Ferroalloy extraction from a Zimbabwean chrome ore using a closed DC furnace

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ABSTRACT

The depletion of high-grade hard and lumpy chrome ores has forced Zimbabwean ferrochrome producers to resort to low-grade friable ores that have an average of 28 per cent fines against a set limit of 12 per cent. Submerged open arc furnaces currently in use are associated with eruptions within the furnace because of the high amounts of fines, high energy consumption, as well as high SO_2 and CO_2 emissions. Traditionally employed agglomeration methods such as pelletizing and the use of coke in the furnace have been discarded because of the pressure to lower operational costs and to achieve higher profit margins. This research aims to develop a method to incorporate low-grade friable ore fines and unprocessed coal into production while lowering CO_2 and SO_2 emissions through a closed direct current (DC) arc furnace. The closed DC arc furnace could incorporate the friable ores and coal while maintaining above 90 per cent reduction rates. Preliminary findings also show that energy consumption could be reduced by up to 35 per cent by incorporating a pre-heating and pre-reduction system using the flue gases from the furnace. In the proposed circuit, the cost of production per ton of ferrochrome might be potentially lowered by an average of 30 per cent.

INTRODUCTION

Global ferrochrome production has been steadily increasing over the past few decades, primarily due to the growing demand for ferrochrome by-products. This trend is expected to continue as industrial and usage demands for ferrochrome continue to rise (Fortune Business Insights, 2023) due to the rise in demand for stainless steel (du Preez, et al., 2023). In Zimbabwe, ferrochrome production is mainly from four smelters (Gumbie, et al., 2011) where the most common method used for ferrochrome production is the Submerged Arc Furnace (SAF) that uses an electric current to increase temperature enhancing the reduction reactions of chromite (Yang, et al., 2023).

The depletion of high-grade lumpy ores has resulted in the utilization of fine ores in open alternative current (AC) furnaces causing frequent eruptions and high energy consumption, approximately 3.8 to 4.0 MWh/MT FeCr (Gumbie, et al., 2011). Agglomeration techniques have proved to be expensive for Zimbabwean ferrochrome producers and as such, there is a need to find new ways to smelt chromite fines to produce ferrochrome using methods that are cost-effective and environmentally friendly. A possible alternative to an open AC furnace is the closed direct current (DC) furnace, which will incorporate pre-treatment techniques and utilize low-grade fine ores.

The conventional smelting of ferrochrome is done in electric reduction furnaces with electrodes submerged in the burden materials, ore-bearing material, a carbonaceous reductant, and fluxes. For high efficiencies, the gas flow should be uniform, and channelling of the CO produced during the reactions should not be avoided. This is achieved by the use of hard lumpy ores that are not friable to avoid excessive degradation and generation of fines in the furnace. The depletion of hard lumpy ores and the increased mechanization in mining operations has resulted in considerable utilization of fine ore material of much smaller grain sizes that require agglomeration, and the use of friable ores that generate fines (Goel, 1997).

Agglomeration of chromite fines and friable material currently utilizes a combination of sintering and briquetting methods (Xiaohai, et al., 2019). This improves the smelting efficiency while simultaneously reducing the power consumption of the furnace. Electrical energy consumption typically ranges from 3.0 to 3.5 MWh/MT alloy for sintered or briquette feed, compared to 3.8 to 4.0 MWh/MT alloy for raw fines (Gumbie, et al., 2011).

The DC arc furnace was developed as a way of allowing for the direct smelting of fine chromite that emanated from highly friable ores without the need for costly agglomeration techniques (Jone and Erwee, 2016). Non-coking coal can also be used as the carbonaceous reductant and theoretically, the smelter power input is independent of the burden composition since the resistance is manipulated by the arc length, making it a low-cost technology for ferrochrome production (Grant, et al., 2010). The DC arc uses a single solid carbon electrode as the cathode and is normally open or semi-submerged (Naiker, 2007). Burden materials can be charged directly into the furnace or through a hollow electrode.

Innovations in smelting technology, such as the Premus process have been developed that can decrease electrical energy consumption during smelting by partly reducing pelletized chromite ores in a rotary kiln using energy obtained from combustion of pulverized coal and hot gases generated from the closed submerged arc furnace (Chima, 2012). Pre-reduction and pre-heating techniques that have been employed enhanced the quality and the grade of the ferrochrome product, by increasing the chromium recovery and the chromium-to-iron ratio and decreasing the carbon and silicon contents. Pre-reduction of pelletised chromite ore was the best option with the lowest specific energy consumption using the submerged arc furnace (Neizel, et al., 2013). Generally, higher temperatures (+ 1000 °C) longer times (+60 minutes), finer particles (-10 mm), and higher reductant ratios are favorable for achieving higher degrees of pre-reduction (Jian, et al., 2014).

In this study, a novel structure of the DC arc furnace is being investigated for ferrochrome production in an attempt to utilize fine chromite ore while maintaining low power consumption. In addition, this new structure is expected to process low-grade fine ore material and friable ores while avoiding agglomeration of the fine ore through sintering and briquetting techniques which are expensive and economically non-viable for local smelters. It is also the aim of this study to investigate the effect of pre-heating and pre-reduction techniques on the recovery of metal using off-gas.

EXPERIMENTAL

Materials

The chromite ore fines used in this study were low-grade (A), high-grade (B) and middle-grade (C). which were obtained from Shurugwi, Mutorashanga, and Ngezi chrome fields. The grades of the ores are classified according to the percentage of Cr_2O_3 as shown in **Table 1**. The coal was obtained from Zambezi Gas Company in Hwange, and the quartz from Kwekwe.

