DC arc smelting of silicon: Is it technically feasible?

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Abstract – The technical feasibility of silicon smelting in a DC arc furnace was investigated. A graphite-lined furnace was operated at power levels of 150 to 160 kW, and 40 to 60 V. The average feed rate during the silicon production period was about 30 kg/h. The recipe consisted of pre-mixed batches of 28% petroleum coke and 72% lumpy silica. The target furnace operating temperature was set at 1600°C and then raised to 1700°C. In total, about 1.6 tons of premixed feed materials was processed over 16 feeding–tapping operations. The feed was manually charged into the furnace in sub-batches of 10 to 40 kg at a time. The total batch mass was set at 90 kg during the initial part of the testwork, and increased to about 120 kg for the rest of the campaign.

In total, about 400 kg of metal was produced, analysing between 18 and 78% Si, with the major impurities being iron, calcium, and aluminium. Silicon extraction (recovery to the metal phase) averaged about 40%, and may have reached more than 45% during the last five batches. In comparison, the metal recovery may vary between 70 and 85% in commercial facilities. The results obtained suggest that DC arc smelting of silicon metal is technically feasible, although some improvements will need to be made. However, extensive optimization work is required before the economic viability can be properly assessed.

INTRODUCTION

Metallurgical-grade silicon (MG-Si) is typically produced by carbothermic reduction of silicon dioxide in a three-electrode AC furnace where pre-baked graphite electrodes are employed. Typically, MG-Si contains 98.5% Si or higher, depending on the raw materials used and the requirements of the end user[1].

The silicon production process requires silica as a raw material, as well as a source of carbon[2-4]. The common sources of silica are quartz and quartzite where the SiO_2 content is more than 95%. Charcoal, coke, low-ash coal, and wood chips are used as reducing agents. The raw materials are carefully selected in order to minimize SiO vapour losses[3,5] to the gas phase, and to reduce the purification requirements of the metal produced.

Calcium compounds (CaO, $CaCO_3$, CaF_2, *etc.*) can also be charged into the reaction zone of the furnace. Addition of calcium compounds increases the rate

of the Acheson reaction (SiO₂ + 3C → SiC +2CO), and may contribute to better utilization of raw materials and energy[6-8].

Silicon smelting in a DC arc furnace has been suggested by several authors[9-13]. The major objective of the reported work was to provide a means of improving silicon recovery by capturing the volatilised silicon monoxide in a coke bed. The bed can be a hollow graphite electrode[9], a dedicated compartment within the furnace where most of the reducing agent is introduced, or a separate vessel on top of the furnace[10]. Some patents tend to take advantage of certain features of the DC arc furnace, including feeding of finer raw materials[9]. The aim of such configurations is to direct the furnace off-gas through a carbon-rich zone so that the contained carbon can react with silicon monoxide to form SiC. The reported data suggest that these complicated arrangements had some success, with the silicon recovery ranging from 30 to 67% at 100–400 kW scale of operation.

Proper control of the AC furnace is a challenge that faces the operators of a silicon furnace, where electrical imbalances can lead to significant metal losses, as SiO(g) and SiC accumulate inside the furnace and create additional operational and production issues. Given the success that Mintek achieved in ferromanganese smelting in a DC arc furnace[14] (where manganese losses to the off-gas were acceptable), and the ease of controlling the electrical parameters, it was decided to carry out an exploratory test to investigate the technical feasibility of silicon smelting in a 200 kW DC arc facility. The major results of the work are summarised in this paper.
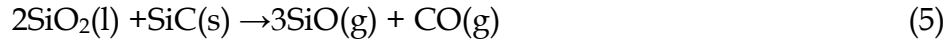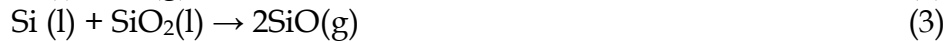
CHEMISTRY OF SILICON PRODUCTION

The production of metallurgical-grade silicon is a complicated process[8,15-16]. The complications are due to high temperatures and the aggressive environment inside the furnace where direct measurements are almost impossible. Therefore, reliance is made on indirect measurements and observations (of the off-gas, tapped metal, movement of electrodes, *etc.*), in order to control the furnace. Other important data that are used to properly operate and control the furnace are: gas distribution at the top of the raw materials, temperatures, the amount of silica in the off-gas, and dynamic changes in the electrical parameters (current, voltage, impedance, load, resistance, harmonics in the current *etc.*).

