



## ジョセフの報告(2)

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館 充

### Iron Ore Preparation and Blast Furnace Practice in Japan

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以下は第1報 鉄と鋼 第71巻 9号 pp. 1085~1093の続きである。

#### Objectives of Iron Ore Preparation Chemical Composition

The broad purpose of iron ore preparation is to obtain more efficient and more uniform blast furnace operation to the end that pig iron of uniform silicon and sulphur content will be produced with a minimum consumption of fuel. Fuel requirements are determined by a number of factors, but one of the most important considerations is the amount of slag forming materials associated with the iron in the ore mixture. The effect of the quantity of slag produced per ton of iron from the ore gangue can be illustrated by a comparison of practice in similar furnaces on prepared Lake Superior iron ore containing 51 percent iron, and on prepared Alabama ore containing 35 percent iron.

Operating Results	Prepared Lake Superior ore	Prepared Alabama ore	Difference
Iron per day, N.T.	1 605	1 268	- 377
Slag per N.T., pounds	968	2 040	+ 1 072
Coke per N.T., pounds	1 510	2 170	+ 660
Carbon per N.T. of iron	1 266	1 852	+ 586

This performance of furnaces of approximately the same size under comparable conditions except the volume of slag shows that an additional 1 072 pound of slag requires 660 pounds of coke or approximately 0.6 pounds of coke per pound of slag. This relationship between slag volume and coke consumption is not necessarily the same for all levels of slag production but illustrates the great amount of heat required to form, melt and superheat the slag.

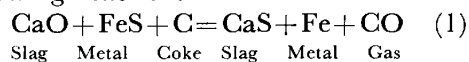
Of course the amount of slag formed per ton of iron depends mostly on the chemical composition of the ore, but the ash in the coke also contributes substantial amounts of slag. Since this ash is released as the coke surface burns at the tuyeres, and because it must be fluxed at or below this level, the slag it produces may require more fuel than the slag produced from the ore gangue. This special case will be discussed later in dealing more

specifically with the coke ash problem.

Control over the amount of phosphorus and manganese in the iron is exercised by regulating the amount of these elements in the ore mixture. This is particularly true of phosphorus because all of this element is reduced and enters the metal. However, high temperatures favor the reduction of manganous oxide and tend to increase the recovery of this element in the metal. Variations of a few tenths of a percent of manganese may therefore result from variations in temperature. Wide variations on the order of 1.0 percent of manganese must however be obtained by varying the amount of manganese in the ore.

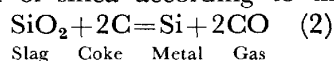
A large excess of carbon exists in the lower part of the furnace. In fact pieces of coke are forced into the iron bath by the weight of the stock column. The iron accordingly becomes saturated with carbon so that the amount taken into solution is a function of the temperature and the amount of other elements present, particularly silicon which reduces the solubility of carbon in iron.

Once the ore mixture is fixed or subject to relatively small variations in manganese and phosphorus, control over the composition of the iron becomes a problem of controlling the silicon and sulphur it contains. Due to an absorption of sulphur above the tuyeres, the iron reaching the crucible of the furnace normally contains several times more sulphur than can be tolerated in the final pig iron. This sulphur is removed by the following reaction.



Like most reactions, this one will proceed at rates depending largely upon the existing temperatures of the contiguous layers of iron and slag. Slag composition has an important though relatively minor effect on the desulphurization of the iron because the activity of the calcium oxide in the slag changes with the amounts of silica and alumina present.

The silicon in the metal is formed by the reduction of silica according to the reaction:



Here again, temperature is the major factor of

control and slag composition which alters the activity of the silica is of secondary importance. The great influence of temperature upon the amount of silicon and sulphur in the iron is shown in Fig.'s 8 and 9 of the attached report, "The Blast Furnace Process and Means of Control."

If each additional pound of slag requires on the order of 0.6 pounds of additional coke, it follows that the variations in the amount of ore gangue as a result of variations in the ore mixture will alter hearth temperatures. Moreover, if such variations occur from hour to hour or from day to day, it becomes utterly impossible for the furnace operator to make suitable adjustments in the amount of flux used because such variations only become apparent 8 to 10 hours after the ore enters the furnace and had time to influence the composition of the slag as tapped. Unless the principal ores are each blended in such a way as to eliminate variations in the amount and character of the gangue, uniform blast furnace operation is impossible. As a "burnt offering" to the unknown, when ore quality is variable, the furnace is usually operated on the hot side with a waste of fuel but nevertheless large changes in iron analysis are inevitable and a product varying widely in composition is passed onto the open hearth operator.

Except for the blending and mixing associated with the concentration of magnetite use at the Kamaishi plant of Fuji Iron and Steel Company, Ltd., no adequate provision is made to blend iron ore in Japan for the purpose of securing constant composition for each ore. The use of a large number of ores and charging them in uniform proportions tends to smooth out variations in burden mixtures, but inadequate sampling of incoming ore, and the lack of availability of chemical analysis until after the ore has been consumed, create a condition which must be corrected if Japanese blast furnace practice is to be brought up to a high level of efficiency.

Problems of iron ore concentration are beyond the scope of this report, but it should be pointed out that slag volumes range from about 1 080 pounds/N.T. iron at Kamaishi, where magnetite concentrates and lump magnetite are used, to 1 560 pounds/N.T. iron at furnaces using substantial amounts of low grade domestic ore. The high ash in the coke, which averages about 14.0 percent, also tends to increase slag volumes. The extent to which various ores can be concentrated is a matter requiring special study. However, it appears that modern methods of magnetic concentration can be used effectively to eliminate silica from iron sand. Also, the concentration of pyrite deposits

by differential flotation of iron sulphide and copper sulphide not only eliminates silica but results in the profitable recovery of copper. Improvement in this practice may prevent copper from reaching the pig iron in excessive amounts. These two problems related to the concentration of iron bearing materials warrant special study and present a field for profitable research.

### Effect of Ore Gangue upon the Value of Iron Ore

The use of siliceous iron ore increases the cost of pig iron because it results in more slag, requires more flux and decreases output per day and coke per ton of pig.<sup>†</sup> A comparison in practice on siliceous ore and a high grade ore will illustrate differences in practice and costs.

Chemical analyses, percent.

	Fe	SiO <sub>2</sub>	Al <sub>2</sub> O <sub>3</sub>	Al <sub>2</sub> O <sub>3</sub> + SiO <sub>2</sub>
Domestic ore	45.39	17.44	1.51	18.95
Imported ore	58.29	6.63	2.33	8.96

A net ton of pig iron will contain 1 840 pounds of iron (92% of 2 000). Therefore 4 050 pounds or 1.81 long tons of domestic ore will be required per ton of pig iron. (1 840 lbs/0.4539=4 050 lbs) On the same basis, 3 160 pounds or 1.41 tons of the imported ore will be required per ton of pig iron. The quantity of slag and coke required in the two cases will differ as follows:

	Tons of ore per N.T. of pig	Pounds acid per N.T. of pig	Pounds of slag, per N.T. of pig	Additional pounds of slag
Domestic ore	1.81	766	1 534	968
Imported ore	1.41	283	566	

Applying a figure of 0.6 lb of coke per additional pound of slag, it follows that 581 pounds more coke will be required to produce a ton of pig iron from the siliceous ore. If coke costs \$ 35.00 per net ton, the additional fuel cost will be \$ 10.15 per ton of pig iron

$$(581/2\ 000 \times \$ 35.00 = \$ 10.15)$$

About 968 pounds of extra flux will be required in the case of the siliceous domestic ore and will cost \$ 1.04 if limestone sells for \$ 2.40 per long ton. A decrease in tonnage will be accompanied by an increase in fixed charges amounting to about 32 percent. If the fixed costs above raw materials total \$ 3.30, an overhead penalty of \$ 1.06, should be applied.

$$(581/1\ 800 = 0.322 \times \$ 3.30 = \$ 1.06)$$

The overall summary can now be given as follows:

	Tons of ore per N.T. of pig	Cost per L.T. of ore	Cost of ore per N.T. of pig	Total penalties	Cost plus penalties
Domestic ore	1.81	\$11.05	\$20.15	\$12.25	\$32.40
Imported ore	1.41	\$23.00	\$32.40		

Extra coke, \$10.15, extra flux, \$1.04, over head 1.06 = \$12.25

<sup>†</sup> コークス比は増大するので "increases" が入るものと思われる。

On the basis of the assumption made, it appears therefore that compared to this imported ore at \$ 23.00, this domestic ore may have a value of \$ 11.05, an amount which should cover mining costs and development work on methods of concentration.

