Production Test of High-carbon Ferromanganese Using a Shaft Furnace with Coke Packed Bed Injected with Highly Oxygen Enriched Air and a Large Quantity of Pulverized Coal

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High-carbon ferromanganese has been produced by electric arc furnace in Japan. But its production cost has increased because of a speedy rising of electricity cost and Japanese ferromanganese has lost competitiveness in the international market. So, for the purpose of decreasing its production cost, high-carbon ferromanganese production tests (production rate=1.9–4.3 t/d) were performed using a melting test furnace with coke packed bed injected with highly oxygen enriched air and a large quantity of pulverized coal. Moreover the operation indices of a commercial plant of 170 t/d in production capacity were estimated by heat and mass balance model, and following results were obtained.

(1) In this test, high-carbon ferromanganese of [Mn]=75% was produced stably using the coke blend with 56% non-coking coal, with coal rate of 1 502 kg/t, coke rate of 1 087 kg/t and productivity of 3.11 t/(d·m³).

(2) The total coal consumption of the commercial plant of this method is estimated to be less than that of the blast furnace in spite of higher fuel rate because a large quantity of pulverized coal is used in this process. Through these investigations, this process seems to have a prospect of being available as a ferromanganese production process instead of electric furnace method.

KEY WORDS: ferromanganese; ferroalloy; smelting reduction process; pulverized coal injection; oxygen enrichment; blast furnace; ironmaking; coke packed bed.

1. Introduction

High-carbon ferromanganese has been produced by electric arc furnace (EAF) in Japan. But it needs electric consumption of about 7 300 MJ/t¹⁾ and its production cost has increased because of a high speedy rising of electricity cost in Japan. So Japanese ferromanganese has lost competitiveness in the international market. Therefore high-carbon ferromanganese production method using no EAF has been investigated.^{2,3)}

A shaft type furnace with coke packed bed injected with highly oxygen enriched air and a large quantity of pulverized coal (PC) seems to be suitable for producing ferromanganese which is finally reduced by smelting reduction reaction. As shown in Fig. 1, the melting furnace of this process is a shaft type furnace with coke packed bed and oxygen and a large quantity of pulverized coal are injected into the furnace. The theoretical flame temperature (T_f) can be easily increased because of the highly oxygen enriched air injection instead of the hot blast and the hot zone effective for reduction and melting in the furnace can be also easily expanded because of the pulverized coal injection in large quantities and low

coke consumption.⁴⁾ Moreover, the height of furnace could be decreased because the role of indirect reduction in shaft is less important in ferromanganese production than in pig iron production as ferromanganese is finally reduced by direct reduction reaction in rather large amount. Therefore it may be possible to use low quality of coke because the coke degradation during the descent in the shaft and the load of burden to coke in the furnace

Fig. 1. Concept of this process to produce high-carbon ferromanganese.

would decrease.

From such a background, the production tests of high-carbon ferromanganese of $[\text{Mn}] = 75\%$ were performed and its smelting characteristics were clarified. And based on these results, the indices of the commercial plant of this process were studied.

2. Outline of the Test Plant

2.1. Equipment

The experimental furnace used here is shown in Fig. 2. This plant was used for the verification of the new ironmaking process which consisted of the shaft type reduction furnace and the cupola type melting furnace.⁵⁾ But Fig. 2 shows only the melting furnace because reduction furnace was not used for this test. The maximum top gas pressure is $2.96 \times 10^5 \text{ Pa}$ and the charging systems are installed at the top of the furnace, three tuyeres⁶⁾ at the wall and a tap hole at the bottom of wall.

Fig. 2. Schematic diagram of the test plant.

Fig. 3. Profile of the melting furnace.

The profile of the melting furnace is shown in Fig. 3. Hearth diameter is 0.9 m, effective height (from stock line level to tuyere level) is 2.1 m and effective inner volume is about 1.4 m^3 . The values with the unit of “°” in this figure mean the circumferential direction.

