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PREREDUCED IRON ORE PELLETS: PREPARATION, PROPERTIES, UTILIZATION

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PREREDUCED IRON ORE PELLETS: PREPARATION, PROPERTIES, UTILIZATION

By M. M. Fine, N. B. Melcher, and others

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PREFACE

In 1961 the Bureau of Mines, Twin Cities Metallurgy Research Center, initiated a study of a new agglomerated raw material: Partially reduced, hardened iron ore pellets. The term "prereduced pellets" was selected as most descriptive of a versatile product which could be adapted to either ironmaking or steelmaking. Started in the laboratory, the investigation was translated smoothly to a small pilot plant and then to the Bureau's experimental blast furnace, Bruceton, Pa. Although the research is still (1969) in progress on a smaller scale and technology may be improved, the major objectives have been accomplished. This is, therefore, an appropriate time to make a permanent record of the noteworthy facts and findings in the form of a Bulletin.

The Bulletin has been subdivided into seven parts preceded by an abstract:

Part 1 reviews pelletizing technology and presents the case for prereduced pellets.

Part 2 reveals the laboratory technique for making prereduced pellets from a variety of iron ore concentrates and solid reductants.

Part 3 describes the problems of production in a small pilot rotary kiln. Approximately 140 tons of pellets were manufactured for subsequent evaluation in the blast furnace.

Part 4 includes data from blast furnace smelting of the 140 tons of prereduced pellets as well as some from an industrial source. For comparison, experiments with other prereduced burdens are included.

Part 5 appraises the economics of hot-metal production from reduced and unreduced iron ore pellets.

Part 6 summarizes the present state of the industrial art. Much of industry's interest in prereduced pellets has been inspired by the Bureau's pioneering efforts.

Part 7, the concluding section, is a brief glimpse into the future.

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PREREDUCED IRON ORE PELLETS: PREPARATION, PROPERTIES, UTILIZATION

by

M. M. Fine,[1] N. B. Melcher,[2] and others

ABSTRACT

A process for simultaneously reducing and indurating iron ore pellets was developed by Bureau of Mines laboratory research and confirmed on a continuous pilot plant basis. Moist iron ore concentrate was balled, mixed with solid reductant, and heated at about 1,100° to 1,150° C. On a small scale, a variety of concentrates and fuels were utilized successfully. The pilot rotary kiln tests were limited to natural magnetite, synthetic magnetite, lignite, and anthracite. In both cases, partially reduced pellets with excellent physical properties were regularly produced with average analyses of at least 80 percent total iron and 60 percent or more metallic iron. These and other prereduced pellets were smelted in the Bureau's experimental blast furnace facility, Bruceton, Pa. With prereduced spheres comprising 85 percent of the burden, the coke rate was approximately half that with standard oxide pellets and production of hot metal was increased by 75 percent. Economic appraisal of the cost of making hot metal indicates that reduced pellets could be more profitable than any other iron raw material in both investment capital and daily cost of operation.

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PART 1. — THE CASE FOR PREREDUCED PELLETS

By M. M. Fine

In the last decade, iron ore pelletizing in the United States has been highly successful. By 1967, 12 years after the first commercial shipment, output had reached 41 million long tons per year. The industry is beginning to acquire worldwide recognition with the construction of pellet plants on other continents. It has been estimated that in 1975, total pellet-production capacity will be 190 million long tons, 71 of which will be in the United States.[3]

Iron ore pellets are now widely recognized and accepted as being superior to natural ore. Pellets are higher in grade and bulk density, and much more uniform in structure and composition. Substitution of pellets for other raw materials in the blast furnace has substantially increased productivity and owing to the higher iron content[4], has decreased coke consumption. Some of the added productivity was realized from the greater volume of wind blown through the highly permeable bed.

There are three principal commercial methods for manufacturing pellets; each method has several variations. Although the unit operation of pelletizing is basically simple, there are several variables which may affect the product quality. These are chemical and physical properties of the ore, duration and temperature of induration, and type of apparatus. The finely ground, moist iron oxide, containing about 10 percent H_2O, is intimately mixed with slightly less than 1 percent bentonite binder and fed into a rotating drum, cone, or disk pelletizer. As the seeds rapidly nucleate, the balance of the mix forms around them to build a rolling mass of green balls. These are screened, if necessary, to separate the preferred size, about 3/8 to 1/2 inch.

In all processes, the green pellets are dried and fired, reaching a maximum temperature of about 2,400° F. Differences in the indurating vessels distinguish the schemes from each other. Shaft furnaces are featured in one process,[5] and traveling grates are featured in others.[6] There are variations in grate machines as well, depending on the draft direction and whether provisions are made to introduce side and hearth layers of fired pellets. Grates are also used in combination with kilns to effect drying and partial induration. The pellets then are calcined at higher temperatures in a rotary kiln to attain the ultimate hardness.[7] A fourth process recently entered the competition and had a trial of about 1 year in a pelletizing plant constructed on the Minnesota iron range.[8] Green balls are conveyed through a convection dryer to a rotary hearth turntable furnace where the pellets are brought to a state of semi-induration by radiant heat. The final step is induration in a vertical stove where the pellet bed moves countercurrent to the flow of air.

Since the operation is conducted in a strongly oxidizing atmosphere, the product pellets are essentially hematite, and when magnetite is the raw material, virtually complete conversion to hematite is indicative of successful induration. One of the significant conclusions resulting from a study of pelletizing variables was that stronger pellets are more oxidized.[9] Thus, pellets prepared in a reducing atmosphere might have inadequate strength.

Manufacturing technology for iron oxide pellets is well established, and these agglomerates are, at the moment, the best available blast furnace burden; however, these pellets can be improved by prereduction; that is, by removing oxygen from ferric iron oxide.[10] As a raw material,

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[3] Harrison, H. S. The Changing Iron Ore Industry. Blast Furnace and Steel Plant, v. 54, No. 1, January, 1966, pp. 40–43.

[4] Haley, K. R. Operating Results Using Taconite Pellets on Armco's Middletown Blast Furnace. Proc. Blast Furnace, Coke Oven, and Raw Materials Conf., Philadelphia, Pa., April 1961. American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, v. 20, 1961, pp. 15–31.

[5] Morawski, F. P. The Story of the Erie Mining Company—The Pelletizing Process. Min. Eng., v. 15, No. 5, May 1963, pp. 48–52.

[6] Haley, K. M., and W. E. Apuli. Pelletizing on a Horizontal Grate Machine. Ch. in Agglomeration, Interscience Publishers, New York, 1962, pp. 931–957.

[7] Berkhahn, R. W., and D. M. Urich. Grate Kiln Pelletizing Process at Humboldt. Proc. 23d Annual Univ. of Minn. Min. Symp. and Ann. Meeting, Minn. Sec., AIME, Duluth, Minn., Jan. 15–17, 1962. Duluth, Minn., 1962, pp. 25–32.

[8] Smith, W., D. Beggs, and J. Rinker. New Process for Producing Oxide Pellets on the Mesabi Range. Min. Eng., v. 18, No. 9, September 1966, pp. 47–52.

[9] Cooke, S. R. B., and T. E. Ban. Agglomerating Iron Ore Concentrates. Chem. Eng. Prog., v. 51, No. 8, 1955, p. 364.

prereduced pellets are indicative of the evolutionary and revolutionary changes taking place in the quality and grade of blast furnace burdens for the past decade. In 1954, the average analysis of ores from U.S. ranges in the Great Lakes area was 50.86 percent Fe (natural), 10.22 percent SiO_2, and 10.90 percent moisture; 10 years later, the average was 56.67 percent Fe (nat.), 8.13 percent SiO_2, and 6.17 percent moisture. The advance in Canada was even more pronounced in a shorter time. In 1957, the average product of Canadian ore-producing areas was 51.62 percent Fe (nat.), 8.59 percent SiO_2, and 8.02 percent moisture; in 1964, the average grade was 59.59 percent Fe (nat.), 6.27 percent SiO_2, and 3.94 percent moisture.[11] These dramatic data are testimony to the giant strides made in iron ore improvement. A logical next step is prereduction.

Prereduction can be considered a beneficiation step in which oxygen is the gangue. Since the ultimate objective of milling and smelting is iron metal, nonmetallic constituents in general, whether oxygen or silica, are contaminants. The research aimed at lowering the silica content of concentrates is succeeding. The Grade Name listings of the American Iron Ore Association[12] show that virtually all commercial pellets contain less than 8 percent silica, many are lower than 6 percent, and some are below 4 percent. When technical evolution increases the number of pellets in the last category, further mechanical concentration will have reached the point of diminishing returns. At this level, oxygen would constitute the only major diluent remaining to challenge the researcher.

There are several factors that potentially favor prereduced pellets over more conventional agglomerates. First of all, grade may range from 80 to over 90 percent Fe, depending on the intensity of reduction and quality of raw material. This in turn engenders other economies such as a savings in transportation costs, assuming prereduction is done at the mine site. It has been discovered that induration takes place at a lower temperature in a reducing environment, so that maximum operating temperature in manufacturing prereduced pellets could be 300° to 500° F less than that for oxide pellets. Secondly, prereduction can be accomplished by low-cost solid fuels which, in effect, replace some of the coke later required in the blast furnace. Prereduced pellets perform exceptionally well in the blast furnace, yielding peak production at minimum coke consumption.

It is interesting to note opinions others have expressed concerning the impact of increasing the metallic iron content of blast furnace feed. In listing various factors that affected coke consumption, Flint[13] said that an increase in charged free metallics would decrease the net carbon rate per ton of hot metal. Although the "free metallics" were primarily scrap, the conclusion should be equally applicable to reduced pellets. Strassburger,[14] in a discussion of " . . . methods which can be used to obtain more iron with lower coke rates . . ." included " . . . a possible super blast furnace burden obtained from direct reduction processes." A similar dissertation by Holowaty[15] suggested that synthetic scrap from direct-iron processes seemed to be " . . . an ideal charge material for the blast furnace." Rice,[16] on the other hand, reasoning by analogy from blast furnaces operating on very high scrap charges, was less sanguine. He predicted as much as a 70-percent increase in unit production, and a steady decrease in coke rate as the proportion of metallized material in the charge was augmented. However, he predicted a minimum, with a 100-percent-metallized charge of about 950 lb coke-equivalent per net ton of pig iron, but warned that operating difficulties with such burdens could render the furnace unworkable.

An appraisal by a French research group concluded that each percent metallization would, under "ideal" operating conditions, increase productivity and decrease coke usage by 0.7 percent. It was also indicated that with natural gas injection, furnace ideality could be maintained up to 58 percent metallization, compared with only 30 percent without it.[17] Agarwal and Pratt[18] approached the problem differently. They studied the possible benefits of prereduced burdens with oxygen removed from the furnace by EGC (endothermic

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[10] Because prereduction is defined as the percent of O_2 removed from Fe_2O_3, prereduction of hematite (that is, of Fe_2O_3) is zero and that of metallic iron is 100 percent.

[11] Skillings Mining Review, Average Analyses—American Iron Ores—1964 Season. V. 54, No. 38, Sept. 18, 1965, pp. 1, 6-7, 26-27.

[12] American Iron Ore Association, Iron Ore, 1967. Cleveland, Ohio, pp. 70-87.

[13] Flint, R. V. Multiple Correlation of Blast Furnace Variables. Proc. Blast Furnace, Coke Oven, and Raw Materials Conf., Pittsburgh, Pa., April 1962, American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, v. 2, 1962, pp. 49-73.

[14] Strassburger, J. H. Use of Oxygen, Moisture and High Top Pressure in Blast Furnace Operation. Journées Internationales de Sidérurgie, Luxembourg, June 18-28, 1958, p. 281.

[15] Holowaty, M. O., and G. H. Craig. The Second Challenge to the Lake Superior District. Proc. 22d Annual Univ. of Minn. Min. Symp. and Ann. Meeting, Minn. Sec., AIME, Duluth, Minn., Jan. 8-11, 1961, Duluth, Minn., 1967, p. 18-A.

[16] Rice, O. R. The Performance of Commercial-Size Blast Furnaces on High Scrap Charges, Being of Possible Significance to the Use of Prereduced Ore. Proc. 23d AIME Ironmaking Conf., Pittsburgh, Pa., April 13-15, 1964, American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, 1965, pp. 247-253.

gasification of carbon), more generally referred to as the "solution loss" reaction: $CO_2 + C \rightarrow 2CO$. Agarwal and Pratt concluded that the lowest coke consumption should be attained with the most highly reduced ore, although this would not necessarily result in the lowest overall energy consumption. For example, if the O_2 removed by EGC was between 10 and 30 percent, the optimum prereduction would be 60 to 70 percent. Similarly, they predicted a production gain of 40 percent at 85 percent prereduction and 45 percent O_2 removal by EGC; they considered productivity increases amounting to 50 to 60 percent highly unlikely.

From the experimental evidence available today, the theorists appear to have been highly conservative in prognosticating the full potential of prereduced pellets in the blast furnace.

Prereduced pellets are attractive for other outlets in iron- and steel-making. Given an exceptionally good ore or concentrate as starting material, it is possible to make highly reduced pellets with 3 to 4 percent SiO_2 or less containing more than 90 percent metallic iron. These could be the ideal charge to a cupola, and a limited amount of research on this concept has been published.[19] Additional work along these lines is being conducted at the Twin Cities Metallurgy Research Center.

A very persuasive case can be made for steelmaking in the electric furnace using reduced pellets (alone or in combination with scrap).[20] This unique idea is about to enter the field of commercial technology; two plants, one in Mobile, Ala., the other in Portland, Oreg., are being constructed to manufacture steel by this method and will go on stream in 1969.[21] Others are being built or planned overseas. There are a number of factors which have contributed to what is apparently the birth of a new industry, but two of the most significant are convenience and cost. Reduced pellets are a uniform raw material, of known and acceptable composition, which are readily adaptable to small-to-medium-scale operation. Furthermore, it is claimed that both capital costs and operating costs are lower for steel using the prereduced pellets-electric furnace method than by way of the blast furnace and basic oxygen process.[22]

Finally, prereduced pellets can be used as a cooling medium in basic oxygen furnaces in substitution for scrap. Presumably, because of the relative sameness in size and shape they would be easier to charge.[23]

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[17] Meysson, N., A. Maaref, and A. Rist. Emploi de Boulettes Préréduites au Haut Forneau (Application of Prereduced Pellets in the Blast Furnace). Étude Graphique, IRSID Station d' Essais, L.M. 65, Re-96, July 1965.

Rist, A., and N. Meysson. Recherches Graphiques de la Mise au Mille Minimale de Haut Fourneau à' Faible Temperature de Vent (Graphic Processes for Determination of Least Consumption of Carbon in Blast Furnaces With Low Blast Temperature). Revue de Metallurgie, v. 61, No. 2, 1964, pp. 121–145.

[18] Agarwal, J. C., and H. R. Pratt. The Thermodynamic Aspects of Using Partially Reduced Burdens. J. Metals, v. 8, No. 2, February 1966, pp. 224–227.

[19] Hafner, R. H., and S. C. Clow. Melting Prereduced Iron-Ore Pellets in the Cupola. Secondary Raw Materials, v. 6, No. 7, July 1968, pp. 7–9, 12.

[20] Tucker, H. A., F. X. Tartaron, H. L. Goldstein, and J. J. Kocisin. Direct Steelmaking With Prereduced Iron Ore in an Electric Arc Furnace. Blast Furnace and Steel Plant, v. 56, No. 12, December 1968, pp. 1070–1084.

Sibakin, J. G., P. H. Hookings, and G. A. Roeder, Electric Arc Steelmaking With Continuously Charged Reduced Pellets. Proc. Internat. Cong. on the Production and Utilization of Reduced Ores, Evian, France, May 1967, pp. 335–350.

[21] Ban, Thomas E., and Donald C. Violetta. D-LM—New Commercial Ironmaking Process. Presented to AISE, Cleveland, Ohio, June 24, 1968, 35 pp. (Available from McDowell-Wellman Eng. Co., Cleveland, Ohio.)

Sturgeon, James H. The Commercial Production of Pre-Reduced Pellets by the Midland-Ross Process. Proc. AIME Ironmaking Conf., Atlantic City, N.J., v. 27, April 1968, American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, 1969, pp. 140–143.

[22] Dailey, W. H. Steelmaking With Metallized Pellets. Proc. AIME Ironmaking Conf., Atlantic City, N.J., April 1968, American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, v. 27, 1969, pp. 132–139.