It is also evident that, if the domestic ore, including sulphides, can be concentrated, considerable concentration cost might be justifiable.

The costs assumed can not fit all cases, but illustrate the order of magnitude of the increased cost of producing pig iron from siliceous ore. On a percent or unit basis, the calculations show that an additional 10 units of silica plus alumina increase production costs by \$ 12.25 per ton pig iron, or an increase of above \$ 1.20 per ton pig iron for an increase of 1.0 percent silica plus alumina in the ore.

The foregoing calculations show that current high prices of imported ore will permit an evaluation of domestic ore at prices sufficient to permit development work and new concentrating plants.

The present dependence on foreign ore and the great need for a safeguard against dislocated water transportation in case of war, should stimulate developments to improve the quality of domestic ore and especially the development of methods to treat various kinds of Japanese iron pyrite.

### Sulphur in Iron Ore

An increase in the amount of sulphur in the raw materials requires adjustments in blast furnace operation that add to the cost of producing pig iron containing acceptable amounts of sulphur. More basic slags and somewhat higher temperatures can be employed to speed up desulphurization but most operators prefer to increase the volume of slag without altering its basicity. Somewhat higher temperatures must be maintained in the furnace hearth to accelerate the transfer of sulphur from the iron to the slag. The severity of all these adjustment will depend upon the sulphur specifications for the iron. The lower the sulphur in the iron, the greater must be the changes to handle increased amounts of this element in the raw materials. Operating data from the blast furnace of the Kaiser Company, Inc., Fontana, California, show that high sulphur iron ore can be converted into a very satisfactory raw material by complete crushing to 5/8 inches and sintering the crushed product with moderate amounts of carbon.

When the sulphur content of the Vulcan ore, used at Fontana, increased to an average of 2.0 percent in 1947, the practice of crushing all of this ore to minus 5/8 inches was adopted although the ore itself was an excellent furnace material except

for the high sulphur. Previous to this time, only the minus 1/4 inch portion of the ore had been sintered. The analyses of the 1/4 inch fines and the crushed ore follow:

Data	SiO <sub>2</sub>	Al <sub>2</sub> O <sub>3</sub>	CaO	MgO	Fe	S	Size
5/43	7.28	1.21	5.13	3.47	49.55	0.105	-1/4'' × 0
11/47	5.50	0.86	3.00	4.37	52.33	1.820	-5/8'' × 0 (crushed)

The sinter produced from these two ores had the following composition:

Data	SiO <sub>2</sub>	Al <sub>2</sub> O <sub>3</sub>	CaO	MgO	Fe	S
5/43	9.11	1.84	5.66	4.47	52.8	0.020
11/47	6.78	1.18	3.16	5.07	57.31	0.064

This reduction from 1.82 percent in the ore to 0.064 percent in the sinter was made when using a mixture of 68 percent crushed ore (-5/8 inches), 25 percent return fines, 3 percent flue dust and 4 percent coke. The low coke in the sinter mixture is very significant as it shows that excessive amounts of carbon must be avoided if the major portion of the sulphur in the ore is to be eliminated during sintering. Carbon in excess of 4 percent, tends to protect the sulphur against oxidation.

The effect of a reduction in the amount of easily reduced, self-fluxing sinter on furnace practice is shown by average monthly operating data for the Kaiser furnace.

	February	March	April
Percent sinter	42.0	26.2	45.3
Tons of iron per day	1 259	1 131	1 231
Flue dust	72	106	81
Coke rate pound per ton	1 406	1 557	1 442

It is of interest to point out that the Kaiser furnace, through a gradual improvement in raw materials and furnace practice, has averaged 1 340 tons iron per day for a period of a month on a coke rate of 1 363 pound per ton. In describing these excellent results Mr. J. D. SAUSSAMAN, Blast Furnace Superintendent, made the following comments:

"In considering the factors that contributed to this operation, one must give a great deal of credit to the preparation and control of raw materials made possible by the modern equipment installed at Fontana. Without crushing, screening, bedding, and reclaiming, and sintering, the ores, it would have been practically impossible to have produced good basic iron from variable western raw materials. Although each of these beneficiation processes had benefited furnace operation the use of sinter in the burden has made the most noticeable contribution". It is important to note that the sinter used in obtaining this excellent furnace performance, was self-fluxing, rather well oxidized, soft burned and easily reduced. These practical results afford unmistakable evidence of the savings to be made

by eliminating sulphur in the production of good sinter.

### Sulphur and Slag Forming Materials in Iron Ore and Coke

As shown in Table 1, Part I, of the appendix, the sulphur in the ores currently used in Japan will range from a trace to several percent. Although substantial amounts of sinter are produced from limonitic ore, the use of large amounts of fuel prevents satisfactory oxidation of sulphur. Except for sinter produced from magnetic concentrates, at the Kamaishi Plant, the sinter currently produced in Japan will average about 0.2 percent sulphur. An appraisal of the over-all situation is rather difficult, but the sulphur content of the slag and the fuel indicates that the iron bearing materials used in Japan will average about 0.3 percent sulphur. The control of sulphur in the iron is not a less serious problem in blast furnace practice, because the coke used will average only about 0.6 percent sulphur. Moreover high slag volumes on the order of 1 400 pounds/N.T. pig iron, which are due partly to high ash coke and partly to high ore-gangue facilitate the control of sulphur. This fortuitous situation does not, however, alter the opportunity to improve a more complete elimination of sulphur. A reduction of sulphur would permit an immediate reduction in fuel but complete benefits will not be realized until slag volumes are reduced by 400 to 500 pounds either by the elimination of siliceous ores and by the use of high iron and low silica sinter made from domestic sulphide ore.

In order to get a more quantitative appraisal of the effect of sulphur in the ore, and of acids in the

coke ash and ore, a comparison has been made in Table 5, between Lake ore practice in the States and the current practice in Japan.

Notwithstanding the presence of 1.0 percent of sulphur in the American coke, the total sulphur charged is about 3.5 pounds under the corresponding figure for Japanese practice. Another major difference is in the amount of coke ash and ore gangue in the two practices. The 14 percent ash coke typical of Japanese practice contains 96 more pounds of ash equivalent to 85 pounds of silica plus alumina or 170 pounds of slag. This amounts to about 39 percent of the 438 pound of excess slag produced in Japanese practice. The remaining 61 percent of the slag or 268 pounds results from the use of more siliceous ore burdens in Japan.