2.2. Test Condition

Main operational indices are summarized in Table 1. Ore was used after riddled into 5–20 mm. Coke for ferromanganese smelting electric furnace blended with 56% of non-coking coal was used after riddled into 20–50 mm. Non-coking coal with about 34% of volatile matter crashed under 200 mesh was injected into the furnace from the tuyeres.

The ratio of pulverized coal to oxygen, the bosh gas volume (V_B) and the top gas pressure were set about 1.0 kg/Nm^3 , $900 \text{ Nm}^3/\text{h}$ and $2.48 \times 10^5 \text{ Pa}$ respectively. Here V_B is the gas volume generated by the combustion in front of the tuyere. T_f was controlled by nitrogen, and T_f shown in this table is adiabatic flame temperature calculated under the assumption that the combustion products are CO , H_2 and N_2 and the temperature of the coke entering the combustion zone is $0.75 \cdot T_f$ according to Ramm's method.⁷⁾ Slag basicity was controlled by limestone which was mixed with ore and coke, and CaO/SiO_2 of Mn ore and limestone mixture which is almost as same as that of coke and limestone mixture is shown in Table 1 as “ CaO/SiO_2 of burden”. The main altering test conditions are theoretical flame temperature T_f and slag basicity CaO/SiO_2 . Five cases were set as test conditions. Cases 1, 2, 3 were operated with T_f of 2 100, 2 300, 2 500°C respectively with the same basicity

Table 1. Compositions and grain sizes of ore, fuels and flux, and operation condition of the test plant.

a) Ore												
	Chemical compositions (%)									Size d_p (mm)	Blending ratio (%)	
	T-Mn	MnO	Mn ₂ O ₃	T-Fe	Fe ₂ O ₃	CaO	SiO ₂	Al ₂ O ₃	MgO			
A-Ore	49.5	3.8	73.6	3.0	4.3	0.1	6.2	4.0	0.1	11.3	36~47	
B-Ore	31.4	13.2	33.5	23.7	33.9	0.5	4.8	8.0	0.2	13.9	12~21	
C-Ore	51.4	42.7	29.0	3.2	4.6	3.7	6.9	3.7	2.2	12.6	31~32	
Limestone	tr	tr	tr	tr	tr	54.5	tr	tr	1.1	13.8	8~13	

b) Fuel												
	Proximate analysis (%)					Ultimate analysis (%)					Size d_p (mm)	DI (%)
	FC	VM	Ash	Mois		C	H	N	O	S		
Coke	87.8	0.9	10.7	0.6	96.7	0.2	1.4	1.0	0.65	32.2	86.3	
Coal	51.1	35.3	10.4	3.2	85.9	4.5	1.8	7.0	0.71	—	325 mesh > 90%	

c) Operation												
Items		Conditions										
Blast	Bosh gas (Nm^3/h)	898 ~ 903										
	T_f (°C)	2093 ~ 2514										
	O ₂ (Nm^3/h)	240 ~ 270										
	N ₂ (Nm^3/h)	150 ~ 170										
	Steam (kg/h)	0 ~ 36										
	Pulverized coal (kg/h)	240 ~ 267										
	PC/O ₂ (kg/ Nm^3)	0.98 ~ 1.00										
	Top pressure (Pa)	2.48×10^5										
CaO/SiO ₂ of burden (-)		0.97 ~ 1.5										
Tapping		6 ~ 10 times/d										

of burden to investigate the T_f influence. Cases 3, 4, 5 was operated with the burden basicity of 1.7, 1.4, 1.2 respectively with T_f of 2500°C to investigate the basicity influence.

