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EXPERIMENTAL PRODUCTION OF PIG IRON FROM A SILICEOUS IRON ORE
(PERFORMED FOR MONPRE MINING CO., NOVEMBER 1958-APRIL 1959)

by

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MINERAL DRESSING AND PROCESS METALLURGY DIVISION

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ABSTRACT

Four different methods were investigated
for the extraction of iron from a low-grade siliceous
iron ore from the Port Arthur, Ontario area. The
methods used were (1) direct smelting of the ore in an
electric furnace; (2) magnetic concentration followed by
sintering of the concentrate and smelting of the sinter
in an electric furnace; (3) preheating and prereduction
of the ore in a rotary kiln, followed by smelting; and
(4) a two-stage smelting process. The last-named
method appeared to offer important advantages compared
to the first three.

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INTRODUCTION

In a program performed in the Mines Branch laboratories
in the spring of 1958, iron ore from the Monpre Mining Company Matawin
property in the Port Arthur Mining Division, N.W. Ontario, had been
treated by a process of fine grinding, magnetic concentration, sintering
of the magnetic concentrate, and smelting of the sinter in an electric
furnace to produce pig iron. The results of this investigation are
described in Internal Report MDT-58-24 dated August 7, 1958. Since the
results obtained in treating the ore by the above method were not as
favourable as had been hoped, the Company wished to investigate another
process for producing pig iron from the ore. In a letter dated
September 4, 1958, Mr. W. D. Paterson, Secretary-Treasurer of the
Monpre Mining Company, outlined the second experimental program which
the Company wished to have performed with the same ore in the Mines
Branch laboratories.

In the new process the ore would be passed through a
rotary kiln where it would be preheated, and prereduced to some extent

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before being fed to the smelting furnace. It was hoped that the kiln could be mounted above the electric smelting furnace so that the hot prereduced ore could be fed directly to the furnace, and that the savings in energy achieved by this technique would lead to a significantly cheaper process for extracting iron from this ore.

At a number of subsequent meetings between officials of the Company and senior officers of the Mineral Dressing and Process Metallurgy Division, decisions were made regarding the details of the experimental program. The rectangular shell of the furnace was replaced with a cylindrical shell, and the electrode arrangement changed from a three-in-line to a triangular configuration. Since the information would be of some value to both the Company and the Department of Mines and Technical Surveys, it was decided to try to ascertain whether or not the cylindrical shell with the triangular electrode configuration was more efficient than the rectangular shell with the three-in-line electrode arrangement. To make this comparison, a short test was performed in which every effort was made to duplicate the procedure used in one of the periods of the previous work (Period No. 5, described in Internal Report MDT-58-24). Matawin ore was ground to minus 100 mesh and magnetically concentrated. The concentrate was sintered to produce two types of product with $CaO:SiO_2$ ratios of 0.8:1 and 0.4:1 respectively, and these were smelted in the 250 KVA furnace to produce pig iron. After this test had been completed, another short test was performed in which unbeneficiated Matawin ore was smelted in the 250 KVA furnace to determine the requirements of electrical energy, flux, reductant, etc. for the production of pig iron from this ore by direct smelting. The above smelting tests are described in Part I of this report. For the sake of

clarity, the work described in Internal Report MDT-58-24 is referred to in the present report as Test No. 1, and the experiments outlined above, as Test No. 2. Subsequent experiments, described in Parts II and III of this report are designated as Tests 3 to 13.

Although it was hoped that the process of preheating and prereducing the ore in a kiln, followed by electric smelting, might be suitable for treating the low-grade Matawin ore, it was realized that such a process would probably work better on a higher grade ore. It was decided to have some high-grade iron ore available during the tests for blending with the low-grade Matawin ore. If the results of the tests indicated that the process worked better with a blend of ores than with the pure Matawin ore, consideration could then be given by the Company to the use of some such blend of ores if and when the process was to be put into commercial practice. Accordingly one carload of high-grade ore was obtained from Steep Rock Iron Mines Limited, Steep Rock, Ont.

Experiments on the reduction of the ore in a gas-fired rotary kiln were begun early in January. Two tests, (No. 3 and No. 5) were performed on the Matawin ore, and one test (No. 4) was done on the Steep Rock ore. During these tests the smelting furnace was not operated, but the hot products from the kiln were quenched in water and sampled for analysis to ascertain the amount of reduction which had occurred.

After some experience had been accumulated on the reduction of the ores in the kiln, the smelting furnace and kiln were operated simultaneously, the ore being preheated and prereduced to some extent in the kiln before it passed into the furnace. In tests No. 6 and No. 7 Matawin ore only was used; in test No. 8 the ore was

a 1:1 blend of Matawin ore and Steep Rock ore; and in test No. 9

Steep Rock ore alone was used. These two series of experiments, No. 3

to No. 9 inclusive, are described in Part II of this report.

The combined prereduction and smelting tests did not

yield as favourable results as had been hoped, mainly because the

metal produced contained too much silicon, and the amounts of electrical

energy, gas for the kiln, and limestone for flux were very high. Another

process for treating the Matawin ore was then tried, in which the ore

was smelted directly to produce pig iron with a high-silicon content, no

attempt being made to completely neutralize the highly siliceous slag.

The amount of limestone required was thus much less than in the earlier

tests. This high-silicon metal was reacted with high-grade ore in the

hope that the silicon in the metal would be oxidized by the hematite in

the ore, and that a low-silicon metal would result. Sufficient lime

was added in the second stage to flux the silica produced by the

oxidation of the silicon, and a highly basic slag was maintained. The

total lime required in this two-stage process was thus much less than

in the one-stage process, because it was necessary to maintain a basic

slag only in the second stage, when the ratio of slag to metal was

relatively low. Three tests were performed in which high-silicon

metal was produced, tests No. 10, 11 and 12. In test No. 13 the high-

silicon metal was reacted with high-grade Steep Rock ore to produce

low-silicon metal. This test was moderately successful, but it was

believed that better results would have been obtained if the furnace

had been equipped with a basic lining rather than the carbon lining

which was used. A number of small-scale tests then were done in the

60 KVA arc furnace using a basic lining. This series of tests, No. 10

to No. 13 inclusive, together with the experiments in the 60 KVA furnace,

are described in Part III of this report.

SUMMARY OF RESULTS

The more important conclusions from this investigation may be summarized as follows:

Part I. Direct Smelting of Sinter, and Matawin Ore

(a) Production of Sinter from Magnetic Concentrate

1. A concentrate produced from the Matawin ore by magnetic concentration, and containing about 41 percent Fe was readily sintered on a Dwight-Lloyd sintering machine.

2. The results were very similar to those obtained previously in sintering a similar grade of concentrate made from the same type of ore. A low-grade semi-anthracite coal, having a fixed carbon content of about 75 percent, was found to be a satisfactory fuel; the moisture content of the feed was found to be quite critical, the optimum value being between 10 and 11 percent; and the addition of limestone to the feed to increase the ratio of $CaO:SiO_2$ in the sinter, increased the amount of fuel required.

(b) Smelting of Sinter

No significant difference was observed in the smelting operation or results, which could be attributed to the change in furnace shape and electrode configuration. Any comparatively small differences were masked by the effects of different sinter bulk densities, different depths of dry top

and different percentages of the limestone burned
into the sinter.

(c) Smelting of Raw Matawin Ore

    1. Pig iron containing about 3.5 percent silicon was
produced by direct smelting of the raw Matawin ore
in the 250 KVA furnace, using a $CaO:SiO_2$ ratio in
the slag of about 1:1.

    2. The requirements of ore, limestone and electrical
energy per ton of metal obtained were about 8500 lb,
6900 lb, and 7700 kwh, respectively in this operation.
These values are very high, mainly because of the
low Fe and high SiO_2 contents of the Matawin ore.

    3. The data obtained in this operation provide a useful
basis for evaluating other methods of treating the
ore, such as sintering of a magnetic concentrate
followed by smelting; kiln treatment followed by
smelting; or a two-stage smelting process.

Part II. Prereduction and Smelting of Ores

    (a) Reduction of Matawin Ore and Steep Rock Ore, in a
Rotary Kiln

        1. Both Matawin and Steep Rock ores were reduced to
a considerable extent when mixed with coal and passed
through a rotary kiln having a maximum temperature
of about $1000^{\circ}C$.

        2. In the most successful of the experiments with the
Matawin ore the greatest reduction was about 17
percent; and an average of 11 percent reduction was

obtained over a period of about 36 hr.

3. The greatest reduction obtained with the Steep Rock ore was about 34 percent, and an average of about 24 percent reduction was obtained in a 42 hr test.

4. In each case the greatest reduction was obtained when the highest temperature in the kiln was kept just below the softening point of the ore; and when the longest possible hot zone was maintained in the kiln.

5. The composition of the kiln gases fluctuated very widely during these tests, and no correlation was found between the gas composition and the degree of reduction of the ore.

6. The amount of fuel used in heating the kiln during these experiments was very large, mainly because the kiln had only a relatively thin refractory lining ($4\frac{1}{2}$ in.) and the heat losses were very high.

(b) Combined Prereduction and Smelting of Matawin and Steep Rock Ores

1. When Matawin ore was preheated in the rotary kiln under oxidizing conditions, and the hot material was smelted, using a $CaO:SiO_2$ ratio in the slag of about 0.6:1, pig iron containing about 12 percent silicon, was produced. The amount of energy required in the electric furnace was about 13 percent less in this operation than when smelting the raw Matawin ore directly, after an approximate allowance was made for the differences in the slag:metal ratios and the metal compositions in the two tests.

However this saving in electrical energy was more
than offset by the large amount of gas required
in the rotary kiln. This was equivalent to about
4600 kwh per ton of metal produced, or about 60
percent of the energy used in the furnace when
smelting the raw ore directly. This large heat
requirement in the kiln was believed to be mainly
due to the high heat losses through the thin
refractory lining.

2. When Matawin ore was prereduced to the extent of
about 5.6 percent in the rotary kiln, and the hot
material was smelted using a $CaO:SiO_2$ ratio in the slag
of about 0.6:1, pig iron containing about 5.8
percent silicon was produced. The amount of
electrical energy required in the furnace was about
25 percent less in this operation than when smelting
raw Matawin ore directly, after an approximate
allowance was made for the differences in slag:metal
ratios and the metal compositions in the two tests.
Again this saving in electrical energy was more than
offset by the large amount of gas required in the
rotary kiln. This was equivalent to about 7300 kwh
per ton of metal produced or about 95 percent of
the energy used in the furnace when smelting the
raw ore directly. Again this high heat requirement
was believed to be due largely to the high heat
losses from the kiln.

3. When a 1:1 mixture of Matawin and Steep Rock ores
was prereduced in the rotary kiln and the hot material
was smelted using a base:acid ratio in the slag of
about 0.95:1, pig iron containing about 4.3 percent
silicon was produced. The amounts of electrical
energy and other materials such as fluxes were very
much less in this operation than when treating
Matawin ore alone, because of the higher iron content
of the Steep Rock ore.

4. A good grade of pig iron was made from the Steep Rock
ore by prereducing it in the rotary kiln and smelting
the hot prereduced material in the furnace. The
amounts of electrical energy and other materials used
per ton of metal were much less in this operation than
in any of the three previous tests, because of the
high iron content of this ore.

Part III. Production of Low-Silicon Iron

(a) Production of High-Silicon Iron from Matawin Ore

1. Pig iron containing up to 20 percent silicon was
readily produced from the Matawin ore, while operating
with a $CaO:SiO_2$ ratio in the slag of about 0.6:1.