Several major differences in practice are apparent from the preceding tabulation. In typical Japanese practice, about 10 pounds or 48 percent of the total sulphur enters the furnace in the ore burden compared to 0.66 pound or 4 percent for the States. The Gumma ore contains from 1.5 to 2.5 percent of sulphur and is difficult to screen at 5/8 inches, a good dividing line for preparing such ore for sintering. This ore could be converted into a very good sinter or perhaps into pellets or nodules if it were crushed to 5/8 inches and agglomerated in a manner that would eliminate sulphur. It contains but 5 to 6 percent of acids and, through the elimination of some 13 percent combined water, the percent of iron could be raised from about 52 percent to about 59.0 percent. A careful study should be made of the distribution of sulphur in the various screen fractions of all high sulphur ores and pyrite cinder so that sulphur can be eliminated properly by crushing screening and sintering.

A review of the ores in Table 1 of the appendix indicates that certain ores in addition to Gumma can be greatly improved by crushing and sintering the entire ore or by proper screening at 5/8 inches with heated screens to separate fines that may contain a major portion of the sulphur. Ores that fall in this category are as follows: Kutchan, Tokushunbetsu, Takai, Matsuo, and Namerikawa. Most of the pyrite cinder (residue from acid production) is elimination of sulphur as it contains from 0.59 to 4.3 percent of this element. (See Table 1 of Appendix).

### The Effect of High Ash Coke on Blast Furnace Practice

The improvement of coke by the washing of coal is difficult to appraise except by blast furnace performance. In a paper published in 1943, C. D. KING, reported the comparative performance

Table 5. Comparison of sulphur distribution and slag forming Materials, coke rates and slag volumes.

	Japan	USA	Differences
Percent Fe in ore	56	56	
Pounds ore/N.T. pig	3 300 <sup>1)</sup>	3 300	
Percent S in ore	0.3	0.02	- 0.28
Pounds from ore	9.9	0.66	- 8.26
Pounds coke/N.T. pig	1 800.0	1 560.00	- 240.0
Percent S in coke	0.6	1.00	+ 0.4
Pounds S in coke	10.8	15.60	+ 4.8
Pounds S charged	20.7	16.26	- 3.44
Percent S from ore	48.0	4.00	- 44.0
Percent S from coke	52.0	96.00	+ 44.0
Pounds S to metal	0.9	0.90	
Pounds S to slag	19.8	15.36	- 3.44
Percent ash in coke	14.0	10.00	- 4.00
Pounds ash in coke	251.0	156.00	- 96.00
Pounds acids in ash	210.0	125.00	- 85.00
Pounds acids in ore	490.0	356.00	-134.00
Pounds total acids	700.00	481.00	-219.00
Pounds equiv. slag	1 400.00	962.00	-438.00
Percent slag from ore	70.00	74.00	+ 4.0
Percent S in slag	1.40	1.60	+ 0.2

1)  $2\ 000 \times 0.92 = 1\ 840$  pound Fe per net ton of pig iron  
 $1\ 850\text{Fe} = 3\ 300$  pound ore.

of blast furnaces working on coke made from unwashed and from washed coal as follows:

An examination of these data shows that the washed coal reduced the coke ash from 227 to 183 pounds per ton of pig. This reduction of 44 lb in ash would correspond to 36 pounds of acids equivalent to about 70 pound of slag or about 36 percent of the total slag reduction of 195 pounds of slag. It would appear therefore that the remaining 125 pounds of slag reduction must have come either from less ore gangue or from a reduction in the bases in the slag. Assuming that the ore gangue contributed about the same quantity

Table 6. Comparative blast furnace performance on coke from unwashed and washed coal (C.D. KING).

	Unwashed	Washed	Effect	
			Amount	Percent
<b>Coke avail (dry)</b>				
Ash percent	11.9	10.4	-1.5	
Sulphur percent	0.98	0.88	-0.10	
Fixed carbon percent	87.8	89.30	+1.5	
<b>Single day variations</b>				
Ash-maximum	14.1	11.0		
Ash-minimum	11.1	10.1		
Sulphur-maximum	1.08	0.97		
Sulphur-minimum	82	0.80		
<b>Blast furnace</b>				
Net tons iron day	781	844	+63	+8.1
Pounds coke, per N.T. iron	1 909	1 760	-149	-7.8
Pounds F.C. per N.T. iron	1 674	1 570	-104	-6.2
Pounds slag per Net ton iron	1 286	1 091	-195	-15.2
Percent casts over 0.04 percent	6.1	3.8	-2.3	
Proc. blast furnace, coke oven and raw materials committee AIME 1943.				

of slag in both cases and that the reduction of slag was due to uniformly lower ash and lower sulphur, it follows that a reduction of 195 pounds of slag was associated with a reduction of 104 pounds of fixed carbon. This indicates a reduction of one pound of slag by reason of changes due to lower ash and sulphur in the coke will permit a reduction of 0.533 pounds of fixed carbon. ( $104/195=0.533$ )

### The Effect of Coke Ash on the Available Fixed Carbon

An increase in the ash content of coke, normally results in a corresponding decrease in the fixed carbon. However, as the ash increases a portion of the fixed carbon must be used to form, melt, and preheat the slag resulting from the additional ash. This effect of coke ash in reducing the amount of available fixed carbon over and above the quantity needed for the heat requirement following the associated ash can be illustrated as follows:

	Coke No.1	Coke No.2	Difference
Ash percent	10.0	15.0	+ 5
Vol. Matter, percent	1.5	1.5	
Fixed Carbon, percent <sup>1)</sup>	88.5	83.5	- 5
Pounds coke, N.T. pig	1 600.0	1 800.0	+200
Pounds, F.C.N.T. pig	1 416.0	1 503.0	+ 87.0
Pounds, ash N.T. pig	160.0	270.0	+110.0
Pounds, acids N.T. pig	128.0	216.0	88.0
Pounds equiv. slag	256.0	432.0	+176.0
Pounds F.C. for ash slag	128.0 <sup>2)</sup>	216.0	+ 88.0
Pounds available F.C.	1 288.0	1 287.0	- 1.0
Value of coke per N.T.	\$36.0	\$32.0	- 4.00
Cost of coke N.T. pig	\$28.80	\$28.80	

1) Moisture and S free basis

2) One-half pound F.C. per lb. of slag

The foregoing tabulation points up several interesting comparisons. It is rather surprising to find that 200 pound of additional 15 percent ash coke contains only 87 pounds of additional fixed carbon. This additional fixed carbon is required to handle slag from the excess silica and alumina derived from the high ash coke. The available carbon is accordingly the same in each case. If the fuel required per ton of pig is to remain constant the 10 percent ash coke has a value of \$ 36.00 when the 15 percent ash material is priced at \$ 32.00 per ton. On the basis of these assumptions, each percent of ash in the coke adds \$ 4.00/5 or \$ 0.80 to the smelting costs by reason of a higher fuel consumption.

Other smaller adjustments should, however, be made for a loss of tonnage and a proportionate increase in these fixed costs which increase as the tonnage decreases. Using the coke rates of 1 600 pounds and 1 800 pounds as an index of tonnage, the output at the 200 pound higher coke rate per net ton will increase fixed charges by 200/1 600 or 12.5 percent.

Such a decrease in output will add ( $0.125 \times \$ 3.30 = \$ 0.41$ ) to production costs. Additional limestone amounting to about 176 pounds will add another \$ 0.19 if flux costs \$ 2.40 per long ton. Thus, we see that the use of 15 percent ash coke adds a total of \$ 4.60 to the cost obtained on 10.0 percent ash coke. The overall figure of additional costs for each increase of 1.0 percent of ash in the coke, accordingly becomes \$ 0.92 ( $\$ 4.60/5$ ).