3. Operational Results

3.1. Transition of the Operation and Results

The operational variations of test cases are shown in Fig. 4 using the mark of ●. The tapping number of each cases are 8, 8, 12, 8, 5 taps respectively. The data are mean values of several taps except the case transition periods. The mark of ○ in the figure is the calculation result of heat and mass balance model mentioned later. The marks of white and black arrow show the response to the operation action of T_f increase and decrease of CaO/SiO₂ of burden respectively. The cases 1 and 2 were operated with the ore blending ratio which was determined on the assumption that Mn partition equilibrium between slag and metal (R_{Mn}) expressed by

Eqs. (1) and (2) which was derived from the Schenck's equation^{8,9)} was constant at 3.3 considering the peeping test results. But [Mn] was about 67 and 70 % respectively and scheduled value 75 % of [Mn] was not obtained. Then the data of cases 1 and 2, and EAF¹⁾ and blast furnace¹¹⁾ were investigated and it was clarified that R_{Mn} decreased remarkably according to the increase of slag basicity as shown in Fig. 5.

$$\log(N_{Mn}/N_{MnO})^* = 1.586(\text{CaO/SiO}_2) - \log P_{CO} + 3.17 \quad \text{.....(1)}$$

$$R_{Mn} = \left[\frac{N_{Mn}}{N_{MnO}} \right]^{\text{experimental}} / \left[\frac{N_{Mn}}{N_{MnO}} \right]^* \quad \text{.....(2)}$$

Therefore aimed value of R_{Mn} was changed as shown by the mark of ★ in Fig. 5 and cases 3–5 were operated under the re-calculated operation conditions. As the result of this change, ferromanganese of [Mn]=75 % was stably produced after case 3 as shown in Fig. 4.

Examples of the results are shown in Table 2. Cases 3 and 5 in the table correspond to the conditions of high and low slag basicity respectively. From these results, it is demonstrated that high carbon ferromanganese can be produced in such a small furnace with fuel rate of 2600–2700 kg/t using non-coking pulverized coal and low quality coke blended with 56 % of non-coking coal.

As the fuel rate of this process is higher than that of EAF, [P] in the metal would increase by phosphorus in coke and coal. But its influence can be reduced by using the coal containing low concentration of phosphorus. It is considered that use of a large quantity of pulverized non-coking coal is favorable to this process in comparison with all coke operation because of increasing flexibility in selecting resources.

3.2. Raw Material Consumption and Heat Loss

Although the fuel rate decreased according to the T_f increase, the decreasing degree became smaller in the higher T_f region as shown in Fig. 4. And in relation to

Fig. 4. Operational variations of test cases.

Fig. 5. Relationship between basicity (N_{CaO}/N_{SiO_2}) and the degree of Mn partition equilibrium (R_{Mn}).

Table 2. Operation results of the test plant.

Case of operation		Case 1	Case 2	Case 3	Case 4	Case 5
Ore	Rate (kg/t)	3219	2546	2421	2567	2678
Blast	O ₂ (Nm ³ /h)	241	255	272	270	271
	N ₂ (Nm ³ /h)	151	151	170	170	170
	Steam (kg/h)	36	21	0	0	0
	PC (kg/h)	237	256	267	263	266
	T _f (°C)	2109	2292	2509	2513	2508
	Blast gas (Nm ³ /h)	897	899	908	902	903
Top gas	Temp. (°C)	650	694	722	720	729
	Pressure (Pa)	2.48 × 10 ⁵	2.48 × 10 ⁵	2.48 × 10 ⁵	2.48 × 10 ⁵	2.48 × 10 ⁵
Fuel rate	Dust conc (g/Nm ³)	35.8	47.8	69.7	69.8	71.9
	Coke (kg/t)	2580	1692	1087	1099	1117
Production	PC (kg/t)	2964	2032	1502	1514	1598
	Fuel (kg/t)	5544	3724	2589	2613	2715
Product	Production (t/d)	1.91	3.03	4.26	4.17	3.99
	Productivity (t/d·m ³)	1.39	2.21	3.11	3.04	2.91
Product	Temp. of hot metal (°C)	1395	1423	1380	1369	1377
	[% Mn] (%)	66.5	71.3	76.2	75.1	75.1
	[% C] (%)	6.6	6.8	6.9	6.8	6.6
	[% Si] (%)	0.13	0.23	0.17	0.37	0.22
	[% P] (%)	0.44	0.35	0.31	0.32	0.34
	(% MnO) (%)	32.1	26.4	23.3	30.0	32.1
	(% Al ₂ O ₃) (%)	15.0	16.7	17.8	15.7	15.8
	Slag CaO/SiO ₂ (-)	1.77	1.79	1.70	1.40	1.29
	Nm ³ /Nm ³ (-)	1.82	2.33	2.73	2.11	1.94
	Mn yield (%)	84.4	87.3	84.9	85.6	79.3