2. When the silicon content of the metal was in the
range of about 16 to 20 percent its sulphur content
was less than 0.01 percent, and the FeO content of
the slag was only about 2 percent.

3. In all cases the amount of electrical energy
required was very high because of the high silicon

content of the metal produced. In two experiments
the amount of energy required, and also the weight
of electrodes used, were further increased by the
fact that the furnace was operated without a roof,
and the heat losses and the oxidation of electrodes
were quite severe.

4. When the "cold dry top" smelting technique was
employed, paper mill bark being used to increase the
porosity and decrease the bulk density of the charge,
the amount of energy used was only about one-half
and the amount of electrodes considerably less than
one-half of the corresponding amounts used in the
two former tests.

(b) Reaction of High-Silicon Iron with High-Grade Ore to
Produce Low-Silicon Iron

1. Metal containing less than 0.4 percent silicon was
obtained by reacting high-silicon metal with high-
grade iron ore, in both the 250 KVA and 60 KVA
furnaces.

2. The carbon and sulphur contents of the metals
obtained were quite low in both cases, due to the
low carbon and sulphur contents of the high-silicon
metals used.

3. A short period ($9\frac{1}{2}$ hr) of good operation was obtained
in the 250 KVA furnace, using a charge containing
75 lb of metal and 105 lb of ore. The results
indicated that if this charge had been used throughout
the test, and if the furnace had been equipped with
a basic lining rather than a carbon lining, this
period of good operation might have been extended

considerably.

4. The experiments performed in the 60 KVA furnace indicated that the silicon was oxidized quite rapidly and completely by the high-grade iron ore when an oxidizing slag and a basic-lined furnace were used. The analyses of the slags obtained in these experiments indicated that some magnesia (MgO) was removed from the lining.

5. The data obtained for the requirements of energy and materials in the production of pig iron by the two-stage smelting process, could not be compared accurately with the data obtained by other methods, because the period of good operation in the 250 KVA furnace was too short and large amounts of metal and slag remained in the furnace from previous operations.

GENERAL CONCLUSIONS

The reduction of the Matawin and Steep Rock ores in the kiln was only moderately successful, and it is believed that some of the difficulties resulted from the fact that the kiln was not designed specifically for this operation, but was used because it was the only one which was available. Heat losses were particularly high; the hot zone was too short; and the composition of the kiln gases fluctuated widely because of variations in burner operation, air leakages and imperfect pressure control. Recent experience in these laboratories has confirmed that, in operating kilns of this type with reducing atmospheres, the kiln gas composition is strongly affected by slight changes in pressure.

These cause changes in the amount of air leaking into the kiln.
Equipment for precise regulation of kiln gas pressures is being procured,
and this, it is anticipated, will result in considerable improvement in
these operations.

The results obtained in the combined kiln treatment and
smelting operation were not as favourable as had been hoped. In this
case the same difficulties were encountered in the operation of the kiln
as outlined above, and in addition other problems arose when trying to
operate the kiln and furnace together. For example, variations in the
amount and temperature of the gases passing through the duct from the
furnace to the kiln, affected the kiln operation and sometimes caused
blocking of the chute; also the hot discharge from the kiln often
accumulated to a considerable depth in the furnace and smelted very
slowly, even though large amounts of electrical energy were supplied to
the furnace. Throughout these tests the furnace roof deteriorated rather
rapidly because of the high temperature and the corrosive action of the
iron-bearing dust contained in the furnace gases. Electrode consumption
was excessively high due to oxidation by the air which was drawn into
the furnace around the electrodes. It appears that coupling a kiln and
furnace together in this manner for large-scale operation might involve
appreciable difficulties in design and operation. Nevertheless much
valuable information was obtained in these experiments, and it is
believed that using properly designed equipment, this process, or a
modification of it, could be operated more successfully. Modifications
which might be advantageous could include scrubbing of the furnace gases
rather than trying to conduct them directly into the kiln, the use of a
deeper furnace in an effort to obtain lower roof temperatures, and
perhaps the cooling of the kiln discharge to some extent before it was
fed to the furnace.

Metal having low contents of silicon and sulphur was
produced from the low-grade siliceous Matawin ore by the two-stage

smelting process, in both the 250 KVA and 60 KVA furnaces. The phosphorus contents were in the range of those of commercial grades of pig iron but the carbon contents were often intermediate between those of pig iron and steel. This however might not be a disadvantage, as metal of somewhat similar composition is being produced and marketed by a Canadian company at present (Quebec Iron and Titanium Corporation). Unfortunately the data obtained for energy and electrode requirements in this operation were excessively high because of the short duration of the experiment ($9\frac{1}{2}$ hr), and the carry-over of metal and slag from the previous tests. Nevertheless metal having attractive composition was obtained, and it is believed that this process, or some modification of it, could be operated more successfully in a plant specifically designed for this operation. In such a plant the high-silicon metal from the first stage could probably be transferred in the liquid state to a basic-lined furnace used in the second stage.

MATERIALS USED

Early in November 1958 three carloads of Matawin ore and one carload of Steep Rock ore were received at the Mines Branch. The Matawin ore was crushed to minus 3/8 in. and a head sample was obtained. A portion of this ore was then ground to minus 100 mesh and magnetically concentrated to yield a concentrate similar to the minus 100 mesh concentrate used in test No. 1. The Steep Rock ore did not require crushing, but was sampled and stored. In addition some Mesabi ore, containing 52 percent iron, was obtained from the Mineral Dressing Section of this Division and was smelted on one occasion while bringing the furnace up to equilibrium temperature, in order to conserve the supplies of the other ores for the main experiments. The analyses of these two ores, and of the magnetic concentrate, are shown in Table 1.

Table 1

Analyses of Ores, and Magnetic Concentrate
(percent)

	Matawin Ore	Magnetic Concentrate from Matawin Ore	Steep Rock Ore
Fe	27.06	41.49	60.90
SiO_2	45.72	34.20	4.34
Al_2O_3	4.13	3.34	1.96
MgO	2.70	1.55	0.16
CaO	1.86	0.63	0.27
Na_2O	1.24	--	0.06
K_2O	1.40	--	0.11
TiO_2	0.35	0.16	0.03
Mn	0.17	0.09	0.092
S	0.07	0.030	0.025
P	0.10	0.07	0.017
Loss on ignition	1.46	--	6.80
Insoluble	51.20	--	--
H_2O	--	7.12	--

A portion of the head sample of the Matawin ore was also submitted for semi-quantitative spectrographic analysis, to ascertain those elements present in minor amounts. The results of this analysis are shown in Table 2.

Table 2

Semi-Quantitative Spectrographic Analysis of Matawin Ore
(percent)

Ni*	Ti	Ba	Cu	Cr	V	B
.25	.15	.1	.09	.03	.009	.005

*doubtful

The limestone used as flux in most of the tests was
obtained from the Michigan Limestone Division, United States Steel
Corporation, Rogers City, Michigan. As was the case in the previous
work (test No. 1) this limestone was chosen because of its high purity
and favourable location with respect to the Monpre orebody. This
material was crushed to minus 3/8 in. and sampled. During some of the
tests some dolomite obtained from the Mineral Dressing Section and
crushed to minus 6 mesh,was used to supply part of the required flux.
The analyses of these two materials are shown in Table 3.

Table 3

Analyses of Limestone and Dolomite
(percent)

	Limestone	Dolomite
$CaCO_3$ (by diff.)	97.3	55.5
SiO_2	0.88	0.40
$MgCO_3$	1.08	41.12
Al_2O_3	0.63	-
Fe	0.36	-
S	0.10	0.083
P	0.01	-
Loss on ignition	43.16	-

The screen analyses of the Matawin ore, the Steep Rock ore and the limestone are shown in Table 4.

Table 4

Screen Analyses of Ores and Limestone

Particle Size	Matawin Ore	Steep Rock Ore	Limestone
+ $\frac{1}{4}$ in.	20.6	21.4	44.0
$-\frac{1}{4}$ in. + 4 mesh	21.5	14.2	13.7
-4 + 10 mesh	34.5	23.6	21.1
-10+ 28 "	11.2	23.6	7.1
-28+ 48 "	4.4	7.0	3.5
-48+ 65 "	1.5	2.5	1.5
-65+ 100 "	1.3	1.8	1.8
-100 "	5.0	5.9	7.3

As was the case in the previous work, (test No. 1) a low volatile coal obtained from Canmore Mines Ltd., Canmore, Alberta, was used as reductant. This material was obtained from a local fuel supplier as required. Two samples were taken at different times for analyses, the results of which are shown in Table 5.

Table 5

Analyses of Canmore Coal
(percent)

	Sample No. 1 (as rec'd)	Sample No. 2 (as rec'd)
Proximate analysis:		
Moisture	2.4	2.5
Ash	7.1	6.9
Volatile matter	14.6	17.3
Fixed carbon (by diff.)	75.9	73.3
Ultimate analysis:		
Sulphur	0.9	0.9

The screen sizes of the two coal samples are shown in Table 6.

Table 6

Screen Sizes of Canmore Coal Samples

	Sample No. 1	Sample No. 2
+ 1/2 in.	7.1	9.2
-1/2 + 3/8 in.	35.7	33.9
-3/8 + 1/4 in.	28.6	27.7
-1/4 + 1/8 in.	14.3	12.3
-1/8 + 1/16 in.	5.6	7.7
-1/16 in.	8.7	9.2

PART I. DIRECT SMELTING OF SINTER AND MATAWIN ORE

(a) Production of Sinter from Magnetic Concentrate

Experimental Work:

Sinter was made from the minus 100 mesh magnetic concentrate in the same manner as in the previous work (test No. 1). A Dwight-Lloyd sintering machine having a width of 12 in. and a length of travel over the windbox of 4 ft was used. Ignition was accomplished with a natural gas burner, and the windbox was connected to an exhaust system capable of producing a vacuum of about 20 in. of water. The ingredients of the charge were mixed in a muller-type mixer and fed onto the grates from an overhead hopper. A layer of sinter returns, approximately $\frac{1}{2}$ in. in size, was used on the grates to protect them against overheating, and to prevent excessive loss of fines into the exhaust system.

In all, about 12 tons of concentrate were sintered, in

two periods of operation. The analysis of this concentrate is shown in Table 1. Limestone from the U. S. Steel Co. shipment, crushed to minus 6 mesh, was added to the charge to adjust the ratio of lime to silica ($CaO:SiO_2$) in the sinter; and Canmore coal, crushed to minus 6 mesh, was used as fuel. The analyses of the limestone and coal are shown in Tables 3 and 5 respectively. The moisture content of the charge was found to be quite critical, the optimum value being about 10 to 11 percent.

The sinter obtained was crushed to minus 3 in. and screened on a double-deck vibrating screen fitted with 3/4 in. and 1/4 in. screens. The plus 3/4 in. material, and some of the minus 3/4 plus 1/4 in. material were stockpiled for furnace feed while the balance of the minus 3/4 plus 1/4 in. material was used for grate bed layer. The minus 1/4 in. material was recirculated by being blended into the charge in the mixer.

Results

The results of the sintering operation are summarized in Table 7. In this summary the operation has been divided into two periods, according to the grade of sinter produced. In Period I sinter with a lime:silica ($CaO:SiO_2$) ratio of 0.8:1 was made, while in Period II sinter having a $CaO:SiO_2$ ratio of 0.4:1 was produced.