TABLE 1- Chemical composition of different chromite ore grades

Sample ID	**% Cr_2O_3**	**% FeO**	**Cr/Fe**	**% SiO_2**	**% MgO**	**% Al_2O_3**	**% CaO**	**% TiO_2**	**% P**	**% Others**
Low Grade fines (A)	25.13	18.71	1.19	17.29	24.45	5.23	1.38	0.17	0.0011	7.64
High Grade Fines(B)	40.23	17.7	2.18	7.46	15.03	10.41	0.43	0.33	0.001	8.41
Middle grade fines (C)	32.71	16.63	1.89	11.73	20.42	7.40	1.24	0.2	0.0013	9.67

Sample preparations and experimental procedures

Ten samples of 1 kg each were randomly taken from stockpiles of fine ores using an ore sample cutter of 3 m height to obtain a good representation of ore. To determine the particle size distribution of the ores, the samples were first dried using drying ovens at 110 °C for 20 hours and then sieved with different sieve sizes to obtain a representative particle size distribution shown in FIG 1.

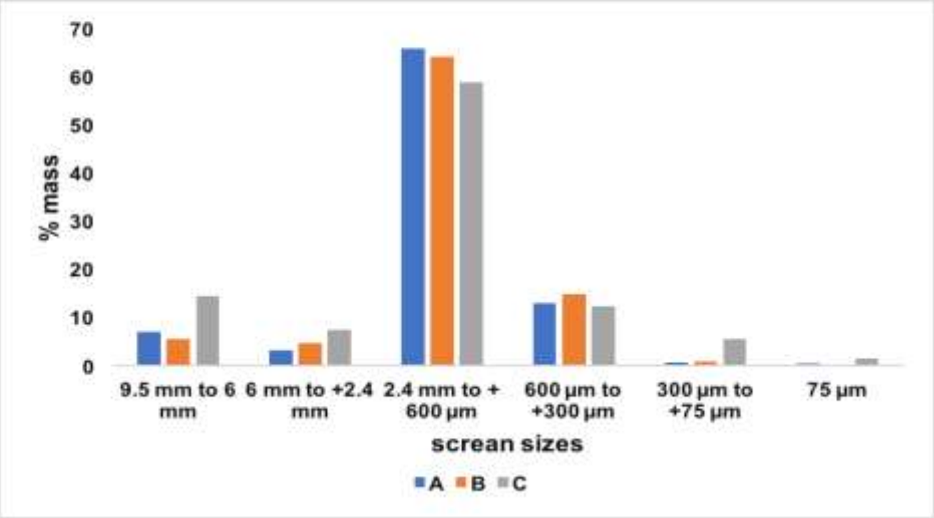

FIG 1 - Particle size distribution of the chromite ore sample

An average of 65.8 per cent of particles fall within the size range of 2.4 mm to +600 µm. This indicates that the chromite ore samples are primarily composed of fine-sized particles. A noticeable 4 per cent of those with sizes of 75 µm and below, indicate that the sample contains a relatively small proportion of ultra-fine particles. The sample also includes a notable proportion of large particles above 6 mm, constituting 18 per cent of the samples.

To obtain a homogeneous feed of less than 2 mm, the chromite ores were first crushed using a laboratory jaw crusher, then pulverized then assayed for the average chrome head grade using a wet chemical analysis and an Inductively Coupled Plasma - Optical Emission Spectroscopy (ICP- OES). The chemical composition of the chromite ore is shown in **Table 1**. Quartz, used for basicity adjustment, was crushed using a laboratory jaw crusher from 40 - 50 mm to below 2 mm and then dried using an oven, followed by pulverization and chemical analysis. The coal fines were sieved and screened to 90 per cent fines in the range of 1 mm to 2 mm. Results from the characterizations of the coal and quartz are shown in **Table 2** and **3** respectively.

TABLE 2 – Coal proximate analysis

Material (wt%)	Volatile Combusted Matter	Ash	Fixed carbon	S	P	Moisture (%)
ZG Coal	22.05	9.92	66.8086	1.01	0.0014	0.21

The proximate analysis for the coal fines was done according to the Indian Standard IS:1350 (Part-I)-1984.

TABLE 3 – Quartz characterisation

Material (wt %)	SiO_2	FeO
Quartz	99.37	0.63

The samples were blended according to the desired mass proportions of different chromite ores as shown in **Table 4** and the chemical composition of the blends is shown in **Table 5.**

TABLE 4 – The mass blend proportions of the chromite ores

Sample ID	Blend 1 (%)	Blend 2 (%)	Blend 3 (%)	Blend 4 (%)	Blend 5 (%)	Blend 6 (%)
A	100	80	70	60	40	0
B	0	10	15	30	50	100
C	0	10	10	10	10	0

TABLE 5 - Chemical composition of the different blends

Sample ID	% Cr_2O_3	% FeO	Cr/Fe	% SiO_2	% MgO	% Al_2O_3	% CaO	% TiO_2
Blend 1	25.13	17.91	1.79	11.70	19.32	8.13	0.85	0.25
Blend 2	27.03	18.71	1.79	12.01	19.85	7.92	0.90	0.24
Blend 3	28.33	19.46	1.84	11.83	19.90	7.97	0.89	0.22
Blend 4	30.73	20.96	1.95	11.46	20.01	8.07	0.88	0.23
Blend 5	33.73	23.21	1.89	11.65	19.95	8.02	0.89	0.23
Blend 6	40.23	17.7	2.18	7.46	15.03	10.41	0.43	8.41