the fuel rate, bosh gas rate V_R decreased according to T_f increase and came to about $5115 \text{ Nm}^3/\text{t}$ (case 3) at 2500°C of T_f .

As shown in Fig. 4, the ore rate of case 2 decreased in spite of the same blending ratio as that of case 1 because Mn oxide reduction was promoted and metal amount produced per 1 ton of ore was increased as the result of T_f increase in case 2.

Although T_f was increased, the top gas temperature did not increased remarkably, and the heat input increase caused by T_f increase worked to the Mn smelting effectively and lead to the increase in production rate. The amount of heat loss obtained from mass and heat balance remained almost constant about 870 MJ/h and did not increased. So heat loss rate (MJ/t) decreased resulting in fuel rate and V_R decrease.

4. Discussion

4.1. Mn Partition between Metal and Slag

Although Mn partition between metal and slag has been already investigated,¹²⁾ Eq. (1) was used in this investigation as a first step because Mn concentration in ferromanganese is much higher than that of pig iron. But as the results of this test, it was clarified that Mn partition ratio between metal and slag depends not only on the slag basicity as expressed by Eq. (1) but also on the temperature and the dependency itself on the slag basicity should be restudied.

(1) Influence of the CaO/SiO_2 of Slag

Figure 6 shows the results of classification of the data by T_f and the coefficient of 0.382 is obtained about the relationship between Mn partition ratio and CaO/SiO_2 under the condition of constant T_f . Therefore Eq. (1) is expressed as follows.

$$\frac{\partial \log(N_{\text{Mn}}/N_{\text{MnO}})}{\partial (\text{CaO}/\text{SiO}_2)} = 0.382 \quad \dots\dots\dots (3)$$

(2) Influence of Temperature

If slag temperature is accepted as the temperature index, and if whole data is accepted as one group without classification by T_f and CaO/SiO_2 the dependency of

whole data on slag temperature shows a similarity to that of the equilibrium constant¹³⁾ as shown in Fig. 7. But the Mn partition ratio shown by the mark of $\blacktriangle$ ($\text{CaO}/\text{SiO}_2 = 1.3\text{--}1.5$) is higher than that shown by $\circ$ ($\text{CaO}/\text{SiO}_2 = 1.7\text{--}1.9$) and this fact is inconsistent with the CaO/SiO_2 dependency expressed by Eq. (3). On the other hand, Mn partition ratio depends also on the T_f as shown in Fig. 8 with the relationship expressed by Eq. (4) and this relationship could be classified by CaO/SiO_2 in this case. It is considered as follows about the reason why T_f has an effect on the Mn partition ratio. The heat exchange is improved by the T_f increase because reduction reaction of MnO is endothermic and this results in the promotion of Mn oxide reduction.

$$\frac{\partial \log(N_{\text{Mn}}/N_{\text{MnO}})}{\partial T_f} = 5.43 \times 10^{-4} \quad \dots\dots\dots (4)$$

And considering the relationship expressed by Eqs. (3) and (4), the inconsistency about the dependency on CaO/SiO_2 in Fig. 7 can be explained by the inclusion of the T_f influence. That is, if the data in Fig. 7 are classified by T_f and CaO/SiO_2 as shown by full line, the dependency on slag temperature expressed by Eq. (5) is

Fig. 7. Influence of slag temperature on the Mn partition ratio between metal and slag.

