upon the decision maker's understanding of the different option available. It can therefore be stated that independent of the design methodology used, certain aids will be required to ensure decision-maker-maker communication. Several models support such decision-making- communication, and can be divided into four general types, which may include physical models, analogue models, schematic models and mathematical models (Blanchard, 2006).

2.6.3.1 Physical Models

The use of physical models during the development of concept designs generate significant benefits in terms of the expression of concept design ideas and interpretation of concept ideas by all the relevant stakeholders or decision makers. Physical models allow a design team or project team to communicate effectively and accurately the relevant aspects of the design or option concept (Kojima *et al.*, 1991). In other words, within the design and project management environment, the purpose of physical models are to ensure sufficient understanding of the design or project to limit complexities and thus risk, associated with fabrication and concept selection (Michealraj, 2009) .

Physical models can take several forms which can be referred as soft-, hard- and presentation models and can be extended to prototypes. Table 2.6-5 presents a breakdown of the different types of physical models that may be used to convey concepts that arise during the course of a product or project development lifecycle.

Table 2.6-5 Classifications of models (after Michealraj, 2009)

Soft Models	Hard Models	Presentation Models	Prototypes
• Rough modelling • Use to assess the overall size, proportion, and shape of many proposed concept. • Fast evaluation of basic sizes and proportions • Reshaped and refined by hand to explore improve its tactile quality	• Technically non-functional yet a replicas of the final design • Made from wood, dense foam, plastic, or painted and textured • Have some "working" features such as button that push or sliders that move	• And matched from CAD data • Complete model and fully detailed composition of the product • Component of this model will be simplified or neglected due to cost or time shortages	• High-quality model or functioning product that is produce to realize a design solution. • Would be tested and evaluated before the product is considered for production.

The progression from a soft models through to presentation models and finally to prototypes can be a seen as increase in information, knowledge or experience, and therefore a decrease in uncertainty in terms of the end-result of the product development. It can be stated that the possibility to influence product development costs decrease as product development progresses through the modelling steps and finally prototyping (Ehrlenspie *et al.*, 2007). Figure 2.6-7 is a presentation of the development costs verses the development timeline related to the sequence of modelling and design. Soft models are therefore associated with the development concepts, hard models with preliminary refinement of these concepts, presentation modelling with high detail models to allow concept selection, with prototyping the final modelling stage to allow physical testing and evaluation of the outcomes of the design process.

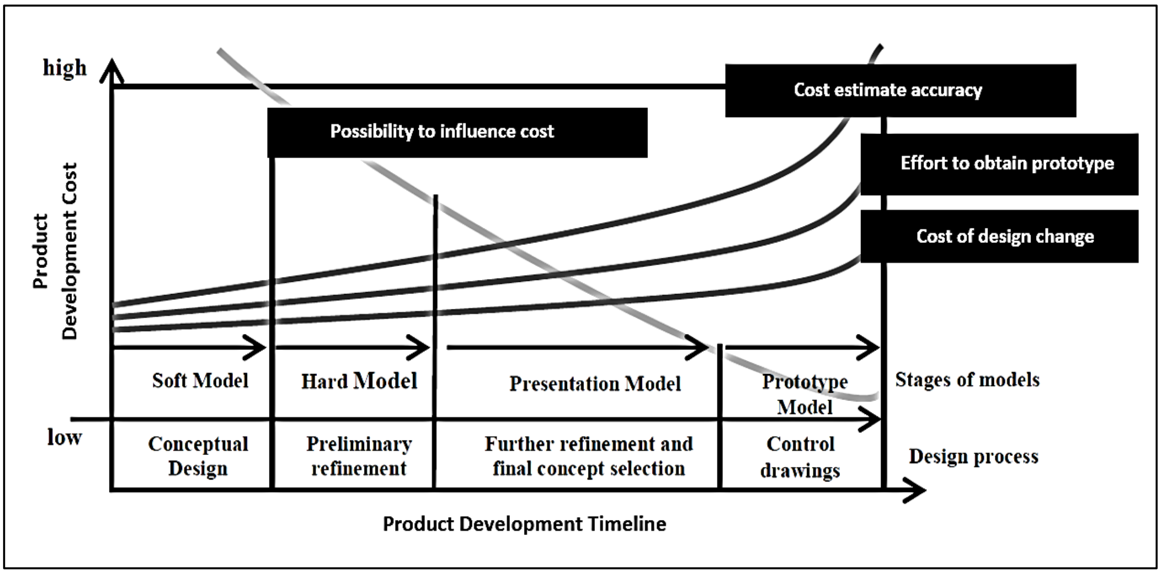


Figure 2.6-7 Graphical presentation of model sequencing as related to development cost as well as product development timeline (adapted from Ehrlenspie *et al.*, 2007)

The graphical presentation in Figure 2.6-7 is in line with the typical phased approach in the managing of projects, where the cost of implementing a change to the project or the outcomes of a project increases dramatically with the progression of the project life (Figure 2.6-8). Similarly the ability to influence the outcomes of projects decreases with the progression of the project life (Figure 2.6-9). When combining the graphs presented in Figure 2.6-8 and Figure 2.6-9 a situation similar to what is presented in Figure 2.6-7 is apparent.

The reason for modelling approaches and its influence on design progression is based on the nature of products or projects. This is because projects are defined as temporary endeavours undertaken to create a unique product, service or result (PMBOK, 2013) and project management techniques are applied to decrease uncertainty in a similar manner as physical modelling decreases uncertainty.

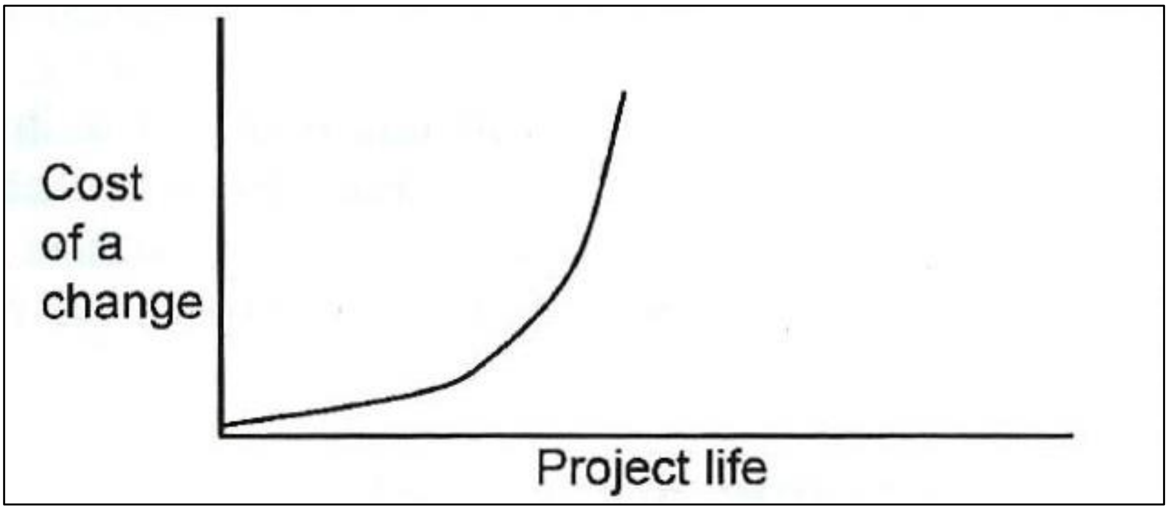


Figure 2.6-8 Graphical presentation of the cost of change during the life of a project (after Steyn, 2015)

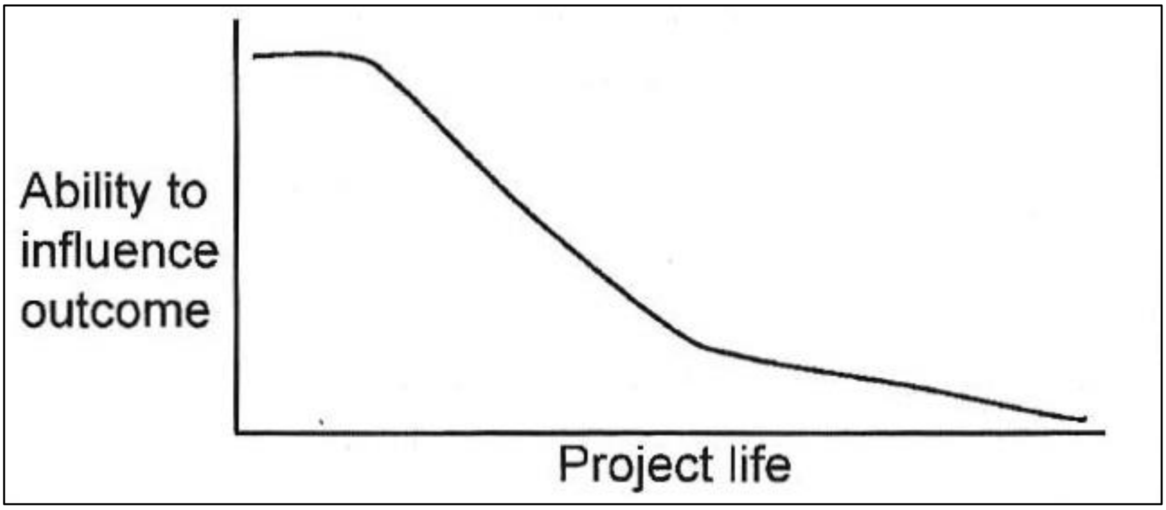


Figure 2.6-9 Graphical presentation of the ability to influence project outcomes during the life of a project (after Steyn, 2015)

There is thus interdependency between the use of different models and prototypes and the overall design process. Therefore, models have functional roles to play within the general design process, as presented in Figure 2.6-10. Moreover, the function of models aids the design process by allowing visualisation of the problem, refining concepts to fit specifications, assist design teams during engineering and development project phases, confirm the resulting design manufacturing, and ensuring the achievement of sought outcomes of the design process (Sæter *et al.*, 2012).

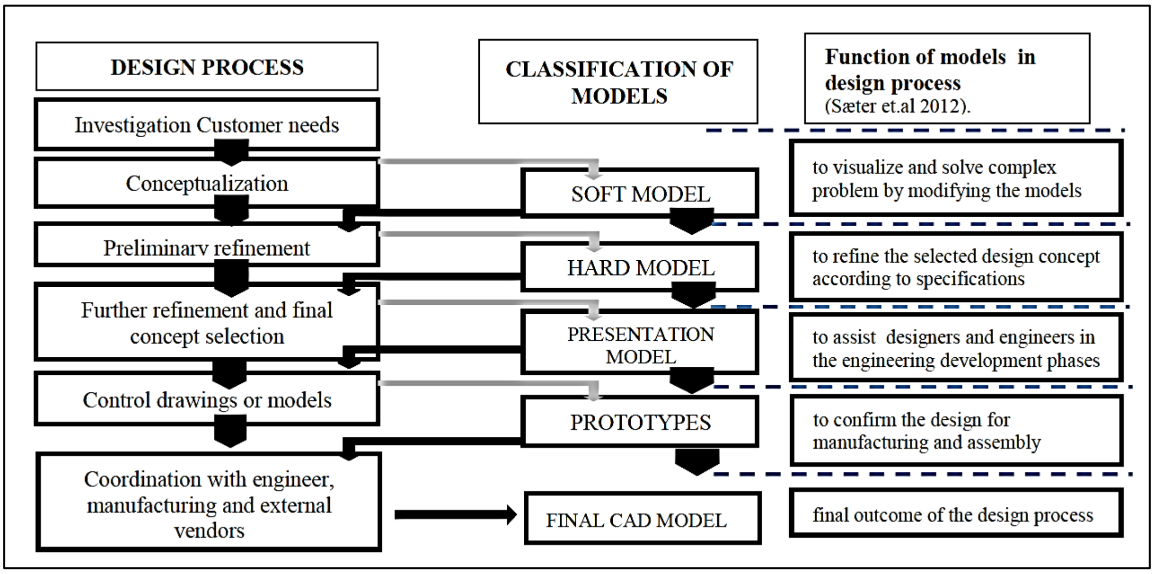


Figure 2.6-10 Interdependency of different models and a design process in terms of the function of the models in the design process steps (after Isa *et al.*, 2014)

2.6.3.2 Schematic Models

Schematic models are an attempt to present a process, state or condition, or an event schematically or diagrammatically (Blanchard, 2006). Schematic models may or may closely resemble the reality.

Schematic models are useful in the presentation of ideas, concepts and can give more meaning to abstract processes such as information management systems (Al-Fedaghi *et al.*, 2015). It finds increasing application in teaching methods where schematic concept mapping is implemented to aid the transfer of knowledge and communication to students (Taşkina *et al.*, 2011).

In process engineering the use of schematic models is a means of communicating process concepts and process control methods. The applications of schematic model in the process engineering discipline have in more recent time become more sophisticated by allowing digital schematics with significant process information to be layered and made available to process engineering design teams. The use of schematic block process-flow-diagrams (PFD's) and piping-instrumentation-diagrams (P&ID's) are now integrated in to plant design software and allows process simulation to be performed.

Schematic models also find significant application in project management and are used to structure activities, interdependencies and responsibilities. Schematic approaches also then allow allocation responsibilities, accountabilities and project funding (Steyn, 2015). Project management fraternity this is referred to a work-breakdown-structure (WBS), and is vital to the success of large multi-activity projects (Brown *et al.*, 2010).

From the literature reviewed, schematic models are used to not only to communicate ideas and concepts, but to also control activities and allow decisions to be made by included decision makers in structured schematic communications.

2.6.3.3 Mathematical Models

The application of mathematical language to describe concepts, systems, processes and organisations is generally referred to as mathematical models. The use of mathematical models allows the incorporation of abstraction, logic and precision, and is utilised to make predictions and to control conditions. This is more prominent where computer aided design and control is implemented during development and decision making. Where systems of processes contain social and economic variables, mathematical models requires the incorporation of probabilistic elements and parameters to enable it to explain actual or real system response deviation from the predicted system response. Furthermore, mathematical model utilised explain or predict existing or expected operations in corporate two types of variables (Blanchard, 2006):

- Variables directly affected or controlled by decision makers
- Variables not direct, or indirectly affected or controlled by decision makers

Thus, mathematical models are beneficial where analyses of systems are required. This aspect of mathematical modelling also implies that the integrity of data that is used as input into the model needs to be of a sufficient level of quality (Winkler, 2004). Where less reliable data is available other modelling approaches may be more apt.

2.7 Conclusions and Purpose of the Study

The production of quality steel requires the use of ferromanganese alloys to enable sufficient refining of steel products to allow it to be used for engineering and other industrial applications. The ferromanganese production chain relies on manganese ore of acceptable consistency to ensure the most cost effective production as well as the lowest impact on the environment.

One way of ensuring the abovementioned, is to decrease the variability of ore and thus, producing sintered or beneficiated ore from Mamatwan mine to downstream smelter facilities in Meyerton. At Mamatwan mine the variability of ore supplied to the sinter plant is currently being managed by the DMS facility. However, the use of this method of ore variability management is evaluated in terms of lower grade ore discarded by the application of the DMS process. This equates to a loss in terms of operational expenditure yielding no return, as the mentioned discard is considered waste and treated as such.

By reconsidering the way in which mining practices are applied at Mamatwan mine, and also reviewing another method of ore variability management, it is deemed possible to manage the mentioned ore variability that is so critically important for the sinter production and downstream smelter production consistencies and efficiencies.

The purpose of this study is therefore focused on selecting an alternative ore variability management method, as an alternative to the DMS method, and consequently to perform a trail project at Mamatwan mine to evaluate the feasibility of implementation of this alternative method.

2.8 Scope of the Study

In this study the Stuart Pugh Selection Option Selection method will be employed to select the best alternative ore variability management method, as an alternative to the existing DMS, as well as to implement a full scale trail project aimed at the production of 18 000 tons of sinter for which the sinter feedstock material will not be treated through the application of the DMS facility, but by the application of an alternative ore variability management methodology.

The sinter produced by the sinter plant will be used directly in the Smelter facilities a Meyerton.

The trial project needs to indicate metrics that will confirm the achievement of ore variability management through the application of a method other than the DMS.

The trial project will also be tasked with all aspects of plant alteration, as well as cross-functional integration of Mamatwan mine and plant personnel, and any other facets of the plant and mine that will aid the implantation and testing of the mentioned alternative ore variability management method.

2.9 Outcomes of the Trial

The following outcomes can be stated for the trial project:

- To verify the mines decision to implement Blend-mining as an ore variability management methodology by the application of an Option Selection method.
- To select the most viable ore variability management implementation methodology via an Option Selection method to allow testing of Blend Mining on Mamatwan mine ore through a trail.
- Implement methodologies to allow evaluation of the selected ore variability management method on ore composition throughout the processing of ore at Mamatwan mine and plant.

CHAPTER 3: EXPERIMENTAL/TRIAL PROCEDURE

3.1 Introduction

After completion of the literature study as presented in Chapter 2 of this dissertation, it was apparent that there is a clear relationship between geological discontinuities and product ore variability, as experienced by the mine when extracting, processing and beneficiating Mamatwan manganese ore. It was further mentioned in Section 2.4.1 that it is deemed possible to control ore variability with methods other than that already implemented at Mamatwan mine, which is the DMS facility or method. From the discussion in Sections 2.2 and 2.4 of this dissertation it is clear that existing ore variability management creates significant complexity in terms of the facilities it requires, namely the DMS facility, but also in terms of the waste generated and resources consumed in the form of water and ferrosilicon as mentioned in Sections 2.2.2 and 2.2.3. Because of these reasons it is pertinent to evaluate possible ore variability management methodologies and to perform a trial to prove the validity of such methodologies.

It is clear from the above paragraph that there are several aspects influencing this trial. *The first set of aspects* are related to the understanding of the ore extracted during mining activities at Mamatwan mine, and the qualification of ore variability (Pre-trial ore variability). *The second set of aspects* are the approach used in the selection of the most viable alternative ore variability management trial method, as well as the consideration of the relevant constraints at Mamatwan mine, together with product specifications and related expectations influencing the selection of a viable alternative ore variability management trial method. *The third aspect* is related to the implementation project to allow the selected alternative to be tested on an industrial scale (production during the trial must be sufficient to allow testing in a commercial smelter). In summary, the project considers the alternative method, how to facilitate its implementation as well as the recording of, verification and validation of data generated during the trial.

The following sections will elucidate upon the aspects mentioned above and give further insight and background in terms of the Mamatwan ore processing facilities.

3.2 Mamatwan Ore Processing

As mentioned in Section 3, the extent of ore variability must be determined in order to establish an ore variability baseline for Mamatwan ore, after some stockpiling and reclaiming has been incurred, chemical analyses data from several ore sampling points were reviewed to reveal the typical variability band for Mamatwan ore.

A detailed diagrammatic representation of the Mamatwan mine ore extraction, liberation, particle-size classification, and beneficiation is presented in Appendix B and is discussed in more detail in Section 2.2.2 of this dissertation. However, a simplified Mamatwan process flow diagram is presented in Figure 3.2-1, and attempts of distinguish between ore extraction, liberation and classification, and lastly beneficiation. Liberation and classification can be further broken down into primary processing or crushing at the In-pit jaw crusher (primary crushing), secondary processing through the Ore-Preparation-Plant (originally referred to as the EVA or "erts-verwerkingsaanleg") where secondary crushing and screen is performed. Beneficiation can be divided into grade control (variability control) through the use of the DMS and carbo-thermal reduction (sintering). The following stockpiles are important:

- Run-of-Mine (ROM) stockpile – blasted ore is crushed through the primary crusher and conveyed to the ROM
- From the ROM, ore is conveyed to the secondary crusher and screened through three sizing and washing screens to classify material into Mamatwan Lumpy ore (M1L) stockpile, tertiary crusher feed stockpile and secondary crushing fines stockpile.
- From the tertiary crusher feed stockpile ore is conveyed to the tertiary crushers and classification screens, after which ore is conveyed to the DMS feed stockpiles.
- After beneficiation through the DMS discard is conveyed to the DMS discard or tailings dump, while beneficiated ore is conveyed to the sinter feed stockpile from it is conveyed into the sinter plant for carbo-thermal reduction (sintering).

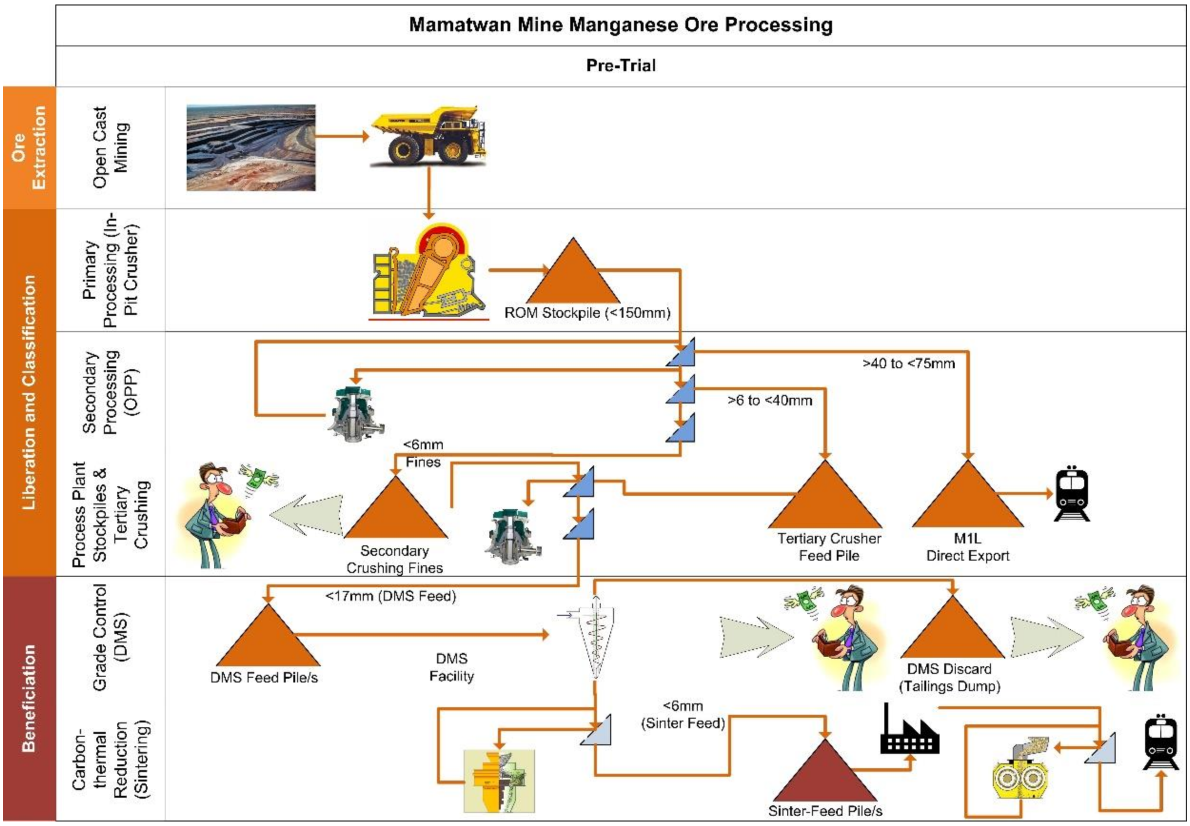


Figure 3.2-1 Simplified Mamatwan mine and plant process flow diagram. Note points where product losses are incurred.

From the flow diagram in Figure 3.2-1 indication of where losses are incurred due to current or existing ore variability management is indicated by the comical figure losing money. This is due to the fact that the ore variability management as it currently is produces none-saleable products. These products represent a loss of operational expenditure and a loss of ore resources.

3.3 Pre-trial Ore Variability Baseline

As indicated in Chapter 2, Section 2.2.1.3, there are certain ranges of geological discontinuities within the Mamatwan manganese resource which leads to variability of the manganese content of ore extracted during mining. However, it is a known that stockpiling and reclaiming of stockpiles, causes intermixing of the ore, with the beneficial effect of decreasing variability to some extent, as indicated in literature. In order to establish an ore variability baseline for Mamatwan ore prior to implementation of alternative variability management, and with some stockpiling and reclaiming incurred, chemical analyses data from several ore sampling points need to be accumulated to reveal the typical manganese compositional variability band for Mamatwan ore. This variability band can then be used as a baseline for the validation and verification of an alternative ore variability management methodology selection and qualification.

A diagrammatic representation of the Mamatwan mine ore extraction, liberation, particle-size classification, and beneficiation is presented in Appendix B and was discussed in detail in Sections 2.2.2 and 3.2 of this dissertation.

To establish the mentioned composition variability band for Mamatwan ore, chemical analyses of material from three sampling points, for a period of twelve days were made available by the mine grade control laboratory. No samples of ore from the pit were available, since this was not part of the Mamatwan mine laboratory sampling schedule. The samples were taken at the following positions on ore preparation plant:

- A daily sample set was taken from the conveyor that delivers material to the Mamatwan Lumpy (M1L – ore sized between 25 and 75mm) stockpile after secondary crushing and screening to remove all ore particle smaller than 25mm (Samples taken were designated as M1L – multiple samples were taken per day and data presented as a daily data set)
- The second sample set was taken from the conveyor that delivers material after secondary crushing and screening to the stockpile that supplies ore material to a tertiary crushing step at Mamatwan plant as part of ore preparation for DMS. This material is sized between 6 and 40mm (Samples taken were designated as ES (tertiary crusher feed secondary crusher fines – Figure 3-2), multiple samples were taken per day and data presented as a daily data set)
- The third sample set was taken from the conveyor that delivers material after secondary crushing and screening to the fine stockpile that supplies ore material to a tertiary crushing step at Mamatwan plant as part of ore preparation for DMS. This material is sized between 6 and 40mm. Samples taken were designated as EF (secondary crusher fines – Figure 3-2). Multiple samples were taken per day and data presented as a daily data set.

Figure 3.3-1 present a simplified process/ore flow diagram based or adapted from Figure 5.2-1; a comprehensive process flow diagram of Mamatan mine, with Figure 5.2-2 an attempt to simply Figure 5.2-1 in Appendix B. In Figure 3.3-1 the mentioned sampling points and stockpiles are indicated.

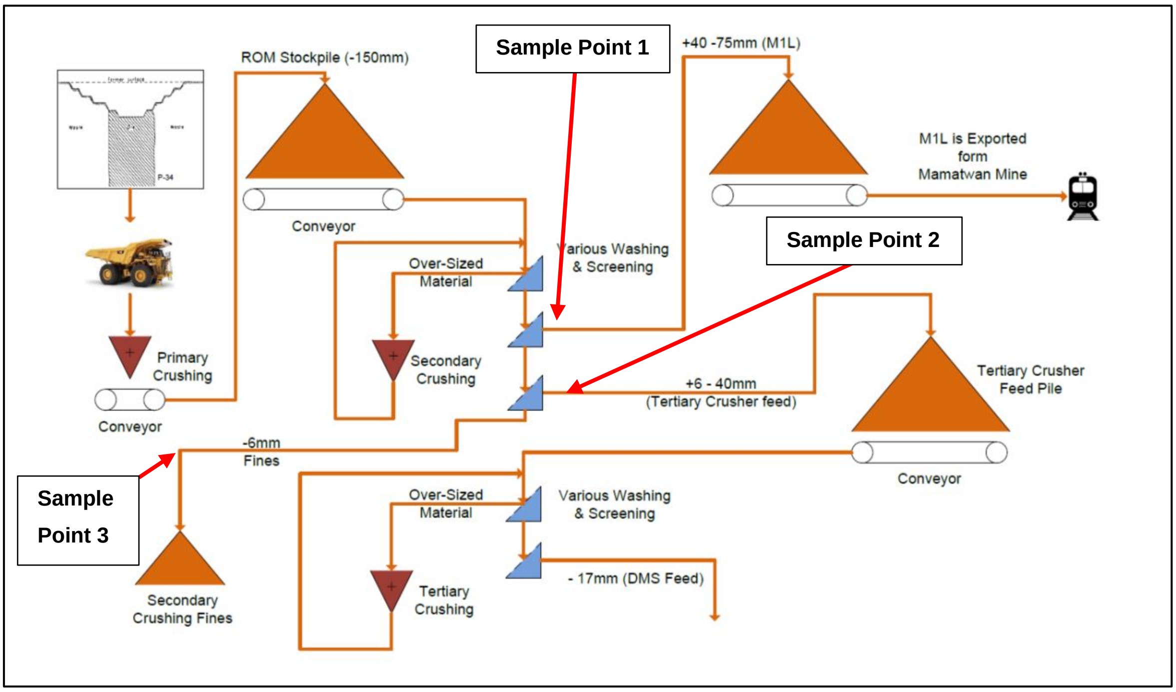


Figure 3.3-1 Diagrammatic presentation of a portion of the Mamatwan mine and plant. The pre-trial sampling mentioned is related to the secondary crushing step, with different screen delivering size classified material to different stockpiles.

Figure 3.3-2 presents an aerial photograph of the plant area where the ROM conveys ore to the secondary crushing and screen facility (known as the OPP – it is also referred to by the mine as the EVA plant, which is an Afrikaans abbreviation for "Erts-verwerkings-aanleg"). These abbreviations are still used by the mine and plant personnel in reports and data sheets. The sampling points and stockpiles mentioned in the discussion related to Figure 3.3-1 are indicated with relevant arrows and descriptions.

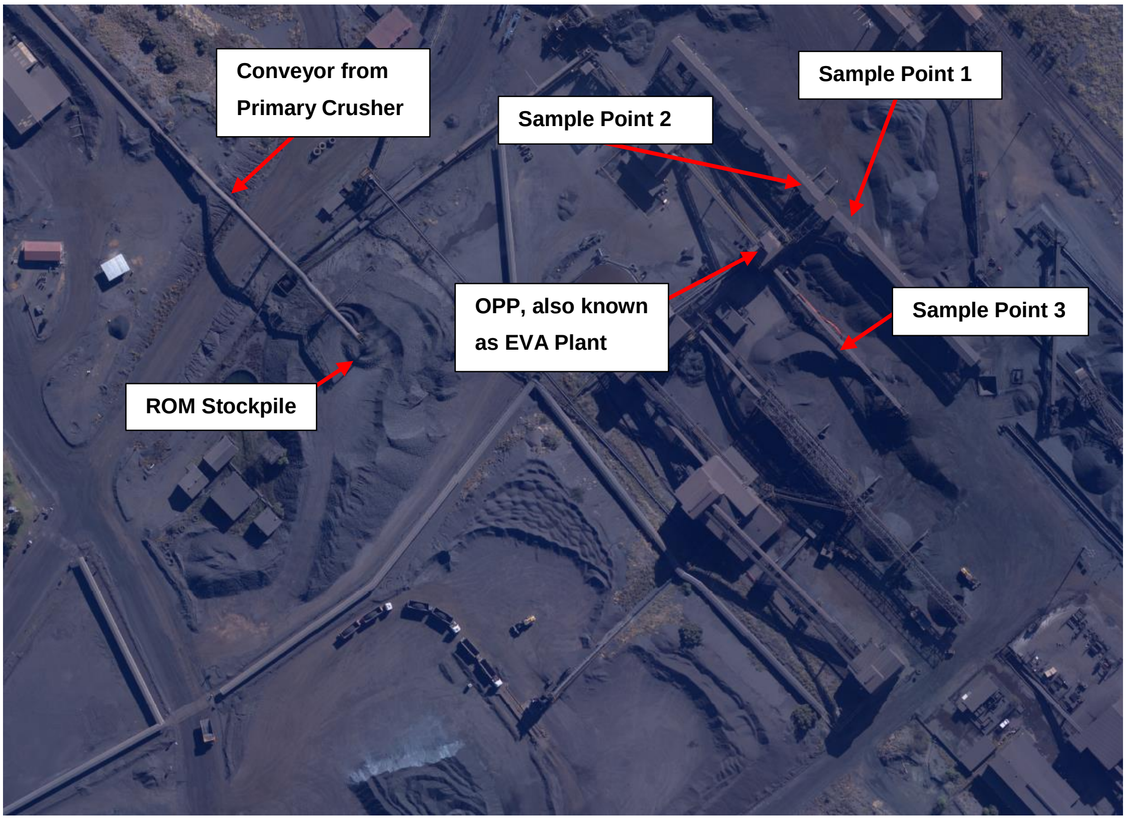


Figure 3.3-2 Aerial photograph of the Mamatwan Plant.