Generally speaking, the basic idea of the process is to use energy and carbon to remove oxygen from silica and thus to produce silicon according to the overall reduction reaction:

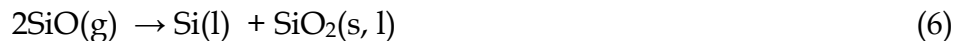$$SiO_2 \text{ (s)} + 2C(s) \rightarrow Si(s, l) + 2CO(g) \qquad (1)$$
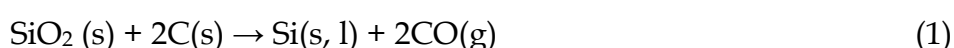
In reality, a number of reactions take place in different regions within the furnace[8] for silicon to be formed. In the lower zone, where the temperature can be 1600–2000°C, the following reactions take place:

$$Si\ (l) \rightarrow Si(g) \tag{2}$$
$$Si\ (l) + SiO_2(l) \rightarrow 2SiO(g) \tag{3}$$
$$SiO(g) + SiC(s) \rightarrow 2Si + CO(g) \tag{4}$$
$$2SiO_2(l) + SiC(s) \rightarrow 3SiO(g) + CO(g) \tag{5}$$

High temperatures tend to enhance the vaporization of liquid silicon and its reaction with SiO_2 (l), and thus lower the Si content in the metal. This is particularly significant in the arc zone where temperatures in excess of 2 000°C can be experienced. On the other hand, high temperatures ensure rapid melting of SiO_2, keeping its activity high enough such that reaction 5 proceeds at an adequate speed. The produced SiO(g) reacts with silicon carbide present in the lower zone (shown in reaction 4) producing silicon, and therefore contributing to high Si-content in the metal.

The middle region of the furnace is filled with raw and partially reacted feed materials, with temperatures between 800-1600°C. In this zone, the important reactions are (in addition to the reverse of reaction 5):

$$2SiO(g) \rightarrow Si(l) + SiO_2(s, l) \tag{6}$$
$$SiO_2\ (s) \rightarrow SiO_2\ (l) \tag{7}$$
$$SiO(g) + 2C \rightarrow SiC(s) + CO(g) \tag{8}$$
$$Si\ (g) \rightarrow Si(l) \tag{9}$$

Carbon monoxide and silicon monoxide are combusted with air in the upper region (freeboard of the furnace):

$$2CO(g) + O_2(g) \rightarrow 2CO_2\ (g) \tag{10}$$
$$2SiO(g) + O_2(g) \rightarrow 2SiO_2(s) \tag{11}$$
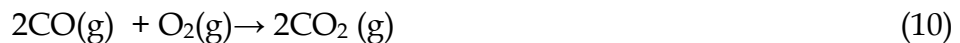
In addition to reactions 2–5, silicon carbide can form in the hot zone of the furnace:

$$Si(g,l) + C \rightarrow SiC(s) \tag{13}$$

This is an undesirable reaction, as it leads to silicon metal losses and build-up of SiC(s). Therefore, un-reacted carbon should be prevented from entering the hot zone, by choosing carbon materials with high SiO reactivity (charcoal and wood chips), so that carbon will be consumed by reaction 8 in the middle zone. Electrodes are a source of carbon, too, therefore, their reactivity towards SiO(g) should be as low as possible.

EXPERIMENTAL EQUIPMENT AND PROCEDURES

The pilot plant, as employed in this investigation, consisted of a feed system, DC power supply, refractory-lined furnace, and an off-gas handling system (Figure 1). The facility was designed for the smelting of various ferro-alloys, and no modifications were undertaken in preparation for the current testwork as it was exploratory in nature.