Fig. 6. Influence of slag CaO/SiO_2 on $M_{\text{Mn}}/N_{\text{MnO}}$.

Fig. 8. Influence of T_f on the Mn partition ratio between metal and slag.

Fig. 9. Comparison of Mn partition ratio between test plant results and calculation results by Eq. (7).

obtained and the relationship in Fig. 7 can be expressed consistently.

$$\frac{\partial \log(N_{\text{Mn}}/N_{\text{MnO}})}{\partial T_{\text{slag}}} = 1.46 \times 10^{-3} \quad \text{.....(5)}$$

Considering Eqs. (3)–(5), and assuming that CaO/SiO_2 , T_f , T_{slag} , are independent variable, the relationship between $\log(N_{\text{Mn}}/N_{\text{MnO}})$ and α shown by Eq. (6) was investigated, and straight line relationship was obtained which gives Eq. (7). The relationship between “ $\log(N_{\text{Mn}}/N_{\text{MnO}})_{\text{experimental}}$ ” obtained from this test and “ $\log(N_{\text{Mn}}/N_{\text{MnO}})_{\text{calculated}}$ ” calculated by Eq. (7) is shown in Fig. 9. Equation (7) seems to be reasonable from this figure.

$$\alpha = 0.382(\text{CaO}/\text{SiO}_2) + 5.43 \times 10^{-4} T_f + 1.46 \times 10^{-3} T_{\text{slag}} - \log P_{\text{CO}} \quad \text{.....(6)}$$

$$\log \frac{N_{\text{Mn}}}{N_{\text{MnO}}} = 0.382 \frac{\text{CaO}}{\text{SiO}_2} + 5.43 \times 10^{-4} T_f + 1.46 \times 10^{-3} T_{\text{slag}} - \log P_{\text{CO}} + 1.545 \quad \text{.....(7)}$$

4.2. Yield of Mn and SiO_2

The relationship between Mn yield and V_R was not obtained. It is considered that the effect of T_f increase to increase the dust concentration is canceled by the effect of V_R decrease to decrease the dust amount. Mn and SiO_2 yield was analyzed by using the following relationship:

$$\text{amount of loss} \propto \text{dust amount} \\ (= \text{dust concentration} \times \text{gas volume})$$

(1) Relationship between Mn and SiO_2 Loss and Dust Amount

The relationship between Mn and SiO_2 loss and dust amount is shown in Fig. 10. Here, Mn and SiO_2 loss are the difference between the input amount in the burden and output amount in metal and slag. Total amount of dust is calculated by Eq. (8) as the summation of the amount of dust caught and not caught at dust catcher.

$$W_{\text{dust}} = W_{d,d} + 10^{-3} W_{s,d} \times V_g \quad \text{.....(8)}$$

From the ratio of loss amount to dust amount, Mn and

Fig. 10. Relationship between Mn-loss, SiO_2 -loss and dust.

Table 3. Properties of dust in the top gas.

a) Size distribution (%)									
mesh	+32	32~60	60~100	100~150	150~200	200~325	325~	$\overline{d_p}$ (μm)	
Dust catcher	2.0	8.3	7.4	5.6	3.7	5.6	67.4	31.0	
Outlet of D.C.	tr	tr	0.02	0.04	0.08	0.35	99.51	22.1	
Total	1.2	5.0	4.4	3.3	2.3	3.4	80.4	27.4	

b) Chemical composition (%)									
	T-Mn	SiO ₂	Al ₂ O ₃	Co	MgO	T-Fe	Na ₂ O	K ₂ O	T-S
Dust catcher	30.1	11.3	6.8	6.4	0.9	2.8	0.5	1.9	0.9
Outlet of D.C.	30.8	15.1	7.9	6.4	1.4	1.1	1.1	3.8	1.7
Total	30.4	12.8	7.2	6.4	1.1	2.1	0.7	2.7	1.2

SiO_2 concentration in the dust were decided to be 45 and 25% respectively as mean values of the region shown by broken line. On the other hand, the Mn and SiO_2 concentration in the dust of this test were 30 and 13% respectively as shown in Table 3. Although there are slight differences between the value from Fig. 10 and Table 3, the former was used in the mass and energy balance model of ferromanganese production because of the difficulty of measuring the dust amount precisely.