### Size Preparation of Iron Ore

Efficient conversion of iron ore to pig iron depends upon establishing contact between an ascending stream of gases and a descending column of stock. The sensible heat of the gases generated at the tuyeres bring the charge up to reacting temperatures and the carbon monoxide converts the ore to metal. One objective of iron ore preparation is to increase the size of particles smaller than 5/8 inches because they offer high resistance to gas flow and contribute all the dust carried out in

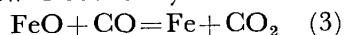
the exit gases (See Fig. 4 of attached report). Due to the segregation of sizes upon charging, and the relative proportions of ore and coke in different parts of the furnace, the flow of gases, existing temperatures, and gas composition vary widely across any level of the furnace shaft, (See Figs. 3, 10, 11, 12, 6, and 7, of attached report.). In addition to reducing large variations in the permeability of the charge across any level of the shaft, elimination of the 5/8 inches or high resistance size fraction, lowers the overall pressure thus permitting more rapid operation and larger tonnages without building up bottom pressures that lead to hanging, slipping, and channeling, of the gases with attendant irregular operation.

Although large particles of iron ore increase the permeability of the stock, they offer a relatively small surface area to absorb heat from the gases and, depending upon their porosity, will not be properly preheated and reduced before reaching the melting zone or in some extreme cases before reaching the tuyeres. Blast furnaces are the largest heat interchange units in industrial operation, yet little attention has been given to this phase of operation.

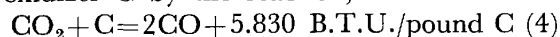
The advantages of small checkers in regenerators such as blast furnace stoves are well understood and the use of such checkers is an accepted feature of modern stove design. In fact total heating surface is an accepted measure of stove capacity. One of the purposes of crushing coarse ore is therefore to increase the surface to facilitate the transfer of heat from the rising gases to the ore particles.

Another purpose of crushing ore is to decrease the time required in the blast furnace for sufficient reducing gas to reach the innermost portion of the ore particle for complete reduction of the iron oxides to metallic iron.

Since crushing produces fine particles that require agglomeration, the porosity and reducibility of the ore must be given consideration in determining the size of crushing for hard, dense compact ores. Such ore resist breakage in mining and handling and therefore normally require crushing to a size that will permit the last stage of reduction to metallic iron to be completed at temperatures below 1 000°C by a reaction as follows:



If reduction of FeO is not completed at temperatures below about 1 000°C, the CO<sub>2</sub> from reaction (3) oxidizes C by the reaction,



This loss of carbon above the tuyeres reduces the heat available from a given amount of carbon charged into. Heat is lost by reaction (4) and less

carbon is available for combustion and heat liberation at the tuyeres. For example if crushing of coarse, dense ore decreases solution loss (reaction (4)) by 100 pounds, this much additional carbon will be burned and over a million additional B.T.U. will be made available to the process.

$$100 \times 4\,370 \text{ B.T.U./pound C} = 437\,000 \text{ B.T.U.}$$

-Heat released

$$100 \times 5\,830 \text{ // /pounds C} = 583\,000 \text{ //}$$

-Heat absorbed

Total additional available

$$\text{heat} = 1\,020\,000 \text{ B.T.U.}$$

A reduction in solution loss has another adverse effect upon fuel consumption and regularity of operation that is difficult to appraise. The chilling effect of the absorption of one-half million B.T.U.'s is more detrimental the lower it occurs in the furnace, as there is less opportunity for the stock to be properly preheated before reaching the crucible. When it occurs in the crucible, the chilling effect upon hearth temperature is particularly undesirable. The most important factor in the efficient use of carbon in the blast furnace is the providing and liberating of the necessary heat at the required potential in the lower part of the furnace. Any steps taken to increase the potential temperature at which heat is developed in the hearth, or to reduce hearth heat requirements, decrease the demand for carbon in the hearth. Full advantage should be taken of ore preparation, as proper pre-treatment of the ore pays dividends in furnace operation. Ideally each ore should be prepared according to its density and reducibility since it is well known that dense hematites reduce more slowly than more porous ones. (See Fig. 3, attached Exhibit A).

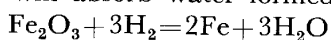
### Reducibility Tests on Iron Ore

One of the great values of reducibility tests is the evidence that such tests give of large differences in the time taken to reduce various ores under comparable conditions of gas flow, temperature, size of specimen and gas composition. Measurable differences on the order of 5 to 1 in the time required for reduction offer definite proof of radically different behavior in the blast furnace.

Reduction with hydrogen of pieces screened between 0.75 and 0.5 inches affords a quick laboratory test, as hydrogen is available in cylinders. A gas-tight quartz or porcelain tube about 1 inch in diameter (I.D.) and 15 to 18 inches long, heated in a small resistance furnace, such as those used in determining the carbon content of steel, provides the essential equipment needed. Such furnaces usually have a central section of fairly uni-

form temperature in which 5 or 6 test pieces previously heated to expell and free and combined water, can be placed.

A thermo-couple extending into the furnace to about the position of the test specimens can be used to measure the temperature. After the furnace is up to temperature, a heat resistant wire gauze boat containing the samples is moved into the furnace, gradually enough to avoid cracking of the pieces. A rubber stopper connection in the inlet end of the quartz tube can now be placed into position and nitrogen poured through the heating tube to sweep out any contained air. Hydrogen is now introduced by a two-way stop-cock, and the exit gas passed through a medium that will absorb water formed by the reaction:



The gas flow through the tube should be sufficient to maintain a large excess of reducing gas throughout the test. By weighing the water formed at 5 to 10 minute intervals, the rate of which the water is formed becomes a measure of the rate of reduction. Elimination of moisture and oxygen from the compressed hydrogen can be accomplished by well known means.

Since carbon monoxide is the active reducing agent in the blast furnace, it offers certain advantages for use in reducibility tests. When blast furnace top gas is available, the  $\text{CO}_2$  can be absorbed leaving a gas of the composition normally used in the lower part of the blast furnace. Such gas can then be passed through a reduction train and the carbon dioxide formed in reduction can be absorbed in towers filled with soda asbestos. If blast furnace gas is not available, pure carbon monoxide can be generated by passing carbon dioxide through a heated column of charcoal. Air can be used in place of carbon dioxide if a mixture of carbon monoxide and nitrogen is desired for reduction.

Blast furnace conditions are difficult to reproduce or simulate. Results from the laboratory will accordingly provide relative values at best. In other words, some base of reference taken from blast furnace operation must be used. For example in preparing Fig. 6, of attached exhibit A, it was known that blast furnace tests had indicated that crushing to pass 2 inches had proven satisfactory for an ore with a porosity of about 25 percent requiring about 40 minutes for 90 percent reduction in the laboratory. Ore having a porosity of 5 percent required about 80 minutes for a comparable degree of reduction. If a size of 2 inches is satisfactory for the more porous ore, it seems reasonable to suppose that the denser ore should be crushed to pass a screen with 1 inch openings.

By similar reasoning 3 inches would be permissible for ore with 40 percent porosity.

The procurement of representative samples always presents a problem when large tonnages of any raw material of variable character are used. Reducibility apparatus which permit following the rate of reduction by the loss of weight of a test pieces not only permits the use of relatively large samples but also provides a quick accurate means of following the loss of oxygen and the rate of reduction of the sample. Such apparatus has been described by C. E. Wood in Report of Investigations 4 569 of the U.S. Bureau of Mines, October 1949, on the Relative Reducibility of Some Iron Oxide Materials.

