

# Conservation of high phosphorous high manganese ores by sintering of dephosphorised concentrate

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## INTRODUCTION

The ores of manganese enjoy a high order of importance since no quality steel can be produced without the addition of small amounts of manganese. In view of its economic importance it becomes imperative to take stock of the facts about this mineral in a comprehensive manner.

According to an IBM estimate, the recoverable reserves of manganese ore as on 1-1-1983 are 138.00 million tonnes comprising 18 million tonnes in measured category 32.00 million tonnes in indicated category and 88.00 million tonnes in inferred category.

According to the projection of iron and steel production in the country furnished by the Steel Authority of India Ltd. for 7th plan period and by Planning Commission for subsequent period up to the end of century, the requirement of low grade and high grade manganese ores would be as given in table 1.

On the basis of the projection it is expected that the consumption of all grades of manganese ore for the period 1985--2000 would exceed 18 million tonnes. Thus the entire measured category of manganese ore would be getting exhausted by the end of the century. To meet such an alarming situation, it is imperative to conserve the manganese ores by every possible means.

**Table - 1**

( '000 Tonnes)			
Year	Low Grade	High Grade	Total
1985-86	730	418	1143
1986-87	737	476	1213
1987-88	863	528	1391
1988-89	935	608	1543
1989-90	981	723	1704
<b>Total</b>	<b>4246</b>	<b>2748</b>	<b>6994</b>
1994-95	1348	790	2138
1999-2000	1712	1018	2730

In respect of metallurgical high grade ore (+45% Mn) which is employed in ferro manganese production, the Indian reserves in measured category are 5.72 million tonnes as per the joint study conducted by GSI & IBM in 1975.

On the basis of probable steel production envisaged, the estimated domestic requirement of bulk ferro alloys in 1990 & 2000<sup>2</sup> would follow the following pattern. ( Table 2 ).

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**Table — 2 : Projected production of ingot steel and ferro-manganese with the estimated requirement of high grade manganese ore, for FeMn production.**

(000'M Tons)

Year	Ingot steel	FeMn	High grade Mn Ore+45
1985-86	9820	165	413
1986-87	10460	190	476
1987-88	11220	211	528
1988-89	11880	243	608
1989-90	14050	289	723
<b>Total</b>	<b>57430</b>	<b>1098</b>	<b>2748</b>
1994-1995	20995	315	790
1999-2000	27115	407	1018

On the basis of the above projection, it is expected that the consumption of high grade manganese ore for 1990 - 1995 & 1995-2000 would be 3.78 million tonnes and 4.52 million tonnes respectively. The total consumption of high grade manganese ore for the period 1985-2000 would be therefore 11 million tonnes. Thus there would be substantial short fall of high grade manganese ore in terms of the measured category and the demand pattern.

This demands top most priority for conserving the high grade manganese ore employed for ferro manganese production.

For production of ferro manganese, ores with high manganese content and phos content below 0.16% are only used. A study on all India basis regarding the mode of occurrence and quality of different manganese deposits has been conducted by IBM which is exhibited in Annexure - I. A major portion of metallurgical grade manganese ores is occurring in Maharashtra, M.P. belt under the leasehold of M/s. Manganese Ore India Ltd. (MOIL), Govt. of India Undertaking. M/s. MOIL has compiled the

estimated reserves of manganese ores mine wise with respect to Mn & Phosphorous content which is furnished in Annexure - II. From the above it is observed that the major portion of manganese ore reserves in our country fall in the high phosphorous high Mn grade reserves.

In order to keep the phosphorous content within limits, selective mining and hand sorting are being practised in various mines in our country. The selective mining is causing fast depletion of low phosphorous reserves while hand sorting is resulting in enormous stockpiling as wastage of low grade Mn ores.

The only solution to the above problem is dephosphorisation. A beginning has been made in this direction by the mineral dressing laboratories in our country. It was established that most of the high phos ore are amenable to dephosphorization by flotation & magnetic separation, after fine grinding for liberation of phosphate minerals. The dephosphorised concentrate in finely ground form needs agglomeration before use in smelters. Agglomeration processes like briquetting and pelletization have serious drawbacks. The former needs sodium based binders which is not acceptable to arc furnaces due to its detrimental effects on the refractories. The latter route requires large quantities of oil which is prohibitively costly. The only alternative was sintering. But extensive R & D studies were necessary for developing a sintering route amenable to the finely ground concentrate. The present sintering studies were sponsored by the Mineral Development Board and the sintering studies were conducted at the R&D Centre attached to the sinter plant at Chandrapur. Indian Bureau of Mines had earlier conducted pilot plant dephosphorisation studies on manganese ore from Tirodi mines, located in Balaghat Dist., M. P. Their concentrates were used for the sintering studies.