Table 6. The amount of raw material used in each blend

Sample ID	Chromite(g)	Coal (g)	Quartz(g)
Blend 1	100	18.608	54.831
Blend 2	100	19.8	53.67
Blend 3	100	20.695	54.45
Blend 4	100	22.39	53.23
Blend 5	100	24.655	54.17
Blend 6	100	25.547	53.987

A mixture of 100 g of chromite and quartz (flux material for separation between gangue and ferrochrome), in proportions shown in Table 6, firstly underwent pre-treatment in a muffle furnace using carbon monoxide from gas cylinders. The temperature within the muffle furnace was deliberately altered in order to observe and analyze the production of metallic chromium and iron (pre-reduction). Then subsequently for smelting in a laboratory-scale closed DC furnace made from the magnisite refacrory paste, coal as reducing agent was added into the mixture. The amount of coal used in every reaction was based on stoichiometry. The operational voltage was established at 500 volts, while a current of 60 amperes was utilized. The power supply employed for this operation was a robust DC welding machine designed for heavy-duty applications. The power-conducting media within the furnace consisted of gauging rods, with 80 per cent graphite content. The furnace was sealed with a lid which was made from the same material, which has two holes where the electrodes are introduced.

Each sample was smelted for 30 seconds after carrying out preliminary experiments on reaction time, taking into consideration the size of the sample and the crucible used, as elucidated by Asish et al, (2020) and Pankaj et al, (2024). The products were discharged from the top of the furnace and allowed to cool down with natural air. Once cooled, the alloy and slag were analyzed using an ICP-OES technique and wet chemical analysis. FIG 2 illustrates the experimental stages.

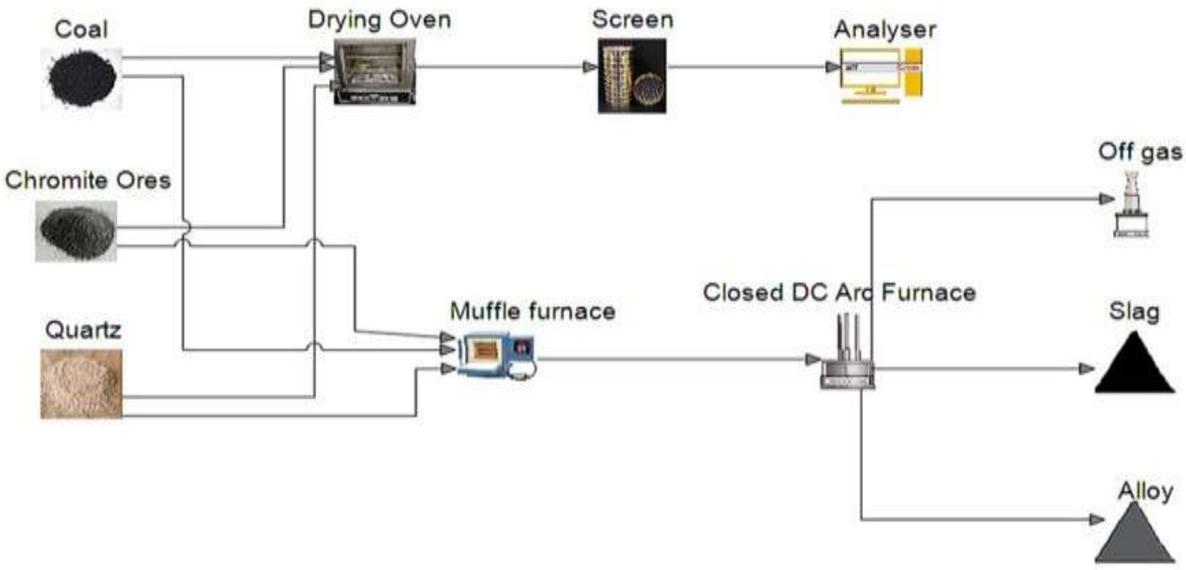

FIG 2 – Schematic presentation of ferrochrome production using DC arc furnace

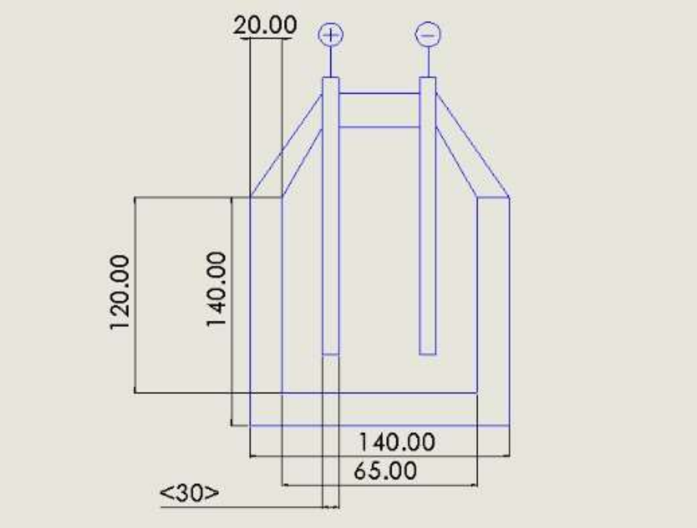

FIG 3 –Technical drawing of the furnace used for experiments (dimensions in mm)

RESULTS AND DISCUSSION

Effect of chrome grade on metal recovery

This was done to investigate the effect of chromite grade on the metal recovery. The chromite grade was achieved by varying the different proportions of the chromite ores, as shown in **Table 4.** The results from the six blends are shown in FIG 4 showing the tapping stage recovery (TSR) based on the changes in blends.Equation 1 How the TSR is determined

$$TSR = \frac{Amount\ of\ Cr\ in\ the\ alloy}{Amount\ of\ Cr\ in\ the\ ore} \times 100 \qquad (1)$$