Magnetite ore reduces much more slowly than hematite. In fact, the rate of reduction of magnetite ore is so slow that the practice of charging lump magnetite has been abandoned in the United States and in Sweden. Elimination of gangue by crushing and magnetic separation has been an important factor in adopting magnetic separation. It appears, however, that the combined effects of the elimination of silica together with the feasibility of producing high grade sinter as at the Kamaishi Plant of Fuji Iron and Steel Co. Ltd, warrant reducibility tests on Kamaishi lump ore. It appears certain that magnetite ore should not be charged in sizes larger than one inch. Table 2 of the appendix shows that about 43.0 percent of Kamaishi lump magnetite falls between 1.6 inches. This is much too coarse for this type of ore which could be improved chemically as well as reducibility wise by concentration and sintering. The current practice is to use Hongkong ore (magnetite) as coarse as 4 inches. At the Tsurumi Plant 11.3 percent of the Hongkong ore is larger than 100 mm (4 inch) and about 79.6 percent is larger than 40 mm (1.6 inch). This Hongkong magnetite also contains a considerable amount of silica which probably should be decreased by crushing and concentrating at the mine. But with no porosity or reducibility data available, it is impossible to make any recommendations as to the crushing practice for any specific ore. From the appearance of the ore upon visual inspection, porosity and or reducibility tests should also be made on those which follows: Calcutta, Utah, Larap, and Samar. Finer crushing of these ores would not only facilitate reduction, but would also provide more ore surface for heat absorption. Certain ores are subject to cracking upon heating and may accordingly be charged in somewhat larger sizes. This tendency to crack usually becomes apparent during reducibility tests.

### Reducibility Tests on Sinter

Inasmuch as the character of sinter can be modified by changes in sintering practice, it would be desirable to make comparative reducibility tests of sinter to establish for each plant those characteristics which give the optimum reducibility. Once these characteristics were established, visual examination would give a general indication of reducibility. If the very slow rate at which hard burned glassy sinter reduces were generally known there would be greater incentive to modify the sintering practice in Japan. The one exception is the excellent sinter produced from magnetic concentrates at the Kamaishi Plant.

### Agglomeration of Fine Iron Ore

The elimination of fine ore particles smaller than about  $3/8$  inches by suitable methods of agglomeration affords one of the best means of improving blast furnace practice. As shown in Figure 4 of the attached paper, the resistance to gas flow increases very sharply as the particle size falls below  $5/8$  inches. Mixtures of fine and coarse particles are particularly undesirable because they result in a minimum percent of voids. (The volume percent of voids is at a maximum in beds of uniform particle size.) The rather rapid decrease in void volume resulting from the addition of coarse particles to beds of fine particles may be seen in Figs. 20 and 21 of the attached paper. Large particles act as single solid obstructions to gas flow, and increasing amounts will continue to reduce the percent of voids as long as the mixture contains sufficient fines to fill the voids between the large lumps, but further admixture of large particles will increase the percent voids. Mixture of particles of approximately the same size — for example, a mixture of 1 inch and 0.5 inch pieces — contain about the same percent of voids for various proportions of each size.

### Sintering Practice in Japan

The necessity of sintering high moisture, sticky, sulphur bearing limonite iron ore is having a very detrimental effect upon the quality of sinter produced in Japan. This difficulty arises largely from the difficulty in screening wet sticky ore at a size small enough for good sintering and for the effective removal of sulphur. The only practical means of screening such ore at about  $5/8$  of an inch is by preliminary drying or by the use of electrically heated screens. One plant in England is drying down to 10 percent moisture to facilitate screening at  $5/8$  of an inch. Heated screens have been used successfully at the *M. A. HANNA* Com-

pany's Portsmouth mine, located at Crosby, Minn. The heated screen is made by the *W. S. TYLER* Company, 3615 Superior Avenue, Cleveland 14, Ohio. A large amperage of electric current, applied at 7.0 volts, heats the screen wire sufficiently to prevent clogging. About 0.1 kWh of power is consumed per ton of ore screened. A 5 foot  $\times$  14 foot vibrating screen with  $5/8$  inches by 5 inch openings will handle from 175 to 25 long tons of ore per hour with a maximum of 25 percent moisture. Heated screens have also been used successfully in screening wet sticky imported hematite ore such as Dungun. Such ore is difficult to handle and tends to disturb uniform charging and in turn uniform arrangement of the charge in the furnace or sintering plant.

The difficulty of screening wet sticky limonitic ore raises the question as to whether all high sulphur ore of this type should be crushed to a size suitable for sintering. The answer to this question, will depend largely upon the degree to which sulphur can be eliminated by sintering pieces smaller than  $5/8$  inches. If most of the sulphur exists in the fine sizes there is no need for crushing the entire ore. Crushing should however be ample to permit removal of about 98 percent of the sulphur when the minus  $5/8$  inch pieces are sintered.

The high specific heat and high latent heat of evaporation of water necessitate considerable heat transfer either during the sintering of high moisture ore or during its heating and reduction in the blast furnace. Pieces of ore with up to 15 percent moisture in the interior and with steam issuing from their pores, heat very slowly. If the pieces are much larger than  $5/8$  inches they will not be heated properly during sintering. It is a common occurrence for sintering mixtures in Japan to contain pieces of limonite larger than 1 inch. Such pieces are not sintered properly. Moreover they tend to segregate in storage bins and charging machines and set up irregularities in the permeability of the sinter bed. They also tend to reduce the overall permeability of the bed because of their relative coarseness. Much of the sinter produced in Japan will be of poor quality as long as the practice of charging ore larger than  $5/8$  inches to the sintering machines continues.

### Permeability of Sinter Bed

There is a great need for closer control of the important factors that influence the rate of burning and the uniformity of burning throughout the bed of material being sintered. This lack of control results in lack of uniformity and in some cases in the production of much unsintered material

† 語句の脱落があると思われる。



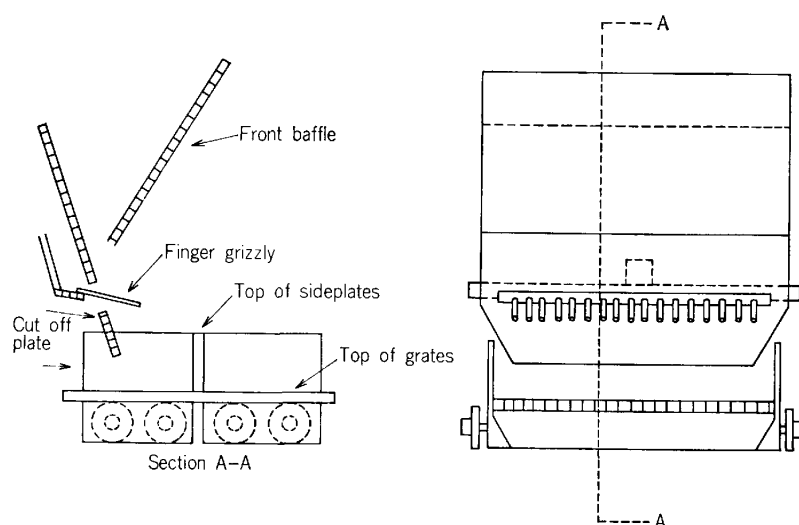


Fig. 1. Details of a finger grizzly.

along with some over-burned material. Results from attempts to sinter large amounts of limonite ore particularly poor at the Muroran Plant of the Fuji Iron and Steel Co. Ltd<sup>†</sup>. Elimination of coarse sizes will correct some of these irregularities but more complete control over permeability of the bed must be obtained to get the best results.

As pointed out earlier in the report, the rate of sintering depends upon the rate at which carbon is burned. Air leakage increases as the over-all resistance of the bed increases. Moreover the volume capacity of the fan is reduced when it work against higher suction caused by higher resistance of the bed to air flow. The rate of air drawn through the bed can accordingly be increased by increasing the permeability of the bed. If the sintering rate is to be uniform throughout the bed it must be of uniform permeability.