The furnace consisted of a refractory-lined cylindrical shell, a domed base, and a conical roof. The furnace shell had an unlined internal diameter of 980 mm. The facility was designed for the smelting of various ferro-alloys, and no modifications were undertaken in preparation for the current testwork as it was exploratory in nature.
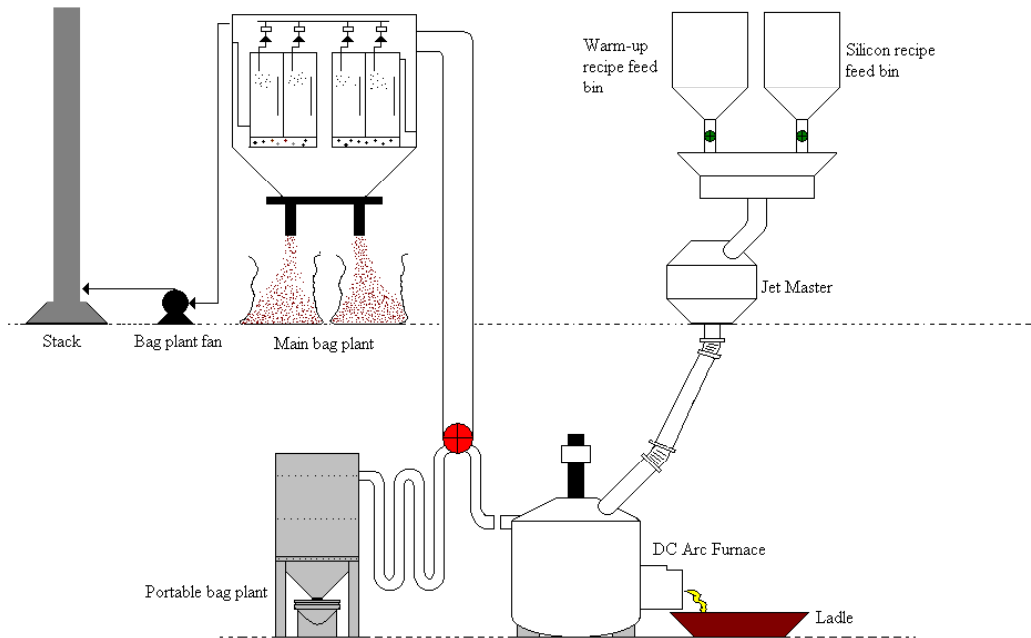

Figure 1: Layout of the 200 kW DC arc furnace facility

The furnace roof contained three ports for feeding, bath inspection, and a central port for a 100 mm graphite electrode (cathode). The furnace shell was lined with Verokast-1800 (Vereeniging Refractories, Table I), and had a graphite crucible with an internal diameter of about 400 mm and a height of 800 mm (see Figure 2). The base and the furnace roof were lined with alumina castable (Verokast-1800, Vereeniging Refractories).

Table I: Chemical analysis of the Verokast 1800, mass %

Component	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	TiO_2	$Na_2O + K_2O$
Verokast-1800	5.3	94.2	0.1	0.1	0.1	0.1	0.3

The return electrode, or anode, consisted of a 100 mm graphite rod bolted to a steel plate that was attached to the hearth dome and was connected to the anode busbars. The electrode and feed pipe were electrically insulated from the feed system. No water-cooling was provided to the shell (natural air-cooling). The conical roof was cooled by means of pressurised water-cooled panels.
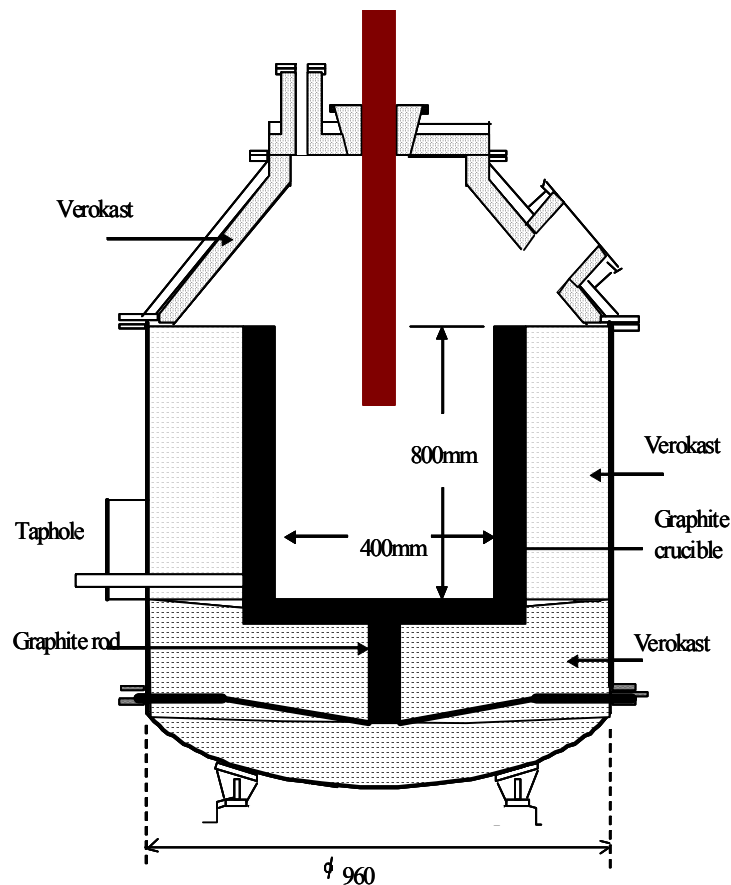
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Figure 2: Furnace refractory layout

The feed system consisted of one hopper to deliver premixed batches of silica and petroleum coke. The hopper was equipped with a variable-speed vibratory feeder and load cells, in order to enable a controlled feed rate to be delivered to the furnace. The feed rate control was linked to a Delta V control system.

