Mathematical Modelling of the base metal matte converter

by

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Declaration

I declare that this thesis, submitted for the degree of Master of Science in Engineering at the University of Cape Town, is my own unaided work. It has not been submitted before, for any degree or examination, at this or at any other university.

Joseph Kabwe

Abstract

The matte converting furnaces are found at the heart of the copper and nickel smelting processes where oxygen contained in air and industrial oxygen is used to oxidise iron in the furnace matte to produce blister copper or the iron free matte suitable for treatment in the downstream processes. A fundamental knowledge of the base metals matte smelting and converting processes system is valuable for maintaining a stable and optimum process operation.

A mathematical model for the Ni-Cu-PGM matte converting process has been developed based on the assumption that the converting process is both in chemical and thermal equilibrium. The converting process demonstrated by the model is the Top Submerged Lance (TSL) base metal continuous converter that treats Ni-Cu-PGM furnace matte similar to the one utilized at one platinum smelter. The TSL PGM converter was chosen for the reasons that the Ausmelt/Isasmelt TSL smelting furnaces are highly productive flexible smelting and converting units and environmentally friendly that requires a low capital cost compared to other smelting processes. The model predicts the masses of the process inputs of air, oxygen, fuel coal or oil, silica flux and the outputs components contained in the three products high grade matte, slag and offgas.

The mathematical model utilises the matrix-inverse method that is used to solve the matrix representing a system of several linear equations to give the values of unknown variables. The mathematical matrix-inverse method provides us with the tool to solve many large linear equations arranged in a square matrix numerically to gives solutions. The matrix-inverse method was chosen because it is part of windows excel formulas as MINVERSE and MMULT formulas which is accessible on most computers with microsoft windows program. The inverse of the matrices is solved using the Gauss -Jordan mathematical method.

The study found that the matrix simulation model is able to give accurate predictions of the masses of the industrial TSL converter process inputs fuel, air, oxygen, and silica flux as well as the process outputs masses of the components of converter matte, slag and offgases. The model also indicated that changes in the feed matte composition, moisture in feed, matte feed rate, oxygen enrichment, bath operating temperature and fuel type results in change in one or more of the process inputs requirement and masses and composition of product phases. The simulated model results compare very well with the industrial plant process parameters sourced from one of the platinum smelters that use the TSL technology for converting furnace matte. The results predicted by the model provides good insight into the operations, control and design of the TSL Ni-Cu matte converter and therefore the model is regarded as an invaluable asset for predicting the process dynamics of the present and future converting units. The model provides a good understanding of the base matte converting process and as such will serve as a useful tool in improving and optimising the base metals converting processes without carrying out extensive and expensive tests.

It is concluded from the study that the mathematical matrix-inverse calculation method is suitable for modelling the smelting and converting processes of the Ni-Cu-Fe-PGM matte in the TSL converter. The model results were validated through comparison with actual plant operating parameters sources from the relevant platinum smelter which indicated an accuracy of over 90% on most parameters. The modelling of the base metal smelting and converting processes has valuably been furthered, however for complete confidence in the model results, further validation is recommended using other simulation softwares and methods.

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Nomenclature

PGM	Platinum Group metals
TSL	Top submerged lance
ACP	Anglo platinum Converting Process
BMR	Base metals refinery
PMR	Precious metals refinery
PSC	Peirce Smith converter
XT	Xstrata technology
Kg	Mass in Kilograms
KW	Power in kilo Watts
MJ	Energy in mega joules
m	length in meters
L	Volume in litres
s	time in seconds
hr	Time in hours
Nm^3	Volume in normal meter cubic
T	Temperature in degrees ^oC or K
VR	Viscosity Ratio constant
μ	Viscosity in Kg/m.s
ρ	Density in Kg/m^3
Cp	heat capacity
A, B, C D	heat capacity constants
H	Enthalpy in MJ
Kpa	Pressure in kilo Pascal
k	Heat loss flux in kW/m^2

Dedication

To my son, wife, parents and sisters for the sacrifices and endurances they made to enable me to finish studies and be able to see the light at the end of the tunnel. To the late great reggae musicians Bob Marley and Peter Tosh for the inspiration messages in their songs that has always given me the strength to conquer so many obstacles and survive through the great storm during my childhood days. To my late brother Moses and sister Ruth who left this world so early. Lastly to the first Zambian president Dr Kenneth Kaunda whose policies enabled most of us to realise our dreams through the free education policy.

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Last but not the least I want to thank my family for their emotional support, inspiration and endurance during the long and demanding study time.

1.0 Introduction

1.1 Sulphide minerals

The purpose of this research project is to demonstrate the simulation of the converting process of the Ni-Cu-Fe-PGM matte using the matrix mathematical model. A comprehensive matrix inverse mathematical model is used to evaluate options for operating a PGM matte converting process for a given set of process conditions.

Copper and nickel extraction from sulphide minerals is mostly accomplished by pyro metallurgical processes known as smelting (Davenport 2002). Pyrometallurgy is an ancient art which has defined significant stages of human development (Davenport 2002). The major sulphide minerals in copper concentrates are chalcopyrite ($CuFeS_2$), bornite (Cu_5FeS_4), chalcocite (Cu_2S), covellite (CuS) and pyrite (FeS_2). Some sulphide concentrates contain minor amounts of oxide copper namely malachite ($CuCO_3.Cu(OH)_2$) and azurite ($CuCO_3.Cu(OH)_2$ (Davenport 2002). In PGM concentrates nickel exists mainly as pentlandite ($Ni_9Fe_9S_8$) in association with other copper sulphide minerals. Other minerals contained in sulphide concentrates are oxides of SiO_2, CaO, MgO, Al_2O_3 and Cr_2O_3 (Jones 1999, Davenport 2002).

Smelting is intended to separate the gangue oxide minerals from the sulphide minerals associated with the noble metals (Davenport 2002). Sulphide copper and PGM concentrates are normally treated in two stages; smelting and converting. The smelting stage is where the low grade matte and slag are produced. The converting stage is where the iron and sulphur in the matte are progressively oxidized with the use of oxygen for the production of blister copper or high grade matte. There is a significant difference in the oxygen potential between the smelting and converting processes. The smelting stage is less oxidizing so that low copper or nickel slag can be produced. The converting stage is strongly oxidizing so that iron and sulphur can be sufficiently removed from the blister or high grade matte as the case with Ni-Cu- PGM concentrates (Davenport 2002, Jones 1999).

Copper or PGM smelting processes can be classified as bath or flash smelting. In bath smelting, smelting and converting occurs predominantly in a molten liquid bath. Concentrate comes in contact with the liquid slag and matte. The matte is converted by air or oxygen enriched air that is injected into or on top of the molten bath. In flash smelting, the concentrate is dispersed into an air or oxygen enriched stream and smelting and converting occurs while the concentrate is suspended in the air stream.

Different types of furnaces are used for smelting and converting worldwide. The old reverberatory smelting furnace which was the most popular furnace for copper sulphide smelting until the 80’s has been replaced with new smelting technologies in almost all the copper smelters around the world and only a few are still in operation. The most common primary smelting and converting furnaces for Cu and Ni-Cu- PGM concentrates are the Outokumpu Flashfurnace, Noranda/Elteniente, Mitsubishi smelting process, Isasmelt/Ausmelt TSL furnaces, Electric furnaces and the Peirce-Smith converters (Davenport 2002).

The mined sulphide ore is crushed and milled into a fine powder and concentrated for further processing by using the froth flotation process. In the concentration process, ground ore is mixed with water and flotation chemicals and passed through the flotation cells where the sulphide mineral particles are separated from the gangue oxide minerals. The grade of copper concentrate ranges between 20-45% Cu. In PGM concentrators the grade of the concentrates produced are quite low and are usually below 5% for both Cu and Ni. The Ni-Cu-Sulphide concentrates are treated by pyro metallurgical process routes to yield high purity metal products or high grade metal sulphides like in copper-nickel smelting processes. Oxide ores which may occur in other parts of the mine, together with other oxidized waste materials, are treated mainly through the hydrometallurgical processes.

During smelting the concentrates are dried and fed into one of several different types of furnaces. At high temperatures, sulphide minerals decompose into different phases and melt to yield a layer of matte which is a mixture of Cu-Ni-Fe sulphides, and slag, which is a mixture of gangue oxide minerals of SiO_2, CaO, FeO, Fe_3O_4, MgO and Al_2O_3 (Davenport 2002). Lime or silica fluxes are blended with the concentrates with a view to balance the charge chemistry that will support the formation of the low melting olivine slag whose liquidus temperatures is around 1100^oC for low MgO slags and 1350^oC for high MgO slags (Jones 1999).

The furnace matte produced have a metal content of 40 to 65% Cu for copper smelting (Davenport 2002) and 5-10% Cu and 12-18% Ni for the PGM sulphide smelting (Jones 1999). Furnace matte produced in the primary smelting furnaces is further processed in the converters by the use of oxygen and air to oxidise iron and sulphur to produce blister copper with copper content of >98% Cu or high grade Ni+Cu matte with metal composition of Ni+Cu>72%. During converting the SO_2 gas generated is cooled, cleaned and converted into sulphuric acid through the contact acid plant process. Blister copper, containing a minimum of 98.5% copper is fire refined to high purity copper in two steps in the anode furnace. The first stage of fire refining is the stage where molten blister copper in the anode furnace is blown with air using one or two lance pipes to remove residual sulphur. The second stage is where natural gas or propanol and air are blown into the molten copper to remove the dissolved oxygen from the molten copper. The molten copper of grade 99.50% Cu is then cast into the copper anodes that are further electro-refined into 99.99% cathode copper (Davenport 2002).

In PGM or nickel converting, the converting process only ends on the slag blow stage which is the iron oxidation stage. The white metal or iron deficient matte of grade Ni+Cu higher than 70% is granulated as performed at Lonmin and Impala platinum mines or cast into bottom cast moulds and slow cooled as done at Anglo platinum (Jones 1999, Jacobs 2006). The high grade matte from the converters is shipped to the BMR refineries for further processing. The common flow sheets of the copper and PGM matte smelting processes are shown on the diagrams indicated in figures 1.1 and figure 1.2.

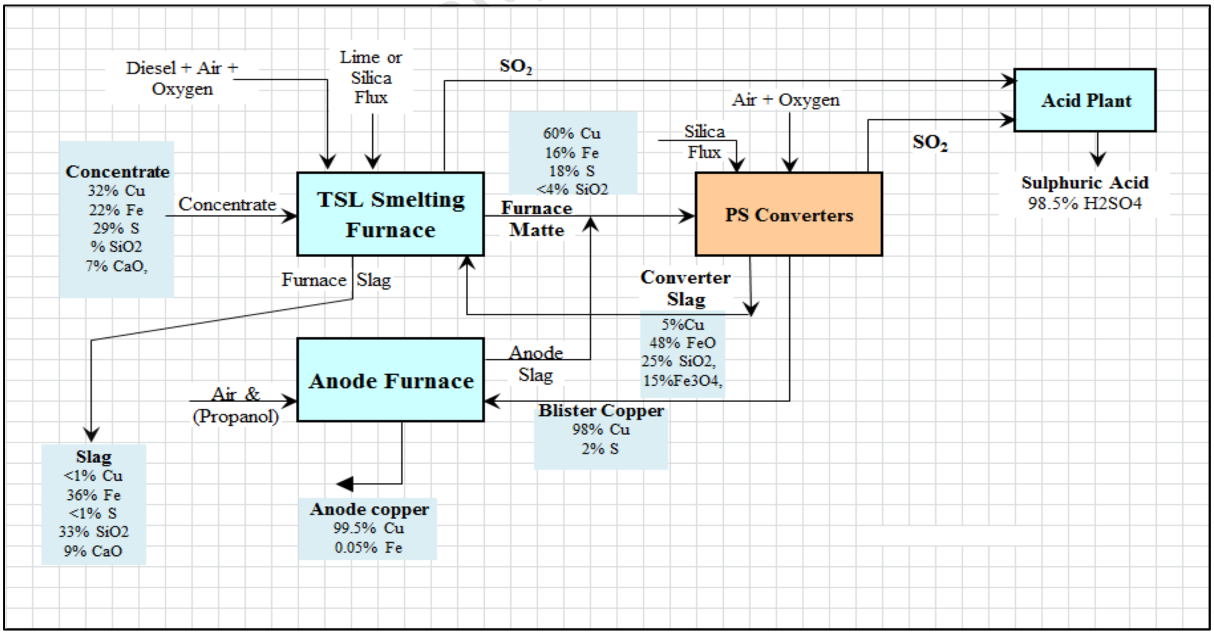


Figure 1.1: Copper smelting flowsheet (Author)

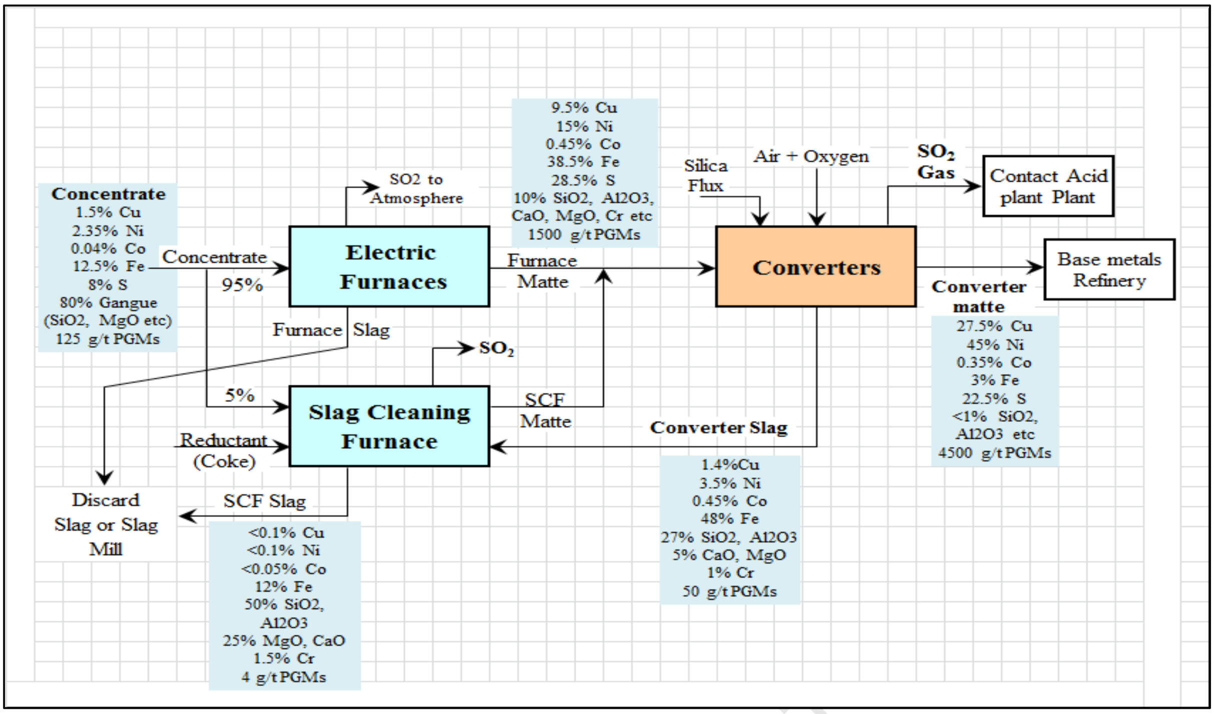


Figure 1.2: Nickel-Copper-PGM concentrate smelting flowsheet(Author)

In matte sulphide converting plants, the old Peirce Smith converter is still the dominant converting unit both in copper and nickel. The Noranda/Teniente converters are used in some copper smelters mostly in South American smelters and produces white metal of copper content 70 -75% (Davenport 2002). The Outokumpu direct to blister flash furnace is used in only three smelters around the world. The TSL converting process has been in existence since 2000 at four smelters, one for copper in China, one at Umicore in Belgium for copper-lead matte, Aurubis in Germany and the PGM nickel-copper matte in South Africa. The TSL converters were developed by Ausmelt and Xstrata technology after the successful development of the TSL primary smelting furnaces (Matusewicz 2005, Bakker 2011).

Anglo platinum replaced six Peirce Smith converters with the TSL converters in order to meet the more stringent sulphur dioxide gas emissions regulations after evaluating different converting technologies. The TSL converter treats more PGM copper-nickel furnace matte in a single furnace than the Peirce Smith converter and the offgas sulphur capture is much higher above 95% (Jacobs 2006).

The TSL converter is viewed as the potential alternative to the Pierce Smith converter because of its high productivity and excellent gas capture of the SO_2 containing offgas (Matusewicz 2005, Bakker 2011). The TSL PGM converter is a continuous process run with high oxygen enrichment resulting in high SO_2 concentration in the offgas. In contrast, the Pierce Smith converter is run on a batch cycle with air and rarely with oxygen enrichment due to high tuyere wear rate (Davenport 2002).

1.2 Problem statement

A process model is a device that mimics the behaviour of an existing or conceptual process. The output of a process modelling development is a process model that can be used to simulate the behaviour of the process and in some cases modelling results of the new process generated by the model.

In plant design, most houses rely on minerals processing simulation software for modelling new plant processes. These softwares are very expensive to obtain and maintain and require specialist skills offered by the software providers to operate them.

The matte converting process is described as a complex process involving many variables. In plant operations, the matte converting process is susceptible to a large number of influences of which the matte feed rate, fuel coal feed rate, oxygen/matte ratio, flux/matte ratio, bath temperature, airflow rate, oxygen enrichment ratio, feed matte quality and moisture in feed matte are the main control parameters. These control parameters need to be continuously monitored and adjusted due to changes in the feeds chemical quality, products quality and slag temperature requirement in order to maintain the plant process steady state status. It is evident that a comprehensive knowledge of the converting process and furnace behaviour is required to stabilise process and to expand and improve the current process operation. The extreme process conditions do not permit uncomplicated evaluation of the furnace interior and one is left with only the external parameters to determine the behaviour of the inside. The major problem encountered in metallurgical plants by operators is the lack of availability of a comprehensive tool to be used other than the tables and charts to correct a major process upset. Inputting inaccurate process set points into the plant process control system may lead to a process failure resulting into loss of production and equipment damage.

It is believed that by taking advantage of the recent advances in computer mathematical programs, metallurgists and process engineers can develop in-house simulation models using some mathematical functions in windows excel or in visio basic (VB) programs which is available on most office computers. These mathematical tools can be utilised to produce fully functional models within a very short period of time and at a low cost. The models can be used effectively for the plant process control, feasibility studies, design, fault-finding, optimisation and re-engineering phases of a process. With the availability of powerful and relatively inexpensive windows computer softwares, it has become feasible to deploy models on-line in the plant. This makes valuable metallurgical/chemical engineering knowledge available to operational personnel around the clock, even when engineers are not present. These on-line models can be used for operator assistance and model-based control to improve the ability of the operation to meet its business related objectives.

1.3 Objectives

The matrix inverse mathematical method was developed to solve a large number of linear equations and has recently found its way to the modelling of processes in the metallurgical and chemical industry. The inverse mathematical method is a powerful engineering toolkit for creating a virtual computer prototypes of processes that would under the actual operating conditions be impossible to predict and quantify. The matrix is based on the computational excel program MINVERSE and MMULT functions and incorporates various equations that represents the mass balances, heats balances and metals distribution coefficients between matte slag and offgas. A mathematical model developed using the Microsoft windows excel program serves as a tool to simulate the oxidation process of the furnace Ni-Cu matte to produce the iron deficient matte and the slag with the desired physical and chemical properties.

As part of the evaluation to utilise the matrix inverse mathematical method to model metallurgical processes and in order to aid in the overall understanding of base metal matte converting process, the primary objective of the study then is to develop the TSL base metals matte converting model using the mathematical matrix inverse method. The model describes the converter process in the form of a set of linear equations that represent the mass and enthalpy balances as well as the distribution of elements and mineral compounds between matte and slag. The linear equations are arranged in the form of matrixes whose solutions are solved by the matrix inverse method. The model is to be developed to simulate the process input requirements and the products quality and quantities at the desired operating conditions.

1.4 Plan of development

In order to meet the project objectives, a converting model representing the converting system as closely as possible has to be created. For this to be achieved, the following must be performed:

- Understanding and knowledge of the furnace operations. The characteristics of the furnace must to be identified in order to determine the modelling scope and strategy.
- Mass balance equations of all elements and compounds in the inputs have to be developed
- Enthalpy balance equations have to be developed for the inputs and outputs at the stated temperatures.
- The heat loss equation has to be developed from the furnace walls and bath.
- Elemental partition coefficients equations for Cu, Ni, Fe and Co between product matte and slag has to be developed.
- The slag FeO and Fe_3O_4 distribution equations within the slag has to be developed.
- Slag quality equation for Fe/SiO_2 in slag has to be developed.
- Product matte Fe content equation have to be developed.
- The model must generate justified numerical results that have to be processed and interpreted in order to formulate a discussion and draw conclusions. Possible validation of the results will strengthen confidence in the model outcome by comparing with actual plant information reported in appendix5.

1.5 Thesis overview

The main body of this thesis is presented in chapters that form the backbone in the development of the model for the TSL base metal matte converting process. In the following chapters the following topics will be discussed.

- Chapter 2
  Literature review of the sulphide mineral Smelting and converting processes
- Chapter 3
  Model development method

- Chapter 4

  Step by step model application and discussion of the results. The effect of change in the operating conditions are also demonstrated in this chapter by the model.
- Chapter 5

  Conclusions drawn from the results of the model.
- Chapter 6

  Recommendations

The references are found in the last chapter followed by the extensive appendix for supporting the claims made in the thesis.

2.0 Literature Review of sulphide Smelting

2.1 Slags

In a smelting process, the main focus is the need to balance the feed blend which will result in the formation of the slag that has a low liquidus temperature and viscosity. The liquidus temperature is the highest temperature at which a solid can exist in the particular slag system (Garbers 2010). The liquidus temperature should be as low as possible to minimise heat loss by radiation, reduce wear of the furnace refractory lining and to prevent super heating of the matte and possible corrosion of the furnace refractory lining (Eric and Hejja 1995, Nell 2004). The slag should also have low solubility for the valuable metals so that they report to the matte for recovery. The optimisation of metallurgical processes requires reliable data of the slag phases. A slag is a molten solution of metal oxides. These oxides include FeO from Fe, SiO_2 or CaO from fluxes blended with concentrates. Oxides commonly found in copper and nickel smelting slags are silica (SiO_2), ferrous oxide (FeO), lime (CaO), ferric oxide (Fe_2O_3), magnesia (MgO) and small amounts of alumina (Al_2O_3) and chrome (Cr_2O_3). The formation of the slag of suitable chemical and physical properties for optimum furnace operation predicates a successful smelting operation (Nell 2004).

The molecular structure of molten slag is described by dividing its oxides into three groups which is acidic, basic and neutral. The best known acid oxides are silica and alumina. When these oxides melt they polymerise, forming long polyions (Mills 1995). These polyions give acidic slags high viscosities making them difficult to work with. Acidic slags also have low solubilities for other acidic oxides and these cause difficulties in copper making as impurities such as As_2O_3, Bi_2O_3, Sb_2O_3 won't be removed in the slag (Mills 2011). Adding basic oxides such as CaO and MgO to acidic slags breaks the polyions into smaller structural units. As a result, basic slags have low viscosities and high solubilities for acidic oxides and the addition of basic oxides to a certain level also lowers the melting point of slags (Mills 1995). Neutral oxides such as FeO and Cu_2O react less strongly with polyions in a molten slag, nevertheless they have much the same effect. FeO and Cu_2O have low melting points and so they tend to lower the slag melting point and viscosity. Viscosity is the property of the fluid to resist the force tending the fluid to cause the flow. More specifically this property reflects the extreme to which relative motion of adjacent liquid layers is retarded and it can be generally regarded as a measure of the internal friction of the liquid (Mills 2011). Many measurements have been made of the viscosities of molten slags. These have been used to develop models which calculate the viscosity as a function of temperature and composition. One example of these measurements is the one done by Utihard and Warczok (1995) indicated by equations 2.01 to 2.04 and relies on the calculation of the viscosity ratio (VR).

$$VR = \frac{Aa}{Bb} \tag{2.01}$$

Aa= (%SiO_2) + 1.5(%Cr_2O_3) + 1.2(%ZrO_2) + 1.8(%Al_2O_3
(2.02)

Bb = 1.28*%FeO + %Fe_2O_3+%PbO + 0.8*%MgO+0.7*%CaO
+ 2.3*(%Na_2O+%K_2O) +0.7*%Cu_2O+1.6*%CaF_2 (2.03)

The VR is the ratio of Aa, an equivalent mass percentage in the slag of acidic oxides to Bb, an equivalent mass percentage of basic oxides. Utigard and Warczok (1995) related VR to viscosity by regression analysis against their existing database and obtained the equation 2.04.

$$\text{Log } \mu \text{ (kg/m.s)} = -0.49-5.1\sqrt{VR} + \frac{-3600+1280\sqrt{VR}}{T(K)} \tag{2.04}$$

Where μ is viscosity, T is temperature in Kelvin and VR is viscosity ratio

The specific gravity of smelting slags ranges between 3.3-3.7 and increases with increasing Fe_2O_3 and SiO_2 content (Utigard 1994) and it increases slightly with increasing temperature. Equations 2.05, 2.06 and 2.07 are some of the empirical equations developed to estimate the density and viscosity of slags by the researchers indicated.

ρslag(Kg/m^3) =2680+581 $* \frac{wt\%\ FeO}{wt\%\ SiO2}$ - 0.03*(T-1450^{o}C)

FeO-MgO-SiO_2 slag system Utigard 1995 (2.05)

ρslag(Kg/m^3) = -0.3*T + 3146 Turkdogan 1983 T in ^{o}C (2.06)

μslag(Kg/m.s) = -15.9748 + $\frac{26361.5}{T}$ Turkdogan 1983 T in ^{o}C (2.07)

An understanding of the thermo physical properties of silicates is fundamental as the silicate structure forms the basis for the slag formation (Yazawa 1953, Mills 2011). It is well known that properties such as viscosity and melting temperature of slags is dependent on the silicate structure in a particular slag system i.e extent of polymerisation of the silicate network. Pure silica consists of 3-dimensional interconnected networks of SiO_4^{4-} tetrahedral in which Si cations are joined together as rings or chains by bridging oxygens and has an extremely high viscosity and a melting temperature. As basic oxides are introduced, the network disintegrates gradually into smaller unit, leading to the increase in the non-bridging oxygen and eventually free oxygen ions O^{2-}. As the basic oxides are added to the SiO_4 tetrahedral, the slag liquidus temperature and the viscosity decreases rapidly (Yazawa and Kameda 1953, Mills 1995). Table 2.1 indicates the influences of different oxides of the metals on the physical and chemical properties of the slag.

Table 2-1: Metals oxide influence on physical and chemical properties of slag (Mills 1995)

Type	Ion & Valency Z	Field Strength Z / r^2 (electro negativity)	Bond Strength kcal/mol	Comments
Network Forming Ions	B^{3+}	56.7	120	**More Acidic- Network formers** High Valance State Small Cations High Electro negativity Open Structure High Bond Strength Covalent Si^{4+} Substitution
	P^{5+}	40.8	90 - 110	
	Si^{4+}	21.6	105	
Amphoteric or Intermediate Ions	Al^{3+}	11.5	100	**Acidic or Basic** Depending upon Environment
	Ti^{4+}	8.7	75	
	Fe^{3+}	7.3		
	Zr^{4+}	6.4	60	
Network Modifying Ions	Mg^{2+}	4.6	37	**More Basic – Network breakers** Low valence State Large cations Low Electro negativity Lattice Distortion Low Bond Strength Ionic (e^- donating) Bond breaking
	Zn^{2+}	3.6	36	
	Fe^{2+}	3.5		
	Ca^{2+}	2.1	32	
	Pb^{2+}	1.4	36	
	Na^{+}	1.1	20	
	K^{+}	0.6	13	

The slags produced in industrial matte smelting processes consist primarily of SiO_2, FeO, Fe_2O_3, CaO, MgO with minor amounts of, Cr_2O_3 and Al_2O_3 (Mills 2011). One important tool used for slag chemistry optimisation is the SiO_2-FeO-CaO ternary system indicated in figure 2.1. The region highlighted in figure 2.1 is the region with the lowest liquidus temperature <1200°C. Since the temperatures attained in the sulphide furnaces are lower at maximum of 1350°C, the concentrates and fluxes must be blended to promote the formation of the slag with the liquidus that fall in the highlighted in figure 2.1. The slag which is contained in the refractory lined vessel should be saturated or close to saturation with the oxide phases which are present in the refractory in order to minimise chemical wear of the lining (Garbers 2010).

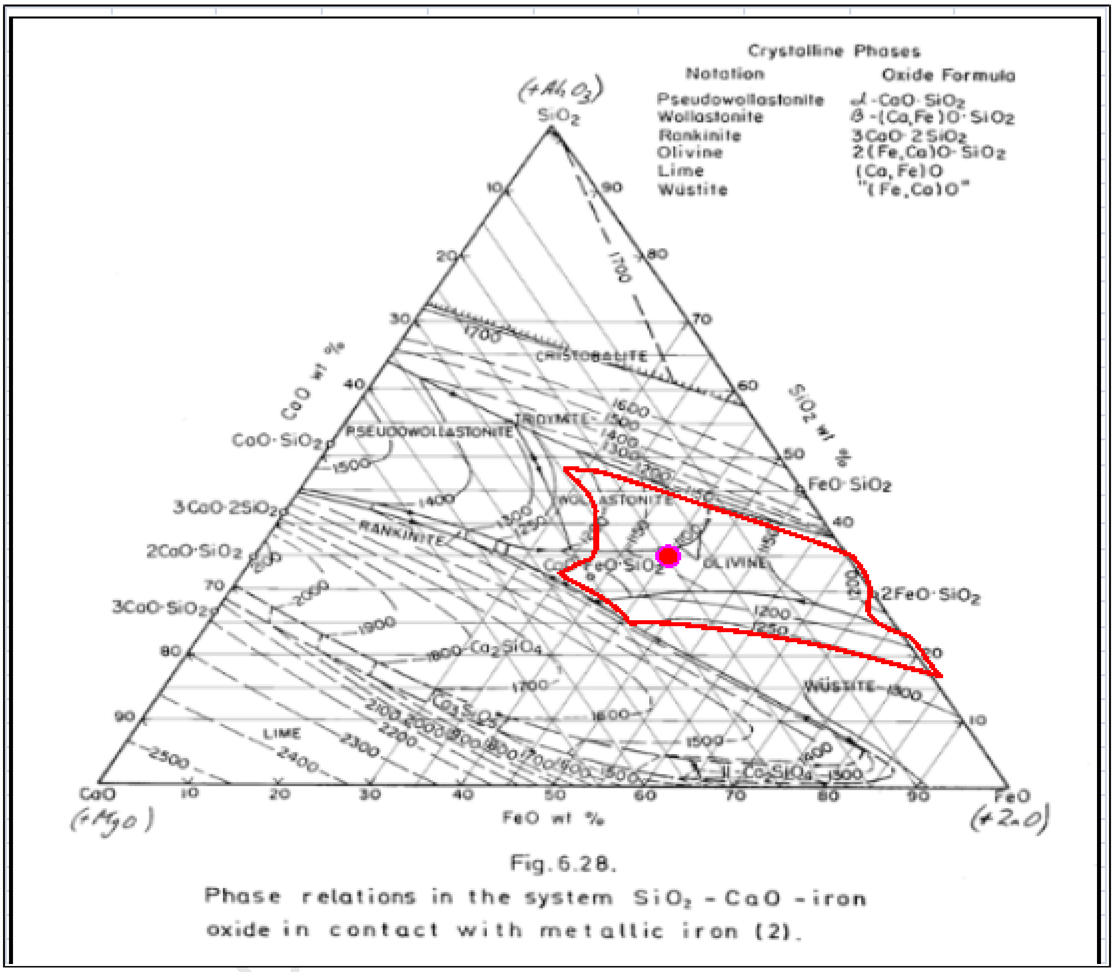


Figure 2.1: Ternary diagram for the SiO2-FeO-CaO slag system (Mills 1995)

Some mathematical equations have also been developed to predict the slag constants that represent slags that has low melting points and viscosities and one example of them is indicated by equation 2.08.

$$MVR = \frac{1.332*\%SiO2+1.005*\%Al2O3}{0.87*\%Fe+0.5*\%CaO+0.56*\%MgO} \qquad (2.08)$$

Where the MVR is called the modified viscosity ratio (Eltringham 1993)

Equation 2.08 has been used at Nkana and Palabora smelters to prepare concentrate charge blends (Eltringham 1993). The desired MVR ranges from 3.5 -5.4 for the slag that is formed on the charge bank and 2.0-2.5 for the bulk slag formed by the charge bank slag and the return slag from the converters. The assays in the MVR formula are the assays of the concentrate feed blends that is charged to the furnace (Eltringham 1993).

The other common formula used in smelting in optimising the slag chemistry is the slag basicity which is the ratio of the basic oxides to the acidic oxides indicated in equation 2.09 (Mills 1995)

$$Basicity = \frac{\%\mathrm{CaO}+\%\mathrm{MgO}}{\%\mathrm{SiO2}} \qquad (2.09)$$

2.2 Refractories

Copper and nickel sulphide smelting furnaces are normally lined with chrome-magnesite and fire brick refractories (Davenport 2002). Refractories are a family of technical ceramics which are combinations of high melting crystalline oxides and chemical stability. The basic principle of the refractory material is the high stability of its oxides and components at elevated temperatures. The material must have a high solidus temperature which is the minimum temperature at which a liquid can exist in a particular refractory material (Garbers 2010). Most refractories are mainly comprised of SiO_2, CaO, MgO, FeO, Fe_2O_3, ZrO_2, Cr_2O_3, C and SiC (Mills 2011, Garbers 2010). The chemical stability of the refractory is determined by the free energy of formation. The more the negative the value, the more stable the oxide. On the Ellingham diagram, the free energy curves located towards the bottom of the graph represents those elements with a strong tendency to remain in the oxide phase eg Ca, Al, Mg Si, Cr and Zr which are common constituents of the refractories (Mills 2011). The stability of the compounds is also considered in respect to one another. A metal with a more stable oxide will tend to reduce the less oxide of another metal. For example the melting of aluminium in the alumina-silicate refractory lined vessel will result in reduction of silica forming aluminium oxide (corundum). The term refractory means stubborn to withstand not only heat but in many applications chemical attack, abrasion, thermo shock and rough handling which points to resistance of refractories to extreme conditions of heat and corrosion when hot and molten material are contained while being transported and or processed (Garbers 2010). The refractories must also exhibit chemical inertness to their environments. Additionally, high mechanical strength at all temperatures is required in order to resist load, impact, abrasion and erosion. Refractory materials also expand when heated or shrink when cooled down. If no permanent changes occur in the original dimensions, this effect is known as the reversible thermo expansion and this change is expected to be very low for a particular refractory. Temperature changes lead to thermo gradients within the refractory causing expansion or contractions that cause distortion and this causes cracking in brittle materials. Refractory linings must also ensure long, trouble free and more importantly an economical refractory installation (Garbers2010). Bricks are usually dense, self-supporting and more resistance to corrosion while monolithics are mostly castable materials that can be mixed at the point of use usually supplied in powdery form.

Mechanical damages as a result of thermo shock and stresses can cause the bricks to fail. Silica bricks undergo phase changes during firing and cooling which are accompanied by considerable volume changes which causes the bricks to crack especially at temperatures below 600°C. During heat up of silica bricks, heating speeds and holding time of the heat up curves must be strictly adhered (Garbers 2010). Table 2.2 indicates the common refractories that are utilised in most copper and nickel sulphide smelting furnaces (Garbers 2010).

Table 2-2: Chemical composition of common bricks (Garbers 2010)

Brick	**Al2O3**	**SiO2**	**MgO**	**CaO**	**Fe2O3**	**Cr2O3**
Silica	0.5	95.5	-	2.5	1.2	-
Fire Clay	42.5	53.5	0.6	0.4	2	-
Andalusite	58.5	37.2	0.3	0.3	0.9	-
Corundum(Alumina)	94.7	4.4	0.1	0.1	0.8	-
Mullite	74.2	27	0.05	0.15	0.15	-
Dolomite	2	1	35.6	51.7	0.8	-
Magnesite	0.3	1	97	1	0.4	-
Direct Bonded Magnesia chrome	10	<1.5	70	<1	5	10
Direct Bonded Chrome Magnesia	18	<2	50	<1	10	20
Burned Chrome Magnesite	22	<5	35	<1.5	-	25

2.3 Matte

Matte is one of the phases that are formed when complex mineral sulphide concentrates are exposed to high temperatures in furnaces. Matte is a molten solution of metal sulphides. Mattes do not consist of polyions like those in slags (Shimpo 1996). They appear instead to be represented as molten salts (Shimpo 1996). Their specific gravity is higher than that of slag (between 3.9 to 5.5) and therefore form the bottom layer in smelting furnaces and their melting points are lower at <1200^{o}C which is lower than that of slags. Matte viscosities are much lower than that of slags e.g. 0.003Kg/m.s for matte versus 0.2-1.0 Kg/m.s. for typical slags and increases with increase in temperature. The specific conductance is higher than that of slag and is between 200 to 1000ohm^{-1}cm^{-1} (Davenport 2002). The equations 2.10 and 2.11 indicated are some of the linear equations that are used to calculate the density of matte (Hyrn, Toguri 1996).

$$\rho matte = 4487.99 - 0.4645(T - 1000^oC) \qquad (2.10)$$

$$\rho matte = 6358 - 76.3[S] + 9.94[Cu + Ni] - 0.4645(T - 1000^oC) \qquad (2.11)$$

Where ρ is in Kg/m^3, T is in ^{o}C and S, Cu and Ni are compositions in mass percentages. The equations is valid for the temperature ranges 1100-1300^{o}C.

2.4 Noranda/Teniente Smelting Converters

The Teniente reactor also known as the Caletones Matte Treatments (CMT) is similar in design to the Noranda reactor and shares many features with the Noranda smelting furnace (Mackey 1995, Harris, 1999). The Noranda and the Teniente furnaces are both submerged-tuyere bath smelting and converting processes. The Noranda/Teniente converters are furnaces designed to smelt copper concentrates by use of the heat of oxidation of FeS in matte with oxygen enriched air to produce high grade matte with grade of 72-75% Cu (Davenport 2002, Mackey 1988).

The Noranda/Teniente smelting converters are large cylindrical furnaces of dimensions, 4.5-5.5m diameter by 18-26m length. The 75mm steel shell is lined with 18 inch magnesite-chrome bricks and has 35-65 tuyere pipes with diameters of 50-65mm and 4-10 special tuyeres for dry concentrate injection (Mackey and Campos 2001). The furnace has two matte tap holes and one slag tap hole and a 6m^2 mouth for adding seed matte and for venting the offgases out of the furnace during operation. The furnaces are rotatable so that their tuyeres can be rolled in and out of the liquids bath when blowing is interrupted (Davenport 2002). The Noranda/Teniente smelting furnaces account for 15 to 20% of world copper smelting. The teniente converter-smelting furnaces are the dominant smelting processes in Chile and are used at some plants around the world (Davenport 2002). The schematic diagram of the Noranda/Teniente converter is illustrated in Figure 2.2.

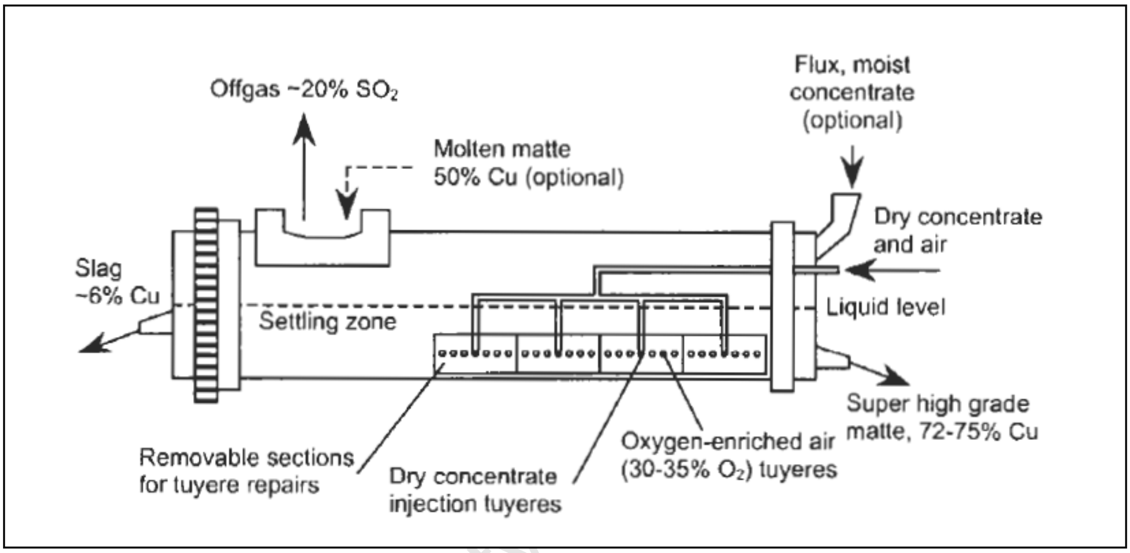


Figure 2.2: Noranda/Teniente converter diagram (Davenport 2002)

The Noranda/Teniente process is a continuous process in which smelting and converting takes place in a single reactor designed to produce high grade copper matte directly from sulphide concentrates (Campos 1993 and Alvarado 1995, 1999). The converting process involves the smelting of bone dried and wet copper concentrates and converting of FeS arising from smelted concentrates in a thermally autogeneous operation. The main difference between the Noranda and Teniente processes is that the Noranda process was initially conceived as a direct-to-copper smelting process. It used to produce molten copper from 1973 to 1975 (Davenport 2002) from wet concentrates containing 8% H2O mixed with coke/coal and silica flux which was charged onto the molten bath using the slinger. It was switched to high-grade matte production as done in the Teniente converter to lower impurity levels in the smelter's anode copper and to increase the smelting rate. The charging of concentrates has also been switched from wet to bone dry and is charge into the molten bath through dedicated tuyeres instead of the surface of the bath by the slinger (Davenport 2002). All Noranda furnaces now produce 72-75% Cu matte (Davenport 2002, Harris 1999).

The bone dried concentrate (<1.0% H_2O) is directly injected into the molten matte bath, with a mixture of oxygen enriched air of 30-50% by volume through specially designed tuyeres. Wet concentrates (8% H2O), silica flux and reverts are fed into the reactor through a garr-gun feeder or slinger onto the slag bath on the smelting and converting section. Coal/oil, flux and reverts are charged during converting to balance the heat and slag chemistry (Mackey 1988, Torres 1998). The converting process generates excessive heat such that external scrap and reverts are also charged to the furnace to maintain the bath temperature at 1250oC. The products of smelting and oxidizing of copper concentrate with flux are two separate molten phases, the high grade matte or white metal with the degree of oxidation being controlled so as to yield white metal with 72-75% Cu, a slag that contains 5-10% Cu and 16-24% Fe3O4 and offgases with SO2 concentration of 12-25% by volume (Davenport 2002). These molten phases are intermittently tapped in a counter current flow pattern, white metal at 1220°C and slag at 1250°C, through especially designed water cooled tap holes located at each side of the endplates. White metal is poured into ladles and transferred to the Peirce Smith Converters where final blowing into blister copper take place. The slag is recycled back to the primary smelting furnace or the slag cleaning furnace for the copper recovery process. The offgas is cooled, cleaned and dried and processed in the contact acid plant to produce 98% H_2SO_4.

Noranda/Teniente smelting converter is started by heating the furnace with hydrocarbon burners. Molten matte from another furnace is then poured in through the furnace mouth with tuyeres elevated or melting pieces of furnace matte in the furnace itself (Torres 1998). Once the matte bath level is > 1m, the tuyere blast is started and the tuyeres are rolled into the molten matte to begin oxidation and heat generation. Once the bath temperature reaches 1250^{o}C, concentrate and flux feeding is then started and normal smelting process is started (Davenport 2002, Alvarado 1995).

2.4.1 Chemical Reactions

During the smelting of concentrates in the Noranda/Teniente converter, the reaction mechanisms are as follows (Mackey 1988, Davenport 2002);

- Sulphide concentrates and SiO_2 flux are fed into the furnace from a slinger belt or garr gun and they are quickly absorbed and melted when they fall into the tuyere-blast stirred matte and slag bath.
- The dense sulphide high grade matte settles to the bottom and low grade matte arising from concentrate smelted is oxidized by oxygen in the blast to form FeO.
- Fe oxides react with SiO_2 flux to form slag which rises to the top of the bath.
- SO_2 from the oxidation reactions rises through the bath and leaves the furnace along with N_2 from the tuyere blast and CO_2/H_2O from hydrocarbon combustion.

Other components of the charge which is scrap and reverts materials melt and undergo oxidation and slagging. Oxides rise to the slag layer while copper and precious metals descend to the matte layer. The probable chemical reactions taking place in the bath are indicated by equations 2.20 -2.25;

$$Cu_2S\text{-}FeS(l) + 3/2O_2(l) = Cu_2S(l) + FeO(l) + SO_2(g) \qquad (2.20)$$

$$2FeO(l) + SiO_2(s) = 2FeO.SiO_2(l) \qquad (2.21)$$

$$3FeO(l) + 1/2O_2(g) = Fe_3O_4(l) \qquad (2.22)$$

$$FeS(l) + 3Fe_3O_4(l) = 10FeO(l) + SO_2(g) \qquad (2.23)$$

$$2CuFeS_2(s) + 4O_2(g) = Cu_2S(l) + 2FeO(l) + 3SO_2(g) \qquad (2.24)$$

$$Cu_5FeS_4(s) + 4O_2(g) = 5/2Cu_2S(l) + FeO(l) + 3/2SO_2(g) \qquad (2.25)$$

The Noranda/Teniente smelting converters treats 90% of its total concentrate as bone dry which is injected into it through dedicated tuyeres pipes while 10% of wet concentrates along with flux and reverts are charged onto the molten bath surface (Davenport 2002, Harris 1999). Tuyere injection of dry concentrates leads to high thermal efficiency and low dust evolution. Submerged blowing of blast causes violent stirring of the matte-slag bath and results in rapid melting and oxidation of the furnace charge (Davenport 2002). It also prevents excessive deposition of solid magnetite in the furnace even under highly oxidizing conditions. The advantages of tuyere injection are:

- Efficient heat transfer as a result of good contact between matte and slag due to turbulence at the tuyeres.
- A small energy requirement due to the absence of H_2O in the bone dry Concentrate
- Little dust carryout, 1% of solid feed.

2.4.2 Operation Process Control

Once steady state operation has been reached, the furnace is controlled with the purpose to achieve the following;

- Smelt concentrates, scrap and other metal-bearing solids at the plant design rates.
- Produce matte and slag of prescribed composition and temperature
- Produce offgas with desired SO_2 concentration
- Maintain constant depths of matte and slag in the furnace.

The operation is controlled to produce matte and slag of specified compositions and temperature, protect the furnace refractories by temperature control and smelting solid feed at a specified or maximum feed rate. The following are the parameters that must be controlled to achieve the targets mentioned above;

1) Slag bath Temperature control

The Bath temperature is monitored continuously with optical pyrometers in two tuyeres (Prevost 1999). The pyrometers are sighted on the slag tapping stream or onto the molten bath itself. Slag temperature of 1220^oC to 1250°C is controlled by adjusting fuel/concentrate ratio and the blast oxygen enrichment level (N_2/O_2 ratio) which is controlled within a range of 30-50% by volume (Davenport 2002 and Alvarado 1995). High slag temperatures are not desired in the furnace as this leads to damage to the furnace bricks and steel shell.

2) Slag bath composition

The Slag chemistry is controlled by adjusting Flux/Concentrate ratio that will maintain the target Fe/SiO_2 ratio of 1.54. This, plus good temperature control gives a slag and a Fe_3O_4 content of 16-24% which maintains a protective (but not excessive) layer of solid magnetite on the furnace refractory (Davenport 2002, Mackey and Alvarado1995).

3) %Cu in product matte (Matte grade)

The desired matte grade controlled by adjusting the Oxygen/Concentrate ratio. This ratio controls the degree of Fe and S oxidation, hence the grade of product matte composition (Davenport 2002, Alvarado 1995, 1999).

4) Matte and slag depth control

Matte and slag depths are measured frequently by inserting a steel rod into the molten bath in the furnace vertically from above. Matte depth is controlled to give of matte above the tuyeres. This ensures efficient use of oxygen injected into the matte layer. The bath height of matte and slag above the tuyeres is also controlled to be as constant as possible. This gives a constant liquid static pressure above the tuyeres, hence a constant flow of blast. The heights are kept constant by adjusting matte and slag tapping frequencies (Mackey 2001).