Balaghat Mines near Tirodi mines generates during, mining high Mn, low phos fines below 6 mm size. Hence it was felt that sintering a blend of Balaghat fines and dephosphorised

concentrate would pave the way for utilisation of both the materials. The present studies were therefore conducted on such a blend.

**Test Procedure :**

Since efficiency of sintering depends on several factors which interact, the factorial design of experimentation was adopted. The second phase of this project would involve production of bulk sinters( about 1000 M.T.) employing our 50 TPD sinter plant. Hence sintering parameters like bed depth, vacuum, ignition etc, were maintained at levels available on the production

strand. The factorial design covered raw material components viz. dephosphorised concentrate, coke, return fines and water. Mn ore fines constituted the balance for maintaining a constant weight of mix for the test.

On the basis of plant practice a base level (BL) was fixed for the four factors. For each factor a higher and lower levels with respect to BL were fixed. The higher level was designated with a +ve sign and the lower level with a -ve sign with four factors and two levels a full set of experiments designated as a "Matrix" was designed as given below :

Sr. No	Coke	Dephosphorised concentrates	Return fines	Water	Yield	R/F Ratio Return fines added Return fines generated
1.	+	+	-	-	Y 1	R 1
2.	-	-	-	-	Y 2	R 2
3.	+	-	-	+	Y 3	R 3
4.	-	+	-	+	Y 4	R 4
5.	+	+	+	+	Y 5	R 5
6.	-	-	+	+	Y 6	R 6
7.	+	-	+	-	Y 7	R 7
8.	-	+	+	-	Y 8	R 8
9.	+	-	-	-	Y 9	R 9
10.	-	-	+	-	Y 10	R 10
11.	-	+	-	-	Y 11	R 11
12.	+	+	+	-	Y 12	R 12
13.	-	-	-	+	Y 13	R 13
14.	+	-	+	+	Y 14	R 14
15.	+	+	-	+	Y 15	R 15
16.	-	+	+	+	Y 16	R 16
17.	Base Level (BL)					

For preliminary test the matrix was limited to eight tests (Sr.No 1-8). The main depended variable was lump yield, being the main purpose of sintering. The effect of each factor on lump yield was evaluated from the 'a' values.

$$a = \frac{\Sigma (+) - \Sigma (-)}{N}$$

Where  $\Sigma (+)$  = Sum of the yield percentages for + levels

$\Sigma (-)$  = Sum of the yield percentages for - levels

N = Number of tests in the matrix.

For each factor in the mix viz. coke, dephosphorised concentrates, return fines and water, 'a' values were designated as a1, a2, a3, & a4 respectively.

The absolute quantity of a particular 'a' value with its sign indicates the effect of that factor of sinter yield. Smaller the quantity of 'a' value closer the factor to the optimum level. Further the sign of 'a' value indicates the direction of the optimum level. This is an example.

- Say optimum level of coke is 8%
- For a test base level of coke fixed at 7%, with + level at 8% and (-) level at 6% then  $\Sigma (+)$  would be greater than  $\Sigma (-)$  hence the 'a' value shall bear (+) sign.
- For a test, base level of coke fixed at 9% with + level at 10% and -ve level at 8% then  $\Sigma (+)$  would be smaller than  $\Sigma (-)$  hence the 'a' value shall bear (-) ve sign.

Based on the 'a' values of a matrix the next matrix is planned. Thus by successively changing base levels and steps, optimisation of each factor controlling sintering is achieved.