The compositional datasets for twelve consecutive days were generated from chemical analyses of samples from sample points 1, 2 and 3 and are available for more detailed review in Appendix D, from Table 5.3-1 through Table 5.3-12. Associated manganese content graphs for each dataset indicating the variation in manganese content for each are also presented in Appendix D (Figure 5.3-1 through Figure 5.3-12). Combining all datasets into one graphical presentation of the manganese content via chemical analyses gives good insight into the manganese content variability that may exist between ore samples and therefore within the ore body at Mamatwan mine. Figure 3.3-3 presents the manganese contents (mass %) of all sample sets analysed. Manganese contents as low as 28.8% and as high as 48.1% was observed for EL (+40 to 75mm), M1L. Manganese contents as low as 34.6% and as high as 48.0% was observed for ES (+6 to 40mm). Manganese contents as low as 35.2% and as high as 42.8% was observed EF (-6mm).

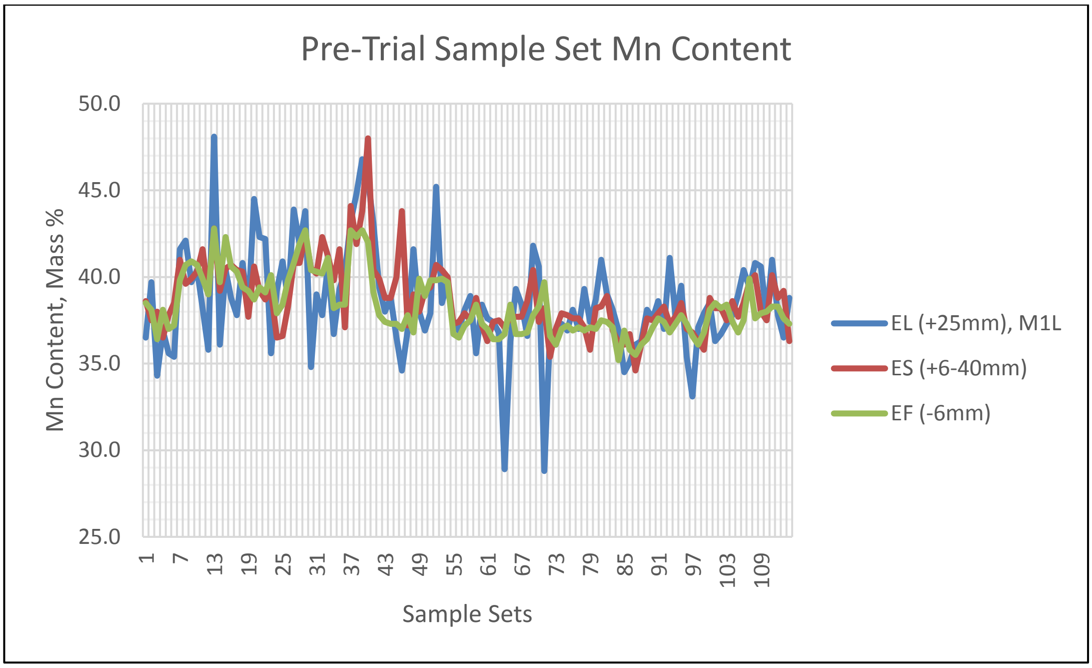


Figure 3.3-3 Graph of manganese content (mass %) for all sample sets collected and analysed over a twelve day pre-trial period.

The data presented in Figure 3.3-3 contains analyses data for 114 sample sets containing three samples each, and interpreting this amount of data can be performed easier by condensing this rather large amount of information in a histogram format, indicating daily sample set % variability. Significant manganese content variability is therefore evident form Figure 3.3-4 for not only EL (+40 to 75mm), M1L samples, but also ES (+6-40mm) samples. This can be further summarised by only considering the minimum and maximum manganese contents recorded during the twelve day pre-trial period (Figure 3.3-5). For EL (+25mm), M1L samples, up to 19.3% manganese content variability was observed. For ES (+6-40mm) samples, up to 13.4% variability and for EF (-6mm) samples, 7.6%. It is therefore clear that significant variability is prevalent, but that commutation decreases variability with each consecutive ore preparation step.

To further display the variability of the Mn content of the ore, the data from the graph in Figure 3.3-3, was further assessed through the presentation of this data in several different manners. The daily sample set variability, the extreme variability within sample sets, as well as the Mn content distribution for all the sample sets were plotted.

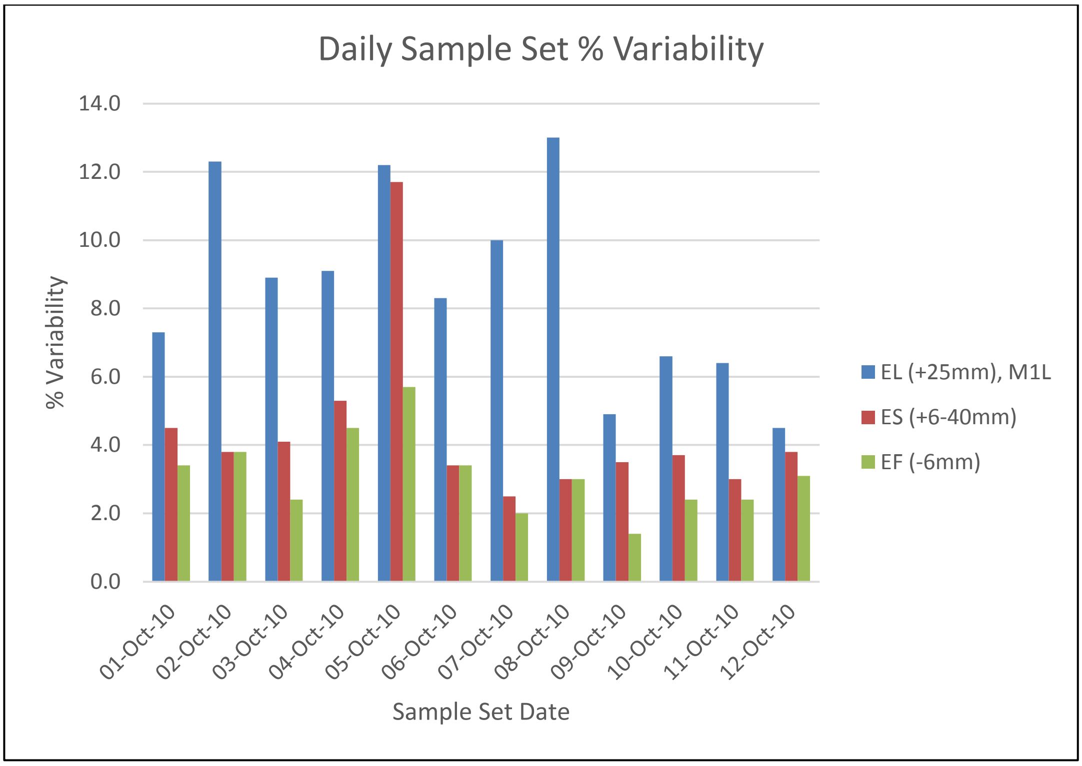


Figure 3.3-4 Histogram presentation of the variability revealed with in the daily sample sets from the three sampling points. Again is observed that variability decreases as commutation increases.

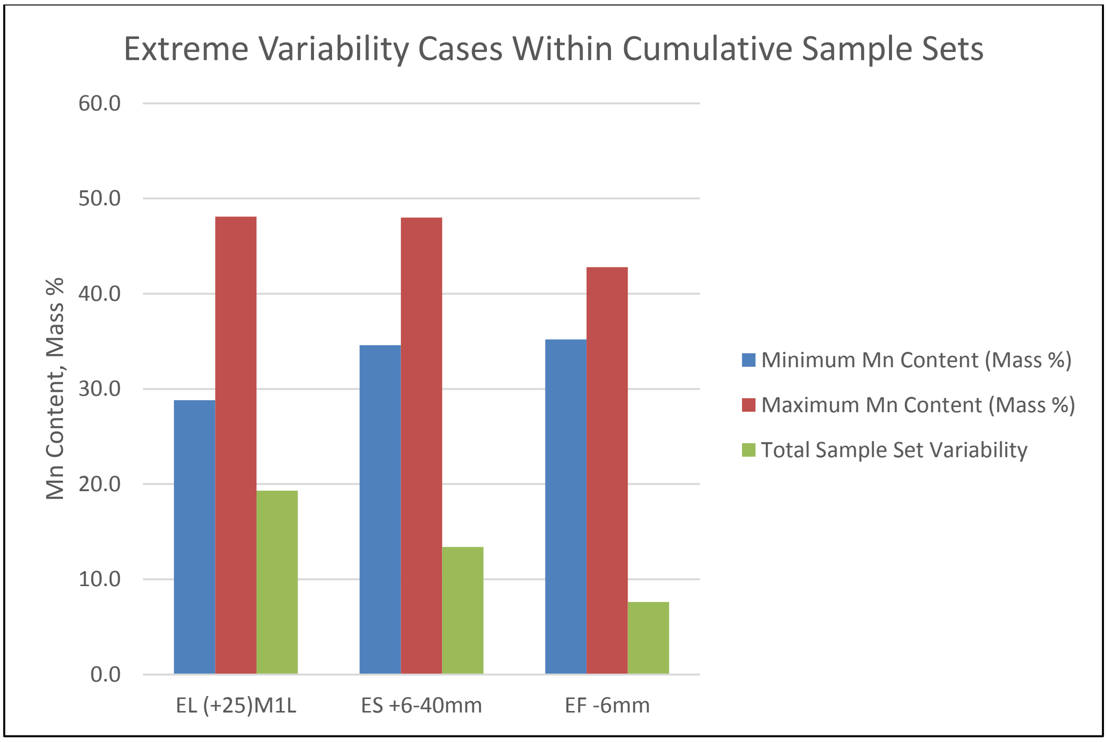


Figure 3.3-5 Histogram of manganese content (mass %) for extreme minimum and maximum sample manganese content for cumulative sample data set collected and analysed over a twelve day pre-trial period.

To give a more profound indication of the effects of commutation on variability, sample compositional distribution graphs were plotted EL, ES and EF in Figure 3.3-6. Clearly visible from these graphs are that commutation of the ore, progressively removes the extreme cases of very low and very high manganese content ore particles, which further explains the decrease in total variability as indicated in the graph in Figure 3.3-5.

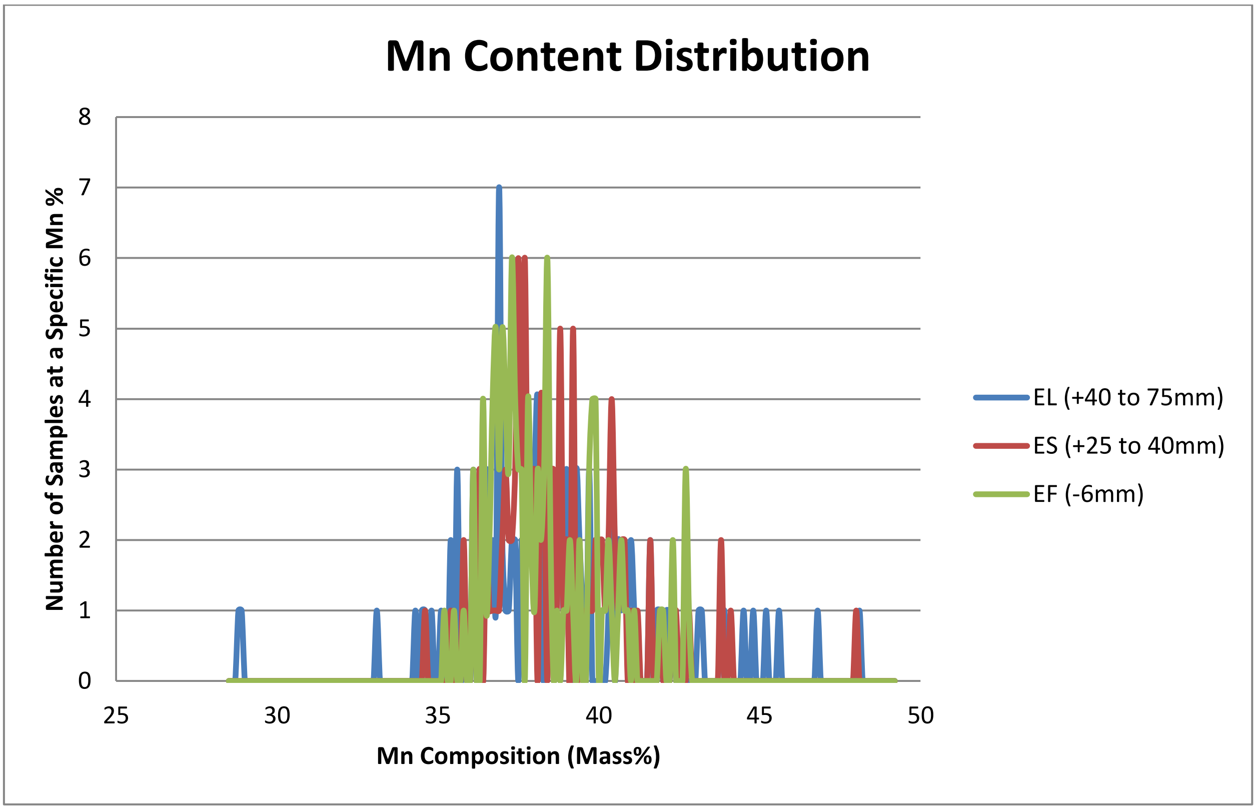


Figure 3.3-6 Plot of the manganese content distribution vs. samples with a particular manganese content for samples taken at sampling point 1.

From the graphs plotted in Figures Figure 3.3-4 through Figure 3.3-6, even though variability is alleviated through commutation, it is not completely removed and still displayed a manganese content variation of 7.6%. This is the variability that originally necessitated the implementation of the DMS facility at the Mamatwan mine and plant.

In an attempt to further study the extent of Mn content in Mamatwan the data as presented the proceeding graphs, Mn content is presented as a variation above or below the 37% Mn content goal. When presenting the data in this manner, the goal of 37% is set to 0 and variation in Mn content is then easily seen as percentile above or below 0.

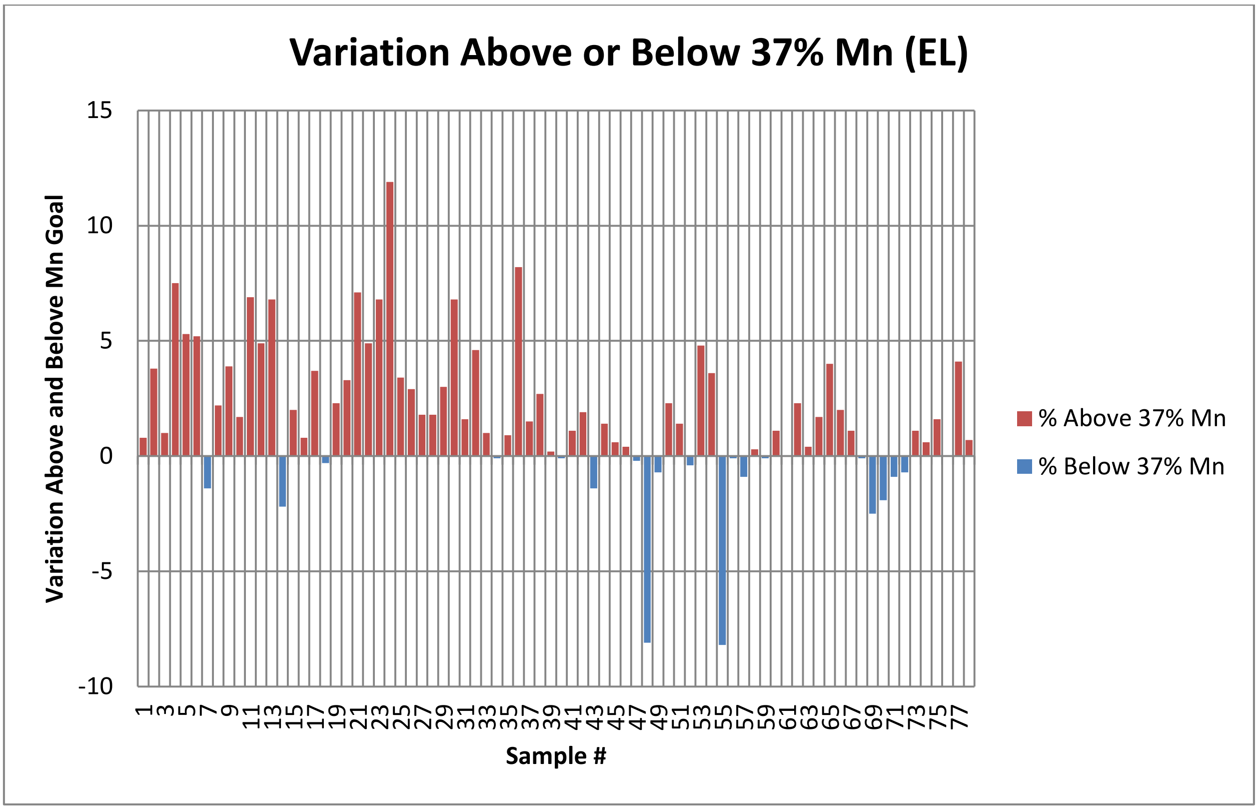


Figure 3.3-7 Graph of the plotted variation around the 37% Mn content goal for tertiary crusher feed (ES).

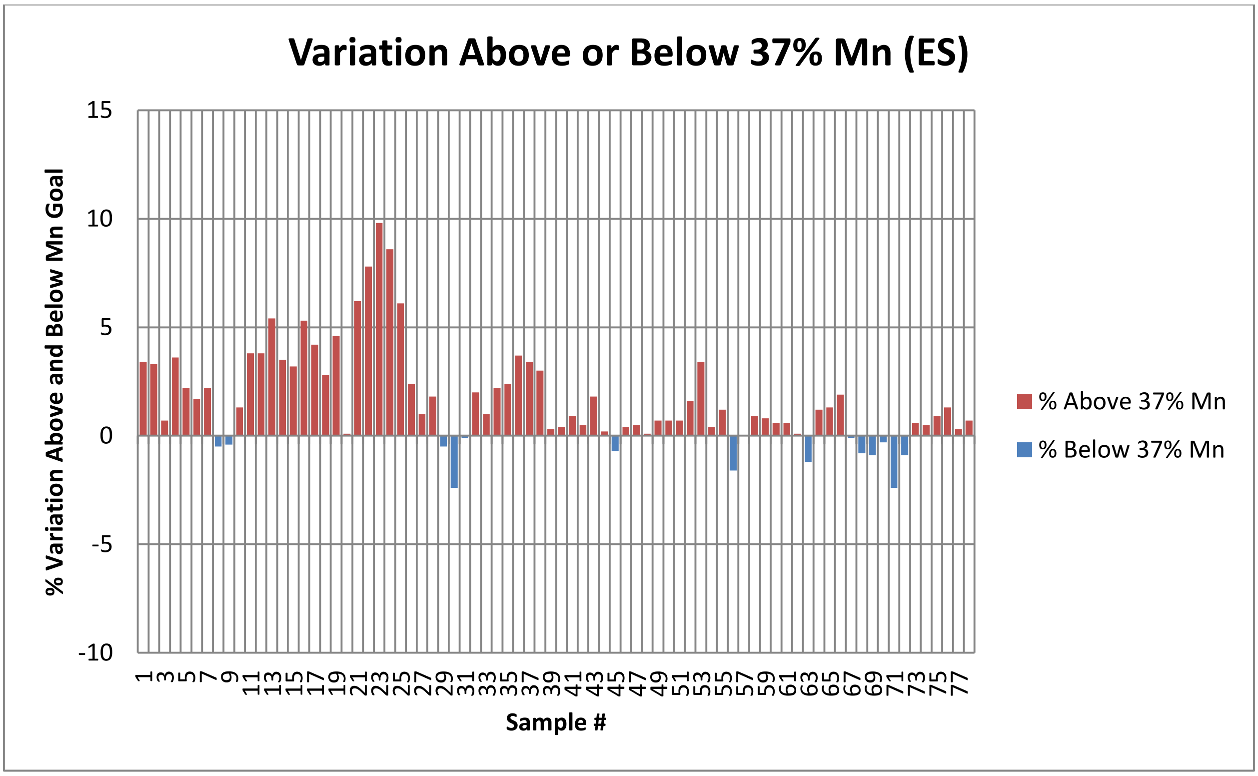


Figure 3.3-8 Graph of the plotted variation around the 37% Mn content goal for tertiary crusher feed (ES). The zero axis on the graph presents a zero deviation from the Mn content goal.

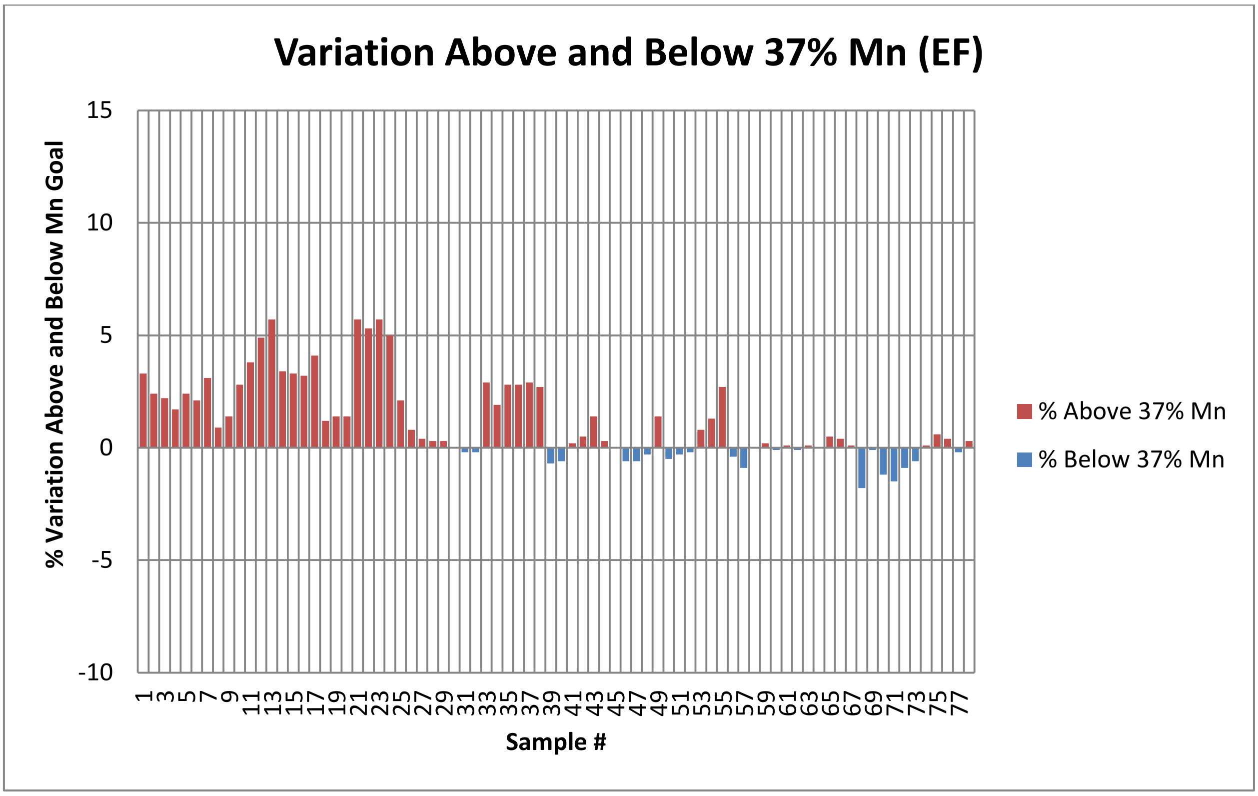


Figure 3.3-9 Graph of the plotted variation around the 37% Mn content goal for tertiary crusher feed (ES). The zero axis on the graph presents a zero deviation from the Mn content goal.

From the presentation of variability in Figures Figure 3.3-7 through Figure 3.3-9 a correlation in terms of variability between EL, ES and EF sample was observed. This is expected as no blending yards are present as Mamatwan Mine and Plant.

In conclusion to the review of pre-trial chemical analyses of ore manganese content; significant variability was evidenced and will serve as a clear validation for the existence of the DMS plant and will serve as a baseline for validation and verification of an alternative variability management methodology. This will be compared to trial data.

When reviewing the bell graphs for the Mn contents, the presence of some higher Mn content samples are quite noticeable. This will be compared with trial data.

When reviewing the bell graphs for the Mn contents, the presence of some higher Mn content samples are quite noticeable. This can be compared with trial data. A summary of the data presented in Figure 3.3-7 through Figure 3.3-9 can be presented in a Table after counting sample below, matching and above the stated goals. From the data presented in this Table and in proceeding graphs, it is clear that significant % of material is above the goal of 37%.

3.4 Option Selection Methodology

Due to the fact that a trial was required in which ore variability without the DMS needed to be performed and proven, Option Selection methodologies were considered in Chapter 2 of this dissertation. It was elected to implement the Stuart Pugh methodology, due the possibility when using this methodology to also take technical and none-technical aspects of Option Selection into consideration. A further motivation for the use of Stuart Pugh is the fact that the mentioned trial needed to be performed through a production facility that operates on 24 hour, 7 days a week basis, and which exists. Again, more than just standard design aspects needed to be taken into consideration during Option Selection processes. The trial would therefore require the assistance of existing plant personnel, which would provide excellent input into for use in the Stuart Pugh selection methodology.

The Pugh methodology is generally applied in a four part approach (Section 2.6.2.3 of this dissertation), which can be structured as follows:

1. Break the design project into six relevant phases
2. Develop a project design specification (PDS), which is related to the outcomes sought
3. List discipline-independent elements in a divergent-convergent manner
4. List discipline-dependent elements in a divergent-convergent manner

This four part approach is difficult to understand when thinking in terms of the four points indicated above. One way of making the Pugh method easier to understand is to consider part 2, 3 and 4 of the Pugh method as a set of criteria the supports the PDS, constraints and opportunities. Therefore, all relevant disciplines involved in the project is allowed to give inputs into options possible as well as constraints that can or must not be forgotten, allowing divergence and convergence to take place as intended by the Pugh method.

After development of concept options and selection criteria, as Pugh selection matrix is developed for each relevant project phase. One of the concept options is usually a product or condition that currently exists. As far a product improvement is concerned, the baseline concept option would be the most recent product model. As far as plant improvement or optimisation design projects are concerned, the baseline concept would be the current plant or operational process.

As discussed in Section 2.6.2.3 for the Pugh method, concept options are considered as being better, worse or similar to the baseline concept option, therefore, indicated with a “+”, “-“ or “S” respectively, and then ranked based in number of “better”, “worse” or similar tick-marks for each concept option.

Also important to note about the Stuart Pugh method is that reaching a "+", "-" or "S" for a certain criteria is not left to a vote. Differently explained, if two members of a design team or even two team disciplines have different opinions when allocating rankings, the matter is resolved by the following (Frey *et al.*, 2009):

- Presentation of facts, experience or even concrete historical data
- Clarification regarding the Option Selection concept which may include another level of discussion:
    - Additional or refinement of criteria definition
    - If agreement ensues, the usual "+" or "-" allocation can be performed. If disagreement remains an "S" is entered into the matrix, denoting either agreement that the concept is similar to the baseline when viewed against the criteria, or disagreement. This dual meaning can lead to distortion of selection outcomes if not managed or just accepted. This is where the Pugh method might experience inadequacy. This shortfall is addressed by the use of an "i" or "?" to indicate that more investigation, design or concept development be performed and selection be repeated (Pahl *et al.*, 1984).

Even though the abovementioned appears to indicate a flaw in the Pugh method it should also be stated that Pugh did not intend his method to be rigid but rather to be adaptable to the team and to allow divergence of concept ideas before facilitating convergence. This convergence is referred to as the Stuart Pugh Controlled Convergence or PuCC (Frey *et al.*, 2009). Multiple case studies conducted by independent authors considering Option Selection for design projects (Muller, 2011) as well as Option Selection for organisational/operational change (Von-Sparr, 2014) and (Burgren *et al.*, 2015) indicated positive Option Selection results and also refers to Pugh method as a Multi-criteria decision making process of MCDM. From the studies performed the main conclusions regarding the use of MCDM which includes the Pugh method, were as follows:

- The lowest ranked options can be eliminated
- The highest ranked option cannot be considered the most suitable go-forward option
- Several high ranking options should or could be considered for further evaluation

Several factors were identified by these authors as important considerations when using MCDM or Pugh selection methods, and are mostly related to human factors:

- Experience
- Knowledge
- Ability to handle information
- Ability to use the models correctly

With these human factor influences in mind it was also elected to modify the Pugh method to some extent.

3.4.1 Pugh Method Modification

The method as proposed by Stuart Pugh uses "+", "-" or "S" as a means of rating and ranking concept options may bring human factors or human emotion or ambiguity into the rating and ranking within selection matrixes. This ambiguity has been identified as having a significant impact of critical control system's design logic and the response of such systems to operational conditions (Kamsties *et al.*, 2001).

Modification of the Pugh method should therefore attempt to decrease ambiguity as far as possible. In the design of computer controlled systems, reducing ambiguity is paramount to ensure the expected response from a system. However, this ambiguity starts as early as the development of design requirements specifications, which is a very early stage of any design project (Popescu *et al.*, 2008). These authors stated that ambiguity is related to the use of language by individuals responsible for defining requirements specifications. As discussed in Section 2.6.2.4 of this dissertation, it is possible to decrease this ambiguity by limiting freedom in terms of rating and ranking concept options.

Based the influence of grammar and long sentences indicated by Popescu and how critical this is in initiating ambiguity it was decided to abstain making statements during workshops, but rather to phrase questions related to Option Selection criteria such that would yield only a yes or a no answer. This is also referred to a ambiguity constraining language (Popescu *et al.*, 2008). This would certainly limit ambiguity to a minimum. However, compiling a set of questions should be based on all aspect of the goals to be achieved.