5) Concentrate feed rate

The concentrate feed rate into the furnace must be high enough that will sustain the autogenous mode of the operation. The concentrate federate is normally increased by increasing oxygen enrichment of the tuyere blast. Oxygen enrichment increases the heat available for smelting as there will be little nitrogen input into the furnace which absorbs and take away a lot of useful heat out of the furnace (Davenport 2002, Mackey 1995).

2.5 TSL Primary bath Smelting process

The Top Submerged Lancing (TSL) bath smelting system was developed over 30 years ago by the Commonwealth Scientific and Industrial Research Organisation (CSIRO) in Australia where it was called High-Temperature Submerged Combustion, then Sirosmelt (Arthur 2003). This technology is marketed by two separate organizations under the names Outotec/Ausmelt and Isasmelt Xstrata technology (XT). Over the years the capacity has increased from 0.25tph to 200tph per unit furnace (Bakker 2011 and Baldock 2000). The reactor is widely used for mainstream base metal smelting and converting. It is also used for treating by-products or industrial wastes in the production of tin, copper, nickel, lead, zinc, gold, silver, PGMs and aluminium. These furnaces and their operating principles are similar and as such they are described together throughout this section.

The TSL furnace is a vertical steel cylinder that is lined with high quality refractory bricks. (Matusewicz 2005, Arthur 2003). Central to the technology is the injection of air, oxygen and fuel directly into the molten slag bath via a vertically suspended lance. The TSL process is extremely efficient because critical process phenomena including feed material dissolution, energy transfer, reaction and primary combustion all take place in the slag layer. The TSL furnace can be operated in two different smelting environments, the low oxygen partial pressure condition (reduction) during smelting and high oxygen partial pressure condition during oxidation known as converting (Matusewicz 2005, Bakker 2011). The injected materials react with metal-bearing raw materials in the slag bath, producing a slag and a semi-finished product, which can be copper matte, high grade Ni-Cu-PGM matte, blister copper or lead bullion. The TSL bath process is simple, flexible and versatile. The same basic furnace, with its detailed design and control systems appropriately optimised, can be used for a variety of different applications. TSL smelting technology is also used for recovering copper from non sulphide materials particularly slags and sludges (Hughes, 2000). Its ability to control air and fuel inputs means that conditions can be changed from oxidizing to reducing without transferring material to a second furnace. This is particularly effective for smelting Cu/Ni hydrometallurgical residues and industrial wastes (Hughes 2000). Commercial scale plants have been built for smelting concentrates of copper, lead, zinc, copper/nickel, waste materials, dusts, metal bearing residues, scrap metal, mattes, drosses, speiss, copper anode slimes, precious metals sweeps, electrical and electronic scrap, shredder materials concentrates, secondary copper materials and for recycling scrap lead (Arthur 2005, Matusewicz 2003). Plant expansions can be accommodated by increasing the fuel and oxygen content of the process air injected down the lance and solid material feeders (Arthur 2005). When a new oxygen plant was constructed at Mount Isa in 1998, the oxygen content of the flow down the TSL lance was increased from approximately 45% to >60%. This resulted in a significant increase in the production capacity of the furnace from 600,000 to 1,100,000tpa (Arthur 2005). The latest innovation with the TSL technology is the continuous copper and Ni-Cu PGM converting technologies. The TSL Smelting and converting processes offers benefits such as (Bakker 2011, Arthur 2005):

- Low investment and operating costs for smelter and acid plant
- High productivity in a small bath volume.
- Continuous process operations, since smelting and converting can be decoupled, resulting in high on-line availability.
- High sulphur recovery with a single continuous high strength SO_2 gas stream to the acid plant.
- Improved in-plant hygiene, the TSL furnace is fully sealed, operated under a slightly negative pressure and ladle transportation of molten materials can be eliminated.
- Flexibility to process wide range of feed materials, including materials high in impurities.
- Simple, compact design ideally suited to retrofit into existing operations.
- Highly automated and quick ramp up to design production rates.

The TSL smelting process has achieved a strong global presence as a reliable, cost effective and environmentally friendly in copper, Ni-Cu PGM and lead smelting technology (Edwards 2007, Matusewicz 2005). In sulphide smelting, the sulphide compounds of the feed partially oxidize and release heat, acting as a fuel for the process with only trimming fuel added to provide for precise bath heat control. Figure 2.3 shows the schematic diagrams of the primary TSL bath smelting furnace.

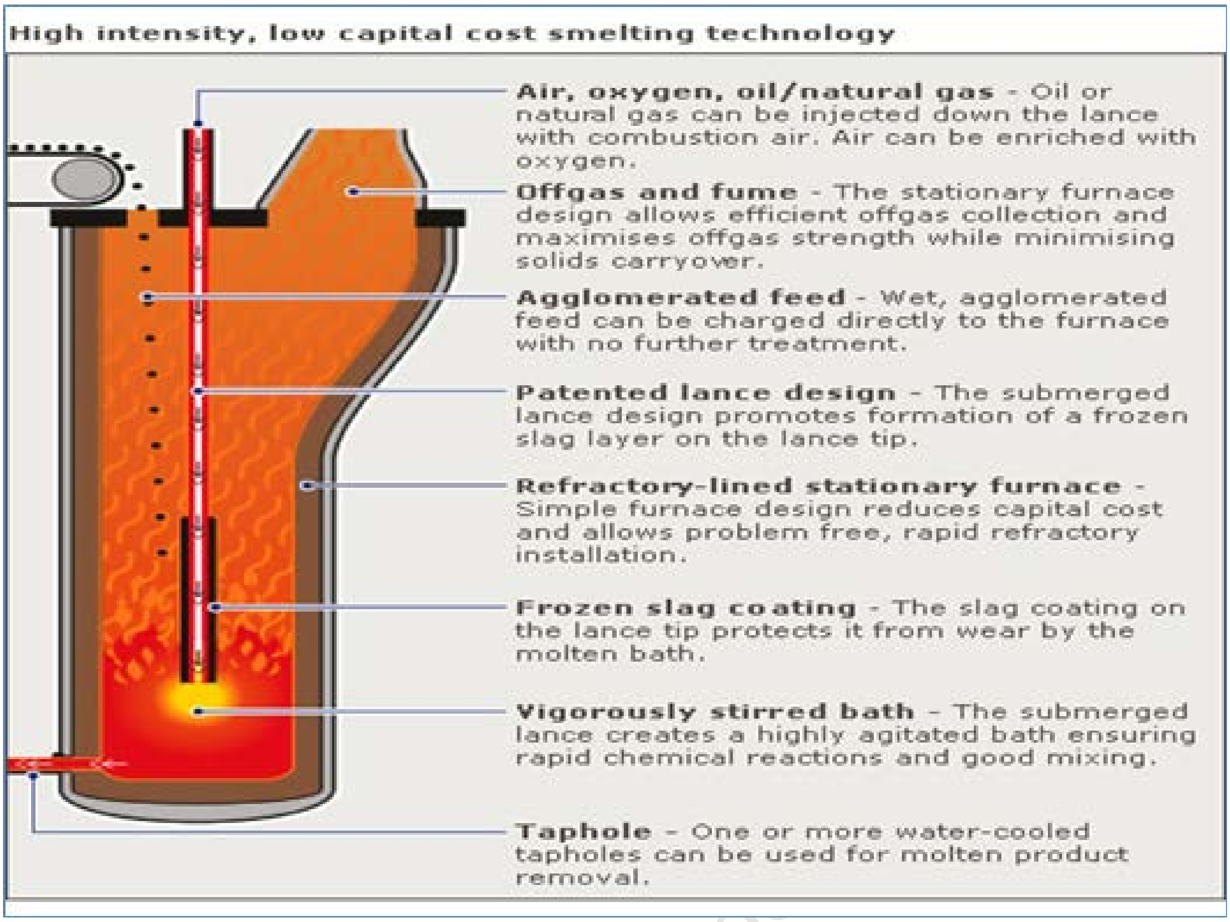


Figure 2.3: TSL smelting furnace diagram (Geraldo 2012)

2.5.1 TSL copper smelting process

The TSL smelting process is a bath smelting stage under which the lance is operated under neutral/reducing conditions (low oxygen partial pressure) to provide a slag bath condition that promote the reduction and decomposition of mineral compounds and partial oxidation of the reaction products. During the smelting process fuel, air and oxygen is injected down the lance into the slag bath. The Isasmelt/Ausmelt furnace is a vertically aligned steel barrel with internal diameter of 4.4-5.0m and 12-17m high lined with 450mm chrome-magnesite bricks or copper coolers on the bottom side (crucible). The bottom chrome-magnesite bricks are backed with copper water cooling blocks. The roof consists of boiler tube panels or steel panels (Matusewicz 2005) and has some openings on the roof for the feed chute, lance insertion, holding burner and uptake for the off gas escape.

The smelting process in the Isasmelt/Ausmelt is started by the preheating the furnace to cure the bricks for a specified heat up time and heating up the furnace to operating temperature, slowly charging and melting solid slag pieces until obtaining a molten bath on 2m height, immersing the smelting lance in the molten slag bath and beginning normal concentrate or residue smelting or matte converting. Smelting is terminated by stopping the solid feed, draining the furnace and turning off the lance (Matusewicz 2005).

The injection of the air and oxygen into the bath creates a high turbulent bath which facilitates rapid smelting reactions and heat transfer (Arthur 2003). This stirring ensures very intimate contact between the feed and the oxygen. Feed materials falling into the bath reacts rapidly, resulting in extremely high productivity for a relatively small bath volume. The lance height is positioned so that its tip is located just under the surface of the molten slag in the furnace. A layer of frozen slag on the outer surface of the lance protects it from the molten bath and this allows the lance to operate submerged in the slag layer for extended periods of days to weeks (Arthur 2003). A swirler inside the lance increases the velocity of the process air and oxygen down the lance thus increasing the heat transfer rate from the steel to the process air (Arthur 2003). The lance can be removed from the furnace easily at any time if maintenance is required and can be started as well in a few minutes. The large 350-450mm diameter lances have proved to be robust (Arthur 2003). Premature failures can occur from matte attack if the lance in inserted too deeply into the matte bath and hence it important to implement the lance positioning system so that the lance immersion in the bath is maintained constant in the slag phase (Matusewicz 2003, 2005).

Steady state operation consists of continuous feeding of solid charge through the roof feed port or the lance and continuous blowing of oxygen-enriched air into the molten bath. The furnace is computer controlled to give a specified concentrate/matte smelting rate while producing matte or metal and slag of desired composition and temperature. The feed rate is adjusted as required by increasing or reducing the fossil fuel supplied through the lance into the molten bath. The compositions are controlled by adjusting the furnace input oxygen and concentrate ratios (Matusewicz 2003).

Feed material is passed through a sealed port on the top of the furnace. The system can accept lump, fine, dry, or wet feed and hence there are minimal feed preparation requirements (Arthur 2005). The capability of moving the lance or adjusting the depth of submergence of the lance in the bath allows the operator to control the degree of turbulence in the bath and the extent of splashing of the slag cascade above the bath. The furnace is easily put on standby at any time for as long as required by removing the lance above the molten bath and allowing fuel to be combusted above the bath (Arthur 2003). The lance can also be used to initially heat the furnace at the start of the smelting or converting process. The matte and slag mixture is tapped periodically into a fuel-fired or electric settling furnace for separation or allowed to settle in the furnace before tapping. The settled matte (60% Cu) is sent to the Peirce-Smith converter while the slag (<0.7% Cu) is discarded (Matusewicz 2003). The offgas with 25% SO_2 concentration is drawn from the top of the smelting furnace through a vertical flue and passed through a waste heat boiler, gas cleaning and on to a sulphuric acid making plant. Most of the energy for smelting comes from oxidizing the part of the concentrate charge. Additional energy is provided by combustion of oil or gas or fine coal blown through the vertical lance and the lump coal premixed with the solid charge. Figure 2.4 below is an illustration of the smelting principle of the TSL copper smelting furnace.

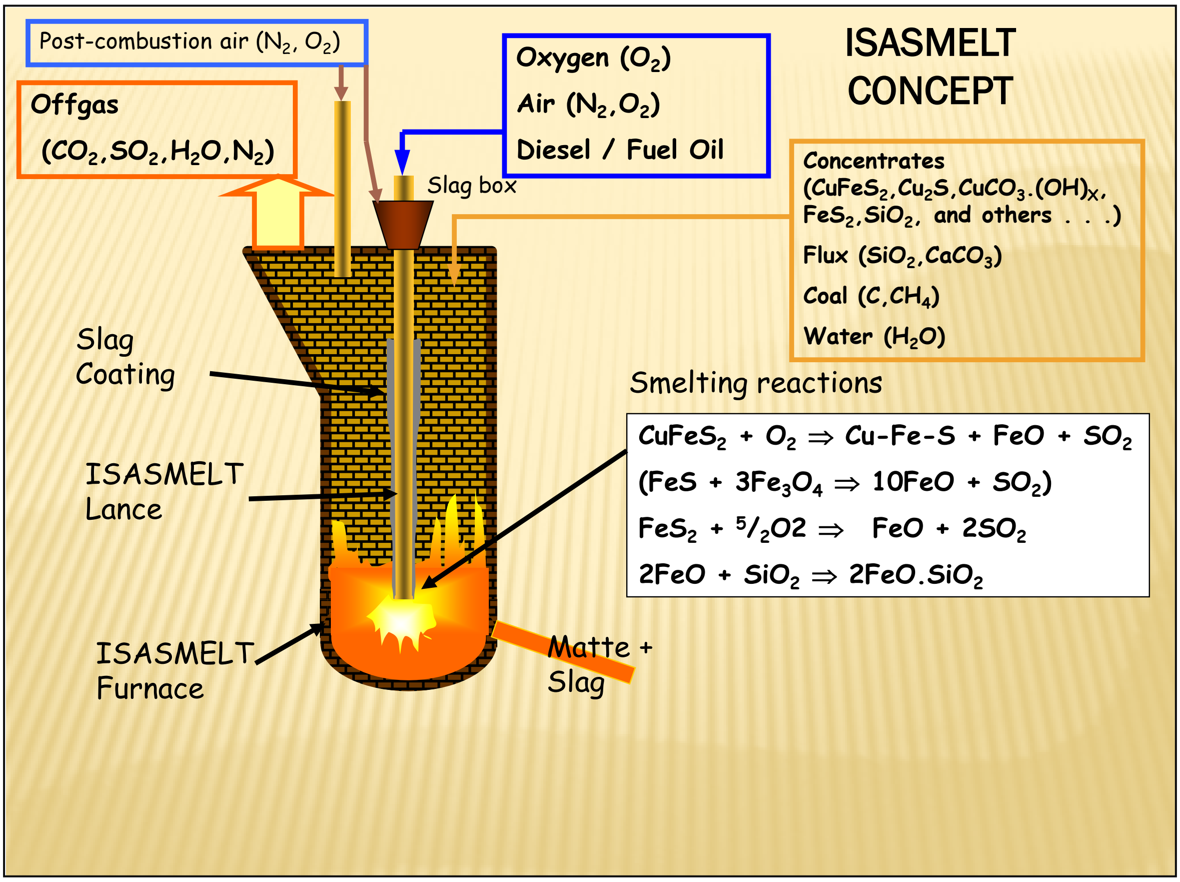


Figure 2.4: TSL copper smelting concept diagram (Ross 2005)

The feed to the TSL Ausmelt/Isasmelt furnaces is composed of moist concentrate, flux and recycle materials and sometimes in pelletized form. Drying of the feed is not necessary because the smelting reactions take place in the matte and slag bath rather than above it. Moist feed also decreases dust evolution. Oxygen enrichment of the air blast into an Ausmelt/Isasmelt furnace bath is standard practice The blast typically contains 50 to 70% by volume and amounts higher than this tend to cause excessive lance wear (Matusewicz 2003). Because of this upper limit on oxygen enrichment and the presence of moisture in the solid feed, auto thermal operation is usually not achieved and to supplement the heat requirement, hydrocarbon fuel is added to keep the slag bath at the desired bath temperature (Edwards 2007). Ausmelt/Isasmelt furnaces are designed to use natural gas or oil or coal. A cool lance tip is important for reducing lance wear. As a result, lump coal is often added to the feed as a partial substitute for flammable fuel oil or natural gas (Matusewicz 2003, Edwards 1998).

2.5.1.1. TSL copper smelting Mechanisms

In the Ausmelt/Isasmelt TSL furnaces, the smelting reactions take place primarily in the molten slag bath at the tip of the lance. It is commonly believed that dissolved magnetite in the slag serves as a catalyst for the overall process (Binegar 1995, Mounsey 1999). The presence of about 5% magnetite in the slag is important (Binegar 1995). This requires a low bath temperature of 1150-1200°C and low Fe/SiO2 ratio of 1.25 for Isasmelt and 1.43 for Ausmelt (Davenport 2002). The probable reactions taking place in the TSL smelting furnace molten bath for copper concentrates are indicated by equations 2.26 to 2.34.

$$C(s) + O_2(g) = CO_2(g) \tag{2.26}$$

$$H_2(g) + 1/2O_2 = H_2O(g) \tag{2.27}$$

$$CuFeS_2(s) = Cu_2S(l) + FeS(l) + 1/2S_2(g) \tag{2.28}$$

$$2Cu_5FeS_4(s) = 5Cu_2S(l) + 2FeS(l) + 1/2S_2(g) \tag{2.29}$$

$$FeS_2(s) = FeS(l) + 1/2S_2(g) \tag{2.30}$$

$$FeS(l) + 3/2O_2(g) = 2FeO(l) + SO_2(g) \tag{2.31}$$

$$3FeO(l) + 1/2O_2(g) = Fe_3O_4(l) \tag{2.32}$$

$$2FeO(l) + SiO_2(s) = 2FeO.SiO_2(l) \tag{2.33}$$

$$3Fe_3O_4(l) + FeS(l) = 10FeO(l) + SO_2(g) \tag{2.34}$$

The following are typical process parameters of the Isasmelt smelting furnace that is used to smelt copper concentrates as from a relevant copper smelter that utilizes the TSL;

Furnace Feeds and Process parameters

- 50-80% O_2 enrichment by 95% oxygen from oxygen plant
- Process air from dedicated blower at 5 Nm^3/s as minimum (design 7 Nm^3/s).
- Minimum lance air ~1.2 Nm^3/s
- 35 L/hr diesel (average during smelting)
- Blend Quality (32%Cu, 22%Fe, 29%S, 7% SiO_2)
- 70-115tph (Design 113tph) at 7-9% H_2O
- 0-6tph Silica flux
- 3tph lump Coal (Design 1- 4.5tph)
- 0 - 25tph Reverts

Products

- Matte temperature 1170-1190°C
- 56-58% Cu in matte
- Fe/SiO_2 – 1.25
- 8% Fe_3O_4 in slag
- <0.7% Cu in slag
- Slag temperature-1250-1300°C
- Offgas flow >50000Nm^3/hr at 25% SO_2.

2.5.1.2 TSL smelting Process Control

The main process control parameters of the Isasmelt/Ausmelt smelting furnace are the following (Matusewicz 2003, Arthur 2005). The TSL copper smelting parameters are very similar to the Noranda/Teniente smelting furnaces and are;

1) **Temperature control**

Bath temperature of 1180°C to 1250°C is controlled by adjusting lance fuel feed rates (fuel/concentrate ratio) and the blast oxygen enrichment (N_2 coolant input rate). The enrichment maintained within a range of 50-70% by volume (Arthur 2003, Matusewicz 2003). High oxygen enrichments are required to minimize the lance fuel.

2) **Slag Quality Control**

The Slag chemistry is controlled by adjusting flux/concentrate ratio that will maintain the desired Fe/SiO_2 target ratio of 1.25-1.45 and keeping the temperature above 1200^{o}C to avoid excessive formation of Fe_3O_4 (Davenport 2002).

3) **%Cu in Matte (Matte grade)**

The desired matte grade controlled by adjusting the Oxygen/Concentrate ratio. This ratio controls the degree of FeS oxidation, and hence the product matte chemical composition (Arthur 2003, Davenport 2002)

4) **Matte and slag depth control**

Matte and slag depths are measured frequently by inserting a steel rod into the molten bath in the furnace vertically from above. This ensures correct immersion of the lance into the slag layer and efficient use of oxygen injected (Matusewicz 2003).

5) **Concentrate Feed rate**

The concentrate feed rate into the furnace must be high enough to generate enough heat to keep the smelting process close to autogenous mode for sulphide concentrate smelting. High smelting rates for sulphide concentrates minimizes external lance fuel combustion to sustain the required bath operating temperature Matusewicz 2005).

2.6 TSL Converting Technology

The TSL converting process is based on the utilization of the feed material’s inherent chemical energy for smelting with the use of oxygen (Errington 1997, Arthur 2003). Peirce-Smith converters are still the dominant furnaces that are utilised for the processing of high iron low grade copper and nickel mattes (Davenport 2002). The TSL converting process offers an alternative converting process to batch converting processes such as the Peirce-Smith converter. It is an attractive option due to its high flexibility to treat various types of solid feeds and high capture of the SO_2 offgas. The TSL continuous converter is an increasing attractive option for smelters due to the requirement to maximise the capture of SO_2 gas, increase converting capacity and reduce the operating costs (Matusewicz 2005). Solid feed combined with the high level of oxygen enrichment in the process air produces a low volume of offgas rich in SO_2 concentration. The submerged lance exhibits all the thermodynamic advantages of a tuyere with none of its engineering disadvantages. Unlike a tuyere, the lance can be raised and lowered automatically or manually so that its tip is always where it is required. The TSL converter capacity expansion can be accommodated by increasing the oxygen content of the process air injected down the lance and increasing the feeder speeds for fuel, matte/concentrate, lump coal, flux and air (Edwards 2007, Matusewicz 2005).

The Ausmelt/Isaconvert Converting processes were originally developed on the basis of a batch approach of the Peirce-Smith converter process but eventually changed into a continuous operation. Although the process can be operated with a continuous molten feed of matte, it is believed the advantages of decoupling the smelting and converting steps by using an intermediate stockpile of dry crushed or granulated matte outweighs the increased energy requirements for remelting the matte. The use of solid matte feed also allows decoupling of the smelting and converting steps, giving added flexibility and simplifying the maintenance and operating aspects of the smelter (Matusewicz, Arthur 2005).

The oxidation of furnace matte proceeds under highly oxidizing conditions (high oxygen partial pressure) so that the low grade sulphide matte is converted to high grade Ni-Cu matte or blister copper. High oxygen enrichment of the lance air is utilised to maintain the autogenous mode of operation with minimum external fuel requirement to maintain the desired bath temperature. The furnace can use different types of fuels either coal or oil or natural gas for balancing the energy requirements (Edwards 1999, Matusewicz 2005). The high oxygen enrichment also helps reduce the offgas volume and increase the SO_2 concentration in the off gas (Davenport 2002, Edwards 1999). The grade of the high grade nickel matte or blister copper is controlled by the oxygen partial pressure in the molten bath using the oxygen/matte feed ratio. The slag chemistry is controlled by the flux/matte feed ratio that satisfies the desired Fe/SiO_2 ratio for the iron silicate slag or $Fe+CaO/SiO_2$ for the ferrous calcium silicate slags (olivine). The bath consists of matte and slag at the product compositions at all times, that is the operating conditions effectively fixes the process at what is for the pierce smith converters the end point of the converting reactions (Floyd 1999). The nickel or copper content of the slag produced in the TSL Converting furnace is relatively high and therefore, the converter slag is treated in a slag cleaning furnace by carbothermic reduction. As mentioned in section 2.5, the dissolved magnetite in the slag is believed to be the main oxygen carrier in the bath as indicated by the following reactions (Binegar 1995, Geraldo 2012).

$$3/2O_2 + 9FeO(l) = 3Fe_3O_4(l)$$

(2.35)

$$2FeS(l) + 3Fe_3O_4(l) = 2FeO(l) + SO_2 (g) \qquad (2.36)$$

$$FeS(l) + 3/2O_2(g) = FeO(l) + SO_2(g)$$

(2.37)

Of the three slag systems potentially available for converting furnace mattes fayalite, calcium ferrite and ferrous calcium silicate, the calcium ferrite system appears to be favoured by others for continuous converting processes (Bakker 2011). The calcium ferrite slag system has been successfully applied in copper continuous converting processes like the Mitsubishi process. The beneficial properties of the calcium ferrite slags for copper converting include the ability of the liquid phase to contain higher ferric iron concentrations at high oxygen potentials, lower slag volumes, lower valuable metal losses and greater fluidity (Yazawa and Takeda 1999, 1981, 2006). Whilst the iron silicate slags have been traditionally used in copper converting, the main concern comes from the effect of high oxygen partial pressures that increase the ferric iron (Fe^{2+}) content in the slag. The limited solubility of ferric iron in pure iron silicate slags leads to magnetite precipitation and increase in viscosity and unless higher temperatures are used, the result can be forming of the slag (Yazawa and Takeda 1999, 2006). A solution to this is the use of the calcium ferrite slags ($CaO\text{-}FeO\text{-}Fe_2O_3$) as used commercially by Mitsubishi smelting and converting process. This is due to two major reasons, the high magnetite solubility limit and the low viscosity (0.03 Pa.s compared to 0.25 Pa.s for silicate slag). However such a low viscosity gives rise to another issue, mainly in the furnace containment and refractory wear as experienced in furnaces operating with this kind of slag. Calcium ferrite slags are only suitable for converting due to the higher affinity for sulphides in the slag (Yazawa and Takeda 1999). In addition, operational difficulties arise in calcium ferrite slags if too much silica is present in the feed. Silica levels of only 5wt% are enough to achieve saturation of dicalcium silicate and subsequent precipitation. If more silica is added, the viscosity of the calcium ferrite slags increases considerably and removal of slag from the furnace becomes difficult (Yazawa and Takeda 2006).

The ferrous calcium silicate slag (olivine) system is also one that has been proposed (Yazawa 1980, 1981). Its properties lies between the calcium ferrite slags and the iron silicate slags. They may avoid most of the problems encountered when using either the iron silicate or the calcium ferrite slags in converting. However, the ferrous calcium silicate slags do present issues with increased slag volume and via this the increase in Ni, Cu and PGM losses to the slag (Matusewicz 2005). Ausmelt from its extensive converter operation has found that the magnetite can be controlled within typical smelter slags (10-20%) for the iron silicate and ferrous calcium silicate type slags without the need to resort to extreme operating temperatures or slag systems which are specially tailored to converting, such as calcium ferrite slags (Matusewicz 2005). The knowledge is reason why continuous converting of low grade furnace matte at ACP is done with the iron silicate slag or ferrous calcium silicate slags (Matusewicz and Sofra 2005). Isaconvert has performed at pilot plant scale the continuous converting of the Ni-Cu PGM matte with the calcium ferrite slag (Bakker 2011). Nickel matte converting using the calcium ferrite slag at commercial scale has been performed by Stillwater mining company using the batch Top Blown Rotary converters (Bakker 2011).

Commercial scale TSL converters have been successfully commissioned at Zhong Tiao Shans Houma Smelter for copper and Anglo American Platinum for the low grade Ni-Cu PGM matte, Umicore in Belgium and Aurubis in German (Bakker 2011) for e-waste and precious metals residues. The advantages of the continuous converting include (Arthur and Matusewicz 2005, Geraldo 2010, 2012);

- Low capital cost compared with multiple batch furnace applications. The saving is due to;
  - The reduced size and number of furnaces
  - Easier hygiene ventilation from the smaller plant
  - Elimination of the expensive converter aisle crane system
  - A smaller size acid plant with continuous flow of converting gas from the TSL converting furnace.

- Very low dust production and generation of a low volume of offgas, rich in SO_2 gives greater environmental control than other available technologies.
- Large scale production of blister copper or high grade Ni-Cu-PGM matte in a single vessel unlike using multiple PSC.
- Blister copper containing 0.03 to 0.3% sulphur can be produced.
- Production of high grade low sulphur blister copper or high grade Ni-Cu-PGM matte.
- No intermittent charging matte, skimming slag, scrap or revert addition by boats, flux addition in boats or pouring blister copper required. (Higher availability >85% compared to 40-60% for Peirce-Smith). Continuous process operation since smelting and converting can be decoupled resulting in high on line availability. The TSL converting process is fed with granulated or crushed furnace matte unlike the Peirce-Smith converters which receive hot molten matte. The converting process takes place in a totally contained vessel, which makes it possible to contain all the gas generated in the process (Bakker 2011, Edwards 2007).

2.6.1 TSL Copper Converting Process

The TSL converting systems produce high grade blister copper containing low sulphur and other impurities from low grade furnace matte. Semi-continuous TSL converters are capable of producing 100,000 tonnes per year of blister copper in a single furnace, whilst the continuous converters are capable of producing 400,000 tonne per year of blister copper (Edwards 2007, Matusewicz 2005).

The semi-continuous TSL converting process uses a two stage process in which FeS in the matte is oxidised during the matte feeding stage to produce white metal which is matte with less than 1% Fe in (stage 1) as indicated by equations 2.38 to 2.41. At the start of the converting cycle, dry granulated matte is fed at a fixed rate while a controlled amount of reaction oxygen/Air is injected through the lance into the molten bath. The oxidation process reduces the content of iron in the feed matte (15-25%) to approximately 1% Fe (white metal). During this stage, silica and sometimes CaO is added to flux the oxidised iron and produce a fluid slag. Once the furnace is full, matte feeding is stopped and the final stage of copper sulphide oxidation begins (Matusewicz, Sofra 2005).

$$FeS(s) + 3/2O_2(g) = FeO(l) + SO_2(g) \qquad (2.38)$$

$3FeO(l) + 1/2O_2(g) = Fe_3O_4(l)$ (2.39)

$FeS(l) + 3Fe_3O_4(l) = 10FeO(l) + SO_2(g)$ (2.40)

$2FeO(l) + SiO_2(s) = 2FeO.SiO_2\ (l)$ (2.41)

The copper sulphide arising from the first stage blow (slag blow) is oxidised further to produce blister copper suitable for anode casting as represented by equations 2.42 to 2.44 below.

$Cu_2S(l) + O_2(g) = 2Cu(l) + SO_2(g)$ (2.42)

$2Cu(l) + 1/2O_2(g) = Cu_2O(l)$ (2.43)

$2Cu_2O(l) + Cu_2S(l) = 6Cu(l) + SO_2(g)$ (2.44)

The TSL copper continuous converting process produces blister copper in the single stage directly from matte, removing the need to produce intermediate white metal in between the processes. This improvement significantly increases the blister copper productivity in the single TSL converting furnace up to 400,000 tonnes per year (Edwards 2007, Matusewicz 2005). The concept of continuous was evidenced during operation of the batch converting process on the occasion when the first white metal stage was accidentally overblown resulting in a separate copper metal phase forming in the furnace (Edwards 1999, 2007). The concept has been applied in some commercial operations where a controlled degree of over blowing has been applied during periods of converting solid matte feed. The development of the TSL copper continuous converting process started with the concept of eliminating the production limiting stage of the semi continuous which is the dead time arising from the final blow to copper, thereby making more time available for feeding matte to the system (Edwards 2009, Matusewicz 2005). The simplified reactions taking place during the continuous copper converting process are indicated by the equations 2.45 to 2.47. The grade of blister copper produced is controlled by monitoring the Cu content in the slag (Matusewicz, Sofra 2005)

$2Cu_2S.FeS(l) + 5O_2(g) + SiO_2(s) = 4Cu(l) + 2FeO.SiO_2\ (l) + 4SO_2(g)$ (2.45)

$2Cu(l) + 1/2O_2(g) = Cu_2O(l)$ 10-15% of Cu
(2.46)

$$FeO(s) + 1/2O_2(g) = Fe_3O_4(l) \qquad \text{10-15\% of FeO} \qquad (2.47)$$

2.6.2 TSL Ni-Cu-PGM converting Process

The feed to the PGM smelter typically consists of low grade Ni-Cu sulphide containing high percentages of refractory oxide materials. The product from smelting PGM smelter feeds is generally a high iron containing low grade matte which is further processed almost exclusively using multiple units of Peirce-Smith converters to produce low iron containing matte often referred to as bessemer matte (Jones 1999). The exception is Anglo Platinum mine in South Africa which utilises the TSL Ausmelt converting process (Jacobs 2006). The six Peirce Smith converters were replaced with the two TSL Ausmelt converters with the primary purpose of minimising the pollution of SO_2 and increasing productivity (Jacobs 2006, Nelson 2006). Figure 2.5 illustrates the difference in the converting mechanisms between the TSL converter and the pierce smith converter. The converting reactions in the TSL takes place in the molten slag bath while in the Peirce-Smith converter reactions take place in the molten matte phase (Davenport 2002).

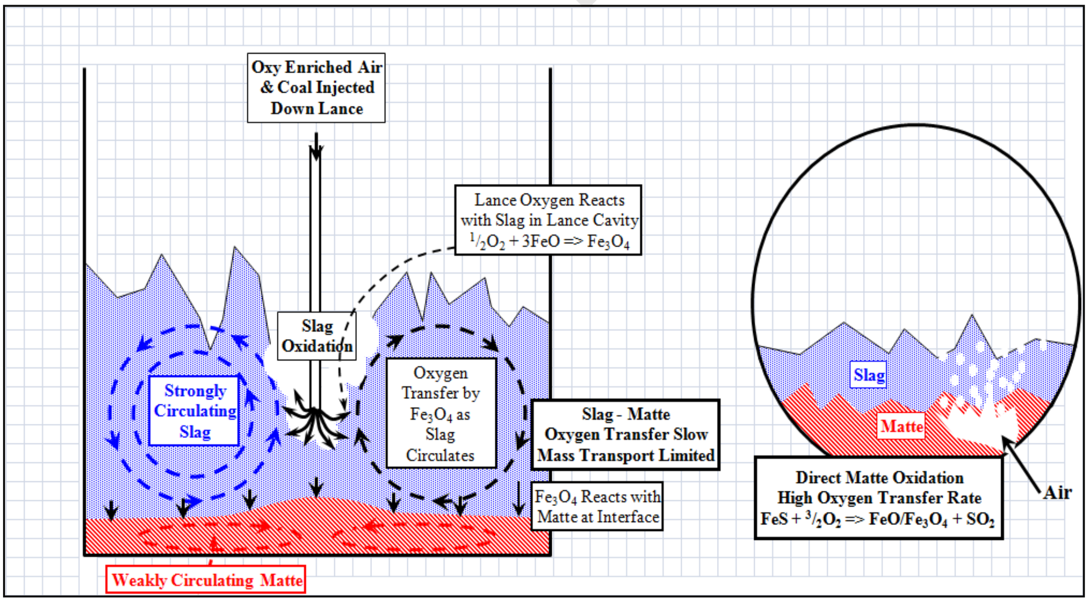


Figure 2.5: Diagram of the TSL and PSC oxidation mechanisms (Matthew 2009).

The original flowsheet for one TSL PGM matte converter involved the two stage batch converting steps to produce bessemer matte. The first stage involved feeding furnace matte and lowering the iron content from 40% to 13% by oxidation until the required volume of 13% Fe containing matte is obtained. The second stage involved reducing the iron from

13% to 3% during which only air, oxygen, coal and flux were fed into the bath (Bakker 2011, Jacobs 2006). Due to operational difficulties experienced in process control in the second stage of converting and explosive forming of the bath at low iron levels and with the need to maintain a constant high strength SO_2 gas stream to the Acid plant, the two stage converting method was modified to the continuous mode. Granulated matte is continuously fed into the furnace and producing converter matte with 3% Fe instantaneously at any given moment (Bakker 2011, Jacobs 2006). Lance air, oxygen, granulated matte and fuel coal are fed into the bath through the lance while silica flux reverts and lump coal is fed through the roof (Matthew 2009). The ACP lance has been designed to allow the injection of matte through the lance into the molten bath (Matthew 2009). The swirlers in the lance increases the velocity of lance air which accelerate the heat transfer from the outer metal to the air resulting in the formation of the slag freezing that increases the lance life (Matusewicz 2005). Figures 2.6 and 2.7 indicate the lance swirlers mounted inside of the inner pipe and the lance assembly sourced from the relevant platinum smelter.

Figure 2.6: Swirlers configuration on lance at a TSL base metal smelter (Matthew 2009)

Figure 2.7: TSL converter lance assembly at a TSL base metal smelter (Matthew 2009)

Using information obtained from one platinum smelter, the TSL PGM converter continuously processes a blend of furnace mattes produced from different electric furnaces to reduce the iron content from an average of 40% down to 3% to produce the PGM containing iron deficient matte that is suitable for treatment at the base metal and the PGM refineries (Jones 1999). The converter furnace feed input ratio of oxygen/matte feed is set to produce bessemer matte and slag that are at product compositions at any time. During the converting process, granulated dry matte, silica flux, fuel coal, lump coal, reverts, air and oxygen are fed continuously into the furnace bath to produce Bessemer matte, ferrous silicate slag (fayalite slag) and SO_2 rich off gas. Xstrata technology has demonstrated the continuous converting process of Ni-Cu-PGM furnace matte in a pilot TSL furnace using the calcium ferrites slag system to produce the bessemer matte with 3% Fe (Bakker 2011). The molten product converter matte and slag are tapped periodically from different tap holes. The off gas from the furnace is cooled, cleaned and dried and converted into H_2SO_4 acid in the contact plant. The bath temperature is controlled between 1300 to 1350^{o}C with most of the energy coming from the iron oxidation reaction and the rest coming from lance fuel coal combustion. Roof coal is used to keep the free board temperature between 1300-1350^{o}C to melt accretions in the uptake section. Converter matte is tapped, slow cooled, crushed and milled before shipping to the BMR for further treatment. Converter slag is granulated, dewatered and dried to <0.5% moisture and further treated in the slag cleaning furnace to recover entrained PGMs in the form of matte (Jacobs 2006).

Operation of the TSL matte converting process is started by heating the furnace to its operating temperature with oil or coal burners. At some smelters the lance is used as a burner for heating up for example at the ACP converters at Anglo platinum (Matthew 2009). Once the operating temperature is achieved, the furnace is fed with converter slag which is melted until the molten bath is above 1000mm after which the lance is inserted 300 - 500mm in the bath and then matte, air, oxygen, fossil fuel and flux feeding into the molten bath is started (Matusewicz 2005). Shutdown consists of turning off of the matte, coal and flux feeders and taking the lance out of the bath and finally stopping the fuel, lance air and oxygen after tapping out the furnace.

2.6.2.1 TSL Ni-Cu-PGM Converter design

The TSL converter at one of the PGM base metals smelters is composed of a crucible made out of a ring of copper coolers (waffle coolers) of 4.0m height and the upper and roof section consist of the high pressure tube wall membrane coolers (Rutanen 2001, Nelson 2006). Due to the high heat fluxes associated with intense matte converting process, the crucible is made up of copper waffle coolers to necessitate the high rate of heat removal by the low pressure cooling water. The inner diameter of the furnace is 4.4m and 17.5m high. There are two rows of waffle coolers with sixteen coolers per row of cast copper coolers with cast-in monel pipes for cooling water flow. The lower part of the copper waffles is lined with ten courses (1200mm) of chrome-magnesite bricks. The water-cooled copper waffle coolers are designed to withstand high heat intensities (Nelson 2006). The waffle coolers have been designed to limit the erosion of refractory within the converter and retain accretion build-up to provide a protective layer in front of the copper coolers (Nelson 2006). A protective layer called a freeze lining is necessary to avoid the copper cooler from directly contacting matte. The cooling system has been designed to maintain a minimum of 10mm sidewall freeze lining thickness. This is achieved by intensive cooling the molten matte and slag at the interface with side wall so that they freeze against the refractory and form a protective barrier between the molten bath and the bricks (Nelson 2006).

Using information sourced from the relevant platinum smelter, the matte and slag tap holes are 75mm in diameter and are located 38mm and 840mm above hearth surface. The maximum allowable molten bath level in the vessel is 2.5m above the hearth. There are 16 transition coolers which are 225mm high, situated at the top of the upper row waffle coolers. They are designed similar to the waffle coolers and also able to handle high heat intensities (Nelson 2006, Jacobs 2006). Transition coolers extend from the top of the upper row waffle coolers to the base of the upper high pressure coolers. Figures 2.8 and 2.9 shows the converter side wall copper waffle configuration and the slag tap hole position from TSL base metal converter obtained from one platinum smelter.

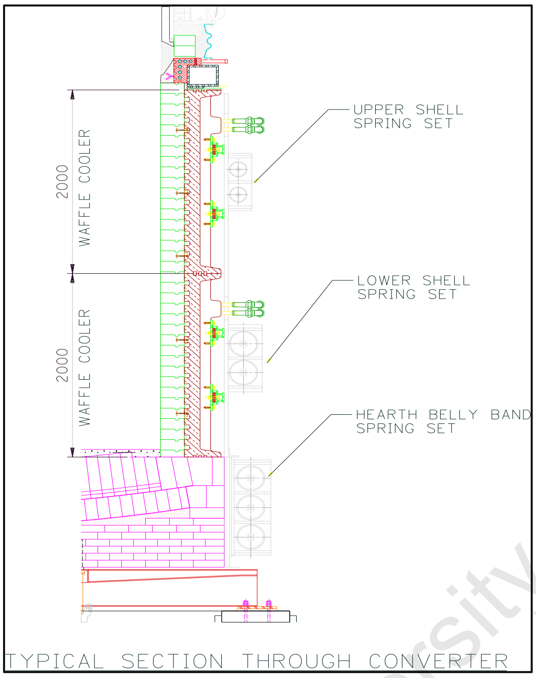


Figure 2.8: Slag tap hole position

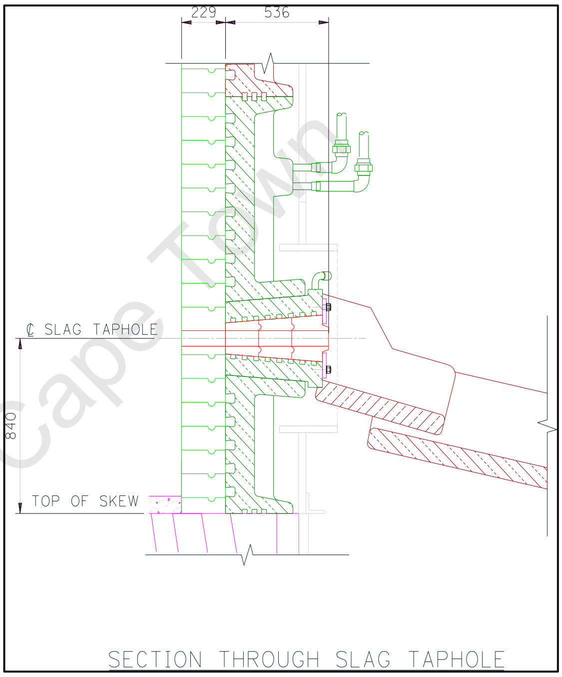


Figure 2.9: Waffle configuration (Mathew 2009)

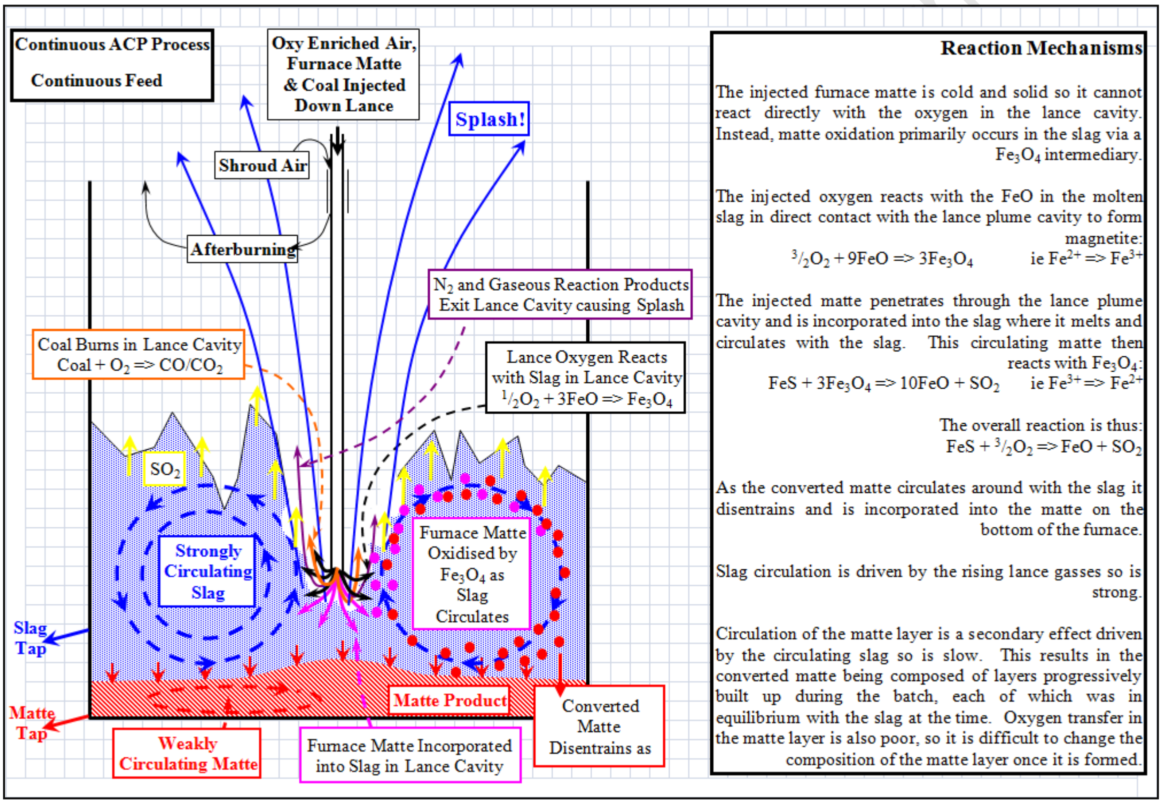
The converter low pressure cooling water system is a semi-closed loop design that provides the intensive cooling of the copper crucible waffles. From the converter, water is pumped through the cooling towers to the supply headers that distribute water to all cooling elements. The increase in water temperature exiting the copper coolers ranges between 2-5°C (35-40°C) and is cooled down in the forced draught cooling tower. The design low pressure cooling water flow rate is between 9000-12000 L/minute at a pressure of 600kPa (Rutanen 2001).

The high pressure cooling system cools the offgas exiting the molten bath in the copper crucible during the process from 1350°C to less than 900°C. The cooling water flowing in the high pressure cooling system is pressured at pressures of 40-70 bars to keep it liquid between

220-260^{o}C. The temperature increase of the water exiting the cooling system averages 10^{o}C (240-250^{o}C) which is cooled down by the fin fan heat exchangers. The design water flow rate in the high pressure cooling system is 1200 -1600m^{3}/hr (Rutanen 2001).

2.6.2.2 TSL Ni-Cu PGM converting process chemistry

Same as in the TSL smelting process, the converting reactions take place in the molten slag bath at the tip of the lance. The dissolved magnetite acts as the carrier of oxygen in the molten slag bath for the overall converting process (Geraldo 2010, 2012). Figure 2.10 illustrates the oxygen transfer concept during the matte converting process.



Reaction Mechanisms

The injected furnace matte is cold and solid so it cannot react directly with the oxygen in the lance cavity. Instead, matte oxidation primarily occurs in the slag via a Fe_3O_4 intermediary.

The injected oxygen reacts with the FeO in the molten slag in direct contact with the lance plume cavity to form magnetite:

$^{3}/_{2}O_2 + 9FeO \Rightarrow 3Fe_3O_4$ ie $Fe^{2+} \Rightarrow Fe^{3+}$

The injected matte penetrates through the lance plume cavity and is incorporated into the slag where it melts and circulates with the slag. This circulating matte then reacts with Fe_3O_4:

$FeS + 3Fe_3O_4 \Rightarrow 10FeO + SO_2$ ie $Fe^{3+} \Rightarrow Fe^{2+}$

The overall reaction is thus:

$FeS + {}^{3}/_{2}O_2 \Rightarrow FeO + SO_2$

As the converted matte circulates around with the slag it disentrains and is incorporated into the matte on the bottom of the furnace.

Slag circulation is driven by the rising lance gasses so is strong.

Circulation of the matte layer is a secondary effect driven by the circulating slag so is slow. This results in the converted matte being composed of layers progressively built up during the batch, each of which was in equilibrium with the slag at the time. Oxygen transfer in the matte layer is also poor, so it is difficult to change the composition of the matte layer once it is formed.

Figure 2.10: TSL converter oxygen transfer mechanism diagram (Matthew 2009).

The simplified chemical reactions that take place in the TSL converter when treating the PGM containing Ni-Cu matte are illustrated by equations 2.48 – 2.55 indicated.