The gas generated in the furnace was expelled through an off-gas port located at the furnace shell – roof flange into the gas extraction system. The gas extraction system comprised trombone coolers, a reverse-pulse bag-filter, a fan, and a stack.

The power supply consisted of an 11 kV vacuum breaker, two isolators, two contactors, two transformers, and two 5 kA DC thyristor rectifiers connected in parallel.

Feeding of pre-mixed batches of silica and coke through the feed system proved to be difficult, as a result of very frequent blockages in the off-gas port and the feed pipe. As a result, the operation could not be maintained for more than 30 minutes without having to switch off the power in order to clean these ports. Therefore, the feed mixture was introduced manually into the furnace throughout most of the campaign.

During the smelting period, the furnace was operated at power levels of 150–160 kW. This power range was based on a specific energy requirement (SER) of

2.96 kWh/kg feed, a total feed rate of 30 kg/h, and an energy losses set-point of 60–70 kW. The voltage set-point was chosen such that the 'arc length' was between 40 and 60 mm. (The arc length refers to the hoist position upon striking, compared to its position after raising the electrode to obtain the target operating voltage). The aim here was to minimize SiO(g) losses to the fume by ensuring that the electrode tip was below the level of the un-reacted feed. Generally, the voltage set-point was between 40 and 60 V.

The raw materials were pre-mixed in a Jones mixer and then charged into the furnace. This was done by feeding a known amount of the Si recipe (10–40 kg) at once, followed by stewing until the target SER was reached. This process was repeated until the batch mass was totally fed. Initially, the batch mass was set at 90 kg (Taps 3–7), and then increased to 120 kg for the rest of the campaign.

Tapping was carried out by drilling through the tap-hole, followed by oxygen lancing. Upon tapping, temperature measurements were taken using an infrared optical pyrometer. Metal and slag samples were taken from the flowing stream, crushed, pulverized, and them chemically analysed. In addition, fume samples were taken from the bag-house and the clean-out of the off-gas port, and analysed.

The Si recipe consisted of lumpy silica (72% by mass) and petroleum coke (Sascarb, 28% by mass). The feed ratio was based on the following reaction:

$$SiO_2 + 2C = SiC + 2CO \qquad (14)$$

and assuming that 15% of the silicon in the feed was lost to the off-gas as SiO(g):

$$SiO_2 + C = SiO(g) + CO \qquad (15)$$

SUMMARY OF TEST CONDITIONS

In addition to the warm-up period, three different conditions were experimented with, as shown in Table II. The first condition (Taps 3–7) involved charging 10 kg of the Si recipe, at once, every 20 minutes. The batch mass during this condition was targeted at 90 kg. In addition, and due to certain difficulties encountered during this period, 140 kg of flux mixture (80% silica, 10% lime, and 10% Hicast) was also charged into the furnace.

In the second condition, (Taps 8–13), the total batch mass was increased to 120 kg, and was fed in sub-batches of 20 kg every 40 minutes. Lime (2.5% of the Si recipe, by mass) was introduced into the furnace during the last tapping period of this condition. In addition, 40 kg of flux was also processed.

The last condition covered Taps 14 to 18. The sub-batch mass was increased to 40 kg every 80 minutes. Lime addition was kept at 2.5% of the Si recipe, bringing the total batch mass to 123 kg.

Table II: Summary of the experimental conditions

Tap range	Condition	Parameter	Batch, kg *	Mass fed, kg**	Flux, kg
1–2	0	Warm-up	60	123	0
3–7	1	Sub-batch = 10 kg	90	505	140
8–13	2	Sub-batch = 20 kg	120	683	40
14–18	3	Sub-batch = 40 kg Lime = 1 kg	123	615	0

*: Target mass of Si recipe to be fed in a given tap
**: Actual total mass of the Si recipe fed during a given condition

RAW MATERIALS

The raw materials used in the testwork consisted of silica sand and petroleum coke (Sascarb). The particle size range of the as-received silica was 10–40 mm. Petroleum coke with a particle size of less than 6 mm was acquired from Sasol. In addition, lime and Hicast (from Vereeniging Refractories) were charged into the furnace on a few occasions, in order to reduce and control the build-up inside the furnace. Chemical analyses of the various raw materials appear in Table III.