3.4.2 Project Phasing

The Stuart Pugh design methods require phasing as presented in Section 2.6.2.3., and can be presented as follows:

- Needs definition for trial
- Trial requirements specification
- Concept development/design for the purpose of the trial
- Detailed design and planning of the trial
- Trial project/design execution
- Trial operation period

As mentioned in Section 2.6.2, the Stuart Pugh phased approach is very similar to standardised project management approaches based on the Project Management Body of Knowledge (PMBOK, 2013) and further simplified in Project Management: A Multi-Disciplinary (Steyn, 2015). The aim of this study is not the develop project management, but to perform Option Selection in such a way that the best possible solution is implemented as a trail for managing ore variability via an alternative method. As such, project management methods will not be discussed in more detail, but rather what happens during project management in terms of option development and Option Selection.

To achieve sensible, and more importantly, a trustworthy Option Selection for implementation, three sets of information are required to define the limits within which concept development can take place as well as yield the best possible Option Selection, and can be stated as follows:

- Statement of selection constraints
- Statement of project design specifications
- Statement of selection criteria

The next several sections of this dissertation will discuss these aspects in more detail.

3.4.3 Option Development and Selection Constraints

When performing a Stuart Pugh based Option Selection, elements of product design specification (PDS) must be considered. In terms of this dissertation, PDS statements can be official as mandated by Mamatwan mine management, is less so, when obvious and as well as less obvious constraints are present at Mamatwan mine and which places additional requirements upon the design or project Option Selection. These constraints and PDS statements will culminate into selection criteria. To enable the compilation of comprehensive selection criteria, the mandated PDS statement as well as obvious and less obvious constraints that exist at Mamatwan mine and plant must be identified and be considered during Option Selection criteria development.

Mamatwan mine and plant was first put into service in 1963. At that stage with the knowledge at hand, a certain plant layout and design philosophy was employed. The plant was initially a dry crushing and screening facility with no DMS or sinter plant. Due to downstream production chain requirements, a sinter plant (1987) and later a DMS (1989) was added to the Mamatwan processing facilities (explained in Section 2.3). The implementation of the DMS required the addition of wet screening and crushing at the Secondary ore preparation plant. This change in the processing method was required to decrease ferrosilicon losses during the DMS processing of ore. With the implementation of the DMS and wet crushing and screening, the addition of a thickener as well as slimes (ultra-fines carried by water) dump was required for storage of these newly produces slimes or waste. Currently the Adams pit as it is referred to by the mine is used for DMS tailings and slimes. After the implementation of these facilities, no space was left for additional facilities while allowing continues stockpile management to be performed by conveyor systems as well as hauling vehicles. Furthermore, the way in which a trial project would interface with the existing facilities and related constraints must be kept in mind as well as constrains due to the current operational schedule of the OPP and sinter plant.

With this stated, the most significant constraints caused by Mamatwan mine's plant and operational activities can be broken down as follows:

- Site Position Constrains:
  - Existing access routes and existing plant infrastructure will make the implementation of additional facilities problematic (constrains expansion to the West).
  - Waste dumps for DMS discard and slimes storage prohibits any facility expansion that may interfere with waste routes (constrains expansion to the North).
  - Overhead conveyor systems associated with existing facilities, as well as rail infrastructure also limits expansion (constrains expansion to the East).
  - Road-transport stockpiling, loading and turning as well as the weight-bridge system associated with road-transport vehicle load verification, negates expansion opportunities within this area (constrains expansion to the West).
  - Previous waste material waste stockpiles areas are far away from the OPP and sinter plant and several materials transport access roads are associated with these old stockpiles (constrains expansion to the South-West).
- Existing Plant Interfacing Constraints:
  - Transportation of ore material after secondary crushing and screening. Interface between OPP and Trial.
  - Trial treatment via commutation or tertiary and quaternary crushing and screening of material from the OPP and supply of this material to the sinter plant. Interface between Trial and Sinter plant.
- Operation Constraints:
  - Mining operations, the OPP, DMS and Sinter plants operate on a 24 hour basis with a 12 hour weekly shutdown for each of the operation (routine preventative maintenance, but scheduled consecutively to maximise production time loses.

The site-constraints as indicated above can be illustrated by adapted aerial pictures depicting these constraints diagrammatically in Figure 3.4-1 through to Figure 3.4-3. In Figure 3.4-3 the green ovals indicate the main plant areas as previously discussed in

Figure 3.3-2 Aerial photograph of the Mamatwan Plant. The photograph is a higher magnification of the photograph in Figure 5.2-6 and focuses on plant areas related to the process flow diagram related to Figure 3.3-1. Sampling positions 1, 2 and 3 is indicated with the numbered arrows in the photograph. Figure 3.3-2 which focussed more on sampling points than constrains. Existing operational plant areas cannot be affected during trials (OPP, DMS and Sinter plant facilities).

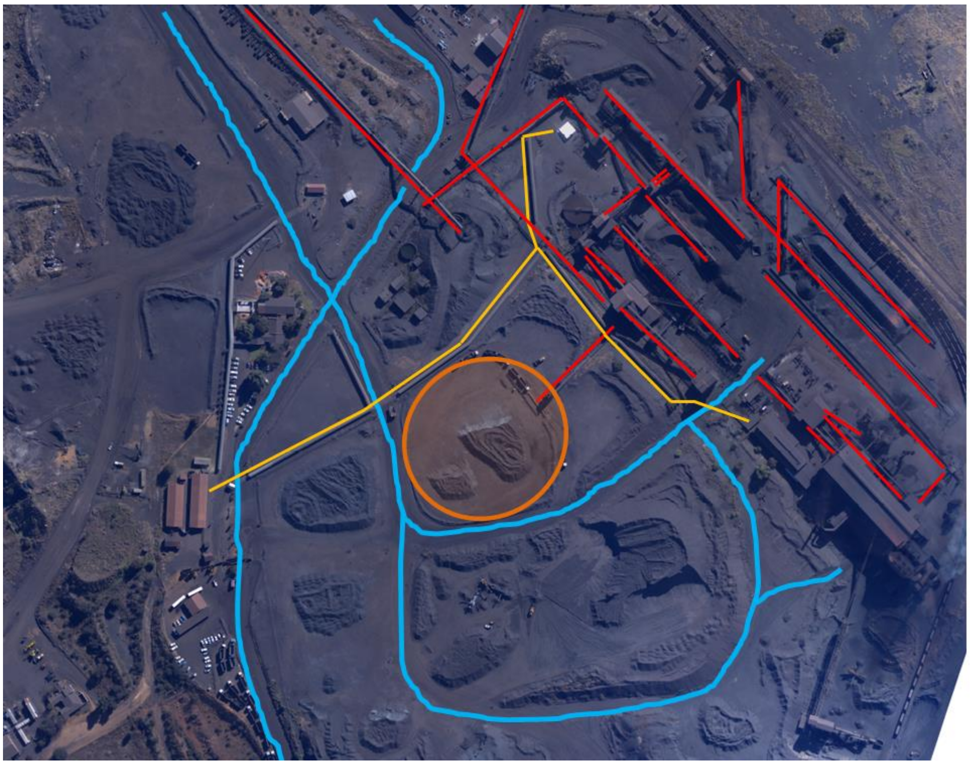


Figure 3.4-1 Aerial photograph indicating personnel access, hauler routes and conveyors at Mamatwan Plant. The blue lines are indicative of hauler routes, while personnel access the plants via walkways indicated in yellow. Stationary conveyor systems indicated are indicated with red. Road-transport load-out area with its road-hauler-weigh-bridge is indicated with.

Figure 3.4-2 Aerial photograph indicating gantry and dumping type stockpiles.

Figure 3.4-3 The green ovals indicate the main plant areas.

Figure 3.4-1 through Figure 3.4-3 presents visually the structures, access routes and stockpile areas that constrain the options that may be selected for a trial project. Several other constraints will be discussed in the next paragraphs.

3.4.4 Project Design Specification (PDS)

The formal requirements for the trial set by Mamatwan mine management can be listed as (please note that this PDS includes many of the constraints discussed in the previous section):

- Produce 18 000 tons of sinter without the use of a DMS facility
- Sinter product produced without the use of the DMS must conform to the following requirements:
  - Goal manganese content of 44%
  - Maximum calcium oxide content of 17%
  - A goal manganese to iron ratio of 8.3
- The trial project must have a zero or limited impact of normal production at Mamatwan plant. In other words, implementation of the trial project must cause the least amount of lost production time.
- The trial project implementation may not impede in any way, the existing plant operations, which includes heavy vehicle movement to and from the plant and stockpile areas.
- Plant personnel access may not be affected.

3.4.5 Selection Criteria Structuring

Based on the statements of Selection Constraints and the Project Design Specifications, it is possible to develop a structure for Selection Criteria. This based on the interpretation of the existing and stated constraint at Mamatwan mine and plant as well as the state specification for the outcomes of the trial project.

The following Selection Criteria structure is recommended:

- Criteria related to plant and operational processing of trial material (ore handling/management)
- Criteria related to the impact and safety of the plant and operational activities (disruption)
- Criteria related to the impact on plant and operational losses (production losses)
- Criteria related to funding in addition to losses (capital costs)
- Criteria related to ease of implementation which would impact time required to reach readiness for Trial commencement (complexity)

As mentioned before, it was also elected to further enhance the Stuart Pugh method by adding levels importance to the selection criteria within the selection matrix as compiled for this trial. The three levels of criteria importance can be expressed as three types of directives and be stated and as follows:

- Criteria under Prim-directives; which are the highest of directives, to which options shall be compliant to with no compromise. Prim-directives are considered more important than other types of directive and none-compliance to prim-directives by options will adversely affect such options in terms of consideration. Prim-directives carry the most weight in the selection matrix.
- Criteria under Directives, which are important to the success of the trial but carry less weight than Prim-directive. None-compliance to criteria considered on a directive level, will have a less adverse impact of options for consideration.
- Criteria under Sub-directives; which are not considered less important than directives, but will most probably have a less adverse impact on the outcomes of the trial. For this reason, criteria under Sub-directives will carry the least amount of all other criteria-types.
- Comparative Directives; even though it was elected to decrease ambiguity by asking questions that can only yield a yes or a no, there are instances where it is required to have some degree of comparison. Comparative directives are aimed as instances where for example cost implications must be assessed.

3.5 Variability Management Option Development and Selection

Even though the trial as described in this dissertation was the result of a mine management decision to find a means to manage ore variability without the use of the DMS, verification of the variability management option selected by the mine management was performed to ensure that the trial efforts are focused on a viable future implementable ore processing change. The following three paragraphs endeavours to explain the options considered by the mine's management.

Option 1 – Current State:

This is the current state at Mamatwan (MMT as it is sometime referred to), performed variability management through the application a hydrometallurgical process called DMS which was explained in Section 2.2.3 of this dissertation. It was also explained that such a processing or beneficiation step requires equipment and resources other than just ore. It is apparent form the discussion that such a hydro-metallurgical processing step will remove a certain amount of low grade ore, apart from the operational complexity and additional raw materials cost. In summary the current state refers to Mamatwan plant as it has been for *the past 2 decades. This option therefore exhibits no benefit in terms of limiting operational complexity, and no benefit in terms of ore losses during DMS beneficiation. Ore variability is thus removed by stripping lower grade material completely and allowing only high grade and sometimes very high grade material to pass to the sintering facility.*

Option 2 - DMS-Less State (Blending Yard):

This option considers the use of replacing the DMS with a Bending-yard. Blending yards are used in several mining endeavours to successfully control ore variability or even providing consistent but different grades of ore to clients with different needs in terms of ore grade and cost. Such Blending-yards require substantial implementation area as well as the addition of controlled deposition, which implies a perfect knowledge of what ore grade is deposited where on the Blending-yard's stockpile, as well as the means to effectively and consistently reclaim differently grade ore material for the Blending-yard stockpile. The conveyor, deposition (stacking) and reclamation infrastructure to facilitate successful Blending-yard operation will bring about significant complication and the mentioned loss of existing area at Mamatwan mine and plant which is one of the constraints already discussed. Computerisation and more proficient operational and maintenance personnel will be required. The capital cost (estimated at R 250 000 000) of such blending yard as well the operational costs brought about by the complex conveyor systems associated with Blending-yard must be taken into account. Obviously such costs would be offset by the none-operation of the DMS facility. Mine management would like to retain the existing DMS facility to accommodate future market demands.

Option 3 - DMS-Less State (Blend Mining):

The third option considered for the replacement of the DMS focused on improving the way in which ore blasted from the ore body is handled or collected. With the variability of Mamatwan ore well validated through several studies, it was envisaged that the DMS could be circumvented if ore collection in the mine pit was such that variability observed within Blast-hole would be managed by controlling where blasted material is collected by haulers and delivered to the primary crushing stage Mamatwan mine. In essence the collection of blasted material would be such that lower grade material will intermix with higher grade material during the collection and supply the primary crusher, that variability will be decreased to such an extent that previously experienced variability will not impact Sinter plant production. In other words, variability will be managed at the pit and not through the beneficiation processing of the DMS plant. This option would require better laboratory capabilities. The capital cost of upgrading the existing Mamatwan laboratory to accommodate Blend-mining practices would be relatively small (R14 000 0000).

As mentioned before in Section 2.6.2.3, the Stuart Pugh approach eventually leads to the compilation of a selection matrix that encompasses all needs, requirements and constraints as selection criteria used for the evaluation of possible concepts developed or available for application to the problem. It was also mentioned that ambiguity can lead to difficulty or complexity in Option Selection, due to vagueness within stated needs, requirements and constraints when stated as selection criteria (Section 2.6.2.4). Based on all of the above mentioned, criteria for selection matrix application in the form of questions, such, that the answers rendered can only be “yes” or “no” are posed. Table 3.5-1 present criteria questions as they are deemed applicable to prim-directive, directives, sub-directives and comparative directives in an effort to verify mine management’s decision to implement Blend-mining on a trial bases. Also note that the current state indicates no implementation of an alternative ore variability management solution other than the existing DMS facility. This could be construed as a base-case for option comparison.

The top or numbered portion of the Table indicates the responses to the questions posed. Note that comparative directives were allowed to have only low (none), medium or high impacts on the selection matrix. From experience with group sessions at the mine it was found that such a comparison is less ambiguous than cases where 1 to 5 or 1 to 10 rating were required.

The bottom portion of the Table give an indication how different directives were counted, weighed and ranked to give the best option for implementation.

The ranking and outputs of the selection process can be seen in the last two rows of the table. Even though Blend-mining was indicated as the best option, the current state with the DMS was ranked as second. The blending yard option was ranked as third or last.

The bottom portion of the table made use of simple Boolean Algebraic function available in Microsoft Excel. A working spread sheet can be provided for more insight into the “If”, “Else”, “True” and “False” logic used to count compliant or “yes” markers, add a weighting the markers form the different directive, and rank the results to provide a very clear Option Selection mostly devoid of ambiguity, which is precisely what is needed during option convergence.

Table 3.5-1 Option Selection matrix for verification of the mine’s decision to perform Blend-mining as an alternative means of ore variability management. Criteria were structured as discussed.

Alternative Variability Management Solution Selection			
Selection Criteria	Concept Variants		
	Current State	DMS-Less State (Blending Yard)	DMS-Less State (Blend Mining)
Prim-Directive	**Compliance**	**Compliance**	**Compliance**

1	**Will Ore Variability be Addressed by the Respective Option's Processing Strategy?**	Yes	Yes	Yes
2	**Will MMT Waste be Avoided?**	No	Yes	Yes
	Directives			
3	**Will Switching of Processing Routes be Supported?**	No	Yes	Yes
	Will Production Expansion be Possible?	Yes	No	Yes
	Sub-Directives			
5	**Does the Process Solution Imply Simplification?**	No	No	Yes
6	**Do the Concept Limit Process Engineering Challenges?**	No	No	Yes
7	**Is Over-all Process Complexity Limited?**	Yes	No	Yes
	Comparative-Directives			
8	**Will capital input cost low (none), medium or high?**	Low	High	Medium
9	**Will Stockpiling area sacrificed be low (none), medium or high?**	Low	High	Medium
	Prim-Directive Yes's	1	2	2
	Prim-Directive No's	1	0	0
	Directive Yes's	1	1	2
	Directive No's	1	1	0
	Sub-Directive Yes's	1	0	3
	Sub-Directive No's	2	3	0
	Prim-Directive Compliance Rating	0	24	24
	Directive Compliance Rating	0	0	16
	Sub-Directive Compliance Rating	-4	-12	12
	Comparative Directive	9	0	3
	Comparative Directive	9	0	3
	Total Compliance Score	14.00	12.00	58.00
	Ranking	**2**	**3**	**1**
	Selection Output (Option for implementation)	**No**	**No**	**Yes**

3.6 Trial Option Development

As part of the trial option development, the management personnel of the mine and plant, as well as lower level operational personnel were consulted in an effort to gain the inputs of all with an interest in the outcomes of the trial without the DMS.

In terms of very high level options there were mainly four views on how to proceed with the trial. One viewpoint was that the trial be performed with Mamatwan ore, but to allow the processed ore to be transported to a third-party sintering facility (concept option 1). Another viewpoint was to transport trial ore, via mine and road haulers to a third party facility for crushing and sintering (complete processing), while normal production at Mamatwan mine and plant continues as normal (concept option 2). A third viewpoint was to establish a temporary mobile ore processing facility that would mimic the existing OPP process but receive material via haulers from the existing ROM (concept option 3). A forth viewpoint was to temporarily reconfigure the existing plant to allow a short term trial to be performed. Such a reconfiguration would be such that no EL (M1L – Mamatwan lumpy would be produced), while also by-passing the DMS (concept option 4).

The options mentioned in the above paragraph can also be construed as the highest possible option concepts for the purpose of initial divergence.

Table 3.6-1 Diagrammatic presentation of the four concept options discussed in the previous paragraph.

Primary Activity	Secondary Activity	Concept Option 1	Concept Option 2	Concept Option 3	Concept Option 4	Concept Option 5	Concept Option 6
Ore Extraction	Drilling and Blasting	Complete Mamatwan Ore Mining and Processing (Primary to Quaternary Crushing & DMS)	Complete Mamatwan Ore Mining and Processing (Primary to Quaternary Crushing)	Mamatwan Drilling & Blasting	Mamatwan Drilling, Blasting & Primary Crushing	Mamatwan Drilling, Blasting & Primary Crushing	Mamatwan Drilling, Blasting & Primary Crushing
Liberation, Classification (processing)	Primary Crushing			Third Party Ore Processing (Primary to Quaternary Crushing)			
	Secondary Crushing				Third Party Ore Processing (Secondary to Quaternary Crushing)	Reconfigured OPP and S7 Conveyor	Reconfigured S7 Conveyor
	Tertiary Crushing						
Beneficiation	Grade Control (DMS)		DMS not part of Trial	DMS not part of Trial	DMS not part of Trial	DMS not part of Trial	DMS not part of Trial
	Carbo-thermal Reduction (Sintering)	Mamatwan Sintering	Third Party Sintering	Third Party Sintering	Mamatwan Sintering	Mamatwan Sintering	Mamatwan Sintering

3.6.1 Concept Option 1

As with Section 3.5 of this dissertation, it was elected to use the current operational configuration of Mamatwan plant as the current state, and to use that as a baseline for comparison with the developed Concept Options 2, 3, 4, 5, 6.

3.6.2 Concept Option 2

Concept option 2 is envisaged to make use of almost all Mamatwan mine and plant's existing infrastructure but to only employ a third-party to accomplish the final sintering step. In other words the sintering will be performed at a sintering facility other than that of the existing sinter plant. This will allow the DMS to be circumvented by removing material from the DMS feed stockpile and transporting this material to another facility. Proponents of this Concept Option maintained that if enough blasted material is available normal production can continue normal production while of the DMS feed is removed for Third-party sintering.

3.6.3 Concept Option 3

Concept Option 3 is very similar to Concept Option 1, but it is envisaged for this option that material will be blasted and directly transported to third-party ore processing and sintering facilities, relieving Mamatwan Mine and Plant of any part in the Trial other that drilling, blasting and hauling material to a road-transport loading point. Proponents of this Concept Option also wanted to involve Mamatwan operational resources as little as possible while the trial commenced. This would suggest that a larger than normal ore block be drilled and blasted.

3.6.4 Concept Option 4

Concept Option 4 is an option that is a mixture of Concept Options 1 and 2, wherein the primary crushing facilities of Mamatwan is utilised while removal of material for Third-party processing is performed at the ROM stockpile. With this Concept Option the difference is also with the establishment of a mobile ore processing facility at Mamatwan. Such a facility will then perform similar crushing and screen as would otherwise be performed by the OPP and quaternary crushing after DMS treatment of material. Mamatwan Sinter plant would be used to perform carbo-thermal-reduction (final beneficiation at Mamatwan). Proponents of this Concept Option maintained that normal production will be minimally disrupted, while partial removal of trial material takes place at the ROM. This would suggest that a larger than normal ore block be drilled and blasted.

3.6.5 Concept Option 5

Concept Option 5 deviates from the three previously stated Concept Options, in that no Third-party is involved in any of the handling of ore material at any stage. This Concept

Option would require some reconfiguration of the screens at the OPP to divert all material that enters the OPP to be processed and supplied to the Mamatwan tertiary and quaternary crushing to be prepared for supply to the Mamatwan Sinter facility. This also implies that only sinter production will be performed while EL (Mamatwan lumpy – M1L) production will not be available. Conveyor S7 will be also reconfigured to circumvent the DMS facility.

3.6.6 Concept Option 6

Concept Option 6 is similar to Concept Option 4 accept for the reconfiguration of the OPP screens. Only reconfiguration of conveyor S7 will be performed to circumvent the DMS facility. This implies that Trial material will be shared between EL production and Trial material production. This would also negate any EL production losses during the trial but would require a larger block of the ore body to be drilled and blasted. Furthermore, it is doubtful that such a configuration will provide sufficient evidence to allow validation and verification of Blend-mining as splitting or ore by the scalping screens will change the effects of commutation on the ore variability during Mamatwan ore processing.

3.6.7 Concept Option Discussion

Without the application of the Stuart Pugh method just yet, some discussion of the stated Concept Options can be performed to allow further insight of the different options. Concept Options that envisages the use of Third-party involvement does necessarily require additional funds for transport of material to and from any Third-party facilities, as well as other raw material and operational costs entailed by the use of such Third-party facilities. Where Third-party facilities are of a mobile sort, funds will be required for the transport to Mamatwan mine and site establishment and de-establishment as well as the operational costs of such mobile ore processing facilities at Mamatwan mine.

Furthermore, it should be noted that Third-party approaches will most probably not cause the same commutation as the commutation resulting from material processed through the Mamatwan ore processing plant (OPP), and may therefore not yield results that are a true reflection of variability control without the DMS when fully implemented at Mamatwan in the future.

From these paragraphs it is rather clear that only Concept Options 4 and 5 should be considered as possible options for the implementation of a Trial without the DMS. However, since it was stated previously that the purpose of following a structure Option Selection approach was to instil trust and cooperation of all involved in the Trial. The modified Stuart Pugh methodology was used to render the best possible option. If this Option Selection methodology proves to be usable it can be utilised in many other applications at the mine and plant at Mamatwan.

Since there are mentioned cost components to the different Concept Options, a Concept Option Discussions or even just trial project considerations cannot be complete without some costing estimates displayed and considered, even if the outcomes of the trial was to evaluate the possibility of an alternative method to the DMS facility to manage ore variability.

Firstly, standard plant ore production costs must be estimated. Please note that the cost values presented are not the exact cost values as provided by Mamatwan mine as such figure are confidential. However, the values used deviates less than +/-25% from actual values provided at the time of the trial. Road transport costs per ton (R210 per ton for 2012 with inflation added this is R336 per ton) were taken from published costs in 2012 (Van Jaarsveld *et al.*, 2013). Mobile plant operating costs provided by a well-known rental plant contractor and was provided as a guideline only to allow Option Selection to be performed. The contractor indicated that for Options 2 and 3, each crushing screening and washing step costs would amount to R36/ton/stpe, with R1.5m and R1m estimated for mobile plant site establishment respectively.

Table 3.6-2 presents close approximations of the costs of each ore processing steps to produce Mamatwan sinter product as per the slightly altered costing information provided by the mine.

Table 3.6-2 Cost estimates for the production of Mamatwan Sinter. Data from this table will be used as input data for trial concept option cost comparison.

Mamatwan - Sinter Cost	
In-pit ore costs	**R/t**
Delivered to Inpit Crusher Cost (Including overheads - loading & hauling)	150.00
Primary Crushing Cost	7.00
In-pit Total	157.00
OPP (Secondary Crushing)	**R/t**
Delivered from In-pit	157.00
Secondary Crushing Cost	9.00
OPP Total (excluding teriary and DMS)	166.00
Teriary Chrusing	**R/t**
Ore Cost as per previous processing step	166.00
Kawa Crusher Cost	9.00
Teriary Crushing Total (Excluding DMS and Quaternary Crushing)	175.00
DMS Cost	**R/t**
DMS Feed as per previous processing step	175.00
DMS Costs (Including Quaternary Crusher - Barmac)	59.00
DMS Total (Including Quaternary Crushing)	234.00
Estimated Quaternary Crusher Costs based in Secondat and Tertiary Crusher costs	9.00
Sintering Cost	**R/t**
Delivered to Sinter	234.00
Sintering Cost	146.00
Delivered from Sinter (Total Cost)	380.00

The cost values presented in this table will enable an estimate of a base-line plant operational cost and will help to offset trial option costs. Note that the cost values takes to some extent inflation and fuel increases into consideration since the performing the trial.

By taking the information presented in Table 3.6-2 into consideration as well as mobile plant costs, estimated plant establishment costs as well as ore transport costs to Third-party facilities, a monetary value for the respective trial option can compiled in one Concept Option cost table (Table 3.7-1). Note that Mamatwan mine and plant costs as well as third party costs are set at a certain value or a zero cost depending on the Concept Option configuration displayed in Table 3.6-1 Diagrammatic presentation of the four concept options discussed in the previous paragraph.. The current state cost (Concept Option 1 – do no trial) was subtracted from the total Concept Option costs to reveal the cost of the Trial Concept Option.

Note that Options 5 and 6 have negative values, which means that those trial options incurred savings in terms of sinter production due to the subtraction of the DMS costs. However, Option 5 did not allow for the continued production of EL (M1L) and that should be considered an EL production loss, which can be brought into the cost equation, but as explained in the in Section 3.6.5, the trial is aimed at validating Blend Mining as an ore variability method. Furthermore, the co-production of EL (M1L) during the trial may impede the integrity of the data generated as the full extent of commutation would become too vague for sensible assessment and interpretation.

Concept Option 2, 3 and 4 would all incur substantial additional costs but would also allow partial production of EL (M1L). Again it should be noted that transportation can influence commutation outcomes due to loading particles and segregation during transport, unloading and reclamation at a Third-party facility. Furthermore, even if Third-party and mobile equipment is similar in operational principle to ore processing at Mamatwan, the geometric differences in Third-party and mobile ore processing equipment could further influence commutation outcomes in terms of variability assessment.

Another aspect not brought into this calculation sheet due to the aim of the trial to validate and verify blend-mining, is the fact that 35% material lost as a result of the DMS operation was not incurred and could be stated as a further saving.

3.7 Trial Option Selection

The Trial Option Selection was performed in a similar fashion as was followed in Section 3.5. However, for the Trial option development and selection significantly more information was collected to assist with the option development and obviously with the stating of criteria for selection. Note that due to the generation of trial concept option costs, options developed were assessed in terms of having cost rankings related to 1st, 2nd, 3rd, 4th and 5th most expensive. No ambiguity can be brought into the selection process by such a ranking as it is stated in cost estimates for the option, but allows mine management to understand the trial impact in monetary terms.

In Table 3.7-2 below, the selection criteria is expressed in questions format related to the criteria structuring already discussed. Note that the focus of the criteria based questions are focused on selection of a concept option that is most likely able to render firstly, validation and verification of Blend-mining, and secondly, to decrease the option impact on safety, access disruption and operational complexity, and thirdly to select the most cost effective concept option bearing the first and second aims of the trial Option Selection in mind and keeping to those priorities.

Table 3.7-1 Cost breakdown for each Concept Option, taking current state, third-party and mobile facilities as well as transport costs and saving in terms of DMS circumvention into consideration

Concept Number	Expected Trial Sinter Tons	Additional Cost to Facilitate Trial Concept Option			Manatwan Mine & Plant Ore Processing Costs per Ton							Third Party Ore Processing Costs per Ton					Total Concept Sinter Product Cost *	Additional Costs incurred by the Trial (not taking EL (M1L) losses into account)
		Shipping Costs (To 3rd Party Facilities) per Ton	Mobile Plant Establishment	Plant Alterations	Blasting & Hauling	Primary Crushing	Secondary Crushing	Tertiary Crushing	DMS Treatment	Quaternaty Crushing	Sintering	Primary Crushing	Secondary Crushing	Tertiary Crushing	Quaternaty Crushing	Sintering		
Concept Option 1	18000	R-	R-	R-	R150.00	R7.00	R9.00	R9.00	R59.00	R9.00	R146.00	R-	R-	R-	R-	R-	R 7 002 000.00	R -
Concept Option 2	18000	R210.00	R-	R-	R150.00	R7.00	R9.00	R9.00	R-	R9.00	R146.00	R-	R-	R-	R-	R-	R 9 720 000.00	R 2 718 000.00
Concept Option 3	18000	R210.00	R1 500 000.00	R-	R150.00	R36.25	R36.25	R36.25	R-	R36.25	R146.00	R8.75	R11.25	R11.25	R11.25	R167.90	R 16 505 200.00	R 9 503 200.00
Concept Option 4	18000	R-	R1 000 000.00	R-	R150.00	R7.00	R36.25	R36.25	R-	R36.25	R146.00	R-	R11.25	R11.25	R11.25	R167.90	R 12 041 200.00	R 5 039 200.00
Concept Option 5	18000	R-	R-	R700 000.00	R150.00	R7.00	R9.00	R9.00	R-	R9.00	R146.00	R-	R-	R-	R-	R-	R 6 640 000.00	-R 362 000.00
Concept Option 6	18000	R-	R-	R700 000.00	R150.00	R7.00	R9.00	R9.00	R-	R9.00	R146.00	R-	R-	R-	R-	R-	R 6 640 000.00	-R 362 000.00

* Current State Cost

Table 3.7-2 Option Selection matrix for Trial Concept selection. Criteria were structured as discussed.