$FeS(s) + 3/2O_2(g) = FeO(l) + SO_2(g)$ 98% of FeS to FeO (2.48)

$2FeO(l) + SiO_2(s) = 2FeO.SiO_2\ (l)$ 85% of FeO to $2FeO.SiO_2$ (2.49)

$6FeO(l) + O_2(g) = 2Fe_3O_4(l)$ 15% of FeO to Fe_3O_4 (2.50)

$$Ni_3S_2(s) + 7/2O_2(g) = 3NiO(l) + 2SO_2(g) \qquad \text{20\% of } Ni_3S_2 \text{ to NiO} \tag{2.51}$$

$$Cu_2S(s) + 3/2O_2(g) = Cu_2O(l) + SO_2(g) \qquad \text{15\% of } Cu_2S \text{ to Cu2O} \tag{2.52}$$

$$CoS(s) + 3/2O_2(g) = CoO(l) + SO_2(g) \qquad \text{76\% of CoS to CoO} \tag{2.53}$$

$$C(s) + O_2(g) = CO_2(g) \qquad \text{complete fuel combustion} \tag{2.54}$$

$$H_2(g) + 1/2O_2(g) = H_2O(g) \qquad \text{complete fuel combustion} \tag{2.55}$$

The converting concept of Ni-Cu PGM furnace matte in the TSL converter is illustrated in figure 2.11 as per the information obtained from one platinum smelter.

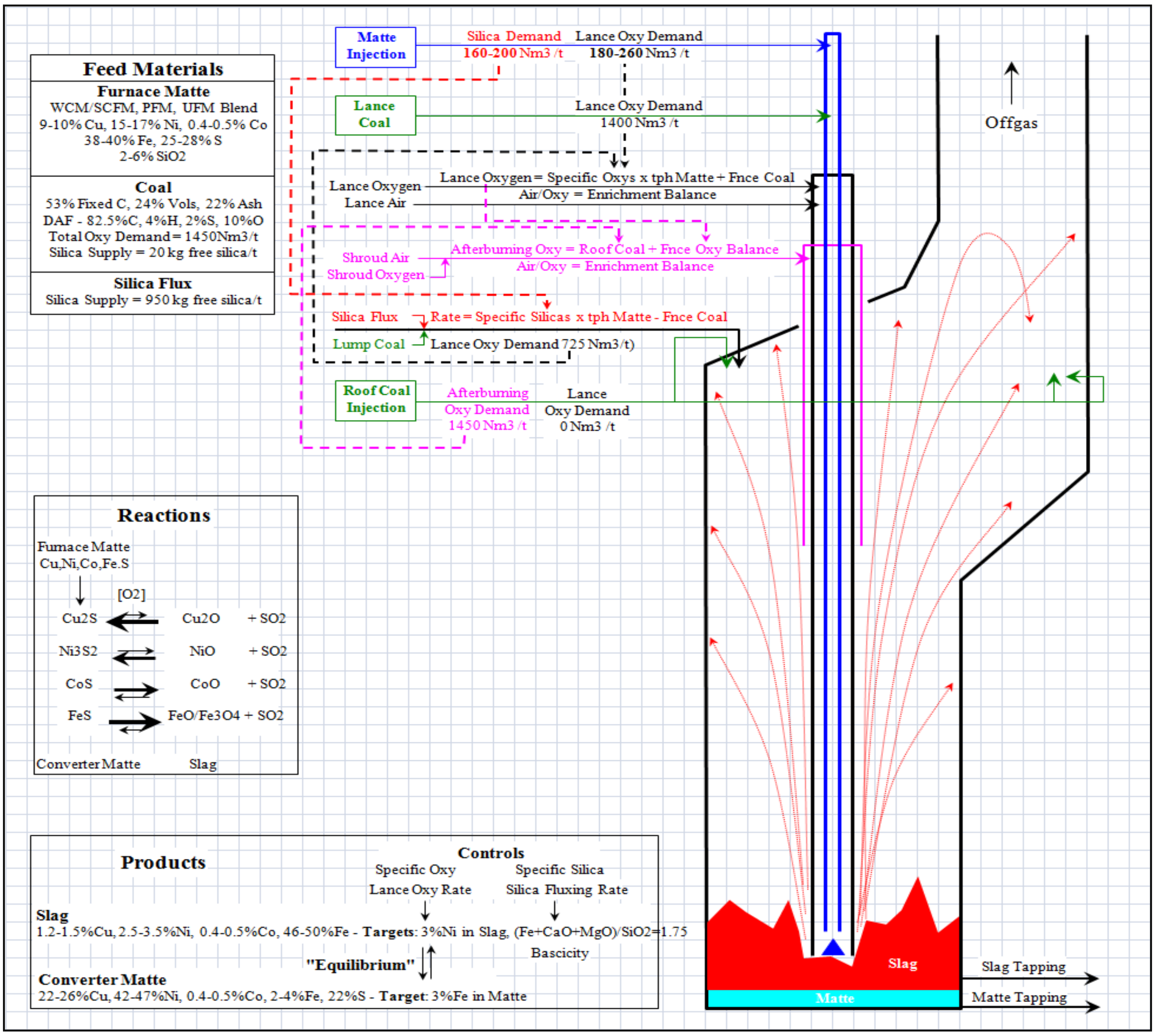


Figure 2.11: Ni-Cu-PGM matte converting process diagram (Matthew 2009).

The typical process parameters obtained from one of the platinum smelters where the TSL furnace is used for converting PGM furnace matte are indicted below.

Process feeds and parameters

- 30-60tph furnace matte (Design 60tph) at <0.5% H_2O
- 30-50% O_2 enrichment with 99% oxygen from oxygen plant
- Maximum supply Process Air(Lance and Shroud) air ~ 30000Nm^3/hr
- 5tph maximum lance fuel coal
- 3tph roof coal as maximum
- 3tph lump Coal (4.5tph as design)
- Matte blend Quality (15% Ni, 8%Cu, 38%Fe, 26%S, 5% SiO_2)
- 0-20tph Silica flux
- 0 - 25tph Reverts

Products

- Converter Matte temperature 1250-1300^{o}C
- Converter slag temperature 1300 – 1350^{o}C
- 2.5-4.5% Fe in converter bessemer matte
- Fe/SiO_2 – 1.70-1.85
- 3.0 - 3.5%Ni in slag
- <15% Fe_3O_4 in slag
- SO_2 in offgas >7%

2.6.2.3 Operational Process Control

The steady state converting process of the PGM furnace matte in the TSL converter is one that is feeding granulated matte into the molten slag bath and producing the fluid converter matte (bessemer matte) with desired Fe content of 2.5-4.5% and fluid slag with the Fe/SiO_2 ratio of 1.70-1.85 (Matthew 2009). From a shutdown, the furnace is started by pre-heating the bricks for five days and on the sixth day when the furnace is hot enough, slag is charged and melted until a molten slag bath of 1000mm is obtained. The lance is submerged 200-500mm in the molten slag bath and then lance air, oxygen, fuel coal or diesel, matte and flux feeding into the furnace is started. In operation, the lance is operated with a submergence of about 200 to 500mm in a molten bath height of 1200 to 2500mm to minimize the splashing of the bath (Matusewicz 2005). The solids build up at the hearth bottom is desirable in the range

200-500mm for protection of the hearth. The lance is 22.46m long and its diameter is 350mm at the tip. The lance movement is computer controlled and the movement up or down is dependent on the charge/matte feed rate and matte/slag flow rate when tapping with the movement ranging from 10-15mm upwards per tonne charge fed or downwards per tonne matte/slag tapped. The main process control parameters for TSL PGM matte converter furnace are indicate below and is based on information sources from one platinum smelter;

1) Temperature control

The slag bath temperature is maintained at the required temperature of 1300-1350^{o}C by the lance fuel coal/diesel feed rate and the level of the oxygen enrichment depending on the matte feed rate. At a higher oxygen enrichment >40%, the converting process attains the autogenous mode with no external fuel needed to be added to the bath. The matte temperature is cooler than slag in the range of 1200-1300^{o}C.

2) Slag Quality Control

The Slag chemistry is controlled by adjusting the Flux/matte feed ratio which is used to maintain the target Fe/SiO_2 ratio of 1.70-1.85 and keeping the slag bath temperature above 1250^{o}C to keep the formation of Fe_3O_4 at the minimum. The target flux demand lies between 150- 240kg per tonne matte feed depending of the quality of the silica flux which is desired to contain >90% SiO_2.

3) Converter Matte grade (%Fe in product matte)

The desired converter matte quality is controlled by adjusting the specific oxygen/matte ratio. This ratio controls the degree of FeS oxidation and hence the product matte composition. Since the grade of matte cannot be checked until the matte tap, the Ni content in the slag as NiO is used to estimate the composition of Fe in product matte at any time during the converting process using the slag bath assays. The equilibrium graph for Ni(slag) Vrs Fe(matte) indicated in figure 2.12 has been developed from real plant process information sourced from a relevant platinum smelter. It is used to control the grade of Fe in product matte. Slag bath samples taken using a dip bar for every 15-20 tonnes of matte fed are analysed for Ni, Cu, S, Fe and SiO_2. The corrected Ni analysis in the slag bath is used to adjust the specific oxygen set point in the programmable logic controller (PLC) control system. The specific oxygen demand lies between 180-250Nm^3/tonne feed furnace matte for the PGM furnace mattes of quality indicated in table 4.2.

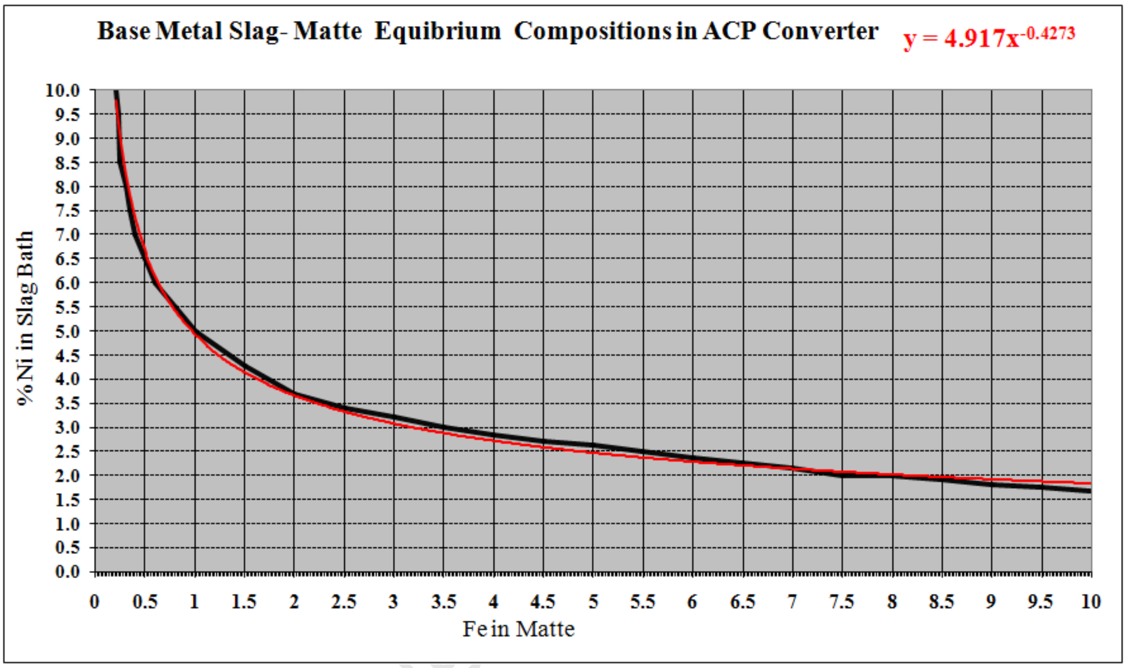


Figure 2.12: Ni-Cu-PGM matte converting equilibrium graph.

4) Fuel Combustion

External fuel is injected into the bath to balance up the heat at low oxygen enrichments and low matte federates. The efficiency of fuel combustion is controlled by the Oxygen/Fuel demand ratio which is about 1475Nm^3 of oxygen per tonne lance fuel coal for the coal of quality indicated in table 4.2. During the bath reduction mode the specific oxygen demand is much less than the stoichiometric requirement at 700Nm^3/tonne lance fuel coal.

5) **Furnace slag bath control**

Matte and slag depths are measured hourly by inserting a steel rod into the molten bath in the furnace vertically from above. This ensures the correct lance immersion in the slag bath and minimise splashing of the molten bath. The molten bath is maintained between 1200 - 2500m.

6) **Concentrate, Reverts, Flux and Oxygen Qualities**

The quality of fuel coal, matte composition and silica flux in term of composition plays a big influence on the performance of the process. A considerable change in the quality with any of these can lead to a major process upset.

2.7 Peirce-Smith Converters

The side-blown Peirce-Smith converter is by far the most dominant converter in copper and nickel sulphide smelting. Peirce-Smith converting technology has been in place since 1910 and is still the mainstay of the copper and nickel-copper PGM smelting industry after more than 100years (Davenport 2002). Many mechanical and metallurgical improvements have been made but the basic design remains the same. The vast majority of the world's copper smelting capacity today, more than 90% involves the use of Peirce Smith converters to transform copper matte into blister copper and furnace matte in PGM smelting into high grade iron deficient matte Ni-Cu matte (Davenport 2002, Eltringham 1993, Jones 1999). The batch-wise nature of PSC, which is sandwiched between a continuous or semi-continuous smelting process and a batch fire refining process, poses a significant challenge to the scheduling of matte, slag, and blister copper transfers to and from the converting aisle (Davenport 2002).

The main raw material for converting is molten Cu-Fe-S or Cu-Ni-Fe-S matte from the primary smelting furnaces. Other raw materials include silica flux, air and industrial oxygen. Several copper bearing materials are recycled to the converter mainly solidified Cu or Ni-Cu bearing reverts and copper scrap in copper converting. The products of converting are;

- Molten blister copper which is sent to anode furnaces for fire- refining or high grade PGM Ni-Cu-S matte which is further treated by hydrometallurgical processes.
- Molten iron-silicate slag which is sent for Cu recovery in a primary smelting furnace.
- SO_2 bearing offgas which is cooled and clean and converted into H_2SO_4 acid.

The bath temperature is sustained entirely by FeS oxidation as the process in highly exothermic. The copper converting process in the Peirce Smith converter is conducted in two stages namely the slag blow and the copper blow (Eltringham 1979, 1993). In PGM Ni-Cu matte converting, only the slag blow stage is conducted as the product is the high grade Ni-Cu converter matte with grade of >75% Ni+Cu in matte (Jones 1999, Wicks 2000).

Industrial Peirce-Smith converters are typically 4m in diameter by 11m long. They consist of a 5cm steel shell lined with 0.5m of $MgO-Cr_2O_3$ refractory brick. Converters of these dimensions treat 300-700 tonnes of matte per day to produce 200-600 tonnes of copper per day. A smelter has two to five converters depending on its overall smelting capacity. Oxygen-enriched air or air is blown into a converter at 600-750 Nm^3/minute at 120KPa gauge pressure (Davenport 2002, Eltringham 1993). It is blown through a single line of 5cm diameter tuyeres which are 40 to 60 in total per converter. Air enters the matte 50cm to 100cm below its surface, nearly horizontal (Lehner 1993). The flowrate per tuyere is about 12-13Nm^3/minute at a velocity of 80 to 120m/s. Blowing rates above 17Nm^3/minute/tuyere cause slopping of matte and slag from the converter and high blowing rates without slopping are favoured by deep tuyere submergence in the matte (Richards, 1986).

About half of the world’s Peirce-Smith converters enrich their air blast with industrial oxygen up to 29% by volume in the blast (Davenport 2002). Peirce-Smith tuyeres are carbon steel or stainless steel pipes embedded in the converter refractory. They are joined to a distribution bustle pipe which is affixed the length of the converter and connected through a rotatable seal to a blast supply flue. Industrial oxygen is added to the supply flue just before it connects to the converter. Steady flow of blast requires periodic clearing (punching) of the tuyeres to remove matte accretions which build up at their tips especially during the slag blow (Bustos 1984). Punching is done by ramming a steel bar completely through the tuyere. It is usually done with a gaspe mobile carriage puncher which runs on rails behind the converter. The puncher is sometimes automatically positioned and operated (Dutton, Simms and Fukushima 1988). Peirce-Smith converter offgas is collected by a steel hood which is usually water cooled which fits as tightly as possible over the converter mouth (Sharma 1979, Pascal 1999).

The gas then passes through a waste heat boiler or water-spray cooler, electrostatic precipitators and a sulphuric acid plant. Peirce-Smith converter off gases contains 7-10% SO_2 by volume during the slag blow. A schematic diagram in figure 2.13 is an illustration of the PSC.

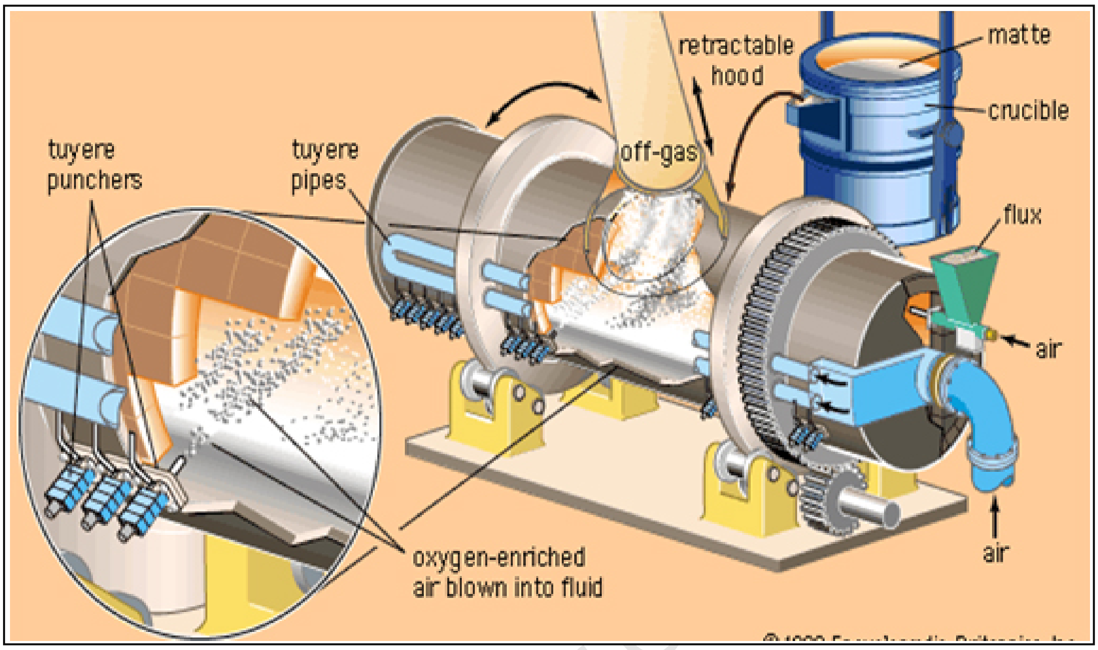


Figure 2.13: Peirce Smith converter diagram (Davenport 2002)

Traditionally converting in the pierce smith converter is a batch process which has several disadvantages such as:

- Intermittent gas flows of variable SO_2 strength and gas flow rate
- Ingress of air around the off-gas collection hood which dilutes the gas
- Ladle transfers and repetitive handling of matte and slag with fugitive emissions
- Thermal cycling and bath composition cycling which reduces refractory life

Converting in the PSC is an oxidation process of molten Cu-Fe-S matte to form molten blister copper (99% Cu) or high grade matte in PGM Ni-Cu-Fe-S matte and fayalite slag. It entails oxidizing Fe and S from the matte with oxygen enriched air or air blast. In most Cu-Ni sulphide smelters the primary smelting process produces a low to medium grade matte. The converter carries out the bulk of the oxidation where Fe and S and other impurities originally in the matte are eliminated. Oxygen enrichment may be used to increase converting capacity, conserve heat and to increase the SO_2 concentration of the offgas. The diminished amount of

N_2 coolant in the blast permits generation of high temperatures even with high Cu grade (low FeS fuel Mattes). The high temperature generated by the high oxygen enrichment of the blast allow the melting of valuable coolants such as Cu or Ni bearing reverts, copper scrap and gold bearing flux (Davenport 2002).

The only disadvantage of high oxygen blast is that it gives high reaction temperatures at the tuyere tips. This leads to rapid refractory erosion in the tuyere area. This erosion is discouraged by blowing at a high velocity which promotes tubular accretion formation and pushes the reaction zone away from the tuyere tip (Bustos 1984).

Flux is added through chutes above the converter mouth or via a high pressure air gun (Garr Gun) at one end of the converter or by using boats that are hoisted by the crane. It is added at a rate that matches the rate of Fe oxidation (usually after an initial delay of 15 minutes at the start of blowing while the converter bath heats up (Davenport 2002). The most reactive fluxes are those with a high percentage of quartz rather than tridymite or feldspar (Casley 1976 and Schonewille 1993). The flux is commonly crushed to 1-5 cm diameter. Sand (0.1 cm) is used in some smelters. Rapid reaction between oxygen, matte and flux to form liquid slag is encouraged by:

- High operating temperature
- Steady input of small and evenly sized flux (Schonewille 1993)
- Deep tuyere placement in the matte
- The vigorous mixing provided by the Pierce-Smith converter
- Reactive flux.

The present upper practical limit of oxygen-enrichment is known to be about 29% oxygen enrichment by volume and this is because above this level, refractory erosion becomes excessive. Tubular accretions do not form in front of the tuyeres above 29% oxygen enrichment. Sonic high-pressure blowing is expected to permit higher oxygen levels, (Davenport 2002).

High grade matte contains little FeS so that it requires less oxygen and time to convert. Rapid blowing of blast, a high oxygen enrichment in blast and a high utilization efficiency lead to rapid oxidation. High oxygen utilization efficiency is obtained by ensuring that the tuyeres are submerged as deeply as possible in the matte. This gives maximum oxygen-in-matte residence time (Davenport 2002).

Another important service of the Peirce-Smith converter is melting of valuable solids with the heat from the converting reactions. The most usual solids are copper or nickel bearing revert materials, scrap copper and gold and silver containing flux (Davenport 2002*)*. Copper concentrates are also smelted in several smelters. Melting of solids is maximized by:

- Maximizing blast oxygen enrichment level.
- Blowing the converter at a rapid rate with the tuyeres deep in the matte.

The solids are added steadily to avoid excessive cooling of the converter liquids. This is easily done with flux and reverts which can be crushed and added at controlled rates from storage bins above the converter. Scrap copper, on the other hand, is often large and uneven in shape. It is usually added in batches by crane with the converter in charging position. This has the disadvantages that blowing must be stopped during scrap addition and the large batch of scrap may excessively cool the molten converter. Several converters have conveyor systems which feed large pieces of copper (scrap anodes and purchased blister copper) at a steady rate during blowing (Fukushima 1988 and Maruyama 1998). This avoids excessive cooling and maximizes the converter scrap melting capability. Up to 30% of a converter's blister copper product comes from copper scrap (Fukushima 1988 and Pannell 1987).

Peirce Smith Converters produce 20000-50000 tonnes of blister copper before they are taken out of service for tuyere-refractory replacement. The replacement takes about two weeks and it is done many times before the converter must be completely relined. Several Chilean smelters (Davenport 2002) remove and replace segments of the tuyere line refractories from the outside of the converter (Mackey 1995, Torres 1993). This lowers converter off-line time to several days but it may weaken the converter shell.

Industrial evidence suggests that oxygen-enrichment up to 25% enhances converter productivity without shortening campaign life. This is especially true if converter blowing rates are high (Verney 1987). Enrichment above this level should be tracked to determine the optimum from the points of view of converter productivity and campaign life.

The copper converting process in the pierce smith converter is conducted in two stages namely the slag blow and the copper blow. In PGM Ni-Cu matte converting, only the slag blow stage is conducted as the product is the high grade Ni-Cu converter matte with grade of >75% Ni+Cu in matte.

2.7.1 Slag blow stage

The slag blow or Slag-forming stage is the stage when the FeS in matte is oxidized to FeO, Fe_3O_4 and SO_2 using oxygen enriched air as indicated by equations 2.59 to 2.63

$Cu_2S\text{-}FeS(l) + 3/2O_2 = (l)Cu_2S(l) + FeO(l) + SO_2(g)$ copper smelting (2.59)

$Ni_3S_2\text{-}Cu_2S\text{-}FeS(l) + 3/2O_2(l) = Ni_3S_2\text{-}Cu_2S(l) + FeO(l) + SO_2(g)$ (2.60)

$2FeO(l) + SiO_2(s) = 2FeO.SiO_2(l)$ (2.61)

$3FeO(l) + 1/2O_2(g) = Fe_3O_4(l)$ (2.62)

$2FeS(l)+3O_2(g)+SiO_2(s) = 2FeO.SiO_2(l)+2SO_2(g)$ (2.63)

The melting points of FeO and Fe_3O_4 are 1385°C and 1597°C so silica flux is added to form a liquid fayalite slag with FeO and Fe_3O_4. The slag forming stage is terminated and slag is poured from the converter when there is about 1% Fe left in the matte. The principal product of the slag-forming stage is impure molten Cu_2S (white metal) in copper smelting or $Ni_3S_2\text{-}Cu_2S$ in PGM- Ni-Cu matte (Jones 1999, Eltringham 1979, 1993).

Industrially, copper matte is charged to the converter in several steps with each step followed by oxidation of FeS from the charge. The end of the slag blow is marked by drop on SO_2 concentration in the offgas, reduced temperature increase during blowing, appearance of white metal in slag samples taken from tuyeres and the turning of the greenish flame in the hood from deep yellow to green (Eltringham 1993). Slag is poured from the converter after each oxidation step and a new matte addition is made. In this way, the amount of Cu or Ni in the converter gradually increases until there is sufficient 100-250 tonnes Cu as molten Cu_2S (white metal) for a final copper making blow or $Ni_3S_2.Cu_2S$ bessemer matte (Davenport 2002, Wicks 2000). In PGM Ni-Cu converting the furnace matte is blown until the Fe in the matte is less than 1% which is determined by the X-ray analysis of the samples.

2.7.2 Copper blow stage

This process stage is only applied in copper smelting. The copper blow stage is the stage where the Cu_2S or white metal obtained from the slag blow in the same converter or the white metal (high grade matte) from the Noranda/Teniente converter is oxidised to blister Cu and SO_2 gas. The copper making reactions of the copper blow are indicated by equations 2.64 to 2.66 (Davenport 2002).

$$Cu_2S(l) + O_2(g) = 2Cu(l) + SO_2(g) \qquad (2.64)$$

$$2Cu(l) + 1/2O_2(g) = Cu_2O(l) \qquad (2.65)$$

$$2Cu_2O(l) + Cu_2S(l) = 6Cu(l) + SO_2(g) \qquad (2.66)$$

White metal is not appreciably oxidized until it is almost devoid of S. It takes place until the sulphur is lowered to 19.6%. Subsequent blowing of air and oxygen beyond 19.6%S causes a second liquid phase to (metallic copper with 1% S) to appear (Davenport 2002). The molten copper phase is dense and sinks to the bottom of the converter. Further blowing oxidizes additional sulphur from the Cu_2S and the amount of molten copper increases at the expense of the white metal. As long as the combined average composition of the system is in the immiscibility range, the converter contains both white metal with 19.6% S and molten copper with 1%S and eventually the white metal becomes S deficient meaning that the sulphide phase disappears and only molten copper with 1% S remains (Davenport 2002). Great care is taken during this period to ensure that the copper is not over oxidized to Cu_2O. This care is necessary because Cu_2S is no longer available to reduce Cu_2O back to Cu by reaction indicate by equation 2.66. Its beginning is marked by a change in the converter hood flame colour from green to clear whitish yellow when metallic copper begins to be oxidized as well as the drop in the SO_2 concentration in the offgas (Shook 1999). This tells the converter operator that the copper blow is nearly finished. The copper converting process is terminated the instant copper oxide begins to appear in samples of the molten copper. The copper is poured from the converter into ladles and craned molten to a fire refining furnace for S and O removal and casting of anodes (Davenport 2002).

The blister copper product from the converting is low in both sulphur and oxygen [0.001-0.03% S, 0.1-0.8% O] (Davenport 2002). If this copper were cast, the S and O would form SO_2 bubbles which leaves blisters or holes which give blister copper its name.

2.7.3 Operational Process Control

1) **Temperature control**

The high temperature during the slag making process is desired to give rapid slag formation, and a fluid slag with a minimum of entrained matte. It also discourages tuyere blockage (Davenport 2002). An upper limit of about 1275°C is imposed to prevent excessive refractory wear. The Bath temperature is monitored continuously with optical pyrometers sighted on the slag bath inside the furnace by two optical pyrometers mounted on both sides of the converter through the dedicated holes or by a two wavelength optical pyrometer periscope sighted through a tuyere (Pelletier 1987). The tuyere pyrometer appears to be more satisfactory because it sights directly on the matte rather than through a dust-laden atmosphere. Bath temperature of 1220^{o}C to 1275°C is controlled by maximizing the air blast flow rate of 550-750Nm^3/min and by oxygen enrichment of 24-29%. Reverts materials are added when the temperature is >1250^{o}C (Eltringham 1993).

2) **Slag Quality Control**

The chief objective of creating a slag in the converter is to liquefy newly formed solid FeO and Fe_3O_4 so they can be separated from the white metal and poured from the converter. SiO_2 bearing flux (quartz, quartzite, sand) is added for this purpose. Enough SiO_2 in flux must be charged to the converter to give Fe/SiO_2 ratio of 1.7-2.0 and acceptable Fe_3O_4 levels of 12-18% (Davenport 2002, Eltringham, 1993) while keeping the temperature between 1250-1275^{o}C. The Slag chemistry is controlled by adjusting oxygen Flow/Flux feed rate ratio. Some smelters use Au and Ag bearing siliceous material as converter flux. The Au and Ag dissolve in the matte and proceed with copper to the electro refinery where they are profitably recovered. These smelters tend to maximize flux input (Davenport 2002).

3) **Bath depth control**

The tuyere submergence in the bath is important in order to maximize the oxygen efficiency. The point of optimum submergence is determine by the tuyere back pressure which is kept between 80-120KPa (Eltringham 1993). This ensures efficient use of oxygen injected into the slag layer.

4) **Reverts and flux Quality**

The quality of reverts and flux in terms of composition plays a big influence on the performance of the process. A considerable change in the quality of the above can lead to a big process upset. For silica flux it must be ensured that the SiO_2 in flux as quartz is above 90% to minimize slag contamination with delirious oxides like MgO and Al_2O_3. The reverts that are charged must be of high grade in metal value and low in gangue components like MgO, Al_2O_3 and Cr_2O_3 which affect the slag quality (Davenport 2002).

2.8 Summary of literature review

A review of the literature of has indicated that the smelting and converting processes are susceptible to a large number of influences which include the concentrate or matte feed rate, fuel feed rate, oxygen/concentrate or matte ratio, flux/matte or concentrate ratio, bath temperature, airflow rate, oxygen enrichment, feed concentrate/ matte quality, flux quality and moisture in concentrate or matte. The major problem encountered in metallurgical plants by operators is the lack of availability of a comprehensive that can be used to predict process outputs when some operating variables such as those listed above change. In most of the plants operators use tables and charts to correct a major process upset. Although this method works, it does not provide an optimal solution because the tables of charts gives a rough indication of what is required while a mathematical model would be more specific if well configured.

The use of computers coupled with the knowledge of the process provides a good platform for process engineers and metallurgists to develop in-house process simulation models. These mathematical tools can be utilised to produce fully functional models which can be used to simulate converting and smelting processes at a low cost. The models can be used effectively for the plant process control, feasibility studies, design, fault finding, optimisation and re-engineering phases of a process.

3.0 Methodology

3.1 Model Development Methodology

The mathematical method and procedure used to develop the TSL base metals converter model used for treating Ni-Cu-PGM furnace matte is described in this chapter. The basic objective of the study is to demonstrate the use of mathematical matrix inverse method to model a TSL base metal converter for the treatment of Ni-Cu-PGM furnace matte. The study further demonstrate how the mathematical model can be modified to determine the process inputs feed rates if the process conditions change to achieve the prescribed set of goals.

The mathematical simulation of an industrial process is a complicated procedure. The actual nature of the process may not be well known and what is happening inside a reaction vessel is often not easily observed or measured. This is especially the case in pyrometallurgical operations which by definition require high temperatures. The main purpose of the computer model is to give a better understanding of the working of the process being simulated. The model should be capable of predicting the effect of changes in operating procedures on the process input requirements and mass and quality of the outputs. In this way the process can be optimised without carrying out extensive and expensive testworks.

Modelling of complex smelting systems as accurately as possible requires careful design of the computational domain and effective use of the numerical model provided by the Matrix inverse mathematical method. For this reason a systematic approach was followed in the model development. The mathematical matrix inverse method is packaged in the windows excel program as MINVERSE and MMULT. These formula commands in excel makes it easy to invert a matrix. In this chapter the reasoning behind the model development is discussed while the details regarding the creation on the matrix from the linear equations and the model specifications are provided.

The modelling process is one of making a number of assumptions, creating an initial model based on them, testing their validity and refining them to achieve a final product. The establishment of a comprehensive and stable model requires numerous initial trials and initial simulations for investigating the effects of different model set ups, model parameters, and assumptions. A stepwise approach with an increasing level of complexity was therefore followed to develop an accurate TSL PGM converting model. The following assumptions were used throughout the model;

- The process is at steady state (chemical and thermo equilibrium attained).
- No dust losses from the process.
- Mass balance calculations are carried out in kilograms.
- Calculations are based on the one hour process duration, i.e. feed rates, production rates and heat loss calculations.
- All stable oxides CaO, SiO_2, CaO, MgO, Al_2O_3 and Cr_2O_3 report to the slag.
- Ni, Cu, Fe and Co is distributed between product matte and slag as Ni_2S, Cu_2S, FeS and CoS in matte and NiO, Cu_2O, CoO, FeO and Fe_2O_3 in slag.
- Heat balances requires the enthalpies of various inputs and outputs including mineralogy for minerals and this information was calculated using thermo heat capacities acquired from the thermodynamics literature and is in MJ/kg.
- Temperature of reaction products chosen based on information from ternary phase diagrams liquidus temperature prediction

In the mass balances, only major substances where quantities in the feed charge exceed %0.01 are taken into consideration. The metals partition coefficient equations between slag and matte are derived from actual plant data as well as the slag chemistry obtained from one platinum smelter.

3.2 Model Operation

The matrix inverse mathematical method gives us a tool to solve several large linear equations. The model utilises the matrix inverse mathematical method to solve several linear equations that have been grouped into a square matrix. In this model, the linear equations represents the mass balances, heat balance, metal distribution equations, product matte grade equation and slag quality equations on the TSL converting process. The model also demonstrates further the change in process parameters due to change in feed matte quality, matte feed rate, blast oxygen enrichment, process operating temperature, moisture in feed matte and replacement of lance coal with diesel oil. A system of linear equations defined from the process mass and heat balances and distribution coefficients of input elements in the process outputs are arranged into a square matrix which is a matrix that has equal number of rolls and columns. The square matrix is solved using the matrix inverse method to give the process conditions as solutions of the matrix. The effect of change in the materials feeds quality, feed rates and the process operating conditions is also demonstrated throughout by changing the relevant linear equations. The matrix solutions obtained by the inverse are interpreted and interpreted while improvements to the matrix model are considered. The Gauss-Jordan elimination method can also be used to find the inverse of a square matrix (Winston 2003).

For example if we want to solve the linear system Ax = b that has m equations and m unknowns and suppose that A^{-1} exists, by multiplying both sides of Ax = b by A^{-1} it is seen that any solution of Ax = b must also satisfy $A^{-1}(Ax) = A^{-1}b$. Using the associative law and the definition of the matrix inverse, equations 3.01, 3.02 and 3.03 are obtained (Winston 2003)

$$(A^{-1}A)\,x = A^{-1}b \qquad (3.01)$$

or $$I_m\,x = A^{-1}b \qquad (3.02)$$

or $$x = A^{-1}b \qquad (3.03)$$

Where A is the square matrix, A^{-1} is an inverse matrix of A, I_m is an identity matrix of A, x is an equation variable and b is the equation constant. This shows that knowing A^{-1} enables to find the unique solution to a square linear system. To solve a single linear equation such as 4x = 3, simply multiply both sides of the equation by the multiplicative inverse of 4 which is 4^{-1} or ¼. This yields $4^{-1}(4x) = 4^{-1}(3)$ or x = ¾ (Winston 2003).

In the model, the manual method of calculating the inverse and solutions of the square matrix is not utilised and instead, the MINVERSE and MMULT formulas functions in excel computer program in utilised as it has been designed to solve the Gauss-Jordan matrix equations and the inverse of the square matrixes. The Gauss-Jordan method utilises elementary row operations (EROs) to solve the linear equation system (Winston 2003).

3.3 Model Development

The study is centered on developing the mathematical model for the TSL PGM converter that operates at 35tph solid matte feed, 40% oxygen enrichment and producing fluid matte and slag at 1325°C at the prescribed product matte and slag qualities. The TSL base metal converter is used to oxidise iron sulphide contained in the furnace matte to produce the iron deficient matte or bessemer matte suitable for treatment at the base metals refineries (BMR) and the precious metals refineries (PMR). The main raw materials of the TSL Ni-Cu matte converter include;

- Low grade Ni-Cu -PGM matte.
- Silica flux
- Air and industrial oxygen
- Fuel coal or oil

3.3.1 Furnace Matte

The source of the furnace matte is the electric furnaces that are used to smelt low grade Ni-Cu-PGM sulphide concentrates from the concentrators. Matte composition, mineralogy and temperature will always be known for a particular operating furnace. The main components of the PGM furnace matte are Ni, Fe, Cu and Co sulphide minerals and minor gangue PGMs metals and oxide compounds of SiO_2, CaO, MgO, Al_2O_2 and Cr_2O_3. The base metals exist in the form of pentladite (Ni_3S_2), pyrrhotite (FeS), chalcocite (Cu_2S), and carollite (CoS) (Jones 1999). The basic TSL converter furnace control strategy is to feed the furnace matte at a prescribed rate and to base all other controls on this feed rate. In this model, a feed rate of 35tph is used as a standard but the model has been tested for other matte feed rates. Furnace matte is considered to enter the converting process at 25°C. Tables A.4 and A.5 in the appendices illustrates the quality of fee matte used at one platinum smelter.

3.2.2 Silica flux

The silica flux is part of the feed because it is required to react with the iron oxides that are formed during the converting process to create a molten slag which can easily be removed from the furnace. The target slag Fe/SiO_2 ratio is obtained by controlling the flux/matte feed rate ratio in Kg flux per tonne of feed matte. The active component in quartz is the SiO_2 which reacts with the FeO formed from the FeS oxidation process. The amount of flux added must satisfy the Fe/SiO_2 ratio of 1.70-1.85 in slag (Davenport 2002). The flux enters the converting process at $25^{o}C$.

3.3.3 Air and Oxygen

The composition and temperature of the blast are important operating parameters of the converting process. Oxygen enrichment is a standard practice for the TSL smelting and converting furnaces. The blast to the TSL furnaces is always a mixture of N_2 and O_2. The blast usually contains 30-60% O_2 by volume (Matusewicz 2005). The O_2 in air and industrial oxygen is used to oxidise the FeS in the matte to FeO and Fe_3O_4. The main purpose of using oxygen in the TSL is to maximise the heat output by decreasing the N_2 from air passing through the furnace that absorbs a lot of useful heat thereby minimising the use of external fossil fuel to balance the bath temperature. Air contains 21% O_2 by volume and industrial oxygen is 95 - 99%. The blast oxygen enrichment equation used in the model is indicated by equation 3.04.

$$\frac{(0.233*\mathrm{Ma}+0.9912*\mathrm{MO2})}{(\mathrm{Ma}+\mathrm{MO2})}*100=\%\ \mathrm{Oxy\ by\ mass\ in\ blast} \qquad (3.04)$$

Where Ma is mass of air and M_{O2} is mass of oxygen in the lance blast. Air and industrial oxygen is considered to enter the converting process at $25^{o}C$. The oxygen enrichment equation is calculated by equation3.7 in the appendix.

3.3.4 Process fuel oil and coal

Lump coal of size -25mm to +15mm is part of the feed in the TSL converter. Its purpose is to reduce unwanted oxides Fe_3O_4 and NiO in the slag to the acceptable concentrations in the instances where the slag bath is over oxidised.

Fuel coal or diesel oil is used to supplement the heat during the converting process normally at low matte feed rates, low oxygen enrichment and during the furnace heat up. Lump fuel coal is pulverised into fine type with particle size of 100% -1mm which is used as lance fuel for bath heating. In the model both coal and diesel oil is assumed to enter the furnace at 25^{o}C.

3.3.5 Converter products

The principle products of the converting process are converter iron deficient matte, slag and off gas. Same as the feed matte, the converter product matte is presented as ideal solutions of metal sulphides mainly Ni_2S and Cu_2S with small amounts of CoS and FeS in the mass and heat balances. The matte composition is not presented as metal sulphide compounds in mass balances but rather as elements as indicated by the lab chemical analysis. The Ni in the product matte is regarded as Ni_2S due to sulphur deficiency nature of the product converter matte. The slag is presented in the mass and heat balances as ideal liquid solutions of metal oxides of FeO, Fe_3O_4, SiO_2 with minor amounts of NiO, Cu_2O, CoO, CaO, MgO, Al_2O_3, Cr_2O_3 (Mills 1995). The offgas composition is also known and is presented as N_2, SO_2, CO_2, H_2O, CO and unreacted O_2 in the model. The appendices A.6, A.7, A.8 and A.9

3.3.6 Steady operation of the TSL converter

Steady state converting of furnace Ni-Cu matte consists of steadily blowing oxygen enriched air, feeding dry granulated matte and flux and sometimes fossil fuel into the molten slag bath while intermittently tapping converter matte and slag. The control strategy of the TSL converter is that the converter must be fed with furnace matte, oxygen, air, coal or oil and flux at prescribed rates in order to meet the following goals whose benefits were described in the literature review;

- Producing converter matte at a grade of 2.5-4.5% Fe suitable for treatments at downstream processes.
- Producing slag with a Fe/SiO_2 ratio of 1.7-1.85 at the temperature of 1300-1350^{o}C.

- Producing offgas at 1325°C with SO_2 content of >7% by volume that can be economically converted into H_2SO_4 in the contact acid plant.

3.3.7 Slag composition control

The FeO and Fe_3O_4 formed from the oxidation of matte and the minor gangue oxide minerals in the matte are fluxed by SiO_2 in the flux. The slag Fe/SiO_2 ratio is kept between 1.7-1.85 which promotes the formation of the slag with a low liquidus temperature. This ratio is controlled by the Flux/matte feed ratio (Davenport 2002). The slag composition equation is calculated by using equation 3.6 in the appendix.

3.3.8 Converter matte Grade control

The purpose of the TSL converting process is to reduce the Fe composition in the furnace matte from 40% to $3 \pm 0.5\%$ on average (Jacobs 2006, Jones 1999). The converter matte grade obtained is controlled by the oxygen/matte feed rate ratio in Nm^3 per tonne of feed matte. This ratio fixes the amounts of Fe and S which are to be oxidised from the feed furnace matte and to a smaller extent Ni, Cu and Co. The Ni content obtained in the slag bath sample analysis is used to adjust the oxygen specific consumption factor in the furnace control program. Equation 3.05 obtained from the graph in figure 2.15 sourced from the relevant smelter is utilised to estimate Fe as FeS concentration in the product matte by using the Ni assay obtained from the slag bath samples that are taken regularly.

$$\%Ni_{\text{slag}} = 4.917 * \%Fe^{-0.4273}_{\text{matte}} \qquad (3.05)$$

Where %Ni is the nickel in slag bath as NiO and %Fe is the iron in converter product matte as FeS. The converter matte Fe grade is calculated with equation3.4 in the appendix.

3.3.9 Slag bath temperature

The slag from the TSL Ni-Cu matte converter is a fayalite (Fe/SiO2 >1.75) type which is desired to be between 1250-1350°C. Slag temperature control is critical during the smelting process since slag has a higher melting point and viscosity than matte. Slag temperature is chosen to be high enough for good slag fluidity and an efficient separation of matte and slag and easy handling (Davenport 2002. The temperature of the slag bath is maintained by adjusting the lance fuel feed rate or by adjusting the blast oxygen enrichment or by increasing the matte feed rate which increases the FeS oxidation rate. In the model the slag and matte bath temperature is assumed to be at 1325^{o}C which is the normal operating temperature for the one TSL PGM converter at one platinum smelter. The offgas components exit at the same temperature of 1325^{o}C from the reaction zone.

3.4 Mass balance and material

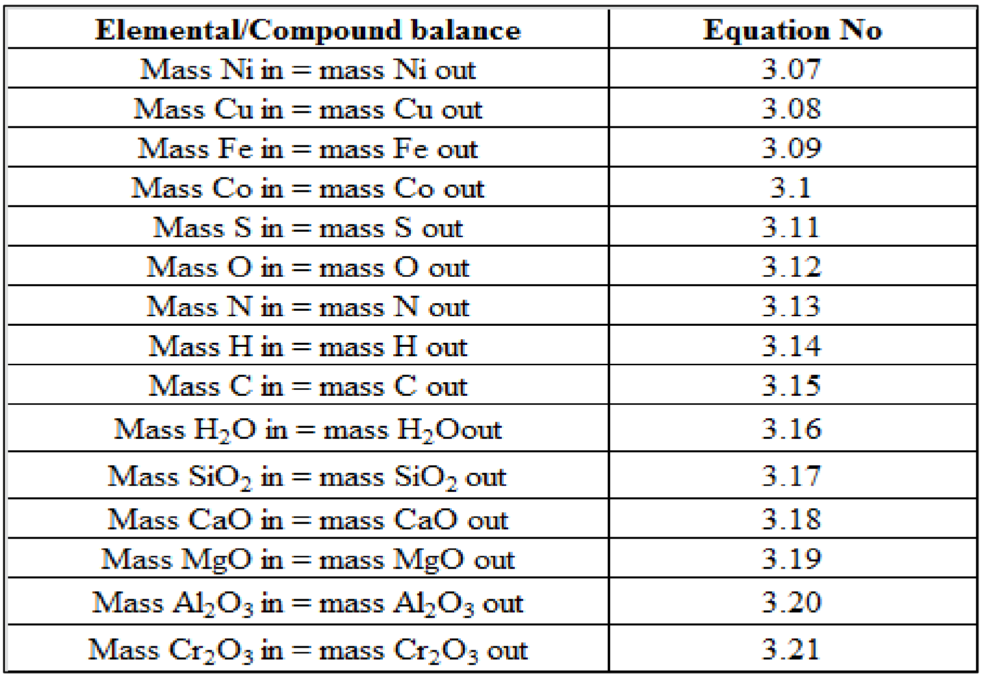
3.4.1 Mass balance

The mass balance in the model takes into consideration the mass conservation law. The general mass balance equation which must be obeyed during the stead state operation is highlighted by equation 3.06

$$\sum Xi \text{ (feed)} = \sum Xi \text{ (products)} \qquad (3.06)$$

Xi represents a specific element or compound in the feeds entering the furnace and in the products exiting the furnace. All equations are developed for the elements and compounds in the feeds and products in kilograms for the one hour operation time. The basic elemental mass balance equations for the converting of Ni-Cu furnace matte in the TSL furnace are represented by equations from 3.07 -3.21 indicated in table 3.1. The balances for Cu, Ni, Co and Fe partitioning in the slag are calculated using equations 3.1, 3.2, 3.3 and 3.4 in the appendix. The information in tables A.1 and A.2 were used to develop the mass balance equations in appendix 3 which have been used in the development of the model.

Table 3-1: Mass balance fundamentals



Elemental/Compound balance	Equation No
Mass Ni in = mass Ni out	3.07
Mass Cu in = mass Cu out	3.08
Mass Fe in = mass Fe out	3.09
Mass Co in = mass Co out	3.1
Mass S in = mass S out	3.11
Mass O in = mass O out	3.12
Mass N in = mass N out	3.13
Mass H in = mass H out	3.14
Mass C in = mass C out	3.15
Mass H_2O in = mass H_2Oout	3.16
Mass SiO_2 in = mass SiO_2 out	3.17
Mass CaO in = mass CaO out	3.18
Mass MgO in = mass MgO out	3.19
Mass Al_2O_3 in = mass Al_2O_3 out	3.20
Mass Cr_2O_3 in = mass Cr_2O_3 out	3.21

Mass balances of other elements and compounds will be introduced later when required. SiO_2 is assumed to behave like an element in the sense that the silicon-oxygen bond is not broken during smelting. Other stable oxides such as CaO, Al_2O_3, MgO, Cr_2O_3 and H_2O are treated in the same way as SiO_2.