Table 7

Summary of Sintering Operation

	Period I $\frac{CaO}{SiO_2} = \frac{0.8}{1.0}$	Period II $\frac{CaO}{SiO_2} = \frac{0.4}{1.0}$	Overall
Concentrate fed (dry basis)(lb)	18,400	5150	23,550
Coal fed (dry basis)(lb)	2345	500	2845
Limestone fed (dry basis)(lb)	10,131	1030	11,161
Sinter produced(lb)	21,703	4914	26,617
Average Fe content of sinter (%)	32.2	35.4	32.8
Recovery of Fe in sintering (%)	92	82	90
Materials used per ton (2000 lb) of sinter produced			
Concentrate (lb)	1696	2096	1769
Limestone (lb)	934	419	839
Coal (lb)	216	204	214
Natural gas (cu ft)	604	698	621

Sampling of the sinter for analysis was accomplished by removing one complete cake from the end of the machine each hour. These cakes were subsequently put through the same crushing and screening circuit as was the bulk of the sinter. Samples for analyses were prepared from each of the two main sizes, and the results of these analyses are shown in Table 8. The coarser sizes (+ 3/4 in.) are identified as Types I C and II C respectively, while the -3/4 + 1/4 in. sizes are identified as Types I F and II F.

Table 8

Analyses of Sinters
(percent)

	Type I C (+ 3/4 in.) $\frac{CaO}{SiO_2} = \frac{0.8}{1.0}$	Type I F (-3/4 + 1/4 in.) $\frac{CaO}{SiO_2} = \frac{0.8}{1.0}$	Type II C (+ 3/4 in.) $\frac{CaO}{SiO_2} = \frac{0.4}{1.0}$	Type II F (-3/4 + 1/4 in.) $\frac{CaO}{SiO_2} = \frac{0.4}{1.0}$
Fe	32.1	32.36	35.48	35.14
CaO	21.90	21.80	15.09	14.99
SiO_2	27.28	26.60	29.20	30.44
Al_2O_3	3.16	3.22	3.06	3.26
MgO	1.60	1.50	1.80	1.40
S	0.089	0.085	0.083	0.083
P	0.063	0.068	0.082	0.042
C	0.68	0.62	0.57	0.45

Strong porous sinter was obtained when the charge contained the optimum contents of fuel and moisture. Under these conditions the plus 3/4 in. material had a bulk density of about 45 lb per cu ft and the minus 3/4 plus 1/4 in. size a bulk density of about 58 lb per cu ft. These values are approximately the same as those obtained in previous work for similar sinters. During this operation an accurate record was made of the gas used in the ignition burner, but it was realized near the end of the test that the minimum amount of gas needed for ignition had not always been used. The heating value of the gas, as quoted by the gas company was 1030 BTU per cu ft.

Discussion

The amounts of concentrate, limestone and coal used in making a ton of each of the two types of sinter are shown in Table 7. These values are similar to those obtained in previous work (test No. 1). Again, more fuel was required when extra lime was incorporated to make a sinter with a $CaO:SiO_2$ ratio of 0.8:1 than when making sinter with a ratio of 0.4:1.

As was the case in previous work, some dust was lost in the exhaust system. This is believed to be the main reason for the relatively low recovery of Fe in the operation (90 percent).

(b) Smelting of Sinter

Experimental Work:

The cylindrical shell which was installed on the 250 KVA furnace was 8 ft in diameter, and was lined with fireclay brick next to the shell, with carbon brick surrounding the hearth. The hearth was formed of rammed carbon paste, and was 44 in. in diameter. The electrodes were 5 1/8 in. graphite, set in an equilateral triangular configuration,

with 19 in. between centres. The metal tap hole was located at the
hearth level, the slag hole $2\frac{1}{2}$ in. above this, and a drain hole was
installed $2\frac{1}{2}$ in. below it. Eleven thermocouples were installed in the
furnace lining to indicate the temperature at various locations.

After the carbon paste had been rammed into place, a
bed of coke was placed on the hearth and the power was turned on to
supply heat to bake the carbon lining. When this was completed, the
coke was not removed, but an estimate was made of the amount present,
and feeding was begun, using Mesabi ore without any additional reductant
for the first few hours until the excess coke was all used. No roof
was used on the furnace. Feeding of Mesabi ore, with Canmore coal as
reductant, and limestone as flux was continued for about 60 hr to bring
the furnace up to a stable operating temperature. This point was
judged to have been reached when the thermocouples in the furnace lining
ceased to show any further increase in temperature. During this
"Warm-up" period the furnace was operated at about 200 KWH input, and
the $CaO:SiO_2$ ratio in the slag was maintained at about 1.25:1. Small
amounts of sinter returns were fed on a few occasions. The optimum
tapping schedule was found to be 2 metal taps for each slag tap, and the
metal was taken on a preheated ladle and poured into pig moulds. The
slag was tapped into large conical shaped slag pots. Samples of metal
for analyses were taken from the flow in iron spoons, and slag samples
were usually obtained by dipping a rod into the pot after the slag had
had a few minutes to settle. Metal samples were prepared by drilling
when this was possible, but samples which were too hard to be drilled
were sometimes prepared by pulverizing. Slag samples were prepared
by pulverizing the material to minus 100 mesh, and cleaning with a
small magnet to remove any particles of metallic iron which might be

present.

After the furnace had reached operating temperature, feeding of Type I sinter was begun. The objective in the test was to reproduce as closely as possible the procedure used in Period No. 5 of test No. 1. Some limestone was added to increase the $CaO:SiO_2$ ratio in the slag. The energy input to the furnace was kept at about 250 kilowatts and a dry top 12 to 15 in. deep was maintained in the furnace. The metal produced contained about 3.9 percent carbon and about 2.4 percent silicon. The FeO content of the slag averaged about 2.2 percent.

The results obtained in test No. 1 had shown that the energy required to smelt sinter depended upon the depth of the dry top which was used, and also to some extent on whether most of the limestone required was burned into the sinter or added directly to the furnace. In the experiment just described about 80 percent of the total limestone had been burned into the sinter whereas in Period No. 5 of test No. 1 only about 60 percent had been introduced in this way. Accordingly, a new experiment was begun in which an appreciable portion of Type II sinter ($CaO:SiO_2 = 0.4:1$) was used in the charge. This meant that a large proportion of the total limestone required had to be added directly to the furnace, since the sinter contained less lime. Also a shallower dry top was maintained in the furnace. The energy input was kept at about 220 kilowatts, and very little change was noticed in the operation except that the silicon content of the metal increased to about 2.7 percent, the carbon content decreased to about 3.8 percent, and the tapping temperatures were somewhat higher than in the previous experiment.

The results obtained in the two periods when sinter
was smelted, are shown in Table 9.

Table 9

Results of Smelting of Sinter and of Matawin Ore (Test No. 2)

	Period I (Sinter)	Period II (Sinter)	Period III (Matawin Ore)
Duration (hr)	18.68	42.12	62.50
Charge			
Ore (1b)	——	——	16300
Sinter Type I "	9000	9300	——
Sinter Type II "	——	6204	——
Coal "	1350	2383	1950
Limestone "	1026	2538	13188
Products			
Metal (1b)	2930	5268	3814
Slag "	5538	9530	17036
Slag:metal ratio	1.89	1.81	4.47
Average metal analyses (%)			
C	3.90	3.83	3.23
Si	2.38	2.73	3.50
S	0.059	0.053	0.047
Average slag analyses (%)			
FeO	2.24	2.29	5.13
CaO	46.3	46.0	41.2
SiO_2	40.6	42.1	42.2
$CaO:SiO_2$ ratio	1.14	1.09	0.98
Energy			
Total energy input (kwh)	4442	9297	14732
Average energy input (kwh/hr)	238	221	236
Av. tapping temp. (°C) Slag	1488	1525	1476
Metal	1248	1350	1376
Recovery of Fe (%) In metal	94.5	94.5	79.6
In slag	3.3	3.3	15.2
Total	97.8	97.8	94.8
Requirements per ton (2000 1b) of metal produced			
Ore (1b)	——	——	8547
Sinter "	6143	5887	——
Coal "	921	905	1023
Limestone "	700	964	6916
Energy (kwh)	3032	3530	7725
Electrodes (1b)	42	46	47

To evaluate the overall operation, it is of interest to compare the requirements in the combined sintering and smelting operation, for the production of a ton of metal from the magnetic concentrates. These data for Periods I and II of test No. 2 are shown in Table 10 along with the corresponding data from Period No. 5 of test No. 1.

Table 10

Requirements per Short Ton of Metal for Production of Pig Iron from Magnetic Concentrates

	Test No. 2		Test No. 1
	Period 1	Period 2	Period 5
CaO:SiO$_2$ ratio	1.14	1.09	1.10
Concentrate (1b)	5209	5463	5166
Coal "	1584	1526	1540
Limestone "	3569	3106	3349
Energy (kwh)	3032	3530	3328
Percent of limestone burned into sinter	80	69	61
FeO in slag (%)	2.24	2.29	4.4
Si in metal "	2.38	2.73	2.51
Slag:metal ratio	1.89	1.81	1.97

Discussion

As mentioned earlier, the main objective in these smelting experiments using sinter was to determine whether the round shell with the triangular electrode configuration was more efficient than the rectangular shell with the electrodes in a three-in-line arrangement. Unfortunately it was not possible to reproduce the conditions used in the previous work as closely as had been hoped.

Because the sinter used in the present experiments did not have as long a period of storage as the sinter used in the previous work, it did not disintegrate as much and consequently had a lower bulk density and greater porosity at the time it was fed to the furnace. Also it was difficult to maintain exactly the same depth of dry top in the furnace as had been used in the previous work, and it was found that the amount of limestone which had to be added to the furnace to maintain the $CaO:SiO_2$ ratio in the slag at the desired level was not as large as had been expected, and consequently the percentage of the limestone burned into the sinter was higher than was intended, particularly in Period I. For these reasons no firm conclusion can be drawn as to whether the round shell was more efficient than the rectangular shell, because the energy required per ton of metal produced was lower in Period I, and higher in Period 2 than in Period 5 of test No. 1, but it would appear that there was little, if any, difference between the two.

(c) Smelting of Raw Matawin Ore

Experimental Work:

When the smelting of sinter was completed, the furnace was shut down for a few hours and the roof was installed. This roof was equipped with several radiamatic heads designed to indicate the temperature of the molten bath, and also of the roof. Smelting of the unbeneficiated Matawin ore was then begun using a charge containing about 80 lb of limestone and 10 lb of coal per 100 lb of ore. Adjustments were made in the proportions of coal and limestone from time to time in order to keep the FeO in the slag as low as possible and the $CaO:SiO_2$ ratio at about 1:1. Because of the low grade of the ore (27% Fe) the slag:metal ratio was very high and a suitable tapping

schedule was found to be 3 slag taps for each metal tap. The furnace was operated at about 235 kilowatts with a phase to phase voltage of 90 volts. The silicon content of the metal increased to about 3.5 percent and the carbon content decreased to about 3.2 percent. The FeO content of the slag remained rather high, about 5 percent, in spite of the high silicon content of the metal. The results of this smelting operation are shown in Table 9.