[23] Von Bogdandy, Ludwig, Heinz-Dieter Pantke, and Ulrich Pohl. The Purofer Direct Reduction Process. Proc. AIME Ironmaking Conf., Pittsburgh, Pa., November-December 1965, American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, v. 24, 1967, pp. 18–22.

PART 2. — LABORATORY PREPARATION OF PREREDUCED PELLETS

By M. M. Fine

Commercial and experimental concentrates from a number of diverse raw iron ore materials have been successfully agglomerated in the laboratory by the induration-reduction technique. Two magnetic taconite samples from Minnesota operations were essentially magnetite with some quartz gangue. Two specular hematites from Michigan jaspilite ore bodies contained hematite and magnetite with quartz and some silicates. The next sample was a flotation concentrate prepared by the Bureau from a western Mesabi nonmagnetic taconite, which was comprised of earthy hematite with goethite in a silica gangue. The final product was a magnetic concentrate from an experimental roasting pilot plant in which nonmagnetic taconite and semitaconite were the feed ores. All were high-grade materials as shown in table 1; sizing analyses are presented in table 2.

Table 1.—Analyses of concentrates and reduced pellets

Concentrate and reductant	Concentrate analysis, percent		Indurated pellets				
			Analysis, percent			Crushing strength, lb	Met. Fe/Fe
	Fe	SiO_2	Fe	Met. Fe	SiO_2		
Magnetic taconite:							
Lignite	64.2	8.8	84.2	80.3	10.8	252	0.95
Bituminous coal	64.2	8.8	82.7	74.4	—	395	.90
Anthracite	66.0	6.7	81.9	56.4	8.2	459	.69
Specular hematite:							
Lignite	63.4	7.9	85.4	80.7	10.5	150	.95
Anthracite	61.6	8.4	75.2	49.4	11.1	113	.66
Nonmagnetic taconite:							
Lignite	60.0	4.9	88.2	82.6	8.0	217	.94
Anthracite	60.0	4.9	80.2	48.8	6.8	334	.61
Synthetic magnetite:							
Lignite	67.4	6.3	88.7	84.2	8.4	396	.95
Anthracite	68.5	4.4	81.9	46.4	5.7	957	.57

Table 2.—Screen sizing of concentrates, weight-percent

Size, mesh	Magnetic taconite 1	Magnetic taconite 2	Specular hematite 1	Specular hematite 2	Nonmagnetic taconite	Synthetic magnetite
Plus 100	1.4	0.6	0.3	0.9	0.0	0.5
100 to 200	2.9	3.9	6.5	8.3	6.1	3.9
200 to 325	10.5	8.8	14.9	21.9	24.1	16.0
Minus 325	85.2	86.7	78.3	68.9	69.8	79.6
Total	100.0	100.0	100.0	100.0	100.0	100.0

A standard pelletizing procedure produces green balls with reproducibly uniform physical properties. A 1,000-gram batch of dry concentrate is carefully blended with 7.5 to 10.0 grams of bentonite and transferred to the stainless steel bowl of an electric household mixer. Water is added slowly with constant stirring to reach an 8- to 10-percent moisture content. The quantities of both bentonite and moisture are functions of the physical and chemical properties of the concentrates. The prepared mix is pelletized in a 16- by 6-inch smooth steel drum revolving at 52 rpm into which moist seed pellets (42 grams, minus 4- plus 5-mesh), made beforehand from more of the same blend, have been placed. The feed material is added gradually over 20 to 25 minutes, accompanied by occasional water sprays from an atomizer to compensate for evaporation. The use of prepared uniform seeds inhibits further nucleation and the number of product pellets is essentially equal to the number of seeds.

In some of the early experiments, soda ash was incorporated with the mix to improve pellet quality; however, with laboratory technique improvements the soda ash was discontinued with no impairment of quality. The previously specified quantities of raw materials yield approximately 250 half-inch green balls. These are (1) screened to exclude the few that are larger or smaller than 9/16 by ½ inch, (2) returned to the drum for 5 minutes, and (3) dried at 100° C. The drum pelletizer is pictured in figure 1.

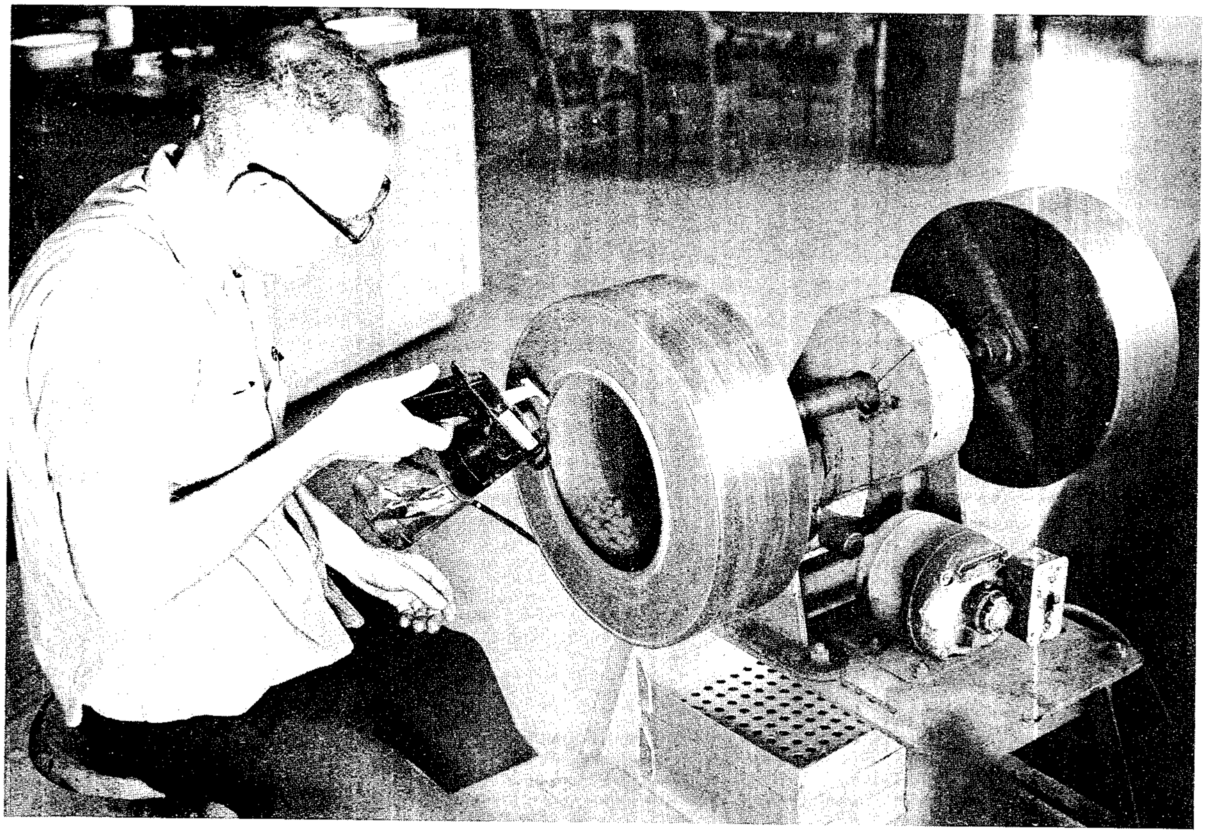

Figure 1.—Laboratory pelletizing drum.

The dry pellets were mixed with reductant (60- and 180-grams per batch, respectively) and transferred to four clay crucibles fitted with loose covers. Table 3 presents proximate analyses of the solid fuels tested as reductants. The crucibles were charged to a muffle furnace which had been preheated to the operating temperature. After a timed interval, the crucibles were withdrawn, cooled to room temperature, and sampled. The reduced pellets were then chemically analyzed for percent of total and metallic iron, and tested for crushing strength, which is the average load in pounds required to fracture the half-inch spheres as determined with laboratory compression testers (fig. 2). Figures 3–5 are plots of typical data showing the time-temperature dependence of metalization and crushing strength for magnetite, and specular hematite pellets reduced with lignite char and coke breeze.

Table 3.—Proximate analyses of fuels

Fuel	Analysis, percent					Heating value, Btu/lb[1]
	Moisture	Volatile matter	Fixed carbon	Ash	Sulfur	
Anthracite	2.8	4.9	79.5	12.8	0.6	12,330
Coke breeze	1.5	3.4	87.1	8.0	.7	13,100
Petroleum coke	.4	9.3	89.7	.6	5.4	14,910
Lignite	31.5	30.2	31.4	6.9	.25	7,190
Lignite char	4.2	15.2	66.3	14.3	1.8	10,690
Bituminous (caking)	1.1	35.7	58.4	4.8	.9	14,450
Do	.6	16.4	76.3	6.7	.6	14,530
Bituminous (noncaking)	9.1	34.4	49.4	7.1	.5	11,600

[1] As received.

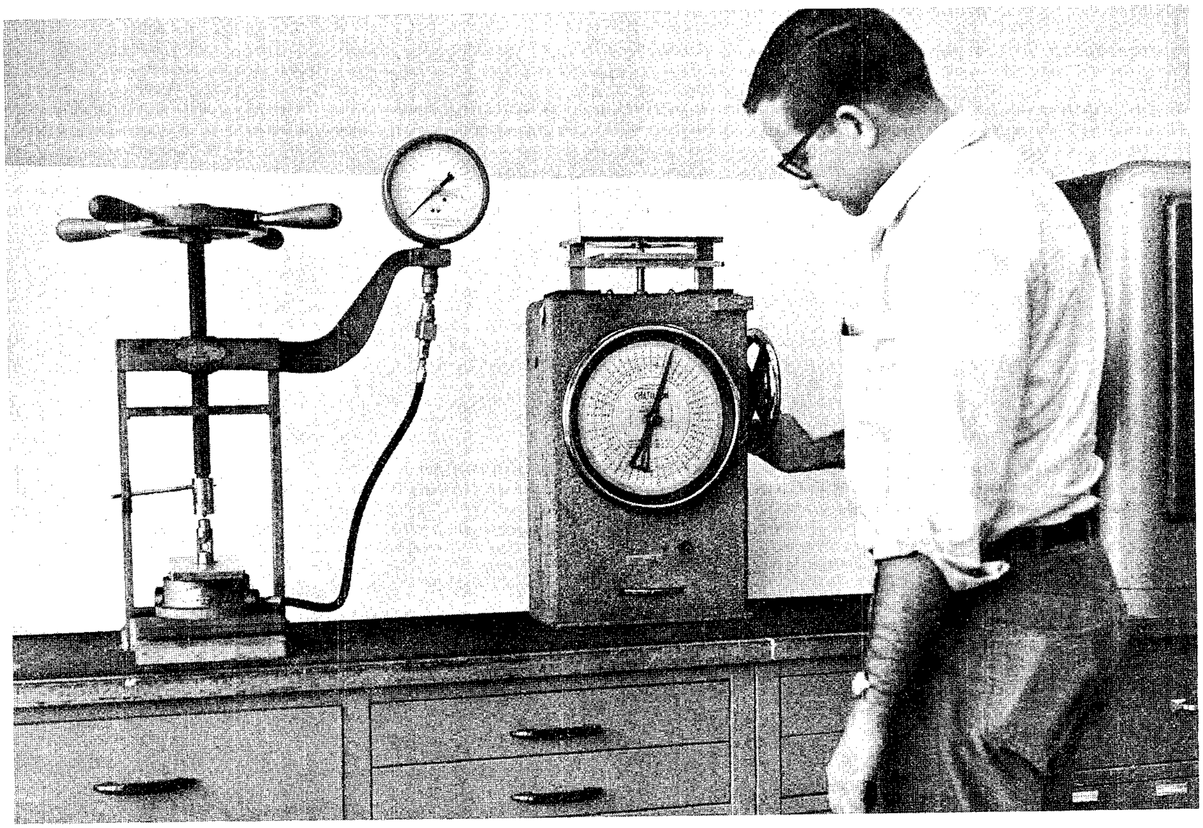

Figure 2.—Compression testers.

Although a very high degree of metallization was not sought, the pellets were reduced rapidly at 1,050° C. Metallization of about 70 to 80 percent was considered a somewhat flexible goal because disproportionate expenditures of energy are required to achieve reduction in the 90- to 100-

percent range. Adequate strength was not developed at the lowest temperature, but as both time and temperature were increased, the pellets could sustain a greater compressive load. There was an occasional departure from this generalization at the highest temperatures. Rapid exposure to 1,200° C or above sometimes created a dense peripheral iron layer which impeded penetration of reducing gases. However, from figure 3, it may be seen that a 1-hour exposure to reducing gases at 1,150° C increased crushing strength to more than 500 lb. This compares favorably with that of commercial iron oxide pellets.

Grain intergrowth of iron oxides and slag bonding contribute to the strength of oxide pellets. Of these, grain growth is much more important and, in the case of magnetic raw materials, is probably accelerated by the concurrent oxidation to hematite. A ceramic bond is created which is uniformly distributed throughout the agglomerate. Grain growth is also the dominant bonding phenomenon in reduced pellets but the mechanism differs. Reduction takes place topochemically from the surface with the successive formation (from hematite) of magnetite, wustite, and, ultimately, metallic iron. At about 1,000° C, the metallic iron content is well developed, but aside from some surface concentration, the iron grains plus those of wustite and silicates are scattered throughout. This results in low mechanical strength. As the pellet proceeds into hotter zones, peripheral concentration of metallic iron increases and the layer becomes compacted. Two of the forces responsible are (1) solid-state diffusion of iron into iron, which

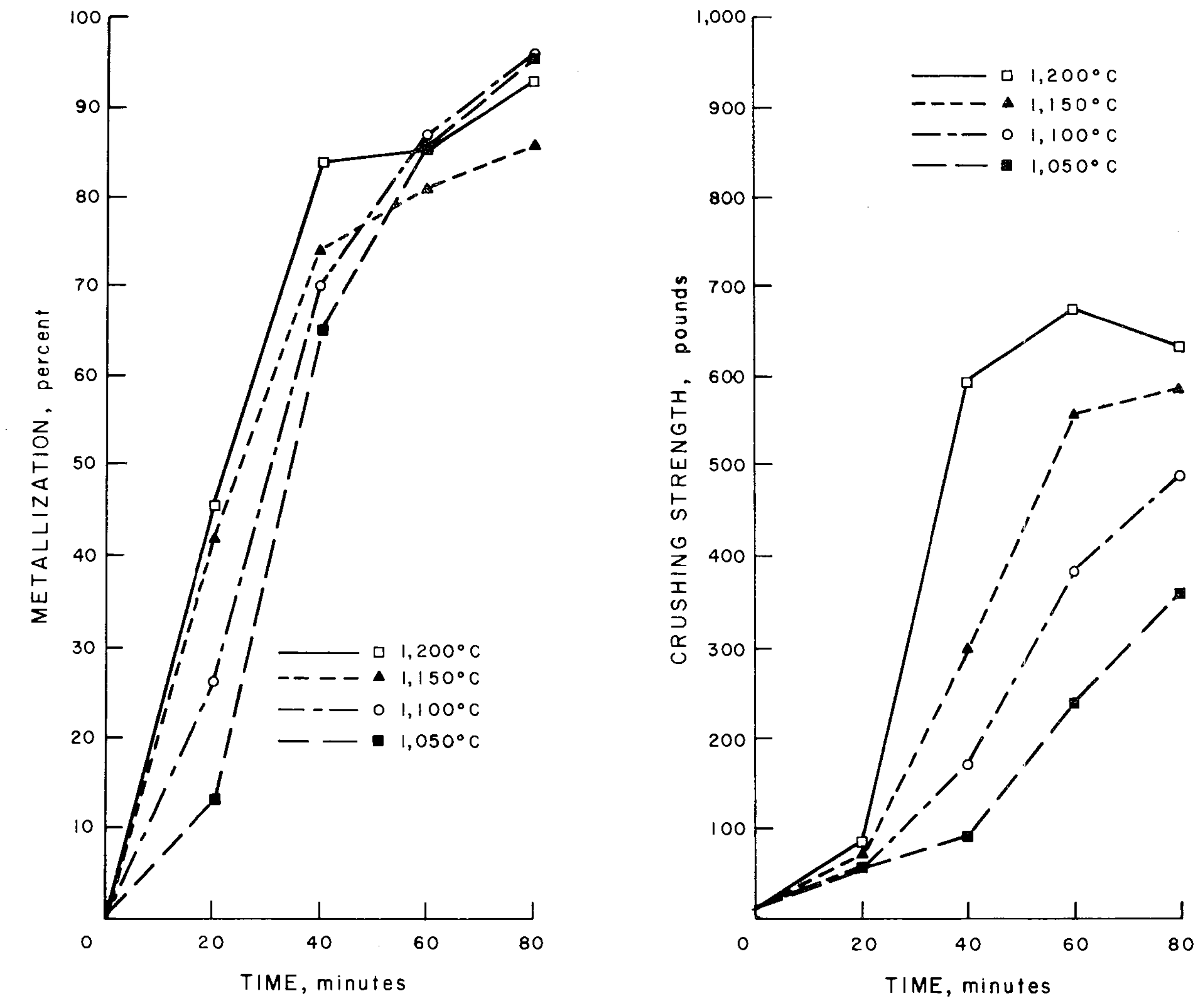

Figure 3.—Metallization and crushing strength of magnetite pellets reduced with lignite char.