As stated before, the feed materials for the TSL base metal converter are Ni-Cu furnace matte, silica flux, blast of oxygen/air mixture and lance fossil fuel. The furnace input matte composition and mineralogy is always known for a given smelting process. The matte composition is normally given in elemental form as Ni, Cu, Fe, Co and S but it must be understood that these exist as compounds of metal sulphides and are presented as ideal liquid solutions of Ni_3S_2, Ni_2S, Cu_2S, CoS and FeS in the mass and heat balances (Bale, Toguri 1976 and Jones 1999). Likewise, the chemical compositions and mineralogy of flux and fuel will be known and presented the same as the feed matte. The blast is always a mixture of N_2 and O_2 from air and industrial oxygen. The composition of the blast is an important operating variable of the TSL smelting process and need to be specified in the model.

3.4.2 Enthalpy or Heat balance

The heat balance considers all major heat inputs and outputs from the converter. The heat balance of the process is made under the following assumptions;

- Enthalpy and masses of feed materials depends on temperature, mineralogy and mass of components.
- Enthalpy of products depends on temperature, mineralogy and mass of compounds that comprises the products slag, matte and the off gas.
- Heat loss is taken as the constant per hour and is based on the calculated converter heat loss using the design heat loss fluxes.

The steady state operation of the furnace stipulates that for the enthalpy balance, the enthalpy entering the furnace is equal to the enthalpy leaving the furnace plus the heat Losses from the furnace. In mathematical terms the heat balance equation is represented by equation 3.22 indicated.

$$\sum H_{reactants} = \sum H_{products} + H_{loss} \qquad (3.22)$$

The $H_{reactants}$ represents the sum of the enthalpies of all feed material entering the furnace and its numerical value depends on the quantity and temperature of each feed material. $H_{products}$ represents the enthalpy of the furnace products in the same way as feeds. H_{loss} is the heat loss from the reactor by conduction through the furnace walls and eventual convection and radiation of this heat away from the furnace or by being transferred into the circulating cooling water in the copper cooler.

The enthalpy of the inputs and outputs components is calculated using the formula indicated by equation 3.23.

$$H_T = \int C_p dT = H° + AT + 1/2\, BT^2 - CT^{-1} + 1/3\, DT^3 \quad \text{MJ/kg} \qquad (3.23)$$

Where H is enthalpy in MJ/kg, T is temperature in Kelvin and A, B, C and D are heat capacity constants. Equation 3.23 has been used to calculate the enthalpies in appendix 4.0

The temperature of the feed and product materials on the TSL converting furnace will either be known or specified for evaluation purposes. The feed materials furnace matte, silica flux, fossil fuel, oxygen and air are assumed to be at the atmospheric temperature of 25^{o}C. The approach taken in the calculations for the enthalpy in the model is that all components of the products matte, slag and offgas leave the furnace at the same temperature of 1325^{o}C. Effects of operating at other temperatures were tested as well. The normal operating bath temperature of the Ni-Cu-Fe-PGM matte converters is between 1300-1350^{o}C (Jones 1999).

Using information sourced from the relevant smelter, the converting process reactions in the TSL furnace takes place in the copper waffles crucible which is 4.4m in diameter and 4.45m high. The copper waffles have been designed to withstand the heat fluxes of 100KW/m^2 on the lower part and 50KW/m^2 on the upper part of the furnace. Low pressure water flowing at 10000L/minute is used to intensively cool the copper coolers to promote the formation of the slag freeze lining of 10-15mm that protects the coolers from coming into contact with the molten bath. The heat loss from the furnace reactor is calculated using the design heat fluxes from different sections of the furnace and is calculated using equation 3.24 as indicated in appendix 4.0.

$$H_{loss} = A * k \tag{3.24}$$

Where H_{loss} is furnace heat loss in kW, A is the surface area in m^2 of the bottom part of the cylindrical furnace where the reactions take place and k is the design heat loss flux from the furnace in KW/m^2.

4.0 Model Application

The model in this study was worked for the Ni-Cu-PGM furnace mattes in South Africa but the same principle has been applied to simulate other pyro metallurgical processes. Similar models have been demonstrated by Davenport 2002 and Kucharski 2007 for the Outokumpu flash smelting furnace and the direct to blister copper flash smelting furnaces.

This chapter is devoted to developing the process linear equations and matrices that represent the mass and heat balances of the steady state industrial process of the TSL PGM furnace matte converter. It then calculates the masses of the process inputs required and the masses of products at a determined process conditions. The subsequent sections of this chapter demonstrate further how the linear equations can be modified when the process feeds quality and operating conditions are altered e.g. feed chemical quality, matte feed rate, oxygen enrichment, bath temperature, moisture in matte, lance fuel type and blast air preheat. The basic objectives of this chapter are to;

- To demonstrate the development of the process linear equations and matrices that describe the mass and heat balances of the TSL base metal converter used to treat PGM furnace matte.
- To indicate how a TSL converter can be operated and controlled so as to consistently give products of prescribed quality and operating temperature.
- To show how the TSL converter can be optimised to achieve the process objectives with minimum expenditure on resources.

The fundamental principle upon which the TSL converter model is developed is based on the steady state mass balances of elements and compounds entering and leaving the furnace and the overall heat balance of the process. The principles highlighted in the appendices 1, 2, 3 and 4 will be used to develop the mass and heat balance equations of the process. The results predicted by the model in the following sections will be compared to the actual plant operating parameters in appendix5 sourced from the relevant base metal smelter that utilises the TSL for converting PGM furnace matte.

4.1 Illustrative process model

In this illustrative model, the TSL converter is assumed to be running at a feed rate of 35tph furnace matte to produce converter matte with 2.78% Fe as FeS and slag with the Fe/SiO_2 ratio of 1.80. The Ni, Cu and Co in the feed report to the converter matte as molten sulphide solutions of Cu_2S, Ni_2S and CoS and partially to the slag as NiO, Cu_2O and CoO. Most of the Fe in the feed reports to the slag as FeO and Fe_3O_4 and a small amount remains in the matte as FeS. Oxides in the feed SiO_2, CaO, Al_2O_3, MgO and Cr_2O_3 all report to the slag as molten solutions. It is assumed that all the feeds i.e matte, blast air and oxygen, silica flux and fuel coal enter the furnace at 25°C and all the furnace products molten matte and slag and off gas exit the furnace at 1325°C. The blast is composed of air and oxygen at 40% oxygen enrichment by volume and the fuel utilised is pulverised coal. The feed matte type and flux used is indicated as Feed 1 and silica flux in table 4.1 and the fuel type is coal indicated in table 4.2. Feed1 type matte in table 4.1 contains more sulphide base metals and less oxide gangue compounds when compared to matte feed types feed2 and feed3. Coal contains some ash which mainly comprises SiO_2 and Al_2O_3 while diesel oil is only C and H with no ash. The compositions of feed matte, flux, coal and diesel is in % by mass. Air is assumed to contain 21% oxygen and 99% for industrial oxygen by volume. The model will calculate the inputs air, oxygen, flux and coal requirements and outputs matte, slag and off gas weights and their compositions.

Table 4-1: Feed matte and flux

	%Cu	%Ni	%Co	%Fe	%S	%SiO_2	%CaO	%MgO	%Al_2O_3	%Cr_2O_3	H_2O	Total
Feed 1	9.38	15.44	0.43	39.61	28.01	4.45	0.16	0.25	0.64	1.62	0.00	100.0
Feed 2	8.38	14.40	0.42	37.50	25.84	9.52	0.65	0.35	1.20	1.74	0.00	100.0
Feed 3	8.04	13.65	0.47	34.80	24.45	14.07	0.81	0.47	1.35	1.89	0.00	100.0
Feed 4	8.73	14.38	0.40	36.88	26.08	4.14	0.15	0.23	0.60	1.51	6.90	100.0
SilicaFlux	0.00	0.00	0.00	0.00	0.00	98.00	0.00	0.00	2.00	0.00	0.00	100.0

Table 4-2 : Coal and diesel assays

	%C	%H	%N	%S	%O	%SiO2	%Al2O3	%CaO	%MgO	%Fe2O3	%H2O	Total
Coal	70.86	3.5	1.49	1.66	5.27	9.15	6.18	0.51	0.27	0.61	0.5	100
Diesel	86	14	0	0	0	0	0	0	0	0	0	100

The mass balance will focus on the main elements and unstable compounds of Cu, Ni, Fe, Co, S, O, N, C, H and the stable compounds SiO_2, CaO, MgO, Al_2O_3, Cr_2O_3 and H_2O. The useful linear equations developed for the PGM converting process that represents the mass and energy balances of the converting process are indicated by equations 4.01 to 4.26.

1) **Copper balance equation**

Copper enters the furnace in solid form in the furnace feed matte and leaves the furnace as molten Cu_2S and Cu_2O in converter matte and slag. The copper balance per 35000kg of feed matte is given as;

Mass Cu in = mass Cu out

$35000* 0.0938$ Cu in matte $= 0.8* Cu_2S + 0.89*Cu_2O$

$$3283 = 0.8* Cu_2S + 0.89*Cu_2O \qquad (4.01)$$

2) **Nickel balance equation**

Nickel enters the furnace in solid form in the furnace matte feed and leaves the furnace as molten Ni_2S and NiO in converter matte and slag.

Mass Ni in = mass Ni out

$35000* 0.1544$ Ni in matte $= 0.78* Ni_2S + 0.79*NiO$

$$5404.78* Ni_2S + 0.79*NiO \qquad (4.02)$$

3) **Cobalt balance equation**

Cobalt enters the furnace in solid form in the furnace matte feed and exits the furnace in molten state as CoS and CoO in converter matte and slag.

Mass Co in = mass Co out

$35000* 0.0043$ Co in matte $= 0.65*CoS + 0.79*CoO$

$$129 = 0.65* CoS+ 0.79*CoO \qquad (4.03)$$

4) **Iron balance equation**

Iron enters the furnace in in solid form in the furnace matte feed and coal and exits the furnace in molten state as FeS in converter matte and as FeO and Fe_3O_4 in slag.

Mass Fe in = mass Fe out

$35000*0.3961$ Fe in matte + $0.00476*$ Fe in Coal = $0.64*$ FeS + $0.78*FeO$ + $0.72*Fe_3O_4$

$$13864 = -0.00476*Coal + 64*FeS+0.78*FeO + 0.72*Fe_3O_4 \qquad (4.04)$$

5) **Oxygen balance equation**

Reaction oxygen enters the furnace in the blast air and oxygen as gases and in solid coal as Fe_2O_3 and exits the furnace as molten FeO, Fe_3O_4, NiO, Cu_2O, CoO in slag and in gaseous state as CO_2, SO_2 and H_2O in the offgas.

Mass O in = mass O out

$$0.233*Air + 0.9912*Oxygen + 0.0544*Coal = 0.22*FeO + 0.28*Fe_3O_4$$
$$+ 0.21*NiO + 0.11*Cu_2O + 0.21*CoO + 0.5*SO_2 + 0.73*CO_2 + 0.89*H_2O$$

$$0 = -0.233*Air - 0.9912*Oxygen - 0.0544*Coal + 0.22*FeO + 0.28*Fe_3O_4$$
$$+ 0.21*NiO + 0.11*Cu_2O + 0.21*CoO + 0.5*SO_2 + 0.73*CO_2$$
$$+ 0.89*H_2O \qquad (4.05)$$

6) Sulphur balance equation

The sulphur enters the furnace in solid metal sulphide solutions in furnace matte feed and coal and exits the furnace as molten Ni_2S, Cu_2S, FeS and CoS in converter matte and in gaseous state as SO_2 in the offgas, therefore the S mass balance equation is;

Mass S in = mass S out

$35000*0.2801$ S in matte $+ 0.0166*$ S Coal $= 0.36*FeS + 0.22*Ni_2S$
$+ 0.2*Cu_2S + 0.35*CoS + 0.5*SO_2$
$9804 = - 0.0166*Coal+0.36*FeS + 0.22*Ni_2S + 0.2*Cu_2S$
$+ 0.35*CoS + 0.5*SO_2$ (4.06)

7) Nitrogen balance equation

Nitrogen enters the furnace in in gaseous state in the blast air, industrial oxygen and coal and it exits the furnace as N_2 in the offgas in gaseous state.

Mass N in = mass N out
$0.767*N_2$ in Air $+ 0.0088* N_2$ in Oxygen $+ 0.0149*Coal = 1.0* N_2$
$0 = -0.767*Air - 0.0088*Oxygen - 0.0149*Coal + 1.0*N_2$ (4.07)

8) Carbon balance equation

Carbon enters the furnace in the fuel coal in solid form and leaves the furnace in gaseous state as CO_2 in the offgas. The carbon mass balance equation is written as;

Mass C in = mass C out
$0.7086*$ C in Coal $= 0.27* CO_2$
$0 = -0.7086*Coal + 0.27* CO_2$ (4.08)

9) Hydrogen balance equation

Hydrogen enters the furnace in the fuel coal as a gas and leaves the furnace as H_2O vapour in the offgas phase and therefore the hydrogen equation is written as;

Mass H in = mass H out
$0.0356*$ H in Coal $= 0.11* H_2O$
$0= -0.0356*Coal + 0.11* H_2O$ (4.09)

10) Hydrogen balance equation

Silica enters the furnace in solid form through furnace matte as an impurity, flux and fuel coal. It leaves the furnace as molten SiO_2 in the slag phase. The silica equation is written as;

$$35000*0.0445* SiO_2 \text{ in matte} + 0.98*Flux + 0.0915*Coal = 1.0*SiO_2$$

$$1558 = -0.98*Flux - 0.0928*Coal + 1.0*SiO_2 \qquad (4.10)$$

11) Alumina balance equation

Alumina enters the furnace in solid form through furnace matte as an impurity, flux and coal and exits in the slag phase as molten Al_2O_3. The Al_2O_3 balance equation is given as;

$$\text{Mass } Al_2O_3 \text{ in} = \text{mass } Al_2O_3 \text{ out}$$

$$35000*0.0064* Al_2O_3 \text{ in matte} + 0.02*Flux + 0.0618*Coal = 1.0* Al_2O_3$$

$$225 = -0.02*Flux - 0.0618*Coal + 1.0*Al_2O_3 \qquad (4.11)$$

12) Calcium oxide balance equation

CaO enters the furnace in solid form through furnace matte as an impurity and coal. It exits as molten CaO in the slag phase and its balance equation is given as;

$$\text{Mass CaO in} = \text{mass CaO out}$$

$$35000*0.00160* CaO \text{ in matte} + 0.0051*Coal = 1.0* CaO$$

$$57 = -0.0051*Coal + 1.0*CaO \qquad (4.12)$$

13) Magnesium Oxide balance equation

MgO enters the furnace in solid form through feed furnace matte as an impurity and coal and exits as molten CaO in the slag phase, therefore the mass balance equation is written as:

Mass MgO in = mass MgO out
35000*0.0025* MgO in matte + 0.0028*Coal = 1.0* MgO
88 = -0.0028*Coal + 1.0* MgO (4.13)

14) Chrome balance equation

Cr_2O_3 enters the furnace in solid form through feed furnace matte as an impurity and exits as molten Cr_2O_3 in the slag phase. The mass balance equation is given as;

Mass Cr_2O_3 in = mass Cr_2O_3 out
30000*0.0162* Cr_2O_3 in matte = 1.0* Cr_2O_3
567 = 1.0* Cr_2O_3 (4.14)

15) Water balance equation

The H_2O enters the furnace in liquid state in the feed furnace matte and leaves as H_2O vapour in the off gas phase.

Mass H_2O in = mass H_2O out
35000*0.00* H_2O in matte = 1.0* H_2O
0 = 1.0* H_2O (4.15)

16) Reverts balance equation

The reverts enters the furnace as solids and the products leave in molten form in the matte and slag and in gaseous form in the offgas. The reverts required are charged into the furnace when the bath temperature goes above the maximum determined safe limit. Mr in the equation represents the mass of reverts.

Mass Reverts in = mass of Reverts components in matte, slag and offgas

$$0 = -0.2875*Mr + -0.445*Mr + -0.032*Mr + -0.218*Mr + -0.0035*Mr + -0.0098*Mr + -0.0018*Mr + -0.0005*Mr + -0.0014*Mr + -0.0005*Mr + Cu_2S + Ni_2S + CoS + FeS + Cu_2O + NiO + CoO + FeO + Fe_3O_4 + SiO_2 + CaO + MgO + Al_2O_3 + Cr_2O_3 + SO_2 \quad (4.16)$$

17) Slag mass equation

The slag is composed of metal oxides and therefore the total slag mass equation is the sum of all the oxide minerals and is given as;

$$\text{Mass Slag} = SiO_2 + FeO + Fe_3O_4 + Cr_2O_3 + MgO + CaO + Al_2O_3 + NiO + Cu_2O + CoO \quad (4.17)$$

18) Converter product matte total mass equation

The product converter matte is composed of Ni_2S, Cu_2S, CoS and FeS and therefore the total mass of converter matte is the sum of all the product sulphide metal species. The product matte mass balance equation is given as;

$$\text{Mass Matte} = Ni_2S + Cu_2S + FeS + CoS \quad (4.18)$$

19) Matte carrier air

Matte carrier air is used to pneumatically move atomized furnace matte from the storage bin to the furnace into the molten bath. The matte carrier air is fixed at 3000Nm^3/hr and therefore, the matte carrier air equation is given as;

3000*1.2869 = Matte carrier air

3861 = Matte carrier air (4.19)

20) Coal carrier air

Coal carrier air is used to pneumatically move pulverized fuel coal from the storage bin to the furnace into the molten bath. The coal carrier air is fixed at 500Nm^3/hr and therefore, the matte carrier air equation is given as;

500*1.2869 = Matte carrier air

643 = Matte carrier air (4.20)

21) Total Air Mass equation

The total converter lance input air is composed of matte carrier air, coal carrier air and lance air and therefore, the equation for the total air flow to the lance tip is given as follows;

Total air = Matte carrier Air + Coal carrier air + lance air (4.21)

22) Slag quality equation

The converter slag quality is represented by the Fe/SiO_2 ratio in the slag bath. The target ratio is 1.80 which is known to promote the formation of the slag that has a low liquidus temperature of 1200^oC. The slag quality equation is given as;

$$0 = 0.78*FeO + 0.72*Fe_3O_4 - 1.80*SiO_2 \qquad (4.22)$$

23) Slag Iron distribution equation

The Fe in the slag is split between FeO and Fe_3O_4. It is assumed that 80% of Fe in the feed is oxidised to FeO and 20% to Fe_3O_4 which is close to most matte sulphide converters performance. The slag Fe distribution equation is given as;

$$0 = 1.0833*FeO – 4.0*Fe_3O_4$$

(4.23)

24) Converter matte iron grade equation

The converter product matte quality is the measure of the level of the Fe concentration left after the oxidation process. The Fe in converter matte exists only in the form of FeS and therefore, for the Fe grade of 2.78% in converter matte, the Fe concentration equation in product matte is given as;

$$0 = – 0.64*FeS + 0.0278*\text{mass of converter matte} \qquad (4.24)$$

25) Nickel concentration equation in slag

Ni in the slag bath is used to determine the Fe grade in the product matte by using the graph in figure 2.12 or using the equation 3.05 which is the equation of the graph in figure 2.12. For the target Fe in matte of 2.5-3.5%, the Ni in the slag bath is targeted between 3.0-3.3%. For the Fe target in converter matte of 2.78%, equation 3.05 calculates 3.18% Ni in the converter slag bath as the target which exists as NiO. Therefore, the Ni in slag equation is given as follows;

$$0 = 0.79*NiO - 0.0318*Mass\ of\ slag \qquad (4.25)$$

26) Copper distribution equation in matte and slag

The distribution of Cu between matte and slag is estimated from the plant data of the TSL PGM converter at one of the platinum smelter. The historical data indicates that at the product Fe grade of 2.78% in converter matte, 85% of Cu in the feed matte report to converter matte as Cu_2S and 15% reports to the slag as Cu_2O. Therefore, the Cu distribution equation between slag and matte is given as;

$$0 = 5.667*Cu_2O - 0.8989*Cu_2S \qquad (4.26)$$

27) Cobalt distribution equation in matte and slag

Same as the copper, the distribution of Co between matte and slag is estimated from the plant data of the TSL PGM converter at a platinum smelter. At 2.78% Fe in converter matte, 26% of Co reports to converter matte as CoS and 74% reports to the slag as CoO. Therefore, the Co distribution equation between slag and matte is given as;

$0 = 0.3514*CoO – 0.823*CoS$ (4.27)

28) Blast Oxygen enrichment equation

The lance air oxygen enrichment is given by equation 3.03 which provides the distribution of oxygen from air and industrial oxygen at a given oxygen enrichment in the blast. At the oxygen enrichment of 40%, the enrichment equation of the blast of air and oxygen is given as;

$0.233* Ma +0.9912* MO_2 = (Ma + MO_2)*0.432$

$0 = 1.2922*MO_2 – 0.4612*Ma$ (4.28)

Where Ma and M_{O2} is mass of air and industrial oxygen

29) Furnace wall Heat loss equation

The heat loss from the furnace is calculated using equation 3.24 for the given design heat loss fluxes of the lower and upper parts of the TSL furnace. The heat loss in MJ/hr as calculated is given as;

$48668 = H_{loss}$ (4.29)

30) Enthalpy balance equation

The enthalpy balance of the converting process is represented by equation 3.21 and the enthalpy of an element or compound at a particular temperature is calculated using equation 3.22. The subscript 25 in H_{25} represents the temperature of inputs and outputs in degrees celcius. The total enthalpy of the inputs at 25°C is;

$\sum$Hreactants = Cu_2S in matte*H25 + Ni_3S_2 in matte* H25
+FeS in matte*H25 + CoS in matte* H25
+ SiO_2 in matte* H25 + CaO in matte*H25
+MgO in matte*H25 + Al_2O_3 in matte*H25
+Cr_2O_3 in matte*H25+Air*H25 + Oxygen* H25

+ mass reverts*H25 + Coal feed*H25

+flux mass*H25

= 4104* -0.51+ 6880* -0.90 + 232* -0.91 + 21628* -1.16

+ 1558*-15.18 + 57*-11.34 + 88*-15.03 + 225*-16.42

+ 567*-6.78 + Air*0 + Oxygen*0 + Reverts*H25

+ Flux*-15.23 + Coal*-2.62

$\sum$ Hreactants = - 66683 + Air*0 + Oxygen*0 + Reverts* -0.960

+ Flux* -15.13+ Coal* -2.62

Same as the inputs, the subscript 1325 in H_{1325} represents the temperature of outputs. The enthalpies of the products at 1325°C are therefore given as;

$\sum$ Hproducts = Cu_2S in matte*H1325 + Ni_2S in matte* H1325

+ FeS in matte*H1325+ CoS in matte* H1325

+ FeO in slag* H1325 + SiO_2 in slag*H1325

+ CaO in slag*H1325 + MgO in slag*H1325

+ Al_2O_3 in slag*H1325+ Cr_2O_3 in slag*H1325

+ Fe_3O_4 in slag*H1325 + NiO in slag*H1325

+ Cu_2O in slag*H1325 + CoO in slag*H1325

+ N_2 in offgas*H1325+ SO_2 in offgas*H1325

+ CO_2 in offgas*H1325 + H_2O in offgas*H1325

+ Hloss

= Cu_2S in matte*0.30 + Ni_2S in matte* 0.33+FeS in matte*0.19

+ CoS in matte* 0.46 + FeO in slag* -2.33

+ SiO_2 in slag*-13.60+ CaO in slag*-8.77

+ MgO in slag*-11.85+ Al_2O_3 in slag*-13.80
+ Cr_2O_3 in slag*-5.15 + Fe_3O_4 in slag*-3.09
+ NiO in slag*-1.44+ Cu_2O in slag*-0.0023
+ CoO in slag*-1.43+ N_2 in offgas*1.50
+ SO_2 in offgas*-3.56 + CO_2 in offgas*-7.41
+ H_2O in offgas*-10.50 + 48668

The overall enthalpy balance of the converting process is given as;

- 66683 = -Air*0 - Oxygen*0 - Reverts* -0.986 - flux* -15.23 - Coal*-2.62
+ Cu_2S in matte*0.30 + Ni_2S in matte* 0.33+ FeS in matte*0.19
+ CoS in matte* 0.46 + FeO in slag* -2.33 + SiO_2 in slag*-13.60
+ CaO in slag*-8.77 + MgO in slag*-11.85 + Al_2O_3 in slag*-13.80
+ Cr_2O_3 in slag*-5.15 + Fe_3O_4 in slag*-3.09 + NiO in slag*-1.44
+ Cu_2O in slag*-0.0023 + CoO in slag*-1.43 + N_2 in offgas*1.50
+ SO_2 in offgas*-3.56 + CO_2 in offgas*-7.41
+ H_2O in offgas*-10.50 +48668 (4.30)

The mathematical problem is restricted to solving of a set of 30 linear equations that represent the converting process conditions described by equations 4.01 to 4.30. The process linear equations described in this section are transformed into a square calculation matrix indicated in table 4.3. The unknown variables in the linear equations that represent the converting process variables are calculated as solutions of the calculation matrix in table 4.3 using the matrix inverse method that uses the Gauss-Jordan method to calculate the inverse of the calculation matrix.

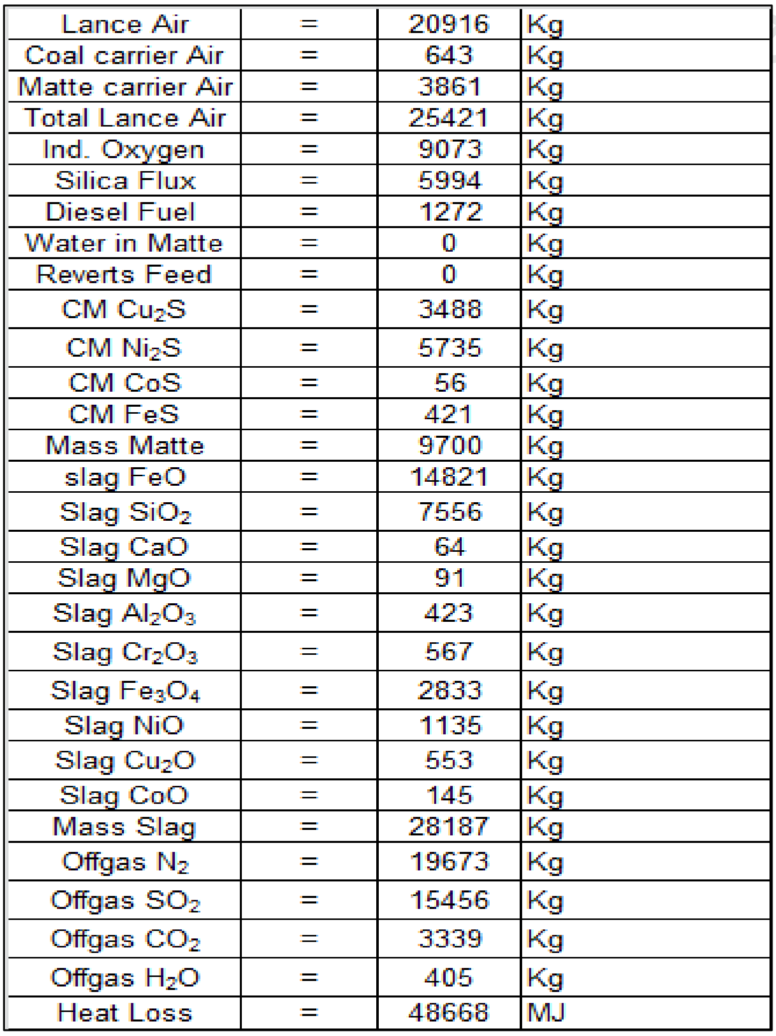
Table 4-3: Calculation matrix for process simulation illustration 4.10

Balance	Value	Lance Air	Coal carrier Air	Matte carrier Air	Total Lance Air	Ind. Oxygen	Silica Flux	Lance Coal	Water in Matte	Reverts Feed	Cu2S	Ni2S	CoS	FeS	Mass Matte	FeO	SiO2	CaO	MgO	Al2O3	Cr2O3	Fe3O4	NiO	Cu2O	CoO	Mass Slag	N2	SO2	CO2	H2O	Heat Loss
Cu	3283	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.288	0.800	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	13864	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	-0.032	0.000	0.000	0.000	0.640	0.000	0.780	0.000	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
S	9804	0.000	0.000	0.000	0.000	0.000	0.000	-0.016	0.000	-0.218	0.200	0.214	0.350	0.360	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.500	0.000	0.000	0.000
O	0	-0.230	-0.230	-0.230	0.000	-0.991	0.000	-0.055	-0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.220	0.000	0.000	0.000	0.000	0.000	0.280	0.210	0.110	0.210	0.000	0.000	0.500	0.730	0.890	0.000
N	0	-0.770	-0.770	-0.770	0.000	-0.009	0.000	-0.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000
Co	151	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	0.000	0.650	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000
Ni	5404	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.445	0.000	0.786	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
C	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.709	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.270	0.000	0.000
H	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.035	-0.110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.110	0.000
SiO2	1558	0.000	0.000	0.000	0.000	0.000	-0.980	-0.098	0.000	-0.010	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al2O3	225	0.000	0.000	0.000	0.000	0.000	-0.020	-0.062	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
CaO	57	0.000	0.000	0.000	0.000	0.000	0.000	-0.005	0.000	-0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Cr2O3	567	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MgO	88	0.000	0.000	0.000	0.000	0.000	0.000	-0.003	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
H2O	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Reverts	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Coal Carrier Air	643	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Carrier Air	3861	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Total Lance Air	0.00	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Grade	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.6400	0.0278	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Slag Comp	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.780	-1.800	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Blast O2 Enrich	0.00	0.000	0.000	0.000	-0.461	1.292	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ni Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.790	0.000	0.000	0.0318	0.000	0.000	0.000	0.000	0.000
Cu Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.899	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Co Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.823	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.316	0.000	0.000	0.000	0.000	0.000	0.000
Slag Fe Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.083	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Mass Slag	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000
Mass Matte	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Heat Loss MJ/hr	48668	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000
Enthalpy(MJ)	-66683	0.000	0.000	0.000	0.000	0.000	15.210	2.620	15.880	0.960	0.305	0.330	0.464	0.190	0.000	-2.333	-13.600	-8.773	-11.849	-13.796	-5.152	-3.092	-1.444	-0.002	-1.428	0.000	1.495	-3.558	-7.410	-10.501	1.000

4.11 Calculation Matrix results and discussion

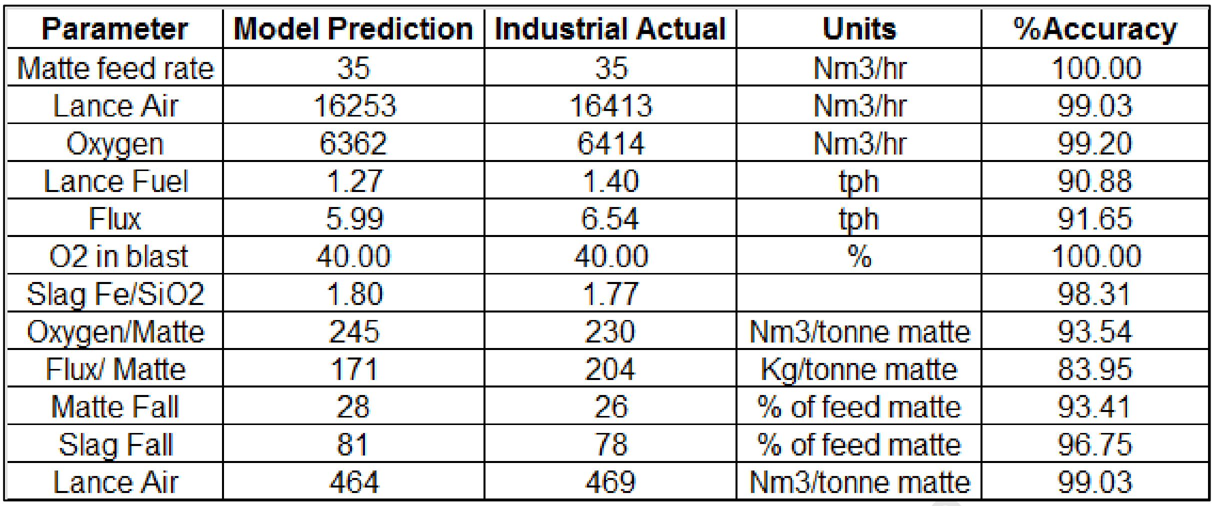
The solution of the calculation matrix in table 4.3 is given in table 4.4 and it represents the calculated mass values of the unknown variables (process inputs and output masses) for the converting process at the operating conditions described in this section. The tables from 4.5 to 4.8 indicate the analysed process information of the matrix results in table 4.4 which is valuable in optimising and designing the converting process. The FeO-Fe_3O_4-SiO_2 ternary diagram in figure 4.1 indicates the liquidus or melting temperature of the slag predicted by the model of the PGM matte converting process. The predicted converter matte and slag qualities are compared with information contained in appendices A.10, A11 and A.12. The tables A.6, A.7, A.8 and A.9 show the slag bath control data in relation with the specific oxygen and flux inputs which are compared with the model predictions.

Table 4-4: Results for calculation matrix in table 4.3



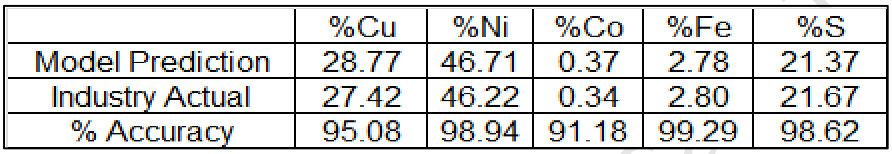
Lance Air	=	20916	Kg
Coal carrier Air	=	643	Kg
Matte carrier Air	=	3861	Kg
Total Lance Air	=	25421	Kg
Ind. Oxygen	=	9073	Kg
Silica Flux	=	5994	Kg
Diesel Fuel	=	1272	Kg
Water in Matte	=	0	Kg
Reverts Feed	=	0	Kg
CM Cu_2S	=	3488	Kg
CM Ni_2S	=	5735	Kg
CM CoS	=	56	Kg
CM FeS	=	421	Kg
Mass Matte	=	9700	Kg
slag FeO	=	14821	Kg
Slag SiO_2	=	7556	Kg
Slag CaO	=	64	Kg
Slag MgO	=	91	Kg
Slag Al_2O_3	=	423	Kg
Slag Cr_2O_3	=	567	Kg
Slag Fe_3O_4	=	2833	Kg
Slag NiO	=	1135	Kg
Slag Cu_2O	=	553	Kg
Slag CoO	=	145	Kg
Mass Slag	=	28187	Kg
Offgas N_2	=	19673	Kg
Offgas SO_2	=	15456	Kg
Offgas CO_2	=	3339	Kg
Offgas H_2O	=	405	Kg
Heat Loss	=	48668	MJ

Table 4-5: Predicted process parameters for simulation 4.1



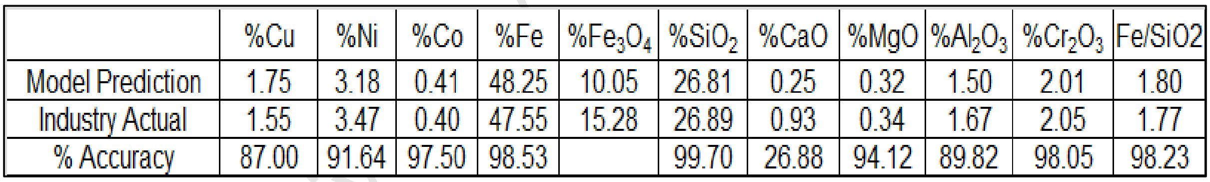
Parameter	Model Prediction	Industrial Actual	Units	%Accuracy
Matte feed rate	35	35	Nm3/hr	100.00
Lance Air	16253	16413	Nm3/hr	99.03
Oxygen	6362	6414	Nm3/hr	99.20
Lance Fuel	1.27	1.40	tph	90.88
Flux	5.99	6.54	tph	91.65
O2 in blast	40.00	40.00	%	100.00
Slag Fe/SiO2	1.80	1.77		98.31
Oxygen/Matte	245	230	Nm3/tonne matte	93.54
Flux/ Matte	171	204	Kg/tonne matte	83.95
Matte Fall	28	26	% of feed matte	93.41
Slag Fall	81	78	% of feed matte	96.75
Lance Air	464	469	Nm3/tonne matte	99.03

Table 4-6: Converter matte composition for simulation 4.1



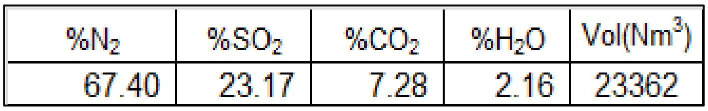
	%Cu	%Ni	%Co	%Fe	%S
Model Prediction	28.77	46.71	0.37	2.78	21.37
Industry Actual	27.42	46.22	0.34	2.80	21.67
% Accuracy	95.08	98.94	91.18	99.29	98.62

Table 4-7: Converter slag composition for simulation 4.1



	%Cu	%Ni	%Co	%Fe	%Fe_3O_4	%SiO_2	%CaO	%MgO	%Al_2O_3	%Cr_2O_3	Fe/SiO2
Model Prediction	1.75	3.18	0.41	48.25	10.05	26.81	0.25	0.32	1.50	2.01	1.80
Industry Actual	1.55	3.47	0.40	47.55	15.28	26.89	0.93	0.34	1.67	2.05	1.77
% Accuracy	87.00	91.64	97.50	98.53		99.70	26.88	94.12	89.82	98.05	98.23

Table 4-8: Offgas composition for simulation 4.1



%N_2	%SO_2	%CO_2	%H_2O	Vol(Nm^3)
67.40	23.17	7.28	2.16	23362

Figure 4.1: FeO-Fe2O3-SiO2 ternary showing slag liquidus temperature for simulation 4.1

It has been demonstrated that the converting process of Ni-Cu-PGM matte in the TSL furnace can be simulated by the mathematical matrix inverse modeling method. The inverse of the calculation matrix that represents various mass, heat and metal distribution linear equations of the process is solved using the Gauss-Jordan method. These equations can be adapted to the raw materials requirements and amounts of products for an industrial TSL base metals converter furnace. The model has shown that given a pyrometallurgical task of producing iron deficient converter matte or blister copper, fluid slag and offgas of specified grades in the base metals converter requires unique amounts of oxygen/air, flux and fuel coal/diesel. Amounts higher than or less than those specified by the model will lead to a process upset or failure

The illustrative converting process model has shown that for the process to run at steady state at the matte feed rate of 35tph at the given process conditions, unique amounts of fuel coal, flux, air and oxygen indicated in table 4.5 is required to maintain the heat balance of the converting process. The model also calculated the masses of the molten and gaseous products from the furnace indicated in table 4.4. Figure 4.1 indicates the melting temperature of the slag predicted by the model on the FeO-SiO_2-Fe_2O_3 ternary diagram. The melting temperature indicated is 1200^oC at a Fe/SiO_2 ratio of 1.80. This temperature is lower than the bath specified operating temperature of 1325^oC and hence the slag will have a super heat of $>100^oC$ which will make the slag sufficiently fluid for easy handling and to minimise valuable metal losses. The matte and slag analysis indicated in tables 4.6 and 4.7 are consistent with the assays from one of the platinum smelters reported by Jones 1999 and the information sourced from the relevant smelter indicated in the appendix. The offgas analysis given in table 4.8 has the SO_2 content of >7% which is economically suitable for conversion into H_2SO_4 acid using the contact acid making process (Davenport 2002). The analysed process parameters in table 4.5 are also consistent with the actual plant operating parameters obtained from one platinum smelter that utilises the TSL converter as indicated by the high accuracy of above 90% on most critical parameters.

The subsequent sections extends the usefulness of the equations and calculations to demonstrate how the process can be optimized when there is a change in feed matte quality, matte feed rate, oxygen enrichment, bath operating temperature, moisture in feed matte and change in fuel type from coal to diesel. The sections represent the strategies by which the goals of the converting process can be achieved with the change.

4.2 Effect of oxygen enrichment on process

All TSL furnaces burn fuel to supplement the energy obtained by the Fe and S oxidation (Arthur 2005). Adjusting the amount of fossil fuel combusted in the furnace is an effective way of controlling the bath temperature. Oxygen enrichment in the TSL smelting furnaces is considered standard practice and this is done so to minimise the fossil fuel and the blast air volume requirements due to the nitrogen influence on the heat balance (Arthur 2005). The nitrogen entering in the blast from the air component must always be heated from its incoming temperature of 25^oC to its exit offgas temperature of 1325^oC and therefore takes away some of the heat from the converting process. In the oxidation process of the Ni-Cu PGM matte in the TSL to produce the iron deficient matte of 2.78% Fe, a small amount of external fuel has to be added to the bath to maintain the desired bath temperature. Nitrogen also increases the volume of gas entering and leaving the furnace and dilutes the SO_2 concentration in the offgas.

The effect of nitrogen in the blast and the method by which it can be quantified is illustrated in this section. This section examines the influence of the blast oxygen enrichment on the;

- Fuel requirement
- Air requirement
- Volume of the offgas
- SO_2 concentration in the offgas

The matte, flux and coal types considered in this section are the same as those in section 4.10 indicated in table 4.1 and 4.2. All the process conditions remain the same as in section 4.10 except blast oxygen enrichment which is changed from 40% to 30%. The furnace matte feed rate is fixed at 35tph. The incoming matte, flux, oxygen and air are at 25^oC and all products of the process namely converter matte, slag and the offgases are maintained at 1325^oC. The target matte and slag compositions are 2.78% Fe and Fe/SiO_2 of 1.80.

4.2.1 Calculation Matrix

All linear equations for this simulation are the same as that given in section 4.10 except for the oxygen enrichment equation 4.24 which is changed due to the change in the oxygen enrichment of the blast to the lance. Equation 4.24 represents the distribution of oxygen from industrial oxygen and oxygen from atmospheric air at a given oxygen enrichment set point. For the oxygen enrichment of 30%, equation 4.24 changes as indicated in equation 4.27;

$$0.233*Ma + 0.9912*MO2 = (Ma + MO_2)*\%O_2 \text{ by mass}/100$$
$$0 = 2.0149* MO_2 - 0.29163* Ma \qquad \text{at } 30\% \ \ O_2 \text{ enrichment} \qquad (4.27)$$

Inserting the updated oxygen enrichment equation 4.27 into the calculation matrix 4.3 alters row number 22 with factors for total lance air and industrial oxygen requirements at the oxygen enrichment level of 30%. The calculations were repeated for the enrichments at 21%, 35%, 40% and 45%. The calculation matrix for this operating scenario with 30% oxygen enrichment is indicated in table 4.9.

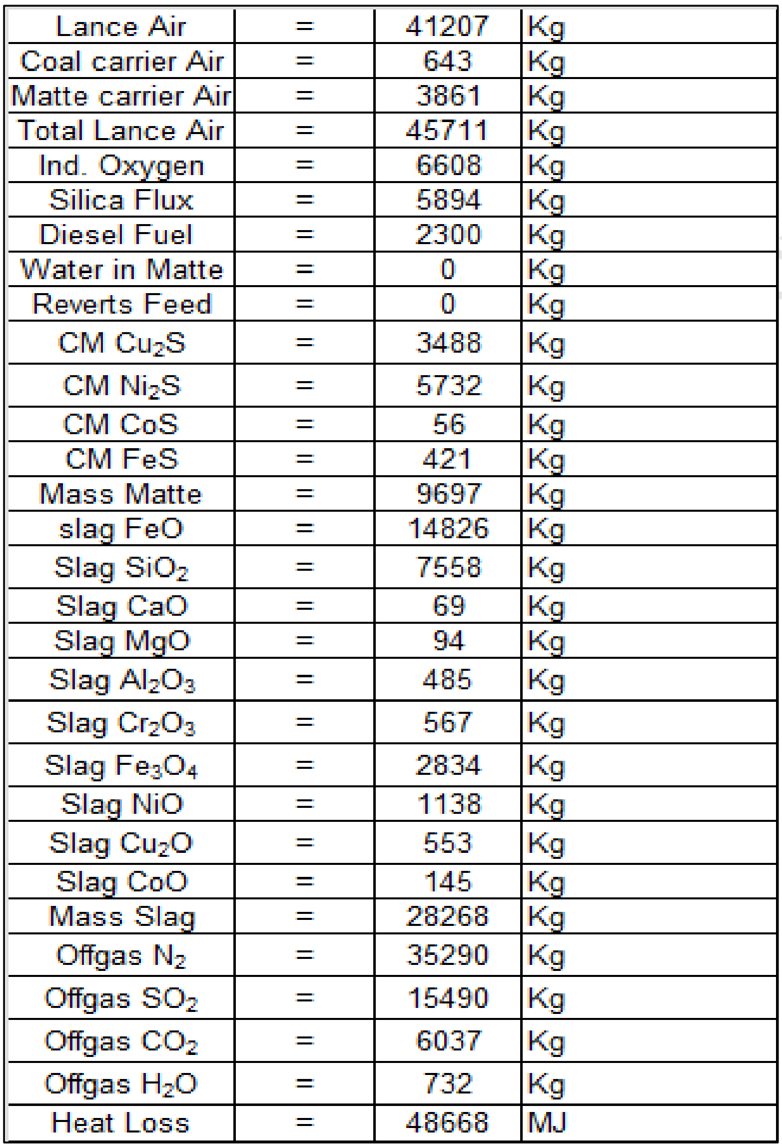
Table 4-9: Process calculation matrix for simulation 4.2

Balance	Value	Lance Air	Coal carrier Air	Matte carrier Air	Total Lance Air	Ind. Oxygen	Silica Flux	Lance Coal	Water in Matte	Reverts Feed	Cu2S	Ni2S	CoS	FeS	Mass Matte	FeO	SiO2	CaO	MgO	Al2O3	Cr2O3	Fe3O4	NiO	Cu2O	CoO	Mass Slag	N2	SO2	CO2	H2O	Heat Loss
Cu	3283	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.288	0.800	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	13864	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	-0.032	0.000	0.000	0.000	0.640	0.000	0.780	0.000	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
S	9804	0.000	0.000	0.000	0.000	0.000	0.000	-0.016	0.000	-0.218	0.200	0.214	0.350	0.360	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.500	0.000	0.000	0.000
O	0	-0.230	-0.230	-0.230	0.000	-0.991	0.000	-0.055	-0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.220	0.000	0.000	0.000	0.000	0.000	0.280	0.210	0.110	0.210	0.000	0.000	0.500	0.730	0.890	0.000
N	0	-0.770	-0.770	-0.770	0.000	-0.009	0.000	-0.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000
Co	151	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	0.000	0.650	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000
Ni	5404	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.445	0.000	0.786	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
C	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.709	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.270	0.000	0.000
H	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.035	-0.110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.110	0.000
SiO2	1558	0.000	0.000	0.000	0.000	0.000	-0.980	-0.098	0.000	-0.010	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al2O3	225	0.000	0.000	0.000	0.000	0.000	-0.020	-0.062	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
CaO	57	0.000	0.000	0.000	0.000	0.000	0.000	-0.005	0.000	-0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Cr2O3	567	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MgO	88	0.000	0.000	0.000	0.000	0.000	0.000	-0.003	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
H2O	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Reverts	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Coal Carrier Air	643	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Carrier Air	3861	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Total Lance Air	0.00	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Grade	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.6400	0.0278	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Slag Comp	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.780	-1.800	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Blast O2 Enrich	0.00	0.000	0.000	0.000	-0.291	2.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ni Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.790	0.000	0.000	0.0318	0.000	0.000	0.000	0.000	0.000
Cu Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.899	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Co Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.823	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.316	0.000	0.000	0.000	0.000	0.000	0.000
Slag Fe Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.083	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Mass Slag	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000
Mass Matte	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Heat Loss MJ/hr	48668	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000
Enthalpy(MJ)	-66683	0.000	0.000	0.000	0.000	0.000	15.210	2.620	15.880	0.960	0.305	0.330	0.464	0.190	0.000	-2.333	-13.600	-8.773	-11.849	-13.796	-5.152	-3.092	-1.444	-0.002	-1.428	0.000	1.495	-3.558	-7.410	-10.501	1.000

4.2.2 Calculation Matrix results and discussion

The results of the calculation matrix in table 4.9 is given in table 4.10 which shows the mass of the variables that represent the process inputs and output masses of the converting process at the operating conditions assumed in this section. The analysis of the matrix results from table 4.10 is highlighted in tables 4.11, 4.12 and 4.13 and in figures 4.2 and 4.3.