Discussion

The most significant features of this experiment were the high requirements of limestone and electrical energy per ton of metal obtained. These results were of course not unexpected in view of the low Fe and high SiO_2 contents of the ore. The $CaO:SiO_2$ ratio in the slag was kept at about 1:1, because previous work had indicated that at lower $CaO:SiO_2$ ratios an excessive amount of silica was reduced and the silicon content of the metal was too high. Even at a $CaO:SiO_2$ ratio of 1:1 in this experiment the silicon content of the metal was about 3.5 percent and the FeO content of the slag was about 5 percent. This high FeO content in the slag, and the high slag:metal ratio (4.5:1) resulted in a rather high loss of iron in the slag. If a larger proportion of the iron had been recovered in the metal instead of being lost in the slag the requirements of ore, limestone and electrical energy, etc. per ton of metal produced would have been considerably lower. However to lower the FeO losses in the slag would have required a much more basic slag which would have increased the amount of limestone required still further. Nevertheless the data obtained provide a useful basis for evaluating other methods of treating the ore, such as concentration followed by sintering and smelting, or kiln treatment followed by smelting.

PART II. PREREDUCTION AND SMELTING OF ORES

(a) Reduction of Matawin Ore and Steep Rock Ore in a Rotary Kiln

Experimental Work:

The kiln used for the reduction experiments was 20 ft long, 2 ft inside diameter, lined with $4\frac{1}{2}$ in. of acid-resisting fireclay brick, and heated by means of a combination gas-and-oil burner mounted in a brick-lined head at the discharge end. A graphite ring about 3 in. wide was installed on the face of the discharge head to provide a gas seal between the head and the rotating portion. The kiln was set with a slope of 5 in. in 20 ft, and rotated by means of a variable speed drive. At the feed end an additional tubular section, 6 ft long, was installed to serve as a preheating zone for the feed. This section was made of light gauge steel, with no refractory lining, but with lifter plates installed on the inside which lifted the feed and permitted it to drop back through the hot gases as they passed outward toward the feed end of the kiln. At the upper end of the kiln an automatic feeder was installed which permitted feeding of the charge at a controlled rate while maintaining a reasonably good gas seal. The exhaust gases were drawn off at the feed end by means of an exhaust fan operated in such a manner as to maintain a slight vacuum in the kiln at all times. The kiln is illustrated diagramatically in Figure 1.
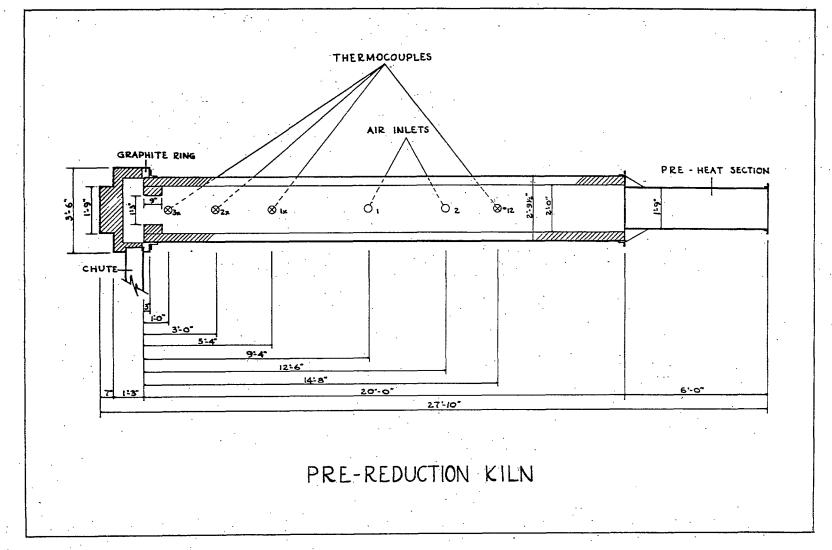
Figure 1. - Kiln Used in Prereduction Experiments.

THERMOCOUPLES

AIR INLETS

PRE - HEAT SECTION

GRAPHITE RING

CHUTE

PRE-REDUCTION KILN

A number of thermocouples were installed at various places along the kiln. These thermocouples are designated as Numbers 1-X, 2-X, 3-X, 12 and 13, and their locations are shown in Figure 1. At a later stage in the program two air inlets were installed part way up the kiln to admit extra air. These inlets were 1 in. diameter pipes inserted to the centre of the kiln, and their locations are also shown in Figure 1. Samples of the exhaust gases were withdrawn for analysis, from the feed end of the kiln.

The charge of ore, limestone and coal was mixed by hand and fed into the hopper of the automatic feeder. After passing through the kiln the hot material was discharged down a vertical pipe, the bottom of which was submerged in a tank of water, so that the hot material was quenched to prevent re-oxidation of the iron. Samples of this discharge were removed from the tank (usually once per hour), dried, and pulverized for analysis. In all, three tests were performed, the first on the Matawin ore, the second on the Steep Rock ore, and the third on the Matawin ore. These are designated as tests 3, 4 and 5, respectively.

Test No. 3   The charge used at the start of this test contained about 80 lb of limestone per 100 lb of ore, which was calculated to yield a product with an average $CaO:SiO_2$ ratio of 1:1. Canmore coal was used as the reducing agent. At the start of the test a screw feeder was used to feed the charge into the kiln, but this was soon found to be unsatisfactory for handling this relatively coarse material. A new feeder was then designed and built, which, although not entirely satisfactory, did feed the charge into the kiln at a fairly uniform rate while maintaining a reasonable gas seal.

During this operation natural gas was used in the burner and the temperature of the kiln was controlled by means of thermocouple No. 1-X. It was found that, to keep this temperature at about 900°C, it was necessary to operate the burner with an oxidizing flame, and consequently the CO content of the kiln gases was not as high as was desired. Observations through a sighting port in the head of the kiln showed that the flame of the burner was relatively short, and that the zone of maximum temperature was also very short and was located close to the discharge end.

Samples of the discharge were analyzed for their contents of metallic, ferrous, ferric and total iron, and the degree of reduction was calculated from the data obtained. The amount of reduction obtained in this test averaged about 4 percent, i.e. about 4 percent of the oxygen associated with iron in the ore was removed. The best reduction obtained was about 7 percent. After about 20 hr of operation the proportion of limestone in the charge was reduced from 80 lb to 5 lb per 100 lb of ore. This charge was used for only a short period of time as the head of the kiln became plugged with fused material about 2 hr later and the test was stopped.

The results obtained are shown in Table 11.

Table 11

Results of Kiln Reduction Tests on Matawin and Steep Rock Ores

	Test No. 3 (Matawin Ore)			Test No. 4 (Steep Rock Ore)			Test No. 5 (Matawin Ore)		
Duration (hr)	22.2			42.6			36.8		
Materials fed:									
Ore (1b)	3500			8500			7800		
Limestone "	2275			635			3120		
Coal "	525			3160			1950		
Feed rate (1b charge per hr)	288			288			349		
Rotation of kiln (RPM) (Av.)	0.8			0.8			1.6		
Natural gas used:									
Total (cu ft)	19590			--			21210[1]		
Cu ft per hr	882			--			576[1]		
Product analyses: (%)									
Metallic iron (Av.)	--			--			--		
Ferrous " "	7.2			34.7			14.2		
Ferric " "	10.4			14.5			9.2		
Total " "	17.6			49.3			23.4		
Percent reduction (Av.)	3.9			23.6			11.3		
" " (Best)	7.1			33.7			16.9		
Kiln temperatures (°C)	Min.	Av.	Max.	Min.	Av.	Max.	Min.	Av.	Max.
1-X	774	871	955						
3-X		(1061)[2]		926[3]	988[3]	1065[3]	954	991	1052
12	410	517	620	375	551	685	670	730	770
13	240	262	300	90	140	195	230	248	275
Stack gas analyses (Av.)(%)									
CO	4.31			1.94			0.07		
CO_2	10.37			8.02			7.85		
O_2	2.71			8.52			9.43		

(1) Some oil was used with the gas during part of this test. The amount of oil used was not measured.

(2) This value was estimated from several pairs of readings of the thermocouples 1-X and 3-X. Facilities were not available to obtain simultaneously a continuous record of each temperature.

(3) These temperatures are only approximate values, as other thermocouple locations were being tried on a few occasions during this test.

Test No. 4

The results obtained in test 3 suggested that better reduction might be obtained if the charge could be kept at a higher average temperature for a longer period of time, while avoiding localized overheating which caused the charge to fuse. In an attempt to increase the average retention time of the charge in the kiln, a brick retaining ring or "lip" was installed at the discharge end so that a minimum depth of $4\frac{1}{2}$ in. of charge was maintained at this end of the kiln. To increase the length of the hot zone in the kiln, a longer flame was obtained from the burner by using fuel oil instead of gas. Because the observations during test 3 had indicated that the maximum temperature in the kiln occurred near the discharge end, thermocouple No. 3-X, which was located one foot from the end, was used for controlling the temperature instead of No. 1-X.

The charge used in this test contained Steep Rock ore, limestone and Canmore coal. Because of the higher grade of ore, the proportion of limestone in the charge was much lower, and that of coal much higher, than was the case with the Matawin ore. The temperature at the discharge end of the kiln (No. 3-X) was kept just below 1000°C, as it was found that the charge began to soften and fuse when the temperature rose just above 1000°C. The use of oil instead of natural gas in the burner resulted in a longer hot zone in the kiln. As in the previous test, it was found that an excess of air had to be used in the burner to keep the kiln temperature up at the desired level; also the composition of the stack gases fluctuated considerably and it was not possible to maintain as high a ratio of $CO:CO_2$ as had been hoped. The amount of reduction obtained in this test was, on the average, much greater than in test 3, reaching values as high as 34 percent, but

fluctuated considerably, and no sustained period of good operation was obtained. The results are shown in Table 11.

Test No. 5

Since considerable experience had been gained in the operation of the kiln in tests 3 and 4, and because the modifications made at the end of test 3 had resulted in considerable improvement, another test was performed using the Matawin ore to try to get better reduction of this material. Before the test was begun two air inlets were installed at 9 ft 4 in. and 12 ft 6 in. respectively from the discharge end. These inlets are indicated in Figure 1. It was hoped that the length of the hot zone could be extended by this means. The charge used contained Matawin ore, limestone and coal, but the proportion of limestone used was somewhat less than was the case in test 3. Again the maximum temperature, as measured by thermocouple No. 3-X, was kept just under $1000^{o}C$ to avoid fusion of the charge. During most of the test both oil and gas were burned simultaneously. The introduction of the extra air at some distance from the burner appeared to help in lengthening the hot zone, as the temperature of thermocouple No. 12, located 14 ft 8 in. from the discharge end, was considerably higher than in either of the previous tests. The composition of the stack gases again varied widely, and the ratio of $CO:CO_2$ was on the average very low. The degree of reduction however was considerably higher than in test 3, reaching 17 percent on one occasion.

The results are shown in Table 11.

Discussion

A significant amount of reduction was obtained in both the Matawin and Steep Rock ores in these kiln experiments, but in neither case was a sustained period of good operation obtained. It was

believed that if the charge could have been kept at the optimum temperature for a longer period of time, and if a strongly reducing atmosphere could have been maintained in the kiln, much better results would have been obtained. The length of the hot zone was extended considerably by burning oil, which gave a much longer flame than that obtained by the use of gas; also the provision of the two extra air inlets at some distance from the burner resulted in an increase in temperature in these regions and helped to extend the length of the hot zone. Unfortunately this particular kiln had a very thin lining ($4\frac{1}{2}$ in.) and the loss of heat by radiation, convection, etc. was very high. As a result the burner usually had to be operated at maximum capacity, with a fairly oxidizing flame, to maintain the desired temperature, and when attempts were made to operate with a reducing flame the amount of heat obtained was insufficient to maintain the desired temperature.