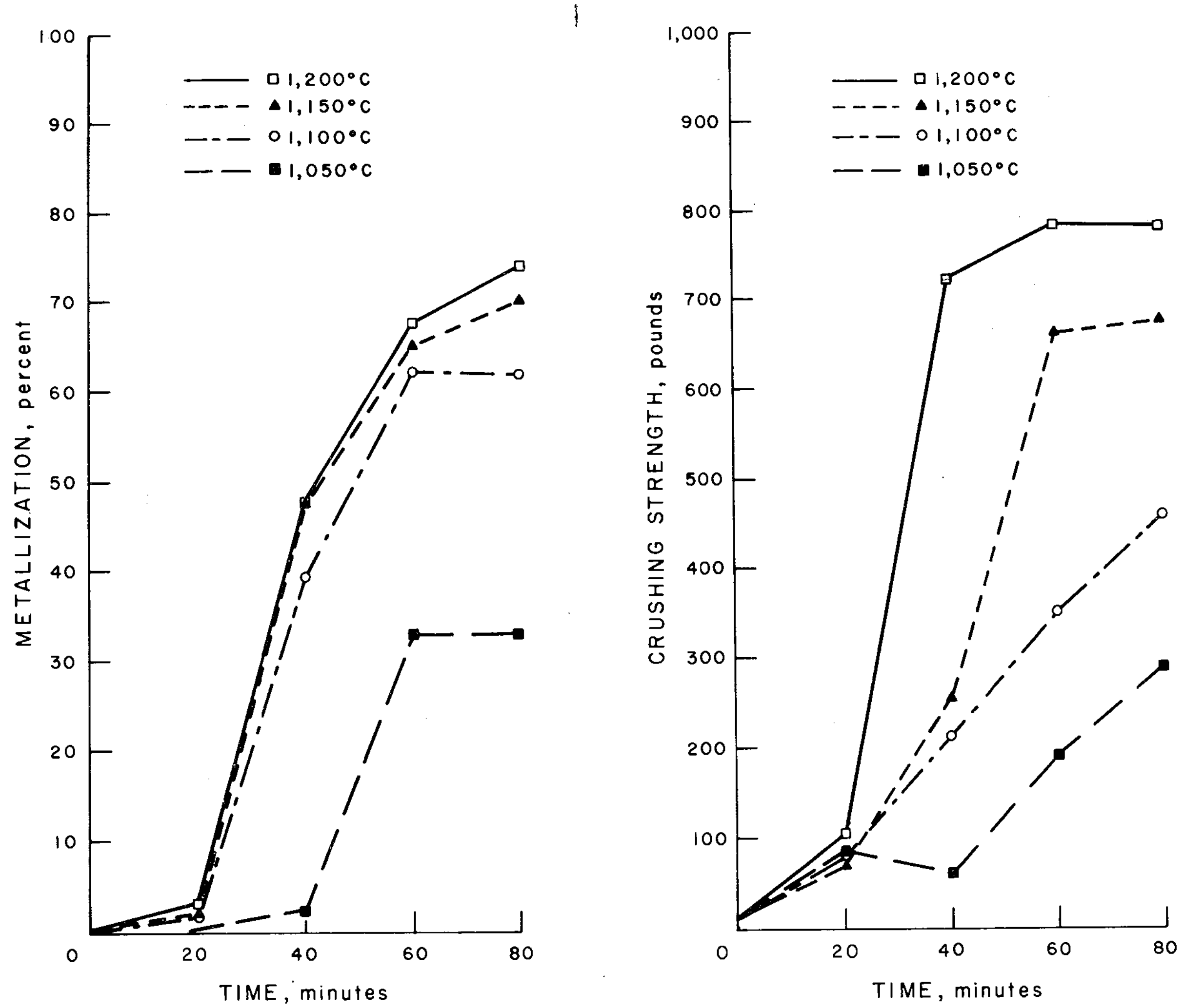

Figure 4.—Metallization and crushing strength of magnetite pellets reduced with coke breeze.

moves the particles away from the center, and (2) welding of adjacent grains at or near the surface. As the iron moves toward the surface, the wustite and slag countermigrate, and cavities develop. If this procedure is carried to extremes by prolonged exposure to high temperature, the metal shell separates physically and the wustite-slag core rattles around like the kernel in a nut.

In addition to time and temperature, a host of variables affect the properties of reduced agglomerates as well as green and fired oxide pellets. For raw material, these include grain size and distribution, specific surface, and content of bentonite. The primary function of the bentonite is to improve green properties; however, it has a salutary effect on the quality of reduced pellets. Figure 6 illustrates this point graphically for a specular hematite raw material. Note that at constant temperature, as the bentonite content of the mix was increased from 0 to 40 lb/ton, the crushing strength of the product pellets went from 100 lb to over 400 lb. The quantity of bentonite used must be balanced against the desired level of physical competence since bentonite is not only costly but adds to the silica content.

Other things being equal, the mineralogy of the concentrate affects both pelletizing and reduction. Geological composition is not a variable in the sense that it can be controlled, but there was a marked difference in behavior of the concentrates studied. Although specular hematite reduced more readily than natural magnetite, the resulting pellets were weaker. Synthetic magnetite yielded a highly reduced product, much more metallized than the previous samples. This was probably due as much to grain size as to differences in mineralogical makeup. The ultimate particles were much smaller than the values disclosed in the screen analysis which, for this cryptocrystalline material, tended to be qualitative rather than absolute. The fineness also created another problem. The rapid evolution of gases engendered a network of cracks in the product reduced by the regular laboratory procedure. A new firing regime with a slower heating rate solved the problem. Similar difficulties were encountered in pelletizing and reducing nonmagnetic taconite concentrate. The component minerals were rather earthy and contained chemically combined water of hydration. The first property appeared to be advantageous because less bentonite was required to form satisfactory green and dry balls. Evolution of the water of crystallization during the normal firing cycle gave rise to many surface cracks, and the resulting product was poorly consolidated. Here again the successful technique was to volatilize the moisture gradually at lower temperatures and accelerate the rate to reach levels required for simultaneous reduction and induration.

Table 1 presents chemical and physical properties of reduced pellets from nine typical batches. These were ½-inch pellets reduced under a slow-firing schedule from 450° C to 1,150° C. All were well metallized; however, those produced from specular hematite were notably weaker than the others. Pellets made with anthracite were substantially lower in metal, but much stronger than those reduced with lignite.

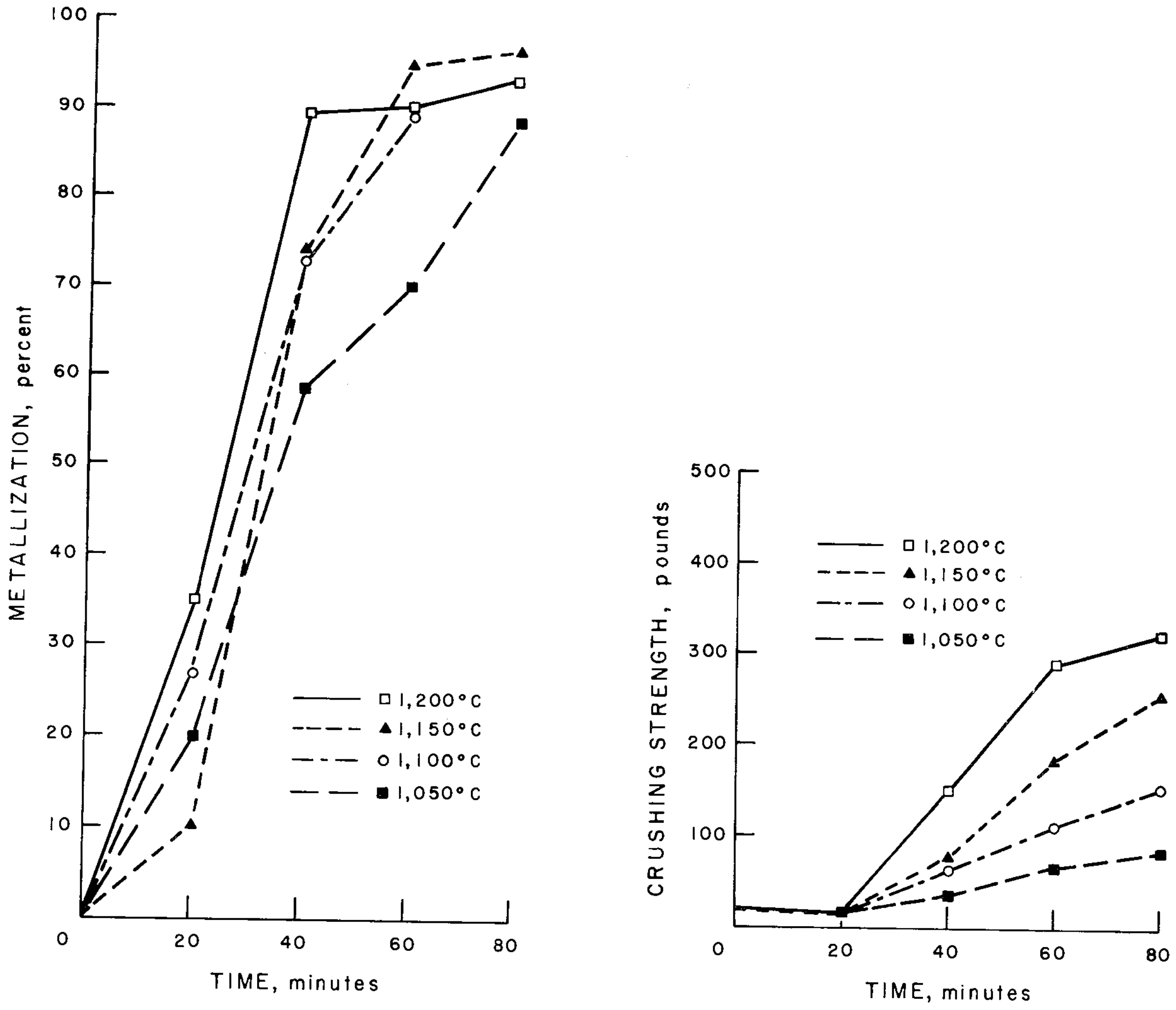

Figure 5.—Metallization and crushing strength of specular hematite concentrate pellets reduced with lignite char.

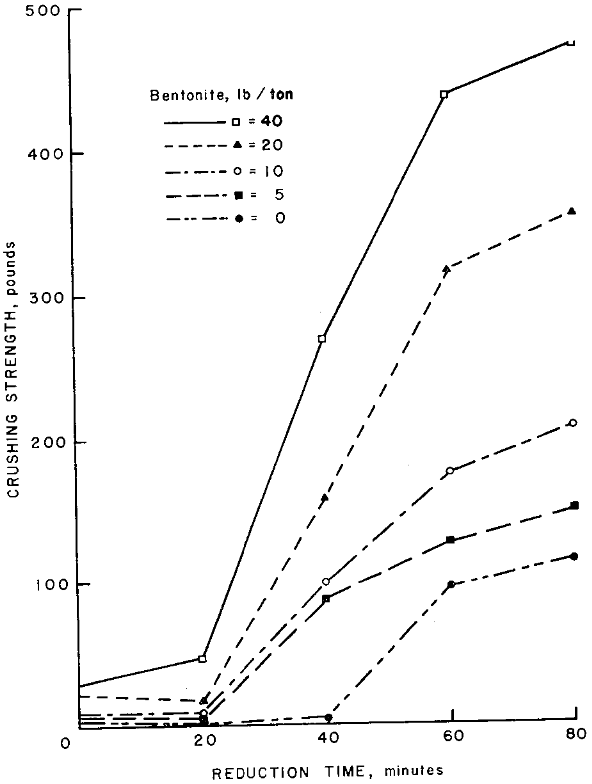

Figure 6.—Effect of bentonite on the crushing strength of prereduced pellets at 1,250° C.

An important variable in the process is the solid reductant, of which the following have been investigated: Lignite, lignite char, anthracite, metallurgical coke, petroleum coke, and bituminous coals. Substantial differences in behavior between some of these in iron ore reduction systems have been observed. In terms of reduction alone, lignite and lignite char were very effective, as was noncaking bituminous coal. Anthracite and metallurgical coke were intermediate; petroleum coke and caking bituminous coal rated lowest. One would expect this to be a function of the CO potential which in turn is related to reactivity of the fuel carbon; however, although anthracite is one of the least reactive fuels, it was one of the better reductants.

There is no simple relationship between the type of reductant and the hardness (expressed as crushing strength) developed. The hardness of highly reduced spheres was affected by both the thickness and the compaction of the peripheral layer, as previously stated. With the better reductants, therefore (lignite and anthracite, for example), satisfactory hardness was ensured by the expenditure of adequate time and temperature. Paradoxically, some of the poorest reductions with other coals resulted in very strong pellets. For example, with a West Virginia bituminous coal, ½-inch magnetite pellets were metallized to only 40 percent in 40 minutes at 1,200° C, but the individual crushing strengths were over 700 pounds. It is possible that the metal shell, although thin as a result of the low reduction, was unusually compact.

The sulfur content of the solid fuel also affects the properties of the reduced pellets. Sulfur is somewhat of a reduction deterrent, but it does catalyze and accelerate sintering of metallic iron. Sulfide sulfur is apparently much more potent in this regard than organic sulfur. For example, lignites containing appreciable pyrite will accomplish induration at 1,000° C as compared with over 1,100° C with other reductants. The rate of reduction is not slowed markedly, but limestone or dolomite must be added late in the process to desulfurize the pellets to an acceptable analysis.[1]

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[1] Fine, M. M., and R. B. Schluter. An Improved Process for Making Prereduced Iron Ore Pellets. Trans. AIME, v. 244, No. 1, March 1969, pp. 71–77.

PART 3. — PILOT PLANT PRODUCTION OF PREREDUCED PELLETS

By M. M. Fine and N. Bernstein [1]

PILOT TEST 1

The laboratory experimental work described in the previous chapter established that simultaneous induration and reduction of a wide variety of iron ore concentrates was technically feasible. It neither proved this could be done continuously on a meaningful scale nor determined the improvement in blast furnace practice through smelting this new product. Hence, in the spring of 1962, in cooperation with Erie Mining Co., 140 tons of prereduced pellets were made in a rotary kiln at the Bureau of Mines Twin Cities Metallurgy Research Center.

Pelletizing Procedure

Raw Materials

Tonnage quantities of concentrate in the form of filter cake were shipped to the Bureau's Twin Cities facility from the Erie Mining Co. plant at Hoyt Lakes, Minn. A typical chemical analysis showed, in percent, 64.2 Fe, 8.8 SiO_2 0.30 Al_2O_3, 0.28 CaO, and 0.40 MgO, and the size structure was substantially that of magnetic taconite (table 2). Limestone, used to scavenge the sulfur in kiln gases, was a local product crushed to pass ¼ inch and screened on 10-mesh prior to use. It contained, in percent, 30.5 CaO, 7.0 MgO, 6.0 Al_2O_3 and 14.8 SiO_2.

Analyses of the reductants, North Dakota Lignite and Pennsylvania anthracite, were comparable to the appropriate data in table 3. The lignite was a mine-run product that ranged from 3-inch lumps to dust; however, only the minus 1-inch portion was introduced to the kiln. The oversize was stockpiled and, after weathering for a week or two, decrepitated sufficiently to provide additional minus 1-inch lignite for the process. Barley-size anthracite (3/16 to 3/32 inch) was combined with lignite in a few runs.

Flowsheet and Equipment

Figure 7 is one of the early flow diagrams for simultaneous induration and reduction of green iron ore pellets. Rotary-kiln treatment of the pellet-lignite mix is followed by recovery of the metallized product by screening and magnetic separation. The individual steps and equipment are described in subsequent paragraphs.