Table 4-10: Results of calculation matrix table 4.9



Lance Air	=	41207	Kg
Coal carrier Air	=	643	Kg
Matte carrier Air	=	3861	Kg
Total Lance Air	=	45711	Kg
Ind. Oxygen	=	6608	Kg
Silica Flux	=	5894	Kg
Diesel Fuel	=	2300	Kg
Water in Matte	=	0	Kg
Reverts Feed	=	0	Kg
CM Cu_2S	=	3488	Kg
CM Ni_2S	=	5732	Kg
CM CoS	=	56	Kg
CM FeS	=	421	Kg
Mass Matte	=	9697	Kg
slag FeO	=	14826	Kg
Slag SiO_2	=	7558	Kg
Slag CaO	=	69	Kg
Slag MgO	=	94	Kg
Slag Al_2O_3	=	485	Kg
Slag Cr_2O_3	=	567	Kg
Slag Fe_3O_4	=	2834	Kg
Slag NiO	=	1138	Kg
Slag Cu_2O	=	553	Kg
Slag CoO	=	145	Kg
Mass Slag	=	28268	Kg
Offgas N_2	=	35290	Kg
Offgas SO_2	=	15490	Kg
Offgas CO_2	=	6037	Kg
Offgas H_2O	=	732	Kg
Heat Loss	=	48668	MJ

Table 4-11: Predicted process parameters for simulation 4.2

Parameter	Section 4.1	Section 4.2	Industrial Actual	Units	% Change	%Accuracy
Matte feed rate	35	35	35	Nm3/hr	0.00	100.00
Lance Air	16253	32020	16413	Nm3/hr	97.01	4.91
Oxygen	6362	4634	6414	Nm3/hr	-27.16	72.25
Lance Fuel	1.27	2.30	1.40	tph	80.81	35.68
Flux	5.99	5.89	6.54	tph	-1.66	90.12
O2 in blast	40.00	30	40.00	%	-25.00	75.00
Slag Fe/SiO2	1.80	1.80	1.77		0.00	98.31
Oxygen/Matte	245	247	230	Nm3/tonne matte	0.97	92.51
Flux/ Matte	171	168	204	Kg/tonne matte	-1.66	82.55
Matte Fall	28	28	26	% of feed matte	-0.04	93.44
Slag Fall	81	81	78	% of feed matte	0.29	96.45
Lance Air	464	915	469	Nm3/tonne matte	97.01	4.91

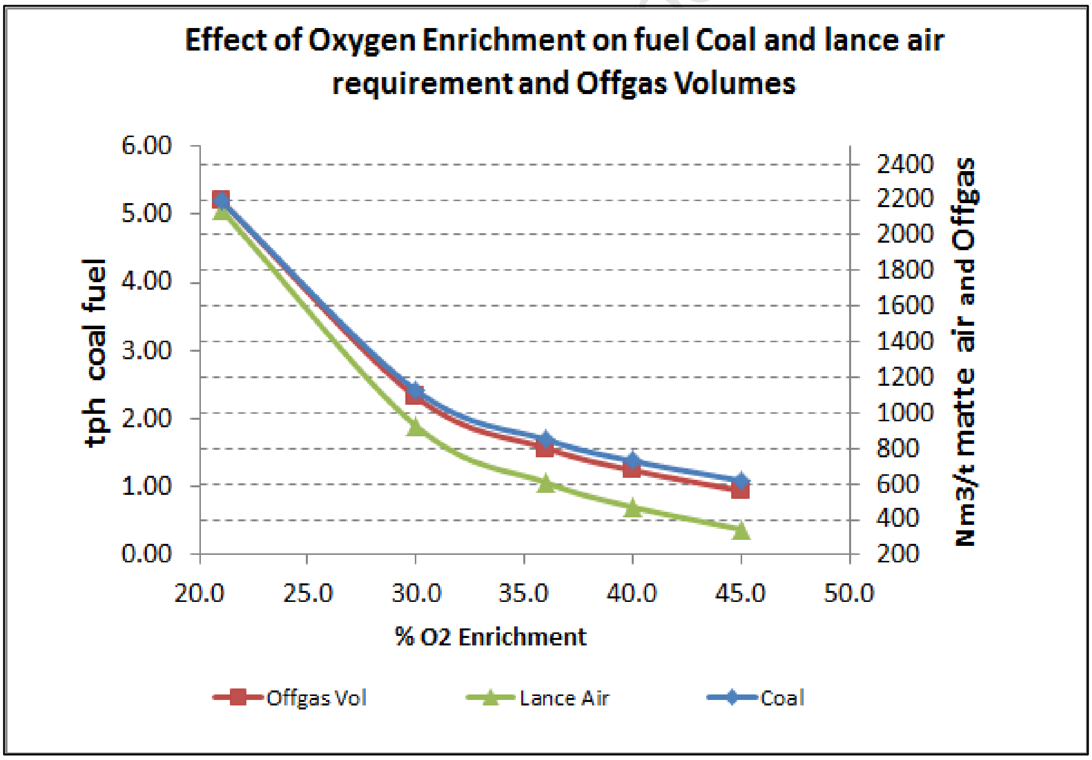


Figure 4.2: Fuel coal and lance air requirement and offgas flow for simulation 4.2

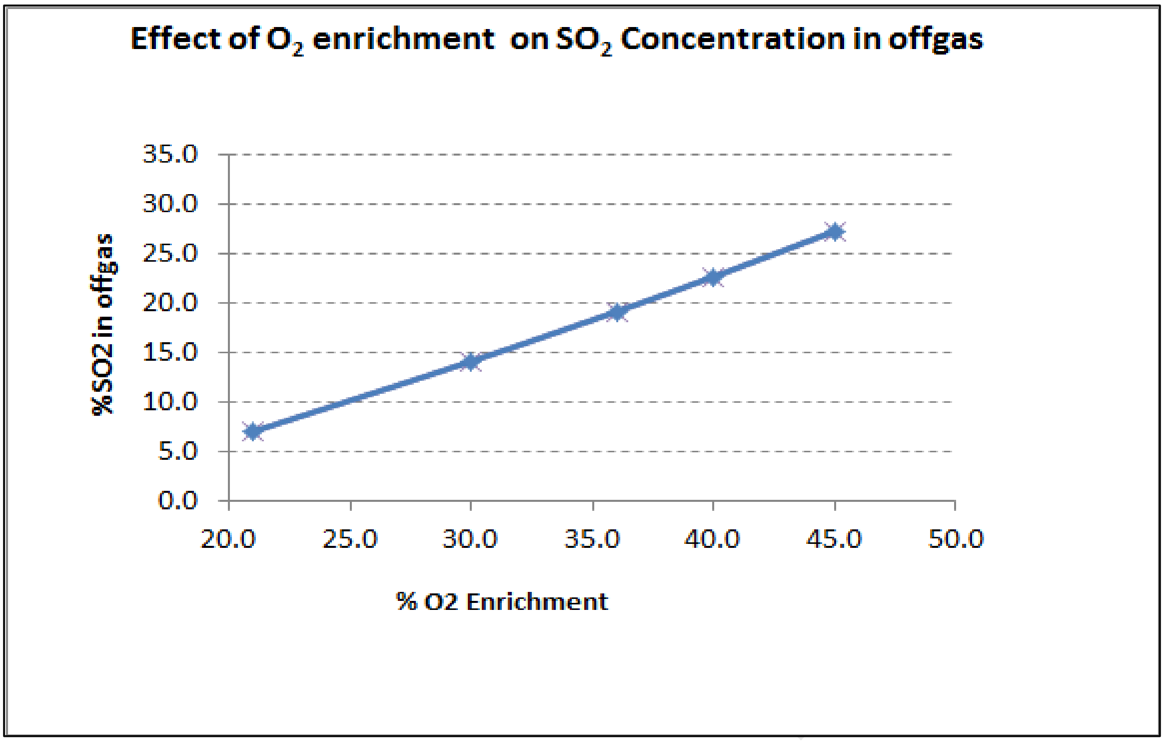


Figure 4.3: SO2 in offgas at different oxygen enrichments set points

Table 4-12: Converter matte quality for simulation 4.20

	%Cu	%Ni	%Co	%Fe	%S
Model Prediction	28.78	46.70	0.37	2.78	21.37
Industry Actual	27.42	46.22	0.34	2.80	21.67
% Accuracy	95.04	98.96	90.45	99.18	98.62

Table 4-13: Converter slag quality for simulation 4.2

	%Cu	%Ni	%Co	%Fe	$\%Fe_3O_4$	$\%SiO_2$	%CaO	%MgO	$\%Al_2O_3$	$\%Cr_2O_3$	Fe/SiO2
Model Prediction	1.74	3.18	0.40	48.13	10.03	26.74	0.24	0.33	1.71	2.01	1.80
Industry Actual	1.55	3.47	0.40	47.55	13.50	26.89	0.93	0.34	1.67	2.05	1.77
% Accuracy	87.51	91.64	98.83	98.79	74.27	99.43	26.16	97.51	97.33	97.85	98.21

Examination of tables 4.12 and 4.13 shows that the converter matte and slag qualities simulated by the model are very close to the ones in section 4.10 which is consistent with the grades from the platinum smelters reported by Jones 1999 and Matthew 2009. Information in table 4.11 and figure 4.02 indicates that the fuel requirement decreases with the increase in oxygen enrichment and vice versa (Davenport 2002). The increase in oxygen enrichment from 30% to 40% decreases the fuel coal requirement from 65.7 to 36.3kg/tonne feed matte

(2.30 -1.27tph) as indicated in figure 4.2 and table 4.11. The decrease in fuel requirement is due to decrease in the nitrogen content in the blast which absorbs a lot of heat from ambient temperature to the operating temperature of the furnace (Davenport 2001, 2002).

Another observation from figure 4.2 is that a larger volume of blast is required at low oxygen enrichment with its associated cost. Figure 4.2 indicates that a decrease in oxygen enrichment from 45 to 21% results in large increase in lance air from 464 to 2122Nm^3/tonne matte and offgas volumes from 667 to 2171Nm^3/tonne matte. Low oxygen enrichment leads to a larger blast volume (Matusewicz 2005, Davenport 2002) and a larger offgas volume that considerably require a larger gas handling system. The SO_2 concentration in the offgas decreases with the decrease in the oxygen enrichment as can be seen from figure 4.3 which makes the acid making process expensive (Davenport 2001).

This section has shown that the effect of changing the blast oxygen enrichment can readily be determined by the calculation matrix by changing the oxygen enrichment equation. The blast oxygen enrichment specification is useful for comparing the alternative energy inputs into the converting or smelting process at the expense of the oxygen making process. The matrix results indicated that the oxygen enrichment of the blast has a big effect on the fuel and lance blast requirements and the offgas volume. The effect of operating the TSL at low oxygen enrichment levels requires more fuel coal or oil to balance the process heat requirement (Matusewicz 2005, Davenport 2001). A large air blast results into process problems of splashing and a large offgas volume with low SO_2 concentration and hence the justification of a high oxygen enrichment of the blast as a standard practice for the TSL processes (Matusewicz 2005).With the feed matte and flux qualities that are consistent, the operator only needs to change the lance fuel feed rate at different oxygen enrichments as the specific consumption factors for oxygen and flux will remain the same (Matthew 2009).

4.3 Effect of feed matte chemical composition

The TSL feed matte vary considerably in chemical composition due to contamination of the matte during tapping and casting. Some PGM smelters cast the furnace matte in silica sand pits leading to contamination of the matte tapped. In some cases matte can be contaminated with slag during tapping.

This section examines the effect of feed matte composition on the fuel, flux and oxygen requirements for operating the TSL converter at the specified temperature and matte feed rate. The section also examines the effect of feed matte composition on the offgas volume, SO_2 in offgas concentration and product matte and slag masses.

The matrix is this section highlights the effect of processing of three different types of matte of different chemical compositions at the same process conditions pointed out in section 4.10. The matte type used in this section is feed3 indicated in table 4.1. Feed3 has less base metal sulphides and high silica compositions as compared to feed1 and feed2 matte types.

4.3.1 Calculation Matrix

Same as in other simulations demonstrated, this converting illustration is going to indicate the change in the fuel, flux and oxygen requirements. The simulation also shows the change in masses of the products converter matte and slag as a ratio of the feed matte.

The change in this matrix model is the input mass balance equations for the components Cu, Ni, Co, Fe, S, SiO_2, CaO, MgO, Al_2O_3 and Cr_2O_3 in feed matte and the enthalpy of the feed matte as indicated in equations 4.28 to 4.38.

$$35000 * 0.0804 \text{ in matte} = 0.8 * Cu_2S + 0.89 * Cu_2O$$
$$2814 = 0.8 * Cu_2S + 0.89 * Cu_2O \quad (4.28)$$

$$35000 * 0.1365 \text{ in matte} = 0.78 * Ni_2S + 0.79 * NiO$$
$$4778 = 0.78 * Ni_2S + 0.79 * NiO \quad (4.29)$$

$35000*0.0047$ in matte $= 0.65*CoS + 0.79*CoO$

$$165 = 0.65*CoS + 0.79*CoO \qquad (4.30)$$

$35000*0.3480$ in matte $+ 0.00476*Coal = 0.64*FeS + 0.78*FeO + 0.72*Fe_3O_4$

$$12180 = -0.00476*Coal + 0.64*FeS + 0.78*FeO + 0.72*Fe_3O_4 \qquad (4.31)$$

$35000*0.2445$ in matte $+ 0.0166*Coal = 0.36*FeS + 0.22*Ni_2S +$
$0.2*Cu_2S + 0.35*CoS+0.5*SO_2$.

$$8558 = -0.0166*Coal + 0.36*FeS + 0.22*Ni_2S + 0.2*Cu_2S + 0.35*CoS + 0.5*SO_2 \qquad (4.32)$$

$35000*0.1407*$mass matte $+ 0.96*Flux + 0.0915*Coal = 1.0*SiO_2$

$$4925 = -0.96*Flux - 0.0928*Coal + 1.0*SiO_2 \qquad (4.33)$$

$35000*0.0135*$mass feed matte $+ 0.04*Flux + 0.0618*Coal = 1.0*Al_2O_3$

$$473 = -0.04*Flux - 0.0618*Coal + 1.0*Al2O3 \qquad (4.34)$$

$35000*0.0081*$mass feed matte $+ 0.0051*Coal = 1.0*CaO$

$$284 = -0.0051*Coal + 1.0*CaO \qquad (4.35)$$

$35000*0.0047*$mass feed matte $+ 0.0028*Coal = 1.0*MgO$

$$165 = -0.0028*Coal + 1.0*MgO \qquad (4.36)$$

$35000*0.0189*$mass feed matte $= 1.0*Cr_2O_3$

$$662 = 1.0*Cr_2O_3 \qquad (4.37)$$

And the heat balance for the process with feeds entering at 25^{o}C and the products exiting the process at 1325^{o}C is;

$\sum$Hreactants = Cu_2S in matte*H25 + Ni_2S in matte* H25 + FeS in matte*H25
+ CoS in matte* H25 + SiO_2 in matte* H25+ CaO in matte*H25
+ MgO in matte*H25 + Al_2O_3 in matte*H25 + Cr_2O_3 in matte*H25
+ Air*H25 + Oxygen* H25 + mass reverts*H25 + Coal feed*H25
+ flux mass*H25

$$\begin{aligned}&= 3518^{*}\,{-0.51} + 6082^{*}\,{-0.90} + 253^{*}\,{-0.91} + 19001^{*}\,{-1.16} + 4925^{*}{-15.18}\\&+ 284^{*}{-11.34} + 165^{*}{-15.03} + 473^{*}{-16.42} + 662^{*}{-6.78} + \text{Air}^{*}0\\&+ \text{Oxygen}^{*}0 + \text{Reverts}^{*}{-0.96} + \text{Flux}^{*}{-15.23} + \text{Coal}^{*}{-2.62}\end{aligned}$$

$$\begin{aligned}\sum H_{\text{reactants}} = &- 122190 + \text{Air}^{*}0 + \text{Oxygen}^{*}0 + \text{Reverts}^{*}\ {-0.960} + \text{Flux}^{*}\ {-15.13}\\&+ \text{Coal}^{*}\ {-2.62}\end{aligned}$$

The numerical terms in the output side of the enthalpy equation are not affected by the change in input matte chemical composition. Therefore, the product compounds and their temperatures are the same as in section 4.10 and 4.20 and hence the enthalpy equation of the products is the same as well. Therefore the overall enthalpy equation for the converting process with feed3 type of matte at1325°C in this model is;

$$\begin{aligned}- 122190 = &-\text{Air}^{*}0 - \text{Oxygen}^{*}0 - \text{Reverts}^{*}\ {-0.986} - \text{flux}^{*}\ {-15.23} - \text{Coal}^{*}\ {-2.62}\\&+ Cu_2S\text{ in matte}^{*}0.30 + Ni_2S\text{ in matte}^{*}\ 0.33 + FeS\text{ in matte}^{*}0.19\\&+ CoS\text{ in matte}^{*}\ 0.46 + FeO\text{ in slag}^{*}\ {-2.33} + SiO_2\text{ in slag}^{*}{-13.60}\\&+CaO\text{ in slag}^{*}{-8.77} + MgO\text{ in slag}^{*}{-11.85} + Al_2O_3\text{ in slag}^{*}{-}\ 13.80\\&+ Cr_2O_3\text{ in slag}^{*}{-5.15} + Fe_3O_4\text{ in slag}^{*}{-3.09} + NiO\text{ in slag}^{*}{-1.44}\\&+ Cu_2O\text{ in slag}^{*}{-0.0023} + CoO\text{ in slag}^{*}{-1.43} + N_2\text{ in offgas}^{*}1.50\\&+ SO_2\text{ in offgas}^{*}{-3.56} + CO_2\text{ in offgas}^{*}{-7.41}\\&+ H_2O\text{ in offgas}^{*}{-10.50} + 48668\end{aligned}\tag{4.38}$$

The equations 4.28 to 4.38 are incorporated into the calculation matrix 4.30 resulting in the new modified matrix indicated in table 4.14 for the operating scenario in this section. The altered equations are represented in rows 1, 2, 3, 6, 7, 10, 11, 12, 13, 14, 15 and 30 in the calculation matrix which reflect the new composition and enthalpy of the feed matte.

Table 4-14: Calculation matrix for simulation 4.30

Balance	Value	Lance Air	Coal carrier Air	Matte carrier Air	Total Lance Air	Ind. Oxygen	Silica Flux	Lance Coal	Water in Matte	Reverts Feed	Cu2S	Ni2S	CoS	FeS	Mass Matte	FeO	SiO2	CaO	MgO	Al2O3	Cr2O3	Fe3O4	NiO	Cu2O	CoO	Mass Slag	N2	SO2	CO2	H2O	Heat Loss
Cu	2814	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.288	0.800	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	12180	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	-0.032	0.000	0.000	0.000	0.640	0.000	0.780	0.000	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
S	8558	0.000	0.000	0.000	0.000	0.000	0.000	-0.016	0.000	-0.218	0.200	0.214	0.350	0.360	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.500	0.000	0.000	0.000
O	0	-0.230	-0.230	-0.230	0.000	-0.991	0.000	-0.055	-0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.220	0.000	0.000	0.000	0.000	0.000	0.280	0.210	0.110	0.210	0.000	0.000	0.500	0.730	0.890	0.000
N	0	-0.770	-0.770	-0.770	0.000	-0.009	0.000	-0.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000
Co	165	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	0.000	0.650	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000
Ni	4778	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.445	0.000	0.786	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
C	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.709	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.270	0.000	0.000
H	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.035	-0.110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.110	0.000
SiO2	4925	0.000	0.000	0.000	0.000	0.000	-0.980	-0.098	0.000	-0.010	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al2O3	473	0.000	0.000	0.000	0.000	0.000	-0.020	-0.062	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
CaO	284	0.000	0.000	0.000	0.000	0.000	0.000	-0.005	0.000	-0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Cr2O3	662	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MgO	165	0.000	0.000	0.000	0.000	0.000	0.000	-0.003	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
H2O	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Reverts	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Coal Carrier Air	643	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Carrier Air	3861	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Total Lance Air	0.00	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Grade	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.6400	0.0278	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Slag Comp	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.780	-1.800	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Blast O2 Enrich	0.00	0.000	0.000	0.000	-0.461	1.292	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ni Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.790	0.000	0.000	0.0318	0.000	0.000	0.000	0.000	0.000
Cu Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.899	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Co Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.823	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.316	0.000	0.000	0.000	0.000	0.000	0.000
Slag Fe Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.083	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Mass Slag	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000
Mass Matte	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Heat Loss MJ/hr	48668	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000
Enthalpy(MJ)	-122190	0.000	0.000	0.000	0.000	0.000	15.210	2.620	15.880	0.960	0.305	0.330	0.464	0.190	0.000	-2.333	-13.600	-8.773	-11.849	-13.796	-5.152	-3.092	-1.444	-0.002	-1.428	0.000	1.495	-3.558	-7.410	-10.501	1.000

4.3.2 Calculation results and discussion

The solution of the calculation matrix in table 4.14 gives the results presented in table 4.15 for the unknown variables at the specified operating conditions in this section. Tables 4.16, 4.17 and 4.18 and figures 4.5 and 4.6 are the analysis of the information from table 4.15 which shows the process operating parameters required to produce products of the specified quality.

Table 4-15: Results of calculation matrix table 4.14

Lance Air	=	19333	Kg
Coal carrier Air	=	643	Kg
Matte carrier Air	=	3861	Kg
Total Lance Air	=	23837	Kg
Ind. Oxygen	=	8508	Kg
Silica Flux	=	1597	Kg
Diesel Fuel	=	1539	Kg
Water in Matte	=	0	Kg
Reverts Feed	=	0	Kg
CM Cu_2S	=	2990	Kg
CM Ni_2S	=	5045	Kg
CM CoS	=	61	Kg
CM FeS	=	367	Kg
Mass Matte	=	8463	Kg
slag FeO	=	13024	Kg
Slag SiO_2	=	6640	Kg
Slag CaO	=	291	Kg
Slag MgO	=	169	Kg
Slag Al_2O_3	=	599	Kg
Slag Cr_2O_3	=	661	Kg
Slag Fe_3O_4	=	2490	Kg
Slag NiO	=	1028	Kg
Slag Cu_2O	=	474	Kg
Slag CoO	=	158	Kg
Mass Slag	=	25535	Kg
Offgas N_2	=	18452	Kg
Offgas SO_2	=	13501	Kg
Offgas CO_2	=	4040	Kg
Offgas H_2O	=	490	Kg
Heat Loss	=	48668	MJ

Table 4-16: Predicted process parameters for simulation 4.3

Parameter	Section 4.1	Section 4.3	Industrial Actual	Units	% Change	%Accuracy
Matte feed rate	35	35	35	Nm3/hr	0.00	100.00
Lance Air	16253	15022	15780	Nm3/hr	-7.57	95.20
Oxygen	6362	5966	6040	Nm3/hr	-6.23	98.77
Lance Fuel	1.27	1.54	1.70	tph	20.98	90.55
Flux	5.99	1.60	2.20	tph	-73.35	72.60
O2 in blast	40.00	40	40.00	%	0.00	100.00
Slag Fe/SiO2	1.80	1.80	1.73		0.00	95.95
Oxygen/Matte	245	215	190	Nm3/tonne matte	-12.19	86.84
Flux/ Matte	171	46	62	Kg/tonne matte	-73.35	73.60
Matte Fall	28	24	26	% of feed matte	-12.75	93.00
Slag Fall	81	73	78	% of feed matte	-9.41	93.54
Lance Air	464	429	469	Nm3/tonne matte	-7.57	91.53

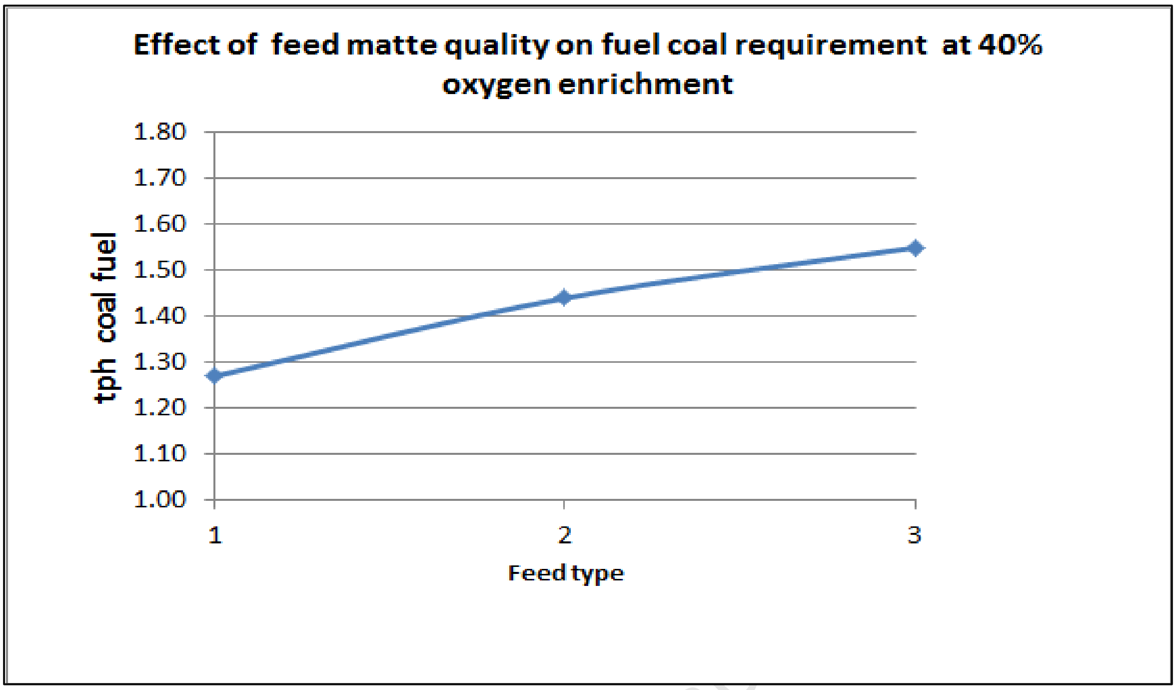


Figure 4.4: Fuel coal for different matte types

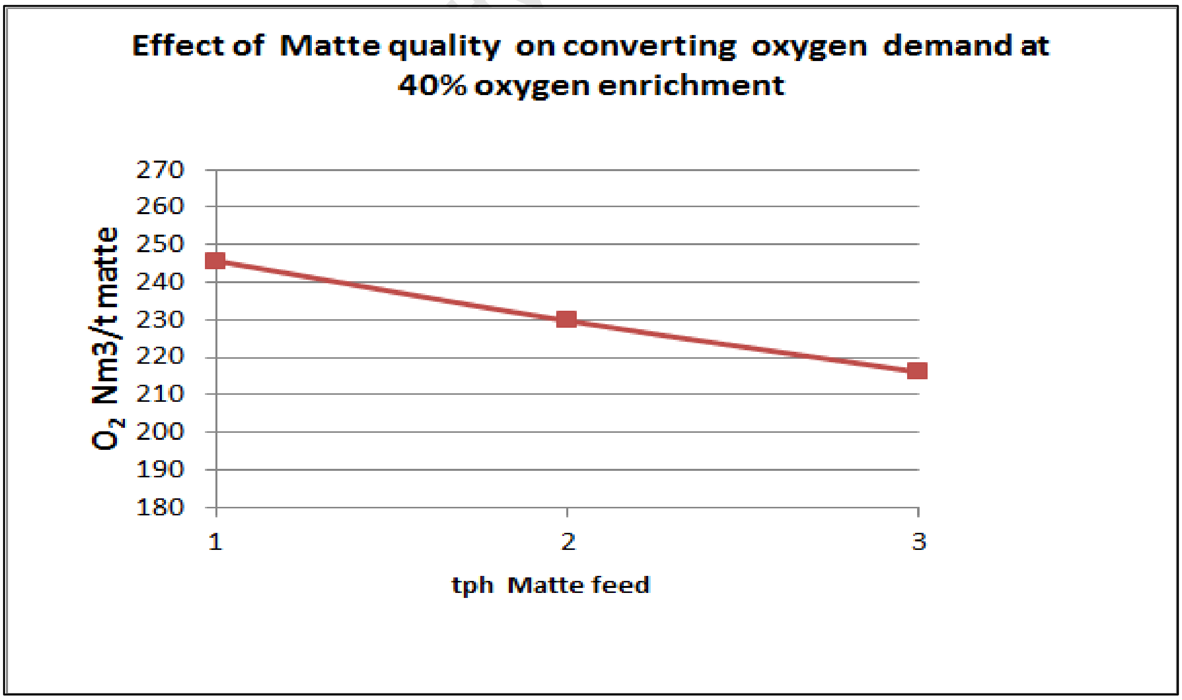


Figure 4.5: Specific oxygen requirement and SO2 composition for different matte types

Table 4-17: Converter matte quality for simulation 4.3

	%Cu	%Ni	%Co	%Fe	%S
Model Prediction	28.26	47.10	0.47	2.78	21.40
Industry Actual	27.42	46.22	0.34	2.80	21.67
% Accuracy	96.92	98.11	62.82	99.18	98.74

Table 4-18: Converter slag quality for simulation 4.3

	%Cu	%Ni	%Co	%Fe	%Fe_3O_4	%SiO_2	%CaO	%MgO	%Al_2O_3	%Cr_2O_3	Fe/SiO2
Model Prediction	1.65	3.18	0.49	46.80	9.75	26.00	1.14	0.66	2.35	2.59	1.80
Industry Actual	1.55	3.47	0.40	47.55	13.50	26.89	0.93	0.34	1.67	2.05	1.77
% Accuracy	93.26	91.64	77.59	98.43		96.70	77.31	5.74	59.44	73.63	98.21

Comparison of the results for the process treating feed1 in section 4.10 and feed3 in this section under the same process conditions of 35tph, 40% oxygen enrichment, 1325°C bath temperatures indicates that;

- The fuel coal requirement increases from 36.2 to 44.3kg/tonne feed matte (1.27-1.54tph) when treating feed3 type of matte as indicated in figure 4.4 and table 4.16. The increase in fuel requirement is due to increase in gangue material SiO_2 a heat consumer and decrease in the in base FeS in the feed matte (Davenport 2002) which generates heat. This fuel coal requirement increase can be minimised by increasing the oxygen in the blast above 40%. The matte and slag chemical compositions indicated in tables 4.17 and 4.18 are in agreement with the actual plant information reported by Jones 1999 and data obtained from the relevant smelter.
- Figure 4.5 and table 4.16 indicates that the specific oxygen requirement for matte oxidation decreases from 245 to 215Nm^3/tonne matte. The reduction is mainly due to reduction in composition of base metals mainly FeS in feed3 matte. The converter matte fall produced reduces from 28 to 24% due to reduced base metals in the feed3 matte.

- The flux requirement to produce the slag of the defined slag quality of Fe/SiO_2 of 1.80 decreases significantly from 172Kg/tonne indicated in section 4.10 to 46 kg/tonne feed matte (5.99 -1.60tph) as can be seen in table 4.16. The reduction is due to increased SiO_2 content and reduced FeS in feed3 matte. The predicted slag to be produced reduces from 81 to 73% due to reduced content of FeS in the feed matte.

It has been demonstrated that the change in the feed matte chemistry can be accommodated by modifying the calculation matrix. Only the numerical term column of the matrix is affected by changes in the feed matte composition and the matte enthalpy term of the feed matte.

The simulation has demonstrated that the change in the matte feed quality requires that the operator of the converting process changes the specific consumption factors of oxygen, flux and fuel coal feed in order to keep the converting process at steady state. This section only considered the change in chemical composition of the elements and compounds of feed matte listed in table 4.1 to illustrate the model predictive capabilities. Other minerals that are not indicated or present in the feed matte or flux can readily be incorporated in the calculation matrix as demonstrated.

4.4 Effect of matte feed rate on fuel requirement

The TSL furnace can be run at a different matte feed rate as required at any particular time. This section examine the effect of operating the TSL converter at a higher matte feed rate of 45tph on the process input requirement fuel coal, silica flux, and oxygen and the SO_2 concentration in the offgas. The simulated process conditions in this section are the same as in section 4.10 and the only adjusted variable is the matte feed rate from 35tph to 45tph. As highlighted in section 4.10 all process inputs are at the temperature of 25^oC and all outputs are at 1325^oC. The slag chemistry and matte quality are the same as in the model given in table 4.3. Two immediate consequences of operating at higher matte feed rates are;

- Increased Fe and S oxidation rates leading to increase in heat generation.
- Increased or decreased oxygen requirement both from air and industrial oxygen

4.4.1 Calculation Matrix

As explained in the previous sections the model is used to calculate masses of the process inputs and outputs fuel coal, flux, oxygen, air, converter matte, slag and the offgas.

The change in this matrix model is the input masses of the components Cu, Ni, Co, Fe, S, SiO_2, CaO, MgO, Al_2O_3 and Cr_2O_3 in feed matte and the enthalpies of the feed matte as indicated in the following mass and heat balance equations 4.39 to 4.49.

$$4221 = 0.8 * Cu_2S + 0.89 * Cu_2O \quad (4.39)$$

$$6948 = 0.78 * Ni_2S + 0.79 * NiO \quad (4.40)$$

$$194 = 0.65 * CoS + 0.79 * CoO \quad (4.41)$$

$$17825 = -0.00476 * Coal + 0.64 * FeS + 0.78 * FeO + 0.72 * Fe_3O_4 \quad (4.42)$$

$$12605 = -0.0166 * Coal + 0.36 * FeS + 0.22 * Ni_2S + 0.2 * Cu_2S + 0.35 * CoS + 0.5 * SO_2 \quad (4.43)$$

$$2003 = -0.96 * Flux - 0.0928 * Coal + 1.0 * SiO_2 \quad (4.44)$$

$$289 = -0.04*Flux - 0.0618*Coal + 1.0*Al_2O_3 \tag{4.45}$$

$$73 = -0.0051*Coal + 1.0*CaO \tag{4.46}$$

$$113 = -0.0028*Coal + 1.0* MgO \tag{4.47}$$

$$729 = 1.0* Cr_2O_3 \tag{4.48}$$

The total overall heat input into the converting process is,

$$\sum H_{reactants} = - 85735 + Air*0 + Oxygen*0 + Reverts* -0.960 + Flux* -15.13 + Coal* -2.62$$

The enthalpies of the products are;

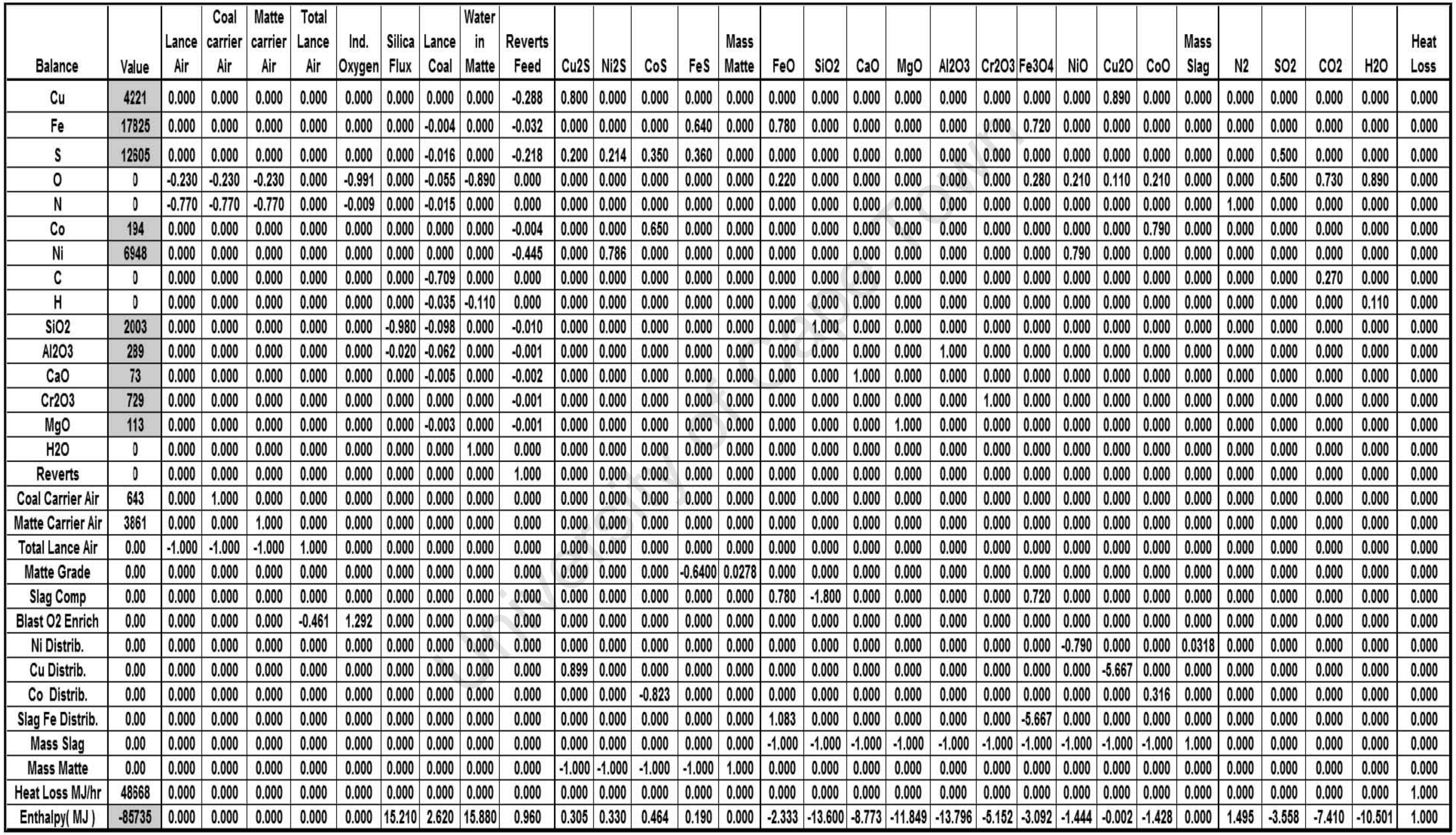
$$\begin{aligned}\sum Hproducts = {} & Cu_2S \text{ in matte}*0.30 + Ni_2S \text{ in matte}* 0.33 + FeS \text{ in matte}*0.19 \\ & + CoS \text{ in matte}* 0.46 + FeO \text{ in slag}* -2.33 + SiO_2 \text{ in slag}*-13.60 \\ & + CaO \text{ in slag}*-8.77 + MgO \text{ in slag}*-11.85+ Al_2O_3 \text{ in slag}*-13.80 \\ & + Cr_2O_3 \text{ in slag}*-5.15 + Fe_3O_4 \text{ in slag}*-3.09 + NiO \text{ in slag}*-1.44 \\ & + Cu_2O \text{ in slag}*-0.0023 + CoO \text{ in slag}*-1.43+ N_2 \text{ in offgas}*1.50 \\ & + SO_2 \text{ in offgas}*-3.56 + CO_2 \text{ in offgas}*-7.41 + H_2O \text{ in offgas}*-10.50 \\ & + 48668\end{aligned}$$

And hence, the overall enthalpy balance is given as;

$$\begin{aligned}- 85735 = {} & -Air*0 - Oxygen*0 - Reverts* -0.986 - flux* -15.23 \\ & - Coal* -2.62 + Cu_2S \text{ in matte}*0.30 + Ni_2S \text{ in matte}* 0.33 \\ & + FeS \text{ in matte}*0.19+ CoS \text{ in matte}* 0.46 + FeO \text{ in slag}* -2.33 \\ & + SiO_2 \text{ in slag}*-13.60+ CaO \text{ in slag}*-8.77 \\ & + MgO \text{ in slag}*-11.85 + Al_2O_3 \text{ in slag}*-13.80 \\ & + Cr_2O_3 \text{ in slag}*-5.15 + Fe_3O_4 \text{ in slag}*-3.09 \\ & + NiO \text{ in slag}*-1.44+ Cu_2O \text{ in slag}*-0.0023 \\ & + CoO \text{ in slag}*-1.43 + N_2 \text{ in offgas}*1.50 \\ & + SO_2 \text{ in offgas}*-3.56 + CO_2 \text{ in offgas}*-7.41 \\ & + H_2O \text{ in offgas}*-10.50 + 48668\end{aligned} \tag{4.49}$$

Equations 4.39 to 4.49 are incorporated into the calculation matrix as shown in table 4.68. The change in the calculation matrix is the numerical terms for Cu, Ni, Co, Fe, S, SiO_2, CaO, MgO, Al_2O_3, Cr_2O_3 and the enthalpy equations to reflect the new values at the new matte feed rate of 45tph.

Table 4-19: Process Calculation matrix for simulation 4.4

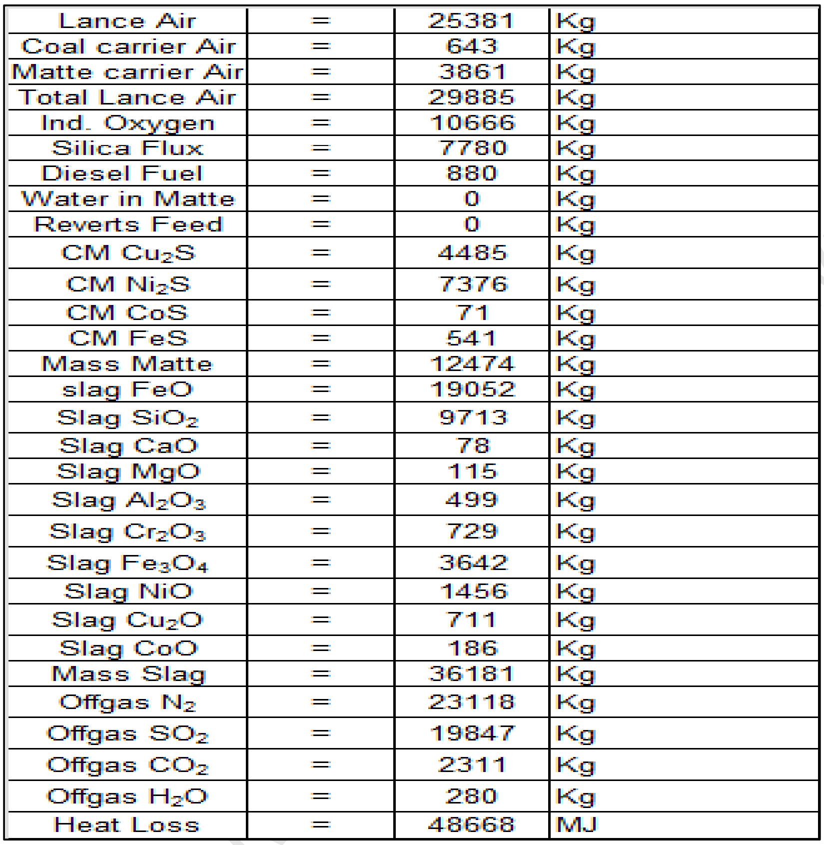


Balance	Value	Lance Air	Coal carrier Air	Matte carrier Air	Total Lance Air	Ind. Oxygen	Silica Flux	Lance Coal	Water in Matte	Reverts Feed	Cu_2S	Ni_2S	CoS	FeS	Mass Matte	FeO	SiO_2	CaO	MgO	Al_2O_3	Cr_2O_3	Fe_3O_4	NiO	Cu_2O	CoO	Mass Slag	N_2	SO_2	CO_2	H_2O	Heat Loss
Cu	4221	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.288	0.800	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	17825	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	-0.032	0.000	0.000	0.000	0.640	0.000	0.780	0.000	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
S	12605	0.000	0.000	0.000	0.000	0.000	0.000	-0.016	0.000	-0.218	0.200	0.214	0.350	0.360	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.500	0.000	0.000	0.000
O	0	-0.230	-0.230	-0.230	0.000	-0.991	0.000	-0.055	-0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.220	0.000	0.000	0.000	0.000	0.000	0.280	0.210	0.110	0.210	0.000	0.000	0.500	0.730	0.890	0.000
N	0	-0.770	-0.770	-0.770	0.000	-0.009	0.000	-0.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000
Co	194	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	0.000	0.650	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000
Ni	6948	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.445	0.000	0.786	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
C	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.709	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.270	0.000	0.000
H	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.035	-0.110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.110	0.000
SiO_2	2003	0.000	0.000	0.000	0.000	0.000	-0.980	-0.098	0.000	-0.010	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al_2O_3	289	0.000	0.000	0.000	0.000	0.000	-0.020	-0.062	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
CaO	73	0.000	0.000	0.000	0.000	0.000	0.000	-0.005	0.000	-0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Cr_2O_3	729	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MgO	113	0.000	0.000	0.000	0.000	0.000	0.000	-0.003	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
H_2O	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Reverts	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Coal Carrier Air	643	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Carrier Air	3861	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Total Lance Air	0.00	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Grade	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.6400	0.0278	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Slag Comp	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.780	-1.800	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Blast O2 Enrich	0.00	0.000	0.000	0.000	-0.461	1.292	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ni Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.790	0.000	0.000	0.0318	0.000	0.000	0.000	0.000	0.000
Cu Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.899	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Co Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.823	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.316	0.000	0.000	0.000	0.000	0.000	0.000
Slag Fe Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.083	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Mass Slag	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000
Mass Matte	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Heat Loss MJ/hr	48668	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000
Enthalpy(MJ)	-85735	0.000	0.000	0.000	0.000	0.000	15.210	2.620	15.880	0.960	0.305	0.330	0.464	0.190	0.000	-2.333	-13.600	-8.773	-11.849	-13.796	-5.152	-3.092	-1.444	-0.002	-1.428	0.000	1.495	-3.558	-7.410	-10.501	1.000

4.42 Calculation results and discussion

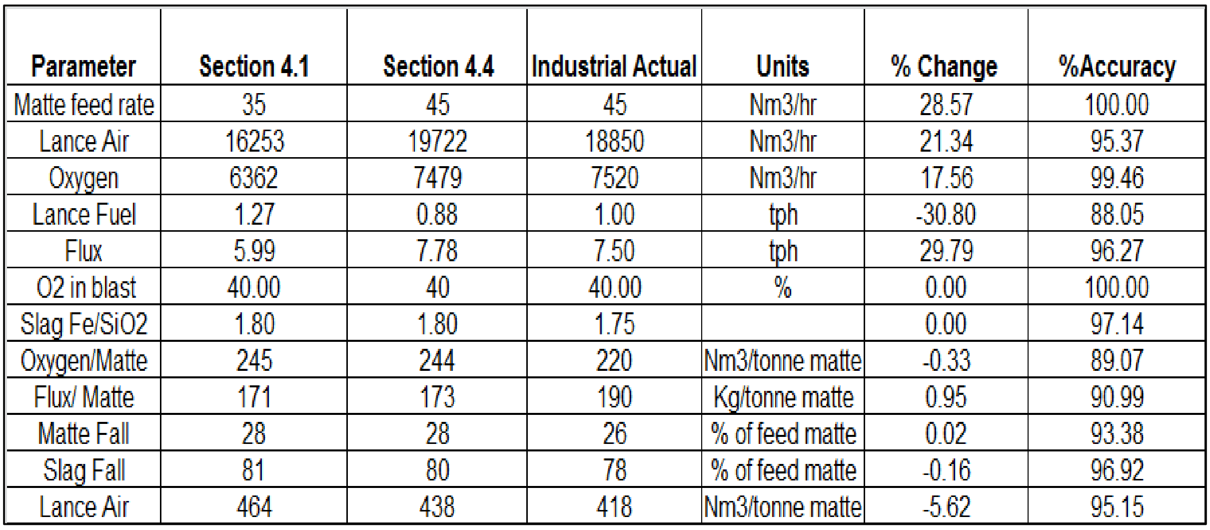
Table 4.20 indicates the values of the unknown variables of the process calculated from the matrix in table 4.19. Figure 4.6 indicate the fuel feed rate requirement and tables 4.21 and 4.22 indicate the predicted assays of matte and slag analysed from table 4.20.