The composition of the kiln gases fluctuated very widely in all experiments, and a high ratio of $CO:CO_2$ was seldom obtained. Variations in the burner adjustments, and air leaks around the head of the kiln, were two factors which contributed to these fluctuations. Also changes in the barometric pressure of the air in the furnace room appeared to affect the kiln operation. The kiln gases were expelled to the stack by a fan operated so as to keep the pressure in the kiln slightly lower than that in the room, the difference in pressure being usually about 0.2 in. of water. With such a small difference in pressure, a slight change in the pressure of the air in the room made an appreciable difference in the amount of air which was drawn into the kiln by leakage.

After test No. 3 was discontinued because the kiln had become plugged with fused material, a number of samples were removed

from the region where the fusion had occurred. These samples, and also
a sample of the untreated Matawin ore were examined by the Physical
and Crystal Chemistry Section of this Division, using microscope
and X-ray diffraction techniques; also the softening temperatures of
all the samples were determined. The softening points of the samples
removed from the kiln were found to range from 1210 to $1250^\circ C$, and
that of the untreated Matawin ore was found to be $1280^\circ C$. These data
indicated that the maximum temperature at the discharge end of the kiln
must have been in excess of $1210^\circ C$ when the trouble occurred. The
temperature of thermocouple No. 1-X at that time was $960^\circ C$, which
confirmed the visual observation made during the test that the maximum
temperature in the kiln was very close to the discharge end, rather
than at the location of thermocouple No. 1-X.

A pyroxene-like material was found to be present as a
minor constituent in the untreated ore, and as a major constituent
in several of the samples from the kiln. It was believed that this
material would melt at a temperature slightly above $1200^\circ C$. Also some
melilite was found in a number of the samples, usually occurring with an
excess of lime (CaO) and it was believed that this material was probably
formed by the reaction of CaO with the pyroxene. The samples in which
the melilite was found had higher softening points than those in which
the pyroxene was the major constituent. It seems probable that the
reduction in the proportion of limestone in the charge, from 80 lb to
5 lb per 100 lb of ore, which was made about 2 hr before the shutdown,
may have contributed to the plugging by permitting the formation of some
material containing a lower proportion of lime. In the melilite solid
solution series the melting point decreases as the percentage of CaO
decreases. The details of this investigation are given in the report

MDT-59-30, by Arnout Jongejan and John F. Rowland of this Division.

It was expected that greater reduction would have been obtained at higher temperatures and at higher ratios of $CO:CO_2$ in the kiln gases. The degree of reduction of each sample was plotted against the kiln temperature, and also against the $CO:CO_2$ ratio of the gases. The points were found to be very widely scattered, particularly in the case of the $CO:CO_2$ ratio. The five samples having the best reduction, and the five samples having the poorest reduction were then chosen from each of the tests, and compared with the corresponding temperatures and gas analyses. This comparison indicated that in each test the five best samples were obtained at a significantly higher average temperature than the five poorest samples. No correlation was found between the degree of reduction and the gas analyses. This is believed to have been due to the wide fluctuations which occurred in the kiln gas composition in each of the tests.

(b) Combined Prereduction and Smelting of Matawin Ore and
      Steep Rock Ore

Experimental Work:

For this operation the kiln and the furnace were coupled together to permit the hot solid material from the kiln to be discharged directly into the furnace. A brick-lined chute was installed between the bottom of the kiln head and the centre of the furnace roof so that the material dropped into the furnace in the centre of the triangle formed by the electrodes. In addition another brick-lined duct was installed, which joined the side of the kiln head and a hole in the furnace roof near the periphery. This duct permitted some of the gases from the furnace to by-pass the feed chute, thus preventing the gas velocity in the feed chute from becoming so high as to carry an

excessive amount of fine material back up the chute. A sampling device
was installed in the discharge chute, by means of which samples of the
kiln product could be withdrawn without letting air into the chute.
Three charging ports near the periphery of the furnace roof were fitted
with small hoppers, each having a ball-cock type stopper in the bottom,
which could be operated by hand. Through these ports additional feed
could be added to the furnace when desired, while maintaining a good
gas seal.

Four tests were performed in which the kiln and furnace
were operated together; these are designated as No. 6, 7, 8 and 9
respectively.

Test No. 6

In this test Matawin ore was treated, and the kiln was
used only to preheat the ore, without doing any prereduction. Gas was
used as fuel in the burner, with an oxidizing flame, and the coal or
coke required for reduction was added directly to the furnace. The
furnace gases were allowed to escape to the atmosphere, instead of being
taken through the kiln. The temperature at the discharge end of the
kiln was kept at about $950^{o}C$, although some fluctuations did occur and
the discharge chute became plugged on one occasion. Analyses of the
samples of kiln product taken from the chute showed that the ore was
being oxidized to some extent in the kiln.

The main difficulty encountered in this test was in
trying to match the throughput of the kiln with that of the furnace.
A feed rate of about 225 lb per hr was used in the kiln in the early
stages, and the furnace was operated at 90 volts phase to phase, with a
power input of about 200 to 225 kilowatts. Under these conditions a rather
unstable furnace operation resulted. At times the arcs at the electrode tips

were quite long, and a large amount of heat was radiated onto the roof, while at other times one or more of the electrodes became deeply immersed in the slag. A lower voltage (78 volts) was tried but the electrodes dipped very deeply into the slag; the character of the smelting changed from arc resistance to slag resistance, and a very thick bed of unsmelted material accumulated in the furnace. At about this time the furnace bottom began to cool down, and a salamander developed. The voltage was again increased to 90 volts, the rate of feed to the kiln was decreased for a time, and the power input to the furnace increased to try to get the two units to operate in balance. Toward the end of the test the feed rate to the kiln was again increased, the furnace voltage was increased to 130 volts and the power input was decreased to about 150 kilowatts. The furnace roof remained very hot throughout the test, and considerable erosion occurred at the point where the feed chute was joined to the roof.

The data and results which pertain to the kiln operation during this test are shown in Table 12. Those which pertain to the furnace operation are given in Table 13, and the materials and energy used per ton of metal obtained in Table 14.

Table 12

Data and Results for Kiln Operation during Tests 6, 7, 8 and 9

	Test No. 6 Matawin Ore	Test No. 7 Matawin Ore	Test No. 8 Mixed Matawin and Steep Rock Ores (50:50)	Test No. 9 Steep Rock Ore
Duration (hr)	63.0	135.1	72.0	24.0
Materials fed:				
Ore (Matawin) (1b)	8961	24970	9000	---
Ore (Steep Rock) "	180	---	9000	7000
Limestone "	3609	9988	4623	560
Dolomite "	---	2694	1800	---
Coal and coke "	1638	4593	4326	2013
Feed rate (1b charge per hr)	228	313	399	399
Rotation of kiln (RPM) (Av.)	1.1	1.3	1.6	1.8
Natural gas used (cu ft)	32596	74699	15930	5400
" " " (cu ft per hr)	517	533	221	225
Product Analyses (%)				
Metallic iron (Av.)	---	0.1	---	2.0
Ferrous " "	6.2	13.6	14.8	37.6
Ferric " "	22.6	15.9	20.4	10.0
Total " "	28.8	29.6	35.2	49.6
Percent Reduction (Av)	-3.81*	5.67	9.20	29.31
" " (Best)	-3.04	13.98	16.58	39.27

Kiln temperatures (°C)	Min.	Av.	Max.	Min.	Av.	Max.	Min.	Av.	Max.	Min.	Av.	Max.
3-X	760	947	1065	885	976	1024	815	938	1024	927	985	1094
12	425	637	735	510	620	730	425	567	675	510	664	1020
13	160	202	265	125	164	220	100	150	175	115	156	205

*
Oxidation

Table 13

Data and Results for Furnace Operation during Tests 6, 7, 8 and 9

	Test No. 6 Matawin Ore	Test No. 7 Matawin Ore	Test No. 8 Mixed Matawin and Steep Rock Ores (50:50)	Test No. 9 Steep Rock Ore
Duration (hr)	66.5	148.5	86.4	26.1
Materials fed directly to furnace:				
Ore (Matawin) (1b)	2940	100	300	--
Ore (Steep Rock) "	--	957	500	100
Ore (Steep Rock) " (Test 4 product)	--	---	1200	--
Limestone "	1307	90	220	--
Dolomite "	--	10	60	--
Coal and coke "	678	18	200	--
Products:				
Metal (1b)	4240	6162	8123	4254
Slag "	7976	23410	9004	1378
Slag:metal ratio	1.88	3.80	1.11	0.32
Average metal analyses (%)				
C	1.44	2.37	2.64	2.58
Si	12.07	5.80	4.26	1.61
S	0.018	0.074	0.087	0.148
Average slag analyses (%)				
FeO	2.35	8.81	7.98	4.0
CaO	31.6	28.1	32.8	34.9
SiO_2	51.6	47.7	41.3	39.4
MgO	--	--	6.14	--
Al_2O_3	--	--	--	12.9
$CaO:SiO_2$ ratio	0.61	0.59	0.94[1]	0.89
Energy:				
Total energy input (kwh)	12727	16447	16205	5135
Av. " " (kwh/hr)	191	111	188	197
Average tapping temp. (°C) Slag	1397	1345	1423	1393
Metal	1268	1276	1273	1283
Recovery of Fe: In metal	95.7	70.9[2]	87.3[3]	93.7
In slag	3.8	22.1	6.5	1.0
Total	99.5	93.0	93.8	94.7

(1) In this case the base:acid ratio was calculated as $\dfrac{CaO + MgO}{SiO_2}$ instead of $\dfrac{CaO}{SiO_2}$

(2) In calculating the recovery of Fe in this test allowance was made for 1037 lb of dust and unsmelted feed which was cleaned up at the end of the test.

(3) In calculating the recovery of Fe in this test, allowance was made for 1163 lb of dust and unsmelted feed which was cleaned up at the end of the test.

Test No. 7

In this test Matawin ore was prereduced in the kiln before it was discharged into the furnace to be smelted. To obtain the reducing atmosphere in the kiln coal was mixed with the charge before it was fed and the burner of the kiln was operated with as little excess air as possible. Natural gas was used as fuel in the burner. The furnace gases were drawn into the kiln through the feed chute and the extra gas chute, by maintaining a slightly lower pressure in the kiln than in the furnace room. This slight pressure difference was controlled mainly by adjusting the speed of the exhaust fan.

Considerable difficulty was encountered in this test in trying to achieve several objectives simultaneously, i.e. maintain in the kiln a strongly reducing atmosphere, a long hot zone, and a stable temperature just below the softening point of the ore; match the throughput of the kiln with that of the furnace; and obtain from the furnace metal and slag of the desired composition. It was found that if the pressure difference between the kiln and the furnace room was too large, an excessive amount of air was drawn into the furnace around the electrodes, causing excessive electrode consumption. Also, under these conditions, much of the carbon monoxide in the furnace gas was burned in the chutes, causing the temperature in the chutes to increase to the point where dust and ore began to fuse and stick. The discharge chute became plugged on several occasions during this test. In order to keep the temperature at the discharge end of the kiln close to $1000^{\circ}C$, it was often found necessary to admit extra air by removing the small clean-out door which was situated directly below the burner. When this door was removed the atmosphere in the kiln became oxidizing instead of reducing. Samples of the kiln gases from the feed end, and from the discharge end,

were taken for analyses each hour. It was found that the composition of these gases varied very widely throughout the test. Samples of the kiln product were removed from the discharge chute each hour and analyzed. It was found that the degree of reduction of the ore in the kiln varied widely throughout the test, but values as high as 14 percent were obtained.