Table III: Chemical analyses of raw materials, mass %

Component	Silica	Sascarb	Lime	Hicast
MgO	<0.08	<0.08	1.01	0.32
Al_2O_3	0.89	0.31	0.38	94.50
SiO_2	98.44	0.46	0.75	0.39
CaO	<0.07	<0.07	92.76	4.48
MnO	<0.06	<0.06	0.86	0.11
Fe_2O_3	0.09	<0.07	0.24	<0.05
S, ppm	40	210	200	100
P, ppm	400	200	100	100
B, ppm	23	25	10	300

RESULTS AND DISCUSSION

Slag

The average slag analyses for the three identified conditions appear in Table IV. Clearly, there is quite a variation in the SiO_2 content from one condition to the other, and this reached about 95% in the last condition. In addition, there seems to be a relationship between the SiO_2 content and the carbon content. This is believed to be related to the presence of silicon carbide (SiC) in the slag. In fact, when the silicon content is expressed as SiC and SiO_2, the total analysis for the last condition comes to about 101% (as compared to a total of 143% if it was to

be expressed as SiO_2 only). These estimates are based on the assumption that all the carbon in the slag is present as SiC (no free carbon, or carbides of any other elements). The suggestion that silicon carbide is present in certain slag samples is supported by XRD analysis of a few slag samples taken from the middle layer of the furnace dig-out. The results suggest that the two major crystalline phases present are SiC and metallic silicon. Semi-quantitative analysis of these samples indicates a SiC content of 62% and a silicon content of 32%. A third and minor phase with a chemical formula of $Ca_2Al(AlSi)O_7$ was also identified.

In spite of the addition of lime during the last condition, the CaO content in the slag was the lowest, compared to the initial two conditions. This is believed to be due to the charging of a certain amount of slag fluxing mixture into the furnace during Conditions 1 and 2.

Table IV: Chemical analyses of the slag (mass %), and tapped masses

Tap	Mass, kg	MgO	Al_2O_3	SiO_2	CaO	B	MnO	FeO	C	P	Total
3(1)	0.0	0.48	11.01	29.79	1.15	0.001	0.06	16.60	0.03	0.05	59.17
4(1)	1.7	0.30	29.47	42.75	33.81	0.001	0.06	2.19	1.17	0.03	109.78
5(1)	1.8										
6(1)	0.0										
7(1)	0.0										
Weighted Average-1		**0.39**	**20.24**	**36.27**	**17.48**	**0.001**	**0.06**	**9.39**	**0.60**	**0.04**	**84.47**
8(2)	8.6										
9(2)	0.1										
10(2)	0.5										
11(2)	0.1	0.43	28.52	31.29	21.00	0.001	0.27	15.18	1.33	0.05	98.07
12(2)	5.0	0.28	22.48	37.50	13.06	0.002	0.42	23.93	1.10	0.05	98.83
13(2)	0.3										
Weighted Average-2		**0.36**	**25.50**	**34.39**	**17.30**	**0.002**	**0.35**	**19.55**	**1.22**	**0.05**	**98.72**
14(3)	0.0										
15(3)	0.2										
16(3)	5.2										
17(3)	0.0										
18(3)											
Dig-out top	21.0	0.08	29.94	62.89	3.49	0.004	0.32	0.75	28.53	0.07	126.08
Dig-out middle	43.0	0.08	8.20	154.61	4.40	0.005	0.54	1.34	13.01	0.05	182.22
Dig-out bottom	14.0	0.08	14.30	118.93	4.68	0.005	0.17	1.94	20.20	0.05	160.35
Dig-out loose	4.0	1.33	2.72	43.29	1.19	0.004	0.15	51.97	1.53	0.09	102.27
Weighted Average-3		**0.39**	**13.79**	**94.93**	**3.44**	**0.005**	**0.30**	**14.00**	**15.82**	**0.06**	**142.74**
Overall		*0.38*	*18.33*	*65.13*	*10.35*	*0.003*	*0.25*	*14.24*	*8.36*	*0.05*	*117.09*

The average FeO content in the slag was always 10% and higher in all three conditions, largely due to oxygen lancing (using steel lance rods) that introduced a significant amount of iron into the furnace products, as well as to possible metallic inclusions in the analysed samples.