The generation of return fines and its addition to the mix has to be equal in sintering process i. e. the ratio

$$R = \frac{\text{R/F added}}{\text{R/F generated}} = 1$$

This aspect is also to be considered in sintering studies. Each pot test consisted of five runs. The first run was conducted with out addition of return fines. Dephosphorised manganese ore fines, dephosphorised concentrate and coke were mixed at preset levels ( totaling 50 kg which is the pot capacity ). The first run was designated as A 11. The sinter cake was broken manually to (-) 75 mm size. The broken material was screened on 6 mm sieve. The -6 mm material constituted the R/F and +6 mm constituted the lump sinter. The return fines generated in run A 11 were used for preparing the sinter mix for next run i. e. A 12. and R/F of

A 12 were used for A 13. The idea of repeating the run three times was to stabilize the quality of R/F.

Employing R/F generated in A 13, the final run A 2 was conducted. Employing the R/F from A2 run, A3 was conducted. During the runs A2 & A3 the proportion of raw mix, was maintained constant. Runs A2 & A3 were considered as duplicate run and their average values of sinter yield was thus computed.

For a matrix of 16 tests 80 pot runs were conducted and 5 additional runs were conducted at the base level. In all 85 pot runs constituted a full matrix and 45 pot runs constituted a limited matrix.

The average yield for each test ( Average of runs A2 & A3 ) were tabulated against the test levels and 'a' values were computed. The Return fines ratio was calculated for each test and tabulated.

#### Experimental Procedure :

After fixing the raw material levels for a particular matrix, corresponding weighed quantities of the raw materials and water were taken.

The Mn ore fines were first charged and mixed dry for 1 min. Then a predetermined quantity of water was added and wet mixing was carried out for 1 min. The dephosphorised concentrate/ground ore was then slowly added so as to coat the surface of wet Mn ore grains. The dephosphorised concentrate was added in stages alternating the same with addition of water. Thus the material formed a coating on the Mn ore grains layer by layer. Thereafter balance quantity of water, return fines and coke were added while mixing was continued for 1 min.

A 25 mm thick bed layer of sinter ( -12 + 8 mm ) was spread over the sinter pot grate. The wet mix was charged over the bed layer.

The burner air and oil supplies were started and the burner was lighted keeping the burner hood over the refractory lined chamber for stabilization.

The sinter fan was started and vacuum was adjusted at 250 mm W. G. by bleeding through a slide valve. The burner hood was rolled over to the sinter pot for ignition, simultaneously starting the timer. An ignition time of 1½ mts was practised for all the tests. After ignition the burner was shut off and burner hood was rolled back.

The wind box temp. was recorded at 1 minute interval. The temperature of the outgoing gases rises slowly and reaches a peak. The time elapsed from beginning of ignition to drop in exhaust gas temperature is taken as sintering time.

The sinter pot side panels were opened and the sinter cake was exposed for visual examination.

The cake was then pushed to a container kept on the floor in order to break it. Bigger

lumps were manually broken to (—) 75 mm. The material was then allowed to cool and sieved on 6 mm screen.

The —6 mm material was used as return fines for subsequent runs of that particular test.

Tumbling index and abrasion index of the sinters were determined as per Indian Standard Specification IS 6495-1972. The + 6mm sinter was subjected to sieve analysis and chemical analysis.

After confirmatory runs, the final sinter was also subjected to (a) Mineralogical analysis. (b) Reducibility (c) X-ray diffraction.

#### Experimental work :

#### Raw Materials :

The chemical and size composition of the raw materials employed is given in Table No. 3

Table No. 3

Radical	Chemical Analysis			Size Analysis			
	Assay %			Size fraction microns	Wt. %		
	Coke	Mn ore fines	Dephos conc.		Dephos conc.	Mn ore fines	Coke
Mn (T)	—	43.38	41.79	+ 2800	—	25.6	5.00
Fe (T)	2.96	7.50	9.41	— 2800 + 2000	—	36.0	21.00
P	0.13	0.12	0.14	— 2000 + 500	—	28.0	38.33
SiO <sub>2</sub>	18.78	9.25	11.80	— 500 + 300	—	3.6	16.67
Al <sub>2</sub> O <sub>3</sub>	9.96	2.07	1.28	— 300 + 150	3.9	2.8	11.66
CaO	—	0.48	3.84	— 150 + 106	5.5	1.8	3.67
Mgo	—	0.32	2.50	— 106 + 75	9.4	1.2	2.00
Fixed Carbon	61.0	—	—	— 75 + 45	11.5	0.6	1.00
Ash	36.0	—	—	— 45	69.7	0.4	0.67
V.M	2.4	—	—	Total	100.0	100.0	100.00

**Sinter Pot Studies :**

Four preliminary matrices were conducted and each time the base levels and step levels were successively changed on the basis of 'a' values obtained from the previous matrix. The

final matrix was conducted on the following conditions.