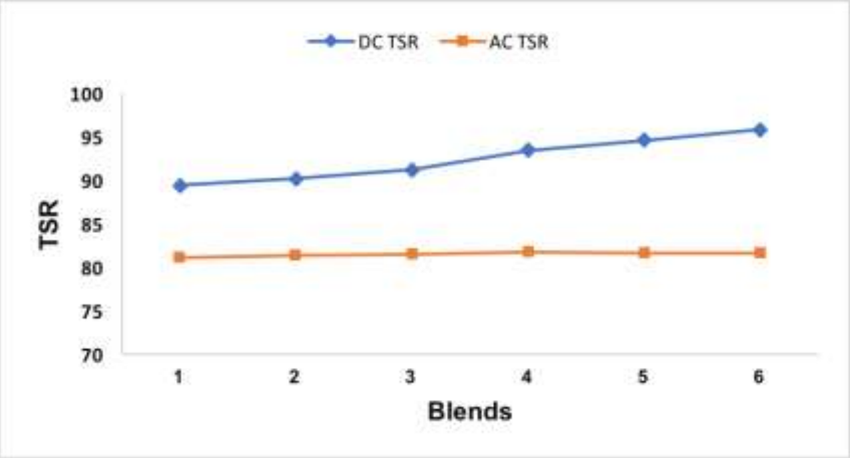

FIG 4 – Trends in tapping stage recovery based on the changes in Cr_2O_3 and the Cr/Fe ratio

It can be observed from FIG 4 that there is an increase in the TSR for the DC arc from about 89 per cent to approximately 96 per cent on the 6th blend, which can be attributed to the increase in both Cr_2O_3 content and the Cr/Fe ratio from blend 1 to blend 6. The results obtained from the AC furnace serve as a reference point for comparison purposes, indicating the high energy conversion efficiency of the closed DC arc furnace. This implies that a significant portion of the input energy is effectively utilized in the DC furnace, resulting in improved recovery rates during the tapping stage. The chemical composition of the tapped alloy is shown in **Table 7**.

TABLE 7 – Alloy chemical composition

Sample ID	Cr (%)	Fe (%)	Cr/Fe	Si (%)	Mn (%)	P (%)	Ni (%)	Co (%)	Ti (%)	V (%)	C (%)
Blend 1	59.75	28.97	2.06	1.00	0.28	0.03	0.33	0.06	0.30	0.20	7.79
Blend 2	61.45	32.27	1.90	1.76	0.20	0.02	0.37	0.06	0.12	0.21	7.52
Blend 3	62.54	33.04	1.89	1.33	0.18	0.02	0.39	0.06	0.08	0.21	7.46
Blend 4	62.86	32.96	1.91	1.29	0.18	0.02	0.39	0.06	0.08	0.21	7.35
Blend 5	64.98	33.70	1.92	1.23	0.17	0.02	0.41	0.07	0.08	0.21	7.09
Blend 6	67.56	33.90	1.99	1.30	0.17	0.02	0.41	0.07	0.08	0.20	7.15

Generally, the percentage of Cr and Fe in the alloy increases as the Cr_2O_3 and FeO grades in the blends increase. There is a noticeable increase in the percentage of Cr and Fe in the alloy from 59.75 per cent to 62.54 per cent and 28 per cent to 33 per cent for blend 1 to blend 3 respectively, but between blend 3 and 4, there is a slight increase due to smaller difference between Cr_2O_3, FeO and the Cr/Fe ratio in these blends. From blend 4 to blend 6, there is a significant increase in Cr and Fe in the alloy because of an increase in Cr/Fe. The amount of Si and C was almost constant due to a constant reductant mass used across all blends. The amount of Cr that reported to slag is shown in FIG 4 **5**.

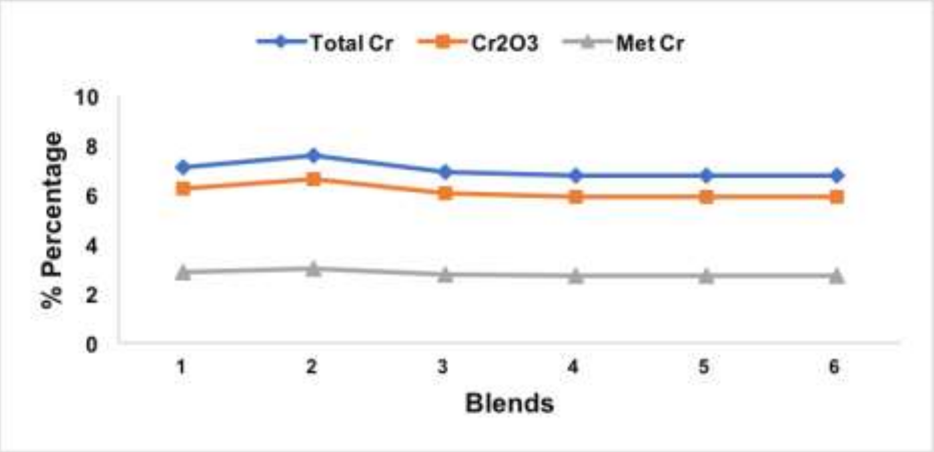

FIG 5 – Slag analysis

FIG 5 shows the slag analysis from the closed DC arc furnaces, indicating the total Cr, unreduced Cr_2O_3, and metallic Cr in the slag. The total Cr content in the slag is uniformly low at around 8 per cent across all the blends, with Cr_2O_3 content less than 7 per cent. This indicates the high reduction efficiency of the closed DC arc furnace. A minimum metallic Cr content remains in the slag at around 3 per cent across all blends, which is a positive indication of effective separation by the flux.