Condition 3 can be divided into two periods. In Taps 14 to 16, the furnace was tapped at the end of each batch, where mostly metal was recovered. The last two batches (Taps 17 and 18) were intentionally left inside the furnace and the products were collected during the dig-out. The slag analysis, therefore, represents that of the 'slag' recovered from the dig-out. This slag included a small amount of loose material that was present at the top of the furnace.

It should be noted that silicon production is theoretically a slag-less process. In this testwork, however, about 100 kg of what is referred to here as slag was produced due to various factors including: 1) Addition of a slag flushing recipe as a means of limiting the build-up inside the furnace. The recipe consisted of lime, alumina, and silica. 2) Oxygen lancing that caused the oxidation of some of the metal, in addition to the formation of iron oxide(s). Both silicon and iron oxides reported to this slag phase. 3) The dig-out slag is not entirely an oxide phase, as revealed by XRD. In fact, it contained a significant proportion of silicon carbide and metallic silicon. 4) Refractory erosion, sample contamination, and others.

The hard phase of the dig-out slag contained between 0.5 and 1.5% Fe. This is significant, as no oxygen lancing was performed at the end of the campaign.

Fume
The silica content of the fume collected from the furnace off-gas port was relatively high, at 95.4% SiO_2, with the other major impurities being Al_2O_3, MnO, carbon, and sulphur (see Table V). Notice that a wooden stick was used to remove the dust build-up in the off-gas port, which might have resulted in the relatively high carbon content. In the meantime, the bag-house fume contained high levels of FeO, TiO_2, CaO, and Al_2O_3, possibly due to contamination from fume produced in earlier campaigns carried out in the same facility.

Table V: Chemical analyses of the fume, mass %

	MgO	Al_2O_3	SiO_2	CaO	MnO	FeO	TiO_2	C	S
Bag-house	1.86	3.50	55.33	4.19	0.33	25.61	5.44	2.23	0.27
Off-gas	0.08	0.59	94.50	0.07	0.27	0.27	0.25	0.56	0.38

Metal
The average chemical analysis of the metal is presented in Table VI. In general, iron levels were very high, as a result of several factors, including oxygen lancing, the use of steel rods in order to distribute the fresh feed around the electrode, and the use of mild steel spoons to take metal samples. These factors are highlighted in Table VII where the overall iron mass balance clearly indicates the significant impact of lancing on the metal quality. In addition, chemical analysis of certain samples proved to be difficult, as precipitation of

the dissolved sample occurred before the concentration of various elements analysis could be measured (using Atomic Absorption technique, AA). Repeat sample preparation and analysis did not improve the outcome, particularly when the metal content in the sample was high. Similar difficulties were experienced with other samples containing high levels of silicon.

It is believed that oxygen lancing added between 140–150 kg of iron into the metal phase. However, it is difficult to assess iron contamination that was caused by the sampling spoons. Obviously, pick-up of iron by such samples would have the effect of decreasing the silicon content. This appears to have been the case where samples taken with alumina or graphite cups tended to show a lower iron content in general (Table VIII). Although the data is not very conclusive, it provides an indication that sampling was a problem when steel spoons were used. As a result, silicon content appeared to be lower, as compared to the samples taken using alumina or graphite spoons. Notice that the alumina crucible gave somewhat higher aluminium contents in certain samples.

Table VI: Chemical analyses of the metal (mass %), and tapped masses per tap (condition)