**Final Matrix. :**

The matrix tests were conducted with the dephosphorised flotation concentrate and the results are as follows :—

**Matrix - V**

Test No.	Coke	Dephos concentrate.	R/F	Water	Yield	RF Ratio RF added RF generated
1.	(-) 5.1	- 6.0	(-) 12.0	(-) 2.5	68.0	0.98
2.	- 5.1	+ 7.0	(-) 12.0	(+) 3.5	69.6	1.05
3.	(-) 5.1	- 6.0	(+) 13.0	(+) 3.5	69.4	1.07
4.	+ 5.5	- 6.0	(-) 12.0	(+) 3.5	69.0	1.08
5.	(-) 5.1	+ 7.0	(+) 13.0	(-) 2.5	69.2	1.08
6.	(+) 5.5	- 6.0	(+) 13.0	(-) 2.5	69.8	1.19
7.	(+) 5.5	+ 7.0	(-) 12.0	(-) 2.5	68.4	1.00
8.	+ 5.5	+ 7.0	(+) 13.0	+ 3.5	69.0	1.10
B. L	5.3	6.5	12.5	3.0	69.6	1.16
		a 1	a 2	a 3	a 4	
'a' Value		0	0	(+) 0.3	(+) 0.20	

From the matrix results it can be seen that under test 7 conditions the return fines balance while yield was 68. 4%

The test No. 7 was therefore repeated thrice and the results read as follows.

Repeat	Coke Kg.	Levels Dephosporised concentrates. Kg.	RF Kg.	Water Kg.	Yield Kg.	RF added RF Generated
1.	5.5	7.0	12.0	2.5	68.3	0.99
2.	5.5	7.0	12.0	2.5	68.5	1.01
3.	5.5	7.0	12.0	2.5	68.5	1.01

Sinter produced during confirmatory runs was subjected to quality testing, chemical analysis reducibility, mineralogical analysis, screen analysis porosity etc.

**Sinter quality testing :**

Chemical analysis of sinter with dephosphorised concentrate.

Radical	Assay %
Mn (T)	45.50
Mn O	48.50
Fe (T)	7.46
SiO <sub>2</sub>	13.25
Al <sub>2</sub> O <sub>3</sub>	3.80
CaO	0.61
MgO	0.68
Mn (M)	0.60
P	0.14

**Tumbling and abrasion indices :**

The TI & AI were determined as per IS : 6495 and the results are given in Table No. 4

**Table No. 4**

Sample	TI	AI
1. Balaghat plain Mn-Ore Sinter	84.5	2.2
2. Dephosphorised concentrate blended sinter	84.6	2.8

**Screen analysis :**

The final sinter was subjected to screen analysis after breking to - 75 mm size. The results are given in Table No. 5

**Table No. 5**

Size range	Wt. %	cum% retained
75 + 40 mm	18.2	18.2
40 + 30 mm	32.8	51.0
30 + 20 mm	22.2	73.2
20 + 10 mm	14.0	87.2
10 + 6 mm	11.9	99.1
6 mm	0.9	—

AV Sinter size by harmonic mean = 20.57 mm

**Porosity :**

The porosity determination was made on Balaghat Mn-ore lumps, Balaghat plain sinter and Mn ore, dephosphorised concentrate mix sinter using Beckman air pycknometer. The results are given in Table No. 6

**Table No. 6**

Sample	AV Porosity
Mn ore lumps	25.27
Plain Mn ore sinter	46.20
Mn ore dephosphorised Concentrate mix sinter	46.35

Note :- Porosity reported is total porosity.

**Mineralogy :**

Comparative mineralogical analysis was conducted on the Balaghat Mn ore, plain Mn ore sinter and sinter from Mn ore, dephosphorised concentrate mix.