Trial Option Solution Selection							
Selection Criteria		Trial Concept Variants					
		Concept Option 1	Concept Option 2	Concept Option 3	Concept Option 4	Concept Option 5	Concept Option 6
Prim-Directives		Compliance	Compliance	Compliance	Compliance	Compliance	Compliance
1	Will blend-mining be fully validated and verified by the Concept Option?	No	Yes	Yes	Yes	Yes	No
2	Will existing stockpiles be unaffected by the Concept Option?	Yes	Yes	Yes	No	Yes	Yes
3	Will existing HMV activity be uninterrupted by this Concept Option?	Yes	Yes	No	No	Yes	Yes
Directives		Compliance	Compliance	Compliance	Compliance	Compliance	Compliance
4	Will operational complexity be decreased by the Concept Option?	Yes	No	No	No	Yes	Yes
5	Existing Access Routes be uninterrupted by the Concept Option?	Yes	No	No	No	Yes	Yes
6	Will existing Production be uninterrupted?	Yes	Yes	Yes	Yes	No	Yes
Sub-Directives		Compliance	Compliance	Compliance	Compliance	Compliance	Compliance
7	Will Implementation be Simplified?	Yes	No	No	No	Yes	Yes
8	Will Logistics be Simplified?	Yes	No	No	No	Yes	Yes
9	Do the Concept Limit Operational Challenges?	Yes	No	No	No	Yes	Yes
Comparative Directives		Ranking	Ranking	Ranking	Ranking	Ranking	Ranking
10	Rank the cost of the Trial Concept Option (1 for highest and 5 for lowest)	4	3	1	2	5	5
	Prim-Directive Yes's	2	3	2	1	3	2
	Prim-Directive No's	1	0	1	2	0	1
	Directive Yes's	3	1	1	1	2	3
	Directive No's	0	2	2	2	1	0
	Sub-Directive Yes's	3	0	0	0	3	3
	Sub-Directive No's	0	3	3	3	0	0
	Prim-Directive Compliance Rating	10	30	10	-10	30	10
	Directive Compliance Rating	24	-8	-8	-8	8	24
	Sub-Directive Compliance Rating	12	-12	-12	-12	12	12
	Comparative Directive Compliance Rating	8	6	2	4	10	10
	Total Compliance Score	*54.00*	*16.00*	*-8.00*	*-26.00*	*60.00*	*56.00*
	Ranking	*3*	*4*	*5*	*6*	*1*	*2*
	Continue	*No*	*No*	*No*	*No*	*Yes*	*Yes*

3.8 Trial Alterations –Hardware, Stockpiles and Processing

3.8.1 Hardware Alterations

The hardware alterations required for the successful completion of the Trial without the DMS is related to the reconfiguring the OPP in such a way that all material from the Blast-Hole eventually ends up Sinter Product, as well as the circumvention of the DMS facility, while still transferring material from the tertiary crusher to the quaternary crusher systems. In summary the following alterations will be performed.

Screening re-configuration at the OPP Screening house which will enable all material fed into the OPP to be delivered as ES (EVA sinter feed destined for tertiary and quaternary crushing and eventual sintering):

- Replace 75mm screens with 60mm screens
- Replace 35mm screens with 50mm screens

The changes in the apertures of the OPP scalping and separation screens will prevent lumpy ore (EL) material from being produced (EL – 40 to 75mm). Only tertiary crusher feed material would then be produced (ES – 6 to 40mm). This is the only material that the OPP will produce under such a plant configuration and therefore all Blast-Hole material will be crushed through tertiary and quaternary crushers to produce SF. OPP fines (EF - <6mm) material will still be produced by the OPP but will be transferred from the EF stockpile to the SF stockpile via front-end loaders.

To circumvent the DMS facility and therefore perform no beneficiation before sintering or carbo-thermal-reduction is achieved, material that normally would be conveyed to the stockpiles feeding the DMS plant had to be conveyed in such a way that the material still reached the quaternary crusher circuit and be delivered to the sinter feed stockpiles. To achieve this it is important to first review normal process flows. The flow of material was discussed in Sections 2.2.2 and 2.2.3 and is augmented with Figure 5.2-1 and Figure 5.2-2 (Figure 5.2-2, which is a simplification of Figure 5.2-1). In this simplified normal process flow diagram, S7 conveyor, and its delivery transfers material from the tertiary crushers and onto conveyor DCV15 which delivers material to the DMS feed stockpile. However, extending the western tip of conveyor S7 to close proximity to conveyor DCV9, and by installing spillage chute extension between an existing spillage chute on conveyor DCV9, which is part of the quaternary crusher circuit directly after the DMS, and by reversing the direction of conveyor S7, material can be transferred directly from the tertiary to the quaternary crusher and thus, circumventing the DMS process completely while still retaining the final crushing step to produce correctly size sinter feed (SF).

3.8.2 Stockpile Positions and Materials Flow

As mentioned before, workshops held to ensure the selection of the best options, but these workshops were also instrumental in understanding not only hardware change-management by also to ensure all personnel as aligned with the stockpiles to be used. The diagram in Figure 3.8-2 presents a simplified process from diagram (PFD) of the Mamatwan plant and is aimed as indicating the relevant stockpiles diagrammatically. This also gives anyone not familiar with the operations of Mamatwan Plant an understanding of the processes involved. Note that the diagram in Figure 3.8-2 is a significantly simplified process flow diagram from more detailed process flow diagrams (PFD's) in Appendix B

Figure 3.8-1 presents an aerial photograph of the plant and indicated the respective piles with the brown ovals.

Figure 3.8-1 Stockpiles of importance in terms of management and sampleing during the trial are indicated with the bown ovals in the aerial picture

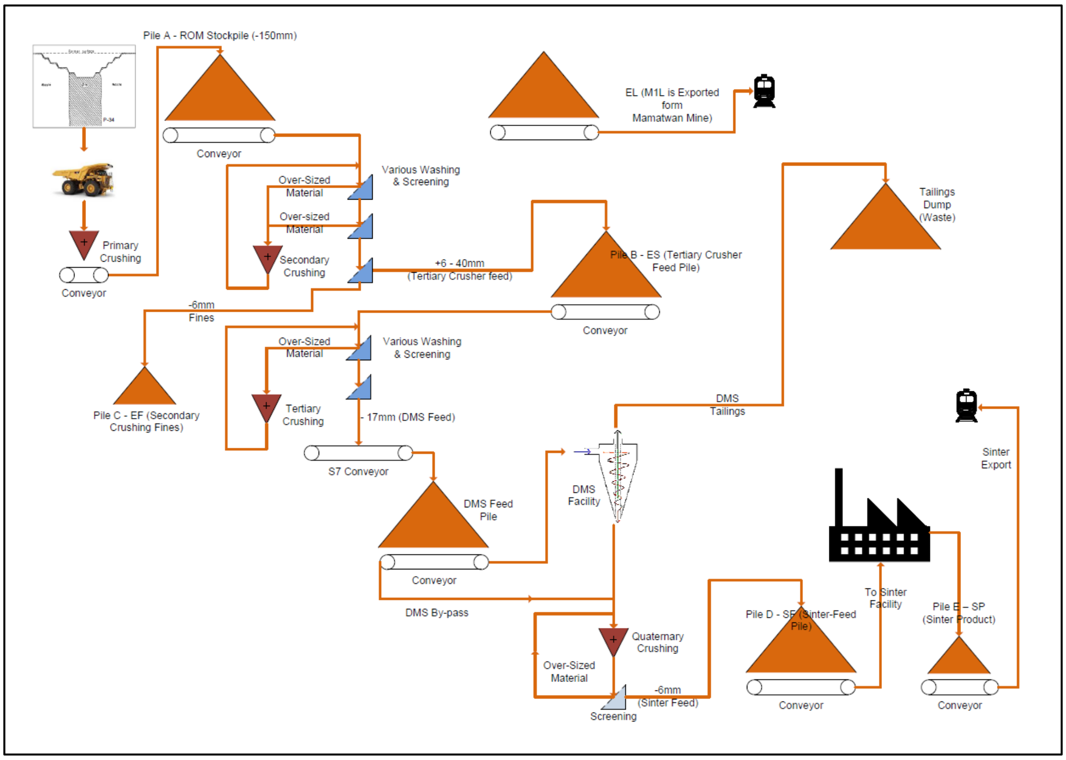


Figure 3.8-2 Simplified ore processing flow diagram (PFD) with indication of the Stockpiles A, B, C, D and E. Note in that in the diagram the word stockpile was substituted with pile due to space or diagram clutter considerations

Stockpiles are of great importance, as material is temporarily stored on stockpile and then transferred to the next processing stage. The stockpiles can be described as follows:

- Stockpile A (ROM) – ROM Stockpile (for this trial the ROM pile will contain Blend Mined Ore)
- Stockpile B (ES) –Tertiary Crusher Feed Pile produced by crushing and screening through re-configure OPP. Material from this pile will be referred to as ES (Eva Sinter feed) material during in this dissertation, and consists of Blend Mined crushed and screen material with particle size ranging between 6 and 40mm.
- Stockpile C (EF) – Material on this pile is considered a by-product of the OPP crushing, washing and screening process. Material from this pile will be referred to as EF (Eva Fines) material during in this dissertation, and consists of Blend Mined crushed and screen OPP by-product material with particle size smaller than 6mm.
- Stockpile D (SF) – Tertiary Crushed and screened material is then transferred directly to the quaternary crushing and screening circuit via the altered S7 conveyor system to be crushed and screened a fourth time. Material generated in this manner will be referred to as SF (Sinter Feed) material during in this dissertation, and consists of Blend Mined crushed and screen material with particle size smaller than 6mm.
- Stockpile E (SP)– Sinter Product Stockpile for sintered Trial without the DMS Material

3.8.3 Stockpile Sizes

For the piles indicated in Section 3.8.2 the sizes and volumes is foreseen is presented in Table 3.8-1.

Table 3.8-1 Trial without the DMS Pile Capacities and Required Plant Surface Areas

Stockpile Designation	Stockpile Capacities (Tons)	Required Surface Area
Stockpile A (ROM)	27 000	1484
Stockpile B (ES)	28 000	2400
Stockpile C(EF)	7 500	750
Stockpile D (SF)	7 500	750
Stockpile E (SP)	12 000	3418

3.8.4 Description of Trial Material Flow

The flow of Blend mined ore through the plant during the Trial without the DMS will be as follows (please refer to Appendix B for more detail on process flows as well as Section 2.3):

- Blend mined ore material will be conveyed from the pit to Pile A (ROM)
- From Pile A ore will be conveyed to the OPP via Conveyor R2

- Material crushed, washed and screened through the re-configured Screening House will be conveyed to Pile B (Area 6)
- Fines from the OPP will be conveyed to Pile C (F2 Fines Area), or directly to the Quaternary Crushers
- Material from Pile B will be transferred to the Tertiary Crushers
- Material crushed through the Tertiary Crushers will then conveyed via Conveyor S7 to Conveyor DCV9, which forms part of the Quaternary Crusher Circuit
- Material Crushed through the Quaternary Crushers will be transferred to Pile D (Area 3), which is Sinter Feed Material
- From Pile D Trail without the DMS material will be supplied to the Sinter Plant
- Material subjected to Sintering will be transferred to SGS Product Pile 1.

3.9 Blend Mining Trial Data Analyses

Normal plant production was stopped and material processed under standard pit conditions were not retrieved anymore. Only a new block of material considered trial only material was considered for processing using Blend-mining. The data presented in this section is related to the Blend-mining trial only with the circumvention of the DMS, and with no production of EL or Mamatwan lumpy, also known as M1L.

Sampling for the trial without the DMS were such, that material was extracted from the Blast-Hole (normally not analysed), tertiary crusher feed (ES +6-40mm), OPP fines (EF -6mm), and sinter-feed without the DMS (SF (Sinter Feed)).

In other words, samples were collected at five sampling points, which is indicated in the process diagram presented in Figure 3.9-1. This diagram is augmented by an aerial picture with arrows and descriptions to further indicated the different sampling points as relevant to the trial (Figure 3.9-2).

In the diagram presented in Figure 3.9-1, the portions of the plant that was not used during the trial is clearly indicated with the red demarcations. The lumpy stockpile (+40 to 75mm – M1L), and it related rail transport was not utilised. This was accomplished by reconfiguring the OPP to only produce tertiary crusher feed.

The DMS was circumvented by reconfiguration of a conveyor system to run in the opposite direction which delivered material not to the DMS facility but to the quaternary crushers directly after tertiary crushing.

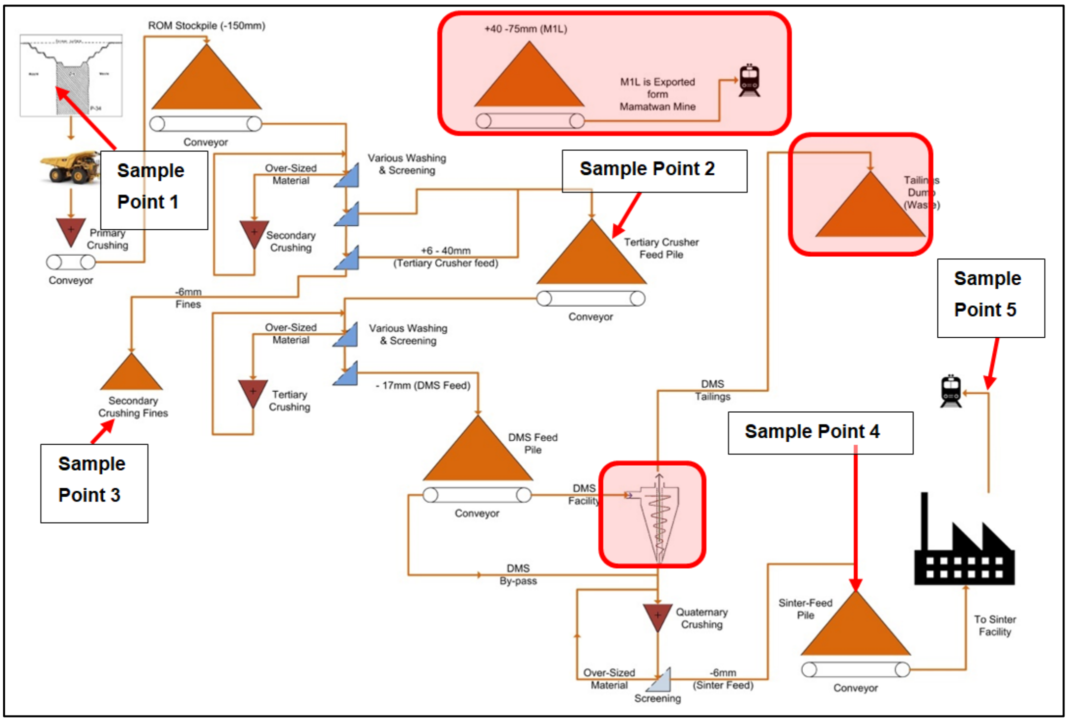


Figure 3.9-1 Diagrammatic presentation of a portion of the Mamatwan mine and plant. The trial sampling mentioned is related to the pit blast-hole, secondary crushing step, with reconfigured screens to deliver on tertiary crusher feed, OPP fines, Sinter Feed, and Sinter Product.

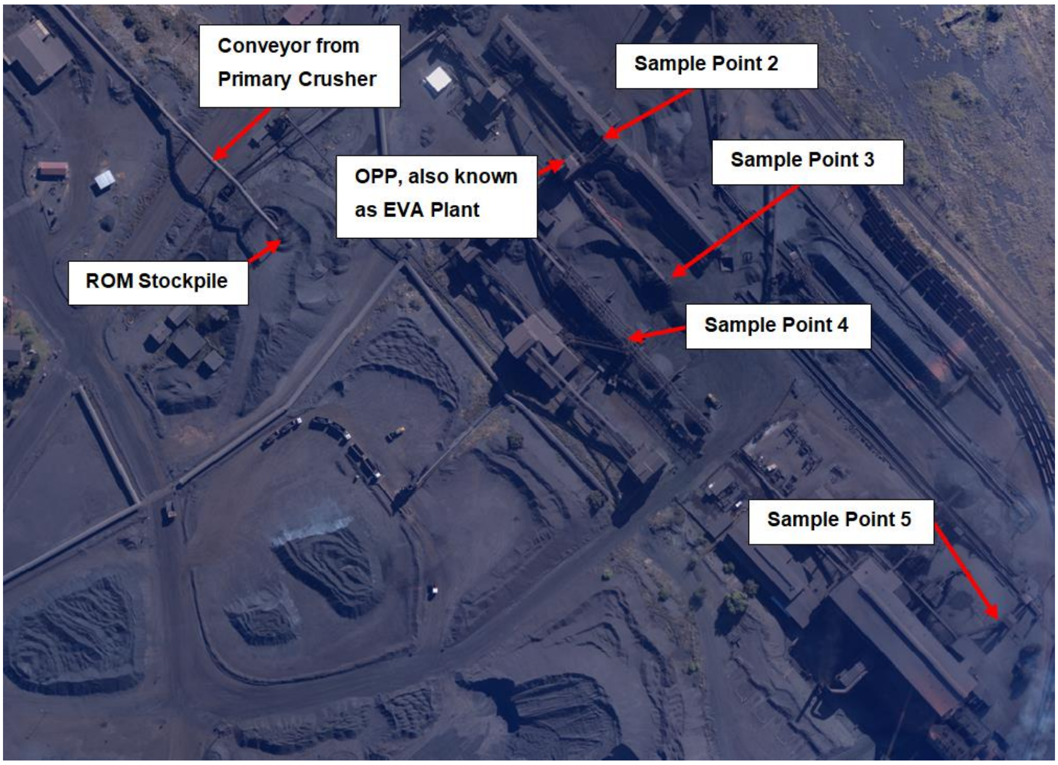


Figure 3.9-2 Aerial photograph of the Mamatwan Plant. **Blend Mining Feed Analyses (In-Pit)**

Blast-hole samples were taken at even intervals along the blasted ore pile and subjected to chemical analyses. The results of the Blast-Hole analyses are presented graphically in

Figure 3.9-3 through Figure 3.9-5, which is related to the manganese (Mn) and iron (Fe) content as well as the calculated Mn:Fe ratio.

The Mn content variation was slightly greater than 10%. The Fe content variation was just more that 3%, with the Mn:Fe ratio variation slightly larger than 3.5%.

These In-pit analyses results clearly displays the level of variability present in the material.

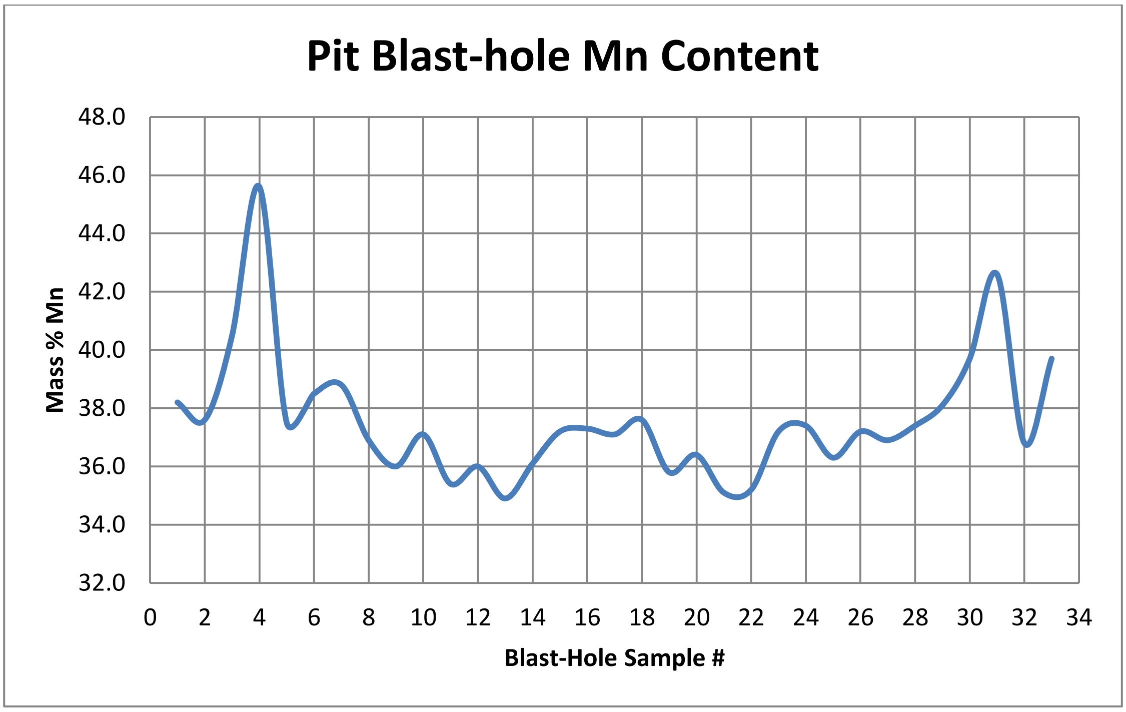


Figure 3.9-3 Graphic presentation of Blast Hole Analyses results. As indicated before, the main variability concern is related to the Mn content.

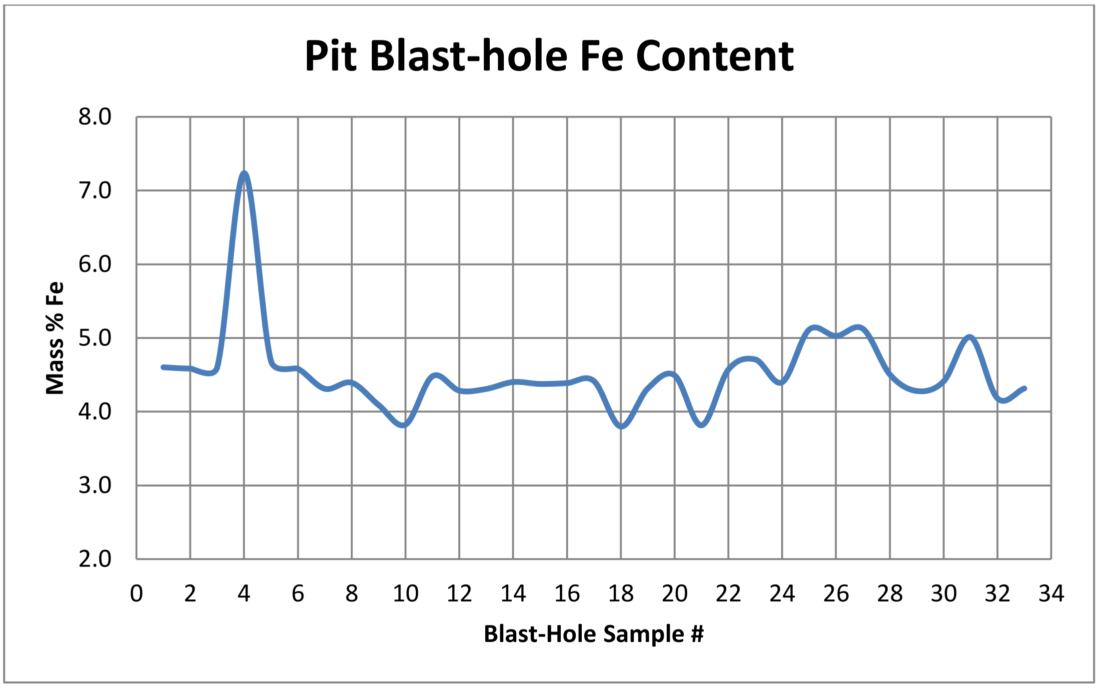


Figure 3.9-4 Graphic presentation of Blast-Hole analyses results. Variability of more than 3% was observed for Blast-Hole-Samples in terms of Fe content.

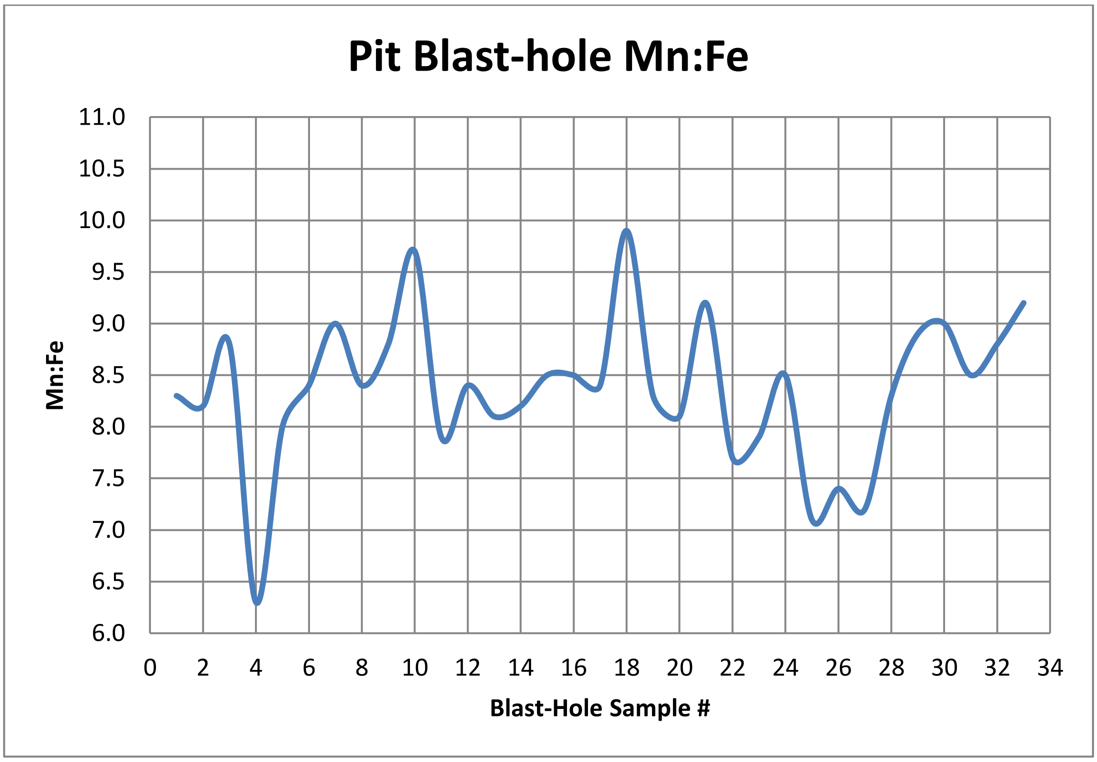


Figure 3.9-5 Graphic presentation of Blast-Hole analyses results. Variability of more than 3.5% was observed for Blast-Hole-Samples in terms of Fe content.

To further express the variation that is present in blasted ore, the data as presented Figure 3.9-3, it was elected to present Mn content as a variation above or below the 37%

Mn content goal. When presenting the data in this manner, the goal of 37% is set to 0 and variation in Mn content is then easily as percentile above and below 0.

Figure 3.9-6 presents the Blast-hole sample Mn content as a variation above or below the goal as stated in the previous paragraph. When reviewing the data in this manner, it is clear the 35% of Blast-hole material are not acceptable and will be removed by beneficiation methods such as the DMS. This value of 35% is in line with DMS tailings produced at Mamatwan Mine. Also more easily observed were the fact that sample of the ore had a Mn content almost 9% higher than the goal of 37%, and almost a 2% lower Mn content than the state goal.

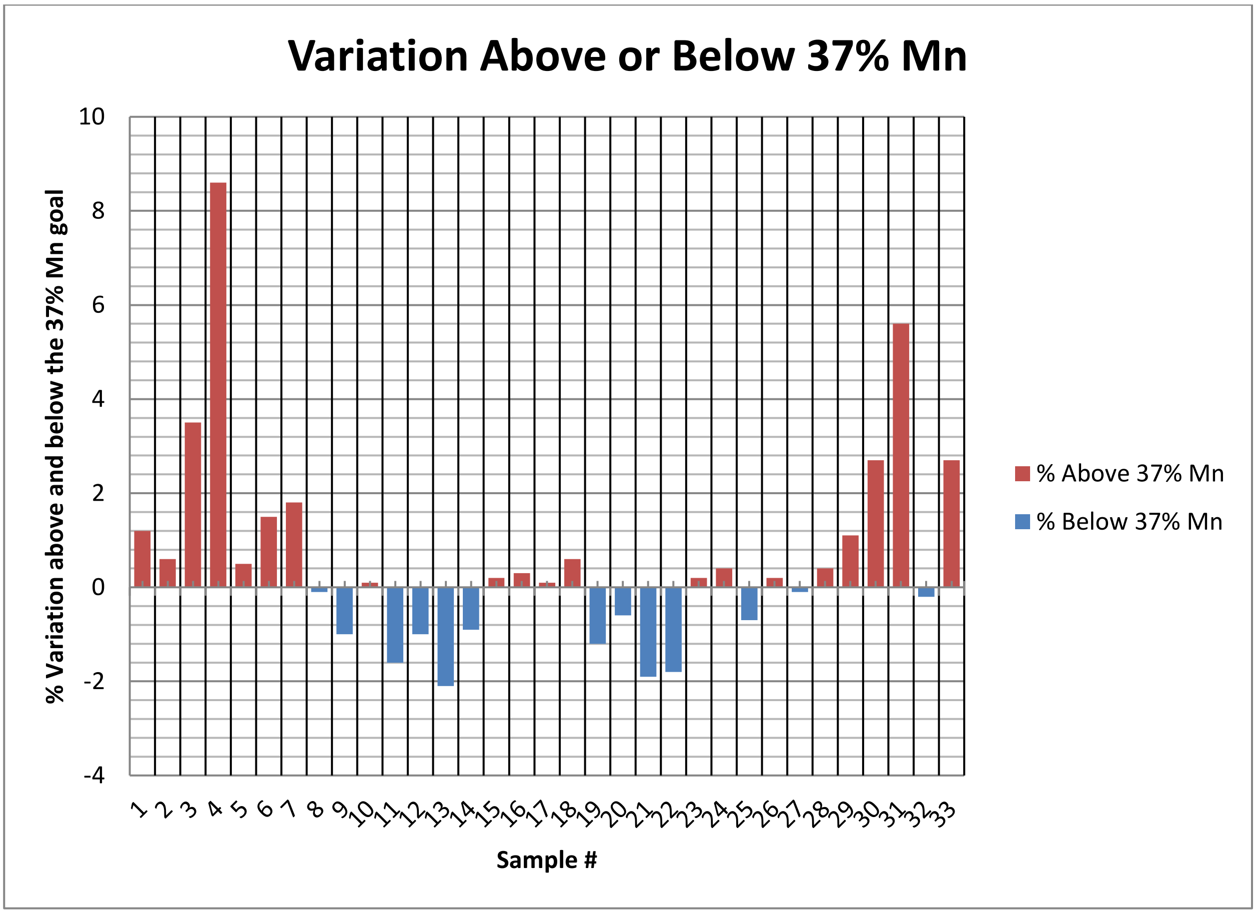


Figure 3.9-6 Variation of ore Mn content expressed as a percentile value above or below the 37% goal which is for the sake of this graph the 0 axis.

A summary of the data presented in Figure 3.9-6 can again be presented in Table 3.9-1 after counting samples below, matching and above the stated goals. From the data presented in this Table and in proceeding graphs, it is clear that 60.6% of material is above the goal of 37%, while 39.4% is below the goal. However, from the distribution evident in Figure 3.9-6, samples high in Mn content were found on opposite sides of the Blast Hole material. In other words, the distribution of Mn content was not evenly distributed throughout the Blast Hole material.

Table 3.9-1 Percentage samples below, above and matching the Trial Goal

	% Below the Goal	% Matching the Goal	% Above the Goal
Blast Hole	39.4%	0.0%	60.6%

3.9.1 Blend Mining Feed Analyses (OPP & Sinter Plant)

Samples taken during the trial was subjected to Compositional Chemical Analyses, which is presented in the following graphical presentations. Figure 3.9-7 through Figure 3.9-13 presents the variance in the composition of the Blend Mining feed material through the processing steps in the plant. Each graphical presentation depicts only one Main Constituent and its variance. Graphs for Manganese (Mn), Iron (Fe), Calcium Oxide (CaO) and Mn:Fe ratio is presented in the set of figures. Please note that the axis's of the graphs in Figures Figure 3.9-8 through Figure 3.9-11 were kept the same as in Figure 3.3-7 through Figure 3.3-9 to allow comparison, with pre-trial sampling data.