Efect of basicity on Separation efficiency

The MgO/SiO_2 and MgO/Al_2O_3 ratios of the blends are shown in FIG 6 and these ratios might affect slag fluidity.

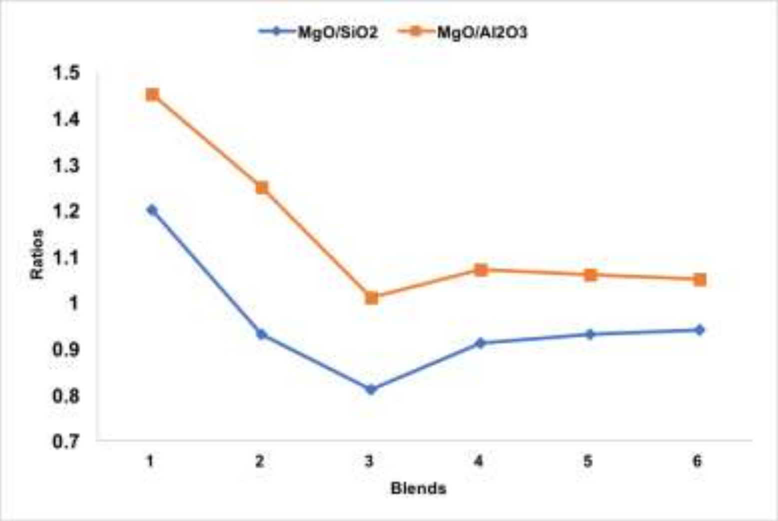

FIG 6 – Slag analysis on Separation efficiency

The MgO/SiO_2 ratio shows a decreasing trend from blend 1 to 3, suggesting a decrease in separation efficiency due to an increase in SiO_2 relative to MgO since there was a corresponding increase of the Cr reporting to slag in blend 2, as shown in FIG 5. This was also supported by the slow increase in tapping stage recovery between Blend 1 and 3 shown in FIG 6 followed by a rapid increase in recovery afterwards. This shift in ratio from blend 1 to 3 might have increased the melting point of slag which results in a specific energy consumpsion (SEC) increase and an increase in viscosity of the slag decreasing the separation efficiency between the slag and alloy. On the other hand, the MgO/Al_2O_3 ratio remains relatively constant, indicating stable separation efficiency and consistent proportions of these compounds across the samples. This implies that the difference in melting points during slag chemistry due to the difference in MgO, Al_2O_3 and SiO_2 affects the separation efficiency.

Specific Energy Required (SEC)

The different trends of SEC with and without pre-treatment techniques is shown in FIG 7. These were calculated as:

Equation 2 How to determine the specific energy required

$$\mathrm{SEC} = \frac{\text{Amount of Electrical energy used}}{\text{amount of alloy produced}} \qquad (2)$$

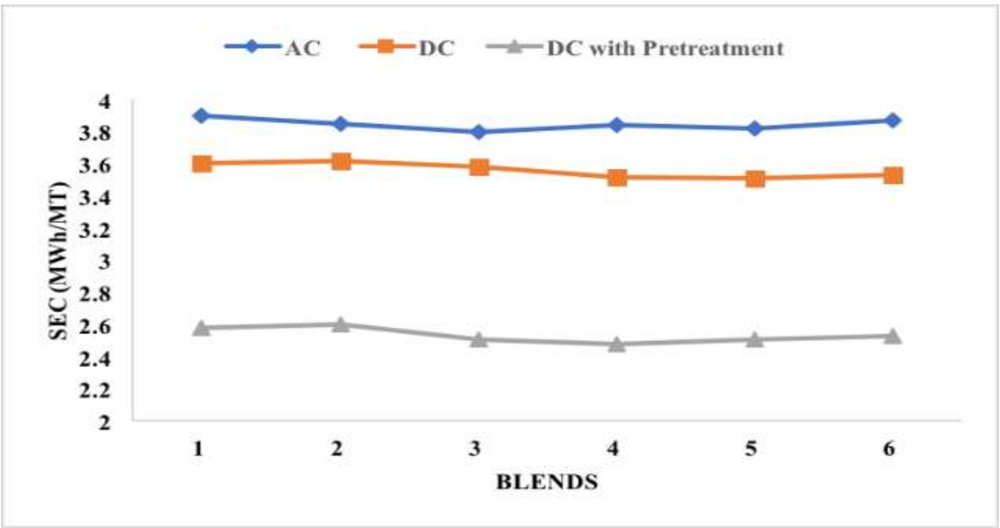

FIG 7 – SEC comparison of with and without pre-treatment methods

FIG 7 compares the SEC in MWh/MT of alloy between AC furnaces, DC furnaces, and DC with pre-treatment. AC furnaces consistently show low SEC, averaging 3.9 MWh/MT due to the inherent characteristics of AC furnaces. DC furnaces exhibit better SEC than AC furnaces due to higher operating temperatures, minimum heat losses and more efficient energy transfer. DC furnaces with pre-heating and pre-reduction have much better SEC, with an average SEC of 2.5 MWh/MT. This suggests that pre-heating and pre-reduction significantly improve the performance of DC arc furnaces, by reducing the energy required for smelting. However, the SEC remains constant throughout the samples, indicating that the efficiency of DC furnaces is not significantly affected by the grade of chromite fines. There was a general increase in SEC from blend 1 to 2 in the DC with a pre-heating curve that corresponded to an increase in the MgO/SiO_2 that resulted in a slight increase in Cr reporting to slag, shown in FIG 6. Blend 4 was then considered the optimal blend based on both recovery and SEC. This blend was used in the subsequent experiments.