(2) Influence Factor on the Dust concentration

It can be considered that T_f and T_{slag} as a temperature index of lower part of the furnace are factors of dust generation and T_g is a dust trap factor of packed bed. As a result of the relationship study between dust concentration and above factors, the relationship shown in Fig. 11 was found and expressed by Eq. (9).

$$D = 0.0875 T_f + 0.1 T_g - 214 \quad \text{.....(9)}$$

Mn and SiO_2 loss can be expressed by following equation using the above mentioned results.

$$D_{\text{Mn}} = \frac{0.45}{1000} D \cdot V_R \quad \text{.....(10)}$$

$$D_{\text{SiO}_2} = \frac{0.25}{1000} D \cdot V_R \quad \text{.....(11)}$$

(3) Possibility of Mn Vaporization Loss

Since boiling temperature of Mn (2050°C) is lower than T_f and dust shape looks like sphere as shown in Fig. 12, Mn vaporization can be considered to have high

Fig. 11. Influence of top gas temperature and T_f on the dust concentration in the top gas.

Fig. 12. Photograph of the dust sampled from the off gas.

Fig. 13. Dissection results of the melting furnace.

potential for Mn loss.³⁾

The furnace was quenched by nitrogen blown through the tuyeres and the intersection study was performed after the smelting test. Its results are shown in Fig. 13. Here, the temperature in the furnace is estimated through the analysis of the crystallite size of sampled coke in the direction of (002). Mn ore is heated rapidly to about 1300°C in the upper part of the furnace and Mn oxide is reduced to MnO there, but final reduction to metallic Mn occurs in the smelting zone higher than 1500°C.

Figure 13 also shows the relationship between T_f and the temperature T_{Mn} which is calculated by Eqs. (12) and

(13), where W_{Mn} is Mn loss calculated as the difference between Mn amount in burden and in metal and slag, and a_{Mn} is calculated according to Schenck's method.⁸⁾ Here it is assumed that Mn loss is caused only by Mn vaporization and fume dust originating from Mn vapor is not trapped in the furnace. Although these assumption may be have some problems,³⁾ it is overlooked here because qualitative effect of T_f on Mn loss is intended to be investigated.

$$P_{Mn} = \frac{22.4 \cdot W_{Mn} \cdot P_{top}}{54.94 \cdot V_B} \dots\dots\dots(12)$$

$$Mn(l) = Mn(g)$$

$$\Delta G^\circ = 235768 - 197.0 \cdot T_{Mn}^{10)} \dots\dots\dots(13)$$

$$= -R \cdot T_{Mn} \cdot \ln(P_{Mn}/a_{Mn})$$

Figure 13 shows that T_{Mn} corresponds to the temperature near the tuyere level where Mn is finally reduced. And the relationship between the T_{Mn} and hot metal temperature is not found but the straight line relationship between T_{Mn} and T_f is obtained. That is, T_{Mn} increased according to the T_f increase. Therefore if T_f is increased, the temperature of tuyere level zone also increase and it results in the increase in Mn vapor pressure leading to Mn loss increase.

4.3. Mass and Energy Balance Model of Smelting Mn Ore

Mass and heat balance model of smelting Mn ore was made taking the above mentioned results into account. In this model, it is assumed that Mn oxide is reduced to MnO by CO at the temperature below 1000°C, and thereafter MnO is reduced to Mn by C. Blast condition is calculated if T_f , PC/O₂, steam rate is given. Its estimation results for cases 1 to 5 are shown in Fig. 4 using the mark of ○ compared with this test's results (●), and its accuracy is acceptable.