Table 4-20: Results of calculation matrix table 4.19



Lance Air	=	25381	Kg
Coal carrier Air	=	643	Kg
Matte carrier Air	=	3861	Kg
Total Lance Air	=	29885	Kg
Ind. Oxygen	=	10666	Kg
Silica Flux	=	7780	Kg
Diesel Fuel	=	880	Kg
Water in Matte	=	0	Kg
Reverts Feed	=	0	Kg
CM Cu_2S	=	4485	Kg
CM Ni_2S	=	7376	Kg
CM CoS	=	71	Kg
CM FeS	=	541	Kg
Mass Matte	=	12474	Kg
slag FeO	=	19052	Kg
Slag SiO_2	=	9713	Kg
Slag CaO	=	78	Kg
Slag MgO	=	115	Kg
Slag Al_2O_3	=	499	Kg
Slag Cr_2O_3	=	729	Kg
Slag Fe_3O_4	=	3642	Kg
Slag NiO	=	1456	Kg
Slag Cu_2O	=	711	Kg
Slag CoO	=	186	Kg
Mass Slag	=	36181	Kg
Offgas N_2	=	23118	Kg
Offgas SO_2	=	19847	Kg
Offgas CO_2	=	2311	Kg
Offgas H_2O	=	280	Kg
Heat Loss	=	48668	MJ

Table 4-21: Process parameters for simulation 4.4



Parameter	Section 4.1	Section 4.4	Industrial Actual	Units	% Change	%Accuracy
Matte feed rate	35	45	45	Nm3/hr	28.57	100.00
Lance Air	16253	19722	18850	Nm3/hr	21.34	95.37
Oxygen	6362	7479	7520	Nm3/hr	17.56	99.46
Lance Fuel	1.27	0.88	1.00	tph	-30.80	88.05
Flux	5.99	7.78	7.50	tph	29.79	96.27
O2 in blast	40.00	40	40.00	%	0.00	100.00
Slag Fe/SiO2	1.80	1.80	1.75		0.00	97.14
Oxygen/Matte	245	244	220	Nm3/tonne matte	-0.33	89.07
Flux/ Matte	171	173	190	Kg/tonne matte	0.95	90.99
Matte Fall	28	28	26	% of feed matte	0.02	93.38
Slag Fall	81	80	78	% of feed matte	-0.16	96.92
Lance Air	464	438	418	Nm3/tonne matte	-5.62	95.15

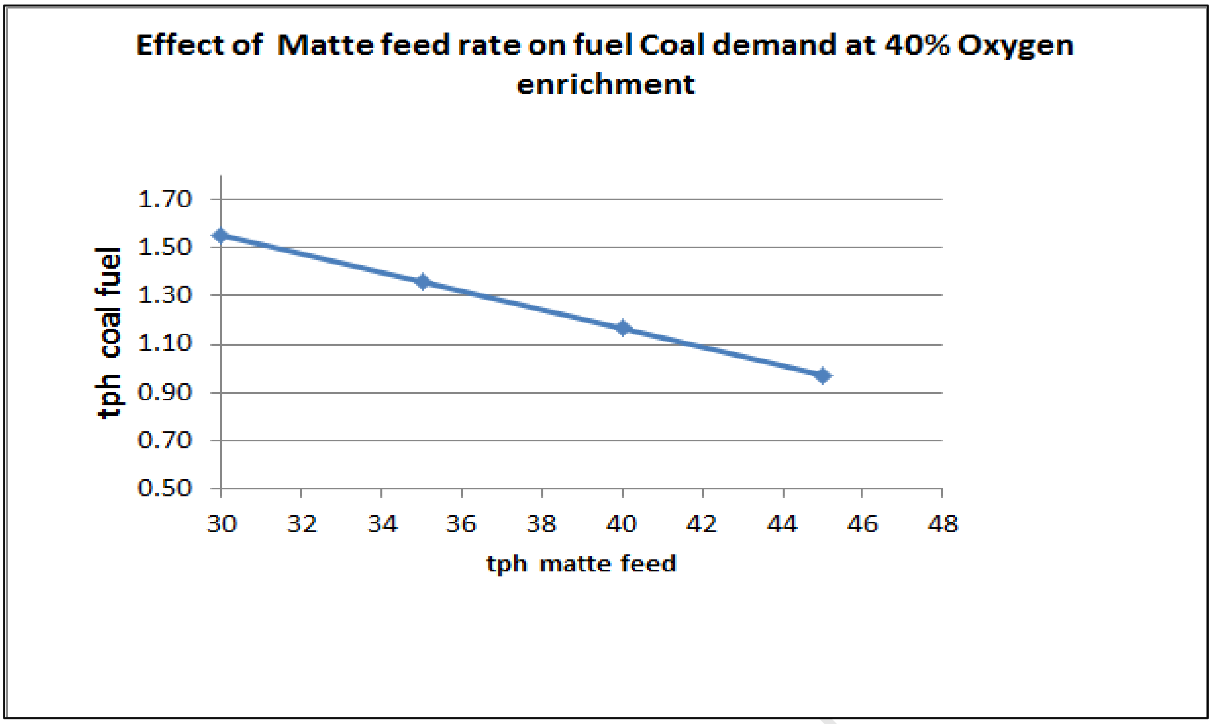


Figure 4.6: Fuel coal and lance air requirement and offgas volume for simulation 4.4

Table 4-22: Converter matte quality for simulation 4.4

	%Cu	%Ni	%Co	%Fe	%S
Model Prediction	28.76	46.72	0.37	2.78	21.37
Industry Actual	27.42	46.22	0.34	2.80	21.67
% Accuracy	95.10	98.93	90.52	99.18	98.62

Table 4-23: Converter slag quality for simulation 4.4

	%Cu	%Ni	%Co	%Fe	%Fe_3O_4	%SiO_2	%CaO	%MgO	%Al_2O_3	%Cr_2O_3	Fe/SiO2
Model Prediction	1.75	3.18	0.41	48.32	10.07	26.84	0.22	0.32	1.38	2.01	1.80
Industry Actual	1.55	3.47	0.40	47.55	13.50	26.89	0.93	0.34	1.67	2.05	1.77
% Accuracy	87.00	91.64	98.38	98.38		99.83	23.13	93.39	82.59	98.29	98.21

From table 4.21, it is indicated that the increase in the matte feed rate leads to the increase in lance air, oxygen and flux requirement due to the increase in the rate of FeS oxidation in the feed matte. In figure 4.6 it is shown that the lance fuel coal requirement decreases significantly from 36.3 to 19.6Kg per tonne matte (1.27-0.88tph) due to the increased rate of oxidation of FeS that generates some extra heat (Matthew 2009). Tables 4.21 and 4.22 indicate the matte and slag assays predicted by the model at the process conditions mentioned in this section and are similar to the assays reported in section 4.1 which are very close to actual plant assays reported by Jones 1999 and Matthew 2009.

This section has indicated that the model is able to predict what would happen to the process with respect to changing the matte feed rate. The matrix calculation of this section has demonstrated that increasing the matte feed rate results in a decrease in fuel coal requirement and increases in lance air, oxygen and flux feed rates. One independent parameter that the operator must change is the fuel coal feed rate as the lance air, oxygen and flux feed rated are automatically increased by the control system using the specific consumption factors that are linked to the matte feed rate.

4.5 Influence of process operating temperature

TSL furnace bath temperature can be varied from 1250 to 1375°C (Davenport 2002). The TSL furnace can be operated at a high bath temperature as required to attain certain objectives. Extra energy must be supplied when a furnace is operated at a high temperature so that some benefit must be gained to justify the practice which may include improving slag fluidity and improved metal recovery. The purpose of this section is to show how the TSL converter fuel and oxygen requirements are affected by operating at a high bath temperature. Compared to section 4.10, the only variable changed in this section is the temperature of the process products components in converter matte, slag and the offgas to a higher temperature from 1325°C to 1375°C.

4.5.1 Calculation Matrix model

All equations in this section are the same as in section 4.10 except the enthalpy balance equation 4.26 which represent the numbers in row number 30 in the calculation matrix table 4.03.

In the enthalpy equation, the enthalpy of the process inputs remain the same as in model 4.10 and only the enthalpies of the product changes due to the increase in the products temperature from 1325°C to 1375°C.

The total enthalpy of the process inputs at 1325°C are;

$$\sum \text{Hreactants} = -66683 + \text{Air}*0 + \text{Oxygen}*0 + \text{Reverts}*-0.960 + \text{Flux}*-15.13 + 0*-15.88 + \text{Coal}*-2.62$$

The enthalpies of the products from the process at 1375°C are;

$$\begin{aligned}\sum \text{Hproducts} = {} & Cu_2S \text{ in matte}*H1375 + Ni_2S \text{ in matte}*H1375 \\ & + FeS \text{ in matte}*H1375 + CoS \text{ in matte}*H1375 \\ & + FeO \text{ in slag}*H1375 + SiO_2 \text{ in slag}*H1375 + CaO \text{ in slag}*H1375 \\ & + MgO \text{ in slag}*H1375 + Al_2O_3 \text{ in slag}*H1375 + Cr_2O_3 \text{ in slag}*H1375 \\ & + Fe_3O_4 \text{ in slag}*H1375 + NiO \text{ in slag}*H1375 \\ & + Cu_2O \text{ in slag}*H1375 + CoO \text{ in slag}*H1375 + N_2 \text{ in offgas}*H1375\end{aligned}$$

$$\begin{aligned}
&+ \text{SO}_2 \text{ in offgas*H1375} + \text{CO}_2 \text{ in offgas*H1375}\\
&+ \text{H}_2\text{O in offgas*H1375+ Hloss}
\end{aligned}$$

The overall modified enthalpy equation of the process operating at the temperature of 1375^{o}C is indicated by equation 4.50;

$$\begin{aligned}
- 66683 = &-\text{Air*0 - Oxygen*0 - Reverts* -0.960 - flux* -15.21 - Coal* -2.62}\\
&- \text{0*-15.88+ Cu}_2\text{S in matte*0.33 + Ni}_2\text{S in matte* 0.37}\\
&+ \text{FeS in matte*0.23 + CoS in matte* 0.50 + FeO in slag* -2.29}\\
&+ \text{SiO}_2\text{ in slag*-13.53 + CaO in slag*-8.72 + MgO in slag*-11.77}\\
&+ \text{Al}_2\text{O}_3\text{ in slag*-13.73 + Cr}_2\text{O}_3\text{ in slag*-5.11 + Fe}_3\text{O}_4\text{ in slag*-3.05}\\
&+ \text{NiO in slag*-1.41 + Cu}_2\text{O in slag*0.03 + CoO in slag*-1.39}\\
&+ \text{N}_2\text{ in offgas*1.56 + SO}_2\text{ in offgas*-3.51 + CO}_2\text{ in offgas*-7.34}\\
&+ \text{H}_2\text{O in offgas*-10.37 + 48668}
\end{aligned} \tag{4.50}$$

The calculation matrix in section 4.10 is modified by replacing the enthalpy equation in row number 30 with the new equation 4.50. The modified calculation matrix is indicated in table 4.24.

Table 4-24: Process calculation matrix for simulation 4.5

Balance	Value	Lance Air	Coal carrier Air	Matte carrier Air	Total Lance Air	Ind. Oxygen	Silica Flux	Lance Coal	Water in Matte	Reverts Feed	Cu2S	Ni2S	CoS	FeS	Mass Matte	FeO	SiO2	CaO	MgO	Al2O3	Cr2O3	Fe3O4	NiO	Cu2O	CoO	Mass Slag	N2	SO2	CO2	H2O	Heat Loss
Cu	3283	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.288	0.800	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	13864	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	-0.032	0.000	0.000	0.000	0.640	0.000	0.780	0.000	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
S	9804	0.000	0.000	0.000	0.000	0.000	0.000	-0.016	0.000	-0.218	0.200	0.214	0.350	0.360	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.500	0.000	0.000	0.000
O	0	-0.230	-0.230	-0.230	0.000	-0.991	0.000	-0.055	-0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.220	0.000	0.000	0.000	0.000	0.000	0.280	0.210	0.110	0.210	0.000	0.000	0.500	0.730	0.890	0.000
N	0	-0.770	-0.770	-0.770	0.000	-0.009	0.000	-0.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000
Co	151	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	0.000	0.650	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000
Ni	5404	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.445	0.000	0.786	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
C	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.709	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.270	0.000	0.000
H	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.035	-0.110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.110	0.000
SiO2	1558	0.000	0.000	0.000	0.000	0.000	-0.980	-0.098	0.000	-0.010	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al2O3	225	0.000	0.000	0.000	0.000	0.000	-0.020	-0.062	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
CaO	57	0.000	0.000	0.000	0.000	0.000	0.000	-0.005	0.000	-0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Cr2O3	567	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MgO	88	0.000	0.000	0.000	0.000	0.000	0.000	-0.003	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
H2O	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Reverts	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Coal Carrier Air	643	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Carrier Air	3861	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Total Lance Air	0.00	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Grade	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.6400	0.0278	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Slag Comp	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.780	-1.800	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Blast O2 Enrich	0.00	0.000	0.000	0.000	-0.461	1.292	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ni Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.790	0.000	0.000	0.0318	0.000	0.000	0.000	0.000	0.000
Cu Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.899	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Co Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.823	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.316	0.000	0.000	0.000	0.000	0.000	0.000
Slag Fe Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.083	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Mass Slag	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000
Mass Matte	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Heat Loss MJ/hr	48668	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000
Enthalpy(MJ)	-66683	0.000	0.000	0.000	0.000	0.000	15.210	2.620	15.880	0.960	0.333	0.370	0.497	0.226	0.000	-2.286	-13.529	-8.717	-11.766	-13.725	-5.106	-3.046	-1.408	0.033	-1.388	0.000	1.558	-3.512	-7.343	-10.367	1.000

4.5.2 Calculation results and discussion

The results of the calculation matrix table 4.24 are not indicated as the numbers are very similar to the ones in table 4.4. The tables 4.25 indicate analysed parameters from the results obtained from the calculation matrix table 4.24. The graph in figure 4.8 shows the effect of operating the converting process at different bath temperatures on the lance fuel coal requirement.

Table 4-25: Predicted process parameters for simulation 4.5

Parameter	Section 4.1	Section 4.5	Industrial Actual	Units	% Change	%Accuracy
Matte feed rate	35	35	35	Nm3/hr	0.00	100.00
Lance Air	16253	16904	16650	Nm3/hr	4.01	98.47
Oxygen	6362	6572	6430	Nm3/hr	3.30	97.80
Lance Fuel	1.27	1.50	1.50	tph	17.78	99.90
Flux	5.99	5.97	6.54	tph	-0.37	91.31
O2 in blast	40.00	40	40.00	%	0.00	100.00
Slag Fe/SiO2	1.80	1.80	1.82		0.00	98.90
Oxygen/Matte	245	245	230	Nm3/tonne matte	0.13	93.41
Flux/ Matte	171	171	186	Kg/tonne matte	-0.37	91.74
Matte Fall	28	28	26	% of feed matte	-0.01	93.41
Slag Fall	81	81	78	% of feed matte	0.06	96.68
Lance Air	464	483	476	Nm3/tonne matte	4.01	98.47

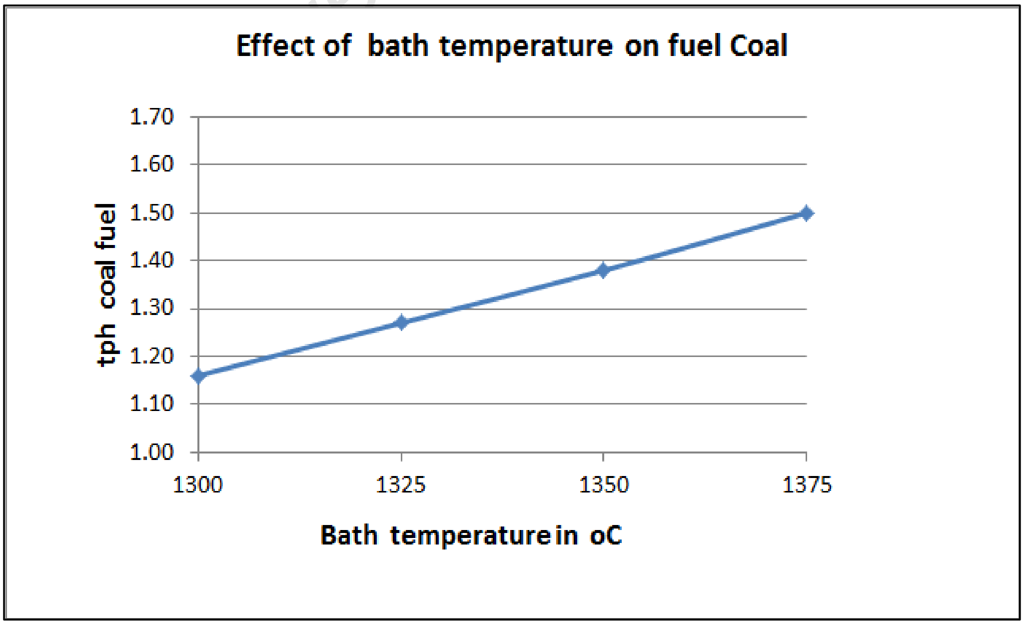


Figure 4.7: Fuel coal requirement for simulation 4.5

Examination of figure 4.8 and table 4.25 shows that the fuel coal requirement increases from 36.3 to 43kg/tonne feed matte (1.27-1.5tph) due to operating the process at a high temperature. Table 4.25 shows that the fuel feed rate is the major parameter required to be changed when the process operating temperature is changed for some reason at the fixed oxygen enrichment. The fuel required can be reduced if the lance oxygen enrichment for the process is increased to levels higher than 40%. The assays of converter matte and slag are not indicated in this section but are similar with the ones reported in the previous sections.

The model has shown that the effect of changing the process operating temperature can be predicted by the model by changing enthalpy equation in the calculation matrix 4.3. The analysed simulation results in table 4.25 and figure 4.8 has indicated that the fuel requirement for producing matte and slag of the desired quality can be minimised when operating the furnace at lower temperatures consistently with adequate slag fluidity (Davenport 2002). Low temperature operation is necessitated by choosing the slag compositions with low freezing points and high fluidities and by treating matte and other materials that are less contaminated by oxide minerals (Davenport 2002). In this simulation the converter operator only needs to change the lance fuel coal feed rate if it is required to change the operating temperature of the process assuming that the feed matte and flux assays and oxygen enrichment do not change (Matthew 2009, Davenport 2001).

4.6 Influence of moisture in feed matte

Liquid water is always present in most feed materials treated in the smelting furnaces. Water exerts a strong cooling effect on the furnace heat balance as the liquid must be vaporised and heated to the furnace offgas temperature and hence it is important that it is accounted for in the TSL smelting calculations. As will be seen from the model in this section, liquid water in the feed increase the energy requirement of the converting process to a large extent.

This section shows the effect of the water in the feed matte on the lance fuel and other process inputs requirement. The base data of section 4.10 has been used in this section and the only change made is the moisture content in the feed matte indicated in table 4.26 which is an extract from table 4.1. The feed matte in this section is feed4 matte in table 4.1 which contains 6.90% moisture which is basically wet feed1 matte.

Table 4-26: Moist feed

	%Cu	%Ni	%Co	%Fe	%S	%SiO_2	%CaO	%MgO	%Al_2O_3	%Cr_2O_3	H_2O	Total
Feed 4	8.73	14.38	0.40	36.88	26.08	4.14	0.15	0.23	0.60	1.51	6.90	100.00

4.6.1 Calculation Matrix

As a result of inclusion of a new compound in feed matte1 which is H_2O, some changes needs to be done to the calculation matrix. The changes are;

- Mass balance for Cu, Ni, Co, Fe, S, SiO_2, Al_2O_3, CaO, MgO, Cr_2O_3 and new input H_2O on the feed side.
- The enthalpy balance of the process due to inclusion of H_2O in the process inputs.

The mass balance of the components contained in wet feed matte becomes at the feed rate of 35tph are indicated in equations 4.51 to 4.61;

$$3057 = 0.8 * Cu_2S + 0.89 * Cu_2O \tag{4.51}$$

$$5031 = 0.78 * Ni_2S + 0.79 * NiO \tag{4.52}$$

$$140 = 0.65 * CoS + 0.79 * CoO \tag{4.53}$$

$$12908 = -0.00476 * Coal + 0.64 * FeS + 0.78 * FeO + 0.72 * Fe_3O_4 \tag{4.54}$$

$$9128 = -\ 0.0166*Coal + 0.36*FeS + 0.22*Ni_2S + 0.2*Cu_2S + 0.35*CoS + 0.5*SO_2 \quad 4.55)$$

$$1450 = -0.96*Flux - 0.0928*Coal + 1.0*SiO_2 \quad (4.56)$$

$$209 = -0.04*Flux - 0.0618*Coal + 1.0*Al_2O_3 \quad (4.57)$$

$$53 = -0.0051*Coal + 1.0*CaO \quad (4.58)$$

$$81 = -0.0028*Coal + 1.0*MgO \quad (4.59)$$

$$528 = 1.0*Cr_2O_3 \quad (4.60)$$

$$2415 = 1.0*H_2O \quad (4.61)$$

The inclusion of water in the feed matte and change in the rate of components in the feed matte, the enthalpies of the inputs to the furnace becomes;

$$\sum Hreactants = -\ 62082 + Air*0 + Oxygen*0 + Reverts* -0.960 + Flux* -15.21 + 2415* -15.88 + Coal* -2.62$$

The enthalpies of the products remain the same as in section 4.10 and therefore, the overall enthalpy equation is;

$$\begin{aligned}-62082 = {} & -Air*0 - Oxygen*0 - Reverts* -0.96 - flux* -15.21 \\ & -2592.7*-15.88 - Coal* -2.62 + Cu_2S\ in\ matte*0.30 + Ni_2S\ in\ matte*0.33 \\ & + FeS\ in\ matte*0.19 + CoS\ in\ matte*0.46 + FeO\ in\ slag* -2.33 \\ & + SiO_2\ in\ slag*-13.60 + CaO\ in\ slag*-8.77 + MgO\ in\ slag*-11.85 \\ & + Al_2O_3\ in\ slag*-13.80 + Cr_2O_3\ in\ slag*-5.15 + Fe_3O_4\ in\ slag*-3.09 \\ & + NiO\ in\ slag*-1.44 + Cu_2O\ in\ slag*-0.0023 + CoO\ in\ slag*-1.43 \\ & + N_2\ in\ offgas*1.50 + SO_2\ in\ offgas*-3.56 + CO_2\ in\ offgas*-7.41\end{aligned}$$

$$+ H_2O \text{ in offgas}^*-10.50 + 48668 \qquad (4.62)$$

The new mass balance equations 4.53 to 4.61 and the enthalpy equation 4.62 are inserted in the calculation matrix in table 4.3 and the modified calculation matrix that represent the operating conditions specified in this section is indicated in table 4.27.

Table 4-27: Process calculation matrix for simulation 4.6

Balance	Value	Lance Air	Coal carrier Air	Matte carrier Air	Total Lance Air	Ind. Oxygen	Silica Flux	Lance Coal	Water in Matte	Reverts Feed	Cu2S	Ni2S	CoS	FeS	Mass Matte	FeO	SiO2	CaO	MgO	Al2O3	Cr2O3	Fe3O4	NiO	Cu2O	CoO	Mass Slag	N2	SO2	CO2	H2O	Heat Loss
Cu	3057	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.288	0.800	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	12908	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	-0.032	0.000	0.000	0.000	0.640	0.000	0.780	0.000	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
S	9128	0.000	0.000	0.000	0.000	0.000	0.000	-0.016	0.000	-0.218	0.200	0.214	0.350	0.360	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.500	0.000	0.000	0.000
O	0	-0.230	-0.230	-0.230	0.000	-0.991	0.000	-0.055	-0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.220	0.000	0.000	0.000	0.000	0.000	0.280	0.210	0.110	0.210	0.000	0.000	0.500	0.730	0.890	0.000
N	0	-0.770	-0.770	-0.770	0.000	-0.009	0.000	-0.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000
Co	140	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	0.000	0.650	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000
Ni	5031	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.445	0.000	0.786	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
C	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.709	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.270	0.000	0.000
H	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.035	-0.110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.110	0.000
SiO2	1450	0.000	0.000	0.000	0.000	0.000	-0.980	-0.098	0.000	-0.010	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al2O3	209	0.000	0.000	0.000	0.000	0.000	-0.020	-0.062	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
CaO	53	0.000	0.000	0.000	0.000	0.000	0.000	-0.005	0.000	-0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Cr2O3	528	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MgO	81	0.000	0.000	0.000	0.000	0.000	0.000	-0.003	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
H2O	2415	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Reverts	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Coal Carrier Air	643	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Carrier Air	3861	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Total Lance Air	0.00	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Grade	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.6400	0.0278	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Slag Comp	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.780	-1.800	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Blast O2 Enrich	0.00	0.000	0.000	0.000	-0.461	1.292	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ni Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.790	0.000	0.000	0.0318	0.000	0.000	0.000	0.000	0.000
Cu Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.899	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Co Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.823	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.316	0.000	0.000	0.000	0.000	0.000	0.000
Slag Fe Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.083	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Mass Slag	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000
Mass Matte	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Heat Loss MJ/hr	48668	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000
Enthalpy(MJ)	-62082	0.000	0.000	0.000	0.000	0.000	15.210	2.620	15.880	0.960	0.305	0.330	0.464	0.190	0.000	-2.333	-13.600	-8.773	-11.849	-13.796	-5.152	-3.092	-1.444	-0.002	-1.428	0.000	1.495	-3.558	-7.410	-10.501	1.000

4.6.2 Calculation results and discussion

Table 4.28 and figures 4.8 and 4.9 indicates the parameters analysed from the results of calculation matrix in table 4.27. Table 4.28 indicates the predicted process parameters which are important in optimizing the process at the conditions mentioned in this section. The graphs in figures 4.8 and 4.9 illustrate the influence of the moisture content of the feed matte on the lance fuel coal and the specific oxygen requirement.

Table 4-28: Process operating parameters for simulation 4.6

Parameter	Section 4.1	Section 4.6	Industrial Actual	Units	% Change	%Accuracy
Matte feed rate	35	35	35	Nm3/hr	0.00	100.00
Lance Air	16253	17447	17435	Nm3/hr	7.34	99.93
Oxygen	6362	6746	6685	Nm3/hr	6.04	99.08
Lance Fuel	1.27	2.07	1.85	tph	62.89	87.97
Flux	5.99	5.49	6.54	tph	-8.34	84.01
O2 in blast	40.00	40	40.00	%	0.00	100.00
Slag Fe/SiO2	1.80	1.80	1.72		0.00	95.35
Oxygen/Matte	245	229	215	Nm3/tonne matte	-6.41	93.41
Flux/ Matte	171	157	187	Kg/tonne matte	-8.34	84.01
Matte Fall	28	26	25	% of feed matte	-6.93	96.82
Slag Fall	81	75	76	% of feed matte	-6.65	98.92
Lance Air	464	498	498	Nm3/tonne matte	7.34	99.93

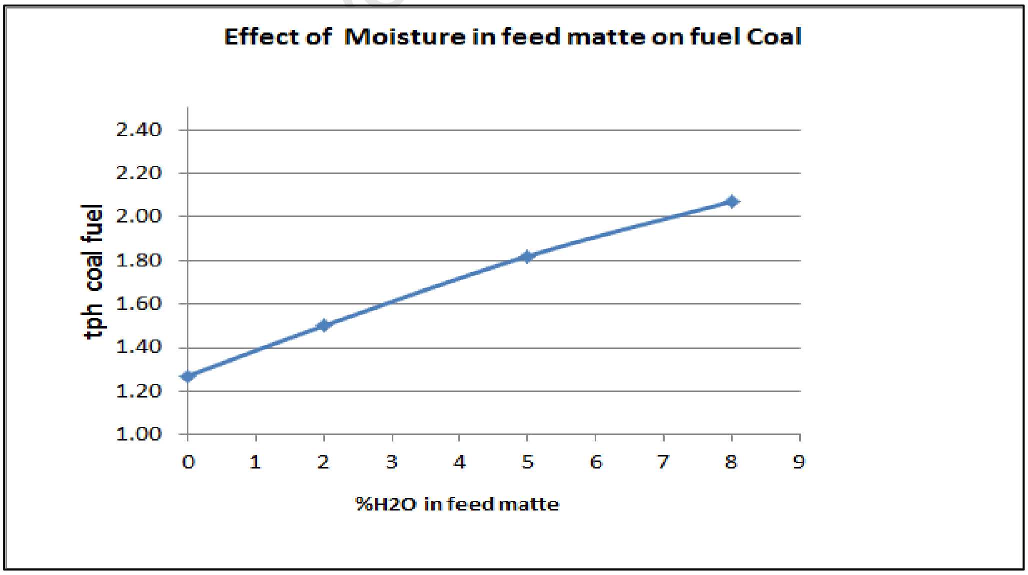


Figure 4.8: Fuel coal requirement for simulation 4.6

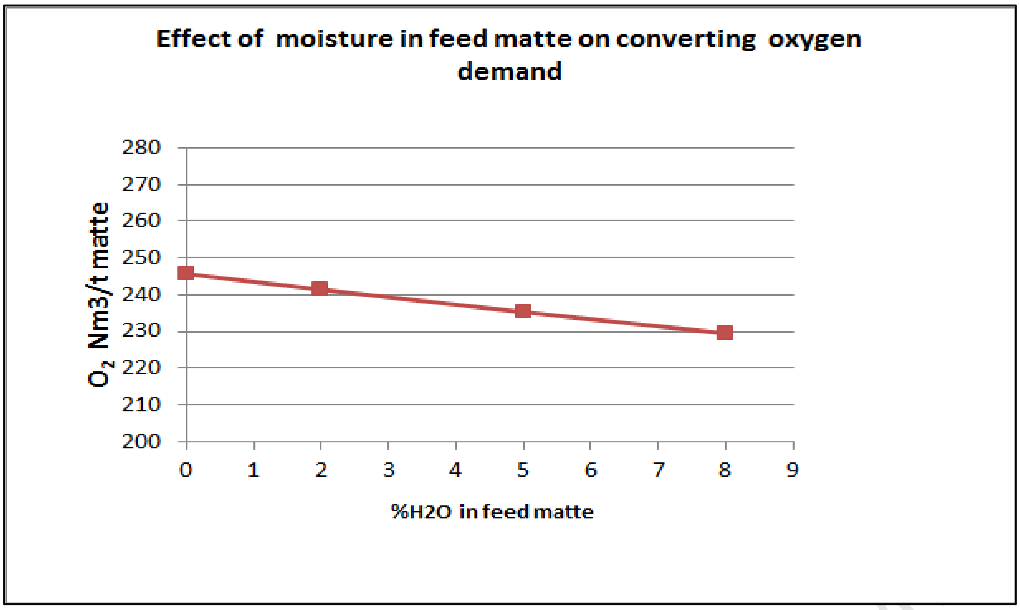


Figure 4.9: Specific oxygen requirement for simulation 4.6

It can be seen from figure 4.8 that an increase in the moisture content from 0 - 6.90% in the feed matte results in the increase in the lance fuel coal demand from 36.3- 59kg/tonne matte (1.27-2.07tph) for the steady state operation of the process. The increase in fuel coal is due to the H_2O in the feed matte that has to be preheated in liquid form, vaporised and again preheated in the gaseous form up to the operating temperature. Figure 4.9 shows that the specific oxygen demand reduces from 245-229Nm^3/tonne matte. It is also noted from table 4.28 that the specific flux demand reduces from 171-157Kg/tonne matte (5.99-5.49tph). The reduction in specific oxygen and flux is due to reduced base metals components in the feed matte resulting from the diluting effect of water in feed matte. At a fixed operating temperature the fuel coal can be minimised by increasing the oxygen enrichment on the air blast (Davenport 2001, 2002) when the moisture in feed matte or flux increases.

It has been demonstrated in this section that the inclusion of moisture in the matte feed or flux can readily be included in the calculation matrix by modifying the relevant equations. The calculation of the matrix including the moisture in the feed has shown that the treatment of wet matte results in the increase in fuel requirement and reduction in the specific oxygen and flux requirement. It is necessary that if for some reason dry matte to be treated becomes wet to some extent, the specific oxygen and flux settings in the control system be changed and the fuel coal also increased for the steady state operation process to be achieved. The other method to cover for the high fuel requirement is to increase the oxygen enrichment.

4.7 Effect of change miscellaneous parameters

During smelting and converting processes, it is necessary some times to change more than two operating parameters at the same moment in order to achieve some objectives.

This section demonstrates the treatment of wet matte indicated as feed 4 in table 4.1. The process oxygen enrichment and the operating temperature are increased from 40 - 50% and 1375^o - 1375^o. Compared with simulation 4.1, the changes in this section is the inputs enthalpy and mass balance of feed matte as in section 4.6, the enthalpy of the products at 1375^o as in section 4.5 and the modification of the oxygen enrichment equation from 40 to 50% in the blast.

4.7.1 Calculation Matrix

The calculation matrix of this section includes the new oxygen enrichment equation, the mass equations 4.51- 4.61 in section 4.6 and the new enthalpy equation 4.64 which is a combination of enthalpy the reactants in equation 4.62 in section 4.6 and the enthalpy of products in equation 4.50 in section 4.5. The modified oxygen enrichment equation is indicated by equation 4.63.

$$0 = 0.8585 * M_{O2} - 0.5631 * Ma \quad \text{at } 50\% \ O_2 \text{ enrichment} \tag{4.63}$$

$$\begin{aligned} -62082 = & -\text{Air}*0 - \text{Oxygen}*0 - \text{Reverts}* -0.960 - \text{flux}* -15.21 - \text{Coal}* -2.62 \\ & - 0*-15.88 + Cu_2S \text{ in matte}*0.33 + Ni_2S \text{ in matte}* 0.37 \\ & + FeS \text{ in matte}*0.23 + CoS \text{ in matte}* 0.50 + FeO \text{ in slag}* -2.29 \\ & + SiO_2 \text{ in slag}*-13.53 + CaO \text{ in slag}*-8.72 + MgO \text{ in slag}*-11.77 \\ & + Al_2O_3 \text{ in slag}*-13.73 + Cr_2O_3 \text{ in slag}*-5.11 + Fe_3O_4 \text{ in slag}*-3.05 \\ & + NiO \text{ in slag}*-1.41 + Cu_2O \text{ in slag}*0.03 + CoO \text{ in slag}*-1.39 \\ & + N_2 \text{ in offgas}*1.56 + SO_2 \text{ in offgas}*-3.51 + CO_2 \text{ in offgas}*-7.34 \\ & + H_2O \text{ in offgas}*-10.37 + 48668 \end{aligned} \tag{4.64}$$

Incorporating equations 4.51- 4.61, 4.63, 4.64 into the calculation matrix in the relevant columns and row results into the new matrix in table 4.29 which represents the process conditions assumed in in this section.

Table 4-29: Process calculation matrix for simulation 4.7

Balance	Value	Lance Air	Coal carrier Air	Matte carrier Air	Total Lance Air	Ind. Oxygen	Silica Flux	Lance Coal	Water in Matte	Reverts Feed	Cu2S	Ni2S	CoS	FeS	Mass Matte	FeO	SiO2	CaO	MgO	Al2O3	Cr2O3	Fe3O4	NiO	Cu2O	CoO	Mass Slag	N2	SO2	CO2	H2O	Heat Loss
Cu	3057	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.288	0.800	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Fe	12908	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	-0.032	0.000	0.000	0.000	0.640	0.000	0.780	0.000	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
S	9128	0.000	0.000	0.000	0.000	0.000	0.000	-0.016	0.000	-0.218	0.200	0.214	0.350	0.360	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.500	0.000	0.000	0.000
O	0	-0.230	-0.230	-0.230	0.000	-0.991	0.000	-0.055	-0.890	0.000	0.000	0.000	0.000	0.000	0.000	0.220	0.000	0.000	0.000	0.000	0.000	0.280	0.210	0.110	0.210	0.000	0.000	0.500	0.730	0.890	0.000
N	0	-0.770	-0.770	-0.770	0.000	-0.009	0.000	-0.015	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000
Co	140	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.004	0.000	0.000	0.650	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000
Ni	5031	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.445	0.000	0.786	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.790	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
C	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.709	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.270	0.000	0.000
H	0	0.000	0.000	0.000	0.000	0.000	0.000	-0.035	-0.110	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.110	0.000
SiO2	1450	0.000	0.000	0.000	0.000	0.000	-0.980	-0.098	0.000	-0.010	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Al2O3	209	0.000	0.000	0.000	0.000	0.000	-0.020	-0.062	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
CaO	53	0.000	0.000	0.000	0.000	0.000	0.000	-0.005	0.000	-0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Cr2O3	528	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
MgO	81	0.000	0.000	0.000	0.000	0.000	0.000	-0.003	0.000	-0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
H2O	2415	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Reverts	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Coal Carrier Air	643	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Carrier Air	3861	0.000	0.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Total Lance Air	0.00	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Matte Grade	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.6400	0.0278	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Slag Comp	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.780	-1.800	0.000	0.000	0.000	0.000	0.720	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Blast O2 Enrich	0.00	0.000	0.000	0.000	-0.563	0.859	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Ni Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.790	0.000	0.000	0.0318	0.000	0.000	0.000	0.000	0.000
Cu Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.899	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Co Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-0.823	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.316	0.000	0.000	0.000	0.000	0.000	0.000
Slag Fe Distrib.	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.083	0.000	0.000	0.000	0.000	0.000	-5.667	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Mass Slag	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000
Mass Matte	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	-1.000	-1.000	-1.000	-1.000	1.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Heat Loss MJ/hr	48668	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.000
Enthalpy(MJ)	-62082	0.000	0.000	0.000	0.000	0.000	15.210	2.620	15.880	0.960	0.333	0.370	0.497	0.226	0.000	-2.286	-13.529	-8.717	-11.766	-13.725	-5.106	-3.046	-1.408	0.033	-1.388	0.000	1.558	-3.512	-7.343	-10.367	1.000

4.7.2 Calculation results and discussion

Table 4.30 shows the predicted specific process parameters analysed from the results of the calculation matrix table 4.29. Table 4.31 shows the chemical assays of converter products matte and slag that were also analysed from the results of the calculation matrix 4.29.

Table 4-30: Predicted process parameters for simulation 4.7

Parameter	Section 4.1	Section 4.7	Industrial Actual	Units	% Change	%Accuracy
Matte feed rate	35	35	35	Nm3/hr	0.00	100.00
Lance Air	16253	9767	11200	Nm3/hr	-39.91	87.21
Oxygen	6362	7853	7785	Nm3/hr	23.44	99.13
Lance Fuel	1.27	1.74	1.70	tph	37.14	97.36
Flux	5.99	5.53	6.54	tph	-7.81	84.50
O2 in blast	40.00	50	40.00	%	25.00	75.00
Slag Fe/SiO2	1.80	1.80	1.72		0.00	95.35
Oxygen/Matte	245	228	215	Nm3/tonne matte	-6.80	93.86
Flux/ Matte	171	158	187	Kg/tonne matte	-7.81	84.50
Matte Fall	28	26	25	% of feed matte	-6.92	96.81
Slag Fall	81	75	76	% of feed matte	-6.74	98.82
Lance Air	464	279	320	Nm3/tonne matte	-39.91	87.21

Table 4-31: Predicted converter matte and slag assays for simulation 4.7

Converter Matte											
	%Cu	%Ni	%Co	%Fe	%S						
Model Prediction	28.78	46.70	0.37	2.78	21.37						
Industry Actual	27.42	46.22	0.34	2.80	21.67						
% Accuracy	95.06	98.95	90.47	99.18	98.62						
Slag											
	%Cu	%Ni	%Co	%Fe	%Fe_3O_4	%SiO_2	%CaO	%MgO	%Al_2O_3	%Cr_2O_3	Fe/SiO2
Model Prediction	1.74	3.18	0.41	48.18	10.04	26.77	0.24	0.33	1.63	2.01	1.80
Industry Actual	1.55	3.47	0.40	47.55	13.50	26.89	0.93	0.34	1.67	2.05	1.77
% Accuracy	87.38	91.64	98.71	98.68		99.54	25.37	96.42	97.39	97.96	98.21

Table 4.30 indicates that there is a significant increase in lance fuel requirement by 37% when the conditions in this section are compared to the ones in section 4.1 and this is due to the increase in the bath operating temperature and the introduction of moisture in feed matte. However the same table 4.30 indicates that there is a large reduction in lance air by 40% and an increase in bulk oxygen by 23%which is due to the increase in the oxygen requirement. The increase in oxygen enrichment is the reason the lance fuel coal requirement is below 2tph when compared with the results in section 4.6 which was run with the same wet matte but at 40% oxygen enrichment and 1325°C bath temperature. However the choice of operating with high oxygen enrichment or high fuel feed rates is a function of the cost between the two but it is believed that there are more benefits when running with high oxygen enrichment than at high fuel feed rates and high air blasts.

It has been demonstrated that the matrix can readily be modified to cater for multiple changes in the converter operating conditions. The model results has indicated that the fuel requirement and lance air can be minimized with high oxygen enrichments in the air blast when operating at high temperatures, high matte feed rates and high moisture content in the feed matte. Oxygen enrichment of the blast is a standard practice and leads to low process fuel requirement, a smaller air blast and high productivity.

5.0 Conclusion

It has been demonstrated in this work that the converting process of Ni-Cu matte in the TSL base metal converter can be simulated using the matrix inverse method. The process is described by a set of linear equations that represent the process mass balances, heat balance, oxygen enrichment, heat loss and metal distribution of the of metals and mineral compounds in converter matte and slag. The equations are arranged in a matrix that is solved using the matrix inverse method to calculate the mass of unknown feed inputs and outputs as well as compositions of the products at the prescribed process conditions. The model has shown that given a unique task like production of converter matte with 2.78% Fe and slag of Fe/SiO_2 ratio of 1.80 at 1325^{o}C from a given type of furnace matte requires a unique amount of air, oxygen, lance fuel and flux of the specific quality.

The linear equations in the form of the matrix representing the converting process can readily be modified to represent the new process operating conditions to give the strategies by which the goals of the process may be attained. The following conclusions were drawn from this work;

- Whatever the change in operating parameters and material quality, the effect on reactants requirements and product masses and composition is readily determined by the operating matrixes by changing the relevant linear equations.
- The matrix can be modified to obtain an optimum alternative method of producing converter matte of the prescribed grade.
- The matrix can be used to optimise the process to give minimum smelting costs, minimum energy input, maximum smelting rate and optimised SO_2 fixation.
- Effects of multiple process changes such as feed quality change, product matte grade, slag quality, bath temperature, heat loss change and feed rate change can also be determined by making appropriate multiple changes to the calculation matrix simultaneously.

The model was tested and the results have shown that manipulation of the linear equations in the matrix due to change in operating conditions yields the expected trends. The model has shown that during the converting process;

- The changes in the chemical composition of the input materials require the change in the feed air, oxygen, lance fuel and flux feed rates necessary for production of products matte, slag and offgas of desired quality and volumes.

- Operating the furnace at a high bath temperature requires the increase in the lance fuel coal or oil. High operating temperatures require more energy to keep the slag viscosity low for easy handling of slag and low valuable metal loss in slag. It is important to treat matte and flux that has low concentration of gangue mineral oxides such as MgO, Al_2O_3 and SiO_2 so that the converting process can be operated at lower bath temperatures.

- Oxygen enrichment decreases the lance fuel and blast requirement for converting. Air as a source of oxygen increases the blast volume and output offgas volume and also dilutes the SO_2 concentration. Increased lance oxygen enrichment minimises the N_2 from air that get heated from ambient temperature to the furnace offgas temperature and this takes away the valuable heat from the furnace much of which comes from the FeS oxidation. High oxygen enrichment is also used to increase productivity.

- Operating the TSL converter at high matte feed rates is desirable as this leads to a reduction in the lance fuel requirement due to the increased FeS oxidation rate.

- The moisture in the feed matte or flux leads to high consumption of lance fuel as a result of the contained water that need to be heated and vaporised to the process operating temperature.

- The converting process require less amount of diesel oil compared to coal due to the high C and H content and absence of ash which is an energy consumer.

The overall conclusion from this work is that the converting process of the nickel-copper PGM matte in the TSL furnace can be simulated effectively with the use of matrix that is solved by the mathematical matrix inverse method. Comparison of the predicted data with the industrial data indicated that the model is above 90% accurate. The model is made up of a system of linear equations that represent smelting process mass and heat balance equations and equilibrium conditions. The simulation program is done in the windows excel program with the help of the MINVERSE and MMULT formulas.

6.0 Recommendations

The matrix model of the TSL base metals converter demonstrated in this study has to a large extent accomplished in predicting the critical process input parameters necessary to produce matte, slag and offgas of desired quality. The model also further discussed the effects of change of operating variables on process inputs requirements and volumes and quality of the outputs. However further work may be required for a complete process representation. To improve on this matrix model, the following is recommended;

- Model validation has been performed through comparison of the matrix model results and the actual plant operating data from one of the smelters. Due to restrictions of change that can be made on the plant, further validation using data from various operations is required to gain complete confidence in the matrix model results.
- The mass and energy balance did not include the dilution air into the process which is a key operating variable in most smelting processes. The effect of dilution air into the furnace environment must be included in future models of the TSL base metal converter.
- The temperatures of the product matte, slag and offgas were assumed to be the same, however in actual plant processes they are at slightly different temperatures hence it is recommended that further modeling must take into consideration the difference in matte, slag and offgas temperatures.
- The mass and heat loss balances used in the model were for the copper crucible of dimensions 4.4m internal diameter and 4.5m height which is situated at the bottom of the TSL furnace where bath process reactions take place. It is recommended that the mass and heat balance equations be expanded on the entire TSL furnace to include the omitted inputs of roof coal, shroud air and oxygen.

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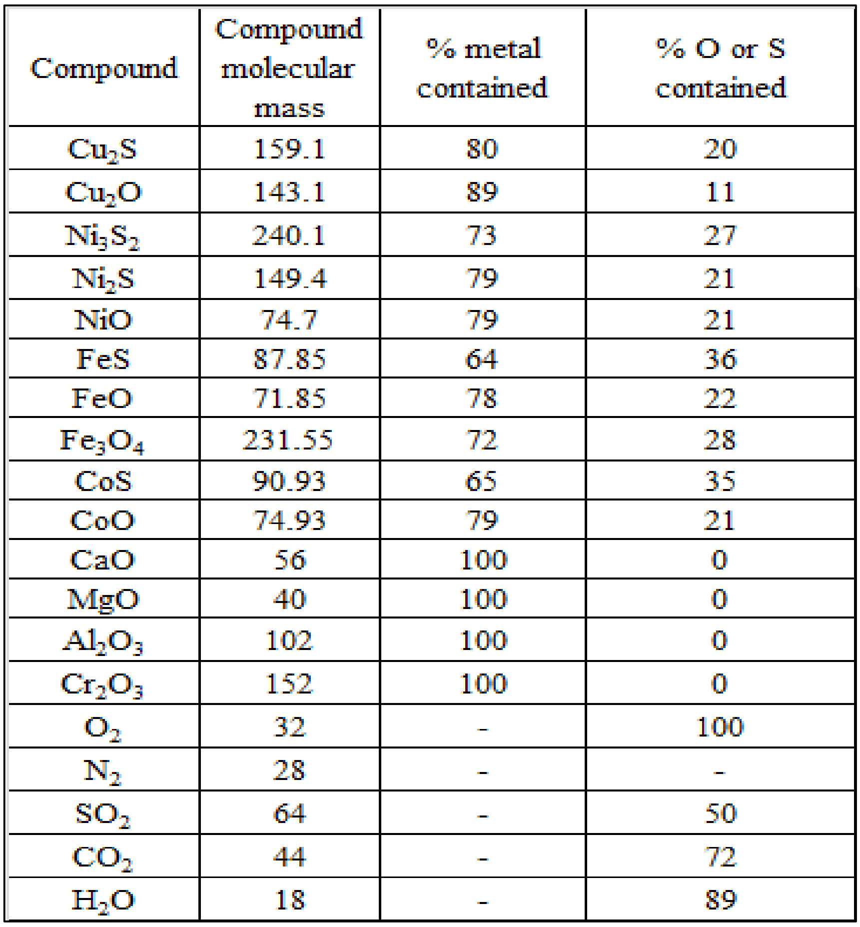
Appendices

Appendix 1.0 Stoichiometric data for elements, minerals and compounds

The oxides of Ca, Al, Mg and Cr are considered to be like elements since they are strongly

Bonded to the metals and do not lose the oxygen in the process.