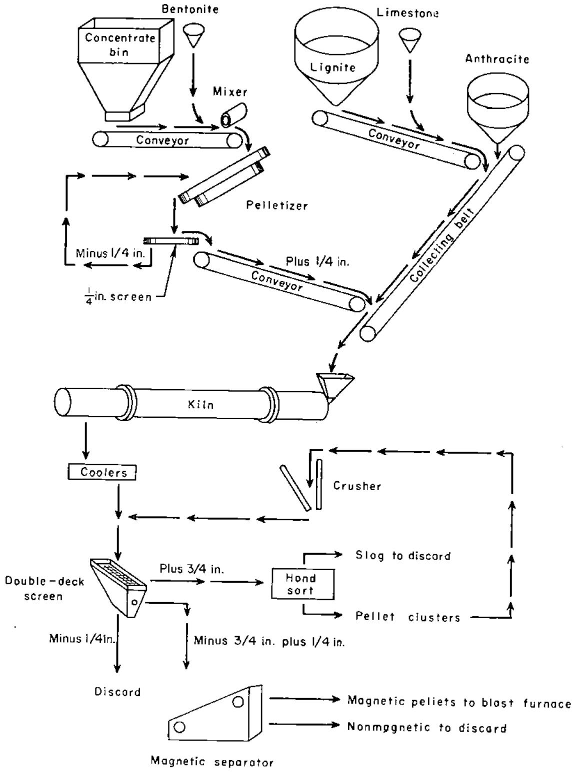

Figure 7.—Schematic flowsheet of process used to simultaneously indurate and metallize iron concentrate pellets.

The concentrate bin had a capacity of 50 cu ft, and lignite was stored in a 180-cu-ft cylindrical bin; both bins discharged to smooth, variable-speed rubber belts. A belt feeder attached to a hopper containing approximately 700 lb of anthracite supplied the system through part of the experiment. Wyoming bentonite was mixed into the concentrate with an enclosed horizontal reel set

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[1] Former Bureau of Mines metallurgist, Twin Cities Metallurgy Research Center, Minneapolis, Minn.; now with Sioux City Compressed Steel Co., Sioux City, Iowa.

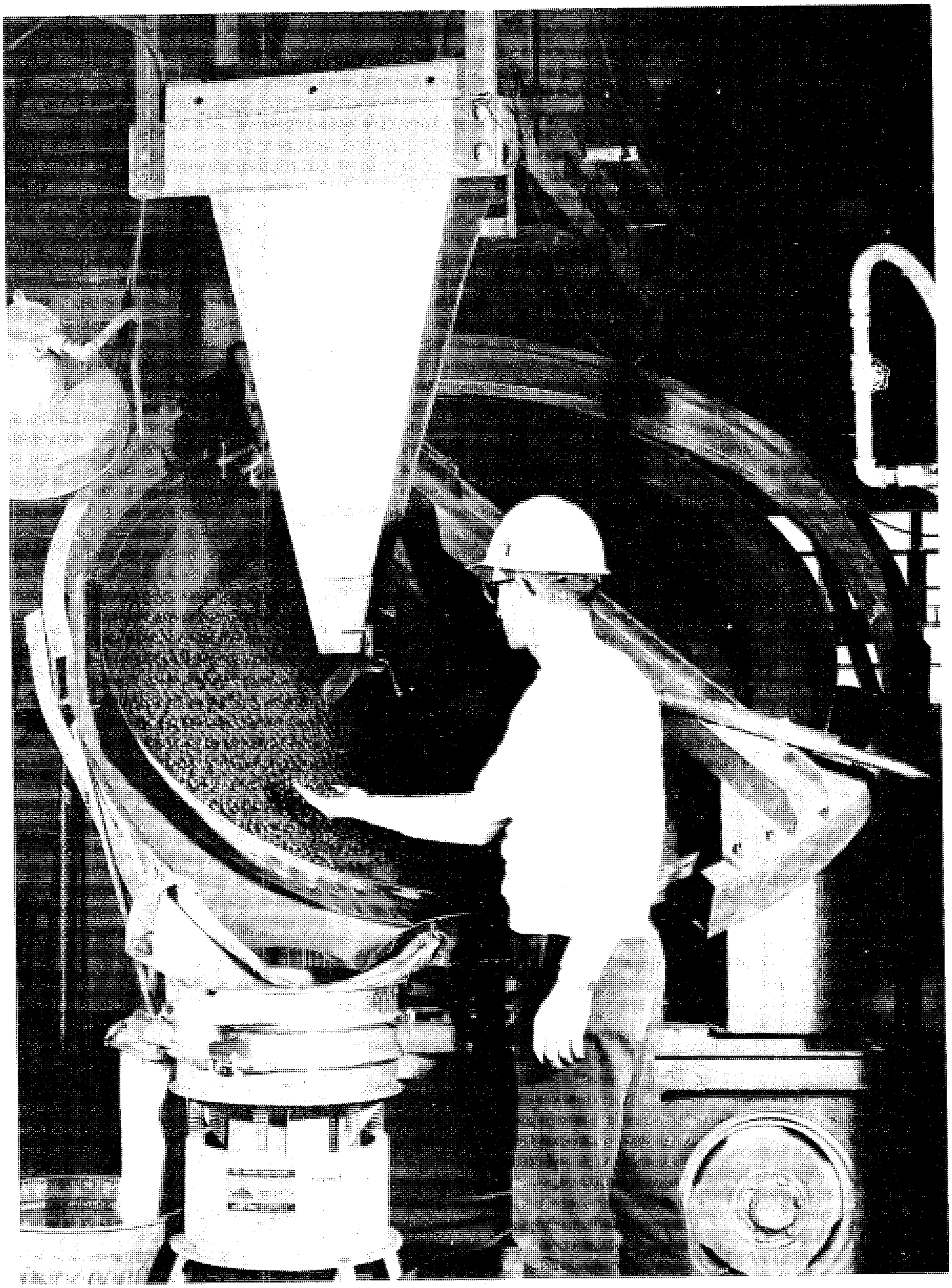

Figure 8.—Sixty-four-inch continuous-pelletizing disk.

slightly above the conveyor belt. The pelletizer illustrated in figure 8 was a 64-inch-diameter, two-step disk set at 45°, turning at 20 rpm, and equipped with a water spray. Green pellets were permitted to build to a maximum size of about ⅝ inch and were screened at ¼ inch on an 18-inch-diameter vibrating screen; the oversize was conveyed to the kiln, and the undersize was returned to the disk. Pellets conveyed from the screen dropped onto the final collecting belt, to which the solid reductant and limestone had also been conveyed. The material then entered the kiln through a 6-foot chute.

The kiln consisted of a steel cylinder 36 feet long and 43 inches in diameter; it was lined with firebrick so that the ultimate inside diameter was 34 inches. The combustion gases left the kiln at the feed end through a steel dust collector and a 14-inch-diameter by 40-foot dampered steel stack. The kiln was chain-driven by a 5-hp motor and had speeds ranging from ¾ to 3¾ rpm made available with a continuous variable-speed drive. A 4½-inch fireclay dam at the discharge end increased retention time of materials through the kiln and deepened the bed. An inspirating burner connected to a 24-psig natural gas line was used to heat the kiln at the discharge end. The burner heated the kiln through a combustion chamber 29 inches deep and 50 inches in diameter. The flame was adjusted so that the hottest part of the kiln was a 2-foot zone extending back from the dam.

Thermocouples in stainless steel protection tubes were located in the kiln at approximately 7-foot intervals. Four were mounted peripherally with measuring junctions 3 inches inside the kiln; contact was made through sliprings. A fifth thermocouple was inserted in the stack. Since the thermocouple closest to the discharge became coated with accretion during the experiment, optical pryometers were used to measure the temperature of the pellets immediately prior to discharge. The kiln is shown in figure 9; typical temperature profiles are shown in figure 10.

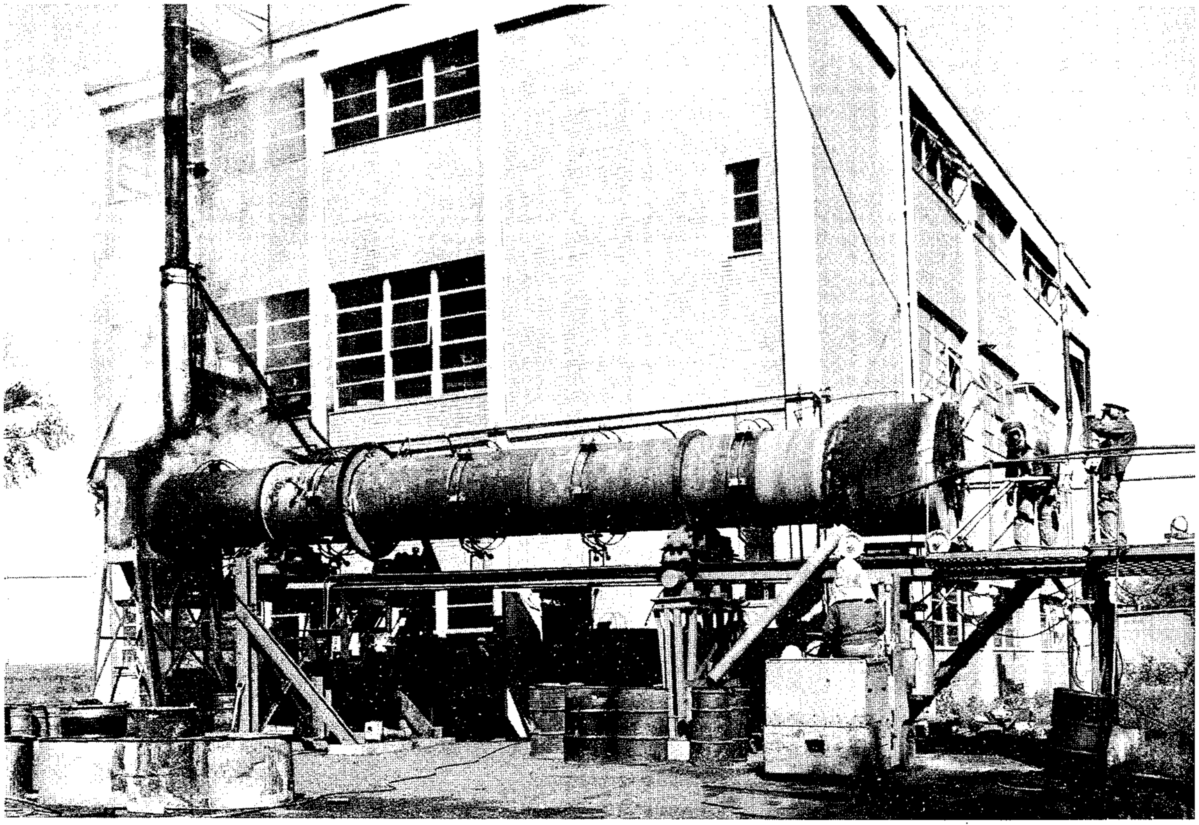

Figure 9.—Rotary kiln used in simultaneous reduction and induration of iron ore pellets.

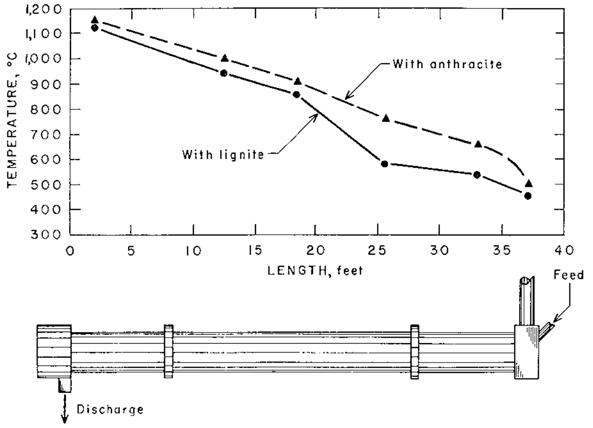

Figure 10.—Typical temperature profiles of 36-foot kiln.

Material was discharged from the kiln through a 12-inch-square steel chute with a butterfly gate. The chute passed through a water-cooled steel plate to which a 55-gallon drum was held in place by a forklift truck. Drums were cooled by water immersion while being filled and for 12 hours thereafter. Kiln discharge was sorted by first screening at ¾ and ¼ inch, and then magnetically separating the minus ¾- plus ¼-inch fraction on a belt magnetic separator. Figure 11 shows the pellet recovery setup.

Operating Techniques

During a campaign conducted in the spring of 1962, about 140 tons of prereduced pellets were manufactured for evaluation in the Bureau's experimental blast furnace. Laboratory data were confirmed, and hard, well-reduced pellets were made.

Moist concentrate initially was fed into the circuit at approximately 320 lb/hr; the rate was gradually increased to 450 lb/hr. Bentonite (0.8 percent of the dry concentrate weight) was added to the concentrate belt ahead of the reel mixer, from which the conditioned mix dropped to the pelletizer. A moisture content of about 8 to 10 percent in the mix resulted in excellent pellets. Half-inch unfired spheres had an average green crushing strength of 6 lb, and an average dry crushing strength of 18 lb; the green balls could sus-

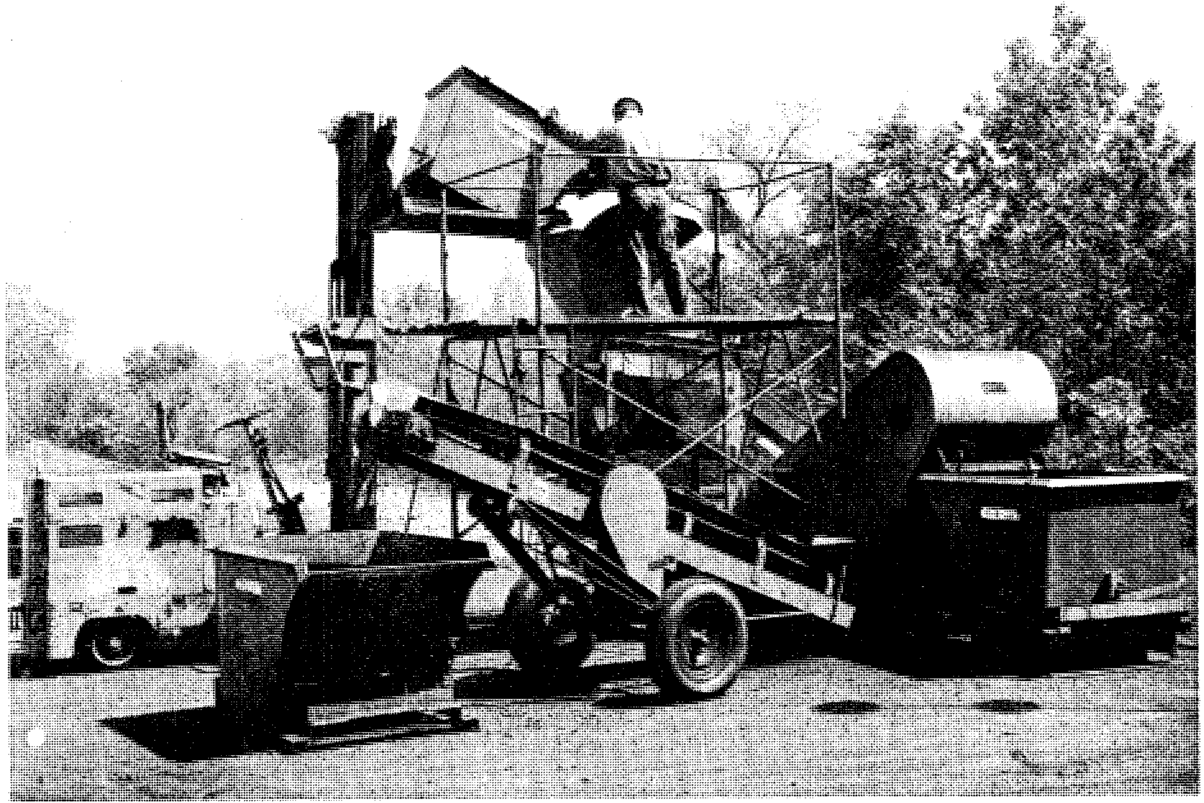

Figure 11.—Pellet recovery equipment.

tain 11 drops from an 18-inch height before shattering. Although variations in water content did not change the physical properties of green pellets appreciably, they affected their ability to withstand the journey through the kiln. Overly dry pellets tended to peel in concentric layers; wet feed resulted in irregularly shaped and highly fragmented pellets because of the rapid (perhaps explosive) escape of vapor in the hot kiln. Under favorable conditions, the pellets traversed the kiln relatively undamaged, and more than 95 percent of the iron in the incoming feed was recovered as usable, plus ¼-inch prereduced pellets. The balance of the iron was largely dust, minus ¼-inch fragments, and kiln accretions, most of which could be reclaimed and returned to the circuit.