**I Manganese Ore :-**

— Manganese ore minerals

- (a) Major : over 15%
  - Hollandite : (BaK)<sub>1.2</sub> MnO<sub>16</sub>
  - Braunite : 3 Mn<sub>2</sub>O<sub>3</sub> Mn<sub>2</sub>O SiO<sub>3</sub>

(b) Subordinate : cryptomelene  
 $K_{1.2}Mn_8O_{16}nH_2O$  ( 7 to 15% )  
 pyrolusite  $MnO_2$

(c) Minor : sitparite bixbyite  $Mn_2O_3$

— Silicate minerals :

(a) Minor :  
 (3—7) Quartz  $SiO_2$

(b) Very minor : Garnet

(c) Traces : Apatite

II. Plain Mn ore sinter :-

— Manganese minerals.

(a) Major : Hausmanite (  $Mn_3O_4$  )  
 ( Over 15% )

(b) Subordinate : Manganosite (  $MnO$  )  
 Silicate glassy slag

(c) Minor : Ferroan tephroite

(d) Very minor : Hematite  $Fe_2O_3$

III. The dephosphorised concentrate mixed sinter.

(a) Major : Hausmanite (  $Mn_3O_4$  )  
 (Over 15%)

(b) Subordinate : Manganosite (  $MnO$  )  
 ( 7 to 15% ) Silicate glassy  
 slag

(c) Minor : None  
 ( 3 to 7% )

(d) Very minor : Hematite  $Fe_2O_3$   
 (1 to 3%) Jacobsite (  $Fe, Mn$  ) $_3O_4$

X-ray Diffractograms of the three samples  
 in Annexure - III

**Reducibility :**

These tests were carried out on Balaghat  
 Mn ore, plain Mn ore sinter and Mn ore dephos-  
 phorised concentrate mix sinter.

The results of reducibility tests are graphi-  
 cally depicted in Annexure - 4. The chemical  
 analysis of the feed samples and reduced pro-  
 ducts are given below.

**Table No : 7**

Sr. No.	Sample description	Assay %			Reducibility	
		Befoer reduction			After reduction	
		Fe	Mn	MnO	MnO	
1.	Balaghat Mn ore	5.00	49.5	11.9	48.28	33.5
2.	Plain Mn ore sinter	8.80	45.8	47.7	58.15	17.1
3.	Dephosphorised concentrate mixed sinter	7.46	45.5	48.5	62.87	19.4

Reducibility is the percentage of oxygen  
 removed in 180 mts of reduction with respect to  
 original oxygen present in the feed computed

on the basis of chemical analysis and loss of  
 weight.



### Discussions of Results :

The optimum sintering conditions were achieved in Test No. 7 recording unity R/F ratio

with sinter yield 68.4. Higher sinter yields could be obtained in other tests but R/F ratio deviated from unity. Optimum raw mix composition was as follows :

Material	Wt. (Kg)	Wt.% Dry basis	Wt.% Wet basis
Coke	5.5	11.0	10.48
Dephosphorised concentrates	7.0	14.0	13.33
Return Fines	12.0	24.0	22.86
Balaghat Mn ore fines	25.5	51.0	48.57
Water	2.5	—	4.76
	52.5	100.0	100.0

The sintering conditions were as follows :

Grate area	=	0.135
Total bed depth	=	150 mm
Hearth layer	=	25 mm
Ignition time	=	1.5 mts
Suction below grate	=	250 mm W.G.
Sintering time	=	16 mts.

Blending of dephosphorised concentrate to the sinter mix did not affect the sinter strength as indicated by the following :

	Plain Mn ore	Dephosphorised concentrate blended sinter
T. I	84.5	84.6
A. I	2.2	2.8

The dephosphorised concentrate blended sinter was in a higher state of reduction containing more MnO phase as compared to predominant  $Mn_3O_4$  phase observed in plain sinter as indicated by mineralogical and X-ray diffraction studies. This may be attributed at first place to the presence of psilomelene in the dephosphori-

sed concentrate compared to hollandite in the Balaghat manganese ore fines and secondly to the increased surface area of the ground concentrate.

The porosity of dephosphorised concentrate blended sinter under optimum conditions is the same as plain Mn ore sinter indicating that blending of finely ground concentrate had no adverse effect.

The reducibility curves of plain sinter and dephosphorised concentrate blended sinter indicate that in the initial stages the rate of loss of oxygen is higher for dephosphorised concentrate. Blended sinter reaches more or less steady weight after 20 minutes of reduction while the plain Mn ore sinter takes 50 minutes to reach the same stage. Thus there would be saving of reduction time in the case of dephosphorised blended sinter.