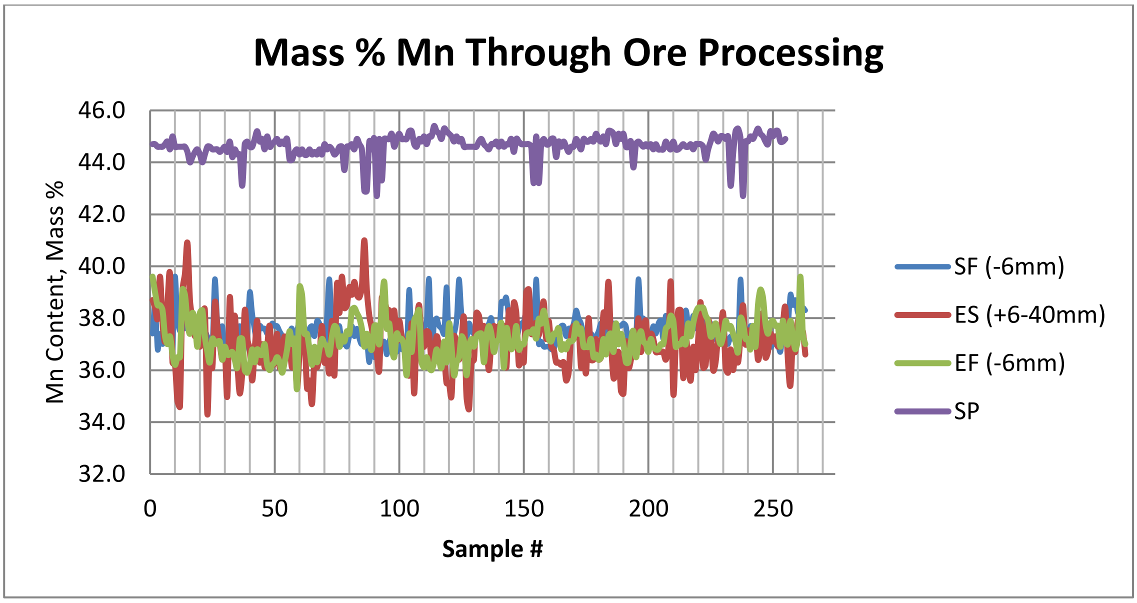


Figure 3.9-7 Mass % Mn in the trial feed from the OPP that prepares tertiary crusher feed (ES), OPP Fines (EF).

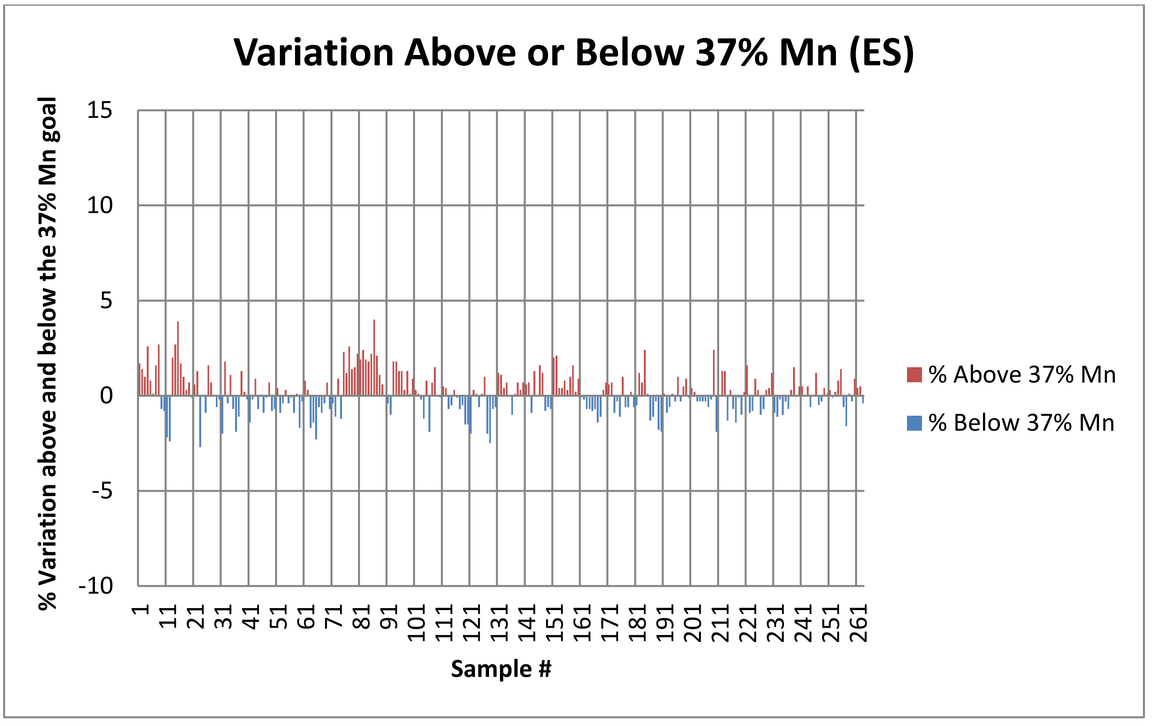


Figure 3.9-8 Graph of the plotted variation around the 37% Mn content goal for tertiary crusher feed (ES).

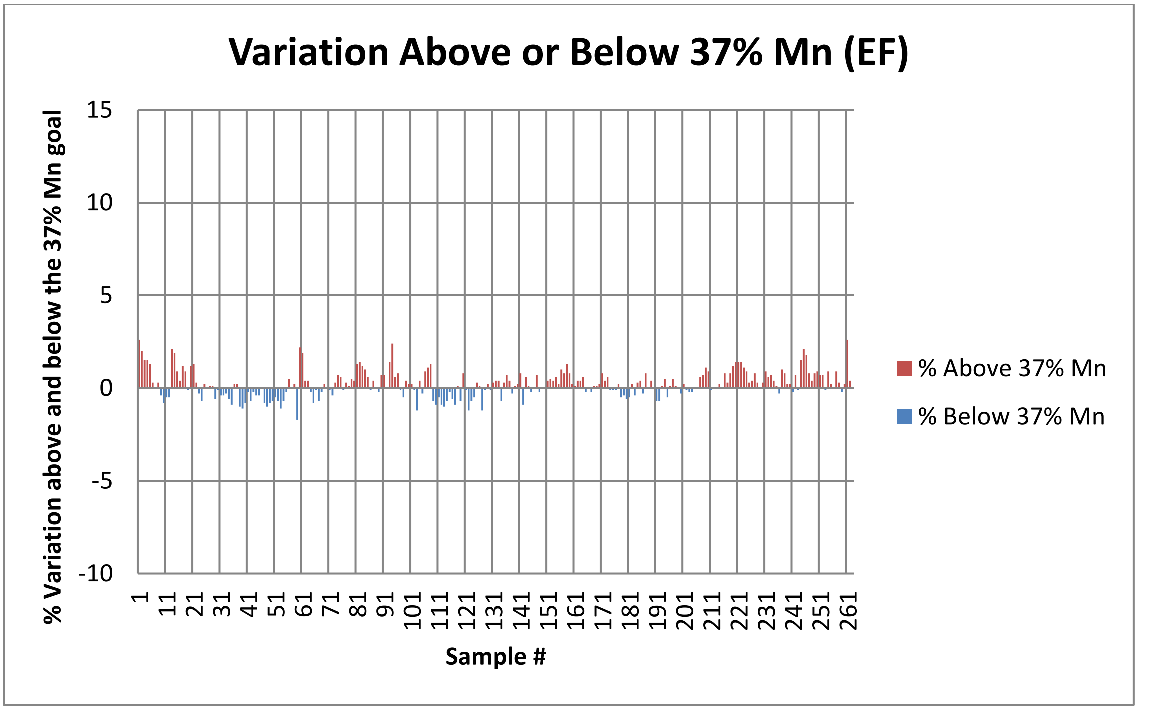


Figure 3.9-9 Graph of the plotted variation around the 37% Mn content goal for OPP Fins (EF).

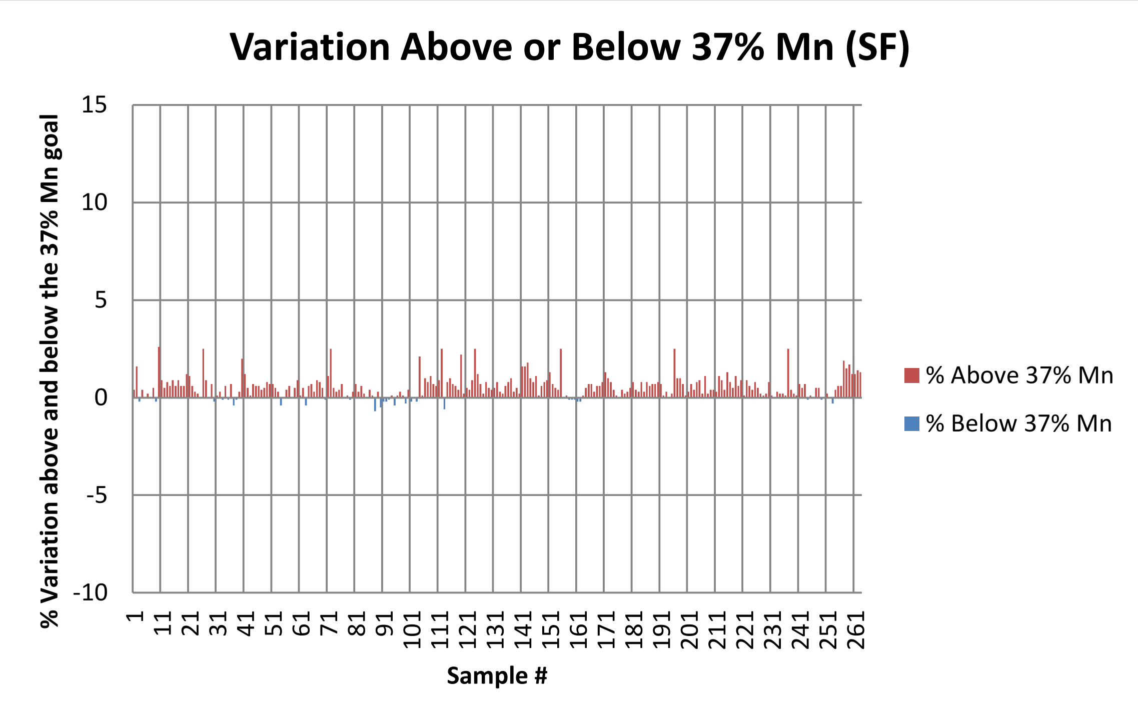


Figure 3.9-10 Graph of the plotted variation around the 37% Mn content goal for Sinter Feed (SF).

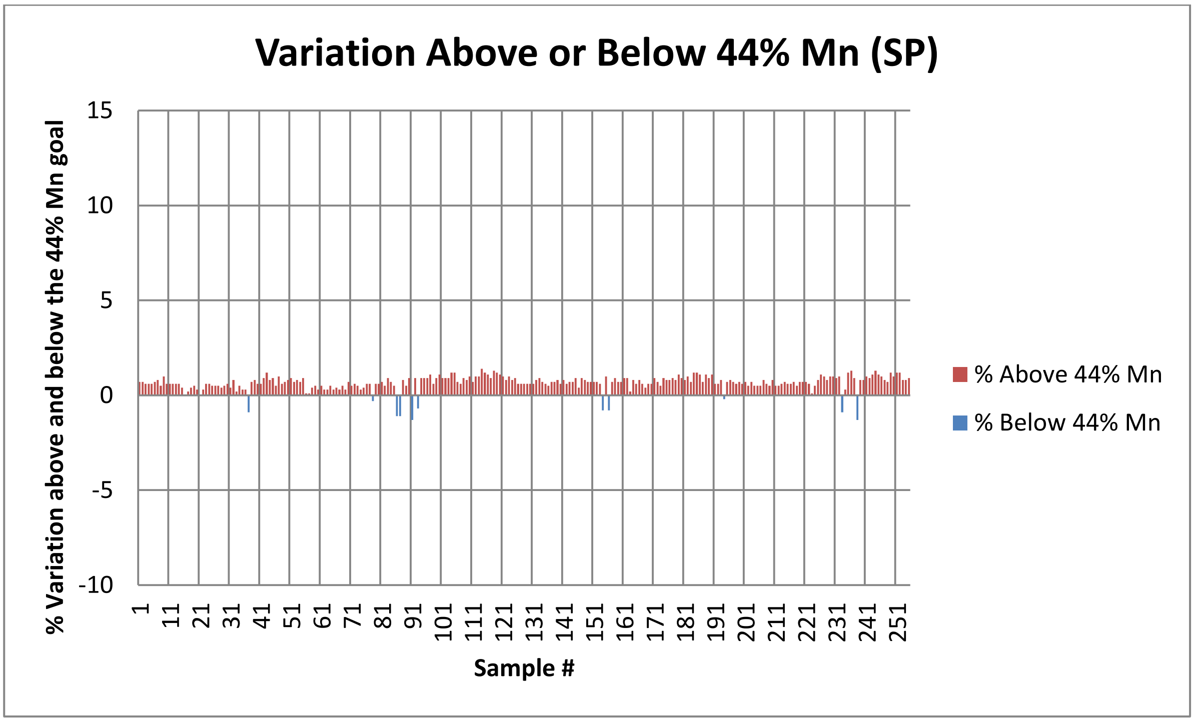


Figure 3.9-11 Graph of the plotted variation around the 44% Mn content goal for Sinter Product (SP).

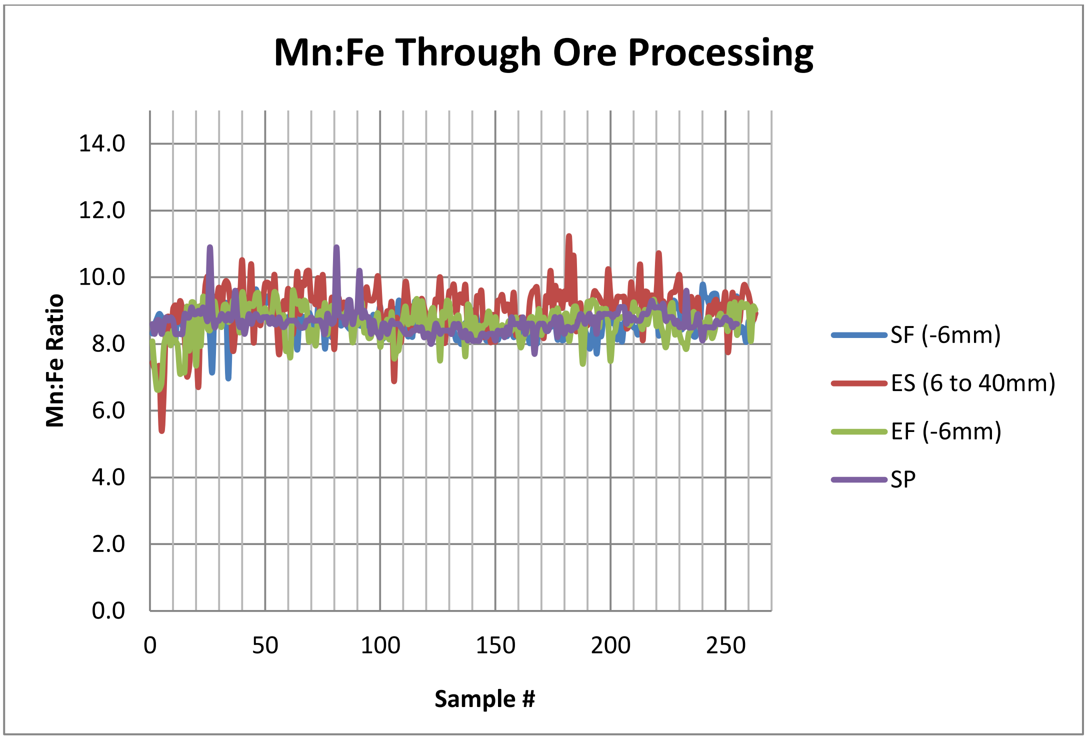


Figure 3.9-12 The Fe content were also determined for the previously mentioned sampling points and is expressed in this graph in terms of the Mn:Fe ratio.

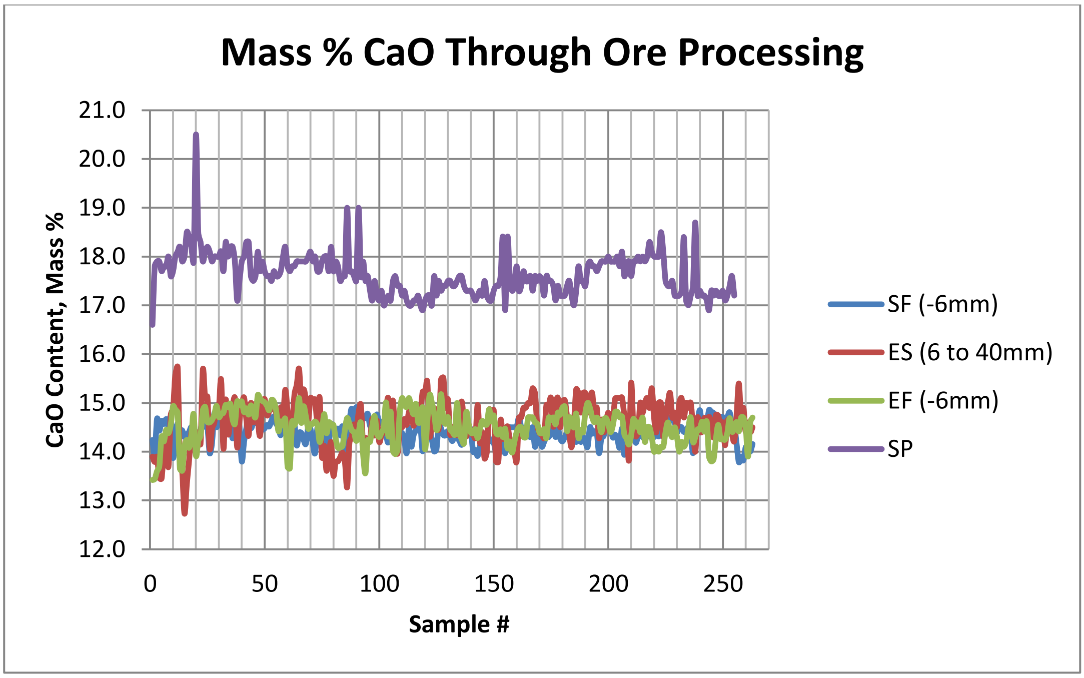


Figure 3.9-13 The CaO content were determined for the previously mentioned sampling points and is expressed in this graph.

A summary of the data presented in Table 3.9-2, and Figure 3.9-8 through Figure 3.9-11 Figure 3.3-9 can be presented in the Table below, after counting samples below, matching and above the stated goals. From the data presented in Table 3.9-2 Summary of graphs presented in and in preceding graphs, and comparing that to Figure 3.3-7 through Figure 3.3-9 and it's preceding graphs, it is clear that Blend Mining achieved a better distribution of samples above en below the goals and also facilitated lower variances than observed in pre-trial sample analyses data when considering ES and EF samples. For SF, it is suggested that commutation and the fact that amount of samples below goal as well as the small level below the goal was responsible for the disappearance of below goal material. This small level of below-goal-material led to further improvements during carbo-thermal reduction or sintering, which is a very effective beneficiation step.

Table 3.9-2 Summary of graphs presented in Figure 3.9-5, Figure 3.9-8 and through Figure 3.9-11.

	% Below the Goal	% Matching the Goal	% Above the Goal
Blast Hole	39.4%	0.0%	60.6%
EL (not produced)	n/a	n/a	n/a
ES	35.4%	8.4%	56.3%
EF	44.9%	4.2%	51.0%
SF	11.0%	6.1%	82.9%
SP	4.3%	0.8%	94.9%

3.9.2 Blend Mining Product Analyses

As indicated before, the following sinter product specifications were elected and set as a Trial without the DMS Goal (Blend Mining). The Trial Specifications were as follows:

- Mn – 44%
- CaO – 17%
- Mn:Fe – 8.3 with 7.8 the lower acceptable limit

To summarise the results in a very easy to ready format, the daily averages for the chemical constituents present in the Blend Mining Sinter product (SP). Note that even though the total trial was performed over 14 days, sinter production was only performed over 7 days. This is related to the build-up of a certain amount of sinter feed before sinter production could commence. Therefore, 7 days were allowed for sinter feed accumulation.

Table 3.9-3 Summary of the daily main constituent content averages and SP production.

Trial Day (SP Production)	Mn	Fe	CaO	Mn/Fe	Tons
1	44.8	5.3	17.7	8.5	2167
2	44.5	5.1	18	8.8	2971
3	44.4	5.1	17.9	8.8	2972
4	44.9	5.3	17.3	8.4	2631
5	44.7	5.3	17.4	8.4	2771
6	44.8	5.1	17.6	8.7	2886
7	44.5	5.1	17.8	8.7	2783
Total Blend Mining Sinter Product Produced					19181

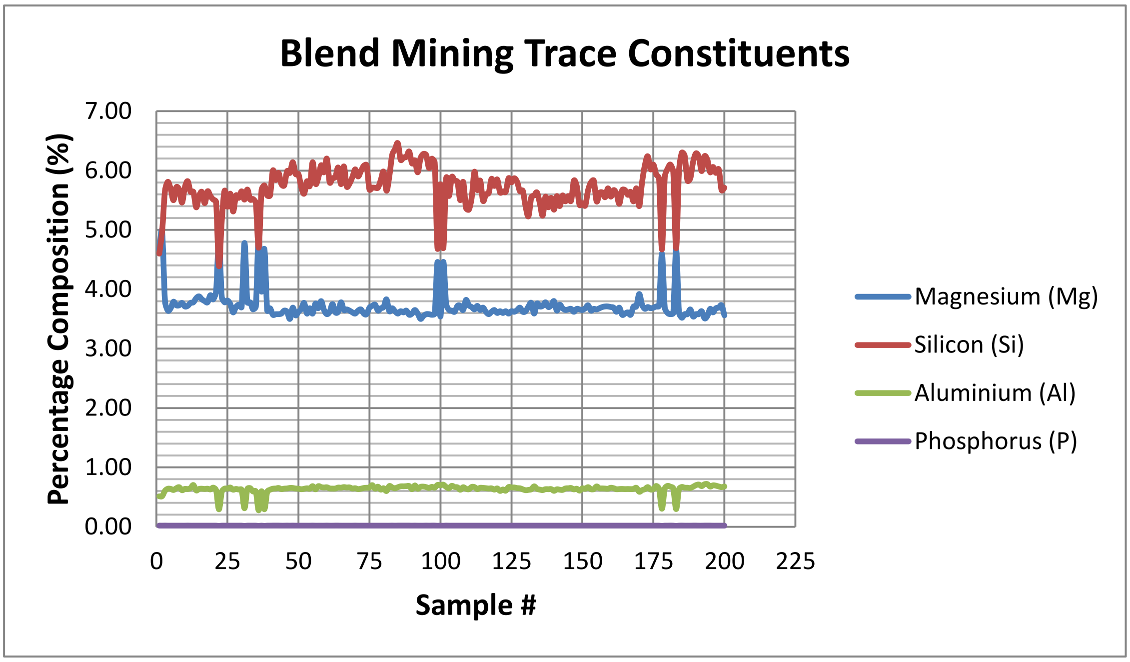


Figure 3.9-14 Presents the composition of the Blending Mining Sinter Product in terms of Magnesium (Mg), Silicon (Si), Aluminium (Al), Phosphorus (P).

Data sheets used for rendering the graphs are available on request.

3.9.3 Analyses Data Distribution and Discussion

To better illustrate the accuracy of the Blend Mining effort, a distribution of sample compositions in terms of Mn content, Mn:Fe ratio and CaO contents are presented graphically in this section. As indicated previously, certain goals were set for the trial and these goals as discussed in Section 3.4.4 are indicated on the graphs with heavy red and orange brown lines (additional legends on the graphs will aid the description of the goals).

The distribution of Mn content, Mn:Fe ratio, and CaO content versus the number of samples are presented via the graphs in Figure 3.9-15 through Figure 3.9-17. Noticeable on these graphs is the profound effect of the sintering process on the beneficiation of the sinter feed in terms of Mn and CaO content.

When reviewing the data presented in these graphs, it is clear that the trial goals were conformed to, accept for the CaO content. However, when taking Figure 3.9-17 into consideration and calculating the average CaO content for the SP, this goals was missed by 0.6%, as the average CaO content for the 19 000 tons of SP produced was 17.6%. This would have some implications for the next step on the ferromanganese production chain as raw materials added to aid the formation of slag during the smelter furnace reaction includes a certain amount CaO to control the basicity of the slag (Sun *et al.*, 2010). CaO is added to control the ratio of basicity ratio between (CaO + MgO)/SiO2. For the production of HCFeMn generally a basicity ratio of close to 1.3 ± 0.1 is targeted (Lagendijk *et al.*, 2010). However, the influence and effects on smelter efficiency and production rate of such a slight excess of CaO over the goal in the trial sinter product should be studied further.

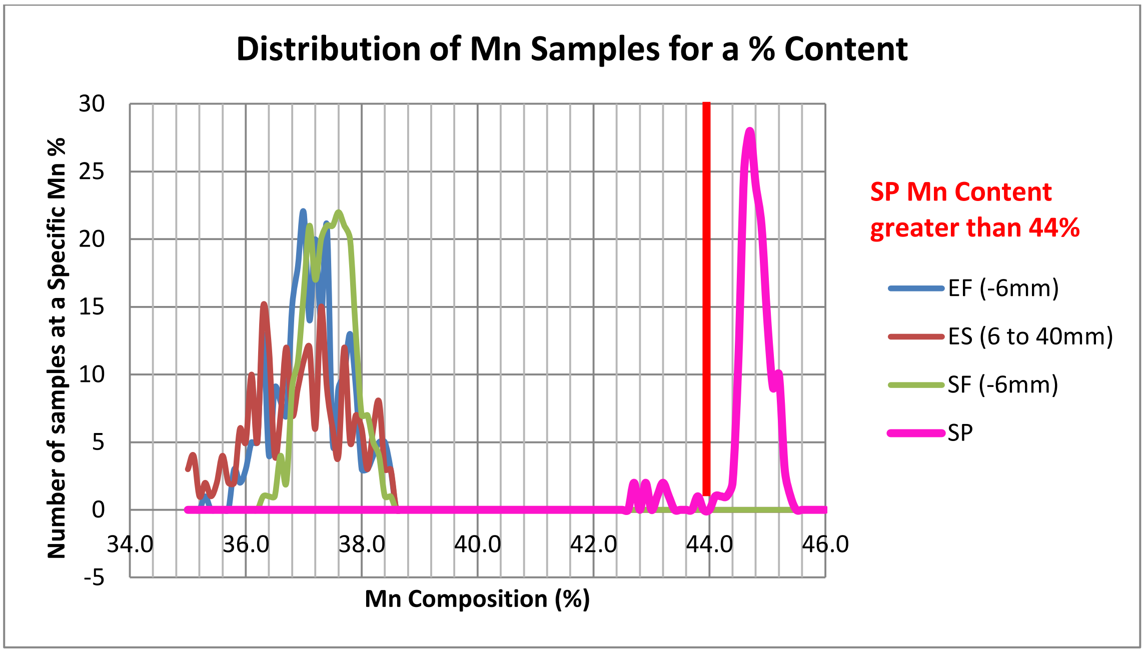


Figure 3.9-15 Distribution of sample compositions in terms of Mn % for ES, EF, SF and SP, as well as the goal set for the trial.

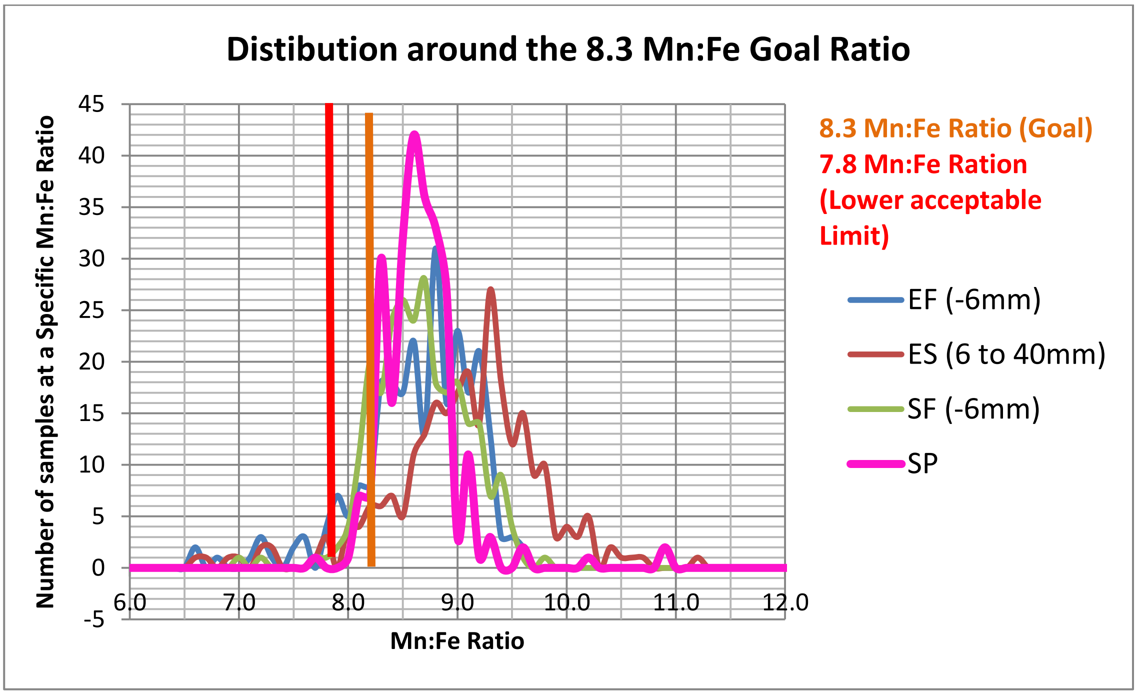


Figure 3.9-16 Distribution of sample compositions in terms of Mn:Fe for ES, EF, SF and SP, as well as the goal set for the trial.

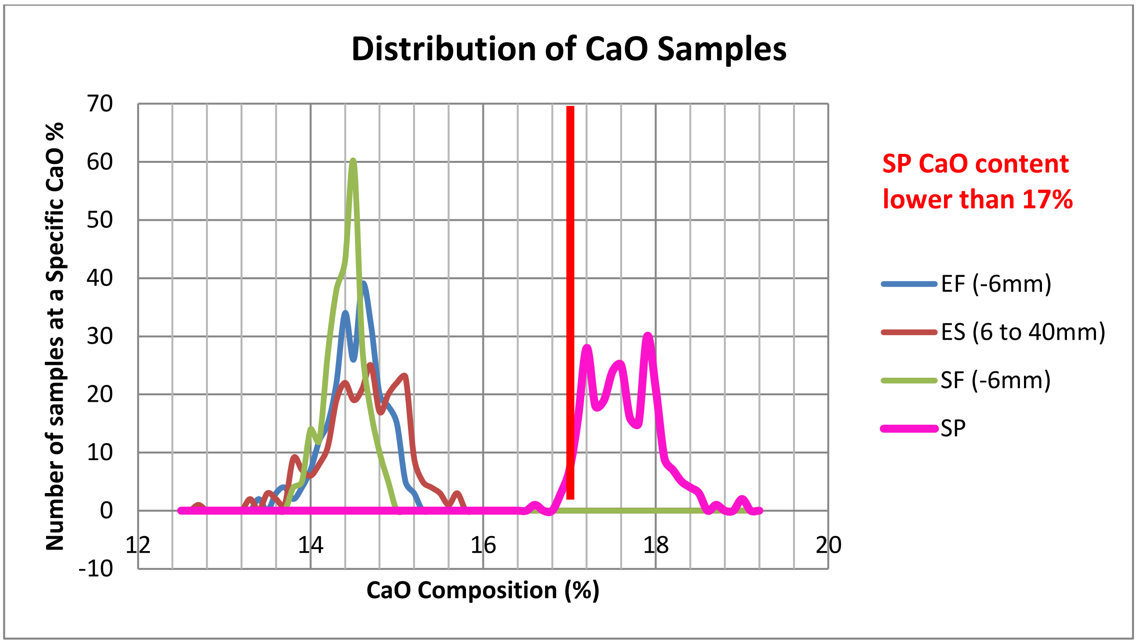


Figure 3.9-17 Distribution of sample compositions in terms of CaO contents for ES, EF, SF and SP.