4.4. Estimation of the Indices of Commercial Plant with Production Capacity of 170 t/d

The indices of commercial plant with production capacity of 170 t/d producing ferromanganese were estimated by using the above mentioned model. Heat loss and the furnace profile of this process and blast furnace BF¹¹⁾ which produced ferromanganese were compared and set as shown in Table 4, and the indices were calculated as shown in Table 5.

(1) Profile of the Furnace

The productivity per unit furnace volume obtained from this test is 3.11 t/(d·m³) and much more than that of BF. But it can be considered that productivity per unit furnace volume of this process, which is an index of heat transfer rate (J/(m³·h)) and reaction rate (kmol/(m³·h)), can be maintained in the large scale furnace because the coke and ore particle size used in this test are 20–50 and 5–20 mm respectively which are not different from those of BF. Therefore the productivity of the commercial plant was fixed at 3.11 t/(d·m³) and furnace volume was calculated to be 55 m³ from the productivity and production capacity.

On the other hand, the productivity per unit of hearth

Table 4. Profile comparison between the blast furnace and this process.

		Blast furnace ⁽¹⁾		This process	
		Case A	Case B*	T. P. *	C. P. *
Production	Production (t/d)	280	336	4.26	170
	Productivity (t/d·m ³)	0.622	0.746	<u>3.11</u>	<u>3.11</u>
	Productivity (t/d·m ³)	13.2	<u>15.8</u>	6.7	<u>15.8</u>
Profile	Volume (m ³)	450		1.37	55
	Hearth dia. (m φ)	5.2		0.9	3.7
	Height (m)	21.2		2.15	5.11
	Wall surface area (m ²)	364.3		6.08	59.4
Heat loss	Heat loss (MJ/h)	30995	29522	870	4971
	Heat flux (MJ/h·m ²)	<u>84.9</u>	81.2	143.1	<u>84</u>
	Rate (MJ/t)	2657	2109	4904	703

* Case B : O₂ enriched blast condition

T. P. : This test plant

C. P. : Commercial plant

Table 5. Rate comparison between the blast furnace and this process.

		Blast Furnace ⁽¹⁾		This Process		
				Test Plant		Commercial Plant
		Case A	Case B *	Basic	Acid	Basic
Material	Ore rate (kg/t)	2035	1968	2421	2497	1981
Fuel	Coke rate (kg/t)	1420	1374	1087	664	711
	PC rate (kg/t)	0	0	1502	1063	826
	Fuel rate (kg/t)	1420	1374	2589	1727	1537
Total coal consumption (kg/t)		2190	2120	3179	2087	1923
Blast	Air (Nm ³ /t)	Hot 4053	Hot 3153	0	0	Cold 575
	O ₂ (Nm ³ /t)	0	154	1530	886	705
	N ₂ (Nm ³ /t)	0	0	926	0	0
	Steam (kg/t)	0	0	0	66	0
	Tf (°C)	2190	2420	2509	2500	2500
	Bosh gas (Nm ³ /t)	4902	3963	5115	2800	2766
	Top pressure (Pa)	2.04×10 ⁵	2.08×10 ⁵	2.48×10 ⁵	2.48×10 ⁵	2.48×10 ⁵
Product	Temp. of metal (°C)	1450	1470	1380	1400	1450
	[% Mn]	(%) 76.4	76.6	76.2	74.5	74.5
	(% MnO)	(%) 12.0	11.1	23.3	44.5	13.8
	Slag CaO/SiO ₂ (-)	1.54	1.56	1.70	0.77	1.85
	Mn yield (%)	89.5	90.8	84.9	91.6	92.2

* Case B : O₂ enriched blast condition

area obtained in this test is 6.7 t/(d·m²) and about half of that of BF. But it was assumed to be 15.8 t/(d·m²) of BF because bosh gas velocity of this test is smaller than that of BF and there are plenty of room for gas velocity. Therefore hearth diameter of the commercial plant was determined to be 3.7 mφ.