SEC and tapping stage recovery

This smelting experiment was performed to check on the tapping stage recovery using blend 4 and pre-calculated SEC. FIG 8 shows the effects of SEC on the taping stage recovery.

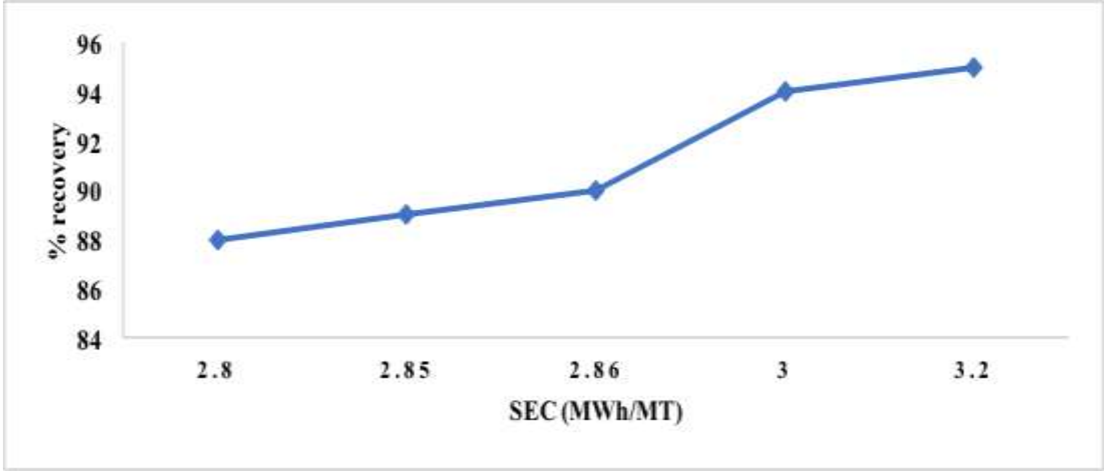

FIG 8 – The effects of Specific energy consumption on Recovery

There is a clear correlation between SEC and percentage recovery in the given data. As the SEC increases from 2.8 MWh/MT to 3.2 MWh/MT, there is a noticeable increase in the percentage recovery from 88 per cent to 95 per cent. The most significant increase in recovery occurs between the SEC levels of 2.86 MWh/MT and 3 MWh/MT, where the recovery surges from 88 per cent to 94 per cent. Although there is only a slight increase in recovery from 3 MWh/MT to 3.2 MWh/MT, it is still evident that better SEC leads to higher recovery rates. However, for optimum operation, the range of 3 MWh/MT to 3.2 MWh/MT appears to be more viable. Within this range, the cost of SEC remains relatively lower, while achieving a high recovery rate of 94 per cent.

Effect of arc length on TSR

FIG 9 shows the recovery of Cr in ferrochrome at different arc lengths during the tapping stage in smelting using a closed DC arc furnace.

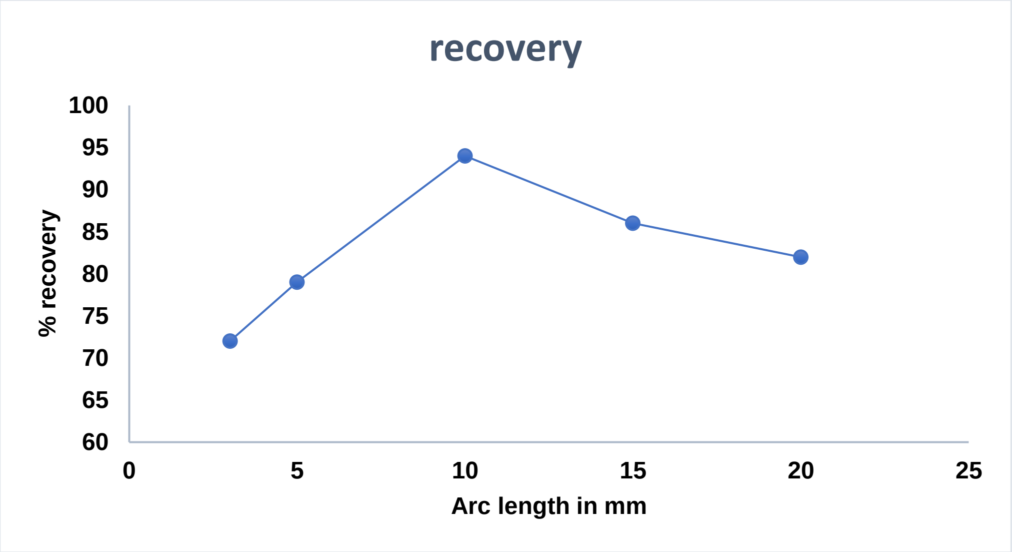

FIG 9– Effects of arc length on the tapping stage recovery.

The recovery increases with an increase in arc length up to 10 mm, after which it starts to decline as the arc length continues to increase. This suggests that 10 mm is an optimal arc length for maximum recovery of Cr in ferrochrome of 94 per cent, this could be due to the optimal balance between heat generation and heat loss at this arc length, which facilitates the reduction of Cr_2O_3 to Cr. A decline in recovery for arc length beyond 10 mm, is due to excessive heat loss at larger arc lengths, which reduces the efficiency of the smelting process.