### Summary and conclusions :

The technical viability of sintering a blend of dephosphorised concentrate fines and reject high grade fines has been established.

The optimum blend of 274.5 Kg. of dephosphorised concentrates with 1 Ton of Mn ore fines was established yielding 68.4% usable sinter. In the optimum conditions the coke and water additions were 10.48% and 4.76% respectively of the wet raw mix.

Addition of fine ground dephosphorised concentrate did not affect the strength of sinters in any way. The TI and AI values were quite adequate for ferromanganese smelting practice. The porosity of dephosphorised concentrate blended sinter was identical to that of the plain Mn ore sinter and far superior to the Mn ore.

The dephosphorised concentrate blended sinter was in a higher stage of reduction compared to plain Mn Ore sinter indicating considerable smelting benefits.

Blending of dephosphorised concentrate shall not only pave the way for utilisation of high phos ores and also reduce the reduction load on the arc furnace. The furnace throughput would be higher thereby improving the overall economics of smelting. Further, appreciable saving in electrical energy, coke, electrode paste and refractory are anticipated adding to the smelting economics.

In India smelters are facing the problem of getting low phos high Mn metallurgical grade manganese ores which constitutes the major furnace burden. Utilisation of dephosphorised concentrate would be a great relief to the smelter operators in view of the scarcity of metallurgical grade ores.

The advantages of using manganese ore sinters in ferromanganese smelting are well known and time is not far off when all the smelters will take up installation of sinter plants to conserve the generated fines. Blending of dephosphorised concentrates to the sinter mix would reduce the

dependence of smelters on lumpy metallurgical grade ores to a large extent.

Since sufficient quantities of high phos Mn ore fines are available as rejects, these can constitute the feed stock for dephosphorisation. On the basis of no cost for these rejects, Rs. 200/- M.T. of concentrate towards dephosphorisation Rs. 180/- for reject high Mn low phos manganese ore fine and Rs. 220/- M. T. towards sintering, the cost of low phos sinter works out at Rs. 400/- M. T. Thus it can be seen that the sintering route of utilizing dephosphorized concentrates is economical as similar grade Mn ore lumps would cost around Rs. 500/- M. T.

Even on the basis high phos lumpy ore as feed stock for dephosphorization and assigning a cost of Rs. 300/- M. T. for this material, the final low phos sinter would cost around Rs. 496/- which is slightly lower than the cost of low phos high Mn Lump ores.

Selective mining and hand sorting has already done extensive damage to our country's manganese ore reserves. Now that the agglomeration bottleneck has been overcome, it is high time to go for dephosphorisation of manganese ores on a big scale for conserving the remaining reserves of this important mineral.