(2) Heat Loss

The heat loss of Thyssen's blast furnace is 29522–30995 MJ/h which corresponds to about 84 MJ/(m²·h) of the heat flux in wall. On the other hand, the heat flux of this test plant is 143 MJ/(m²·h) and slightly higher than that of the blast furnace. In the case of commercial plant, 84 MJ/(m²·h) was adopted because heat flux of this process would decrease according to the scale up. As the result of this, heat loss of commercial plant become to 4971 MJ/h.

(3) Indices Comparison between This Process and the Blast Furnace

The indices was estimated for following two operation conditions as is performed in EAF process, i.e. acid operation: MnO is left in slag in rather high level to be

used as the material for producing silicomanganese, basic operation: MnO is reduced as much as possible and slag is abandoned. "Basic" and "Acid" in Table 5 correspond to "basic operation" and "acid operation" respectively. The estimation results showed that high T_f was favorable in both operational conditions from a viewpoint of fuel rate and ore rate. Moreover it was found by model estimation that partial pressure decrease of CO by air addition to oxygen blowing was effective to reduce MnO further. So the operational conditions shown in Table 5 were chosen as appropriate ones and various indices were estimated as shown in the table.

Although this process is equivalent to the blast furnace process from a viewpoint of Mn yield and ore rate, this process is superior than blast furnace from a viewpoint of fuel source because non-coking pulverized coal and low quality coke containing 56% of non-coking coal can be used in this process. Moreover coal consumption of this process can be reduced less than that of the blast furnace because a plenty of the pulverized coal is used in this process. Through these investigations, this process has been confirmed to be useful one as a ferromanganese production process instead of electric furnace method and blast furnace method.

6. Conclusion

Experiments of high-carbon ferromanganese production were performed using an experimental melting furnace with coke packed bed injected with oxygen and pulverized coal and following results were obtained.

(1) In the experimental plant, high-carbon ferromanganese of [Mn]=75% was produced stably, using the coke blended with 56% non-coking coal, with coal rate of 1502 kg/t, coke rate of 1087 kg/t and productivity of 3.11 t/(d·m³).

(2) On the basis of the results obtained by this tests and blast furnace operation, the indices of a commercial plant of 170 t/d in production capacity was estimated. Although the fuel rate is higher than that of blast furnace method but the total coal consumption is less than that of blast furnace because a large quantity of pulverized

coal is used in this process. Through these investigations, this process has been confirmed to be useful one as a ferromanganese production process instead of EAF method and BF method.

Nomenclature

a_{Mn}	activity of Mn in ferromanganese (—)
CaO	weight percent of CaO in slag (wt%)
D	dust concentration in the raw gas (g/Nm ³)
D_{Mn}	Mn loss (kg/t)
D_{SiO_2}	SiO ₂ loss (kg/t)
N_i	molar fraction of i component (—)
P_i	partial pressure of i component (Pa)
P_{top}	top gas pressure (Pa)
R	gas constant (8.314 J/mol·K)
R_{Mn}	degree of Mn partition equilibrium between metal and slag (—)
SiO ₂	weight percent of SiO ₂ in slag (wt%)
T_g	top gas temperature (°C)
T_{slag}	slag temperature (°C)
V_B	bosh gas volume (Nm ³ /h)
V_g	off gas volume (Nm ³ /tap)
V_R	bosh gas rate (Nm ³ /t)
W_{dust}	dust amount (kg/tap)
$W_{d,d}$	dust amount trapped by dust catcher (kg/tap)
$W_{s,d}$	dust concentration in the gas after dust catcher (g/Nm ³)

W_{Mn}	Mn loss (kg/h)
ΔG°	Gibbs free energy (J)

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