Effect of pre-heating and pre-reduction

FIG 10 shows the % Cr and Fe reduced after pre-treatment of the sample at different temperatures using CO as a reducing agent in blend 4. During the pre-heating and pre-reduction process, the temperature of the chromite ore fines raised to approximately 450 °C, when pretreated with CO at a temperature of 700 °C which is the estimated temperature of the off-gas leaving the furnace. The difference in temperature between the off gas and the ore is due to the partial energy lost during the transfer of CO to ore and the energy used to raise the temperature of the ores. This therefore limits the overall reduction efficiency. At 400 °C, only 10 per cent of FeO was reduced of Fe and an insignificant amount of Cr_2O_3 was reduced.

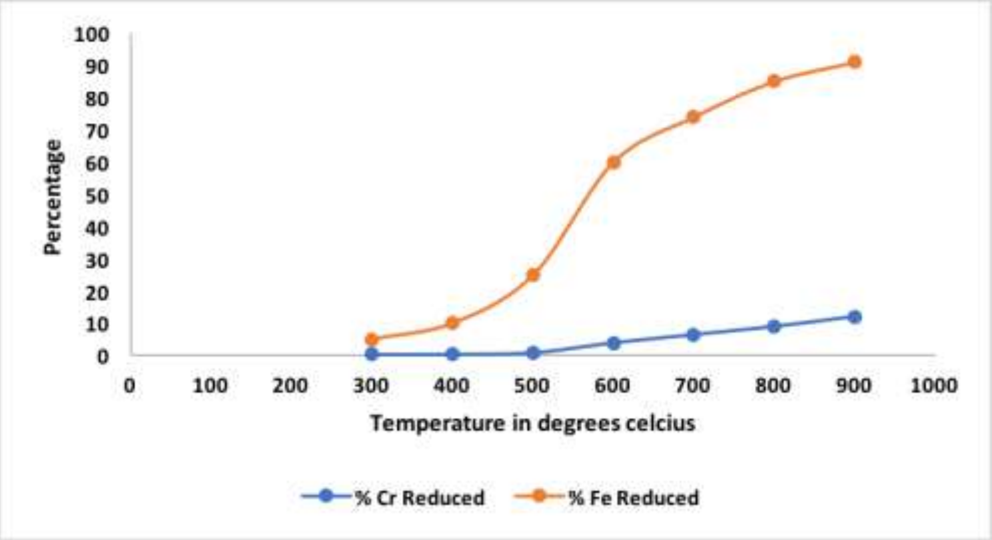

FIG 10 – Pre-reduction percentages of Cr and Fe using CO

There is minimal reduction in both Cr and Fe at low temperatures less than 300 °C as indicated in FIG 10. Above 400 °C, Fe shows a significant increase in metallic percentage of Cr and Fe as temperature increases, reaching close to 100 per cent reduction at 1000 °C. The reduction of FeO to Fe is highly temperature-dependent and becomes more efficient at higher temperatures. In contrast, Cr reduction is relatively steady and minimal across the entire temperature range. The reduction of Cr_2O_3 to Cr is less efficient than that of FeO to Fe under the same conditions as can easily be seen by the thermodynamic data for equations (3) and (5) (Lee, 1999).

$$\mathrm{FeO}(s) + \mathrm{CO}(s) = \mathrm{Fe}(s) + \mathrm{CO}_2(g) \quad (3)$$

$$\Delta G° = 11000 - 156.9T \text{ J/mol (T= 298- 973 K)} \quad (4)$$

$$\mathrm{Cr}_2\mathrm{O}_3(s) + 3\mathrm{CO}(s) = 2\mathrm{Cr}(s) + 3\mathrm{CO}_2(g) \quad (5)$$

$$\Delta G° = 279400 - 452.7T \text{ J/mol (T=298-973)} \quad (6)$$

Energy requirements for ferrochrome smelting

The probable reactions that may occur during the smelting of chromite fines with pre-heating and pre-reduction effects were predicted using basic thermodynamics, taking into account the chemical composition of the blends. It should be emphasized that the thermodynamic analysis was conducted to provide theoretical insight on the energy requirement despite the use of natural chromite ore with varying chemical interactions.

In order to determine the energy required in FIG 10 to elevate the temperature of each compound from the reference temperature to the pre-heating temperature using the off gas from the furnace, the specific heat capacities of each compound within the chromite ore were employed. Shomate constant data obtained from the thermodynamics database (Chase M., 1998) as depicted in Table 8, which shows the specific heat capacity (Cp) and total enthaplies (H-H298). The masses of the compounds were ascertained by computing the proportions of each compound based on the percentage composition of the 0.1 kg sample utilized.

Table 8 Specific heat capacity and total enthalpy from shamote data (Chase, 1998)

Compound	Temp (K)	Cp (J/mol.K)	$H-H_{298}$
Preheating			
FeO		56.11	-
Cr_2O_3		122.7	-
SiO_2	700	68.77	-
CaO		51.54	-
Al_2O_3		116.5	-
MgO		148.69	-
Pre-reduction			
FeO		56.11	21.42
CO		7.45	2.87
CO_2	700	49.57	17.75
Fe		34.48	12.08
Smelting Zone			
FeO		-	104.5
Cr_2O_3		-	216.9
SiO_2		-	111.5
Cr		-	61.06
Fe	2000	-	68.85
Si		-	46.28
CO		-	13.56
CaO		58.48	-
Al_2O_3		136.6	-
MgO		55.70	-

$$Q = mc_p\Delta T \qquad (7)$$

The Cp (J/mol.K) data from Table 8, was converted to Cp (Kwh/Kg.K). The specific energy in **Table 9** was calculated using equation 7 and calculated Cp (Kwh/Kg.K). The total energy consumption for pre-heating the chromite ore to 700 K was found to be **0.012386 kWh**. This value represents the cumulative energy needed to heat all the components in the ore to the desired pre-heating temperature.