3.9.4 Particle-Size-Distribution

A Particle Size Distribution sample was taken from SF on 22/10/2010 during the Trial, as well as on 16/11/2010 during normal production, to determine the distribution of fines material in the final product before sintering. Figure 3.9-18 below indicates the PSD's for the mentioned samples.

From the available data, it would appear that PSD for the Trial sample was not as even as the PSD for the normal Production sample. However, a greater percentage of the larger size fractions were evidenced for the Trial sample. A decrease in the percentage +500 micron particles for the Trial sample, as well as an increase in percentage -500 micron particles were observed.

From the limited number of PSD samples taken during the trial, any further discussion regarding PSD would be speculative.

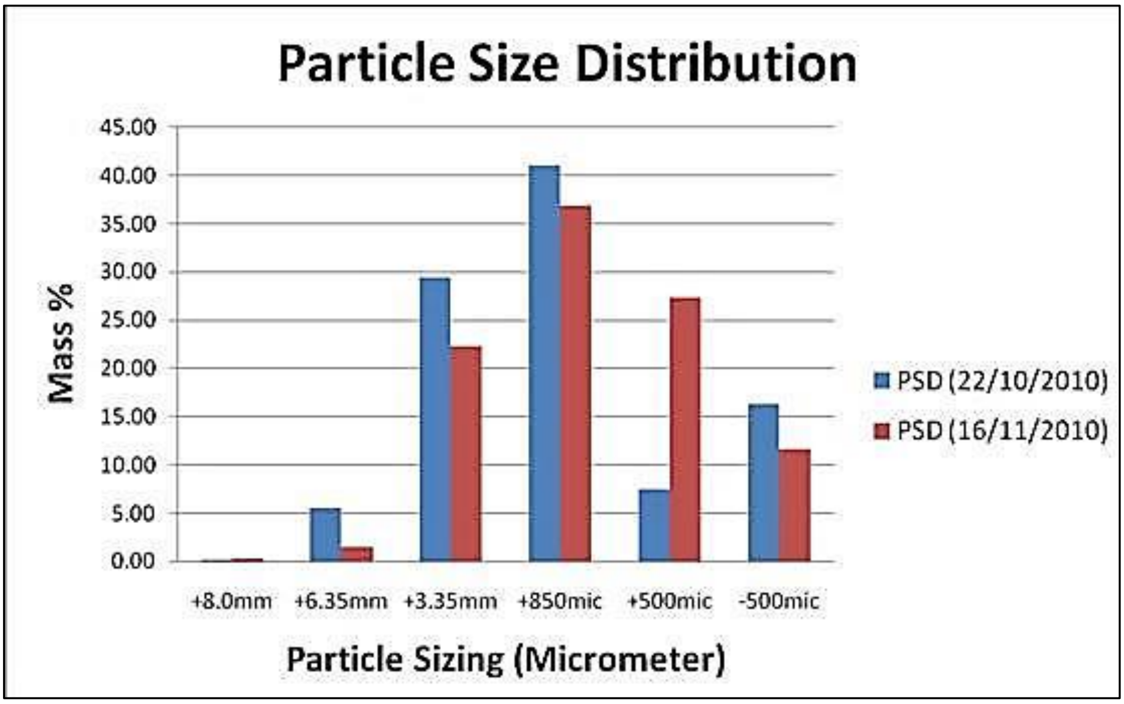


Figure 3.9-18 PSD for the Blend Mining Trial without the DMS, as well as a normal operational Base Case Sample PSD.

3.9.5 Sinter Return Fines

After sintering is complete, sintered ore is crushed to smaller than 40mm. During this crushing process, just as in crushing steps fines are generated. However, when considering sintered material, the generation of fines is an indication of how effective sintering was in agglomerating sinter feed material during the carbon-thermal reduction process, and can give some indication of the mechanical properties of the trial generated sinter compared to sinter generated using the DMS plant.

During the initial stage of the Trial without the DMS a substantial percentage (32%) of return fines were produced. However, as the trial continued return fines generation was found to be an average of 23.1%. Please refer to Figure 3.9-19 for graph depicting the PSD of the train sinter return fines.

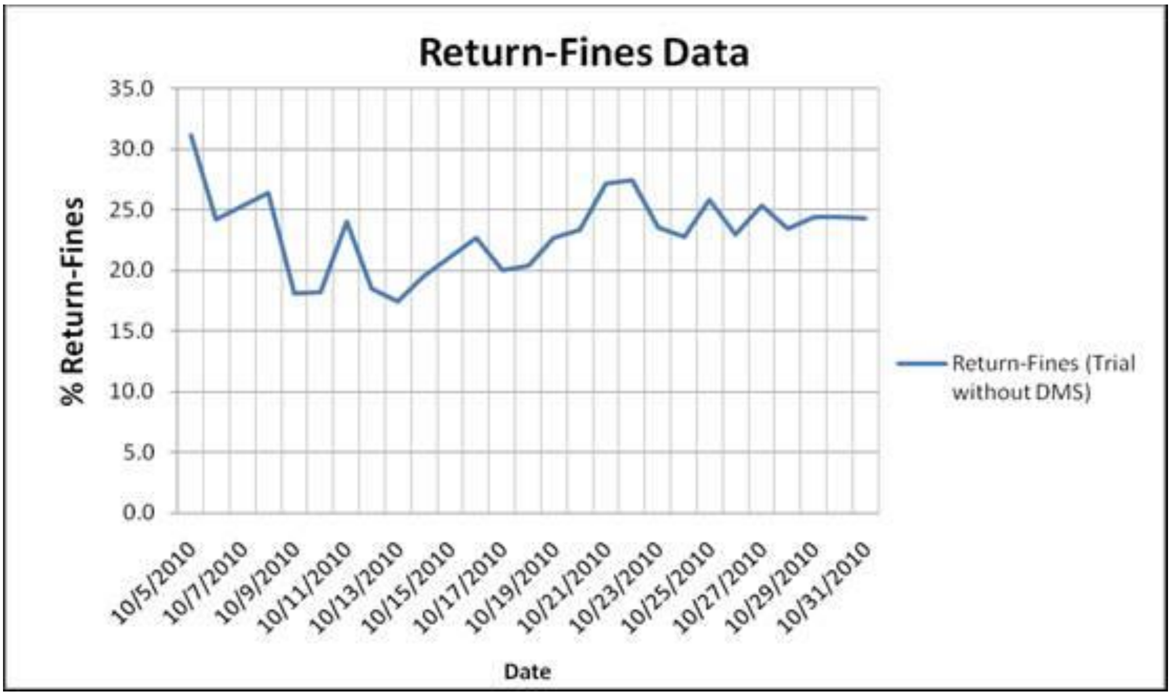


Figure 3.9-19 % Return-Fines generated during the Trial

Return fines generation under non-trial conditions was found to an average of 26%. Please refer to Figure 3.9-20 for more detail.

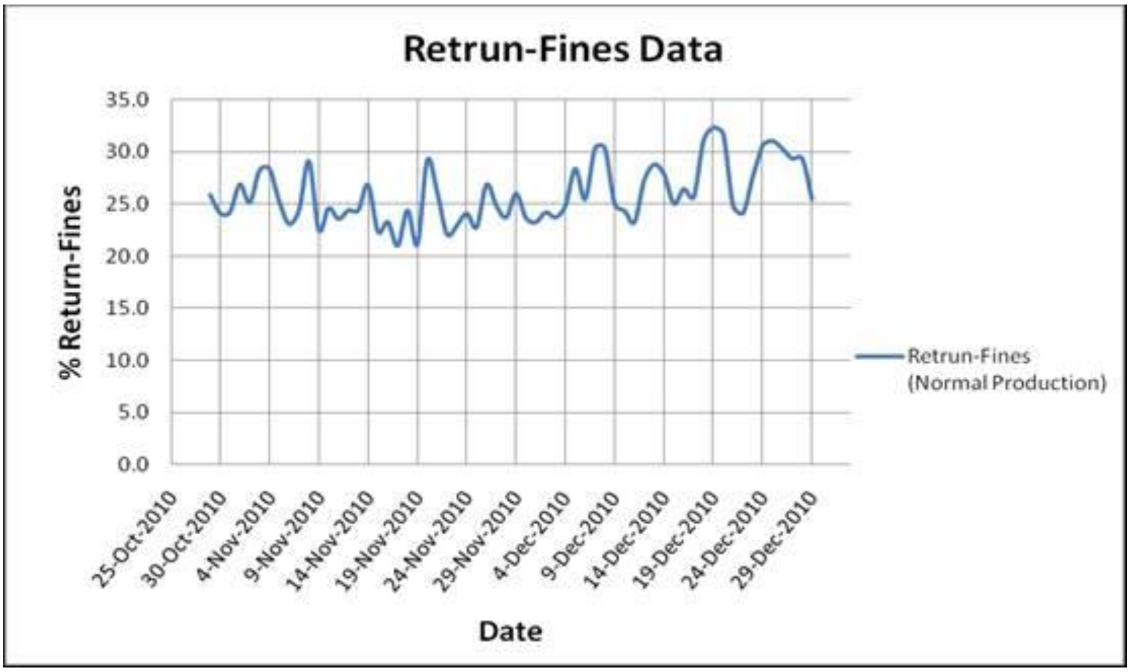


Figure 3.9-20 Percentage return-fines generated during normal non-trial conditions.

CHAPTER 4: DISCUSSION, CONCLUSIONS AND RECOMMENDATIONS

4.1 Discussion

From the onset of this dissertation it was stated that its outcomes were focused on the existence of variability within naturally occurring ore bodies in the crust of the earth, and the impact is such variability was expressed in terms of the complexity it causes in certain industrial production chains.

This complexity was further delineated by stating ore preparation methods generally used when extracting valuable ore material from ore bodies, as well as stating additional ore preparation which increases the contents of certain constituents and decreasing contents of less desirable constituent contents within prepared ore. Such further processing was referred to as beneficiation (ore up-grading).

Since the project presented in this dissertation was performed at a manganese mine which is part of a greater ferromanganese production chain, the variability of the ore body related to this specific mine was reviewed geologically. Published data from several authors were studied and presented in Section 2.2 of this dissertation, which gives verification of variability present in the respective manganese ore (Mamatwan Mine).

This variability has a very real impact of ore processing complexity and the costs incurred to produce acceptable raw materials for use in ferromanganese production chains. The complexity caused at Mamatwan Mine was and is in the form of intermediary ore beneficiations steps. Specifically the DMS plant was instituted at Mamatwan Mine to address variability. This additional facility that was not part of the original plant and was not even present when the Mamatwan Sinter Plant was commissioned to minimize the impact of ore variability on the ferromanganese production chain.

With the advent of additional methods for managing ore variability implemented at worldwide mining sites, the existence of the DMS facility at Mamatwan Mine was questioned in terms of the footprint required for the facility, as well as the manpower and electrical requirements for the operation thereof. Another important resource required by the DMS is a substantial amount of water, as hydro-DMS facilities rely heavily on the availability of water, which in the arid conditions in the Northern-Cape is considered a very valuable. Another negative aspect of the DMS is the fact that a rather large amount of ore is wasted (ore with a Mn content below 37% is discarded as stated before), which destroys value added during all preceding ore preparation steps. Generally 40% of ore prepared before the DMS is discarded. This brings about a further negative aspect of the DMS; waste generation (tailings).

The purpose of the trial project presented in this dissertation is to qualify an alternative method of managing ore variability to possibly replace the DMS. It could be stated that the aim is to validate the stated Mamatwan mine ore variability, which would also then validate the existence of the Mamatwan DMS facility as part of a ferromanganese production chain, and then also validate and verify an alternative method for managing the ore variability at Mamatwan mine such that the DMS facility could possibly be circumvented permanently in the future.

In terms of the implementation of an alternative ore management trial methodology, the development of suitable trial methodologies as well as means for selection of a single suitable trial methodology required review of several option development and subsequent Option Selection strategies and tactics. Since the trial had to be performed in a brownfields (where existing facilities and plant operations are existent) area, and since the trial would require the use of available plant personnel and equipment where possible, it was elected to implement a modified Stuart Pugh methods for option development and selection. Even though there are authors that dispute the Stuart Pugh methods, recent studies indicated that the Stuart Pugh methods are viable if implemented correctly, and in cases where inputs from large existing teams were required. Literature also revealed that ambiguity in selection criteria development could be a cause for failures experienced in many Option Selection efforts and methods as well as in coding of programmable logic controllers. The main modification to the Stuart Pugh method of option development and selection was the use of specific taxonomy to limit ambiguity, and to use yes and no instead of + and -. No provision was made for an “S” which allows for disagreement within the Option Selection team. Another change implemented in terms of the Stuart Pugh methods was that criteria were considered in three levels of importance with a further forth level of importance that allowed comparison on a very limited basis. This allowed certain criteria to have greater weights than other criteria. It was this approach that was considered to be sufficient to remove the usual “S” associated with the Stuart Pugh method. Option Selection criteria was developed with the aid of different high ranking and experienced mining and plant operational personnel, and discussed during group sessions (workshops). It was during these workshops that criteria related questions were posed to the group, which required a unanimous yes or no from the group to allow inputs into the option ranking matrix and subsequent automated ranking and selection via Boolean algebra built into the option ranking matrix. Microsoft Excel was used as an automated matrix.

After selection of a viable option for trial implementation, the planning and coordination of the entire facility was performed. Accountability and responsibility was assigned to relevant personnel and a contractor was approached to assist with relevant plant alterations to allow the trial to be performed. As stated before in this dissertation, the main requirement was that all activities for the trial shall not impact normal plant production until the commencement of the trial.

All plant alterations were performed as described in Section 3.8, and was such that final integration of these alterations occurred during a normal tertiary crusher maintenance shutdown. The final integration required 17 hours and comprised the following:

- Lengthening of the conveyor belt via double splicing and vulcanisation
- Changing the drive direction of the conveyor motor and modification of the conveyor gearbox to run in the reverse direction.
- Training (setting of conveyor guide rollers) to ensure the conveyor runs within positional limits.
- Configuration of hard-wired conveyor interlocks between S7 and S8 conveyors to allow conveyor trips to stop pre- and proceeding conveyor systems.
- Cold-commissioning.
- Start of trial – day 1

The trial was requested to generate data to allow validation and verification of an alternative method for managing the ore variability already explained. It was therefore crucial that sampling points be selected and sampling performed for future decision making, while impacting the normal plant activities to a minimum. Pre-trial chemical analyses was made available by the plant and used as a baseline for comparison against data generated via the trial (Section 3.3). The pre-trial data included limited constituents (Mn and Fe contents) but was sufficient for validation of the existence of ore variability and the existence of the DMS plant. Trial data was substantially more comprehensive since all laboratory resources were directed for a few days only at generating trial analyses data. This was further facilitated due to fact that the DMS plant were not operational and that no product of 40 to 75mm (EL, M1L or lumpy) were produced. Trial data included Mn and Fe content. CaO content as well as trace constituent contents in the final product were also performed. Comparisons between Pre-trial sample data, trial Blast-Hole material and trial ore processing sample data were made. Blast-Hole sample analyses data confirmed substantial variability of ore material in the pit. However, it was clear from graphs plotted using the analyses data (Section 3.9), that Blend-mining did substantially decrease variability by spreading high Mn content material evenly through the material processed through the plant.

4.2 Conclusions

Trial option development, review and selection were performed implementing a modified Stuart Pugh method, which did manage to render quality options for Option Selection. It accumulated usable input from knowledgeable personnel, to gain the trust and cooperation of all team members during the planning and execution of the trial project. The option indicated as the best go-forward option did not disrupt plant activities before commencement of the trial. The selected option was cost effective (only R500k for plant alterations, but this also caused a profit of R320k due to DMS operational costs not relevant – DMS costs for 18 000 tons of ore would amount to more than R1m) and the trial was deemed a success in terms of delivery of material and the time within which the results were achieved. Furthermore, the option implemented remains and can be used for future trials. From this it is clear that the full implementation of Blend mining would generate an annual operational saving as well as waste generation saving of close to R40m since more than 1000 000 tons of sinter is produced (data from Table 3.6-2 was used to derive these saving estimates). The life-of-mine (LOM) would be extended since 400 000 tons of waste is no longer generated.

Blend-mining managed to limit variability to significantly lower levels than material processed under normal operational conditions (none-blend-mining). The goals set for the Sinter Product (SP) in Section 3.4.4 of this dissertation were exceeded. However, slightly higher levels of CaO were achieved than sought in Section 3.4.4. The higher levels of CaO is related to gangue material remaining due to the circumvention of the DMS facility, which would otherwise have been removed with ore particle that contains high Mn content portions with some integrated gangue $CaCO_3$. Slightly higher level of CaO will require minor alteration of smelter raw materials feed. The financial impact of such a small change is considered part of another trial project. Therefore, Blend Mining produced a standard sinter grade product, with acceptable Mn content, low variability, without the DMS facility and without normal DMS tailings (waste).

Additional conclusions can be drawn from the particle-size-analyses on trial sinter feed material and none-trial sinter feed material. It must be remembered that under normal operational conditions where the DMS facility is used for beneficiation, additional commutation and washing takes place, which would remove very fine material as slimes, showing a statistically higher amount of larger particles. During the trials no washing was performed during or after tertiary crushing. It is suggested that washing after the tertiary crushing step would remove a large amount of fines and may even remove $CaCO_3$ to yield a lower CaO content in the final sinter product (SP).

4.3 Recommendations

After completion of the trial, with considerations given to pre-trial data and experience gained during the trial several recommendations can be made for future trial project purposes.

Firstly, it is considered paramount to accumulate plant sample analyses data spanning several months. Even though the period over which pre-trial data was accumulated did include more than 20 000 tons of processed material, higher resolution multi-blast-hole material analyses data will not only serve to further validate the existence of variability, but also verify the true variability band that exists in the Mamatwan ore body. It will also validate and verify the impact of commutation on variability.

Secondly, it is recommended to review the implementation of a washing step after the tertiary crushers. This can be accomplished by the addition of spray water washers to the quaternary crusher's vibrating screen, and possible dewatering of the screen underflow via a hydro-cyclone. This would be a relatively inexpensive and be a low impact addition to the plant to facilitate better results. By implementing such a washing stage during the trial, liberation of gangue material during quaternary crushing and removal via washing can be further studied. It is suspected that such a washing stage will remove some of the $CaCO_3$ and lower the CaO in the final sinter product (SP) to within expectations.

Thirdly, perform structured follow-up trials to further improve the resolution of data generated during this trial and instil trust of facilities down-stream in the ferromanganese production chain in terms of Blend Mining.

Expand upon the sampling points and efforts (type of sampling and analyses) to also include the following:

- Tumble Index Sampling and Analyses which will indicate the mechanical properties of the trial final product.
- Particle-Size-Distribution of Sinter Feed Material.
- Sinter Plant Return Fines.
- Sinter Feed Moisture Content.

Generating such information will greatly enhance insight during future decision making.

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APPENDIXES

5.1 Appendix A – Tabulated data supporting literature survey manganese and batteries

Tables indicating Comparative Battery Characteristics

Table 5.1-1 Comparison of Primary Batteries (after Linden *et al.*, 2002)

System	Voltage	Specific energy (gravimetric)	Power density	Flat discharge profile	Low-temperature operation	High-temperature operation	Shelf life	Cost
Zinc/carbon	5	4	4	4	5	6	8	1
Zinc/alkaline/manganese dioxide	5	3	2	3	4	4	7	2
Magnesium/manganese dioxide	3	3	2	2	4	3	4	3
Zinc/mercuric oxide	5	3	2	2	5	3	5	5
Cadmium/mercuric oxide	6	5	2	2	3	2	3	6
Zinc/silver oxide	4	3	2	2	4	3	6	6
Zinc/air	5	2	3	2	5	5	—	3
Lithium/soluble cathode	1	1	1	1	1	2	2	6
Lithium/solid cathode	1	1	2	2	2	3	2	4
Lithium/solid electrolyte	2	1	5	2	6	1	1	7

* 1 to 8—best to poorest.

Table 5.1-2 Comparison of Electrochemical, Limitations, Production Status and more is presented

System	Zinc-carbon (Leclanché)	Zinc-carbon (zinc chloride)	Mg/MnO_2	$Zn/Alk./MnO_2$	Zn/HgO	Cd/HgO
Chemistry:						
Anode	Zn	Zn	Mg	Zn	Zn	Cd
Cathode	MnO_2	MnO_2	MnO_2	MnO_2	HgO	HgO
Electrolyte	NH_4Cl and $ZnCl_2$ (aqueous solution)	$ZnCl_2$ (aqueous solution)	$MgBr_2$ or $Mg(ClO_4)$ (aqueous solution)	KOH (aqueous solution)	KOH or NaOH (aqueous solution)	KOH (aqueous solution)
Cell voltage, V§:						
Nominal	1.5	1.5	1.6	1.5	1.35	0.9
Open-circuit	1.5–1.75	1.6	1.9–2.0	1.5–1.6	1.35	0.9
Midpoint	1.25–1.1	1.25–1.1	1.8–1.6	1.25–1.15	1.3–1.2	0.85–0.75
End	0.9	0.9	1.2	0.9	0.9	0.6
Operating temperature, °C	−5 to 45	−10 to 50	−20 to 60	−20 to 55	0 to 55	−55 to 80
Energy density at 20°C§:						
Button size:						
Wh/kg				80	100	55
Wh/L				360	470	230
Cylindrical size:						
Wh/kg	65	85	100	145	105	
Wh/L	100	165	195	400	325	
Discharge profile (relative)	Sloping	Sloping	Moderate slope	Moderate slope	Flat	Flat
Power density	Low	Low to moderate	Moderate	Moderate	Moderate	Moderate
Self-discharge rate at 20°C, % loss per year‡	10	7	3	4	4	3
Advantages	Lowest cost; good for noncritical use under moderate conditions; variety of shapes and sizes; availability	Low cost; better performance than regular zinc-carbon	High capacity compared with zinc-carbon; good shelf life (undischarged)	High capacity compared with zinc-carbon; good low-temperature, high-rate performance	High volumetric energy density; flat discharge; stable voltage	Good performance at high and low temperatures; long shelf life
Limitations	Low energy density; poor low-temperature, high-rate performance	High gassing rate; performance lower than premium alkaline batteries	High gassing (H_2) on discharge; delayed voltage	Moderate cost but most cost effective at high rates	Expensive moderate gravimetric energy density, poor-low-temperature performance	Expensive, low-energy density
Status	High production, but losing market share	High production, but losing market share	Moderate production, mainly military	High production, most popular primary battery	Being phased out because of toxic mercury	In limited production being phased out because of toxic components except for some special applications
Major types available	Cylindrical single-cell bobbin and multicell batteries (see Tables 8.9 and 8.10)	Cylindrical single-cell bobbin and multicell batteries (see Table 8.9)	Cylindrical single-cell bobbin and multicell batteries (see Table 9.3)	Button cylindrical and multicell batteries (see Tables 10.9 and 10.10)	NLA*	NLA*

*No longer readily available commercially
† See Chap. 14 for other lithium primary batteries.
‡ Rate of self-discharge usually decreases with time of storage.
§ Data presented are for 20°C, under favorable discharge condition. See details in appropriate chapter.

Table 5.1-3 Table 5.1-2 continued

Zn/Ag_2O*	Zinc/air	Li/SO_2†	$Li/SOCl_2$†	Li/MnO_2†	Li/FeS_2†	Solid state
Ag_2O or AgO KOH or NaOH (aqueous solution)	Zn O_2 (air) KOH (aqueous solution	Li SO_2 Organic solvent, salt solution	Li SOC/2 $SOCl_2$ w/$AlCl_4$	Li MnO_2 Organic solvent, salt solution	Li FeS_2 Organic solvent, salt solution	Li I_2(P2VP) Solid
1.5 1.6 1.6–1.5 1.0	1.5 1.45 1.3–1.1 0.9	3.0 3.1 2.9–2.75 2.0	3.6 3.65 3.6–3.3 3.0	3.0 3.3 3.0–2.7 2.0	1.5 1.8 1.6–1.4 1.0	2.8 2.8 2.8–2.6 2.0
0 to 55	0 to 50	−55 to 70	−60 to 85	−20 to 55	−20 to 60	0 to 200
135 530	370 1300			230 545		
	Prismatic 300 Prismatic 800	260 415	380 715	230 535	260 500	220–280 820–1030
Flat	Flat	Very flat	Flat	Flat	Initial drop than flat medium to high	Moderately flat (at low discharge rates)
Moderate	Low	High	Medium (but dependent on specific design)	Moderate	Medium to high	Very low
6	3 (is sealed)	2	1–2	1–2	1–2	<1
High energy density; good high-rate performance	High volumetric energy density; long shelf life (sealed)	High energy density; best low-temperature, high-rate performance; long shelf life	High energy densitty, long shelf life because of protective film	High energy density; good low-temperature, high-rate performance; cost-effective replacement for small conventional type cells	Replacement for Zu/alkaline/MnO_2 batteries for high rate performance	Excellent shelf life (10–20 y); wide operating temperature range (to 200°C)
Expensive, but cost-effective on button battery applications	Not independent of environment—flooding, drying out; limited power output	Pressurized system Potential safety problems, toxic components Shipment regulated	Voltage delay after storage	Available in small sizes; larger sizes being considered Shipment regulated	Higher cost than alkaline batteries	For very low discharge rates; poor low temperature performance
In production	Moderate production, key use in hearing aids	Moderate production, mainly military	Produced in wide range of sizes and designs, mainly for special applications	Increasing consumer production	Produced in "AA" size	In production for special applications
Button batteries (see Table 12.3)	(See Tables 13.2 and 13.3, also Chap. 38)	Cylindrical batteries (see Tables 14.9 and 14.10)	See section 14.6 and Tables 14.11 to 14.13	Button and small cylindrical batteries (see Tables 14.19 to 14.21)	Produced in "AA" size (see Table 14.18)	(See Table 15.6)

5.2 Appendix B – Process Flow Diagrams and Aerial Site Pictures

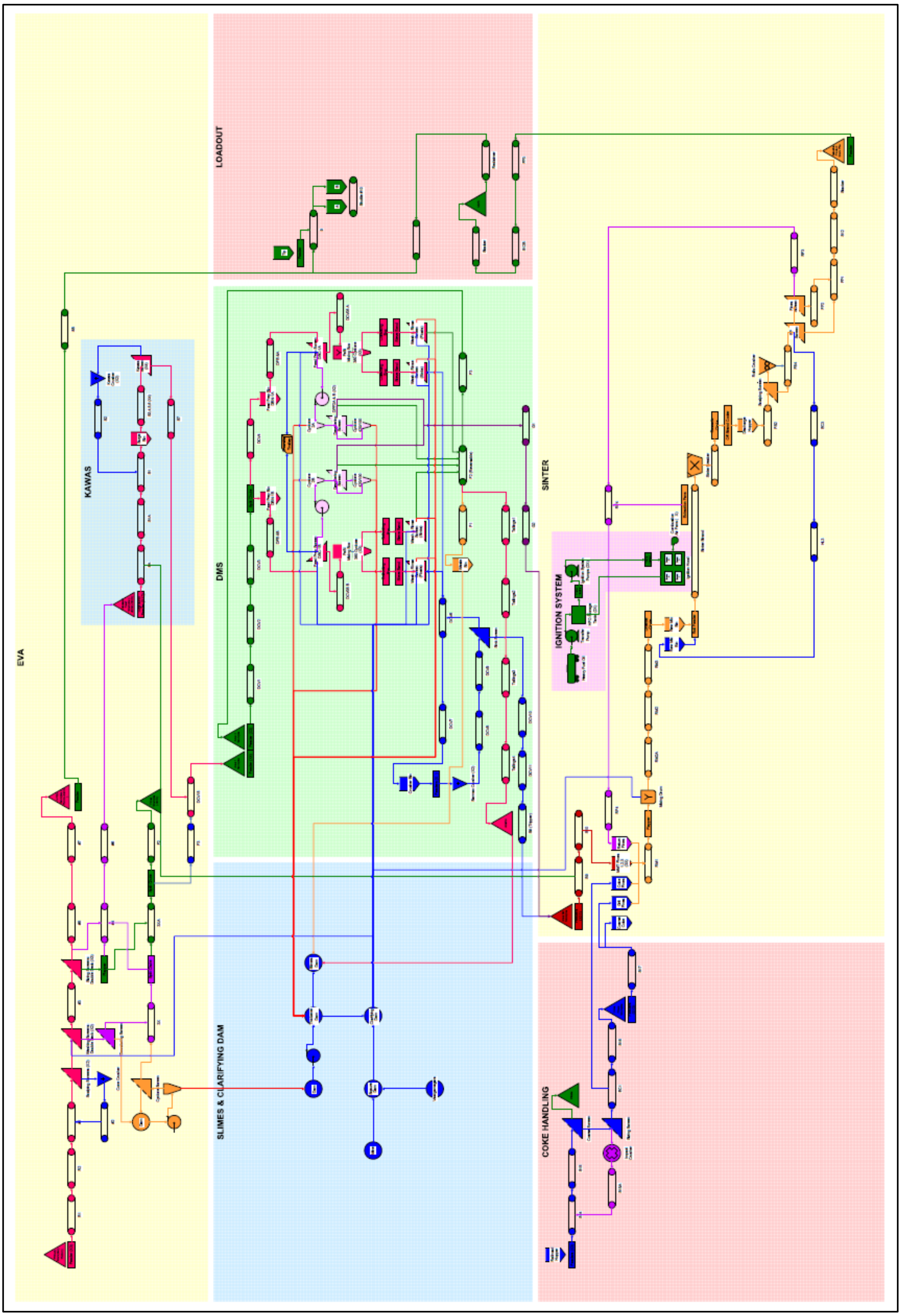

Figure 5.2-1 Comprehensive diagrammatic presentation of the Mamatwna plant process. All equipment and stockpiles are presented in the one figure but does not contain instrumentation control aspects. A higher quality diagram in Microsoft Vision format is available. Note that EVA (OPP), Kawas (Tertiary Crushers), DMS (Dense Medium Separation), Slimes water clarification, Coke handling, and the entire sinter as well as load-out circuits are included.