Table A.1: Metals molecular masses and contained metals or O or S

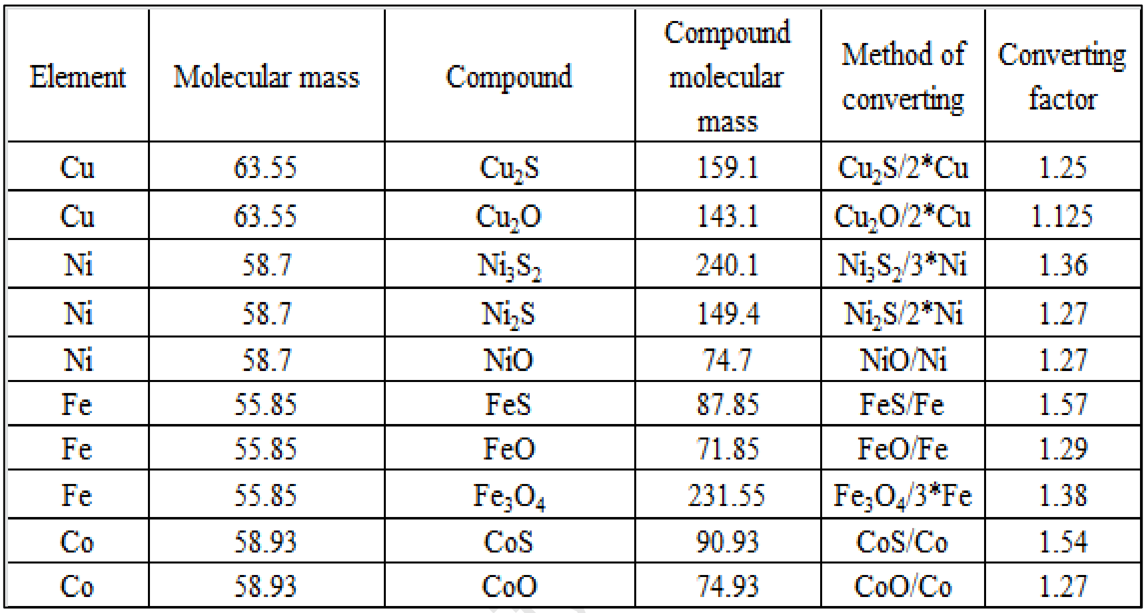


Compound	Compound molecular mass	% metal contained	% O or S contained
Cu_2S	159.1	80	20
Cu_2O	143.1	89	11
Ni_3S_2	240.1	73	27
Ni_2S	149.4	79	21
NiO	74.7	79	21
FeS	87.85	64	36
FeO	71.85	78	22
Fe_3O_4	231.55	72	28
CoS	90.93	65	35
CoO	74.93	79	21
CaO	56	100	0
MgO	40	100	0
Al_2O_3	102	100	0
Cr_2O_3	152	100	0
O_2	32	-	100
N_2	28	-	-
SO_2	64	-	50
CO_2	44	-	72
H_2O	18	-	89

Appendix 2.0 Minerals conversion factors

In the smelting processes of sulphide minerals, the XRF machine is used to quickly analyse the feed and product samples to control the process. The analyses are usually presented in elementals form for base metals but the metals exist as mineral compounds in the matte and slag. To perform the heats balance calculations require that these minerals be converted into their assumed metals compounds. The minerals in the study were converted as illustrated in the table A.2.

Table A2: Metals molecular masses and contained metals or O or S



Element	Molecular mass	Compound	Compound molecular mass	Method of converting	Converting factor
Cu	63.55	Cu_2S	159.1	$Cu_2S/2*Cu$	1.25
Cu	63.55	Cu_2O	143.1	$Cu_2O/2*Cu$	1.125
Ni	58.7	Ni_3S_2	240.1	$Ni_3S_2/3*Ni$	1.36
Ni	58.7	Ni_2S	149.4	$Ni_2S/2*Ni$	1.27
Ni	58.7	NiO	74.7	NiO/Ni	1.27
Fe	55.85	FeS	87.85	FeS/Fe	1.57
Fe	55.85	FeO	71.85	FeO/Fe	1.29
Fe	55.85	Fe_3O_4	231.55	$Fe_3O_4/3*Fe$	1.38
Co	58.93	CoS	90.93	CoS/Co	1.54
Co	58.93	CoO	74.93	CoO/Co	1.27

Appendix 3.0 Calculation of metal partition coefficients, slag quality, Carrier airs and product matte grade equations

The metal partition equations and the slag and matte grade equations used in the study were developed using the information obtained from the real plant data at one of the platinum smelters that uses the TSL for converting furnace matte. Using the real plant process information, the equations were developed as follows;

3.1 Copper in converter matte and slag

85% of the copper in the feed reports to the converter matte and 15% reports to the slag. Copper in the matte and slag exists as Cu_2S and Cu_2O and using the information from appendix1, the partition equation is;

$$0.80 * Cu_2S / 0.89 * Cu_2O = 85/15$$

3.2 Nickel in converter slag

Nickel in the converter slag is assumed to exist as NiO and the composition is 3.18% of the total slag mass. The slag Ni equation is given as;

$$0.79 * NiO = 0.0318 * \text{Mass slag}$$

3.3 Cobalt in converter matte and slag

26% of the cobalt in the feed reports to the converter matte and 74% reports to the converter slag. Cobalt in matte and slag exist is assumed to exist a CoS and CoO and hence the partition equation is calculated as;

$$0.65 * CoS / 0.79 * CoO = 26/74$$

3.4 Iron partition equation in the converter slag as FeO and Fe_3O_4

85% of the Fe oxidised into the slag exist as FeO and 15% as Fe_3O_4. Therefore the iron partition equation in the slag is;

$0.78*FeO/0.72*Fe_3O_4 = 85/15$

3.5 Converter matte grade as % Fe in the product converter matte

Iron in the converter matte is assumed to exist as FeS and the composition is 2.78% of the total mass of converter matte. The product Fe equation is given as;

0.64*FeS in matte = 0.0278* Fe in converter matte

3.6 Slag Fe/SiO_2 ratio equation

The optimum Fe/SiO_2 ratio in the slag varies from 1.70 to 1.80 and Fe in the slag exists as FeO and Fe_3O_4, for the target ratio of 1.80, the slag quality equation is given as;

$$(0.78*FeO + 0.72*Fe_3O_4) /SiO_2 = 1.80$$

3.7 Blast Air oxygen enrichment equation

Industrial oxygen and atmospheric air considered in this study contains 99% and 21% by volume which is 99.12 and 23.30% by weight. For the lance blast containing 40% O_2 by volume (43.2% by weight), the blast oxygen enrichment equation is given as

$$(0.233* Ma +0.9912* MO_2) / (Ma + MO_2) *100 = 43.2\%$$

Appendix 4.0 Enthalpy calculation and heat loss calculation

The enthalpies or heat contents of elements and mineral compounds in the model at given temperatures are calculated using the formula indicated below. The heat capacity constants and enthalpies of different materials at various temperatures that have been used in the model are indicated in table A.3.

$$H_T = H° + A*(T_2-T_1) + 1/2\ B*\ (T_2^2 – T_1^2) –C/(T_2 – T_1) + 1/3\ D*(T_2^{3-} T_1^3) \quad \text{MJ/Kg}$$

Where temperature is in K

The heat loss (H_{loss}) from the furnace wall is calculated using equation 3.24 which is the design heat loss fluxes obtained for one of the TSL converter used for converting furnace mattes at one of the platinum smelters.

Furnace diameter = 4.4m

Furnace hearth area = 15.21m^2

Furnace circumference = 13.82m

Furnace height = 15m

Effective bath height of furnace bottom = 4.45m

Design heat flux loss bottom furnace = 100KW/m^2

Design heat flux loss hearth area = 5KW/m^2

Design heat flux furnace top (Uptake) = 50KW/m^2

Heat loss bottom of furnace = (13.82*4.45*100) + (15.21*5) *3.6 = 22418MJ/hr

Heat loss furnace top = 13.82(15 - 4.45)*50*3600*3.6 = 26250MJ/hr

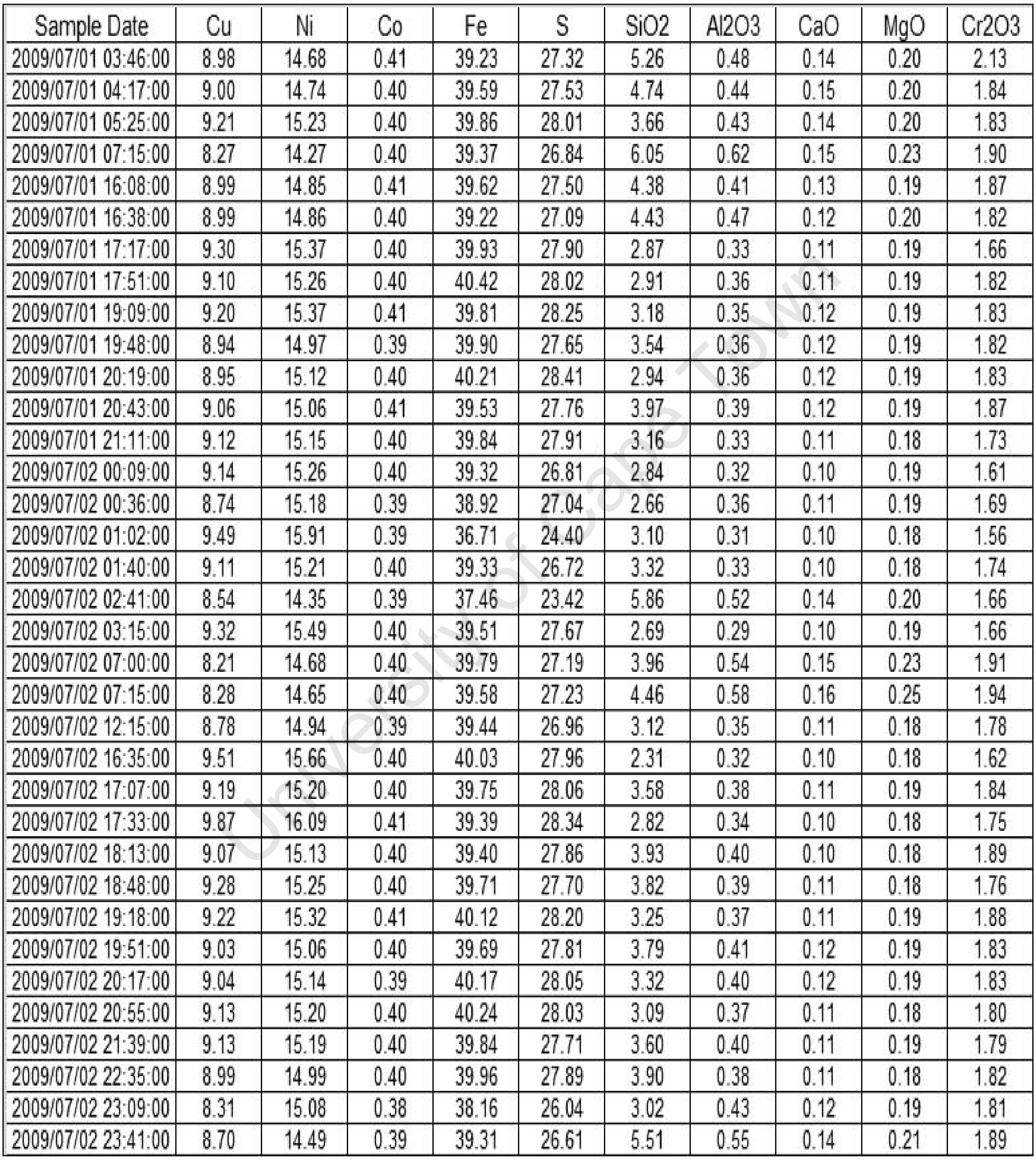
Total furnace wall heat loss = 48668MJ/hr

Table A.3: Heat capacity constants and enthalpies of elements and compounds at different temperatures

		H°	S°	A	B	C	D	Enthalpies at different temperatures			
				1	x 10^{-3}	1x 10^{5}	1 x 10^{-6}	25°C	1325°C	1350°C	1375°C
	MW	kcal/mol	cal/(mol.K)	cal/(mol.K)	cal/(mol.K)	cal/(mol.K)	cal/(mol.K)	MJ/kg	MJ/kg	MJ/kg	MJ/kg
O2(g)	32	0.000	49.031	7.412	1.002	-0.683	-0.092	0.000	1.381	1.409	1.438
N2(g)	28	0.000	45.782	5.390	3.157	0.748	-0.810	0.000	1.495	1.527	1.558
SO2(g)	64	-70.743	59.889	13.093	0.801	-5.914	-0.058	-4.625	-3.558	-3.535	-3.512
CO2(g)	44	-93.777	51.865	12.700	1.152	-6.974	-0.129	-8.917	-7.410	-7.377	-7.343
H2O(g)	18	-57.880	44.905	5.227	5.392	2.029	-0.956	-13.454	-10.501	-10.434	-10.367
H2O(l)	18	-68.315	16.718	14.266	5.500	1.403	5.918	-15.879	-8.044	-7.820	-7.593
CuFeS2(s)	183.35	-45.500	29.870	20.790	12.800	-1.340	0.000	-1.038	-0.070	-0.046	-0.022
Cu2S(s)	159	-19.400	27.761	11.457	23.225	0.000	0.000	-0.511	0.635	0.667	0.699
Cu2S(l)	159	-16.276	31.658	21.430	0.000	0.000	0.000	-0.428	0.305	0.319	0.333
Ni3S2(s)	240.1	-51.700	32.000	27.056	11.622	-2.117	-0.002	-0.901	-0.048	-0.029	-0.009
Ni3S2(l)	240.1	-40.667	38.391	45.840	0.000	0.000	0.000	-0.709	0.330	0.350	0.370
CoS(s)	90.93	-19.800	14.900	10.600	2.510	0.000	0.000	-0.911	-0.135	-0.118	-0.101
CoS(l)	90.93	-6.800	24.900	10.600	2.510	0.000	0.000	-0.313	0.464	0.480	0.497
FeS(s)	87.85	-24.300	14.417	-65.313	186.229	19.417	0.000	-1.157	5.984	6.264	6.550
FeS(l)	87.85	-15.447	21.799	14.950	0.000	0.000	0.000	-0.736	0.190	0.208	0.226
FeS2(s)	119.85	-41.000	12.650	16.461	3.346	-2.320	0.000	-1.431	-0.562	-0.543	-0.524
FeS2(l)	119.85	-30.000	22.650	16.461	3.346	-2.320	0.000	-1.047	-0.178	-0.159	-0.140
Cu2O(s)	143	-40.800	22.070	18.045	-1.621	-2.612	3.295	-1.194	-0.456	-0.439	-0.421
Cu2O(l)	143	-31.124	23.040	23.880	0.000	0.000	0.000	-0.911	-0.002	0.015	0.033
NiO(s)	74.71	-57.290	9.080	77.873	-250.595	-15.223	276.180	-3.208	5.838	6.384	6.953
NiO(l)	74.71	-42.693	15.630	13.000	0.000	0.000	0.000	-2.391	-1.444	-1.426	-1.408
CoO(s)	74.93	-56.870	12.660	12.268	0.065	0.665	1.036	-3.176	-2.192	-2.171	-2.150
CoO(l)	74.93	-44.425	17.255	14.500	0.000	0.000	0.000	-2.481	-1.428	-1.408	-1.388
FeO(s)	71.85	-63.879	13.764	12.142	2.059	-0.791	0.000	-3.720	-2.665	-2.643	-2.620
FeO(l)	71.85	-61.256	12.488	16.300	0.000	0.000	0.000	-3.567	-2.333	-2.309	-2.286
Fe3O4(s)	231.55	-267.300	34.930	113.579	-208.811	-28.805	191.379	-4.830	-2.280	-2.157	-2.029
Fe3O4(l)	231.55	-237.407	47.414	51.000	0.000	0.000	0.000	-4.290	-3.092	-3.069	-3.046
Fe2O3(s)	159.7	-197.000	20.890	23.490	18.600	-3.550	0.000	-5.161	-3.786	-3.751	-3.716
SiO2(s)	60	-217.700	9.910	13.882	-0.008	-3.408	6.745	-15.181	-13.352	-13.298	-13.242
SiO2(l)	60	-221.684	2.225	20.500	0.000	0.000	0.000	-15.459	-13.600	-13.565	-13.529
Al2O3(s)	102	-400.400	12.174	24.821	6.278	-6.953	0.000	-16.424	-14.861	-14.826	-14.790
Al2O3(l)	102	-381.341	10.790	34.623	0.000	0.000	0.000	-15.642	-13.796	-13.761	-13.725
CaO(s)	56	-151.790	9.100	11.860	1.080	-1.660	0.000	-11.341	-10.123	-10.098	-10.073
CaO(l)	56	-136.925	9.794	15.000	0.000	0.000	0.000	-10.230	-8.773	-8.745	-8.717
MgO(s)	40	-143.700	6.435	11.442	1.092	-2.596	-0.062	-15.031	-13.417	-13.383	-13.349
MgO(l)	40	-134.083	3.067	16.000	0.000	0.000	0.000	-14.025	-11.849	-11.808	-11.766
CaCO3(s)	100	-288.461	22.204	23.821	6.445	-5.149	0.000	-12.069	-10.500	-10.464	-10.429
MgCO3(s)	84	-261.902	15.703	18.620	13.800	-4.160	0.000	-13.045	-11.049	-10.998	-10.947
Cr2O3(s)	168	-272.395	19.407	24.930	5.844	1.814	-0.910	-6.784	-5.816	-5.796	-5.776
Cr2O3(l)	168	-255.631	15.513	37.500	0.000	0.000	0.000	-6.366	-5.152	-5.129	-5.106
Cu5FeS4(s)	501.35	-90.900	86.600	49.760	35.080	-1.350	0.000	-0.759	0.139	0.161	0.184
CuS(s)	95.5	-12.690	15.890	10.600	2.640	0.000	0.000	-0.556	0.190	0.207	0.223
2FeO.SiO2(l)	203.7	-348.515	23.255	57.500	0.000	0.000	0.000	-7.159	-5.623	-5.594	-5.564
C10H22(l)	142	-68.786	83.748	64.627	0.000	0.000	0.000	-2.027	0.449	0.496	0.544
C11H24(l)	156	-73.518	80.545	72.419	0.000	0.000	0.000	-1.972	0.553	0.602	0.650
C12H26(l)	170	-78.155	89.388	80.306	0.000	0.000	0.000	-1.924	0.646	0.695	0.745

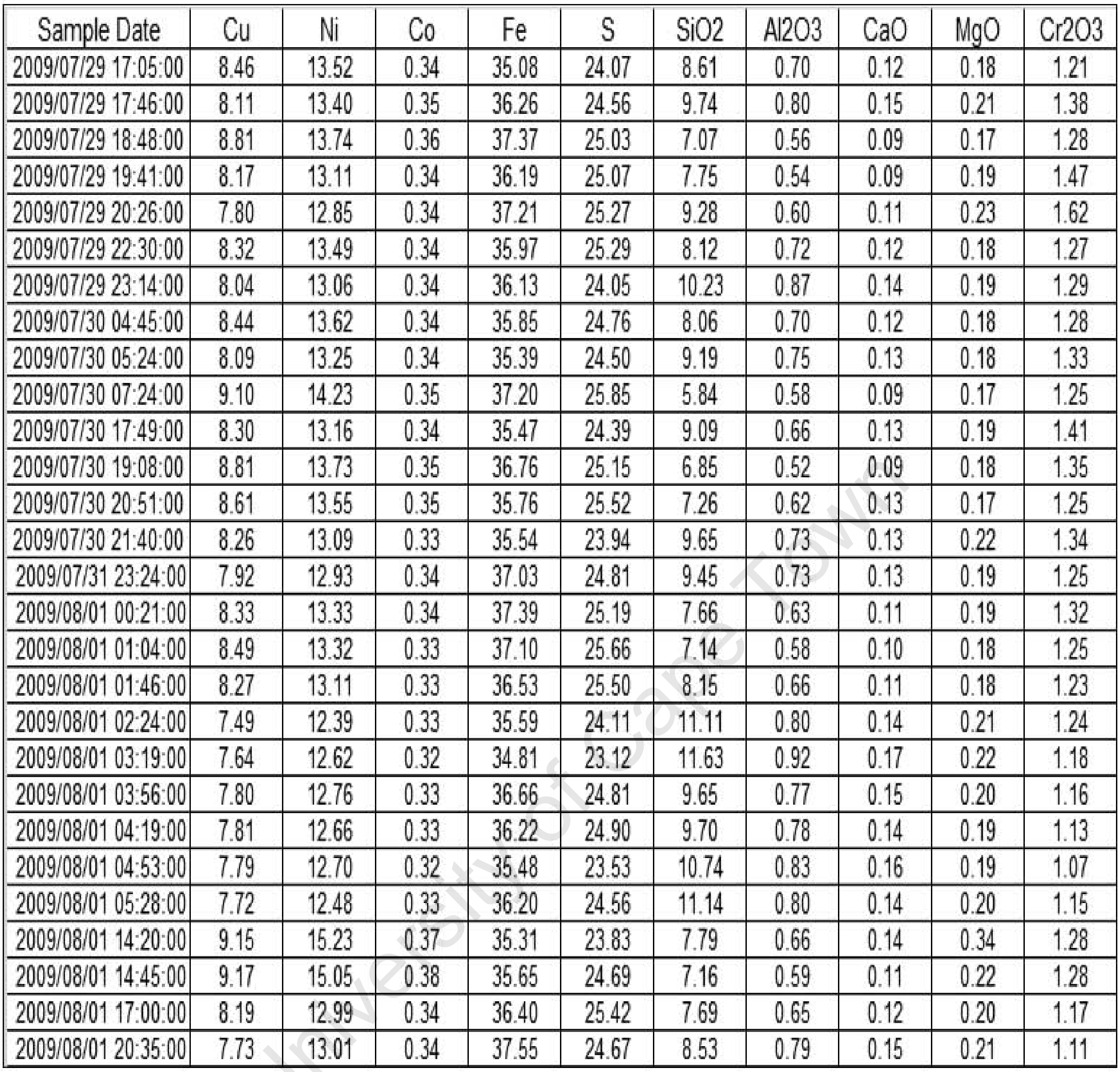
Appendix 5.0 Feed matte, slag and converter slag assays from a platinum smelter treating Ni-Cu-PGM matte (Matthew 2009)

Table A4: Converter feed PGM furnace matte chemical assays from one platinum smelter



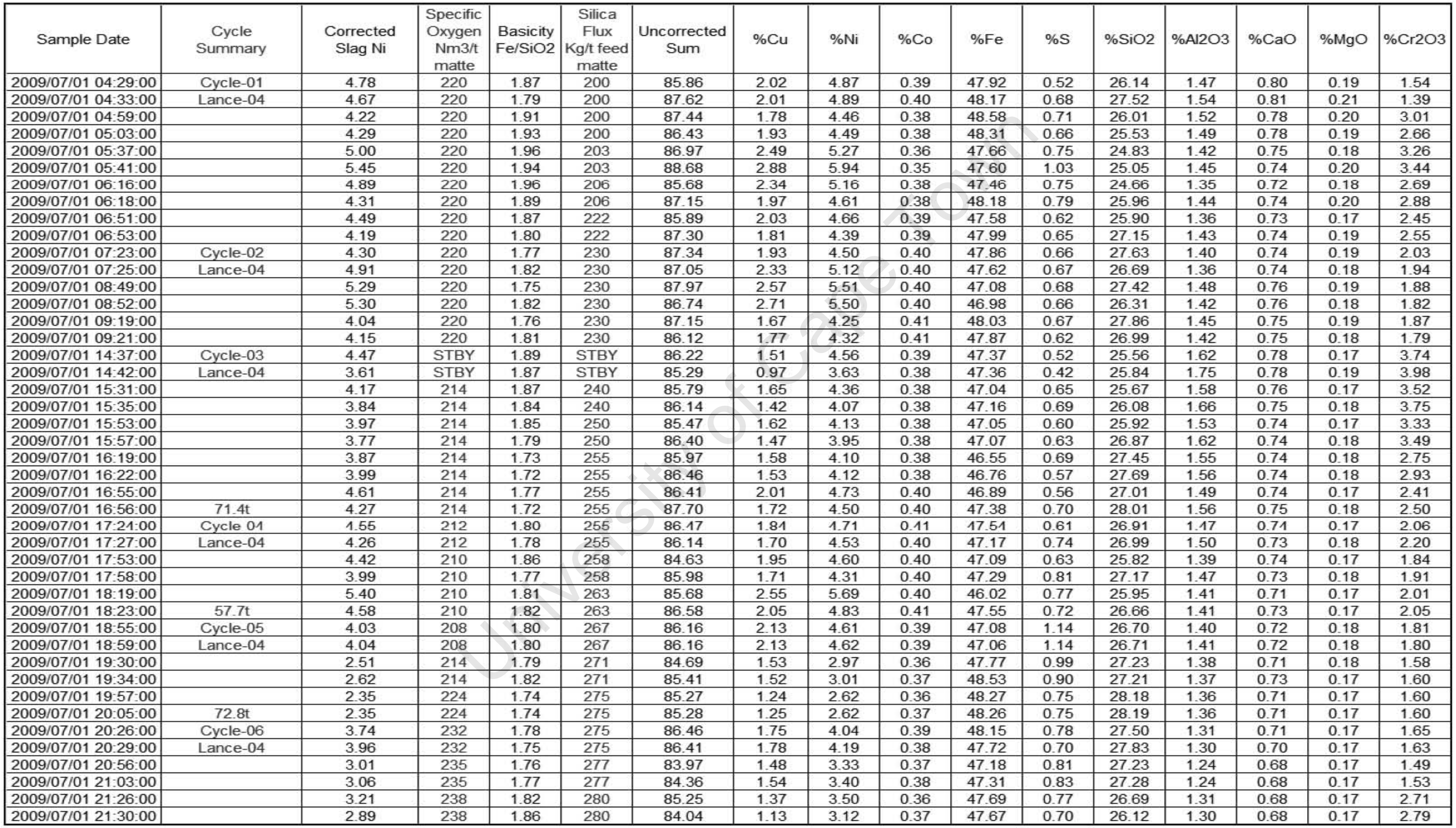
Sample Date	Cu	Ni	Co	Fe	S	SiO2	Al2O3	CaO	MgO	Cr2O3
2009/07/01 03:46:00	8.98	14.68	0.41	39.23	27.32	5.26	0.48	0.14	0.20	2.13
2009/07/01 04:17:00	9.00	14.74	0.40	39.59	27.53	4.74	0.44	0.15	0.20	1.84
2009/07/01 05:25:00	9.21	15.23	0.40	39.86	28.01	3.66	0.43	0.14	0.20	1.83
2009/07/01 07:15:00	8.27	14.27	0.40	39.37	26.84	6.05	0.62	0.15	0.23	1.90
2009/07/01 16:08:00	8.99	14.85	0.41	39.62	27.50	4.38	0.41	0.13	0.19	1.87
2009/07/01 16:38:00	8.99	14.86	0.40	39.22	27.09	4.43	0.47	0.12	0.20	1.82
2009/07/01 17:17:00	9.30	15.37	0.40	39.93	27.90	2.87	0.33	0.11	0.19	1.66
2009/07/01 17:51:00	9.10	15.26	0.40	40.42	28.02	2.91	0.36	0.11	0.19	1.82
2009/07/01 19:09:00	9.20	15.37	0.41	39.81	28.25	3.18	0.35	0.12	0.19	1.83
2009/07/01 19:48:00	8.94	14.97	0.39	39.90	27.65	3.54	0.36	0.12	0.19	1.82
2009/07/01 20:19:00	8.95	15.12	0.40	40.21	28.41	2.94	0.36	0.12	0.19	1.83
2009/07/01 20:43:00	9.06	15.06	0.41	39.53	27.76	3.97	0.39	0.12	0.19	1.87
2009/07/01 21:11:00	9.12	15.15	0.40	39.84	27.91	3.16	0.33	0.11	0.18	1.73
2009/07/02 00:09:00	9.14	15.26	0.40	39.32	26.81	2.84	0.32	0.10	0.19	1.61
2009/07/02 00:36:00	8.74	15.18	0.39	38.92	27.04	2.66	0.36	0.11	0.19	1.69
2009/07/02 01:02:00	9.49	15.91	0.39	36.71	24.40	3.10	0.31	0.10	0.18	1.56
2009/07/02 01:40:00	9.11	15.21	0.40	39.33	26.72	3.32	0.33	0.10	0.18	1.74
2009/07/02 02:41:00	8.54	14.35	0.39	37.46	23.42	5.86	0.52	0.14	0.20	1.66
2009/07/02 03:15:00	9.32	15.49	0.40	39.51	27.67	2.69	0.29	0.10	0.19	1.66
2009/07/02 07:00:00	8.21	14.68	0.40	39.79	27.19	3.96	0.54	0.15	0.23	1.91
2009/07/02 07:15:00	8.28	14.65	0.40	39.58	27.23	4.46	0.58	0.16	0.25	1.94
2009/07/02 12:15:00	8.78	14.94	0.39	39.44	26.96	3.12	0.35	0.11	0.18	1.78
2009/07/02 16:35:00	9.51	15.66	0.40	40.03	27.96	2.31	0.32	0.10	0.18	1.62
2009/07/02 17:07:00	9.19	15.20	0.40	39.75	28.06	3.58	0.38	0.11	0.19	1.84
2009/07/02 17:33:00	9.87	16.09	0.41	39.39	28.34	2.82	0.34	0.10	0.18	1.75
2009/07/02 18:13:00	9.07	15.13	0.40	39.40	27.86	3.93	0.40	0.10	0.18	1.89
2009/07/02 18:48:00	9.28	15.25	0.40	39.71	27.70	3.82	0.39	0.11	0.18	1.76
2009/07/02 19:18:00	9.22	15.32	0.41	40.12	28.20	3.25	0.37	0.11	0.19	1.88
2009/07/02 19:51:00	9.03	15.06	0.40	39.69	27.81	3.79	0.41	0.12	0.19	1.83
2009/07/02 20:17:00	9.04	15.14	0.39	40.17	28.05	3.32	0.40	0.12	0.19	1.83
2009/07/02 20:55:00	9.13	15.20	0.40	40.24	28.03	3.09	0.37	0.11	0.18	1.80
2009/07/02 21:39:00	9.13	15.19	0.40	39.84	27.71	3.60	0.40	0.11	0.19	1.79
2009/07/02 22:35:00	8.99	14.99	0.40	39.96	27.89	3.90	0.38	0.11	0.18	1.82
2009/07/02 23:09:00	8.31	15.08	0.38	38.16	26.04	3.02	0.43	0.12	0.19	1.81
2009/07/02 23:41:00	8.70	14.49	0.39	39.31	26.61	5.51	0.55	0.14	0.21	1.89

Table A5: Converter feed PGM furnace matte chemical assays from one platinum smelter



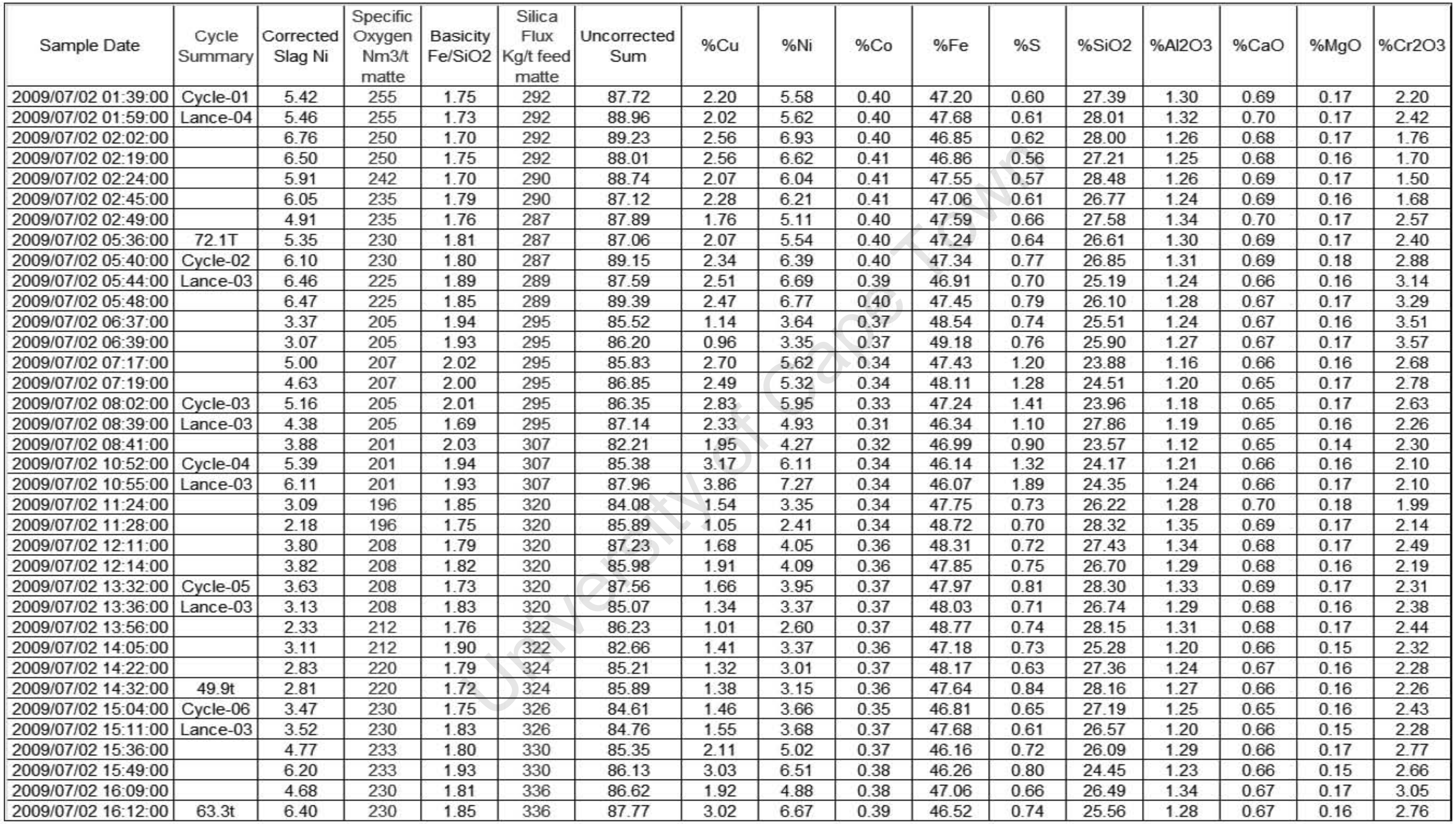
Sample Date	Cu	Ni	Co	Fe	S	SiO2	Al2O3	CaO	MgO	Cr2O3
2009/07/29 17:05:00	8.46	13.52	0.34	35.08	24.07	8.61	0.70	0.12	0.18	1.21
2009/07/29 17:46:00	8.11	13.40	0.35	36.26	24.56	9.74	0.80	0.15	0.21	1.38
2009/07/29 18:48:00	8.81	13.74	0.36	37.37	25.03	7.07	0.56	0.09	0.17	1.28
2009/07/29 19:41:00	8.17	13.11	0.34	36.19	25.07	7.75	0.54	0.09	0.19	1.47
2009/07/29 20:26:00	7.80	12.85	0.34	37.21	25.27	9.28	0.60	0.11	0.23	1.62
2009/07/29 22:30:00	8.32	13.49	0.34	35.97	25.29	8.12	0.72	0.12	0.18	1.27
2009/07/29 23:14:00	8.04	13.06	0.34	36.13	24.05	10.23	0.87	0.14	0.19	1.29
2009/07/30 04:45:00	8.44	13.62	0.34	35.85	24.76	8.06	0.70	0.12	0.18	1.28
2009/07/30 05:24:00	8.09	13.25	0.34	35.39	24.50	9.19	0.75	0.13	0.18	1.33
2009/07/30 07:24:00	9.10	14.23	0.35	37.20	25.85	5.84	0.58	0.09	0.17	1.25
2009/07/30 17:49:00	8.30	13.16	0.34	35.47	24.39	9.09	0.66	0.13	0.19	1.41
2009/07/30 19:08:00	8.81	13.73	0.35	36.76	25.15	6.85	0.52	0.09	0.18	1.35
2009/07/30 20:51:00	8.61	13.55	0.35	35.76	25.52	7.26	0.62	0.13	0.17	1.25
2009/07/30 21:40:00	8.26	13.09	0.33	35.54	23.94	9.65	0.73	0.13	0.22	1.34
2009/07/31 23:24:00	7.92	12.93	0.34	37.03	24.81	9.45	0.73	0.13	0.19	1.25
2009/08/01 00:21:00	8.33	13.33	0.34	37.39	25.19	7.66	0.63	0.11	0.19	1.32
2009/08/01 01:04:00	8.49	13.32	0.33	37.10	25.66	7.14	0.58	0.10	0.18	1.25
2009/08/01 01:46:00	8.27	13.11	0.33	36.53	25.50	8.15	0.66	0.11	0.18	1.23
2009/08/01 02:24:00	7.49	12.39	0.33	35.59	24.11	11.11	0.80	0.14	0.21	1.24
2009/08/01 03:19:00	7.64	12.62	0.32	34.81	23.12	11.63	0.92	0.17	0.22	1.18
2009/08/01 03:56:00	7.80	12.76	0.33	36.66	24.81	9.65	0.77	0.15	0.20	1.16
2009/08/01 04:19:00	7.81	12.66	0.33	36.22	24.90	9.70	0.78	0.14	0.19	1.13
2009/08/01 04:53:00	7.79	12.70	0.32	35.48	23.53	10.74	0.83	0.16	0.19	1.07
2009/08/01 05:28:00	7.72	12.48	0.33	36.20	24.56	11.14	0.80	0.14	0.20	1.15
2009/08/01 14:20:00	9.15	15.23	0.37	35.31	23.83	7.79	0.66	0.14	0.34	1.28
2009/08/01 14:45:00	9.17	15.05	0.38	35.65	24.69	7.16	0.59	0.11	0.22	1.28
2009/08/01 17:00:00	8.19	12.99	0.34	36.40	25.42	7.69	0.65	0.12	0.20	1.17
2009/08/01 20:35:00	7.73	13.01	0.34	37.55	24.67	8.53	0.79	0.15	0.21	1.11

Table A6: Converting process slag bath chemical assays control from one platinum smelter



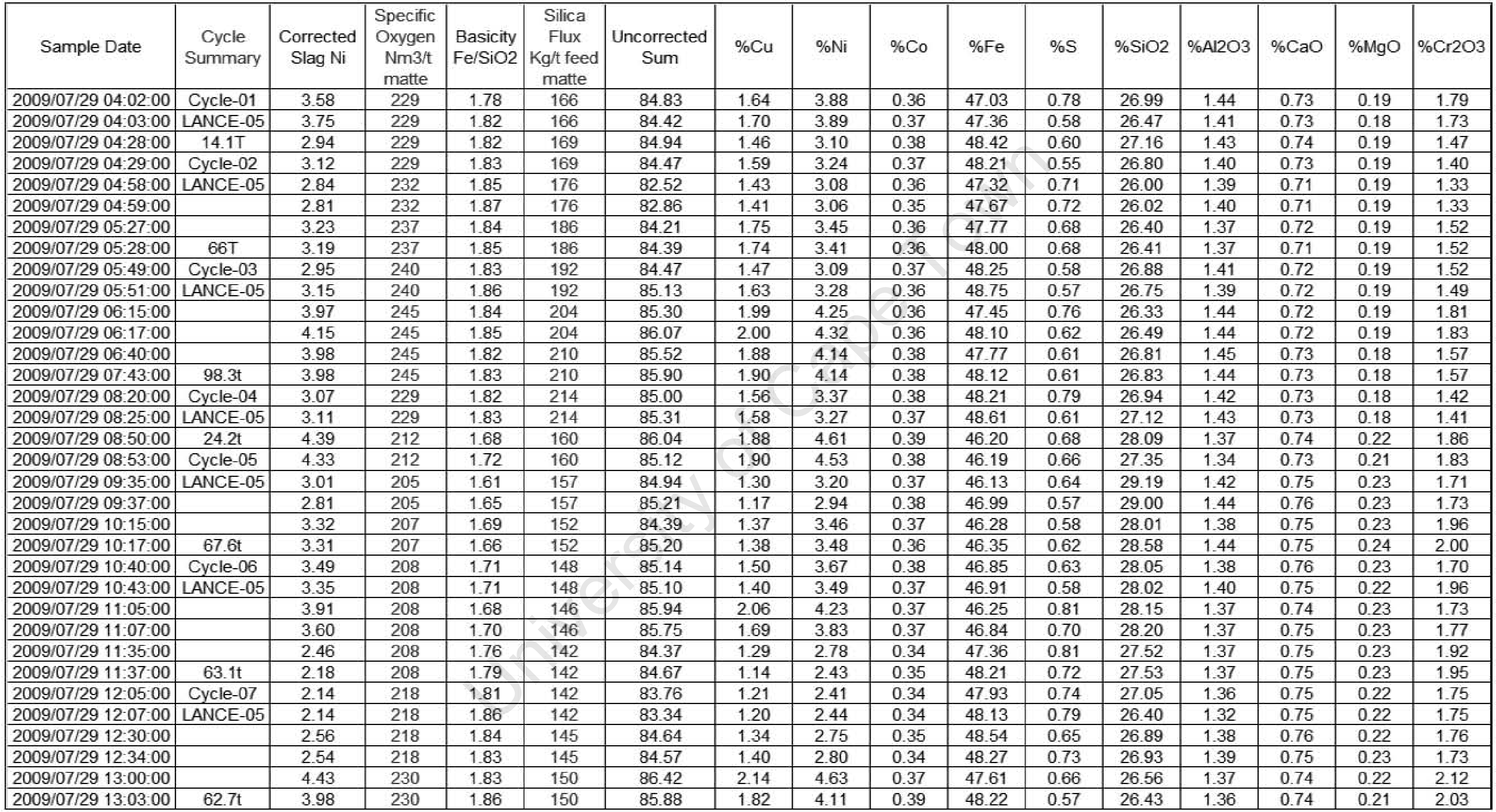
Sample Date	Cycle Summary	Corrected Slag Ni	Specific Oxygen Nm3/t matte	Basicity Fe/SiO2	Silica Flux Kg/t feed matte	Uncorrected Sum	%Cu	%Ni	%Co	%Fe	%S	%SiO2	%Al2O3	%CaO	%MgO	%Cr2O3
2009/07/01 04:29:00	Cycle-01	4.78	220	1.87	200	85.86	2.02	4.87	0.39	47.92	0.52	26.14	1.47	0.80	0.19	1.54
2009/07/01 04:33:00	Lance-04	4.67	220	1.79	200	87.62	2.01	4.89	0.40	48.17	0.68	27.52	1.54	0.81	0.21	1.39
2009/07/01 04:59:00		4.22	220	1.91	200	87.44	1.78	4.46	0.38	48.58	0.71	26.01	1.52	0.78	0.20	3.01
2009/07/01 05:03:00		4.29	220	1.93	200	86.43	1.93	4.49	0.38	48.31	0.66	25.53	1.49	0.78	0.19	2.66
2009/07/01 05:37:00		5.00	220	1.96	203	86.97	2.49	5.27	0.36	47.66	0.75	24.83	1.42	0.75	0.18	3.26
2009/07/01 05:41:00		5.45	220	1.94	203	88.68	2.88	5.94	0.35	47.60	1.03	25.05	1.45	0.74	0.20	3.44
2009/07/01 06:16:00		4.89	220	1.96	206	85.68	2.34	5.16	0.38	47.46	0.75	24.66	1.35	0.72	0.18	2.69
2009/07/01 06:18:00		4.31	220	1.89	206	87.15	1.97	4.61	0.38	48.18	0.79	25.96	1.44	0.74	0.20	2.88
2009/07/01 06:51:00		4.49	220	1.87	222	85.89	2.03	4.66	0.39	47.58	0.62	25.90	1.36	0.73	0.17	2.45
2009/07/01 06:53:00		4.19	220	1.80	222	87.30	1.81	4.39	0.39	47.99	0.65	27.15	1.43	0.74	0.19	2.55
2009/07/01 07:23:00	Cycle-02	4.30	220	1.77	230	87.34	1.93	4.50	0.40	47.86	0.66	27.63	1.40	0.74	0.19	2.03
2009/07/01 07:25:00	Lance-04	4.91	220	1.82	230	87.05	2.33	5.12	0.40	47.62	0.67	26.69	1.36	0.74	0.18	1.94
2009/07/01 08:49:00		5.29	220	1.75	230	87.97	2.57	5.51	0.40	47.08	0.68	27.42	1.48	0.76	0.19	1.88
2009/07/01 08:52:00		5.30	220	1.82	230	86.74	2.71	5.50	0.40	46.98	0.66	26.31	1.42	0.76	0.18	1.82
2009/07/01 09:19:00		4.04	220	1.76	230	87.15	1.67	4.25	0.41	48.03	0.67	27.86	1.45	0.75	0.19	1.87
2009/07/01 09:21:00		4.15	220	1.81	230	86.12	1.77	4.32	0.41	47.87	0.62	26.99	1.42	0.75	0.18	1.79
2009/07/01 14:37:00	Cycle-03	4.47	STBY	1.89	STBY	86.22	1.51	4.56	0.39	47.37	0.52	25.56	1.62	0.78	0.17	3.74
2009/07/01 14:42:00	Lance-04	3.61	STBY	1.87	STBY	85.29	0.97	3.63	0.38	47.36	0.42	25.84	1.75	0.78	0.19	3.98
2009/07/01 15:31:00		4.17	214	1.87	240	85.79	1.65	4.36	0.38	47.04	0.65	25.67	1.58	0.76	0.17	3.52
2009/07/01 15:35:00		3.84	214	1.84	240	86.14	1.42	4.07	0.38	47.16	0.69	26.08	1.66	0.75	0.18	3.75
2009/07/01 15:53:00		3.97	214	1.85	250	85.47	1.62	4.13	0.38	47.05	0.60	25.92	1.53	0.74	0.17	3.33
2009/07/01 15:57:00		3.77	214	1.79	250	86.40	1.47	3.95	0.38	47.07	0.63	26.87	1.62	0.74	0.18	3.49
2009/07/01 16:19:00		3.87	214	1.73	255	85.97	1.58	4.10	0.38	46.55	0.69	27.45	1.55	0.74	0.18	2.75
2009/07/01 16:22:00		3.99	214	1.72	255	86.46	1.53	4.12	0.38	46.76	0.57	27.69	1.56	0.74	0.18	2.93
2009/07/01 16:55:00		4.61	214	1.77	255	86.41	2.01	4.73	0.40	46.89	0.56	27.01	1.49	0.74	0.17	2.41
2009/07/01 16:56:00	71.4t	4.27	214	1.72	255	87.70	1.72	4.50	0.40	47.38	0.70	28.01	1.56	0.75	0.18	2.50
2009/07/01 17:24:00	Cycle 04	4.55	212	1.80	255	86.47	1.84	4.71	0.41	47.54	0.61	26.91	1.47	0.74	0.17	2.06
2009/07/01 17:27:00	Lance-04	4.26	212	1.78	255	86.14	1.70	4.53	0.40	47.17	0.74	26.99	1.50	0.73	0.18	2.20
2009/07/01 17:53:00		4.42	210	1.86	258	84.63	1.95	4.60	0.40	47.09	0.63	25.82	1.39	0.74	0.17	1.84
2009/07/01 17:58:00		3.99	210	1.77	258	85.98	1.71	4.31	0.40	47.29	0.81	27.17	1.47	0.73	0.18	1.91
2009/07/01 18:19:00		5.40	210	1.81	263	85.68	2.55	5.69	0.40	46.02	0.77	25.95	1.41	0.71	0.17	2.01
2009/07/01 18:23:00	57.7t	4.58	210	1.82	263	86.58	2.05	4.83	0.41	47.55	0.72	26.66	1.41	0.73	0.17	2.05
2009/07/01 18:55:00	Cycle-05	4.03	208	1.80	267	86.16	2.13	4.61	0.39	47.08	1.14	26.70	1.40	0.72	0.18	1.81
2009/07/01 18:59:00	Lance-04	4.04	208	1.80	267	86.16	2.13	4.62	0.39	47.06	1.14	26.71	1.41	0.72	0.18	1.80
2009/07/01 19:30:00		2.51	214	1.79	271	84.69	1.53	2.97	0.36	47.77	0.99	27.23	1.38	0.71	0.18	1.58
2009/07/01 19:34:00		2.62	214	1.82	271	85.41	1.52	3.01	0.37	48.53	0.90	27.21	1.37	0.73	0.17	1.60
2009/07/01 19:57:00		2.35	224	1.74	275	85.27	1.24	2.62	0.36	48.27	0.75	28.18	1.36	0.71	0.17	1.60
2009/07/01 20:05:00	72.8t	2.35	224	1.74	275	85.28	1.25	2.62	0.37	48.26	0.75	28.19	1.36	0.71	0.17	1.60
2009/07/01 20:26:00	Cycle-06	3.74	232	1.78	275	86.46	1.75	4.04	0.39	48.15	0.78	27.50	1.31	0.71	0.17	1.65
2009/07/01 20:29:00	Lance-04	3.96	232	1.75	275	86.41	1.78	4.19	0.38	47.72	0.70	27.83	1.30	0.70	0.17	1.63
2009/07/01 20:56:00		3.01	235	1.76	277	83.97	1.48	3.33	0.37	47.18	0.81	27.23	1.24	0.68	0.17	1.49
2009/07/01 21:03:00		3.06	235	1.77	277	84.36	1.54	3.40	0.38	47.31	0.83	27.28	1.24	0.68	0.17	1.53
2009/07/01 21:26:00		3.21	238	1.82	280	85.25	1.37	3.50	0.36	47.69	0.77	26.69	1.31	0.68	0.17	2.71
2009/07/01 21:30:00		2.89	238	1.86	280	84.04	1.13	3.12	0.37	47.67	0.70	26.12	1.30	0.68	0.17	2.79