TABLE 9 - Energy required for pre-heating to 700 K

Compounds	**Mass (Kg)**	***Cp*10^{-5}*** **(kWh/Kg.K)**	**Energy (kWh)**
FeO	0.011558	21.61	0.0010166
Cr_2O_3	0.033812	22.42	0.0030853
SiO_2	0.0115	31.83	0.0014898
MgO	0.01737	48.321	0.003416
Al_2O_3	0.01186	31.73	0.0015316
CaO	0.0139	32.64	0.001847
Total energy			**0.012386**

To determine the energy required for pre-reduction using the off gas from the furnace and the smelting process inside the furnace, the total enthalpies of each component within the chromite ore were employed. These enthalpies were obtained from Shomate constant data as presented in Table 8 (Chase M., 1998).

$$\text{Energy per mol} = \text{total enthalpies of the final products} - \text{total enthalpies of the reactants} \quad (8)$$

$$\text{Specific energy required(SER)} = \text{Energy per mol} \times \text{molecular mass of the species} \quad (9)$$

$$\text{Energy used} = \text{SER} \times \text{mass of the species} \quad (10)$$

Equation 8, 9 and 10 were used in the evaluation of the values obtained in **Table 10 and 11**. The energy/mol was calculated using enthalpies data from Table 8 and equation 8. Reaction for the pre-reduction stage shown in equation 11 and its associated energy consumption are shown in **Table 10.** The mass of Fe is based on the percentage of metallic iron produced in the pre-reduction graph. By combining the specific energy requirement and the mass of Fe reduced, an estimation of the energy consumption for Fe reduction during pre-reduction can be obtained. This energy consumption value is typically expressed in kWh and quantifies the amount of electrical energy needed to facilitate the reduction reaction.

$$FeO + CO \Rightarrow Fe + CO_2 \quad (11)$$

TABLE 10 - Energy used in the pre-reduction

Product species	**Mass of Product species (Kg)**	**Energy/mol (kWh/mol)**	**SER (kWh/kg)**	**Energy (kWh)**	
				Alloy	**Off-gas**
Fe	0.001631	0.001539	0.0275	0.000044825	
CO_2	0.00128	0.001539	0.03498		0.000044825

The energy/mol required for smelting are shown in Table 11, were calculated using enthalpies data from Table 8 and equation 8.

TABLE 11 - Reactions and Energyrequired in the smelting phase

Reduction reactions	**Product species**	**Product species Mass (kg)**	**Energy/mol (kWh/mol)**	**SER (kWh/kg)**	**Energy (KWh)**	
					Alloy	**Off-gas**
$Cr_2O_3 + 3C \Rightarrow$	Cr	0.01997	-0.0446	0.7964	0.0159	
$2Cr + 3CO$	CO	0.015	-0.0446	1.014		0.0159
$FeO + C \Rightarrow$	Fe	0.013885	-0.01598	0.2854	0.00396	
$Fe + CO$	CO	0.0069	-0.01598	0.5707		0.00396
$SiO_2 + 2C \Rightarrow$	Si	0.00114	-0.0303	1.0809	0.00123	
$Si + 2CO$	CO	0.00113	-0.0303	1.0802		0.00123
Total Energy					**0.02109**	

Table 12 - Amount of Energy used in raising temperature of materials which reported to slag

Compounds	**Mass (Kg)**	**$Cp*10^{-5}$ (kWh/Kg.K)**	**Energy (kWh)**
CaO	0.00089	40.61	0.0006059
Al_2O_3	0.00807	37.2	0.005295
MgO	0.0201	38.7	0.01317
SiO_2	0.054	36.11	0.03319
Total			**0.05226**

From Tables 11 and 12, the amount of energy required for ferrochrome production is **0.05226** kWh. However, after subtracting the pre-heating and pre-reduction energies, the overall energy required for smelting becomes 0.03983 KWh. Based on the nature of the experiments, which were performed on a small scale, considering this factor, the approximate amount of energy converted to smelting is 52% of the total energy supplied. Thus, the total energy required for smelting is:

$$\frac{0.03983}{0.52} = 0.07659\ kWh$$

The amount of ferrochrome alloy produced in the reaction amounted to 0.03051 kg. The SEC was therefore evaluated to be 2.51 MWh/MT using Equation (1).

Conclusions

The results shown in this study provide a baseline for the viability of the transition from the traditionally used AC furnace to the closed DC arc furnace with pre-treatment stages. This confirms that fine chromite ores can be processed using a closed DC arc furnace without going through agglomeration techniques.

The DC arc furnace proved to be efficient regardless of the quantity of fines in the blend but rather recovery was determined by the grade of the ore. Comparison between the DC and AC furnace TSR indicated high energy conversion efficiency of the closed DC arc furnace because the input energy is effectively utilized in the DC furnace, resulting in improved recovery rates during the tapping stage.

SEC was significantly lowered by 35.8 per cent from 3.9 MWh/MT to 2.5 MWh/MT when pre-treatment techniques were employed. Also, the implementation of coal as the reductant and use of low-grade fines chromite ore aid in lowering the overall cost of production by an average of 30 per cent. This significant difference is important because it shows the potential of the closed DC arc furnace to achieve the goal of lowering energy consumption in ferrochrome production.

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