In the early part of the test the power input to the furnace was kept at about 150 KW and the feed rate to the kiln at about 300 lb per hour. Under these conditions the furnace roof became quite hot, and the furnace appeared to be underfed. Consequently the power input to the furnace was reduced to about 100 KW and kept at that level for the remainder of the test. The metal obtained in the early stages contained about 3 percent carbon and 3 percent silicon, and the slag contained about 4 percent FeO, but as the test progressed the carbon content of the metal decreased, the silicon content increased, and the FeO content of the slag increased to values as high as 16 percent. The charge contained 40 lb of limestone per 100 lb of ore, and in the early stages the $CaO:SiO_2$ ratio in the slag was about 0.5:1, but as the amount of silicon reduced from SiO_2 increased, the $CaO:SiO_2$ ratio increased to values as high as 1:1. As the $CaO:SiO_2$ ratio increased the sulphur content of the metal decreased and the FeO content of the slag decreased. These effects are in agreement with those found in the smelting of the sintered magnetic concentrates in test No. 1. However after about 6 days of operation it became evident that even with a fairly high $CaO:SiO_2$ ratio, e.g. 0.8:1 or 0.9:1, the silicon content of the metal was too high and the carbon content too low, and the test was discontinued. The operating data and the results of this test are given in Tables 12, 13 and 14.

During this test an attempt was made to measure the average retention time of the charge in the kiln by the radioactive tracer technique. The tracer used for the first test was Ag-110, four millicuries of which were added as silver nitrate solution by sprinkling it over successive layers of mixed charge before it was fed to the kiln. Samples were then taken of the discharge from the kiln, and also the slag tapped from the furnace, for radioactivity measurements.

The average retention time in the kiln was found to be about 4.5 hr, and that in the furnace about 5.8 hr. The latter figure however is strongly influenced by the tapping schedule and the amount of material contained in the furnace during this particular interval.

The results suggested that all the constituents of the charge were not moving through the kiln at the same rate. Accordingly two more tests were performed later, during Furnace Test No. 8 in which tracers were added separately to the coal, and to the iron ore. These tests showed that the iron ore was moving through the kiln faster than the coal.

These tests are fully described in Mines Branch Investigation Report IR 59-25 "Measurement of Kiln and Furnace Residence Times by Means of Radioactive Tracers" by G. G. Eichholz.

Test No. 8

The charge used during this test contained one-half Matawin ore and one-half Steep Rock ore and, as in test No. 7, the material was prereduced in the kiln before being discharged into the smelting furnace. The kiln was operated in the same manner as in test No. 7, i.e. the reducing atmosphere was obtained by mixing coal with the charge, and the burner was operated with as little excess air as possible using natural gas as fuel. About 1200 lb of Steep Rock ore, which had been partially

reduced in test No. 4, were smelted in the furnace to increase its temperature before feeding of the kiln was begun.

The same operating difficulties were encountered in this test as in test No. 7. The difference in pressure between the kiln and the furnace room was found to be quite critical and difficult to stabilize; the electrodes were severely oxidized by air drawn into the furnace; combustion of furnace gas often occurred in the discharge chute causing sticking and blocking due to overheating of the charge; the amount of air admitted at the head of the kiln often had to be varied to maintain the desired kiln temperature, and wide fluctuations occurred in the composition of the kiln gases. Consequently the amount of reduction obtained also varied quite widely, but values up to 16 percent were obtained.

The rate of feed to the kiln during this test was about 400 lb per hour, and the furnace was operated with an energy input of about 200 KW. The volume of gas produced in the furnace was much higher than in the previous test, due to the higher iron oxide content of the feed, and both the feed chute and the extra gas chute became plugged much more frequently than in test No. 7. The plugging and subsequent cleaning of the feed chute disrupted the smooth flow of feed and probably contributed to the erratic results obtained in the furnace. The FeO content of the slag varied very widely, ranging from 2 to 20 percent, and the base:acid ratio also varied widely. In this test some of the limestone was replaced with dolomite in the hope that the mixture might be superior to limestone alone, consequently the base:acid ratio in this case was calculated as $\frac{CaO + MgO}{SiO_2}$. The amount of limestone used in the charge was adjusted several times in an attempt to maintain a constant base:acid ratio, but in spite of this

the ratio fluctuated from about 0.7:1 to about 1.2:1. The average carbon content of the metal was a little higher and the silicon content a little lower than in the previous test. As in test No. 7 the temperature of the furnace roof remained very high and severe erosion occurred where the feed chute was joined to the furnace roof. At the end of the operation several holes in this area had to be patched in order to continue with test No. 9. The data and results are shown in Tables 12, 13 and 14.

Test No. 9

The charge used in test No. 9 contained only Steep Rock ore, limestone and coal and, as in the two previous tests, the ore was prereduced as much as possible in the kiln before it passed into the smelting furnace. The same operating difficulties were experienced in this test as in tests No. 7 and 8, but because of the higher grade of the Steep Rock ore, and also because of experience gained in the two previous tests, the degree of reduction was much higher. Because of the large volume of gas produced in the furnace the desired temperature in the kiln was maintained much more easily, without having to admit as much air around the head of the kiln. A much longer hot zone was obtained in the kiln which helped to promote the reduction of the ore. The degree of reduction averaged 30 percent, and values as high as 37 percent were obtained, with as much as 7 percent metallic iron.

The feed rate to the kiln was about 400 lb per hr and the furnace was operated at about 200 KW. The silicon content of the metal was significantly lower in this test than in the previous three tests, having an average value of 1.6 percent, but the carbon content was also rather low (2.5 percent). The FeO content and the $CaO:SiO_2$ ratio of the slag varied rather widely. The furnace roof remained

hot throughout the test, erosion at the feed chute became progressively worse and, after about 24 hr of operation, the test was discontinued. The data and the results are shown in Tables 12, 13 and 14.

Table 14

Materials and Energy Used per Ton of Metal Produced in Tests 6, 7, 8 and 9

	Test No. 6 Matawin Ore (preheating only)	Test No. 7 Matawin Ore (prereducing)	Test No. 8 Mixed Matawin and Steep Rock Ores (50:50) (prereducing)	Test No. 9 Steep Rock Ore (prereducing)
Matawin ore (lb)	5614	8137	2287	---
Steep Rock ore (lb)	85	311	2336	3338
Steep Rock calcined ore (Test No. 4) (lb)	---	---	295	---
Limestone (lb)	2319	3271	1190	263
Dolomite (lb)	---	878	457	---
Coal and coke (lb)	1092	1497	1113	946
Energy (kwh)	6003	5338	3990	2414
Electrodes (lb)	151	197	100	76
Natural gas (cu ft)	15375	24245	3917	2539

Discussion

The results obtained in this series of experiments were not as favourable as had been hoped for. In no case was metal of the desired composition obtained, and in all cases the amounts of electrical energy and electrodes required were much greater than was expected. It had been anticipated that a large saving in the electrical energy required for smelting would result when the ore was preheated and prereduced in the kiln before it was fed to the furnace. The data given in Tables 9 and 14 show that in the case of the Matawin ore the amount of energy required did decrease from about 7700 kwh for the smelting of the raw ore to about 6000 kwh for the smelting of the preheated ore, and to about 5340 kwh when the ore was prereduced to about 5.5 percent.

The metal produced during these tests (No. 6 and No. 7) contained 12 percent and 5.8 percent silicon respectively while that produced when smelting the raw Matawin ore (Test No. 2, Period 3) contained about 3.5 percent silicon. Also the slag:metal ratio in the latter test (4.5:1) was much higher than in tests 6 or 7 (1.9: and 3.8:1) respectively.) In order to compare the two methods of producing pig iron from the ore, an allowance should be made for the above differences, since more energy was expended in reducing silicon in tests 6 and 7 than when smelting the raw Matawin ore, whereas more energy was used in slag formation in the latter case. When an approximate allowance is made for these differences, the energy requirements are altered from 6000 kwh to about 6700 kwh for test No. 6, and from 5340 kwh to about 5730 kwh for test No. 7.

Although these amounts of electrical energy were considerably lower than that used when smelting the raw ore directly, they were still much larger than was expected, and in addition, the amounts

of natural gas used in preheating and prereducing the ore were very large. When the high-grade hematite (Steep Rock) ore was used in whole or in part instead of the low-grade Matawin ore the amounts of electrical energy and natural gas required were both significantly less. These results were not unexpected since the higher grade ore contained much less gangue material and the slag:metal ratio was much lower than was the case with the Matawin ore.

Considerable difficulty was experienced in operating the kiln and the furnace together, particularly when the furnace gases were being fed back into the kiln to aid in the preheating and prereduction of the ore. Under these conditions neither the kiln nor the furnace could be controlled independently, i.e. when a change was made in the operation of one piece of equipment this automatically had an effect on the operation of the other unit. For example when the degree of reduction in the kiln decreased, more reduction had to be done in the furnace: this caused an increase in the amount of carbon monoxide going to the kiln, which in turn altered the temperature and the atmosphere in the kiln. Manual adjustment of the amount of air admitted at the discharge end of the kiln was often necessary to control the temperature, but these adjustments caused wide fluctuations in the kiln gas analyses.

Unfortunately this kiln had a relatively thin refractory lining ($4\frac{1}{2}$ in.) and a large amount of heat was lost from it by radiation and convection. Also the loss of heat through the furnace roof was very high even when the power input was reduced because the furnace appeared to be underfed. In conventional electric smelting much of the heat radiating from the molten bath is absorbed by the cold feed descending along the furnace walls. In this series of tests the feed entering the furnace was already between 900 and 1000°C and consequently did not

absorb much additional heat but allowed it to radiate onto the roof.
However in spite of the fact that the feed was very hot and to some
extent prereduced, it was necessary to supply a large amount of electrical
energy to complete the smelting. When the energy input to the furnace
was reduced without a reduction in the feed rate, a thick layer of
unsmelted and semi-smelted material accumulated in the furnace, and the
energy input had to be increased again to smelt this material.

In tests 6, 7 and 8 in which Matawin ore was used, in whole
or in part, the silicon content of the metal was too high, the carbon
content too low, and the FeO content of the slag was higher than
desirable. These tests were all performed using a base:acid ratio
of less than 1:1, and the above results are in general quite similar
to those obtained in previous work. For example, in test No. 1 in
which the sintered magnetic concentrates were smelted, it was found
that the silicon content of the metal was too high, the carbon content
too low and the FeO content of the slag was too high when low $CaO:SiO_2$
ratios were used. Similar results were obtained in smelting the raw
Matawin ore, as described in test No. 2 in this report. It had been
hoped that when hot partially prereduced ore was fed to the furnace
better equilibria might be obtained between the metal and slag, and that
the silicon in the metal and the FeO in the slag might have been lower,
and the carbon content of the metal higher. Unfortunately this was not
the case and these results appear to confirm the conclusion reached
after test No. 1 was completed, that in order to obtain metal of
acceptable composition from this ore, a rather high base:acid ratio,
e.g. 1:1 or higher, is required. Unfortunately the use of high base:acid
ratios for smelting this ore would be uneconomic because of the excessive
amounts of limestone which would be needed. For this reason this

investigation was restricted to an exploration of the process when using acid slags.