High moisture sintering mixtures tend to pack as they are deposited on the bed from the charging machine. With the exception of Plant No. 2, the moisture in the sintering mixtures ranges from 12 to 17 percent. (See Table 4 of Appendix). Some provision should be made to break the fall of the material as it is discharged onto the bed. Various ways are available to accomplish this result such as the Morgan Fluffer which is used extensively in the States. A paper describing this fluffer mechanism was presented before the April 1951 Meeting of the Blast Furnace, Coke Oven and Raw Materials Conference of A.I.M.E. Details can also be obtained from Arthur G. McKEE and Company, 2 300 Chester Avenue, Cleveland, Ohio.

Finger Grizzlies of the type shown in Fig. 1 can also be used for this purpose. Such devices also tend to separate larger sizes and cascade them on

to the grates or to the lower part of the bed. A gradation of sizes with the fines at the top is produced. These devices function more effectively when activated by a vibrator. In addition to improving the permeability of the bed, this disposition of the coarse pieces tends to increase the life of grates.

### Size of Fuel

Uniform distribution of the fuel requires that it be crushed to minus 0.125 inches or minus about 3 mm<sup>†2</sup>. The coke used in Japanese sintering practice will range from 5 mm × 0 to 10 mm × 0, which top size is considerably coarser than is desirable. Finer crushing of the fuel would the sintering practice in all of the plants in Japan except at Kamaishi<sup>†3</sup>.

### Return Fines

Grizzly screens in general do a very poor job of screening sinter. Due to their inherent inefficiency, they are usually made with larger openings than would be used in better screens. They are the source of a large amount of fines in the sinter product and also the cause of the unnecessary recycling of much sinter which is large enough for use in the blast furnace.

Return fines arise from rejected material because of inefficiencies of the sintering process and from a deliberate recirculation of good product in order to assist the sintering process. Calcined but unsintered materials produced because particle size is too large, or because of a localized shortage of fuel in parts of the bed of material, due to size segregation, decrease the effective output of the machine, and add emphasis to the importance of adequate mixing, proper fineness of fuel and dangers

††<sup>2</sup> 語句の脱落があると思われる。

†<sup>3</sup> would の後に "improve" が入るとと思われる。

of size segregation. The return of small sizes of sinter may be an asset to the process, but the return of unsintered material is a liability. Poor sintering practice tends therefore to be self-perpetuating which points to the need of careful control.

**Cooling and Handling of Sinter**

It is generally assumed that the last few feet of the sintering machine are used for cooling. If this is so the machine is not being operated to its full capacity. Moreover, fluctuations in the permeability of the bed and in the amount of air drawn through it are such that the sintering rate is so variable that it is quite impossible to end the sintering process at any fixed point along the machine. Water cooling has therefore been adopted. It is now known, however, that such treatment is the cause of excessive breakage as shown by the following screen tests on air cooled and water cooled sinter from the same operation.

Screen tests air cooled.  
(Cumulative percents, except minus 1/16")

Test No.	+4"	+2"	+1 1/2"	+1"	+3/4"	+1/2"
1	10.9	26.6	34.6	45.6	57.2	70.5
2	17.0	40.2	51.8	63.7	72.3	82.8
	+1/4"	+1/8"	+1/16"	-1/16"		
1	93.0	95.2	98.6	1.4		
2	96.2	97.9	99.3	0.4		

Screen tests water cooled.

1	1.0	3.2	8.8	18.8	36.9
2	0.5	0.9	2.2	7.6	28.4
1	83.0	88.3	98.2	1.8	
2	77.9	84.2	95.5	4.5	

These results were obtained by cooling alternate pallets. One was subjected to a continuous stream of water, "water cooled" and the next one was allowed to cool without artificial means, "air cooled". About 77 percent of the "air cooled" product was larger than 1/2 inch, compared with a similar value of 33 percent for the "water cooled" sinter. Such a difference would not necessarily hold for all types of sinter but the results do show that water cooling makes the sinter brittle and susceptible to breakage on handling. Screen Samples of sinter made at the plant of the Kaiser Co. in California, showed 23 percent of the sinter was minus 1/8 inches before air cooling practice the minus 1/8 inch material has been reported as low as 3 percent and has seldom exceeded 8 percent†.

Mr. J. D. SAUSSAMAN, Superintendent of Blast Furnace has described the rotary cooler of the Kaiser Plant as follows:

"The rotary cooling table is a large, circular doughnutshaped bin, 46 feet in diameter and capable of carrying approximately 200 tons of

sinter. This rotary cooler was designed and built by the Kaiser organization. We feel that the introduction of this unit is one of the outstanding developments in sintering plant design in recent years. The side is 8 feet high but resembles a collar with 1 foot, 6 inch opening at the bottom through which the stationary plow extends. As the table rotates at the rate of 400 feet per hour, cool sinter is removed from the bottom of the table by the plow while hot sinter is fed onto the top of the table. The hot sinter stays on the table approximately two hours during which time it cools sufficiently to prevent it from damaging the conveyer which carry it to the stockpile. A small amount of water is used at the cooling table to control the dust but none is used for the purpose of cooling. The sinter stock pile is 275 feet long and has a storage capacity of approximately 12 500 tons. A conveyor is provided under the entire length of the pile onto which the sinter can be fed through any one of the eleven discharge gates located in the coiling of the conveyor tunnel. This conveyor carries the sinter through an automatic sampling station and then to the furnace bins without any further handling."

The use of such a cooling system is particularly applicable to Japan because it was developed to eliminate sulphur and facilitate the production of soft burned easily reduced sinter for the use in a blast furnace operating on coke made from high volatile coal. In most of the plants in Japan, the sinter is water quenched and then subjected to rough handling. This leads inevitably to excessive breakage and to a tendency to over-burn the sinter to make it strong rather than reducible and to the loss of much of the potential advantage to be gained from good sintering practice.

**Suction**

The suction obtained by the fan at any time during normal operation is determined by the resistance offered by the sinter bed. It follows therefore that the principal control of suction must come through the raw materials and the manner in which they are deposited on the sintering grate. Since suction is an effect and not a cause, efforts to control the process should center around those factors that control the uniformity and overall permeability of the bed. This has been discussed in this report under the sub-title, "Permeability of Sinter Bed".

**Ignition**

Uniform ignition is essential to good sintering practice. Poor ignition was observed in a number of plants as a result of low gas pressures or poor

† この文章には語句の脱落があると思われる。

condition of burners. The results of poor ignition are easily detected and need no elaboration. A thin layer of sawdust is spread over the pan in one plant to facilitate uniform ignition. All that is necessary is to heat the coke particles in the bed up to about 450°C, the ignition temperature of coke.

A strong penetrating ignition acting under the effects of flame pressure from the ignition furnace and suction from the main fan may burn out the carbon in the top few inches of the bed before sintering has had time to take place.

### Moisture Content

Within certain limits moisture promotes the permeability of fine material by its action in promoting the formation of balls or loosely held aggregates of fine particles. Water beyond the optimum amount required to form such aggregates serves no useful purpose and requires fuel for its evaporation. Sinter mixtures containing high percentages of water are susceptible to packing on the bed which means that the water should be carefully controlled. Because of its high specific heat and the relatively large amount of heat required for evaporation, excess water increases the fuel requirements very sharply. For these reasons it has a strong influence upon the temperature reached in the bed and should be carefully controlled.

The use of a rapid moisture tester of the calcium carbide type is extremely useful in controlling this important variable in sinter practice.

### Sintering Plant Maintenance

Neglected maintenance asserts itself in production difficulties and in poor quality of sinter. Many maintenance faults are reflected sooner or later in return fines practice. Holes and gaps in grate bars return more fines. Wide gaps at the end of Dwight-Lloyd machine, causing severe impact between pallets, tend to increase damage to pallets (thus also making more return fines) and to cause greatly increased abrasion of the fan impeller. Incorrectly trained belt conveyors lead to the segregation of coarse pieces. Worn-out fan impellers lead to lower outputs. In fact there is hardly a piece of equipment around a sintering plant which will not react adversely on sinter quality if neglected. Maintenance in most of the sintering plants in Japan could be greatly improved.