#### **Acknowledgement :**

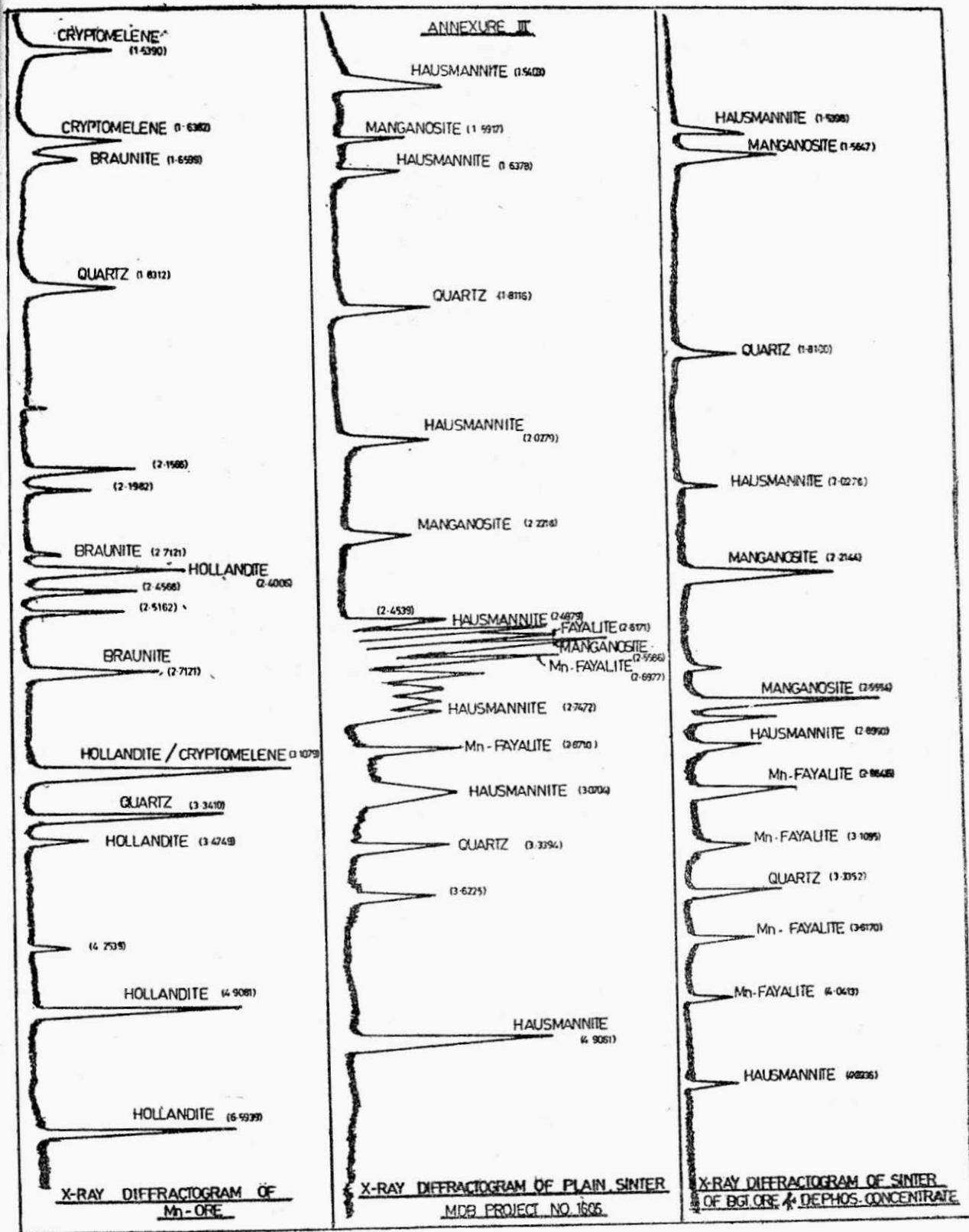
The authors are thankful to the Mineral Development Board for sponsoring this investigation and for permission to present this paper.

The active co-operation of M/s. Maharashtra Electosmelt Ltd. for conducting this investigation is acknowledged.

The authors are thankful to the Indian Bureau of Mines and National Mineral Development Corporation for mineralogical investigation and reducibility tests respectively.