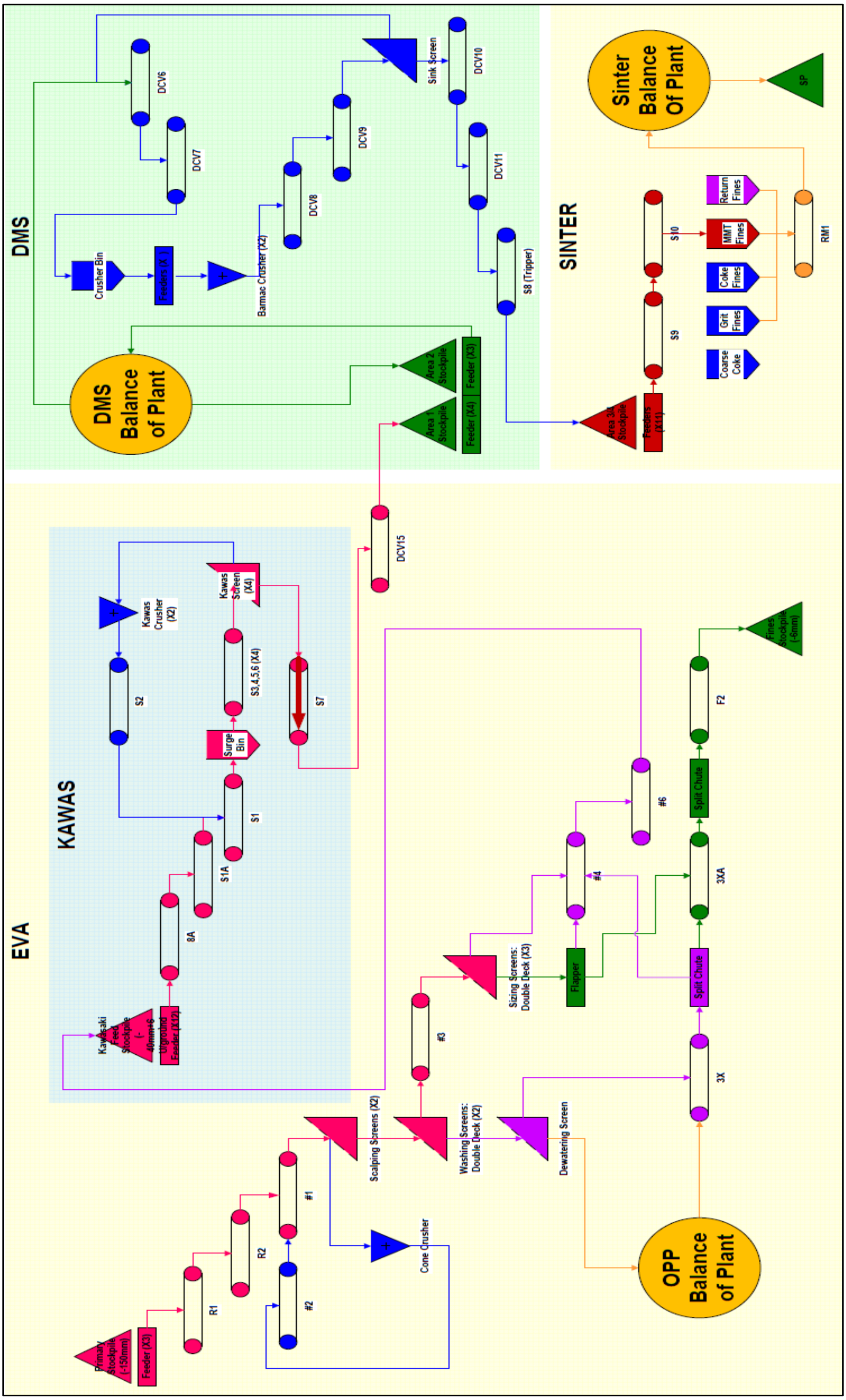

Figure 5.2-2 Diagrammatic presentation of the pre-trial process equipment (conveyors, crushers, feeders, screens, cyclones and stockpiles at Mamatwan Plant). Note that this diagram differs from the diagram Figure 5.2-1, in some portions of the plant were removed and are presented with the ovals indicated as Balance of Plant, in an effort to decrease cluttering and ease discussion. Note the direction of conveyor S7 and its delivery to DCV15.

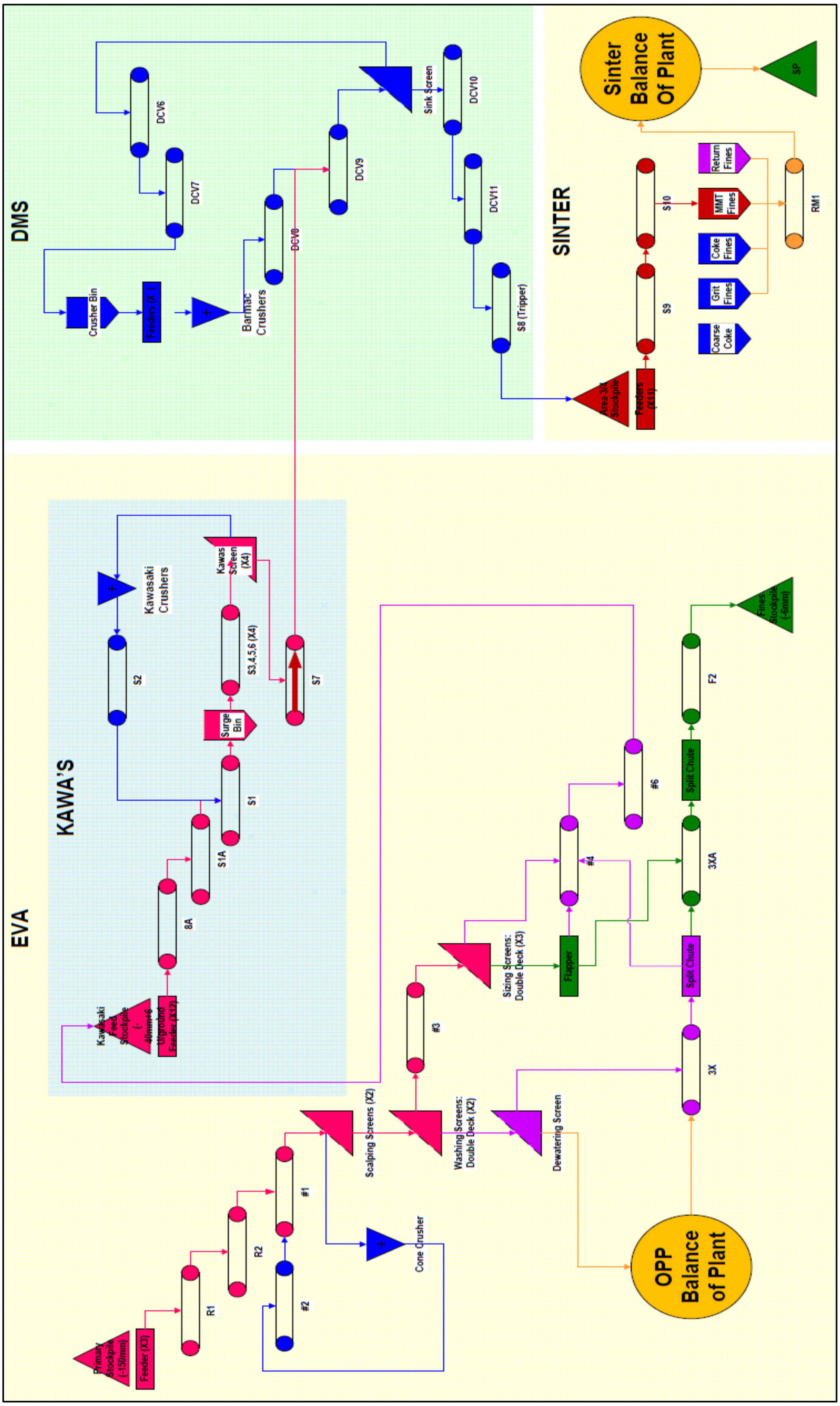

Figure 5.2-3 Trial Diagrammatic presentation of the trial process equipment (conveyors, crushers, feeders, screens, cyclones and stockpiles at Mamatwan Plant). Note that this diagram differs from the diagram Figure 5.2-1, in some portions of the plant were removed and are presented with the ovals indicated as Balance of Plant, in an effort to decrease cluttering and ease discussion. Note the direction of conveyor S7 and its delivery to DCV9.

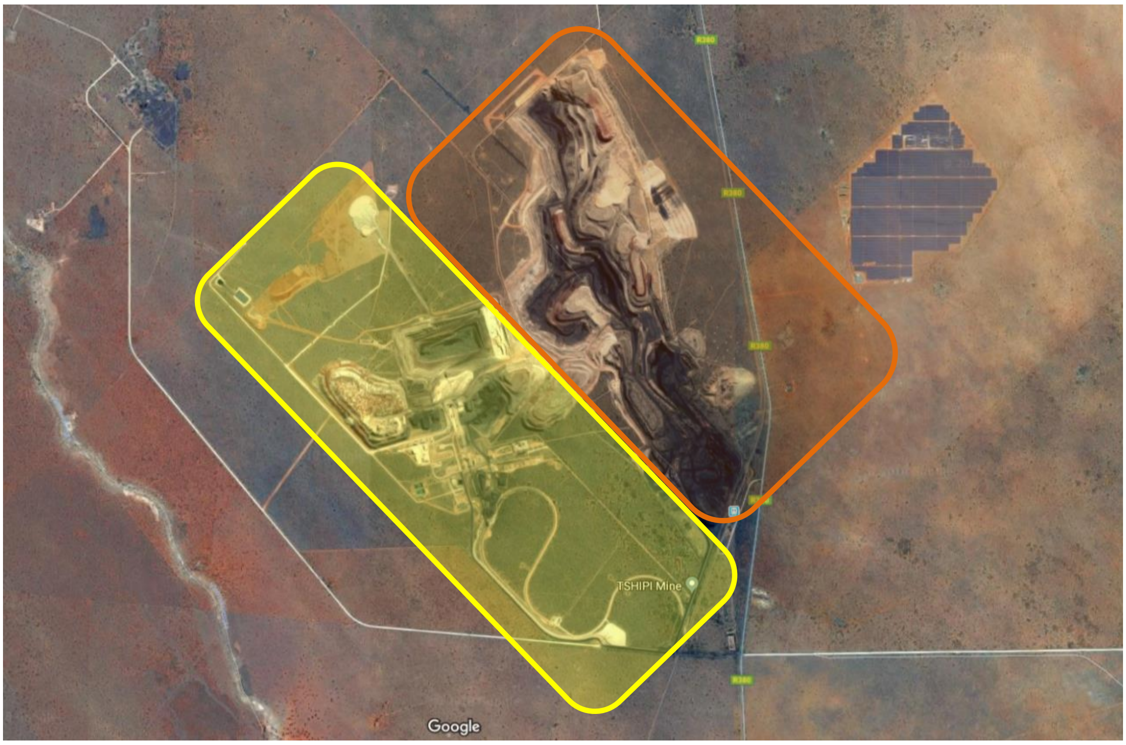

Figure 5.2-4 Google Maps Satellite Picture of Mamatwan Mine and Plant (Transparent orange rectangle) as well as Tshipi Borwa Mine and Plant (Transparent yellow rectangle). The Tshipi Borwa Mine and Plant is operated by Tshipi é Ntle Manganese Mining (Pty) Ltd

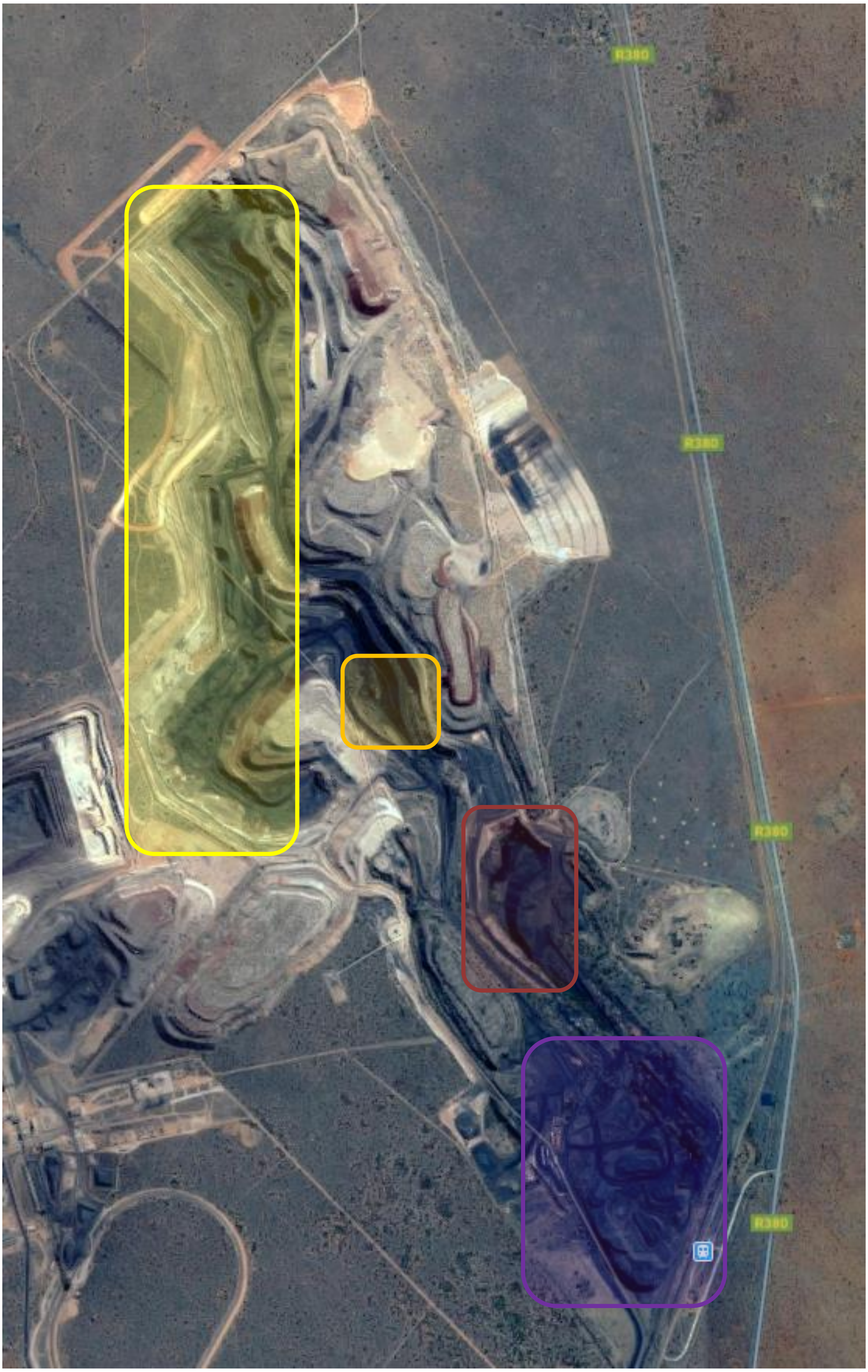

Figure 5.2-5 Higher magnification Google Maps satellite picture of Mamatwan Mine and Plant. Mamatwan Plant is indicted with the purple block, Adams pit (DMS tailings dump) is indicated with the red block, the in-pit crusher with the orange block and the actively mined area with the yellow block

Figure 5.2-6 Higher magnification Google Maps satellite picture of Mamatwan Plant (red, yellow and brown ovals – OPP, DMS Sinter resepctively), product and waste stockpile (green oval) area.

5.3 Appendix C – Pre-trial Analyses Data

Table 5.3-1 Ore Manganese compositions for sample sets taken on 1 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines

OPP DAILY REPORT

bhpbilliton
resourcing the future

DATE: 1-Oct-10

Sample Set #	PLANT / TIME	EL (+25)M1L Mn %	EL (+25)M1L Fe %	EL (+25)M1L Mn /Fe	ES +6-40mm Mn %	ES +6-40mm Fe %	ES +6-40mm Mn/Fe	EF -6mm Mn %	EF -6mm Fe %	EF -6mm Mn/Fe	Average Mn/Fe	ANALYST
1	02:00-04:00	36.5	3.9	9.4	38.6	4.8	8.0	38.5	4.7	8.2	8.5	Matthews
2	04:00-06:00	39.7	4.3	9.2	37.5	4.3	8.7	38.1	4.8	7.9	8.6	Bontle
3	14:00-16:00	34.3	3.7	9.3	38.0	3.9	9.7	36.4	4.2	8.7	9.2	RT
4	16:00-18:00	36.9	3.7	10.0	36.5	3.8	9.6	38.1	4.6	8.3	9.3	RT
5	18:00-20:00	35.6	3.4	10.5	37.8	4.4	8.6	37.0	4.9	7.6	8.9	RT
6	20:00-22:00	35.4	3.7	9.6	38.5	4.2	9.2	37.2	5.1	7.3	8.7	RT
7	22:00-00:00	41.6	4.5	9.2	41.0	5.2	7.9	39.8	5.3	7.5	8.2	Matthews
	AVERAGE	37.1	3.9	9.6	38.3	4.4	8.8	37.9	4.8	7.9	8.7	

	01-Oct-10		
Min Mn %	34.3	36.5	36.4
Max Mn %	41.6	41.0	39.8
Daily Mn Variability %	7.3	4.5	3.4

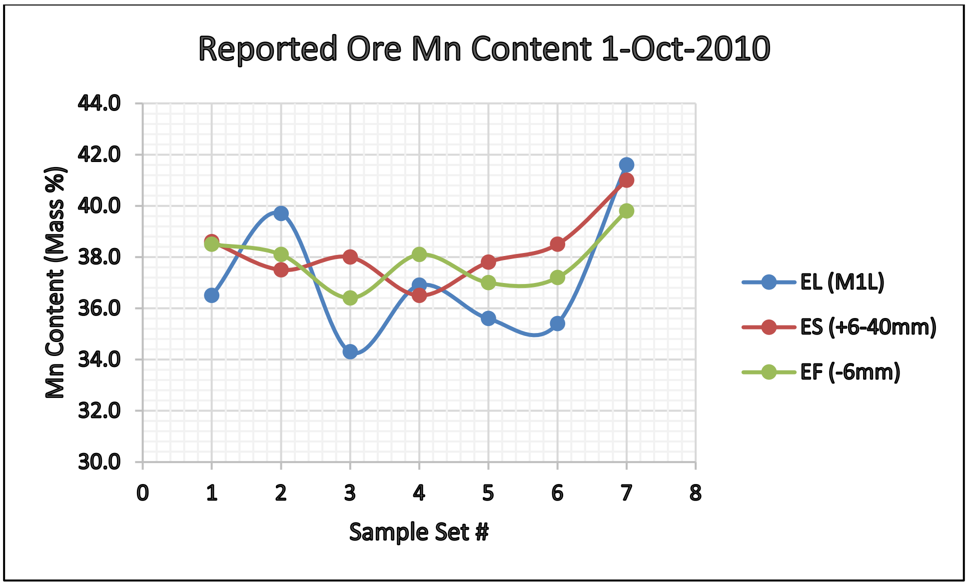

Figure 5.3-1 Plotted graph of the ore manganese content for sample sets taken on 1 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 7.3% was evident.

Table 5.3-2 Ore Manganese compositions for sample sets taken on 2 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

HMM Hotazel Manganese Mines

OPP DAILY REPORT

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DATE: 2-Oct-10

Sample Set #	1. PLANT TIME	EL (+25)M1L Mn	EL (+25)M1L Fe	EL (+25)M1L Mn /Fe	ES +6-40mm Mn	ES +6-40mm Fe	ES +6-40mm Mn/Fe	EF -6mm Mn	EF -6mm Fe	EF -6mm Mn/Fe	Average Mn/Fe	ANALYST
1	00:00-02:00	42.1	4.6	9.2	39.6	4.6	8.6	40.7	5.4	7.5	8.4	MATTHEWS
2	02:00-04:00	39.7	4.3	9.2	39.9	4.3	9.3	40.9	5.2	7.9	8.8	BONTLE
3	04:00-06:00	40.4	3.9	10.4	40.3	4.6	8.8	40.7	5.3	7.7	8.9	BONTLE
4	12:00-14:00	38.2	4.3	8.9	41.6	4.4	9.5	39.9	4.9	8.1	8.8	MOL
5	14:00-16:00	35.8	3.4	10.5	39.0	3.8	10.3	39.0	4.8	8.1	9.6	HOLLY
6	16:00-18:00	48.1	4.6	10.5	42.8	5.2	8.2	42.8	6.3	6.8	8.5	RT
7	18:00-20:00	36.1	4.3	8.4	39.2	4.1	9.6	39.7	5.3	7.5	8.5	HOLLY
8	20:00-22:00	40.4	4.8	8.4	40.5	4.3	9.4	42.3	5.4	7.8	8.6	RT
9	22:00-00:00	38.8	4.0	9.7	40.7	4.7	8.7	40.6	5.1	8.0	8.8	BONTLE
	AVERAGE	40.0	4.2	9.4	40.4	4.4	9.1	40.7	5.3	7.7	8.7	

	02-Oct-10		
Min Mn %	35.8	39.0	39.0
Max Mn %	48.1	42.8	42.8
Daily Mn Variability %	12.3	3.8	3.8

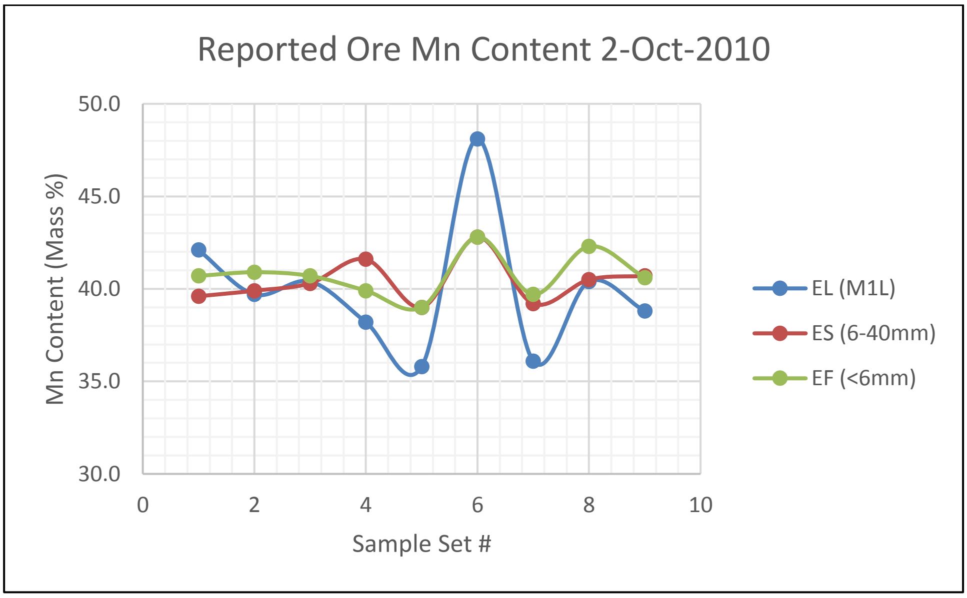

Figure 5.3-2 Plotted graph of the ore manganese content for sample sets taken on 2 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 12.3% was evident.

Table 5.3-3 Ore Manganese compositions for sample sets taken on 3 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines

OPP DAILY REPORT

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DATE: 3-Oct-10

Sample Set #	1. PLANT	EL (+25)M1L			ES +6-40mm			EF -6mm			Average	ANALYST
	TIME	Mn	Fe	Mn /Fe	Mn	Fe	Mn/Fe	Mn	Fe	Mn/Fe	Mn/Fe	
1	00:00-02:00	37.8	4.4	8.6	40.4	4.5	9.0	40.3	4.9	8.2	8.6	MATTHEWS
2	04:00-06:00	40.8	4.1	10.0	40.3	4.8	8.4	39.4	4.8	8.2	8.9	MATTHEWS
3	06:00-08:00	38.0	4.3	8.8	37.7	4.6	8.2	39.2	4.9	8.0	8.3	JOHN
4	10:00-12:00	44.5	4.2	10.6	40.6	6.1	6.7	38.7	6.2	6.2	7.8	JOHN
5	12:00-14:00	42.3	6.1	6.9	39.2	4.8	8.2	39.4	6.0	6.6	7.2	JOHN
6	14:00-16:00	42.2	4.5	9.4	38.7	6.6	5.9	39.1	5.6	7.0	7.4	HOLLY
7	16:00-18:00	35.6	6.6	5.4	39.2	4.8	8.2	40.1	5.6	7.2	6.9	HOLLY
8	18:00-20:00	39.2	4.3	9.1	36.5	4.7	7.8	37.9	4.6	8.2	8.4	HOLLY
9	20:00-22:00	40.9	4.4	9.3	36.6	4.4	8.3	38.4	5.1	7.5	8.4	HOLLY
10	22:00-00:00	38.7	4.3	9.0	38.3	4.4	8.7	39.8	5.0	8.0	8.6	BONTLE
	AVERAGE	40.0	4.7	8.5	38.8	5.0	7.8	39.2	5.3	7.4	7.9	

	03-Oct-10		
Min Mn %	35.6	36.5	37.9
Max Mn %	44.5	40.6	40.3
Daily Mn Variability %	8.9	4.1	2.4

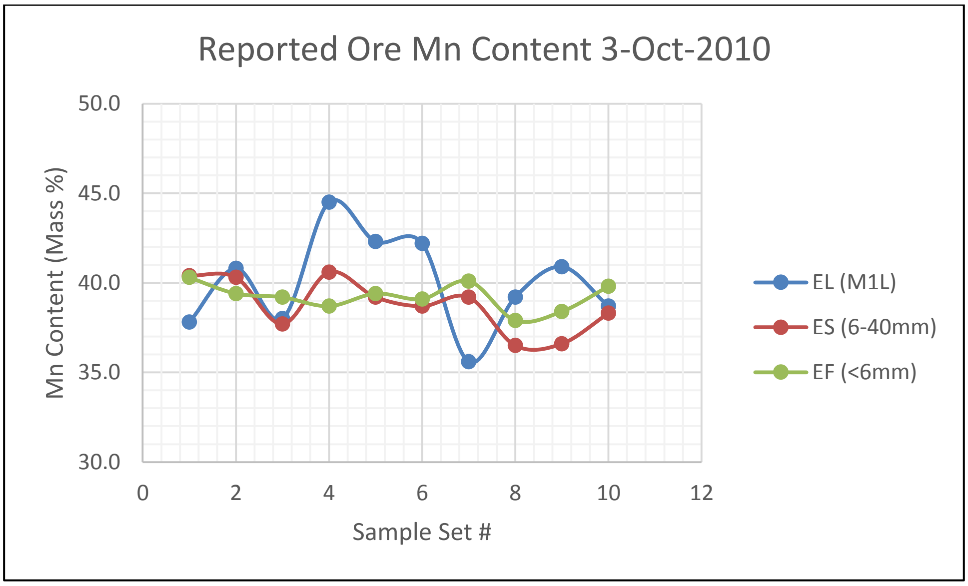

Figure 5.3-3 Plotted graph of the ore manganese content for sample sets taken on 3 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 8.9% was evident.

Table 5.3-4 Ore Manganese compositions for sample sets taken on 4 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines — OPP DAILY REPORT — bhpbilliton resourcing the future

DATE: 4-Oct-10

Sample Set #	1. PLANT / TIME	EL (+25)M1L Mn	EL (+25)M1L Fe	EL (+25)M1L Mn /Fe	ES +6-40mm Mn	ES +6-40mm Fe	ES +6-40mm Mn/Fe	EF -6mm Mn	EF -6mm Fe	EF -6mm Mn/Fe	Average Mn/Fe	ANALYST
1	00:00-02:00	43.9	5.1	8.6	40.8	4.4	9.3	40.8	5.4	7.6	8.5	JOHN
2	02:00-04:00	41.9	4.7	8.9	40.8	5.2	7.8	41.9	5.6	7.5	8.1	MC MOL
3	04:00-06:00	43.8	5.5	8.0	42.4	4.5	9.4	42.7	5.6	7.6	8.3	MC MOL
4	06:00-08:00	34.8	4.0	8.7	40.5	5.0	8.1	40.4	5.6	7.2	8.0	Martie
5	08:00-10:00	39.0	4.7	8.3	40.2	4.1	9.8	40.3	5.3	7.6	8.6	Martie
6	10:00-12:00	37.8	4.4	8.6	42.3	5.4	7.8	40.2	5.3	7.6	8.0	Martie
7	14:00-16:00	40.7	6.7	6.1	41.2	4.9	8.4	41.1	5.7	7.2	7.2	TJ
8	16:00-18:00	36.7	3.9	9.4	39.8	4.4	9.0	38.2	4.6	8.3	8.9	TJ
9	18:00-20:00	39.3	5.9	6.7	41.6	5.2	8.0	38.4	4.8	8.0	7.6	TJ
10	20:00-22:00	40.3	4.1	9.8	37.1	4.2	8.8	38.4	5.1	7.5	8.7	TJ
	AVERAGE	39.8	4.9	8.1	40.7	4.7	8.6	40.2	5.3	7.6	8.1	

	04-Oct-10		
Min Mn %	34.8	37.1	38.2
Max Mn %	43.9	42.4	42.7
Daily Mn Variability %	9.1	5.3	4.5

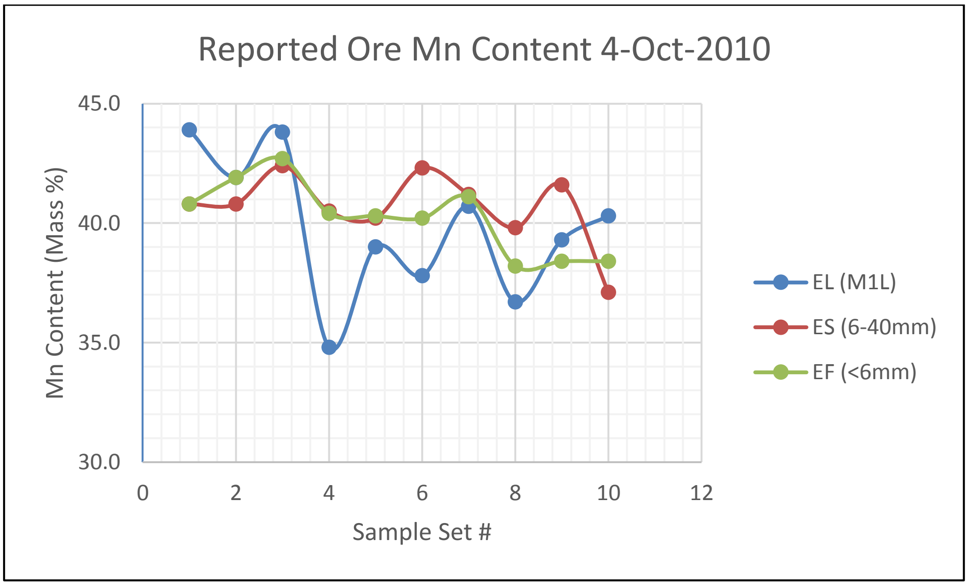

Figure 5.3-4 Plotted graph of the ore manganese content for sample sets taken on 4 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 9.1% was evident.