### Integration of the Sintering Plant with Crushing and Screening Equipment

The original sintering plants designed to treat a steady flow of uniformly fine materials have led to over simplified conception of the present role of

sintering in iron ore preparation. In modern practice the sintering machine is an integral part of an ore preparation plant. It must be supplemented with suitable crushing and screening equipment and its operation must be closely integrated with these facilities. There is a great need in Japan for an extension and rearrangement of ore handling, crushing, and screening, to provide a more regular and properly sized feed to the sintering plant. Unfortunately, the material available is not always used to the best advantage. The rather common practice of sending the central part of cone shaped piles to the sintering machine should be abandoned.

### Sinter Quality

One Plant in Japan is making sinter of high quality. In justice to other plants it must be pointed out that the use of a mixture composed largely of magnetite concentrates, magnetite fines and some iron sand is a large factor in the success of this plant. The sinter produced in this plant has all the characteristics of good sinter. It is relatively free from glazed surfaces, it is relatively low in FeO, and because of its physical make-up and the presence of a large part of the iron as hematite it no doubt is easily reduced. Due partly to air cooling and careful handling it is charged in much coarser sizes than the over-burned water quenched glassy and roughly handled sinter made in most of the other plants in Japan.

### Reducibility Tests on Sinter

Reducibility tests on sinter should be made to get a perspective of how widely this property can change. With a little experience it is not difficult to detect the characteristics and appearance of easily reduced sinter in contrast to material that is difficult to reduce. Once the appearance of easily reduced sinter is established much could be accomplished by training sintering plant operators to recognize the appearance of easily reduced sinter and by encouraging them to modify their practice to produce this type of product.

### Nodulizing of Fine Iron Ore

Rotary kilns have been used to agglomerate iron ore for many years. The process has not however been used widely due largely to problems associated with the formation of rings or accretions on the kiln lining. As the charge moves through the kiln towards the fire end, it is heated to the softening or sticky point where upon small masses collect into roughly spherical shape as a result of the rolling action in the kiln.

Several cement kilns were used to nodulize fine iron ore during the World War II in the States

but these operations were abandoned when hostilities ceased. It should be noted, however, that the U. S. Steel Corporation, through its Oliver Iron Mining Company, will operate a new rotary kiln near Virginia, Minnesota, this summer and autumn. Fines will be screened from a rather high moisture ore of fine structure. Some saving in freight, which amounts to \$2.40 per long ton from the mine to Lower Lake Ports will be realized through the elimination of moisture. A Dwight Lloyd sintering machine will also be operated at the same site for the purpose of comparing the results obtained with sintering and with nodulizing. As the plants of the Oliver Iron Mining Company to treat taconite develop, it appears likely that magnetic taconite concentrates may be mixed with material screened from fine Mesabi ore for agglomeration by sintering or by nodulizing. The results of these tests will be of great interest to the steel industry in the States and elsewhere, the taconite being a very low grade magnetic iron ore containing a large proportion of hard siliceous material.

**Nodulizing Tests of the Kokan Kogyo Mining Company on Jarosite-Limonite Iron Ore**

Experimental work has been undertaken by this company to study the nodulizing of mixtures of Jarosite  $K_2O \cdot 3Fe_2O_3 \cdot 4SO_3 \cdot 6H_2O$ ) and limonite from the Gumma mine estimated to contain 400 000 tons of Jarosite and 1 500 000 tons of limonitic ore. A powdered lignite fired rotary kiln 3.93 ft in diameter and 57 feet long, in which maximum temperatures from 1 200 to 1 250°C are reached, has been used in experimental work. Two larger kilns scheduled for operation in the fall are under construction at the Gumma mine. These kilns, each with a rated capacity of 100 tons per day, will be 125 feet long with an I.D. of 6.56 ft. and an O.D. of 7.55 ft.

Over a test period of 61 days, the following materials were processed in the experimental kiln:

Limonite and Jarosite	395.88 M.T.
Limonite	78.25 M.T.
Limonite fines	271.55 M.T.
Total	745.68 M.T.

Chemical composition of typical ore and nodules.

	Ore	Nodules
Iron percent	43.24	58.24
Sulphur percent	4.98	0.97
Phosphorus percent	1.04	1.20
Silica percent	5.98	10.23

Return fines amounting to 51.14 tons were recycled during the production of 479.50 metric tons of nodules, giving a final product yield of 64 per-

cent. The iron increased 34.7 percent and 80 percent of the sulphur was eliminated. During the test period 137.3 metric tons of powdered lignite was used as heating fuel. A total of 44.7 tons of coke breeze, amounting to about 6 percent of the ore, was mixed into the charge during the test period. The reducing action of this fine coke produced nodules containing as much as 45 percent ferrous oxide.

While the use of coke in the charge may facilitate softening by the fluxing of FeO and SiO<sub>2</sub>, the use of a reducing fuel is not desirable as it prevents the oxidation of sulphur which averaged 0.9 percent in the product for the 61 day test period.

Laboratory tests made in a small electrically heated furnace without the use of coke indicate that a very good desulphurization can be obtained if coke is omitted from the charge.

Results from Laboratory Experiments on Limonite-Jarosite Ore from the Gumma Mine (minus 50 mesh)

Percentage composition of original ore.

Fe-	42.94	Al <sub>2</sub> O <sub>3</sub> -	1.65	K <sub>2</sub> O-	3.75
S-	5.59	CaO-	0.137	P-	0.885
SiO <sub>2</sub> -	3.43	MgO-	0.118	Loss on ignition-	27.05

Percentage composition of product.

Temp°C	Fe	FeO	S	K <sub>2</sub> O	SiO <sub>2</sub>
1 200	59.37	3.36	0.07	4.59	4.79
1 250	59.45	4.25	0.05	4.72	4.88
1 300	59.69	3.06	0.01	4.72	4.78
1 350	59.81	3.94	0.01	4.43	4.78
1 400	61.34	23.64	0.04	3.18	5.76

Data from Kokan Kogyo Chemical Laboratory

These results show an excellent removal of sulphur when the coke is omitted from the feed. They also indicate that, except for the K<sub>2</sub>O, a very acceptable product chemically can be made by dehydrating limonitic ore. Volatilization of the K<sub>2</sub>O during nodulizing and subsequent recovery would improve the product and enhance the economics of treatment. Several preliminary tests with the experimental kiln without the use of carbon and with small additions of limestone have given very encouraging results in kiln operation and in as low as 0.093 percent of sulphur in the nodules.

Nodulizing Tests with No Coke Breeze in Charge

Chemical Analysis, Percent

Test No. 1, June 19 to 21, 1951				
Material	Fe	FeO	S	CaO
Ore	45.24		3.69	
Nodules	58.13	8.53	0.23	0.58
Test No. 2, June 22, 1951				
Ore	42.18		3.34	2.95
Nodules	55.66	8.23	0.09	1.82
Test No. 3, June 23, 1951				
Ore	43.52		3.75	1.23
Nodules	57.50	8.34	0.16	1.84

Note: The nodules in previous tests contained 0.9 percent sulphur.

In view of the fact that no difficulties were encountered when coke breeze was omitted from the charge and small amounts of limestone were added, the low sulphur content of the nodules made in these test is very encouraging.

The nodules are high in iron, and low in silica, and appear to be a very satisfactory material for use in blast furnaces. There is every indication that very poor limonitic ore can be converted into a good blast furnace feed by nodulizing. If domestic limonites were nodulized at the mine, much more sinter could be produced with existing equipment at the furnaces.