Table A7: Converting process slag bath chemical assays control from one platinum smelter



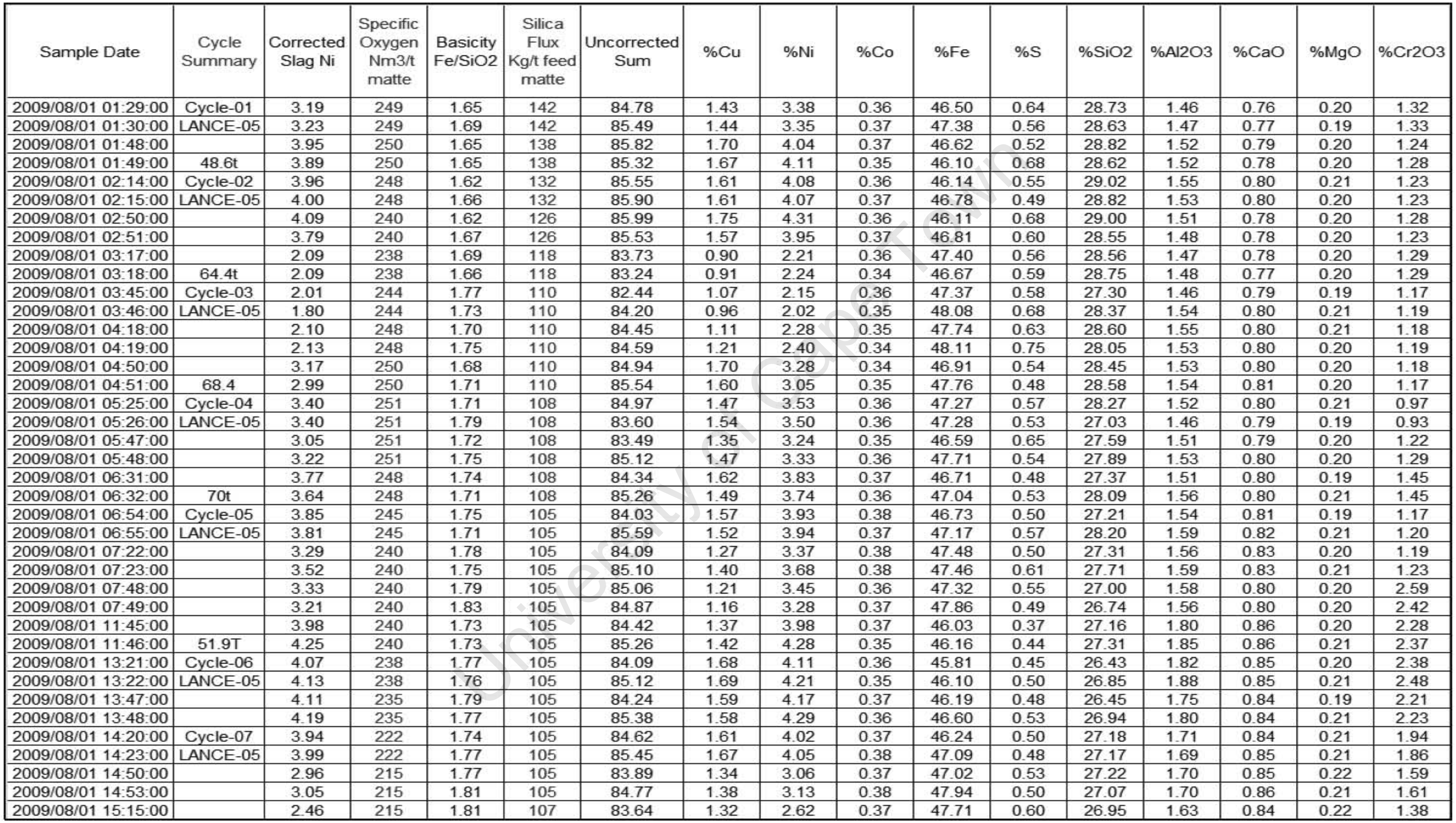
Sample Date	Cycle Summary	Corrected Slag Ni	Specific Oxygen Nm3/t matte	Basicity Fe/SiO2	Silica Flux Kg/t feed matte	Uncorrected Sum	%Cu	%Ni	%Co	%Fe	%S	%SiO2	%Al2O3	%CaO	%MgO	%Cr2O3
2009/07/02 01:39:00	Cycle-01	5.42	255	1.75	292	87.72	2.20	5.58	0.40	47.20	0.60	27.39	1.30	0.69	0.17	2.20
2009/07/02 01:59:00	Lance-04	5.46	255	1.73	292	88.96	2.02	5.62	0.40	47.68	0.61	28.01	1.32	0.70	0.17	2.42
2009/07/02 02:02:00		6.76	250	1.70	292	89.23	2.56	6.93	0.40	46.85	0.62	28.00	1.26	0.68	0.17	1.76
2009/07/02 02:19:00		6.50	250	1.75	292	88.01	2.56	6.62	0.41	46.86	0.56	27.21	1.25	0.68	0.16	1.70
2009/07/02 02:24:00		5.91	242	1.70	290	88.74	2.07	6.04	0.41	47.55	0.57	28.48	1.26	0.69	0.17	1.50
2009/07/02 02:45:00		6.05	235	1.79	290	87.12	2.28	6.21	0.41	47.06	0.61	26.77	1.24	0.69	0.16	1.68
2009/07/02 02:49:00		4.91	235	1.76	287	87.89	1.76	5.11	0.40	47.59	0.66	27.58	1.34	0.70	0.17	2.57
2009/07/02 05:36:00	72.1T	5.35	230	1.81	287	87.06	2.07	5.54	0.40	47.24	0.64	26.61	1.30	0.69	0.17	2.40
2009/07/02 05:40:00	Cycle-02	6.10	230	1.80	287	89.15	2.34	6.39	0.40	47.34	0.77	26.85	1.31	0.69	0.18	2.88
2009/07/02 05:44:00	Lance-03	6.46	225	1.89	289	87.59	2.51	6.69	0.39	46.91	0.70	25.19	1.24	0.66	0.16	3.14
2009/07/02 05:48:00		6.47	225	1.85	289	89.39	2.47	6.77	0.40	47.45	0.79	26.10	1.28	0.67	0.17	3.29
2009/07/02 06:37:00		3.37	205	1.94	295	85.52	1.14	3.64	0.37	48.54	0.74	25.51	1.24	0.67	0.16	3.51
2009/07/02 06:39:00		3.07	205	1.93	295	86.20	0.96	3.35	0.37	49.18	0.76	25.90	1.27	0.67	0.17	3.57
2009/07/02 07:17:00		5.00	207	2.02	295	85.83	2.70	5.62	0.34	47.43	1.20	23.88	1.16	0.66	0.16	2.68
2009/07/02 07:19:00		4.63	207	2.00	295	86.85	2.49	5.32	0.34	48.11	1.28	24.51	1.20	0.65	0.17	2.78
2009/07/02 08:02:00	Cycle-03	5.16	205	2.01	295	86.35	2.83	5.95	0.33	47.24	1.41	23.96	1.18	0.65	0.17	2.63
2009/07/02 08:39:00	Lance-03	4.38	205	1.69	295	87.14	2.33	4.93	0.31	46.34	1.10	27.86	1.19	0.65	0.16	2.26
2009/07/02 08:41:00		3.88	201	2.03	307	82.21	1.95	4.27	0.32	46.99	0.90	23.57	1.12	0.65	0.14	2.30
2009/07/02 10:52:00	Cycle-04	5.39	201	1.94	307	85.38	3.17	6.11	0.34	46.14	1.32	24.17	1.21	0.66	0.16	2.10
2009/07/02 10:55:00	Lance-03	6.11	201	1.93	307	87.96	3.86	7.27	0.34	46.07	1.89	24.35	1.24	0.66	0.17	2.10
2009/07/02 11:24:00		3.09	196	1.85	320	84.08	1.54	3.35	0.34	47.75	0.73	26.22	1.28	0.70	0.18	1.99
2009/07/02 11:28:00		2.18	196	1.75	320	85.89	1.05	2.41	0.34	48.72	0.70	28.32	1.35	0.69	0.17	2.14
2009/07/02 12:11:00		3.80	208	1.79	320	87.23	1.68	4.05	0.36	48.31	0.72	27.43	1.34	0.68	0.17	2.49
2009/07/02 12:14:00		3.82	208	1.82	320	85.98	1.91	4.09	0.36	47.85	0.75	26.70	1.29	0.68	0.16	2.19
2009/07/02 13:32:00	Cycle-05	3.63	208	1.73	320	87.56	1.66	3.95	0.37	47.97	0.81	28.30	1.33	0.69	0.17	2.31
2009/07/02 13:36:00	Lance-03	3.13	208	1.83	320	85.07	1.34	3.37	0.37	48.03	0.71	26.74	1.29	0.68	0.16	2.38
2009/07/02 13:56:00		2.33	212	1.76	322	86.23	1.01	2.60	0.37	48.77	0.74	28.15	1.31	0.68	0.17	2.44
2009/07/02 14:05:00		3.11	212	1.90	322	82.66	1.41	3.37	0.36	47.18	0.73	25.28	1.20	0.66	0.15	2.32
2009/07/02 14:22:00		2.83	220	1.79	324	85.21	1.32	3.01	0.37	48.17	0.63	27.36	1.24	0.67	0.16	2.28
2009/07/02 14:32:00	49.9t	2.81	220	1.72	324	85.89	1.38	3.15	0.36	47.64	0.84	28.16	1.27	0.66	0.16	2.26
2009/07/02 15:04:00	Cycle-06	3.47	230	1.75	326	84.61	1.46	3.66	0.35	46.81	0.65	27.19	1.25	0.65	0.16	2.43
2009/07/02 15:11:00	Lance-03	3.52	230	1.83	326	84.76	1.55	3.68	0.37	47.68	0.61	26.57	1.20	0.66	0.15	2.28
2009/07/02 15:36:00		4.77	233	1.80	330	85.35	2.11	5.02	0.37	46.16	0.72	26.09	1.29	0.66	0.17	2.77
2009/07/02 15:49:00		6.20	233	1.93	330	86.13	3.03	6.51	0.38	46.26	0.80	24.45	1.23	0.66	0.15	2.66
2009/07/02 16:09:00		4.68	230	1.81	336	86.62	1.92	4.88	0.38	47.06	0.66	26.49	1.34	0.67	0.17	3.05
2009/07/02 16:12:00	63.3t	6.40	230	1.85	336	87.77	3.02	6.67	0.39	46.52	0.74	25.56	1.28	0.67	0.16	2.76

Table A8: Converting process slag bath chemical assays control from one platinum smelter



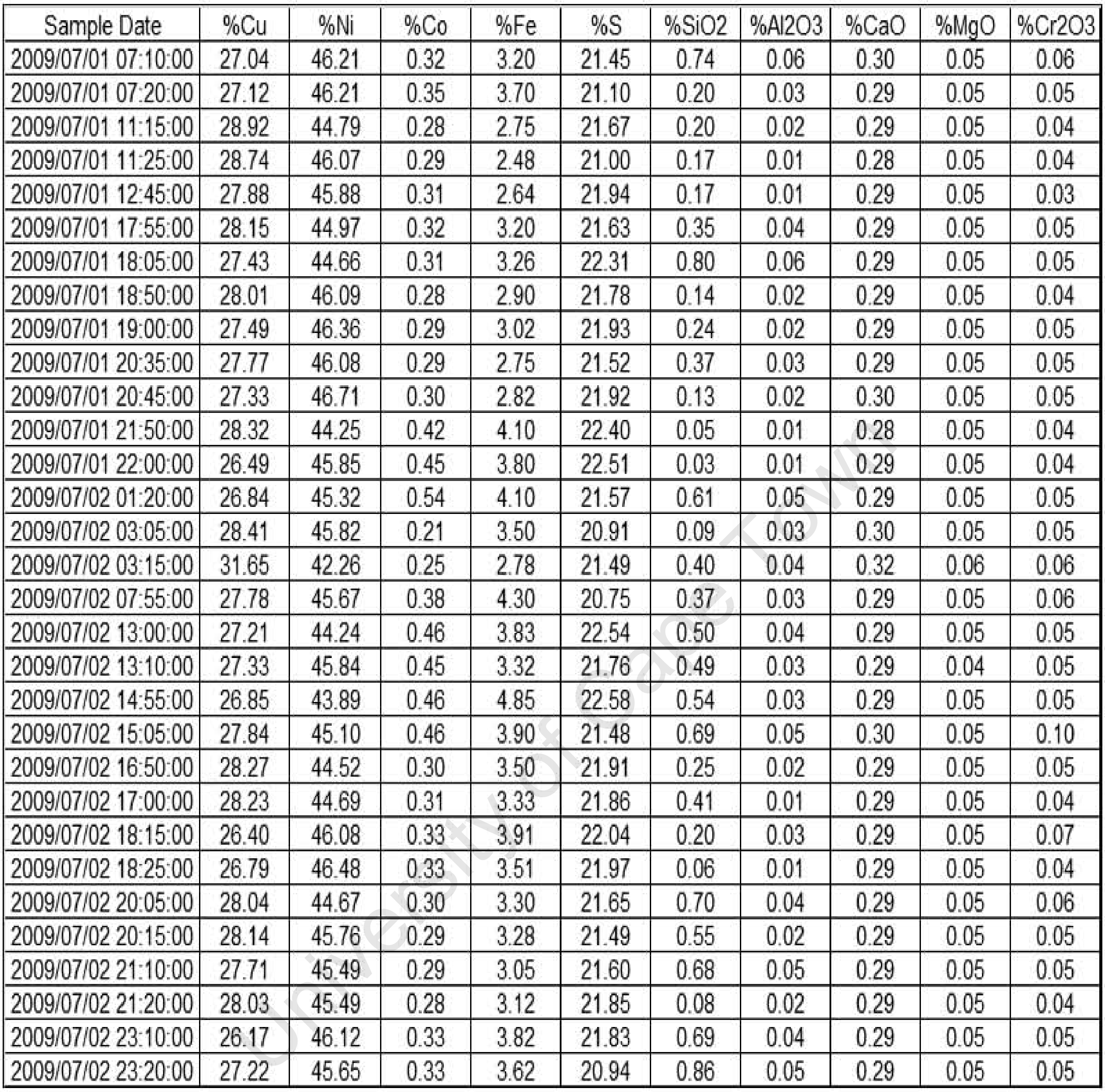
Sample Date	Cycle Summary	Corrected Slag Ni	Specific Oxygen Nm3/t matte	Basicity Fe/SiO2	Silica Flux Kg/t feed matte	Uncorrected Sum	%Cu	%Ni	%Co	%Fe	%S	%SiO2	%Al2O3	%CaO	%MgO	%Cr2O3
2009/07/29 04:02:00	Cycle-01	3.58	229	1.78	166	84.83	1.64	3.88	0.36	47.03	0.78	26.99	1.44	0.73	0.19	1.79
2009/07/29 04:03:00	LANCE-05	3.75	229	1.82	166	84.42	1.70	3.89	0.37	47.36	0.58	26.47	1.41	0.73	0.18	1.73
2009/07/29 04:28:00	14.1T	2.94	229	1.82	169	84.94	1.46	3.10	0.38	48.42	0.60	27.16	1.43	0.74	0.19	1.47
2009/07/29 04:29:00	Cycle-02	3.12	229	1.83	169	84.47	1.59	3.24	0.37	48.21	0.55	26.80	1.40	0.73	0.19	1.40
2009/07/29 04:58:00	LANCE-05	2.84	232	1.85	176	82.52	1.43	3.08	0.36	47.32	0.71	26.00	1.39	0.71	0.19	1.33
2009/07/29 04:59:00		2.81	232	1.87	176	82.86	1.41	3.06	0.35	47.67	0.72	26.02	1.40	0.71	0.19	1.33
2009/07/29 05:27:00		3.23	237	1.84	186	84.21	1.75	3.45	0.36	47.77	0.68	26.40	1.37	0.72	0.19	1.52
2009/07/29 05:28:00	66T	3.19	237	1.85	186	84.39	1.74	3.41	0.36	48.00	0.68	26.41	1.37	0.71	0.19	1.52
2009/07/29 05:49:00	Cycle-03	2.95	240	1.83	192	84.47	1.47	3.09	0.37	48.25	0.58	26.88	1.41	0.72	0.19	1.52
2009/07/29 05:51:00	LANCE-05	3.15	240	1.86	192	85.13	1.63	3.28	0.36	48.75	0.57	26.75	1.39	0.72	0.19	1.49
2009/07/29 06:15:00		3.97	245	1.84	204	85.30	1.99	4.25	0.36	47.45	0.76	26.33	1.44	0.72	0.19	1.81
2009/07/29 06:17:00		4.15	245	1.85	204	86.07	2.00	4.32	0.36	48.10	0.62	26.49	1.44	0.72	0.19	1.83
2009/07/29 06:40:00		3.98	245	1.82	210	85.52	1.88	4.14	0.38	47.77	0.61	26.81	1.45	0.73	0.18	1.57
2009/07/29 07:43:00	98.3t	3.98	245	1.83	210	85.90	1.90	4.14	0.38	48.12	0.61	26.83	1.44	0.73	0.18	1.57
2009/07/29 08:20:00	Cycle-04	3.07	229	1.82	214	85.00	1.56	3.37	0.38	48.21	0.79	26.94	1.42	0.73	0.18	1.42
2009/07/29 08:25:00	LANCE-05	3.11	229	1.83	214	85.31	1.58	3.27	0.37	48.61	0.61	27.12	1.43	0.73	0.18	1.41
2009/07/29 08:50:00	24.2t	4.39	212	1.68	160	86.04	1.88	4.61	0.39	46.20	0.68	28.09	1.37	0.74	0.22	1.86
2009/07/29 08:53:00	Cycle-05	4.33	212	1.72	160	85.12	1.90	4.53	0.38	46.19	0.66	27.35	1.34	0.73	0.21	1.83
2009/07/29 09:35:00	LANCE-05	3.01	205	1.61	157	84.94	1.30	3.20	0.37	46.13	0.64	29.19	1.42	0.75	0.23	1.71
2009/07/29 09:37:00		2.81	205	1.65	157	85.21	1.17	2.94	0.38	46.99	0.57	29.00	1.44	0.76	0.23	1.73
2009/07/29 10:15:00		3.32	207	1.69	152	84.39	1.37	3.46	0.37	46.28	0.58	28.01	1.38	0.75	0.23	1.96
2009/07/29 10:17:00	67.6t	3.31	207	1.66	152	85.20	1.38	3.48	0.36	46.35	0.62	28.58	1.44	0.75	0.24	2.00
2009/07/29 10:40:00	Cycle-06	3.49	208	1.71	148	85.14	1.50	3.67	0.38	46.85	0.63	28.05	1.38	0.76	0.23	1.70
2009/07/29 10:43:00	LANCE-05	3.35	208	1.71	148	85.10	1.40	3.49	0.37	46.91	0.58	28.02	1.40	0.75	0.22	1.96
2009/07/29 11:05:00		3.91	208	1.68	146	85.94	2.06	4.23	0.37	46.25	0.81	28.15	1.37	0.74	0.23	1.73
2009/07/29 11:07:00		3.60	208	1.70	146	85.75	1.69	3.83	0.37	46.84	0.70	28.20	1.37	0.75	0.23	1.77
2009/07/29 11:35:00		2.46	208	1.76	142	84.37	1.29	2.78	0.34	47.36	0.81	27.52	1.37	0.75	0.23	1.92
2009/07/29 11:37:00	63.1t	2.18	208	1.79	142	84.67	1.14	2.43	0.35	48.21	0.72	27.53	1.37	0.75	0.23	1.95
2009/07/29 12:05:00	Cycle-07	2.14	218	1.81	142	83.76	1.21	2.41	0.34	47.93	0.74	27.05	1.36	0.75	0.22	1.75
2009/07/29 12:07:00	LANCE-05	2.14	218	1.86	142	83.34	1.20	2.44	0.34	48.13	0.79	26.40	1.32	0.75	0.22	1.75
2009/07/29 12:30:00		2.56	218	1.84	145	84.64	1.34	2.75	0.35	48.54	0.65	26.89	1.38	0.76	0.22	1.76
2009/07/29 12:34:00		2.54	218	1.83	145	84.57	1.40	2.80	0.34	48.27	0.73	26.93	1.39	0.75	0.23	1.73
2009/07/29 13:00:00		4.43	230	1.83	150	86.42	2.14	4.63	0.37	47.61	0.66	26.56	1.37	0.74	0.22	2.12
2009/07/29 13:03:00	62.7t	3.98	230	1.86	150	85.88	1.82	4.11	0.39	48.22	0.57	26.43	1.36	0.74	0.21	2.03

Table A9: Converting process slag bath chemical assays control from one platinum smelter



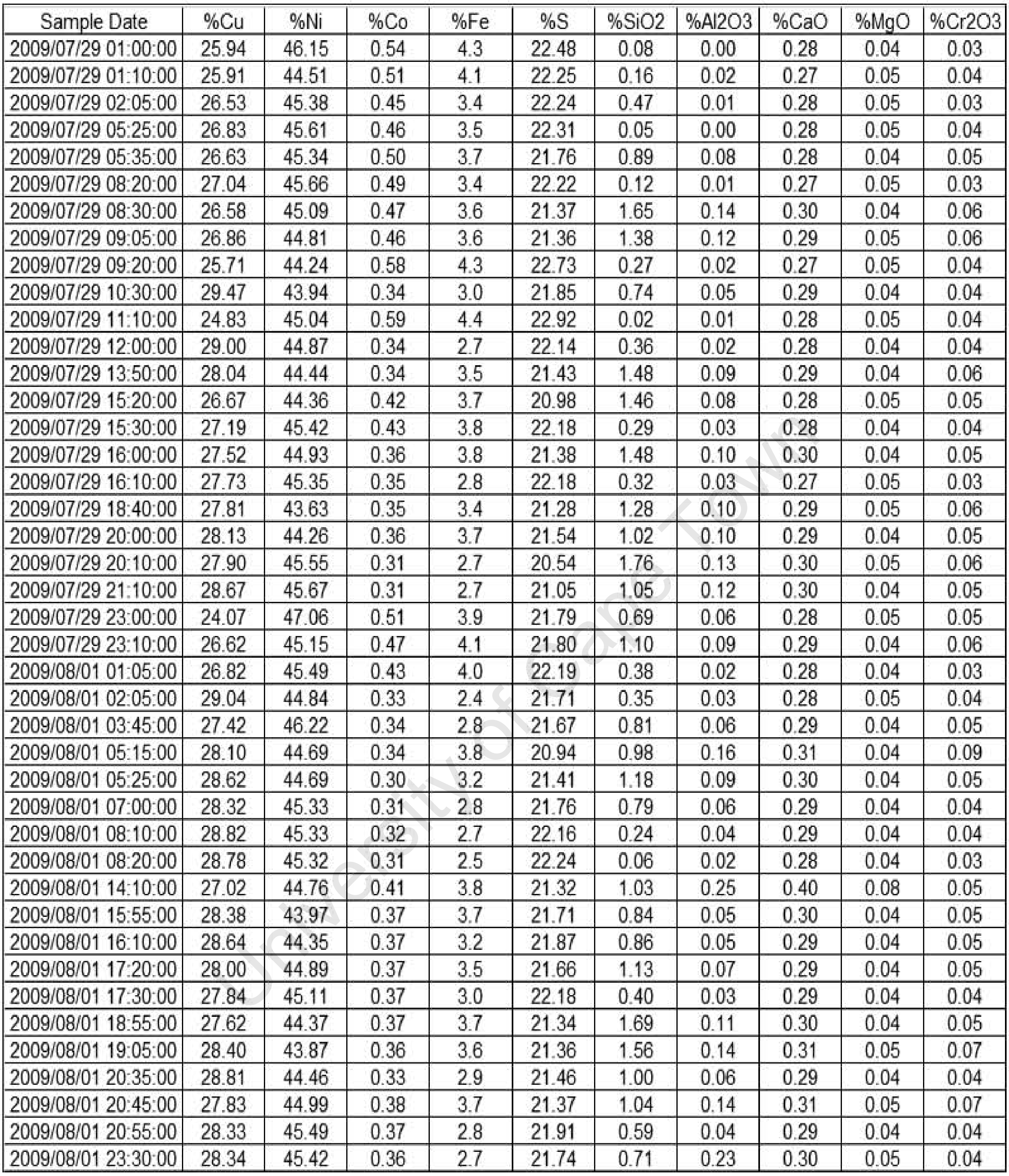
Sample Date	Cycle Summary	Corrected Slag Ni	Specific Oxygen Nm3/t matte	Basicity Fe/SiO2	Silica Flux Kg/t feed matte	Uncorrected Sum	%Cu	%Ni	%Co	%Fe	%S	%SiO2	%Al2O3	%CaO	%MgO	%Cr2O3
2009/08/01 01:29:00	Cycle-01	3.19	249	1.65	142	84.78	1.43	3.38	0.36	46.50	0.64	28.73	1.46	0.76	0.20	1.32
2009/08/01 01:30:00	LANCE-05	3.23	249	1.69	142	85.49	1.44	3.35	0.37	47.38	0.56	28.63	1.47	0.77	0.19	1.33
2009/08/01 01:48:00		3.95	250	1.65	138	85.82	1.70	4.04	0.37	46.62	0.52	28.82	1.52	0.79	0.20	1.24
2009/08/01 01:49:00	48.6t	3.89	250	1.65	138	85.32	1.67	4.11	0.35	46.10	0.68	28.62	1.52	0.78	0.20	1.28
2009/08/01 02:14:00	Cycle-02	3.96	248	1.62	132	85.55	1.61	4.08	0.36	46.14	0.55	29.02	1.55	0.80	0.21	1.23
2009/08/01 02:15:00	LANCE-05	4.00	248	1.66	132	85.90	1.61	4.07	0.37	46.78	0.49	28.82	1.53	0.80	0.20	1.23
2009/08/01 02:50:00		4.09	240	1.62	126	85.99	1.75	4.31	0.36	46.11	0.68	29.00	1.51	0.78	0.20	1.28
2009/08/01 02:51:00		3.79	240	1.67	126	85.53	1.57	3.95	0.37	46.81	0.60	28.55	1.48	0.78	0.20	1.23
2009/08/01 03:17:00		2.09	238	1.69	118	83.73	0.90	2.21	0.36	47.40	0.56	28.56	1.47	0.78	0.20	1.29
2009/08/01 03:18:00	64.4t	2.09	238	1.66	118	83.24	0.91	2.24	0.34	46.67	0.59	28.75	1.48	0.77	0.20	1.29
2009/08/01 03:45:00	Cycle-03	2.01	244	1.77	110	82.44	1.07	2.15	0.36	47.37	0.58	27.30	1.46	0.79	0.19	1.17
2009/08/01 03:46:00	LANCE-05	1.80	244	1.73	110	84.20	0.96	2.02	0.35	48.08	0.68	28.37	1.54	0.80	0.21	1.19
2009/08/01 04:18:00		2.10	248	1.70	110	84.45	1.11	2.28	0.35	47.74	0.63	28.60	1.55	0.80	0.21	1.18
2009/08/01 04:19:00		2.13	248	1.75	110	84.59	1.21	2.40	0.34	48.11	0.75	28.05	1.53	0.80	0.20	1.19
2009/08/01 04:50:00		3.17	250	1.68	110	84.94	1.70	3.28	0.34	46.91	0.54	28.45	1.53	0.80	0.20	1.18
2009/08/01 04:51:00	68.4	2.99	250	1.71	110	85.54	1.60	3.05	0.35	47.76	0.48	28.58	1.54	0.81	0.20	1.17
2009/08/01 05:25:00	Cycle-04	3.40	251	1.71	108	84.97	1.47	3.53	0.36	47.27	0.57	28.27	1.52	0.80	0.21	0.97
2009/08/01 05:26:00	LANCE-05	3.40	251	1.79	108	83.60	1.54	3.50	0.36	47.28	0.53	27.03	1.46	0.79	0.19	0.93
2009/08/01 05:47:00		3.05	251	1.72	108	83.49	1.35	3.24	0.35	46.59	0.65	27.59	1.51	0.79	0.20	1.22
2009/08/01 05:48:00		3.22	251	1.75	108	85.12	1.47	3.33	0.36	47.71	0.54	27.89	1.53	0.80	0.20	1.29
2009/08/01 06:31:00		3.77	248	1.74	108	84.34	1.62	3.83	0.37	46.71	0.48	27.37	1.51	0.80	0.19	1.45
2009/08/01 06:32:00	70t	3.64	248	1.71	108	85.26	1.49	3.74	0.36	47.04	0.53	28.09	1.56	0.80	0.21	1.45
2009/08/01 06:54:00	Cycle-05	3.85	245	1.75	105	84.03	1.57	3.93	0.38	46.73	0.50	27.21	1.54	0.81	0.19	1.17
2009/08/01 06:55:00	LANCE-05	3.81	245	1.71	105	85.59	1.52	3.94	0.37	47.17	0.57	28.20	1.59	0.82	0.21	1.20
2009/08/01 07:22:00		3.29	240	1.78	105	84.09	1.27	3.37	0.38	47.48	0.50	27.31	1.56	0.83	0.20	1.19
2009/08/01 07:23:00		3.52	240	1.75	105	85.10	1.40	3.68	0.38	47.46	0.61	27.71	1.59	0.83	0.21	1.23
2009/08/01 07:48:00		3.33	240	1.79	105	85.06	1.21	3.45	0.36	47.32	0.55	27.00	1.58	0.80	0.20	2.59
2009/08/01 07:49:00		3.21	240	1.83	105	84.87	1.16	3.28	0.37	47.86	0.49	26.74	1.56	0.80	0.20	2.42
2009/08/01 11:45:00		3.98	240	1.73	105	84.42	1.37	3.98	0.37	46.03	0.37	27.16	1.80	0.86	0.20	2.28
2009/08/01 11:46:00	51.9T	4.25	240	1.73	105	85.26	1.42	4.28	0.35	46.16	0.44	27.31	1.85	0.86	0.21	2.37
2009/08/01 13:21:00	Cycle-06	4.07	238	1.77	105	84.09	1.68	4.11	0.36	45.81	0.45	26.43	1.82	0.85	0.20	2.38
2009/08/01 13:22:00	LANCE-05	4.13	238	1.76	105	85.12	1.69	4.21	0.35	46.10	0.50	26.85	1.88	0.85	0.21	2.48
2009/08/01 13:47:00		4.11	235	1.79	105	84.24	1.59	4.17	0.37	46.19	0.48	26.45	1.75	0.84	0.19	2.21
2009/08/01 13:48:00		4.19	235	1.77	105	85.38	1.58	4.29	0.36	46.60	0.53	26.94	1.80	0.84	0.21	2.23
2009/08/01 14:20:00	Cycle-07	3.94	222	1.74	105	84.62	1.61	4.02	0.37	46.24	0.50	27.18	1.71	0.84	0.21	1.94
2009/08/01 14:23:00	LANCE-05	3.99	222	1.77	105	85.45	1.67	4.05	0.38	47.09	0.48	27.17	1.69	0.85	0.21	1.86
2009/08/01 14:50:00		2.96	215	1.77	105	83.89	1.34	3.06	0.37	47.02	0.53	27.22	1.70	0.85	0.22	1.59
2009/08/01 14:53:00		3.05	215	1.81	105	84.77	1.38	3.13	0.38	47.94	0.50	27.07	1.70	0.86	0.21	1.61
2009/08/01 15:15:00		2.46	215	1.81	107	83.64	1.32	2.62	0.37	47.71	0.60	26.95	1.63	0.84	0.22	1.38

Table A10: Converter product matte chemical assays from one platinum smelter



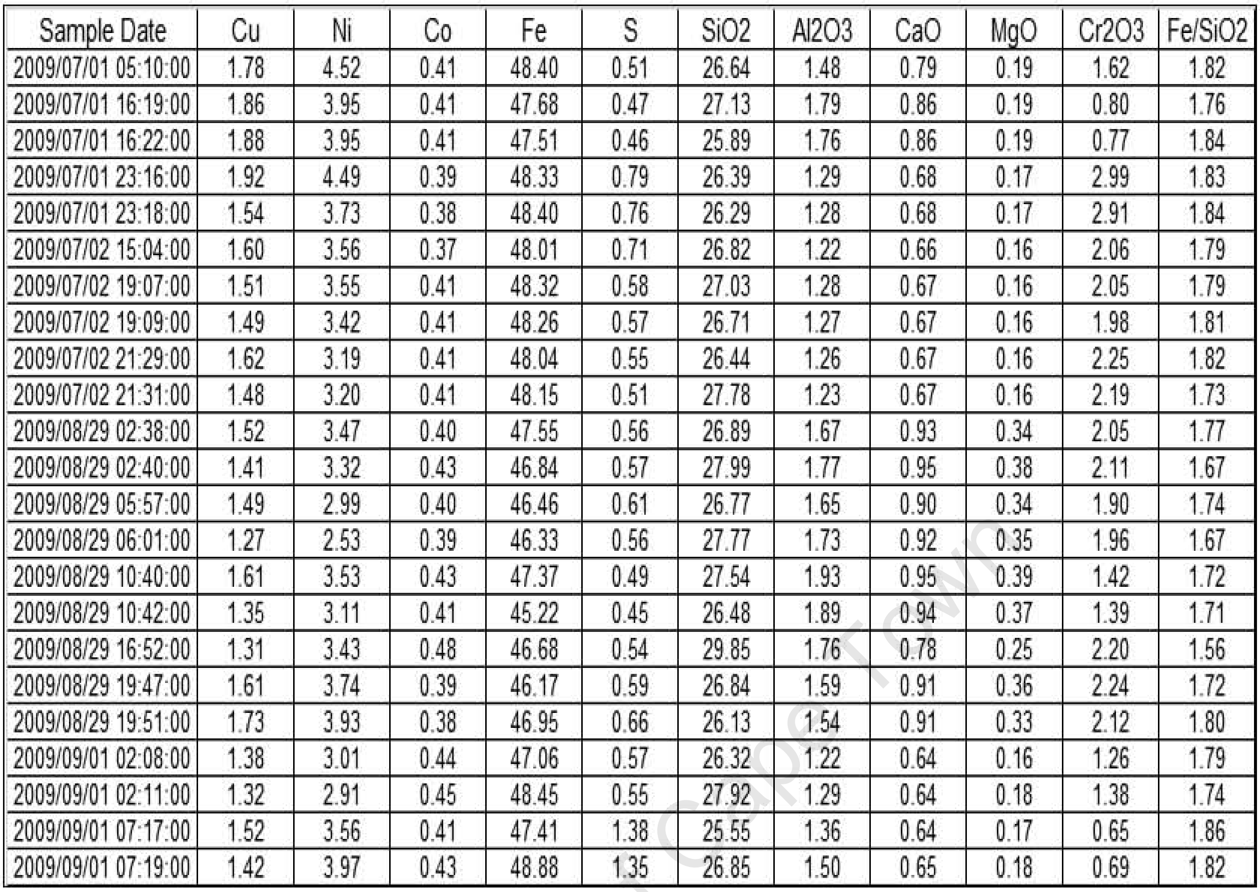
Sample Date	%Cu	%Ni	%Co	%Fe	%S	%SiO2	%Al2O3	%CaO	%MgO	%Cr2O3
2009/07/01 07:10:00	27.04	46.21	0.32	3.20	21.45	0.74	0.06	0.30	0.05	0.06
2009/07/01 07:20:00	27.12	46.21	0.35	3.70	21.10	0.20	0.03	0.29	0.05	0.05
2009/07/01 11:15:00	28.92	44.79	0.28	2.75	21.67	0.20	0.02	0.29	0.05	0.04
2009/07/01 11:25:00	28.74	46.07	0.29	2.48	21.00	0.17	0.01	0.28	0.05	0.04
2009/07/01 12:45:00	27.88	45.88	0.31	2.64	21.94	0.17	0.01	0.29	0.05	0.03
2009/07/01 17:55:00	28.15	44.97	0.32	3.20	21.63	0.35	0.04	0.29	0.05	0.05
2009/07/01 18:05:00	27.43	44.66	0.31	3.26	22.31	0.80	0.06	0.29	0.05	0.05
2009/07/01 18:50:00	28.01	46.09	0.28	2.90	21.78	0.14	0.02	0.29	0.05	0.04
2009/07/01 19:00:00	27.49	46.36	0.29	3.02	21.93	0.24	0.02	0.29	0.05	0.05
2009/07/01 20:35:00	27.77	46.08	0.29	2.75	21.52	0.37	0.03	0.29	0.05	0.05
2009/07/01 20:45:00	27.33	46.71	0.30	2.82	21.92	0.13	0.02	0.30	0.05	0.05
2009/07/01 21:50:00	28.32	44.25	0.42	4.10	22.40	0.05	0.01	0.28	0.05	0.04
2009/07/01 22:00:00	26.49	45.85	0.45	3.80	22.51	0.03	0.01	0.29	0.05	0.04
2009/07/02 01:20:00	26.84	45.32	0.54	4.10	21.57	0.61	0.05	0.29	0.05	0.05
2009/07/02 03:05:00	28.41	45.82	0.21	3.50	20.91	0.09	0.03	0.30	0.05	0.05
2009/07/02 03:15:00	31.65	42.26	0.25	2.78	21.49	0.40	0.04	0.32	0.06	0.06
2009/07/02 07:55:00	27.78	45.67	0.38	4.30	20.75	0.37	0.03	0.29	0.05	0.06
2009/07/02 13:00:00	27.21	44.24	0.46	3.83	22.54	0.50	0.04	0.29	0.05	0.05
2009/07/02 13:10:00	27.33	45.84	0.45	3.32	21.76	0.49	0.03	0.29	0.04	0.05
2009/07/02 14:55:00	26.85	43.89	0.46	4.85	22.58	0.54	0.03	0.29	0.05	0.05
2009/07/02 15:05:00	27.84	45.10	0.46	3.90	21.48	0.69	0.05	0.30	0.05	0.10
2009/07/02 16:50:00	28.27	44.52	0.30	3.50	21.91	0.25	0.02	0.29	0.05	0.05
2009/07/02 17:00:00	28.23	44.69	0.31	3.33	21.86	0.41	0.01	0.29	0.05	0.04
2009/07/02 18:15:00	26.40	46.08	0.33	3.91	22.04	0.20	0.03	0.29	0.05	0.07
2009/07/02 18:25:00	26.79	46.48	0.33	3.51	21.97	0.06	0.01	0.29	0.05	0.04
2009/07/02 20:05:00	28.04	44.67	0.30	3.30	21.65	0.70	0.04	0.29	0.05	0.06
2009/07/02 20:15:00	28.14	45.76	0.29	3.28	21.49	0.55	0.02	0.29	0.05	0.05
2009/07/02 21:10:00	27.71	45.49	0.29	3.05	21.60	0.68	0.05	0.29	0.05	0.05
2009/07/02 21:20:00	28.03	45.49	0.28	3.12	21.85	0.08	0.02	0.29	0.05	0.04
2009/07/02 23:10:00	26.17	46.12	0.33	3.82	21.83	0.69	0.04	0.29	0.05	0.05
2009/07/02 23:20:00	27.22	45.65	0.33	3.62	20.94	0.86	0.05	0.29	0.05	0.05

Table A11: Converter product matte chemical assays from one platinum smelter



Sample Date	%Cu	%Ni	%Co	%Fe	%S	%SiO2	%Al2O3	%CaO	%MgO	%Cr2O3
2009/07/29 01:00:00	25.94	46.15	0.54	4.3	22.48	0.08	0.00	0.28	0.04	0.03
2009/07/29 01:10:00	25.91	44.51	0.51	4.1	22.25	0.16	0.02	0.27	0.05	0.04
2009/07/29 02:05:00	26.53	45.38	0.45	3.4	22.24	0.47	0.01	0.28	0.05	0.03
2009/07/29 05:25:00	26.83	45.61	0.46	3.5	22.31	0.05	0.00	0.28	0.05	0.04
2009/07/29 05:35:00	26.63	45.34	0.50	3.7	21.76	0.89	0.08	0.28	0.04	0.05
2009/07/29 08:20:00	27.04	45.66	0.49	3.4	22.22	0.12	0.01	0.27	0.05	0.03
2009/07/29 08:30:00	26.58	45.09	0.47	3.6	21.37	1.65	0.14	0.30	0.04	0.06
2009/07/29 09:05:00	26.86	44.81	0.46	3.6	21.36	1.38	0.12	0.29	0.05	0.06
2009/07/29 09:20:00	25.71	44.24	0.58	4.3	22.73	0.27	0.02	0.27	0.05	0.04
2009/07/29 10:30:00	29.47	43.94	0.34	3.0	21.85	0.74	0.05	0.29	0.04	0.04
2009/07/29 11:10:00	24.83	45.04	0.59	4.4	22.92	0.02	0.01	0.28	0.05	0.04
2009/07/29 12:00:00	29.00	44.87	0.34	2.7	22.14	0.36	0.02	0.28	0.04	0.04
2009/07/29 13:50:00	28.04	44.44	0.34	3.5	21.43	1.48	0.09	0.29	0.04	0.06
2009/07/29 15:20:00	26.67	44.36	0.42	3.7	20.98	1.46	0.08	0.28	0.05	0.05
2009/07/29 15:30:00	27.19	45.42	0.43	3.8	22.18	0.29	0.03	0.28	0.04	0.04
2009/07/29 16:00:00	27.52	44.93	0.36	3.8	21.38	1.48	0.10	0.30	0.04	0.05
2009/07/29 16:10:00	27.73	45.35	0.35	2.8	22.18	0.32	0.03	0.27	0.05	0.03
2009/07/29 18:40:00	27.81	43.63	0.35	3.4	21.28	1.28	0.10	0.29	0.05	0.06
2009/07/29 20:00:00	28.13	44.26	0.36	3.7	21.54	1.02	0.10	0.29	0.04	0.05
2009/07/29 20:10:00	27.90	45.55	0.31	2.7	20.54	1.76	0.13	0.30	0.05	0.06
2009/07/29 21:10:00	28.67	45.67	0.31	2.7	21.05	1.05	0.12	0.30	0.04	0.05
2009/07/29 23:00:00	24.07	47.06	0.51	3.9	21.79	0.69	0.06	0.28	0.05	0.05
2009/07/29 23:10:00	26.62	45.15	0.47	4.1	21.80	1.10	0.09	0.29	0.04	0.06
2009/08/01 01:05:00	26.82	45.49	0.43	4.0	22.19	0.38	0.02	0.28	0.04	0.03
2009/08/01 02:05:00	29.04	44.84	0.33	2.4	21.71	0.35	0.03	0.28	0.05	0.04
2009/08/01 03:45:00	27.42	46.22	0.34	2.8	21.67	0.81	0.06	0.29	0.04	0.05
2009/08/01 05:15:00	28.10	44.69	0.34	3.8	20.94	0.98	0.16	0.31	0.04	0.09
2009/08/01 05:25:00	28.62	44.69	0.30	3.2	21.41	1.18	0.09	0.30	0.04	0.05
2009/08/01 07:00:00	28.32	45.33	0.31	2.8	21.76	0.79	0.06	0.29	0.04	0.04
2009/08/01 08:10:00	28.82	45.33	0.32	2.7	22.16	0.24	0.04	0.29	0.04	0.04
2009/08/01 08:20:00	28.78	45.32	0.31	2.5	22.24	0.06	0.02	0.28	0.04	0.03
2009/08/01 14:10:00	27.02	44.76	0.41	3.8	21.32	1.03	0.25	0.40	0.08	0.05
2009/08/01 15:55:00	28.38	43.97	0.37	3.7	21.71	0.84	0.05	0.30	0.04	0.05
2009/08/01 16:10:00	28.64	44.35	0.37	3.2	21.87	0.86	0.05	0.29	0.04	0.05
2009/08/01 17:20:00	28.00	44.89	0.37	3.5	21.66	1.13	0.07	0.29	0.04	0.05
2009/08/01 17:30:00	27.84	45.11	0.37	3.0	22.18	0.40	0.03	0.29	0.04	0.04
2009/08/01 18:55:00	27.62	44.37	0.37	3.7	21.34	1.69	0.11	0.30	0.04	0.05
2009/08/01 19:05:00	28.40	43.87	0.36	3.6	21.36	1.56	0.14	0.31	0.05	0.07
2009/08/01 20:35:00	28.81	44.46	0.33	2.9	21.46	1.00	0.06	0.29	0.04	0.04
2009/08/01 20:45:00	27.83	44.99	0.38	3.7	21.37	1.04	0.14	0.31	0.05	0.07
2009/08/01 20:55:00	28.33	45.49	0.37	2.8	21.91	0.59	0.04	0.29	0.04	0.04
2009/08/01 23:30:00	28.34	45.42	0.36	2.7	21.74	0.71	0.23	0.30	0.05	0.04

Table A12: Converter slag chemical assays from one platinum smelter



Sample Date	Cu	Ni	Co	Fe	S	SiO2	Al2O3	CaO	MgO	Cr2O3	Fe/SiO2
2009/07/01 05:10:00	1.78	4.52	0.41	48.40	0.51	26.64	1.48	0.79	0.19	1.62	1.82
2009/07/01 16:19:00	1.86	3.95	0.41	47.68	0.47	27.13	1.79	0.86	0.19	0.80	1.76
2009/07/01 16:22:00	1.88	3.95	0.41	47.51	0.46	25.89	1.76	0.86	0.19	0.77	1.84
2009/07/01 23:16:00	1.92	4.49	0.39	48.33	0.79	26.39	1.29	0.68	0.17	2.99	1.83
2009/07/01 23:18:00	1.54	3.73	0.38	48.40	0.76	26.29	1.28	0.68	0.17	2.91	1.84
2009/07/02 15:04:00	1.60	3.56	0.37	48.01	0.71	26.82	1.22	0.66	0.16	2.06	1.79
2009/07/02 19:07:00	1.51	3.55	0.41	48.32	0.58	27.03	1.28	0.67	0.16	2.05	1.79
2009/07/02 19:09:00	1.49	3.42	0.41	48.26	0.57	26.71	1.27	0.67	0.16	1.98	1.81
2009/07/02 21:29:00	1.62	3.19	0.41	48.04	0.55	26.44	1.26	0.67	0.16	2.25	1.82
2009/07/02 21:31:00	1.48	3.20	0.41	48.15	0.51	27.78	1.23	0.67	0.16	2.19	1.73
2009/08/29 02:38:00	1.52	3.47	0.40	47.55	0.56	26.89	1.67	0.93	0.34	2.05	1.77
2009/08/29 02:40:00	1.41	3.32	0.43	46.84	0.57	27.99	1.77	0.95	0.38	2.11	1.67
2009/08/29 05:57:00	1.49	2.99	0.40	46.46	0.61	26.77	1.65	0.90	0.34	1.90	1.74
2009/08/29 06:01:00	1.27	2.53	0.39	46.33	0.56	27.77	1.73	0.92	0.35	1.96	1.67
2009/08/29 10:40:00	1.61	3.53	0.43	47.37	0.49	27.54	1.93	0.95	0.39	1.42	1.72
2009/08/29 10:42:00	1.35	3.11	0.41	45.22	0.45	26.48	1.89	0.94	0.37	1.39	1.71
2009/08/29 16:52:00	1.31	3.43	0.48	46.68	0.54	29.85	1.76	0.78	0.25	2.20	1.56
2009/08/29 19:47:00	1.61	3.74	0.39	46.17	0.59	26.84	1.59	0.91	0.36	2.24	1.72
2009/08/29 19:51:00	1.73	3.93	0.38	46.95	0.66	26.13	1.54	0.91	0.33	2.12	1.80
2009/09/01 02:08:00	1.38	3.01	0.44	47.06	0.57	26.32	1.22	0.64	0.16	1.26	1.79
2009/09/01 02:11:00	1.32	2.91	0.45	48.45	0.55	27.92	1.29	0.64	0.18	1.38	1.74
2009/09/01 07:17:00	1.52	3.56	0.41	47.41	1.38	25.55	1.36	0.64	0.17	0.65	1.86
2009/09/01 07:19:00	1.42	3.97	0.43	48.88	1.35	26.85	1.50	0.65	0.18	0.69	1.82