**ANNEXURE — I : Different manganese belts / areas their mode of occurrences and mineralogy**

Belt/area	Mode of occurrence	State / District	Ore Mineral	Remarks
Madhya Pradesh Maharashtra.	Primary bedded deposits (Gondite Series) and supergene enrichments of bedded or reef deposits.	Madhya Pradesh: Chindwara, Balaghat and Jabua Maharashtra : Nagpur and Bhandara.	Braunite, psilomelane, pyrolusite, cryptomelane.	Except 2 deposits namely Ukwa and Bharweli all other deposits in the belt are high in Phosphorous (0.18)
Singhbhum Keonjhar Bonai belt.	Supergene enrichment and replacement deposit. Lateritoid ore is also common.	Bihar:Singhbhum:Orissa: Keonjhar and Bonai.	Pyrolusite and psilomelane with wad.	High iron and low phosphorus.
Ganapur.	Primary ore bodies of typical 'Gondite' series and supergene oxidation.	Orissa : Sundergarh.	Primary braunite, rhodinite and other manganese silicates with pyrolusite and psilomelane.	High phosphorous.
Koraput Kala- handi Patna.	Supergene enrichment of primary ore bodies (Khondalite).	Orissa:Koraput Kalahandi and Bolangir.	Psilomelane, pyrolusite wad.	High Iron and phosphorus.
Bellary/Hospet.	Supergene enrichment and replacement deposits. Lateritoid ore also common.	Mysore : Bellary.	Psilomelane, and wad with subordinate pyrolusite type oxides.	High iron, low phosphorus, and silica.
North Kanara.	Lateritoid and surfacial deposit.	Mysore : North Kanara.	Psilomelane, pyrolusite and little braunite.	High iron, low silica and phosphorus.
Shimoga-Chital- durg-Tumkur.	'Lateritoid' and replacement deposit.	Mysore	Pyrolusite and wad.	High iron and silica, low phosphorus.
Visakhapatnam Srikakulam.	Supergene enrichment of primary bedded ore bodies (Kodurite series) Khondalite.	Andhra Pradesh : Srikakulam and Visakhapatnam	Psilomelane with subordinate braunite magnetite and mixture of jacobsite and hausmanite.	High iron, phosphorus and silica.
Udaipur Banswara.	Surfacial enrichment of bedded ore bodies associated with rock or Aravalli system.	Rajasthan : Banswara	Braunite, pyrolusite psilomelane.	High silica and phosphorus.
Panchmahal Baroda.	Supergene enrichment of primary manganese-bearing lodes	Gujarat	Polianite, pyrolusite, psilomelane and braunite.	High phosphorus.
Goa.	Local enrichment of manganese in laterites and surficial replacement.	South Goa	Psilomelane	Low phosphorus and iron.

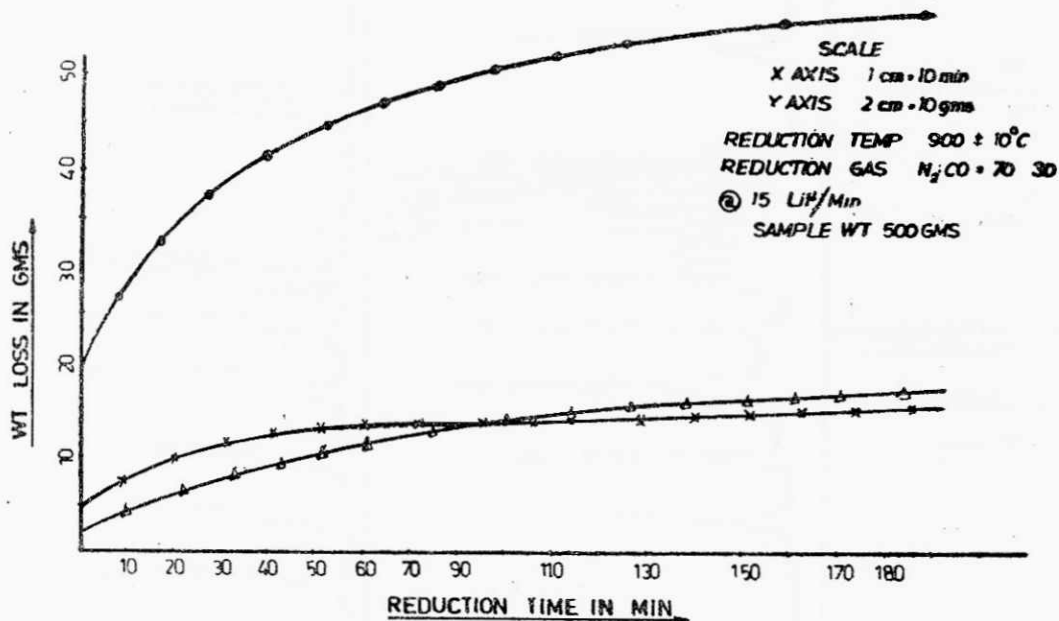


### ANNEXURE - II

The chemical analysis and reserves of the manganese ores from Madhya Pradesh - Maharashtra belt is given below :

Area	Assay %		Estimated reserves Million Tons
	Mn	Phosphorous	
Balaghat	46-50	0.08-0.30	6.50
Ukwa	50	0.08-0.15	12.50
Kandri	31-50	0.20-0.27	1.24
Tirodi	43-50	0.18-0.30	0.75
Chikla	35-48.5	0.10-0.24	4.71
Dongri Buzurg	32-49.5	0.20-0.26	6.20
Total ...			31.90

### ANNEXURE - IV



### REDUCIBILITY CURVES

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BGT Mn ORE

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PLAIN SINTER

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