At the outset, the ratio of lignite to pellets was 1 to 1, but towards the end of the run it was approximately 0.85 to 1. Excess lignite above the stoichiometric quantity required for reduction cushioned the green pellets in the kiln. Industrially, the reductant consumption would be less because some could be recirculated. The temperatures characteristic of the operation were as follows: 1,150° C at the discharge, 450° C at the feed end, and 260° C in the stack. The bed depth at discharge was about 4.5 inches. Precise retention times in the kiln were not measured, but calculations suggest that if no dam were present, retention time would have been 120 minutes. The duration of exposure to elevated tempertures was more significant. Williams and coworkers[2] noted that the rate of reduction of iron ore with carbon increases rapidly with rising temperature and approaches practical values above 850° C. It is estimated that the material passing through the kiln was in this temperature range for approximately 1 hour.

Pellet samples were taken hourly through a port in the discharge chute. Ten pellets were tested in compression, and the crushed fragments were chemically analyzed. Percent metallization, used to measure the effectiveness of the reduction, was the quotient of 100 times metallic iron divided by total iron content. Pellets produced between sample intervals were accepted for further processing in the blast furnace if the sample had average compression strength above 200 lb, at least 77 percent total iron, and 55 percent metallic iron.

It was necessary to limit ingress of air to maintain a highly reducing atmosphere in the kiln and to discourage undue combustion of the lignite. Access openings were fitted with doors, the port was sealed around the burner, and the discharge drums were held tightly against the chute opening, as shown in figure 12. The natural gas burner was operated so that a slight positive pressure was maintained in the kiln. It was virtually impossible to take reliable samples of kiln gases, but the stack steam was sampled occasionally for the presence of excess reducing gases.

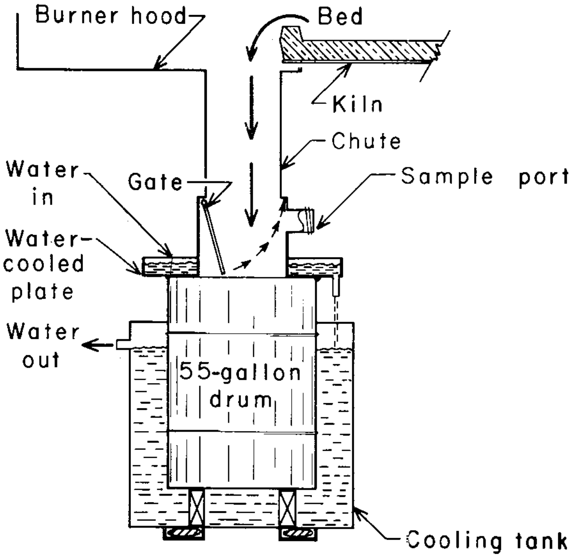

Figure 12.–Kiln discharge and product sampling point.

Screening the kiln product on ¾ inch removed a few oversize pellet clusters and fragments of accretion scraped from the kiln wall during operation. The clusters were later crushed and added to the final product; slag nodules were discarded. A second screening, on ¼ inch, separated fine charred lignite, ash, lime, and pellet fragments.

Finally the ¾- to ¼-inch product was magnetically separated to recover the prereduced pellets, 140 tons of which were shipped to Bruceton, Pa., for smelting in the experimental blast furnace. Samples of the three-car shipment had the following average chemical analysis, in percent: 79.0 total Fe, 60.6 metallic Fe, and 11.9 SiO_2.

Experimental Results

Operations

Table 4 gives the results of 14 consecutive days of satisfactory kiln operation. For the first 8 days, lignite was the sole reductant at an average feed rate of 350 lb/hr. Moist pellets were introduced into the kiln at a rate ranging from 320 to 440 lb/hr. Hourly samples of the prereduced pellets

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[2] Williams, C. E., E. P. Barrett, and B. M. Larsen. Production of Sponge Iron. BuMines Bull. 270, 1927, p. 91.

were tested in compression, and the resulting fragments accumulated during 4 hours, were pulverized for chemical analysis. Hence, for each 24-hour period the reported value of crushing strength was the mean of 240 determinations, and the chemical analyses were averages of six independent samples. In days 1 to 8, highly metallized pellets with adequate crushing strength were consistently produced. The total iron contents of the samples were almost 80 percent, and most of the values for metallic iron were well over 60 percent. Even at the highest feed rate, the excess lignite helped to maintain the quality of the product.

Table 4.—Results of 14 days' prereduced-pellet production

Test day (24-hr periods)	Lignite, lb/hr	Anthracite, lb/hr	Wet pellets, lb/hr	Prereduced pellets			
				Analysis, percent			Crushing strength, lb
				Total Fe	Metallic Fe	Metallization	
1	355	0	345	80.0	63.8	80	380
2	315	0	370	77.5	54.9	71	435
3	320	0	320	77.7	58.3	75	355
4	355	0	345	80.5	63.6	79	380
5	350	0	370	79.6	59.0	74	425
6	350	0	355	80.6	62.3	77	340
7	350	0	400	80.7	63.0	78	320
8	380	0	440	81.1	65.7	81	320
9	310	95	445	80.0	61.1	76	345
10	320	100	440	78.4	56.3	72	285
11	305	100	430	79.7	59.7	75	300
12	310	95	485	80.4	61.7	77	380
13	215	195	470	75.2	44.7	59	475
14	230	200	455	74.5	43.0	58	415

For the next 4 days, anthracite comprised one-fourth the total reductant weight; the same high green-pellet rate was maintained. As shown in table 4 (days 9 to 12), the product continued to be highly satisfactory, both physically and chemically. The average metallization was slightly lower than in the previous 8 days but not enough to be significant.

The weight-percent of anthracite in the reductant was increased to near 50 percent for the next 48 hours (days 13 and 14, table 4). Chemical quality of the product deteriorated drastically, with a metallization of only 44 percent and a total iron content of about 75 percent. In a general way, the lower conversion to metal confirmed the laboratory findings that, for a given time and temperature, anthracite was about 25 percent less effective than lignite. On the basis of the limited data, it seemed that the 50–50 mixture was even less efficient. Anthracite in the kiln discharge appeared to be unburned and, undoubtedly, the lignite was consumed preferentially. Since the quantity of lignite in the mixture was no longer an adequate excess, the pellets were underreduced. The crushing strength of the product pellets was notably greater than that of the more metallized spheres from the previous days. This perplexing phenomenon had been encountered in laboratory tests with poor reductants like petroleum coke. It had not, however, been characteristic of either lignite or anthracite.

The effects of decreased retention time coupled with a shallow bed were investigated by removing the discharge dam from the kiln while holding other conditions constant. Table 5 shows the results of a 5-day run with no dam and the following 5 days after the dam was restored. Little choice could be made between the two periods concerning crushing strength, but the 50-percent metallization achieved without the dam was unsatisfactory. This demonstrated the necessity of having a bed at least several inches deep. Although decreased retention time contributed to poorer reduction, the shallow depth was equally detrimental. As noted by Williams and coworkers,[3] the kiln atmosphere is less reducing above than within the bed, and individual pellets are exposed more frequently in a thin layer.

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[3] Page 125 of work cited in footnote 2.

Table 5.—Results of 10 days' prereduced-pellet production

Test day (24-hr periods)	Lignite, lb/hr	Wet pellets, lb/hr	Prereduced pellets			
			Analysis, percent			Crushing strength, lb
			Total Fe	Metallic Fe	Metallization	
Without dam:						
1	400	390	70.8	32.7	46	250
2	380	380	73.3	37.0	51	425
3	390	350	73.2	38.2	52	525
4	380	325	71.7	37.0	52	325
5	380	380	73.8	35.3	48	350
Average	—	—	72.6	36.0	50	375
Dam replaced:						
6	355	355	80.0	63.8	80	380
7	315	370	77.5	54.9	71	435
8	320	320	77.7	58.3	75	355
9	355	345	80.5	63.6	79	380
10	350	370	79.6	59.0	74	425
Average	—	—	79.1	59.9	76	395

During periods of satisfactory operation, sulfur content of the product pellets was no problem. The addition of a few percent crushed limestone into the kiln feed held the average sulfur content at 0.02 to 0.04 percent. However, when the use of limestone was discontinued, or if the limestone became damp and lumpy which made feeding difficult, the product acquired as much as 0.15 percent of sulfur.

Properties of Product

Among the properties used to evaluate iron ore agglomerates are resistance to compression, abrasion, shattering, and reduction disintegration. These were determined on the reduced pellets and compared, whenever possible, with similar measurements on commercial oxide pellets made from the same raw material. Compression resistance was measured in a manually operated tester (fig. 2) with a hydraulic load cell, in which individual pellets were compressed between parallel steel surfaces until they cracked. The reading was reported as the crushing strength per pellet. This value is subject to wide variations and is influenced by many factors; pellet size is one of the most important. Table 6 presents the average screen analysis of four sample batches taken from a number of product drums; a screen analysis of Erie oxide pellets is also included. The reduced pellets essentially ranged from 0.624 to 0.371 inch; the Erie product was approximately one screen fraction smaller.

Selected sized pellets (at least 80 of each size) were tested in compression with the results given in table 7. In general, although the average crushing strengths of the reduced pellets were satisfactory, they were somewhat lower than those for the corresponding sizes of oxide. Regardless of crushing strength, reduced pellets are somewhat malleable and tend to yield by plastic deformation. Pellets that break tend to remain in sizable fragments; greater deterioration was suffered by oxide pellets. This property was particularly noticeable in the drop test, the results of which are given later.

Crushing strength is a simple and convenient control determination, but obviously is not a measurement of high precision. Wide variations within batches, as indicated by the standard deviations, were characteristic of both reduced and oxide pellets. This variation could be expected, considering the number of factors that affect crushing strength, such as size, shape, surface irregularities, time and temperature of processing, quantity of bentonite and uniformity of distribution throughout the pellet, and the idiosyncrasies of the testing device and its operator.

Resistance to abrasion was determined by tumbling sized samples in a 3-foot-diameter steel drum fitted with 2-inch lifters and rotating at 25 rpm, similar to the ASTM coke tumbler.[4] Samples were screened after 200 revolutions; the percent retained on a 3/8-inch sieve was defined as the

Table 6.—Screen analyses of prereduced and oxide pellets, weight-percent

Size, inch	Prereduced pellets	Erie pellets
Plus 0.624	5.0	1.9
0.624 to 0.525	27.7	4.9
.525 to .441	40.0	25.0
.441 to .371	17.7	52.8
.371 to .263	7.7	15.3
Minus 0.263	1.9	.1
Total	100.0	100.0

Table 7.—Crushing strengths of prereduced and oxide pellets

Size, inch	Prereduced pellets			Erie pellets		
	Strength, lb	Standard deviation	Confidence limits[1]	Strength, lb	Standard deviation	Confidence limits[1]
0.624 to 0.525	422	159	35	—	—	—
.525 to .441	312	105	23	388	172	53
.441 to .371	219	101	22	325	123	38
.371 to .263	—	—	—	292	160	50

[1] Plus or minus the figure listed for 90-percent confidence.

tumble index, and that retained on 28-mesh was the abrasion index. Table 8 presents the composite results of tumbling four 10-pound batches of plus ⅜-inch reduced pellets and one of the commercial oxide variety. Both had abrasion indices of over 99 percent; very little dust was created from either. Prereduced pellets were more resistant to fragmentation, as indicated by a tumble index of over 97 percent compared with about 94 percent for the others.

Table 8.—Summary of tumbler tests

Size	Prereduced pellets		Erie pellets	
	Weight-percent	Cumulative weight-percent	Weight-percent	Cumulative weight-percent
Plus ⅜-inch	97.5	97.5	93.8	93.8
⅜-inch to 28-mesh	1.9	99.4	5.6	99.4
Minus 28-mesh	.6	100.0	.6	100.0
Total	100.0	—	100.0	—

The malleability of metallized spheres also imparts better resistance to shattering. This property was measured by dropping 1,000 grams of plus ⅜-inch pellets 15 times down a cylindrical pipe 15 feet high and screening the fragments. The results are shown in table 9. The tests indicate the resistance of pellets to handling and shipping.

Information is also gained by measuring the resistance of pellets to reduction disintegration (physical deterioration in a reducing atmosphere), such as that encountered in a blast furnace. This resistance is determined with the Linder test,[5] in which

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[4] American Society for Testing and Materials. Tentative Method of Tumbler Test for Coke. D 294-63T in 1964 Book of ASTM Standards.
Part 19, Gaseous Fuels, Coal and Coke. Philadelphia, Pa., 1964, pp. 56–58.

Table 9.—Results of drop tests

Size	Prereduced pellets		Erie pellets	
	Weight-percent	Cumulative weight-percent	Weight-percent	Cumulative weight-percent
Plus 0.441-inch	75.9	75.9	19.1	19.1
0.441- to 0.371-inch	22.8	98.7	50.6	69.7
0.371-inch to 4-mesh	.6	99.3	14.7	84.4
4- to 8-mesh	.3	99.6	8.8	93.2
Minus 8-mesh	.4	100.0	6.8	100.0
Total	100.0	—	100.0	—

Table 10.—Screen analyses of concentrates, weight-percent

Size, mesh	Natural magnetite	Synthetic magnetite
Plus 100	0.8	1.2
100 to 200	3.5	7.5
200 to 325	9.6	15.5
Minus 325	86.1	75.8
Total	100.0	100.0

Table 11.—Chemical analyses of concentrates, percent

Product	Fe	SiO_2	Al_2O_3	CaO	MgO
Natural magnetite	66.0	6.7	Trace	0.6	0.4
Synthetic magnetite	67.9	4.8	0.2	.5	Trace

a sized sample is tumbled in a 130-mm drum while programed temperature and reducing atmosphere are applied. Subsequent screening defines the amount of disintegration. Prereduced pellets are subject to very little reduction disintegration. This was confirmed by the Pickands-Mather Co., Hibbing, Minn., laboratory that found only 0.1 percent minus 28-mesh material resulting from the Linder test. Oxide pellets may display 20 percent or more minus 28-mesh material after the Linder test.

PILOT TEST 2

Continuous campaigns in the large rotary kiln were carried on intermittently to determine the applicability of the process to other iron ore concentrates and its operability with other solid fuels, and to improve and simplify the procedure. This section summarizes some of the more recent findings.

Pelletizing Procedure

Raw Materials

The concentrates evaluated in this campaign were a natural magnetite and a synthetic magnetite with screen and chemical analyses as given in tables 10 and 11. The natural magnetite was a filter cake from Erie Mining Co. similar to but higher in grade than the concentrate used by the Bureau in the earlier prereduced pellet work. It is particularly adaptable to pelletizing, and the commercial oxide pellets produced therefrom can sustain a compressive load of 300 to 400 lb per ½-inch sphere.

The synthetic magnetite was an industrial pilot plant product resulting from reduction roasting and magnetic separation of nonmagnetic taconite. This material makes green pellets that are very sensitive to rapid heating, according to Bunge,[6] and the preparation of oxide pellets requires carefully programed and controlled firing.

The reductants studied were raw North Dakota lignite and Pennsylvania anthracite. The lignite, received as minus 2-inch, was sized shortly before use to exclude the plus 1-inch and minus ¼-inch portions; barley-sized anthracite was used as received. Analyses of the reductants are given in table 12. Limestone was charged to the kiln to prevent sulfur from combining with the nascent metallic iron. The stone was sized between 4- and 10-mesh and contained 45 percent CaO, 9 per-

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[5] Linder, Rolf. Programme-Controlled Reduction Test for Blast-Furnace Burdens. Journees Internationales de Siderurgie, Luxembourg, June 18–28, 1958, pp. 312–325 (reprint; copies available from Belgian Iron and Steel Institute).

[6] Bunge, F. H., and J. H. Wakeman. Pelletizing Butler, Groveland, and Carol Lake Concentrates. Proc. 23d Annual Univ. of Minn. Min. Symp. and Ann. Meeting, Minn. Sec., AIME, Duluth, Minn., January 1962, Duluth, Minn, 1962, p. 55.