Table 5.3-5 Ore Manganese compositions for sample sets taken on 5 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines

OPP DAILY REPORT

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DATE: 5-Oct-10

Sample Set #	1. PLANT TIME	EL (+25)M1L Mn	EL (+25)M1L Fe	EL (+25)M1L Mn /Fe	ES +6-40mm Mn	ES +6-40mm Fe	ES +6-40mm Mn/Fe	EF -6mm Mn	EF -6mm Fe	EF -6mm Mn/Fe	Average Mn/Fe	ANALYST
1	00:00-02:00	43.2	7.5	5.8	44.1	5.8	7.6	42.7	5.8	7.4	6.9	JOHN
2	02:00-04:00	44.8	5.5	8.1	41.9	5.9	7.1	42.3	6.0	7.1	7.4	JOHN
3	04:00-06:00	46.8	5.9	7.9	43.8	5.4	8.1	42.7	6.0	7.1	7.7	JOHN
4	06:00-08:00	45.6	5.9	7.7	48.9	6.3	7.8	42.0	7.0	6.0	7.2	FELICIA
5	08:00-10:00	43.1	5.7	7.6	40.4	5.7	7.1	39.1	5.1	7.7	7.4	FELICIA
6	12:00-14:00	39.4	3.5	11.3	39.9	4.0	10.0	37.8	4.4	8.6	9.9	MARTIE
7	14:00-16:00	38.0	3.4	11.2	38.8	3.8	10.2	37.4	4.2	8.9	10.1	MOL
8	16:00-18:00	38.8	3.4	11.4	38.8	3.8	10.2	37.3	4.3	8.7	10.1	DINEO
9	18:00-20:00	36.5	3.8	9.6	40.0	3.6	11.1	37.3	4.1	9.1	9.9	MOL
10	20:00-22:00	34.6	3.8	9.1	43.8	3.8	11.5	37.0	4.1	9.0	9.9	DINEO
11	22:00-00:00	36.9	4.0	9.2	37.2	3.9	9.5	37.8	4	9.5	9.4	JOHN
	AVERAGE	40.7	4.8	8.5	41.6	4.7	8.8	39.4	5.0	7.9	8.4	

	05-Oct-10		
Min Mn %	34.6	37.2	37.0
Max Mn %	46.8	48.9	42.7
Daily Mn Variability %	12.2	11.7	5.7

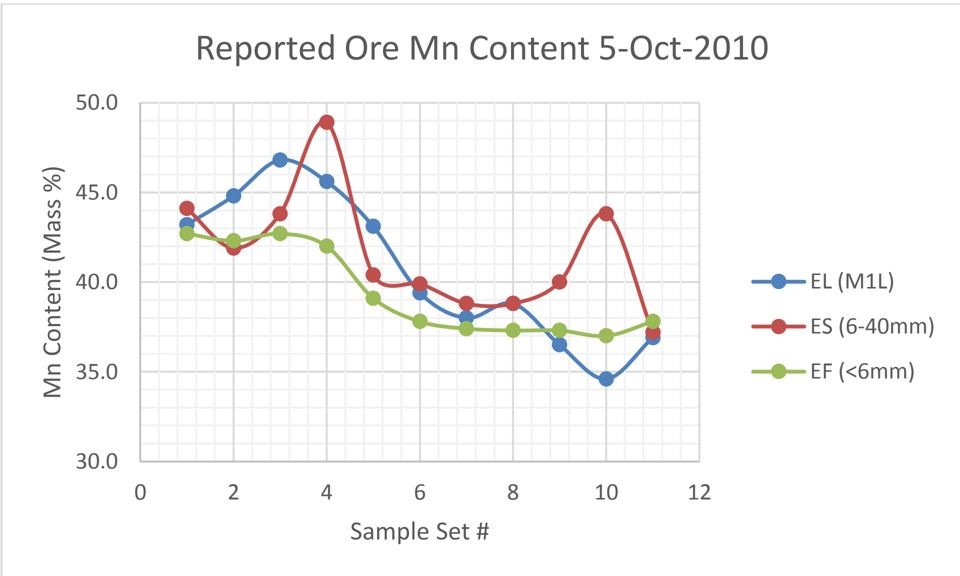

Figure 5.3-5 Plotted graph of the ore manganese content for sample sets taken on 5 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 12.2% was evident.

Table 5.3-6 Ore Manganese compositions for sample sets taken on 6 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

OPP DAILY REPORT

DATE: 6-Oct-10

Sample Set #	1. PLANT TIME	EL (+25)M1L Mn	EL (+25)M1L Fe	EL (+25)M1L Mn /Fe	ES +6-40mm Mn	ES +6-40mm Fe	ES +6-40mm Mn/Fe	EF -6mm Mn	EF -6mm Fe	EF -6mm Mn/Fe	Average Mn/Fe	ANALYST
1	00:00-02:00	41.6	3.9	10.7	39.0	3.9	10.0	36.8	4.1	9.0	9.9	JOHN
2	02:00-04:00	38.0	5.0	7.6	38.0	4.7	8.1	39.9	4.1	9.7	8.5	JOHN
3	04:00-06:00	36.9	3.9	9.5	39.2	4.2	9.3	38.9	4.7	8.3	9.0	JOHN
4	06:00-08:00	37.9	4.0	9.5	39.4	4.5	8.8	39.8	4.6	8.7	9.0	RT
5	08:00-10:00	45.2	5.9	7.7	40.7	4.5	9.0	39.8	4.9	8.1	8.3	Holly
6	14:00-16:00	38.5	4.6	8.4	40.4	5.3	7.6	39.9	5.3	7.5	7.8	DINEO
7	16:00-18:00	39.7	3.9	10.2	40.0	4.5	8.9	39.7	5.4	7.4	8.8	DINEO
8	18:00-20:00	37.2	3.4	10.9	37.3	3.4	11.0	36.7	3.7	9.9	10.6	DINEO
9	20:00-22:00	36.9	3.5	10.5	37.4	3.6	10.4	36.5	3.6	10.1	10.4	TJ
	AVERAGE	39.1	4.2	9.2	39.0	4.3	9.2	38.7	4.5	8.7	9.1	

	06-Oct-10		
Min Mn %	36.9	37.3	36.5
Max Mn %	45.2	40.7	39.9
Daily Mn Variability %	8.3	3.4	3.4

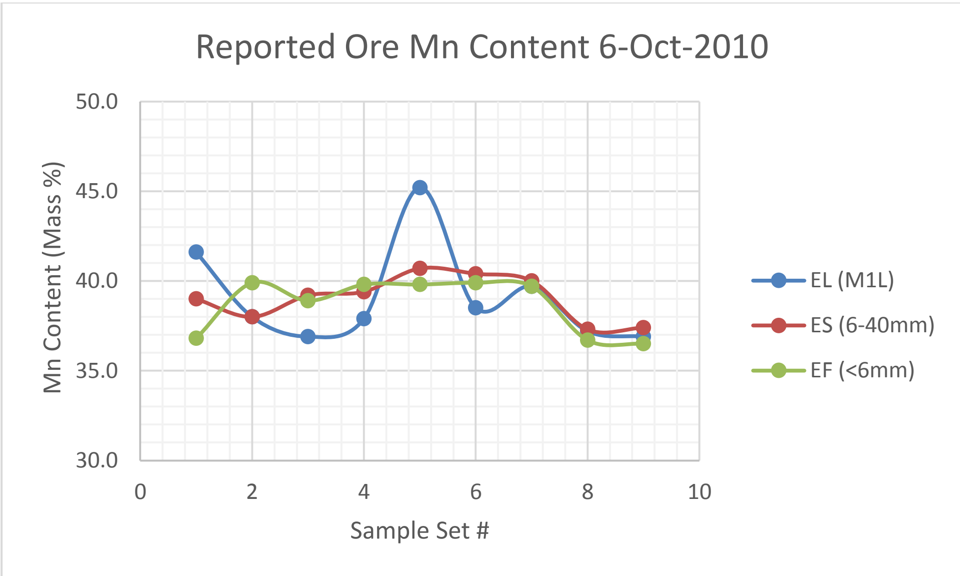

Figure 5.3-6 Plotted graph of the ore manganese content for sample sets taken on 6 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 8.3% was evident.

Table 5.3-7 Ore Manganese compositions for sample sets taken on 7 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

HMM Hotazel Manganese Mines

OPP DAILY REPORT

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DATE: *7-Oct-10*

Sample Set #	1. PLANT TIME	EL (+25)M1L Mn	EL (+25)M1L Fe	EL (+25)M1L Mn /Fe	ES +6-40mm Mn	ES +6-40mm Fe	ES +6-40mm Mn/Fe	EF -6mm Mn	EF -6mm Fe	EF -6mm Mn/Fe	Average Mn/Fe	ANALYST
1	02:00-04:00	38.1	3.6	10.6	37.9	3.8	10.0	37.2	4.4	8.5	9.7	MOL
2	04:00-06:00	38.9	4.7	8.3	37.5	4.1	9.1	37.5	4.1	9.1	8.9	HOLLY/ MOL
3	06:00-08:00	35.6	3.5	10.2	38.8	4.2	9.2	38.4	3.9	9.8	9.8	HOLLY
4	08:00-10:00	38.4	3.9	9.8	37.2	4.3	8.7	37.3	4.4	8.5	9.0	HOLLY
5	16:00-18:00	37.6	4.3	8.7	36.3	4.0	9.1	37.0	3.9	9.5	9.1	HOLLY
6	18:00-20:00	37.4	3.7	10.1	37.4	3.9	9.6	36.4	4.0	9.1	9.6	HOLLY
7	20:00-22:00	36.8	4.6	8.0	37.5	3.8	9.9	36.4	3.9	9.3	9.1	HOLLY
8	22:00-00:00	28.9	3.7	7.8	37.1	4.9	7.6	36.7	4.6	8.0	7.8	MOL
	AVERAGE	36.5	4.0	9.1	37.5	4.1	9.1	37.1	4.2	8.9	9.0	

	07-Oct-10		
Min Mn %	28.9	36.3	36.4
Max Mn %	38.9	38.8	38.4
Daily Mn Variability %	10.0	2.5	2.0

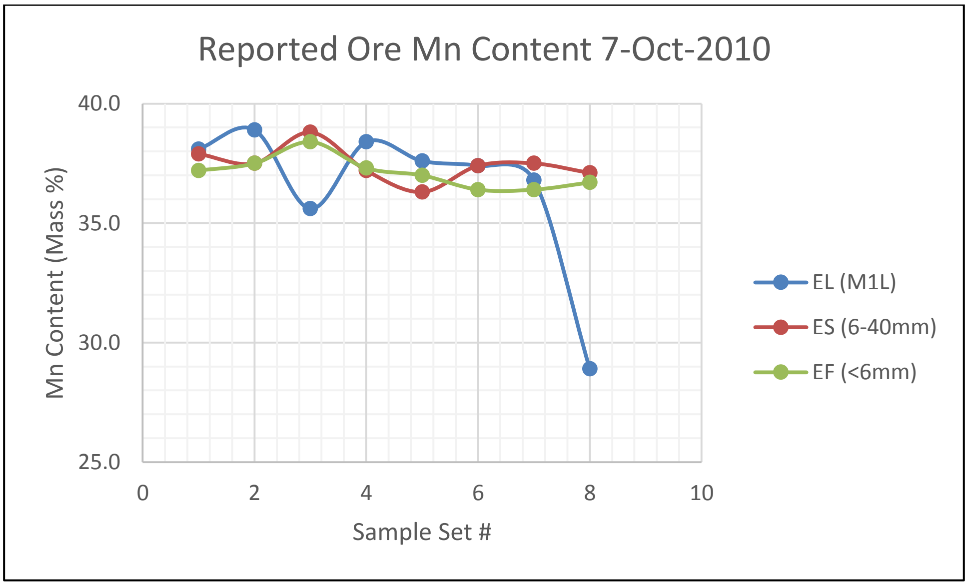

Figure 5.3-7 Plotted graph of the ore manganese content for sample sets taken on 7 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 10.0% was evident.

Table 5.3-8 Ore Manganese compositions for sample sets taken on 8 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines

OPP DAILY REPORT

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DATE: 8-Oct-10

Sample Set #	1. PLANT	EL (+25)M1L			ES +6-40mm			EF -6mm			Average	ANALYST
	TIME	Mn	Fe	Mn /Fe	Mn	Fe	Mn/Fe	Mn	Fe	Mn/Fe	Mn/Fe	
1	00:00-02:00	36.3	4.2	8.6	37.7	4.3	8.8	38.4	4.6	8.3	8.6	MOL
2	02:00-04:00	39.3	4.2	9.4	37.7	3.9	9.7	36.7	4.6	8.0	9.0	MOL
3	04:00-06:00	38.4	4.2	9.1	37.7	4.3	8.8	36.7	4.2	8.7	8.9	MOL
4	06:00-08:00	36.6	3.8	9.6	38.6	4.2	9.2	36.8	4.7	7.8	8.9	TJ
5	18:00-20:00	41.8	4.5	9.3	40.4	3.9	10.4	37.8	4.8	7.9	9.2	M M
6	20:00-22:00	40.6	4.1	9.9	37.4	5.0	7.5	38.3	4.7	8.1	8.5	M M
7	22:00-00:00	28.8	8.9	3.2	38.2	5.1	7.5	39.7	5.5	7.2	6.0	MOL
	AVERAGE	37.4	4.8	7.7	38.2	4.4	8.7	37.8	4.7	8.0	8.1	

	08-Oct-10		
Min Mn %	28.8	37.4	36.7
Max Mn %	41.8	40.4	39.7
Daily Mn Variability %	13.0	3.0	3.0

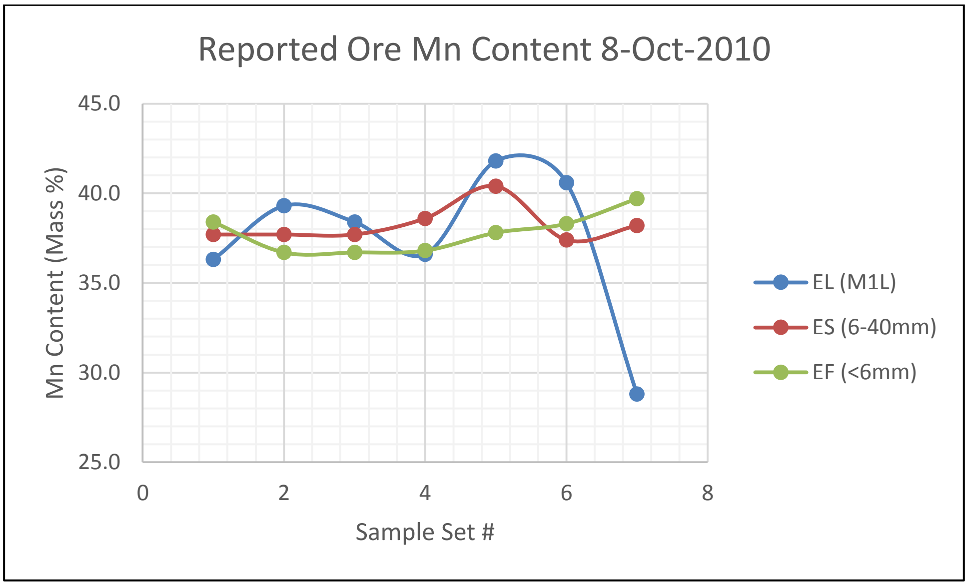

Figure 5.3-8 Plotted graph of the ore manganese content for sample sets taken on 8 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 13.0% was evident.

Table 5.3-9 Ore Manganese compositions for sample sets taken on 9 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines

OPP DAILY REPORT

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DATE: 9-Oct-10

Sample Set #	PLANT	EL (+25)M1L			ES +6-40mm			EF -6mm			Average	ANALYST
	TIME	Mn	Fe	Mn /Fe	Mn	Fe	Mn/Fe	Mn	Fe	Mn/Fe	Mn/Fe	
1	02:00-04:00	36.9	6.2	6.0	35.4	5.3	6.7	36.6	5.0	7.3	6.7	HOLLY
2	04:00-06:00	36.1	4.7	7.7	37.0	4.4	8.4	36.1	4.8	7.5	7.9	RT
3	06:00-08:00	37.3	4.5	8.3	37.9	4.4	8.6	37.0	4.5	8.2	8.4	Martha
4	08:00-10:00	36.9	4.4	8.4	37.8	4.6	8.2	37.2	4.6	8.1	8.2	Martha
5	10:00-12:00	38.1	4.0	9.5	37.6	4.2	9.0	36.9	4.3	8.6	9.0	Martha
6	12:00-14:00	37.0	3.8	9.7	37.6	4.2	9.0	37.1	4.4	8.4	9.0	Martha
7	14:00-16:00	39.3	4.4	8.9	37.1	4.0	9.3	36.9	4.4	8.4	8.9	Bontle
8	16:00-18:00	37.4	7.2	5.2	35.8	4.5	8.0	37.1	4.5	8.2	7.1	Bontle
9	18:00-20:00	38.7	4.0	9.7	38.2	4.4	8.7	37.0	4.4	8.4	8.9	Bontle
10	20:00-22:00	41.0	5.8	7.1	38.3	4.5	8.5	37.5	4.7	8.0	7.9	Bontle
11	22:00-00:00	39.0	4.5	8.7	38.9	4.2	9.3	37.4	4.9	7.6	8.5	RT
	AVERAGE	38.0	4.9	7.8	37.4	4.4	8.5	37.0	4.6	8.1	8.1	

	09-Oct-10		
Min Mn %	36.1	35.4	36.1
Max Mn %	41.0	38.9	37.5
Daily Mn Variability %	4.9	3.5	1.4

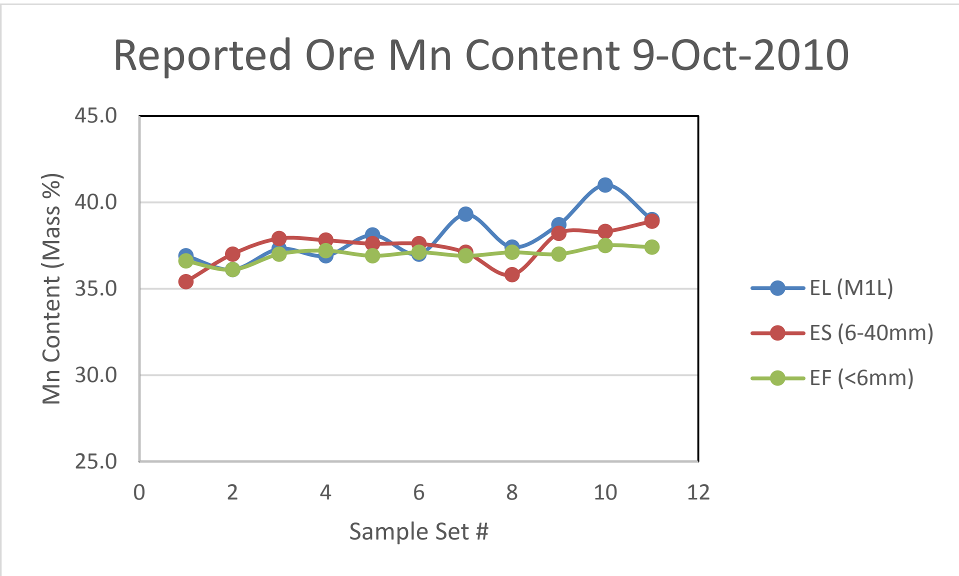

Figure 5.3-9 Plotted graph of the ore manganese content for sample sets taken on 9 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 4.9% was evident.

Table 5.3-10 Ore Manganese compositions for sample sets taken on 10 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines

OPP DAILY REPORT

DATE: 10-Oct-10

Sample Set #	1. PLANT	EL (+25)M1L			ES +6-40mm			EF -6mm			Average	ANALYST
	TIME	Mn	Fe	Mn /Fe	Mn	Fe	Mn/Fe	Mn	Fe	Mn/Fe	Mn/Fe	
1	00:00-02:00	38.1	3.9	9.8	36.9	4.0	9.2	37.1	4.0	9.3	9.4	RT
2	02:00-04:00	36.9	4.0	9.2	36.2	4.2	8.6	35.2	4.4	8.0	8.6	RT
3	04:00-06:00	34.5	4.5	7.7	36.1	4.2	8.6	36.9	4.1	9.0	8.4	HOLLY
4	06:00-08:00	35.1	4.0	8.9	36.7	3.7	9.9	35.8	3.4	10.5	9.8	Felicia
5	08:00-10:00	36.1	3.8	9.5	34.6	4.0	8.7	35.5	3.5	10.1	9.4	M M
6	10:00-12:00	36.3	3.3	11.0	36.1	3.4	10.6	36.1	4.1	8.8	10.1	M M
7	12:00-14:00	38.1	3.6	10.6	37.6	3.9	9.6	36.4	3.8	9.6	9.9	M M
8	14:00-16:00	37.6	4.1	9.2	37.5	4.5	8.3	37.1	4.8	7.7	8.4	Bontle
9	16:00-18:00	38.6	5.6	6.9	37.9	4.1	9.2	37.6	4.4	8.5	8.2	MATTHEWS
10	18:00-20:00	37.0	3.7	10.0	38.3	4.3	8.9	37.4	4.6	8.1	9.0	Bontle
11	20:00-22:00	41.1	4.0	10.3	37.3	3.9	9.6	36.8	4.0	9.2	9.7	Bontle
12	22:00-00:00	37.7	4.3	8.8	37.7	4.6	8.2	37.3	4.6	8.1	8.4	HOLLY
	AVERAGE	37.3	4.1	9.2	36.9	4.1	9.1	36.6	4.1	8.8	9.0	

	10-Oct-10		
Min Mn %	34.5	34.6	35.2
Max Mn %	41.1	38.3	37.6
Daily Mn Variability %	6.6	3.7	2.4

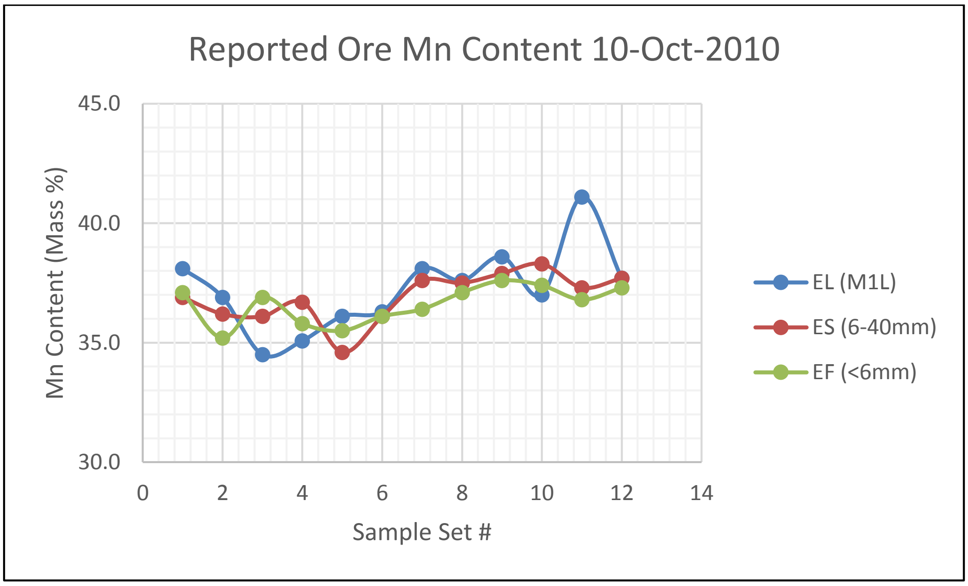

Figure 5.3-10 Plotted graph of the ore manganese content for sample sets taken on 10 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 6.6% was evident.

Table 5.3-11 Ore Manganese compositions for sample sets taken on 11 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

OPP DAILY REPORT

DATE: 11-Oct-10

Sample Set #	1. PLANT	EL (+25)M1L			ES +6-40mm			EF -6mm			Average	ANALYST
	TIME	Mn	Fe	Mn /Fe	Mn	Fe	Mn/Fe	Mn	Fe	Mn/Fe	Mn/Fe	
1	00:00-02:00	39.5	4.5	8.8	38.5	4.7	8.2	37.8	5.2	7.3	8.1	HOLLY
2	02:00-04:00	35.4	3.7	9.6	37.0	4.5	8.2	37.3	4.8	7.8	8.5	RT
3	04:00-06:00	33.1	5.0	6.6	36.8	4.3	8.6	36.6	4.3	8.5	7.9	RT
4	06:00-08:00	37.1	4.2	8.8	36.3	4.2	8.6	36.1	4.7	7.7	8.4	BONTLE
5	12:00-14:00	37.9	4.2	9.0	35.8	4.0	9.0	36.8	4.7	7.8	8.6	BONTLE
6	14:00-16:00	38.5	4.1	9.4	38.8	4.2	9.2	38.1	4.1	9.3	9.3	MATTHEWS
7	16:00-18:00	36.3	4.1	8.9	38.2	4.4	8.7	38.5	4.1	9.4	9.0	MATTHEWS
8	18:00-20:00	36.7	3.7	9.9	38.2	4.4	8.7	38.2	4.2	9.1	9.2	BONTLE
9	20:00-22:00	37.3	3.8	9.8	37.5	4.4	8.5	38.4	4.8	8.0	8.8	MATTHEWS
10	22:00-00:00	37.8	4.4	8.6	38.6	4.4	8.8	37.4	4.5	8.3	8.6	HOLLY
	AVERAGE	37.0	4.2	8.9	37.6	4.4	8.6	37.5	4.5	8.3	8.6	

	11-Oct-10		
Min Mn %	33.1	35.8	36.1
Max Mn %	39.5	38.8	38.5
Daily Mn Variability %	6.4	3.0	2.4

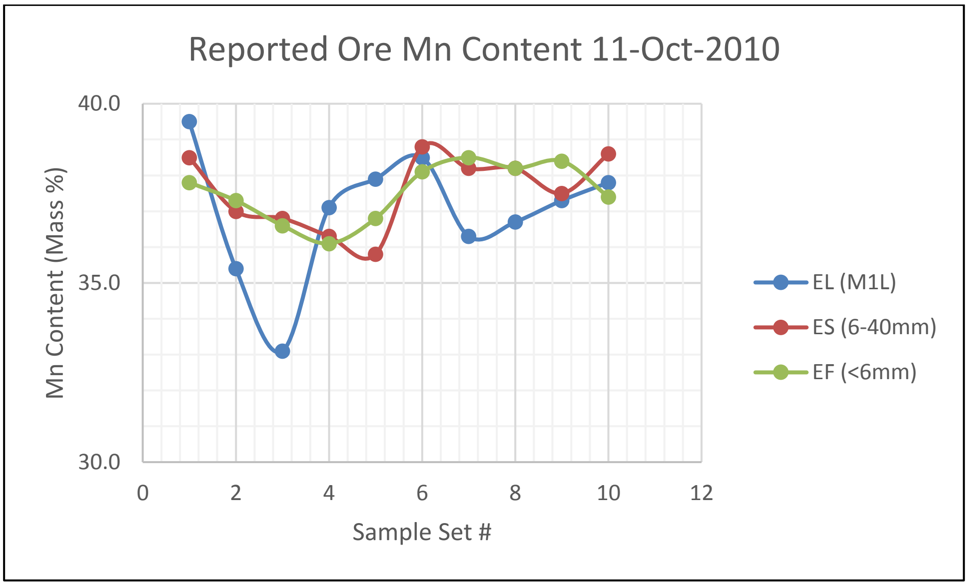

Figure 5.3-11 Plotted graph of the ore manganese content for sample sets taken on 11 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 6.4% was evident.

Table 5.3-12 Ore Manganese compositions for sample sets taken on 12 Oct 2010 from the three pre-trial sampling points. Minimum and maximum manganese %, as well as daily manganese variability is indicated.

Hotazel Manganese Mines

OPP DAILY REPORT

bhpbilliton resourcing the future

DATE: 12-Oct-10

Sample Set #	1. PLANT TIME	EL (+25)M1L Mn	EL (+25)M1L Fe	EL (+25)M1L Mn /Fe	ES +6-40mm Mn	ES +6-40mm Fe	ES +6-40mm Mn/Fe	EF -6mm Mn	EF -6mm Fe	EF -6mm Mn/Fe	Average Mn/Fe	ANALYST
1	00:00-02:00	39.0	4.1	9.5	37.7	4.9	7.7	36.8	4.8	7.7	8.3	RT
2	02:00-04:00	40.4	4.9	8.2	38.5	4.4	8.8	37.5	4.8	7.8	8.3	HOLLY
3	04:00-06:00	39.4	4.2	9.4	39.7	4.5	8.8	39.9	5.2	7.7	8.6	HOLLY
4	08:00-10:00	40.8	4.8	8.5	40.1	4.2	9.5	37.6	4.9	7.7	8.6	M.M
5	10:00-12:00	40.6	4.8	8.5	38.0	4.4	8.6	37.9	4.8	7.9	8.3	M.M
6	12:00-14:00	37.7	4.0	9.4	37.5	4.9	7.7	38.0	4.8	7.9	8.3	M.M
7	14:00-16:00	41.0	4.2	9.8	40.1	4.6	8.7	38.3	5.0	7.7	8.7	BONTLE
8	16:00-18:00	37.8	3.9	9.7	38.8	4.8	8.1	38.3	4.8	8.0	8.6	MATTHEWS
9	20:00-22:00	36.5	4.1	8.9	39.2	4.8	8.2	37.6	4.8	7.8	8.3	BONTLE
10	22:00-00:00	38.8	4.5	8.6	36.3	6.3	5.8	37.3	4.9	7.6	7.3	HOLLY
	AVERAGE	39.2	4.4	9.0	38.6	4.8	8.1	37.9	4.9	7.8	8.3	

	12-Oct-10		
Min Mn %	36.5	36.3	36.8
Max Mn %	41.0	40.1	39.9
Daily Mn Variability %	4.5	3.8	3.1

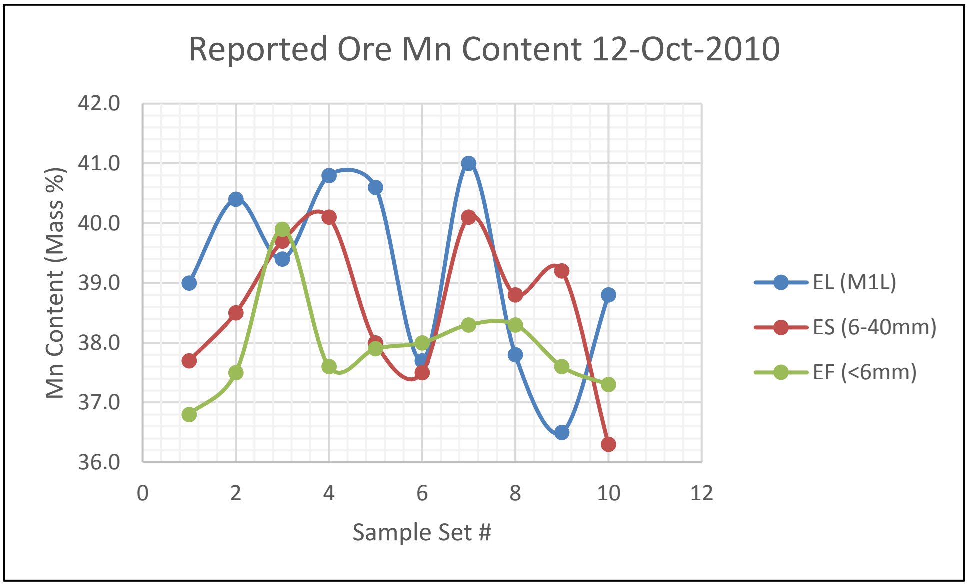

Figure 5.3-12 Plotted graph of the ore manganese content for sample sets taken on 11 Oct 2010 from the three pre-trial sampling points. A maximum ore manganese variability band of 4.5% was evident.