PART III.   PRODUCTION OF LOW-SILICON IRON

In all the previous tests in which Matawin ore was used, the metal produced contained too much silicon, and it appeared that in order to obtain metal with an acceptably low silicon content by direct smelting, excessively large additions of limestone would be required. Consequently it was decided to investigate the possibility of producing an acceptable grade of iron from this ore by a two-stage process. In the first stage of this process the raw Matawin ore would be smelted, using an acid slag and an excess of reductant, to produce metal with a high silicon content, e.g. 15 to 20 percent. In the second stage, this high-silicon metal would be smelted with high-grade iron ore, to produce metal with low silicon content.

(a)   Production of High-Silicon Iron from Matawin Ore

Experimental Work:

Three tests were performed in which high-silicon metal was produced from Matawin ore. In one test (No. 11) the ore which had been prereduced in test No. 5 was used as furnace feed, in order to conserve the supply of raw Matawin ore. In another case (test No. 12) paper-mill bark was added to the charge to investigate the "dry top" smelting technique for this particular operation.

Test No. 10

In this operation raw Matawin ore was smelted, using coal as reductant, and an acid slag. The furnace was operated without a roof. The objective at the start of the test was to produce metal containing 20 percent silicon, while operating with a $CaO:SiO_2$ ratio of 0.5:1, but it was soon found that slag with this $CaO:SiO_2$ ratio was

very viscous and difficult to handle. The furnace voltage was increased to about 130 volts, and although the fluidity at the surface of the bath increased somewhat, the slag remained difficult to tap. Attempts were made at this time to establish a "hot dry top" type of operation in the hope that heat losses might be reduced and a higher slag temperature obtained, but these were not successful, as the bath persisted in frothing and the electrodes rose to the top of the charge each time. More limestone was then added to the charge to bring the CaO:SiO$_2$ ratio up to about 0.6:1, and some fluorspar was added to the bath just before tapping to increase the slag fluidity. These procedures resulted in some improvement in furnace operation.

During this test limestone was calcined in the kiln, and when a sufficient supply was available the limestone in the charge was replaced with burnt lime, in the hope that the amount of energy required in the furnace might be somewhat reduced when electrical energy was not required for calcining the limestone. No significant reduction in the electrical energy required was observed. Near the end of the test the electrodes were moved closer together to reduce the effective size of the hearth, and get more power per unit of hearth area, but no significant improvement was noticed in the operation as a result of this change.

The results of the test are shown in Table 16.

Test No. 11

In this test the Matawin ore which had been prereduced in the kiln in test No. 5 was used as furnace feed in order to conserve the supply of raw Matawin ore. The analysis of this material is shown in Table 15.

Table 15

Analysis of Prereduced Matawin Ore from Test No. 5
(percent)

Constituent	Percent
Total iron	20.28
Ferrous iron	12.87
Ferric iron	7.41
Metallic iron	Trace
SiO_2	41.4
CaO	9.8
MgO	5.2
CO_2	2.1
Fixed carbon	9.7

Again the furnace was operated without a roof. At the start of the test the objective was to obtain metal with about 20 percent silicon while operating with a $CaO:SiO_2$ ratio in the slag of about 0.7:1. Unfortunately too much coal was used in the early hours and an excess of reductant accumulated in the furnace. Under these conditions an excessive amount of silicon was reduced, the silicon content of the metal reaching 28 percent on one occasion, and the $CaO:SiO_2$ ratio in the slag increased very markedly. For a time the size of the molten bath decreased very appreciably and it was suspected that a carbide-type slag was present in the furnace. In order to enlarge the bath it was necessary to feed a number of charges without any reductant. The results obtained are given in Table 16.

Table 16

Results of Production of High-Silicon Iron

		Test No. 10 (Raw Matawin Ore)	Test No. 11 (Prereduced Matawin Ore)	Test No. 12 (Matawin Ore Hog Fuel)
Duration (hr)		91.20	50.58	61.63
Charge				
Raw Matawin ore	(1b)	11600	750	14405
Steep Rock ore	"	400	---	---
Prereduced Matawin ore (from Test 5)	"	---	7000	---
Limestone	"	1120	---	1531
Burnt lime	"	1239	441	1624
Coal	"	2920	545	2531
Bark (70% H_2O)	"	---	---	14730
Products				
Metal	(1b)	3819	1757	4672
Slag	"	7156	3470	8703
Slag:metal ratio		1.87	1.97	1.86
Average metal analysis	(%)			
C		0.69	0.44	0.39
Si		16.4	21.1	18.9
S		0.008	0.006	0.008
P		0.26	0.12	0.12
Average slag analysis	(%)			
FeO		2.21	1.69	2.0
CaO		30.3	28.2	30.5
SiO_2		50.1	52.4	50.4
MgO		4.57	6.20	5.7
CaO:SiO_2 ratio		0.60	0.54	0.61
Energy				
Total energy input (kwh)		22232	12531	15255
Average " " (kwh/hr)		244	248	248
Av. tapping temp. (°C) Slag		1478	1573	1566
Metal		1382	1220	1236
Recovery of Fe (%) In metal		92.7	83.1	95.9
In slag		3.6	2.7	3.5
Total		96.3	85.8	99.4
Requirements per ton (2000 1b) of metal produced				
Raw Matawin ore	(1b)	6074	854	6166
Steep Rock ore	"	209	---	---
Calcined Matawin ore (Test 5)	"	---	7968	---
Limestone	"	586	---	655
Burnt lime	"	649	502	695
Coal	"	1529	620	1083
Energy (kwh)		11643	14264	6530
Electrodes	(1b)	211	315	82
Bark	"	---	---	6305

Test No. 12

     The furnace was operated under cold dry top conditions during this test. Paper-mill bark, obtained from the E.B. Eddy Company, Hull, Quebec, was mixed with the charge to reduce the bulk density and increase the porosity. The $CaO:SiO_2$ ratio in the slag was kept in the range of 0.5 to 0.6, and the slag was found to be very viscous and hard to tap from the furnace. The "dry top" operation worked reasonably well although there was some tendency for the furnace to froth in the early stages when the $CaO:SiO_2$ ratio was very low, e.g. less than 0.5:1, and also some difficulty was experienced with bridging of the charge in the furnace. A period of bridging was usually followed by a sudden collapse of the charge, with frothing of the bath, vigorous evolution of furnace gases, and a temporary loss of the "dry top" operation.

     The results of this test are shown in Table 16.

Discussion

     Metal with a high silicon content was obtained without difficulty in each of the above tests. Also the carbon and sulphur contents of the metals were low, and so also were the FeO contents of the slags. The amounts of limestone and/or burnt lime used as flux were considerably less, per ton of metal obtained, than in previous tests because very siliceous slags were used in each case. The amounts of energy used per ton of metal obtained, were very high in tests No. 10 and No. 11, where an open-top furnace was used, but the amount was much less in test No. 12 where the furnace was operated under "cold dry top" conditions. These results are in accord with those of previous smelting operations in these laboratories, where it has always been found that large savings in energy are obtained in "dry top"

operations. The high silicon contents of the metals produced in these tests were also partly responsible for the large energy requirements since about 5 times as much energy is required to reduce a pound of silicon from SiO_2 as to reduce a pound of iron from Fe_3O_4.

Another factor which made the requirements of energy and materials per ton of metal appear very high in some cases was the carry-over of metal in the furnace from one test to another. Considerable difficulty was encountered in tapping the materials from the furnace in these tests, particularly the highly viscous slags, and at times considerable metal remained frozen or entrapped in the furnace. In test No. 11 in particular considerably more material was fed to the furnace than was removed as the recovery of Fe was only 85 percent. If all the iron had been tapped, the yield of metal would have been considerably higher and the calculated requirements of energy etc. per ton considerably lower. This metal remained in the furnace until the latter stages of test No. 13 when it finally became molten and was removed causing the yield of iron in this test to exceed the input.

The electrode consumption in tests No. 10 and 11 was very high, partly because of the factor described above, and partly due to the fact that no roof was used on the furnace, and oxidation of electrodes was quite severe. The electrode consumption in test No. 12 was much lower because the furnace was operated under "cold dry top" conditions.

(b) Reaction of High-Silicon Iron with High-Grade Ore to
    Produce Low-Silicon Iron

(1) Experimental Work in the 250 KVA Furnace:

The objective in this test was to produce low-silicon metal by reacting Steep Rock high-grade ore with the high-silicon metal

produced in previous tests. The furnace was operated with a roof on,
and with its existing carbon lining, although it was appreciated that
a basic lining would have been more desirable. Before commencing the
test the furnace was warmed for about 40 hr by smelting Steep Rock ore.
During this warm-up period the $CaO:SiO_2$ ratio in the slag was adjusted
to about 1:1 and fluorspar was added frequently in the hope that the
residue of highly siliceous slag from the previous tests might be
removed. This technique was only partially successful, as heavy banks
persisted in the furnace, and some effort was devoted to chipping these
banks with bars in the later hours of this warm-up period in order
to have the furnace as clean as possible before beginning the test.
The metal obtained in this warm-up period contained about 2.5 percent
silicon and about 3.5 percent carbon. The recovery of Fe in this
period was about 103 percent indicating that more material was removed
from the furnace than was fed in. The data from this warm-up period,
along with those from the two succeeding periods of this test which
are designated as Periods I and II, are shown in Tables 18 and 19.

Period I:

At the beginning of this period some high silicon metal
which had been produced in test No. 6 was used as feed. This material
had been broken with a sledge hammer to pieces about minus 2 in. in
size, but was not brittle enough to be crushed to a smaller size in
a jaw crusher. Hence it was appreciated that this material would
probably not react as well with the ore as would finer metal. However,
because of the short supply of high silicon metal which could be
crushed to minus 3/8 in., this coarse material was used for the initial
exploration of the process. Later in the period metal from test No. 10,
and some from tests 11 and 12 were used. The analyses of these three

lots of metal are shown in Table 17. The analyses given for the
test No. 6 metal were calculated from the analyses of the individual
taps. The metals from test No. 10, and from tests 11 and 12 combined,
were crushed to minus 3/8 in. and a sample of each of these two lots
was obtained by riffling, hence the data given for these two lots are
based on actual sample analyses.

<center>Table 17</center>

<center>Analyses of Three Lots of High-Silicon Metal Used in Test No. 13</center>
<center>(percent)</center>

	Lot No. 1 (Test No. 6)	Lot No. 2 (Test No. 10)	Lot No. 3 (Tests 11 and 12)
C	1.44	0.68	0.48
Si	12.1	16.16	18.39
S	0.018	0.010	0.010
P	-	0.27	0.12

The charge used while feeding the test No. 6 metal contained
75 lb of Steep Rock ore and 50 lb of burnt lime per 100 lb of metal.
By calculation this was sufficient lime to yield a slag with a
$CaO:SiO_2$ ratio of about 1.7:1. A small amount of coal was added to
the charge in the hope that this reductant, floating on the bath,
would keep the FeO in the slag down to a very low level. The furnace
was operated at about 240 kilowatts input, and for most of the period
a fairly high voltage was used. The reaction between the ore and the
large pieces of metal appeared to be rather slow; a good fluid bath
was seldom obtained except when some additional raw Matawin ore was
added occasionally to increase the fluidity. This was made necessary
by the fact that the silicon in the metal was not all being oxidized
by the ore, and consequently less SiO_2 was being produced than was

expected and slag with a very high $CaO:SiO_2$ ratio was obtained for a time, e.g. 1.9:1.