Table 12.—Analyses of reductants

Constituents	Analysis (as received)	
	Lignite	Anthracite
Proximate percent:		
Moisture	31.5	1.9
Volatile	30.2	5.5
Fixed carbon	31.4	81.4
Ash	6.9	11.2
Ultimate percent:		
Hydrogen	6.2	2.5
Carbon	43.5	80.6
Nitrogen	0.7	0.9
Oxygen	42.3	4.1
Organic sulfur	0.24	0.35
Pyritic sulfur	0.00	0.38
Total sulfur	0.25	0.74
Heating value, Btu/lb as received	7,190	12,960
Ash fusibility, ° C:		
Deformation	1,250	1,600
Softening	1,290	—
Fluid	1,330	—

Table 13.—Raw material feed rates

Test No.	Lb/hr				Percent of dry concentrate	
	Natural magnetite	Synthetic magnetite	Lignite	Anthracite	Bentonite	Limestone
1	300	0	275	0	1	2
2	300	0	0	225	1	2
3	0	300	275	0	1	2
4	0	300	0	225	1	2

cent MgO, and 1 percent silica. A swelling variety of Wyoming bentonite was used to bond the green pellets. Table 13 is a listing of nominal raw material feed rates.

Process Description

The process depicted in figure 13 consisted of forming green pellets in a conventional manner as before and charging them to a rotary kiln along with coal and limestone. It was, however, a considerable simplification over that of figure 7 because magnetic separation was not required to clean and reclaim the product pellets. Furthermore, the number of clusters and slag fragments coarser than ¾ inch was so few that special handling for the size was not required. These gains were achieved by using lignite with higher ash fusion temperature and excluding lignite fines from the feed, both of which diminished kiln accretions. Mechanical improvements included use of a foundry-type muller to mix concentrate, bentonite, and water in 300-lb batches. Before entering the pelletizing disk, the mix was hammermilled to pulverize any lumps.

Results

Each run was evaluated on the following factors: Average crushing resistance of the product pellets, size indices from a tumble test and from a 15-foot drop test, extent of metallization, efficiency of iron recovery, and the rate of accretion. Procedures for the crushing, tumble, and drop tests were described earlier, along with a definition of metallization. The recovery was calculated from the proportion of iron reporting in the product pellets as compared with the total amount of accounted iron. This included, aside from the pellets, minus ¼-inch discharge material, accretions removed from the kiln walls after each test, and stack dust. The rate of accretion was important because the buildup decreases the effi-

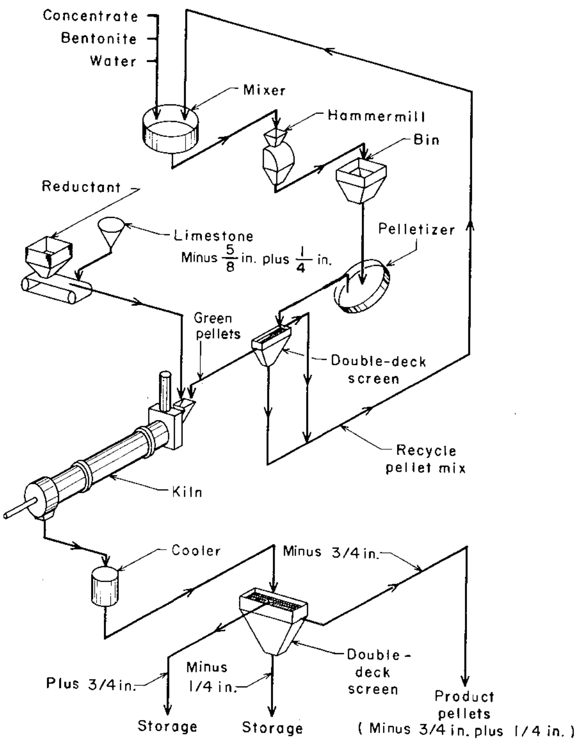

Figure 13.—Improved flowsheet for producing prereduced pellets.

ciency of rotary kiln operation from a recovery point of view, as well as necessitating continuous removal. This may be done at elevated temperatures for some time, but eventually requires cooling the kiln and chiseling the ring free with power tools. The rate of permanent accretion was determined for each test by weighing the material removed after the kiln was cooled and apportioning it over the operating time.

The ideal situation would be to produce pellets with high crushing strengths, tumble indices, drop indices, metallization, and iron recoveries, as well as small accretion rates. Table 14 summarizes the results of the four tests. No universally accepted standards exist for indurated pellets, but a tumbler abrasion index of 97 percent, a drop index of 97 percent, and a 200-lb minimum-compression strength were considered reasonable. The pellets from all runs surpassed these requirements.

Iron recoveries are depicted graphically in figures 14 through 17. Material over ¾ inch was included with the accretions. Recoveries of 91.0 to 95.8 perent of the accounted iron units were credited to the product pellets. The balance was mostly minus ¼-inch discharge material and stack dust. Treatment of both items would be comparatively simple because after magnetic sepa-

Table 14.—Test results and pellet properties

Test No.	Fe recovery in pellets, percent	Accretion rate, lb/hr	Pellet properties					Chemical analysis, percent					
			Crushing strength, lb	Tumble test, percent		Drop test, percent							
				+8	+28	+8	+28	Total Fe	Metallic Fe	SiO_2	CaO	MgO	S
1	91.0	6	300	99.3	99.5	99.5	99.8	82.1	68.4	9.3	1.0	0.5	0.03
2	95.7	2	520	99.8	99.8	99.9	99.9	79.8	47.4	8.1	1.2	.3	.08
3	95.8	1	220	98.0	98.9	99.1	99.7	90.8	84.4	6.8	.2	trace	.01
4	95.1	Negligible	275	99.6	99.6	99.9	99.9	87.3	71.0	6.7	.2	trace	.10

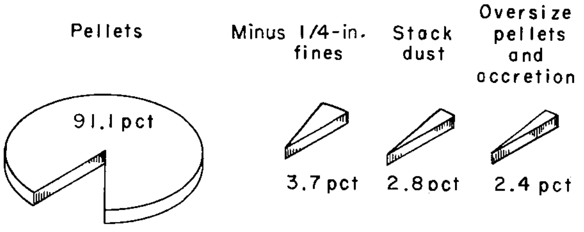

Figure 14.—Iron balance of discharge—test 1.

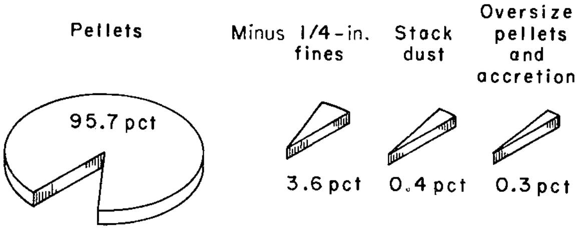

Figure 15.—Iron balance of discharge—test 2.

ration, they resemble the pellet product or the original concentrate, respectively.

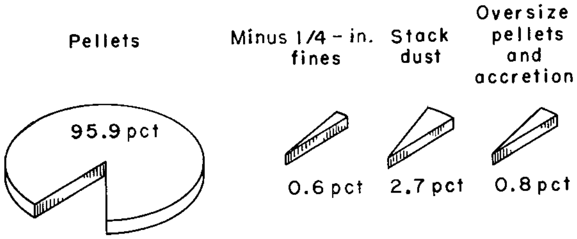

Figure 16.—Iron balance of discharge—test 3.

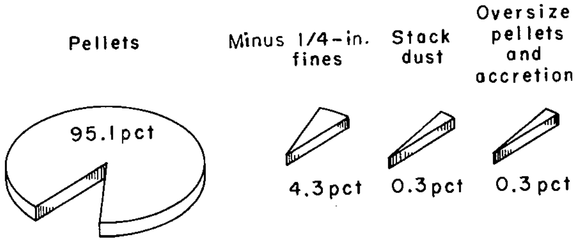

Figure 17.—Iron balance of discharge—test 4.

Separation of Iron and Carbon in Fines

The minus ¼-inch kiln product remaining after the pellets are screened contains reclaimable metallic iron and char. The average iron and carbon distributions of sized and magnetically separated portions of the minus ¼-inch fraction from several tests are shown in table 15. The magnetic material larger than 6-mesh is chemically similar to the prereduced pellets and might be accepted by the consumer as screened; otherwise it could be briquetted. This portion contains about half the minus ¼-inch iron discharged from the kiln. Magnetic material in the 6- to 28-mesh size, being lower in iron and not excessively high in silica, could be briquetted or returned to the grinding circuit for recycling. The small quantity of minus 28-mesh material could well be discarded, since it is generally low in iron, both chemically and distribution-wise. In tests 1 and 4 (as seen in table 15) iron distribution in the minus 28-mesh magnetics amounts to 0.8 and 4.1 percent, respectively, based on the minus ¼-inch discharge; if calcu-

Table 15.—Magnetic separation of minus ¼-inch kiln discharge

Product	Weight, percent	Chemical analysis, percent			Distribution, percent	
		Fe	SiO_2	C	Fe	C[1]
TEST 1 (NATURAL MAGNETITE-LIGNITE)						
¼-inch to 6-mesh:						
Magnetic	9.9	80.8	10.0	—	63.3	—
Nonmagnetic	10.5	.7	—	43.0	.6	9.3
6- to 28-mesh:						
Magnetic	5.3	67.5	9.7	—	28.3	—
Nonmagnetic	67.2	1.1	—	62.0	5.8	85.4
Minus 28-mesh:						
Magnetic	.3	31.9	12.2	—	.8	—
Nonmagnetic	6.8	2.3	—	37.9	1.2	5.3
Total	100.0	12.6		48.8	100.0	100.0
TEST 4 (SYNTHETIC MAGNETITE-ANTHRACITE)						
¼-inch to 6-mesh:						
Magnetic	1.5	89.6	7.0	—	32.0	—
Nonmagnetic	17.6	.7	—	81.8	2.9	18.4
6- to 28-mesh:						
Magnetic	2.6	76.7	8.5	—	47.6	—
Nonmagnetic	76.2	.7	—	82.4	12.7	80.0
Minus 28-mesh:						
Magnetic	.3	56.1	14.0	—	4.1	—
Nonmagnetic	1.8	1.4	—	68.8	.7	1.6
Total	100.0	4.2		78.4	100.0	100.0

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[1] Neglecting carbon content of magnetic fractions.

lated on the basis of total discharged iron, the minus 28-mesh magnetics are only 0.03 and 0.18 percent.

The nonmagnetic fractions must be water-washed to slake and remove the sulfur-bearing lime and then rescreened. The oversize would be recycled as reductant and the undersize discarded. Finer size nonmagnetics are low in carbon and would tend to create an accretion problem. Selection of exact screen sizes could be based on the behavior of individual concentrates and reductants.

Weather Resistance

A final evaluation was based on resistance to weathering. Pellets stored indoors and analyzed 6 months after the first pilot campaign had the same iron content as the initial sample; however, those exposed to the elements showed progressive oxidation from month to month. Reduced-pellet samples in 6-inch-deep wood boxes with slotted bottoms to permit drainage were placed in unsheltered, outdoor locations. One box per month was withdrawn, and the contents were dried, tested, and chemically analyzed. The results of the 6-month exposure, shown in table 16, indicated that the crushing strength and tumble index are relatively unchanged. The metallic iron analysis after a sharp initial drop, slowly decreased; starting from 54.0 percent, the metallic iron in the sample analyzed 43.9 percent after 7 months.

Table 16.—Results of weathering prereduced pellets

Period ending	Days of exposure	Crushing strength, lb	Tumble index	Percent			Precipitation,[1] inches	Average temperature,[2] ° F
				Total Fe	Metallic Fe	Metallization		
Aug. 2	0	417	99.0	78.9	54.0	69	—	—
Aug. 31	29	392	98.3	76.9	50.5	66	3.47	68.3
Oct. 1	60	369	98.9	74.9	45.9	61	2.46	56.4
Oct. 31	90	361	99.2	75.9	46.2	61	1.69	50.2
Dec. 3	123	415	99.2	75.7	44.8	58	.52	35.0
Jan. 2	153	487	99.8	75.1	43.4	58	.26	19.0
Mar. 1	211	384	98.6	75.2	43.9	58	.87	7.5

[1] Precipitation subsequent to previous sample, inches of water.
[2] Average temperature in period subsequent to previous sample.

It is possible to weatherproof metallized pellets by simple and inexpensive means. In a number of qualitative experiments, reduced pellets made from magnetite and synthetic magnetite were coated with one of the following: Bunker "C" oil, petroleum wax, lime, or lime plus 10 percent cement. The samples were weathered as before for a 213-day period extending from August 1964 to April 1965. This testing was purely qualitative since the coating material was generously applied. Table 17 presents the data in terms of residual percent metallization after timed exposure periods. Both the petroleum wax and Bunker "C" oil inhibited oxidation of prereduced pellets made from natural magnetite; the wax worked well on those made from synthetic magnetite. The protection imparted by lime was of limited duration, and lime plus cement was not much better.

Table 17.—Results of weathering coated prereduced pellets—qualitative tests

Coating	Percent metallization after listed days of exposure					
	0	52	85	128	158	213
Natural magnetite:						
Petroleum wax	80	76	77	82	89	79
Bunker "C" oil	80	82	80	83	82	78
Synthetic magnetite:						
Petroleum wax	92	90	87	92	83	71
Lime	92	89	79	74	74	72
Lime plus 10 percent cement	92	89	74	74	80	82

Table 18.—Results of weathering coated prereduced pellets—quantitative tests

Amount of coating oil, lb/ton	Analysis of pellets, percent, after listed days of exposure				
	0	22	125	190	240
No coating oil:					
Fe	82.2	81.2	79.0	76.3	75.0
Fe°	67.8	63.8	61.8	52.0	47.0
Fe°/Fe	.83	.79	.78	.68	.63
8 lb coating oil per ton:					
Fe	82.2	81.7	80.7	79.8	81.5
Fe°	67.8	66.4	65.3	62.6	63.6
Fe°/Fe	.83	.81	.81	.79	.78
13.5 lb coating oil per ton:					
Fe	82.2	81.7	79.5	80.2	82.2
Fe°	67.8	64.5	65.7	62.8	65.2
Fe°/Fe	.83	.79	.83	.78	.79
16.5 lb coating oil per ton:					
Fe	82.2	82.1	79.6	79.9	81.4
Fe°	67.8	68.3	66.0	61.5	66.3
Fe°/Fe	.83	.83	.83	.77	.81

To quantify the requirements for antioxidant, weighed batches of prereduced pellets made from Erie Mining Co. magnetic taconite concentrate were coated with known amounts of Bunker "C". The oil was heated to 100° C and sprayed onto pellets in a rotating metal drum. The quantity of oil retained was determined by back-weighing the pellets. Weathering was performed as before during a 240-day period beginning October 1965. The data are summarized in table 18.

Sampling these batches of prereduced pellets was difficult owing to wide variations in analyses of individual spheres. Other confounding factors were the probable nonuniformity in the coating and the resulting divergence in oxidation rates from pellet to pellet. These engendered some variations in chemical analyses of pellet groups so that trends, rather than individual values, were significant. As the pellets weather and acquire oxygen, dilution diminishes the percentage of all other elements. However, if both total iron and metallic iron are plotted against oxygen, the rate of decline for metallic iron is much more rapid. This is because in a pellet completely reoxidized to hematite metallic iron content is zero, while the total iron would be about 64 percent.

In evaluating the merits of antioxidant coatings, the trends in table 18 reflected by metallic iron analyses and metallization are most revealing. Eight pounds of Bunker "C" oil imparts a high degree of oxidation resistance to 1 ton of reduced pellets. The treatment is estimated to cost 10 to 20 cents per ton and to be effective for at least 4 months of outdoor storage.

Although this means of weatherproofing is simple and inexpensive, prolonged storage of prereduced pellets is not recommended. Slight oxidation takes place during the life of the coating and accelerates thereafter. Any loss of metallic iron diminishes the value of the product, and since prereduced pellets are a relatively costly raw material, the industry should strive for minimum time between production and consumption.

Discussion

Stronger reduced pellets were made from natural magnetite in these campaigns than from synthetic magnetite, although the latter yielded a more metallized product. The average compression value of lignite-reduced natural magnetite pellets was 300 lb, compared with 220 lb for the synthetic; with anthracite it was 520 lb for the natural compared with 275 lb for the synthetic. Although pellets from synthetic magnetite are weaker, they are still a highly acceptable product.

The lignite-reduced pellets display somewhat lower compression resistance than commercial oxide pellets, but the malleability of the metallic variety more than compensates for the difference. Metallic pellets break into a few large fragments after yielding under compressive loads, while oxide pellets disintegrate into many smaller pieces and dust. The tumble and drop tests predict more accurately the ability of pellets to withstand handling. The lowest tumble test on reduced pellets made with 1 percent bentonite yielded 98.9 weight-percent retained on 28-mesh, which is virtually the same as the 99.4 percent displayed by a typical commercial oxide pellet. Other metallized pellets were slightly better than the oxide pellets in the tumble test.