Tap	Mass,kg	Al	Si	Ca	B	Mn	Fe	C	P	Total	Si *
1 (1)											
2 (1)											
3(1)											
4(1)	14.0	2.30	47.50	2.14	0.003	0.20	41.80	0.14	0.06	95.67	53.11
5(1)	65.5	1.58	36.37	1.22	0.001	0.22	60.33	0.09	0.04	100.34	36.44
6(1)											
7(1)	26.1	2.03	46.60	0.57	0.001	0.20	54.00	0.05	0.02	103.92	43.08
Average-1		**1.97**	**43.49**	**1.31**	**0.001**	**0.21**	**52.04**	**0.09**	**0.04**	**99.29**	**44.21**
8(2)	28.2	2.36	33.80	1.20	0.002	0.15	54.40	0.10	0.01	92.40	41.73
9(2)	11.7	1.06	43.35	0.24	0.001	0.17	54.48	0.05	0.01	99.69	43.94
10(2)	35.4	3.98	18.50	1.43	0.001	0.05	20.00	0.19	0.04	44.53	74.28
11(2)	20.6	2.70	25.97	2.01	0.001	0.05	20.00	0.09	0.03	51.18	75.12
12(2)	11.3	2.62	50.13	1.86	0.002	0.08	20.47		0.04	75.66	74.88
13(2)	32.1	0.71	48.87	0.33	0.001	0.09	20.50	0.06	0.05	70.98	78.2
Average-2		**2.24**	**36.77**	**1.18**	**0.001**	**0.10**	**31.64**	**0.10**	**0.03**	**72.10**	**64.67**
14(3)	9.9	0.95	48.90	1.16	0.001	0.10	20.50	0.09	0.04	72.07	77.11
15(3)	18.3	0.87	53.90	0.67	0.002	0.01	36.28	0.06	0.04	92.16	62.02
16(3)	33.9	0.83	40.23	0.20	0.001	0.18	60.40	0.12	0.02	102.34	38.19
17(3)											
18(3)											
Dig-out Top	7.3	1.53	66.45	1.27	0.003	0.07	0.85	24.97	0.05	95.51	71.21
Dig-out middle	5.1	2.18	65.45	2.31	0.007	0.08	2.18	17.94	0.03	90.49	75.22
Dig-out bottom	69.7	3.30	53.30	2.74	0.005	0.10	28.50	9.06	0.04	97.36	56.2
Dig-out loose	8.0	0.81	37.10	0.24	0.007	0.14	61.20	0.14	0.06	100.07	37.29
Average-3		**1.51**	**52.27**	**1.16**	**0.001**	**0.10**	**32.18**	**6.58**	**0.06**	**93.92**	**58.36**
Overall		*1.85*	*45.25*	*1.19*	*0.001*	*0.12*	*35.49*	*3.34*	*0.04*	*87.34*	*57.91*

* by difference: %Si = 100-∑Analysis of other elements.   Average = weighted average

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As Table VI suggests, the total metal analysis was significantly less than 100% in several taps. In fact, the total was only about 45% in Tap 10. Assuming that the silicon analytical results are not accurate, and that the actual silicon content can be calculated by difference, then the silicon content may have been as high as 78% (Tap 13). This is significant, as it indicates that a much better silicon metal quality might have been produced than what the analytical results appear to suggest. In addition, the calculated silicon content has a major impact on both its extraction and accountability, as will be seen later.

Table VII: Iron mass balance

In, kg		Out, kg	
Feed	2.4	Slag	3.3
Lance rods*	147.0	Metal	155.4
Iron bars*	1.2	Fume	13.1
Total	150.6	Total	171.8
Accountability = 114.1%	Surplus = 21.2 kg		

* based on initial and final masses

Table VIII: Effect of sampling spoons on the chemical analyses of Si, Fe, and Al, mass %

Tap No.	Sample cup type	Si	Fe	Al
10	Mild steel	17.3	39.3	3.2
	Alumina	20.5	23.9	4.0
14	Graphite	49.0	<0.05	1.0
	Alumina	23.3	<0.05	1.9

As stated before, oxygen lancing was largely responsible for the relatively low silicon analysis in the tapped metal. As shown in Table IX, a sample of metal was recovered from the inside of the furnace towards the end of Tap 3 and analysed by XRF. The results suggest that the total impurities in this sample are about 5%, with the balance being silicon. This is an indication that relatively good-quality silicon metal was being made in the furnace.

Table IX: Chemical analysis of a silicon metal sample recovered from inside the furnace at the end of Tap 3, mass %

Elements	Si	Fe	Al	Ca	Cr	Mn	S
Mass%	94.71	2.35	1.56	0.20	0.04	0.04	0.84

The other major impurities present in the metal are aluminium and calcium. Boron levels are consistently low, at about 10 ppm, while those of phosphorus range from 300 to 400 ppm on average. Most of the tapped metal contained about 0.1% C. On the other hand, the dig-out metal was high in carbon content, which reached a high value of about 25% in the top and hard section of this product. The presence of silicon carbide in the dig-out material was discussed previously. Nevertheless, the results clearly indicate that silicon production in

a DC arc furnace is technically feasible. However, further and extensive testwork is required in order to optimize the feed recipe in terms of the nature of the reducing agents, and their chemical and physical characteristics, with a view to achieving acceptable silicon recovery. Doing so will allow a proper assessment to be made of the economic viability of the process.