A smoother furnace operation was obtained when the test No. 10 metal was fed. The charge used at this time contained 85 lb of Steep Rock ore and 47 lb of burnt lime per 100 lb of metal. The addition of coal to the charge, to try to keep the FeO content of the slag at a low level, was continued. The $CaO:SiO_2$ ratio in the slag varied between 1.46:1 and 1.63:1, and although a larger and more fluid bath was obtained, some difficulty was again experienced with crust formation in the furnace. Additions of Matawin ore were necessary on several occasions to restore sufficient fluidity. The silicon content of the metal remained relatively high (4 to 5 percent) and the FeO content of the slag varied between 2 and 4 percent.

The charge used when feeding the metal from tests 11 and 12 contained 95 lb of Steep Rock ore, 40 lb of burnt lime, 30 lb of dolomite, (unburnt) and 12 lb of coal per 100 lb of metal. It was hoped that the dolomite might help to increase the slag fluidity. Again the slag remained rather viscous and considerable difficulty was experienced with crust formation in the furnace. The remedy used in this case was the addition of extra Steep Rock ore occasionally, and small amounts of fluorspar, particularly just before tapping. The silicon content of the metal produced remained relatively high, from 3 to 7 percent and the FeO content of the slag remained quite high, from 5 to 20 percent. Near the end of the period the amount of coal in the charge was reduced from 12 lb to 6 lb, and shortly afterward the proportions of metal: Steep Rock ore in the charge were changed from 100:95 to 75:105. Shortly afterward a distinct change occurred in the character of the operation. The results obtained are given in Tables 18 and 19.

Table 18

Results of Smelting of High-Silicon Metal with High-Grade Iron Ore

		Warm-up Period	Period I	Period II
Duration (hr)		40.58	40.77	9.65
Charge				
Steep Rock ore	(lb)	7700	7929	2100
Matawin ore	"	---	825	---
Metal (Test No. 6)	"	---	1765	---
Metal (Test No. 10)	"	---	2700	---
Metal (Tests No. 11 and 12)	"	---	3825	1500
Limestone	"	998	---	---
Burnt lime	"	---	3448	470
Dolomite	"	---	1200	600
Coal or coke	"	2032	962	120
Products				
Metal	(lb)	5012	11822	2936
Slag	"	1857	5614	2100
Slag:metal ratio		0.37	0.47	0.72
Average metal analyses (%)				
C		2.61	2.59	0.86
Si		3.44	5.65	0.23
S		0.112	0.019	0.085
P		---	0.242	0.119
Average slag analyses (%)				
FeO		10.3	8.9	6.9
CaO		37.0	46.5	38.2
SiO_2		34.3	32.9	38.3
MgO		---	7.7	6.5
Al_2O_3		11.2	5.2	4.5
$CaO:SiO_2$ ratio		1.08	1.41	1.00
Energy				
Total energy input (kwh)		9807	9796	1934
Average " " (kwh/hr)		242	240	200
Av. tapping temp. (°C) Slag		1509	1582	1443
Metal		1365	1379	1405
Recovery of Fe (%) In metal		99.8	90.6	116.0
In slag		3.2	3.2	4.5
Total		103.0	93.8	120.5

Table 19

Requirements of Materials and Energy per Ton of Metal Product in Test No. 13

		Warm-up	Period I	Period II
Steep Rock ore	(1b)	3072	1341	1430
Matawin ore	"	---	140	---
Metal (Test No. 6)	"	---	298	---
Metal (Test No. 10)	"	---	457	---
Metal (Tests 11 and 12)	"	---	647	1022
Limestone	"	398	---	---
Burnt lime	"	---	583	320
Dolomite	"	---	203	409
Coal or coke	"	811	163	82
Energy (kwh)		3913	1657	1317
Electrodes	(1b)	78	49	42

Period II

This period of operation was very short, (9.6 hr) and only 5 taps of metal were obtained, but the silicon content of the metal was quite low, varying only from 0.06 to 0.33 percent. The charge used contained 75 lb of metal from tests 11 and 12, 105 lb of Steep Rock ore, 20 lb of burnt lime, 30 lb of dolomite and 6 lb of coal. In spite of the higher ratio of ore to metal and the lower amount of coal, the FeO content of the slag decreased from an average of 8.9 percent in Period I to an average of 6.9 percent in this period. The bath was much more fluid than in the preceding period and the power input to the furnace had to be reduced to about 200 kilowatts. Most of the crusts

and heavy banks which had been present in the furnace during the previous operation were melted out, and the erosion around the "B" tap hole became so severe that this hole could no longer be used. The metal and slag were then tapped from "C" tap hole which was originally designed as the drain hole. At the end of the test the furnace was drained as dry as possible.

After the furnace had cooled down the metal and slag which had remained in the furnace were dug out, separated and analyzed. From the data obtained it was calculated that the metal and slag contained 1895 lb. and 5 lb, respectively of Fe. A decision was made to apportion these materials among the yields of metal and slag of the three periods of test No. 13. The materials were divided among the three periods according to the relative weights of iron contained in the metal and slag originally tapped in these periods. The data given in Table 18 for yields of metal and slag in the three periods of this test thus include the proportional amounts from this clean-up operation, as well as the amounts originally tapped from the furnace.

(2) Experimental Work in 60 KVA Furnace:

It was suspected that the difficulty encountered in trying to produce low-silicon iron by reacting high-grade ore with high-silicon iron in the 250 KVA furnace had been due, in part at least, to the fact that this furnace was equipped with a carbon lining. For this reason four experiments were performed in the 60 KVA furnace, using magnesite linings, to determine whether the silicon could be removed more thoroughly from the metal under these conditions.

The experiments are designated as Numbers 1, 2, 3 and 4 respectively. In experiments 1, 2 and 3 the high-silicon metal used was that produced in tests 11 and 12. In test No. 4 some metal containing only 6.6 percent silicon was used to determine whether this

material could be treated as successfully as the material containing

18 percent silicon. In all cases Steep Rock ore was used as the

oxidizing agent, and burnt lime and dolomite were added to flux

the silica produced by the oxidation of the silicon. In Experiment

No. 1, no coal was added to the charge. In Experiment No. 2 the amount

of ore used was about double the amount required to oxidize the silicon

and sufficient ooal was added to reduce the iron from the additional ore.

In Experiment No. 3 the amount of additional ore used was about 40 percent

greater than that required to oxidize the silicon, and again a suitable

amount of coal was added to reduce the iron from the additional ore.

In Experiments 1, 2 and 3 the high-silicon metal,

which was all minus 3/8 in., was mixed with the other ingredients before

charging. In Experiment No. 4 the metal, in the form of two large pieces,

was melted first, then the mixture of ore and flux was added to the furnace.

In this case no coke and additional ore were used. In all cases the

materials smelted readily and the slags obtained were very fluid. When

it was estimated that the reaction was completed the metal and slag were

poured together into a preheated cast iron pot and separated after cooling.

In all cases the slag was crushed and magnetically cleaned to remove fine

metal-shot before a sample was removed for analysis. The metals obtained

were all found to be ductile and samples were readily obtained by drilling.

The results obtained in the four experiments are shown

in Table 20.

Table 20

Results of Production of Low-Silicon Iron in the 60 KVA Furnace

	Exp. No. 1	Exp. No. 2	Exp. No. 3	Exp. No. 4
Charge				
Metal (Tests 11 & 12) (1b)	11.25	11.25	11.25	---
Metal (Test No. 7, Tap 37)$\ast$ "	---	---	---	15.56
Steep Rock ore "	10.5	20.5	14.5	5.25
Burnt lime "	2.0	2.0	2.0	4.0
Dolomite (unburnt) "	3.0	3.0	3.0	1.5
Coke "	---	2.2	0.88	---
Silica "	---	---	---	1.5
Analysis of metal obtained (%)				
C	0.19	0.09	0.04	1.06
Si	0.33	0.02	0.01	0.39
S	0.010	0.04	0.037	0.014
P	0.21	0.084	0.15	0.15
Analysis of slag obtained (%)				
FeO	2.20	8.88	9.13	3.73
CaO	20.64	18.24	16.29	26.28
SiO_2	37.20	35.44	32.84	32.40
MgO	32.60	36.28	36.28	30.20

$\ast$ This metal contained 6.6% Si; 2.2% C; and 0.07% S.

Discussion

Low-silicon metal was successfully produced by reacting high-silicon metal with high-grade iron ore in the 250 KVA furnace, but the period of good operation was relatively short (9.5 hr). During Period I, which lasted for about 40 hr, the metal produced contained too much silicon. It is believed that several factors contributed to these results. As previosuly mentioned, the furnace had a carbon lining during this operation and this tended to maintain reducing conditions in the furnace bottom rather than oxidizing conditions which were more desirable since oxidation of the silicon in the metal was the major objective in the tests. An indication that the presence of the carbon lining had an important effect on retarding the oxidation of the silicon is given by the results obtained in the magnesite-lined 60 KVA furnace, where low-silicon metal was readily obtained in all four experiments.

Another factor which probably contributed to the poor results obtained in Period I of test 13 was the fact that when this test was begun a considerable amount of high-silicon metal was present in the furnace. At the end of each smelting operation it was never possible to drain the furnace completely of all metal and slag; a certain amount of each always remained frozen around the walls of the crucible, and on the bottom, and was carried over into the succeeding test. It was obvious from visual observations at the start of test 13 that a considerable amount of metal and slag were carried over from the previous tests. It was suspected that the reaction between the high-silicon metal and the iron ore in the feed, which under equilibrium conditions should have yielded low-silicon metal, was being retarded by the reserve of relatively cold high-silicon metal frozen on the bottom of the furnace. In Period II of test No. 13, this reserve of

high-silicon metal appeared to have been all removed from the furnace, because the bath at that time was large and very fluid.

The coal which was used in test No. 13 in the hope of keeping the FeO content of the slag down to low values, did not appear to have been very effective for this purpose, as the FeO contents of the slags remained rather high (7 to 10 percent). In addition the coal may have had a detrimental effect by maintaining a reducing atmosphere and reducing conditions in the slag, when oxidizing conditions were required for removal of the silicon from the metal. It is probably significant that the amount of coal used in Period II of test No. 13 when low-silicon metal was obtained, was only about one-half of that used in Period I, and in addition the ratio of ore to metal was higher in Period II than in Period I. The conditions in the furnace might thus be described as much less strongly reducing during Period II, when good results were obtained, than during Period I when high-silicon metal was produced.

Low-silicon metal was obtained in each of the four experiments performed in the magnesite-lined 60 KVA furnace. The use of coke in Experiments No. 2 and 3 appeared to make little difference in the silicon content of the metal obtained, but apparently increased the FeO contents of the slags rather than decreasing them as was expected. These results are in accord with those obtained in the larger furnace, the FeO content of the slag being lower in Period II where conditions were less strongly reducing than in Period I where more coke was used.

The results obtained in Experiment No. 4 suggested that good equilibrium could be obtained between the silicon in the metal and the iron oxide of the ore if the slag was fluid, since in this case

the metal and ore were not mixed before they were fed to the furnace, but rather the reaction must have taken place across the slag:metal interface.

In all cases the sulphur contents of the metals were quite low. This was due largely to the low sulphur content of the high-silicon iron, and is an attractive feature of the process.

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GEV:RAC:GVS:GNB:RRR/DV