Porosity and reducibility tests on the product would give valuable information on its fitness physically for use in the blast furnace. Particle size would be satisfactory and the bulk density would exceed that of sinter. Nodules would be preferred to sinter of the same reducibility and chemical analysis because of their higher weight per cubic foot and consequent higher heat capacity per unit of bulk volume.

### The Pelletizing Process

The pelletizing process developed at the University of Minnesota to agglomerate fine magnetic concentrates produced in the concentration of taconite, is described in the attached report. Briefly the process consists of the balling up of fine damp material (8 to 10 percent  $H_2O$ ) in a rotating drum and the subsequent drying, controlled heating and baking of the moist balls in a shaft furnace having an upward current of hot gases. The baked balls are called pellets. Due to the difficulty of sintering material 70 to 80 percent of which is finer than 325 mesh, a new method of agglomeration was needed. The process has been undergoing development for the past 15 years and when properly carried out will produce excellent material for open hearth charge ore and for the blast furnace. Pellets have a higher bulk density than sinter, and are more easily reduced because bonding is mainly accomplished by an intergrowth of grains rather than by slagging as in the typical sinter. The bonding temperature of pellets is lower and under better control than is practical in the sintering process.

Granular material such as magnetite which is free of slimes must be ground fine before it will collect into balls of satisfactory strength in the forming drum. The fineness of grinding is determined by amount needed to unlock silica before the ore is concentrated. It has been observed, however, that, the presence of fine colloidal material facilitates the forming of balls and also raises the strength of the balls after drying and before a

bond has developed by heating. Small amounts of bentonite are being used effectively as a binder in some pelletizing plants in the States. The tendency of wet sticky ores such as the limonite of Japan to ball up on conveyor belts and during other methods of handling is well known. This tendency to form balls which is due to the clay like characteristics of limonite has been observed in the sinter-feed mixing machine at the Manari plant of the Imperial Iron Company, Japan.

Some of the balls retained their identity after passing over the sintering pan and could be seen in the sinter.

Due to the difficulty of screening high moisture limonites such as Gumma and Dungun ores at sizes small enough to permit good sintering practice and a satisfactory removal of sulphur, the application of the pelletizing process to such material should be investigated. Small scale laboratory tests would show whether sulphur and arsenic can be eliminated and whether the balls have sufficient strength between 200 and 600°C for their burning without breakage due to the load they must bear in a shaft furnace before complete bonding occurs. The presence of a few tenths of one percent of carbon in the pellets during the early stages of heating may promote the removal of arsenic. Such amounts of carbon as fine coke are being added in some of the pelletizing plants in the States.

Pelletizing is much more efficient thermally than sintering which is an inefficient process because both the gases and the product leave the machine at relatively high temperatures.

### Pelletizing Iron Sands

Some interesting, well-planned and significant tests have been conducted on the pelletizing of iron sands at the Yasugi Works of Hitachi Ltd. Plans are developing to use pellets rather than iron sand concentrates in a small charcoal blast furnace. Torigami white pig iron of high purity, made in the charcoal blast furnace, is used as melting stock in electric furnaces producing high speed steel and other quality products.

Experimental work by Hitachi Ltd has demonstrated the feasibility of balling up iron sand concentrates and also that when these balls are baked at a temperature of about 800°C the pellets have ample strength for use in a seven ton charcoal blast furnace with a rectangular hearth. The development has included careful studies of the grain size of iron sand concentrates, as well as the dropping strength of pellets roasted at temperatures from 200 to 1 000°C. The iron in the pellets occurs largely as hematite after roasting, which makes them more easily reduced than unroasted pellets in

which most of iron is present as magnetite. It is of interest to note that iron sand concentrates contain sufficient colloidal material to permit balling and the development of as much dropping strength at 500°C as is developed when heated to 1 000°C. The increase in strength with increased temperature must, however be determined for each kind of material.

Similar tests on arsenic bearing limonites would be of great interest as they would indicate whether the temperature and atmosphere which are subject to rather close control in the pelletizing process can be adjusted to remove arsenic as well as sulphur. For further details on the pelletizing process see the attached report, "Agglomeration by the Pelletizing Process.

### The Evaluation of Iron Ore from Chemical Composition

The value of iron ore depends upon its iron content and the amount of gangue or harmful elements associated with the iron. Iron content furnishes a good basis for evaluation if proper adjustments are made for additional transportation costs as the iron decreases, and for extra smelting costs when processing siliceous iron ore.

A net ton of pig iron (about 92 percent Fe) contains approximately 1840 pounds of iron, which will require for its production the use of 1.64 long tons of 50 percent ore or 1.37 tons of 60 percent ore.

$$\left( \frac{2\ 000 \times 0.92}{0.50 \times 2\ 240} = 1.64 \text{ and } \frac{2\ 000 \times 0.92}{0.60 \times 2\ 240} = 1.37 \right)$$

If a ton of the 50 percent ore is worth \$ 5.00 at the mine and the total freight to the furnace is \$ 10.00 the 60 percent ore can be appraised at \$ 7.95 per ton at the mine without increasing the ore cost per ton of pig.

<u>50 percent ore</u>	
1.64 tons × \$ 5.00 =	\$ 8.20
1.64 tons × \$ 10.00 =	\$ 16.40
Ore cost N.T.Pig =	\$ 24.60

<u>60 percent ore</u>	
1.37 tons × \$ 7.95 =	\$ 10.90
1.37 tons × \$ 10.00 =	\$ 13.70
Ore cost N.T.Pig =	\$ 24.60

Because of lower freight the 60 percent ore is worth about \$ 2.95 more per long ton at the mine. Thus the 50 units of iron in the 50 percent ore

will be worth only 10 ¢ per unit, compared to 13 1/4 ¢ per unit for the 60 percent ore. Such an increase in value per unit of iron should be recognized and some reasonable part of it should be made available to mine operators as an incentive to improve their product by concentration. Once some base value for iron content is established, it is a matter of straight forward calculations to appraise the value of an iron ore in terms of transportation costs. Such an appraisal is particularly important when transportation cost are high.

The silica and alumina often dilute the iron, but moisture can also act as a diluent. After the value of the ore at the mine is appraised in terms of iron content and transportation charges, penalties should be applied for high gangue content, normally silica. In the case of the 50 percent ore cited above, each percent of silica contributes about 37 pounds the 1.64 tons of ore required per ton of pig (1.64 × 2 240 × 0.01 = 36.8). Since the slag contains about 35 percent silica, the volume of slag per ton of iron will increase by about 105 pounds for each percent of silica added to the ore, assuming that the additional silica is accompanied by the usual proportion of alumina and that the basicity of the slag is unchanged. Applying a figure of 0.6 pound of coke per pound of slag, the additional 63 pounds of coke will increase the cost of pig iron by about \$ 1.10 if coke costs \$ 35.00 per net ton or ¢ 1.75 per pound. The additional 100 pounds of limestone will add a further cost of about 11 ¢ per ton of pig if the flux costs \$ 2.40 per long ton. Costs above raw materials (also known as conversion or processing costs) that increases as output decreases, will increase about 3.5 percent (63/1 800) or \$ 0.14 if such costs total \$ 4.00 per ton of pig iron. We see therefore that an additional percent of silica in the ore will, under the conditions assumed, increase the cost of pig iron by \$ 1.35. Since these additional costs arise from the use of 1.64 tons of ore per ton of pig, the cost per unit of additional silica in the ore will be 82 ¢.

The calculations should, of course, be modified to meet the costs and practices of individual plants. It is very clear, however, that when using expensive coke the costly effect of siliceous ore and high slag volumes should be appraised most carefully. Such problems should be studied by industry wide committees who could work out standard procedures for evaluating iron ore. (次号に続く)