Based on the silicon analysis of the produced metal, silicon accountability during the smelting period is relatively low at 51.4% (see Table X). It improves to about 60% when the silicon content of the metal is determined by difference. Nevertheless, the calculated silicon accountability is relatively low. It is believed that a large proportion of the deficit is present in un-collected bag-house fume.

Table X: Silicon mass balance

In, kg		Out, kg	(1)	(2)
Sascarb	0.42	Slag	47.16	47.16
Silica	591.16	Metal	177.87	226.60
Lime	0.19	Fume	79.05	79.05
Alumina	0.06			
Total	591.84	Total	304.08	352.81
			Accountability. = 51.4% Deficit = 287.76 kg	Accountability = 59.6% Deficit = 239.03 kg

(1) Based on Si analytical results
(2) Based on Si content as calculated by difference

Silicon Extraction

Table XI shows the silicon extraction, or recovery, defined as (Si in metal) / (Si in feed) to the metal phase for the three conditions. Based on the analytical results, the extraction averaged about 34% for the entire campaign, and reached more than 40% in the last condition (Taps 14-18). However, when the silicon content in the metal is calculated by difference, the overall silicon extraction increases to 44.2% on average.

The calculated silicon extraction tends to suggest that with larger feed sub-batches, silicon recovery improved moderately from 38.1% to 45.3%. It should be noted that other factors might have played a role, including a smoother operation as the testwork progressed, particularly during the last condition, lime addition, and possibly metal carry-over from the previous condition(s).

Table XI: Silicon extraction based on Si-producing recipe, %

Tap range	Extraction	
	Analytical results	Si-by difference
3-7	38.1	38.1
8-13	24.3	42.5
14-18	40.2	45.3
Overall	34.0	44.2

The calculated extraction figures appear to be low. However, considering that silicon recovery in a commercial facility can be as low as 70%, the achieved values are reasonable given that this was essentially a first-time experiment in silicon production using a simple DC arc furnace.

Electrode consumption

The furnace was operated at power levels of 150–160 kW throughout the smelting period, resulting in tapping temperatures of between 1630 and 1750°C in most taps. The calculated power flux ranged from 1.19 to 1.27 MW/m² of the original hearth area. In addition, the graphite electrode current density ranged from about 31 to 50 A/cm². In spite of the high power flux and, in particular, the current density, the electrode consumption was just over 5 kg/MWh. In comparison, the electrode consumption in industrial furnaces is reported to be about 9–13 kg/MWh [4].

CONCLUSIONS

The production of silicon metal in a DC arc furnace was demonstrated experimentally. The testwork involved smelting of a silicon-producing recipe at 30 kg/h and power levels of 150–160 kW. The target operating temperature was 1700°C.

About 16 batches (90–120 kg per batch) were processed during the testwork, consisting of about 1.15 tons of lumpy silica and about 450 kg of Sascarb, producing about 400 kg of metal and 100 kg of slag, in addition to about 340 kg of bag-house fume.

The analytical results indicated that the silicon content in the metal varied between 18.5 and 66.5%, with the major impurities being iron, aluminium, and calcium. The silicon content increased to almost 78% when it was calculated based on difference. The presence of iron as the major impurity was largely due to oxygen lancing using steel lance rods. Sample contamination by mild-steel spoons, and the use of steel rods to inspect the furnace contributed to the high iron levels in the metal.

Silicon recovery to the metal phase ranged from 24 to 45%, depending on the experimental condition and on the assumed silicon analysis. The balance appeared to have reported to the fume phase, to a large extent. These findings are based on the calculated silicon accountability of 51 to 60%.

In comparison, the recovery of silicon in industrial furnaces, operating at 10–40 MW, averages about 80%. It can be as high as 85% during good operating periods in modern and well-designed facilities, provided that the raw materials are selected properly and that the operation is well controlled. Otherwise, the metal recovery can drop to 70-75% levels, with the added undesirable consequence of lower-purity metal being produced. In comparison, the current testwork indicated a silicon recovery of 45% in the last five batches of the campaign. This is believed to be a relatively good achievement, given the scale

of operation and the fact that this was the first attempt at silicon production in a simple DC arc furnace.

Future testwork on silicon production in a DC arc furnace can focus on certain factors as a means of improving silicon recovery and its quality. The factors include the target operating temperature, mode of feeding (sequential, pre-mixed, and sub-batch mass), nature of the reductant, and particle size distribution of the raw materials.

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