An excess of reductant was present in all experiments to ensure well-reduced pellets; however, the primary objectives at this stage were not to define the limiting quantities of reactants, but to prepare and evaluate the products from several raw materials. It was even more important, perhaps, to account for the iron units entering and leaving the kiln and to determine what fraction of the output would be comprised of usable prereduced pellets. Green iron ore pellets are notoriously fragile, and dried pellets are particularly vulnerable to abrasion; therefore, high losses due to degradation in the kiln would not have been surprising. In addition, there was a 12-foot difference in elevation from the feed conveyor to the kiln floor, so the green pellets were subjected to several changes in velocity and direction. These factors emphasized the need (at least in this pilot plant) for green agglomerates that are perhaps a little stronger than the norm. This achievement was evidenced by the high proportion (about 95 percent) of usable plus ¼-inch prereduced pellets in the kiln discharge.

The use of 0.8 to 1.0 percent bentonite in the pellet mix contributed to the production of better green pellets. This is about 0.2 percent more bentonite than the quantity used to make commercial oxide pellets; however, it appeared to be a small price for such large gains. The higher bentonite has a beneficial effect on the resistance to abrasion and spalling of unfired pellets so that more of them come out of the kiln undamaged, and the formation of accretions and rings is diminished. The extra binder compensates for the potentially damaging motion of the kiln and its burden.

Both lignite and anthracite are satisfactory reductants, but both have their shortcomings as well. Anthracite produced stronger pellets from each concentrate, and the recovery of whole pellets was greater. It is, however, less reactive and to achieve the same degree of metallization as lignite, the feed rate must be reduced or the treatment prolonged. Anthracite also creates an environment higher in sulfur because most of that element is in pyritic form and persists to temperatures at which metallic iron is generated. Lignite sulfur, much of it organic, is volatilized at comparatively low temperatures and carried harmlessly up the stack. Sulfur in the product pellets can be controlled, even with anthracite, by using a sufficient amount of fine limestone or dolomite. This was demonstrated in Canada during pilot production of sponge iron from commercial oxide pellets, using a rotary kiln and anthracite.[7] However, the proportion of limestone to pellets required for this purpose would be higher for anthracite than for lignite.

The primary factor favoring anthracite reductants is the elevated ash-fusion temperature, which is well above the 1,150° C used in the anthracite experiments, so that kiln interior remained relatively free of rings. Lignite, on the other hand, contains readily fusible ash, and could withstand a maximum temperature of only 1,130° C. Up to this point, the operation was quite satisfactory, but the rate of accretion increased rapidly with temperature increase, and above 1,150° C the process was difficult to operate using lignite. Had it been possible to maintain a more highly reducing atmosphere above the bed at the firing end of the kiln, the operating range could have been extended. Other lignite deposits with higher ash-fusing temperatures are in North Dakota and Montana; these deposits merit consideration.

The outstanding attribute of lignite is its reactivity. Under the conditions established in laboratory and pilot plant research on prereduced pellets, lignite effects more reduction in a shorter time and at lower temperatures than other solid fuels. It is an inexpensive commodity, and lignite originating in North Dakota would enjoy lower transportation costs to reduction plants on the Minnesota iron ranges.

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[7] Sibakin, J. G. Development of the SL Direct Reduction Process. Blast Furnace and Steel Plant, v. 50, No. 10, October 1962, pp. 984–985.

PART 4. — BLAST FURNACE SMELTING

N. B. Melcher and P. L. Woolf [1]

BLAST FURNACE CAMPAIGN 1

The previous chapter described the successful production of 140 tons of prereduced pellets from magnetic taconite concentrate for consumption in the Bureau's experimental blast furnace at Bruceton, Pa. The unit, which has since been rebuilt, conformed to the general lines of a full-scale furnace. It had a hearth diameter of 4 feet and a working volume of 305 cubic feet and was approximately 20 feet from tuyeres to stockline. A detailed description of the furnace was given by Melcher and coworkers.[2]

Raw Materials

The first campaign in the blast furnace took place in the spring of 1962 with the 140 tons of Bureau-produced metallized pellets as a principal raw material. Commercial iron oxide pellets were compared with reduced pellets manufactured by The Steel Co. of Canada, Ltd., using the SL/RN process which will be described in part 6.

The coke was purchased at the nominal size of 1⅞ inches by ⅞ inch. It was screened over a ½ inch slotted screen prior to charging. The limestone and dolomite were 2 inches by 1 inch. All the pellets were screened on ⅛-inch mesh prior to use. The chemical analyses of all the materials are shown in table 19, along with an identification key. For simplicity, the pellets will be referred to as A, B, C, D, and E throughout: Pellet A is the commercial Erie pellet, B is the prereduced pellet made by the Bureau from Erie concentrates, C is a commercial pellet manufactured by Hilton mines, Shawville, Quebec, and D and E, respectively, are partially and highly reduced Hilton pellets, made by The Steel Co. of Canada, Ltd.

Table 19.—Identification and analyses of raw materials used in comparing prereduced and oxide pellets in the Bureau of Mines experimental blast furnace [1]

Description	Designation	Analysis, percent				
		Fe	SiO_2	Al_2O_3	CaO	MgO
Commercial oxide pellets from Hoyt Lakes, Minn., concentrate	A	62.8	8.5	0.45	0.63	0.39
Prereduced pellets from Hoyt Lakes, Minn., concentrate	B	79.0	11.9	.77	1.08	.58
Commercial oxide pellets from Hilton Mines, Shawville, Quebec	C	66.7	2.1	.51	.30	1.71
Hilton pellets, partially reduced by The Steel Co. of Canada	D	83.8	3.1	.66	.58	2.06
Hilton pellets, highly reduced by The Steel Co. of Canada	E	91.3	3.0	.64	.39	2.40
Calcite	—	.19	.83	.29	54.6	.64
Dolomite	—	.67	.93	—	29.7	21.6
Limestone	—	.75	3.5	.61	52.3	.56
Gravel	—	3.2	81.8	4.9	2.3	.41
Coke ash	—	6.8	51.0	29.6	3.2	1.3

[1] The proximate analysis of the coke was, in percent, volatile matter, 0.8; fixed carbon, 92.0; and ash, 7.2.

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[1] Former Bureau of Mines supervisory metallurgist, Pittsburgh Metallurgy Research Laboratory, Bruceton, Pa.; now with American Chemical Co., Coraopolis, Pa.

[2] Melcher, Norwood B., J. P. Morris, E. J. Ostrowski, and P. L. Woolf. Use of Natural Gas in an Experimental Blast Furnace. BuMines Rept. of Inv. 5621, 1960, 15 pp.

Experimental Procedure and Results

Four separate tests were made on burdens with varying degrees of reduction at different blast conditions. Complete operating data for all tests are given in table 20. Summary tables containing more specific data will be presented as the results of individual tests are reported.

Table 20.—Complete blast furnace operating data

	Test 1		Test 2		Test 3					Test 4	
	Base	Trial	Base	Trial	Base	Trial 1	Trial 2	Trial 3	Trial 4[1]	Base	Tr ial
Raw materials, pounds per round:											
Pellet:											
A	598.4	300.0	695.5	—	77.5	100.1	—	119.9	58.9	74.0	90.0
B	—	242.0	—	621.2	—	—	—	—	—	—	—
C	—	—	—	—	518.5	—	—	—	—	506.6	316.0
D	—	—	—	—	—	549.9	255.9	—	129.7	—	—
E	—	—	—	—	—	—	445.1	590.1	512.4	—	217.0
Coke	225.5	174.6	246.5	190.5	230.0	195.0	200.0	200.0	172.2	240.0	218.5
Dolomite	50.0	54.4	69.7	85.9	—	—	—	13.0	—	—	—
Calcite	57.2	54.6	—	10.5	33.1	31.4	51.0	48.0	48.0	—	33.7
Limestone	—	—	68.5	67.9	—	—	—	—	—	35.3	—
Gravel	—	—	—	—	—	4.0	27.0	—	27.8	—	5.0
Blast air:											
Rate, scfm	787	795	791	778	785	791	788	785	783	1,389	1,399
Moisture, gr/cu ft	7.0	7.0	7.0	5.4	6.9	7.0	7.0	7.0	7.0	7.0	7.0
Temperature, ° F	1,526	1,561	1,976	2,001	1,734	1,700	1,729	1,750	1,860	1,630	1,593
Fuel:											
Coke, lb/ton	1,128	900	1,075	723	1,105	732	625	621	550	1,220	867
Natural gas, vol pct	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	—	—
Natural gas, cu ft/ton	1,505	1,220	1,344	944	1,315	948	867	849	—	—	—
Metal:											
Production, ton/day	16.19	19.98	18.06	24.47	18.11	24.99	27.42	27.96	32.27	31.80	42.49
Silicon, wt pct	1.15	0.92	0.97	0.86	0.88	0.91	1.05	0.95	0.82	1.04	1.15
Sulfur, wt pct	0.035	0.050	0.028	0.046	0.033	0.044	0.039	0.041	0.049	0.028	0.031
Slag:											
Volume, lb/ton	643	655	634	738	280	243	297	282	308	277	250
SiO_2, wt pct	40.6	41.2	40.3	40.9	32.0	36.3	36.8	37.2	38.5	32.6	34.1
Al_2O_3, wt pct	7.5	6.0	6.3	6.1	12.8	11.9	11.2	11.3	10.3	13.9	13.0
CaO, wt pct	39.9	39.9	39.9	38.4	37.0	31.8	32.6	32.7	30.9	37.1	32.2
MgO, wt pct	10.6	10.9	11.6	11.8	15.7	17.7	17.2	16.3	17.8	14.6	18.1
FeO, wt pct	0.33	0.34	0.28	0.32	0.34	0.26	0.24	0.27	0.30	0.28	0.30
Sulfur, wt pct	0.87	0.72	0.81	0.67	2.26	2.01	1.44	1.42	1.29	2.08	1.78
Basicity:											
$CaO+MgO/SiO_2$	1.24	1.23	1.28	1.23	1.65	1.36	1.35	1.32	—	1.59	1.54
$CaO+MgO/SiO_2+Al_2O_3$	1.05	1.08	1.10	1.07	1.17	1.03	1.04	1.01	1.00	1.11	1.07
Top gas											
Temperature, ° F	534	497	425	392	681	685	633	644	526	543	508
CO, vol pct	20.9	23.5	19.3	30.1	22.3	26.6	27.4	28.9	—	24.3	24.8
CO_2, vol pct	17.1	13.6	20.2	7.0	16.5	10.2	8.8	5.7	—	15.9	14.1
H_2, vol pct	2.4	3.2	3.9	4.3	2.6	3.6	4.1	4.0	—	1.5	1.2
N_2, vol pct	59.6	59.7	56.6	58.6	58.6	59.6	59.7	61.4	—	58.3	59.9
Miscellaneous:											
Dry dust, lb/ton	1.3	1.3	1.9	1.1	2.5	.6	.6	—	.6	1.6	1.8
Shroud air, scfm	10	10	5	—	—	—	—	—	—	—	—
Tuyere pressure, psi	3.34	3.36	3.88	3.59	3.52	3.47	3.75	3.79	—	5.08	5.11

[1] 24-hour trial at high blast temperature.

The first test comprised a base period with Erie pellets (A) followed by a trial period using 50 percent iron from pellet A and 50 perecnt from Bureau-prepared prereduced pellets (B), which comprised a 42-percent/reduced burden. This run was conducted with the hot blast at approximately 1,550° F and enriched with 2.1 percent natural gas. The furnace operation was good, and substantial improvements in both coke and production rates were achieved. The production rate increased 23.5 percent, and the coke rate dropped from 1,128 to 900 lb/ton of metal. The test data are shown in table 21.

The second test included a base period with pellet A and a trial period with a burden consisting of 100 percent pellet B. The blast temperature was raised to 2,000° F, but the natural gas rate was unchanged. Table 21 gives the results of this test. Note that the coke rate was 1,075 lb; when the burden changed to 100 percent pellet B, the coke rate dropped to 723 lb. The production rate increase was 35.4 percent. The slag rate was about 100 lb more with the 352-lb coke-rate decrease. About half of this was owing to the silica pickup during the manufacture of the reduced pellets; the balance was owing to the fact that the Bureau pellets were made from concentrates somewhat higher in silica than the Erie A pellets. Had there been no silica contamination from the kiln accretion, the silica-to-iron ratio in the Bureau product would still have been higher than that in the A pellets.

Owing to the limited supply of Bureau-prepared pellets and the desire to expand the work with par-

Table 21.—Comparison of prereduced-pellet burden with an oxide pellet burden

	Base period	Trial period
TEST 1 (PARTIAL PREREDUCED-PELLET BURDEN)		
Wind rate.....scfm..	800	800
Blast temperature.....° F..	1,561	1,561
Burden, pounds per charge:		
A pellets	598.4	300.0
B pellets	—	242.0
Average Fe in burden.....pct..	62.8	70.0
Burden ratio.....lb Fe/lb coke..	1.67	2.18
Slag rate.....lb/ton pig iron..	643	655
Top gas:		
Excess CO_2, pct of total CO_2[1]	—	8.2
Temperature.....° F..	534	497
CO_2.....vol pct..	17.1	13.6
CO-to-CO_2 ratio	1.22	1.73
Production rate:		
Pig iron.....net tons/day..	16.2	20.0
Increase.....pct..	—	23.5
Coke rate:		
Pounds per net ton pig iron	1,128	900
Decrease.....pct..	—	20.2
Natural gas injection:		
Cubic feet per ton pig iron	1,505	1,220
Wind blown.....pct..	2.1	2.1
Solution loss.....lb C/ton pig iron..	14	−26
TEST 2 (ALL PREREDUCED-PELLET BURDEN)		
Wind rate.....scfm..	800	800
Blast temperature.....° F..	1,976	2,001
Burden, pounds per charge:		
A pellets	695.5	—
B pellets	—	621.2
Average Fe in burden.....pct..	62.8	79.0
Burden ratio.....lb Fe/lb coke..	1.77	2.58
Slag rate.....lb/ton pig iron..	634	738
Top gas:		
Excess CO_2, pct of total CO_2[1]	—	24.2
Temperature.....° F..	425	392
CO_2.....vol pct..	20.2	7.0
CO-to-CO_2 ratio	0.96	4.30
Production rate:		
Pig iron.....net tons/day..	18.1	24.5
Increase.....pct..	—	35.4
Coke rate:		
Pounds per net ton pig iron	1,075	723
Decrease.....pct..	—	32.7
Natural gas injection:		
Cubic feet per ton pig iron	1,344	944
Wind blown.....pct..	2.1	2.1
Solution loss.....lb C/ton pig iron..	60	−32

[1] That portion of CO_2 in top gas originating from sources other than reduction of iron oxides and flux decomposition.

tially reduced burdens, additional tests were conducted with pellets manufactured by the SL/RN process.

In test 3, the base period was conducted with 13 percent pellet A and 87 percent pellet C. Three trial periods followed, the first with a blend of pellets A and D to yield a 62-percent reduced burden, and the second and third trials using 87- and 83-percent reduced burdens, respectively. In this context, reduction refers to oxygen loss rather than metallization. In the second trial, the burden was composed of pellets A and E; in the third trial, it was composed of pellets D and E. Thus, a comparison was made between a burden of partial reduced pellets and one composed of mixed oxidized and reduced pellets, both at similar degrees of reduction. Natural gas was again injected at 2.1 percent of the blast air. The results of this test are shown in table 22. Trial 1 had a coke savings of 373 lb and a production rate increase of 38.1 percent. Before the furnace was stabilized for the two tests using a high percentage of reduced burdens (above 80 percent), there was at least a day of constant operation with an 85-percent reduced burden, at a blast temperature of 1,860 F. The results of these trials are spectacular—nearly 500 lb of coke less than the base coke rate of 1,105 lb was used; in both cases, production increased more than 50 percent. These results were very much in agreement with those of trials 2 and 3 presented in table 22.

Table 22.—Comparison of partial-prereduced and all-prereduced pellet burden with an oxide pellet burden (test 3)

	Base period	Trial 1	Trial 2	Trial 3
Wind rate....scfm..	800	800	800	800
Blast temperature....° F..	1,734	1,700	1,729	1,750
Burden, lb per charge:				
A pellets	77	100	—	120
C pellets	519	—	—	—
D pellets	—	550	256	—
E pellets	—	—	445	590
Average Fe in burden....pct..	66.2	80.6	88.5	86.5
Burden ratio lb Fe/lb coke in charge	1.72	2.68	3.10	3.07
Slag rate....lb/ton pig iron..	280	243	297	282
Top gas:				
Excess CO_2 pct of total CO_2[1]	—	11.6	42.4	78.4
Temperature....° F..	681	685	633	644
CO_2....vol pct..	16.5	10.2	8.8	5.7
CO-to-CO_2 ratio	1.35	2.61	3.11	5.07
Production rate:				
Pig iron....net tons/day..	18.1	25.0	27.4	28.0
Increase....pct..	—	38.1	51.4	54.7
Coke rate:				
Pounds per net ton pig iron	1,105	732	625	621
Decrease....pct..	—	33.8	43.4	43.8
Natural gas injection:				
Cubic feet per ton pig iron	1,315	948	867	849
Wind blown....pct..	2.1	2.1	2.1	2.1
Solution loss, lb C/ton pig iron	73	−22	−63	−72

[1] That portion of CO_2 in top gas originating from sources other than reduction of iron oxides and flux decomposition.

In test 4, the wind rate was raised from 800 to 1,400 cfm, the blast temperature was 1,600 F, and no natural gas was injected. The base burden was composed of the same 87-to-13 mixture of C and A pellets. For the trial, a portion of the C pellets was replaced by E pellets to yield a 41-percent reduced burden. The purpose of this test was to determine whether some of the benefits of natural gas might have been diminished with a partial reduced burden. As shown in table 23, the coke decreased by 353 lb/ton; the production rate increased 33.6 percent.

DISCUSSION

There are several ways to present furnace data as a function of prereduction. Important statistics such as coke and production rates may be plotted against (1) metallization of the burden, (2) prereduction, that is, the precent of available oxygen removed, (3) total iron analysis of the burden, or (4) simply the percent of prereduced pellets in the

Table 23.—Comparison of a partial prereduced-pellet burden with an oxide pellet burden (test 4) (no natural gas injection)

		Base period	Trial period
Wind rate	scfm	1,400	1,400
Blast temperature	°F	1,630	1,593
Average Fe in burden	pct	66.2	74.7
Burden ratio	lb Fe/lb coke	1.60	2.13
Slag rate	lb/ton pig iron	277	250
Production rate:			
Pig iron	net tons/day	31.8	42.5
Increase	pct	—	33.6
Coke rate:			
Pounds per net ton pig iron		1,220	867
Decrease	pct	—	28.9
Solution loss	lb C/ton pig iron	168	50

burden. Actually, all four yield similar plots. In the graphical presentation of the data, oxygen removed, or percent prereduction, was used as the independent variable. Prereduction may be calculated from the formula

$$R = (100 - 2.33 \frac{O_r}{Fe_r})$$

where R = percent prereduction, Fe_r = iron content of the burden, and O_r = oxygen combined with Fe_r. As O_r/Fe_r nears the oxygen-to-iron ratio of hematite, R becomes zero. Similarly, as the burden becomes more metallic, O_r approaches zero and R approaches 100.

Figure 18 shows a plot of the decrease in coke rate in pounds per ton of hot metal (lb/THM) versus prereduction of the burden for the tests with natural gas injection. The relationship is linear; each percent of available oxygen removed by the prereduction results in a coke savings of 5.7 lb. The savings is not the entire fuel economy since there was also a reduction in the natural gas utilized per ton of metal to the extent of 5 cubic feet per percent of oxygen removed. In figure 18, the point farthest above the line is the trial with the burden consisting entirely of Bureau-prepared pellets. Most of the difference can be attributed to the high slag volume. The fourth test at a higher wind rate (without natural gas) yielded a greater saving in coke per unit of oxygen removed. The coke rate was lowered 353 lb for 41-percent oxygen removal, or 8.6 lb for each percent, compared with 5.7 lb of coke and 5 cubic feet of natural gas for the other tests. This suggests that some of the benefits of natural gas may be negated with partially reduced burdens.

The effect of prereduction on productivity is shown in figure 19. Production increases approximately 0.6 percent for each percent of available oxygen removed from the burden.

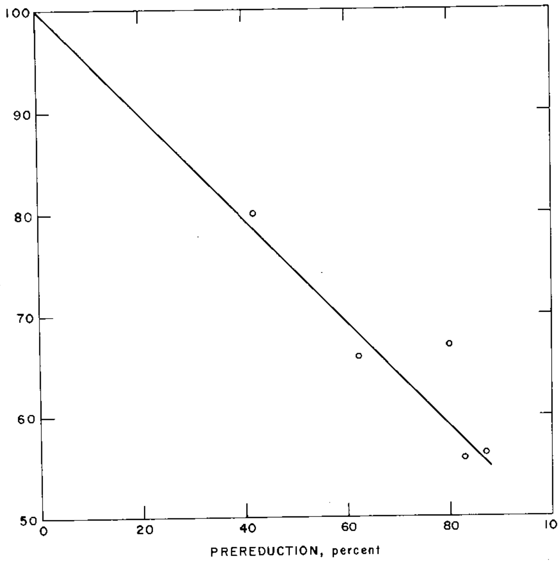

Figure 18.—Effect of prereduction on coke rate.

The comparison of the blast furnace operation on highly reduced burdens and cupola operation is inevitable. If the coke rates in these tests were extrapolated to fully reduced burdens, they would have been much higher than those normal to a cupola. In cupola melting, the coke size and reactivity are controlled to obtain the highest heat per unit of coke consumed at the tuyeres; that is, much of the coke is burned to CO_2. In normal blast furnace operation any CO_2 is immediately reduced by carbon to CO; the net heat effect is that of coke burning to CO. The latter yields less than one-third the heat of complete combustion. Thus, the cupola utilizes its available fuel more effectively in terms

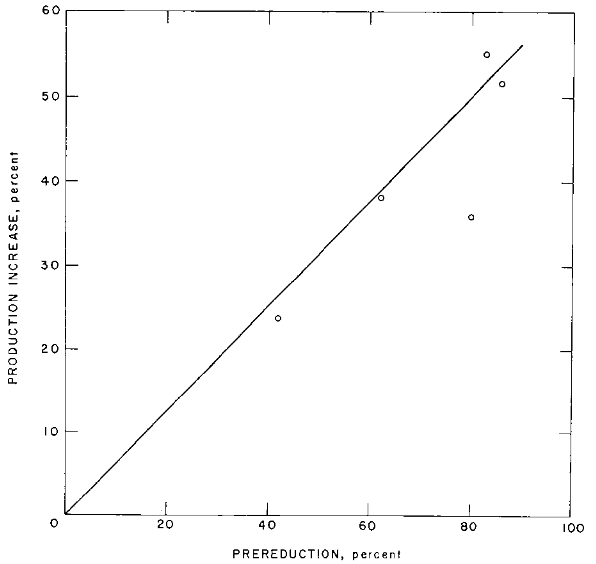

Figure 19.—Effect of prereduction on production rate.

of heat production. Another reason for the lower fuel rates with cupolas is that virtually no reduction of silica and manganese oxide takes place.

As the cupola operation is approached and the proportion of coke in the blast furnace charge decreases some of the coke may burn to CO_2 and leave the tuyere zone without reduction to CO. In some of these tests, with the highly reduced burdens, the CO_2 content of the top gas, as calculated by the wind and coke rates, was higher than could be accounted for by the CO_2 in the flux and the formation of CO_2 from indirect reduction in the stack. This phenomenon may be referred to as negative solution loss and is the result of either carbon deposition ($2CO \rightarrow C + CO_2$) or burning of the carbon at the tuyeres to CO_2 ($C + O_2 \rightarrow CO_2$). The two reactions are indistinguishable from analysis of the data. No furnace probing was performed; this might have provided the answer. Conditions were more favorable for carbon deposition in these tests than in any other blast furnace operation; the CO-to-CO_2 ratio in the stack was very high, and the reduced iron pellets would probably catalyze the reaction. The second explanation would account for the analogy to the cupola and the relatively poor desulfurization and higher slag manganese. This would incate a higher oxygen potential in the hearth.

It has been mentioned that the full benefit of natural gas injection is not realized when the burden is prereduced The greater coke savings per unit of reduction has been cited. Perhaps another indication is the change in hydrogen utilization. Figure 20 is a plot of hydrogen utilization versus degree of reduction. The tests with natural gas show a decrease as the prereduction of the burden increases, whereas the opposite appears to be true without natural gas.

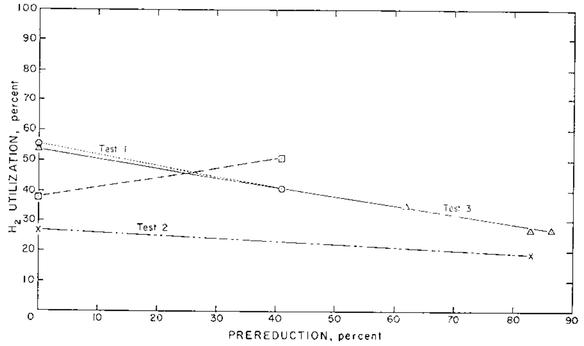

Figure 20.—Hydrogen utilization versus prereduction of burden.

The values for solution loss have been calculated using the wind and coke rates. They do not always agree with values calculated from the top-gas analysis. In the experimental furnace it is assumed that wind rate measurement is more reliable than top-gas analysis. For this reason values such as the CO-to-CO_2 ratio and hydrogen utilization should be considered approximate. This is particularly true when the CO_2 content of the top gas is very low.

Table 24 presents comparative heat balances for test 2, in which a base burden was entirely replaced by Bureau-prepared reduced pellets. Almost 1,400,000 fewer Btu were required with the prereduced burden. There are three main sources for this gain: (1) The net heat of reduction, which yields a saving of 600,000 Btu; (2) the sensible heat of the top gas, which is 345,000 Btu less with the reduced burden; and (3) cooling losses, in that 500,000 Btu is saved by the higher productivity. However, the sensible heat of the slag from the oxide pellets was 80,000 Btu less than that obtained with the reduced pellets. As stated, this was owing in part to SiO_2 contamination in this lot of prereduced pellets; improvement in manufacturing technique will result in a lower coke rate and higher productivity than were found in this test.

As partially reduced pellets were charged to the furnace, the permeability of the burden increased even though the coke rate decreased. This is indicated by the decreased resistivity index for the first trial of the third test as shown in table 25. With a further increase in the level of prereduction, the burden resistivity increased as coke rate

Table 24.—Comparative heat balances for smelting oxide and partially reduced pellets (thousand Btu per ton)

	100 percent pellet A	100 percent pellet B
Input:		
Sensible heat of blast	2,411.5	1,781.1
C(coke) + $\frac{1}{2}O_2$ = CO	3,054.6	2,232.0
C(coke) + H_2O = CO + H_2	−195.5	−109.6
CH_4 + $\frac{1}{2}O_2$ = CO + $2H_4$	54.2	40.1
Total	5,324.8	3,943.6
Consumption:		
Net heat of reduction of iron oxides	321.3	−279.1
SiO_2, MnO, and P_2O_5 + C	172.2	158.2
Sensible heat of dry top gas	591.2	361.0
Sensible heat of top-gas water	135.3	19.8
Sensible heat of metal	1,080.0	1,080.0
Sensible heat of slag	507.2	590.4
Net slag reactions	295.8	292.0
Cooling losses by difference	2,221.8	1,721.3
Total	5,324.8	3,943.6

decreased, to a value similar to the base. These results suggest that ore fines were not generated during further reduction in the stack. The permeability improvement was then offset by the decrease in coke rate when the prereduction was increased from 62 to 85 percent.

Table 25.—Resistivity[1] of test burdens[2] in test 3

Trial	Prereduction, percent	Resistivity index	Coke rate, lb/ton
Base	0	0.51 x 10^{-3}	1,105
1	62	.32 x 10^{-3}	732
2	87	.49 x 10^{-3}	625
3	83	.47 x 10^{-3}	621

[1] Resistivity index $= \frac{(P_b^2 - P_t^2)\ (WV)^2}{(WR)^2\ \mathrm{X}\ (WH)^3}$

where P_b = blast pressure, psia,
P_t = top pressure, psia,
WR = wind rate, scfm,
WH = working height, ft,
and WV = working volume, $ft.^3$

[2] Stapleton, J. M., R. L. Stephenson, and D. H. Regelin. Future of Ironmaking. J. Metals, v. 14, No. 10, 1962, pp. 749–753.

BLAST FURNACE CAMPAIGN 2

The study of prereduced burdens in the blast furnace was resumed in 1965. The improved operating results achieved in the earlier tests had generated considerable interest throughout the industry so that additional investigative effort was warranted. Consequently, experimentation with prereduced materials was included as part of an extensive blast furnace investigation conducted under a cooperative agreement between the Bureau of Mines and Blast Furnace Research, Inc.

Raw Materials

Three very different prereduced materials were used in these tests: Prereduced pellets prepared by the SL/RN process from high-grade concentrates, prereduced ore and briquets produced from Wabana ore from Newfoundland, Canada, and iron powder prepared from mill scale by hydrogen reduction.

The prereduced Wabana ore and briquets were specially produced for these tests. They were prepared from sized heavy medium concentrate (1 inch by 3/16 inch) by the following procedure: The sized ore was charged to a rotary kiln; Nova Scotia bituminous coal, added at intervals along the length of the kiln, was used partly for fuel and partly to supply a reducing atmosphere. A natural gas burner was used at the discharge end of the kiln. The charge was heated to 1,800° F, maintained at that temperature for 5 to 6 hours, dropped into a rotary cooler, and then screened at 3/16 inch to remove fine coke, ash, and ore particles. The plus 3/16 inch material was separated into two products by a single-drum magnetic separator. One of these products, the concentrate, was the reduced ore used in the blast furnace tests. The other product (tailings) was sent to a grinding and magnetic separation circuit to produce a middling which was briquetted for other blast furnace tests.

Iron powder was prepared from mill scale by the H-Iron process.[3] The mill scale, which was heated to 600° F to dry it and burn off any oil, was screened and magnetically separated to remove waste materials. After grinding and classifying, the scale was reduced in three stages using hydrogen at 500 psi and a temperature somewhat below 1,000° F. After reduction was complete and the system had been purged with nitrogen, the reduced powder was rendered nonpyrophoric by heating it in a passivator to 1,300° F.

The reference materials were Carol Lake pellets and Wabana ore concentrate. The average chemical analyses of the raw materials including coke and fluxes are shown in table 26. The pellets, ore, and briquets were screened on ⅛-inch to yield the size analyses shown in table 27. The prereduced briquets were nominally 1⅞ by 1½ inch; the coke was minus 1½ plus ½ inch, and fluxes were minus 1 plus ⅜ inch. The powdered iron screen analysis is given in table 28.

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[3] Statler, H. H., and R. A. Lubker. Performance of First Commercial H-Iron Reduction Plant. Ch. in Iron Ore Reduction, ed. by R. R. Rogers. The Macmillan Co., New York, 1962, pp. 163–173.

Table 26.—Chemical analyses of raw materials, percent

Material	Fe	SiO_2	Al_2O_3	CaO	MgO	Mn	P
Carol Lake pellets	64.7	5.47	0.41	0.64	0.20	0.79	0.013
S–L pellets	89.5	3.95	.68	.47	2.05	.06	.001
Wabana ore	51.7	12.7	4.75	2.75	.50	.10	.84
Prereduced Wabana ore	69.6	13.4	5.13	2.62	.51	.09	.79
Prereduced Wabana briquets	67.9	9.84	4.87	2.79	.60	.09	.80
Powdered iron	98.7	.15	—	—	—	.59	.008
Calcite	.40	1.32	.44	54.1	.96	—	—
Dolomite	.48	1.14	.27	29.5	21.4	—	—
Gravel	.41	71.2	9.65	7.61	1.47	—	—
Coke ash[1]	9.63	49.8	27.8	2.56	.99	—	—

[1] The proximate analysis of the coke was, in percent, 1.2 volatile matter, 89.8 fixed carbon; and 9.8 ash.

Table 27.—Screen analyses of raw materials, weight-percent

Size	Carol Lake pellets	S-L pellets	Wabana ore	
			Untreated	Prereduced
Plus 1 inch	—	—	7.4	—
Minus 1 inch plus ½ inch	35.2	11.3	59.6	22.7
Minus ½ inch plus ⅜ inch	57.1	68.8	20.0	33.6
Minus ⅜ inch plus 3 mesh	4.4	17.7	7.5	29.4
Minus 3 mesh plus 10 mesh	1.0	2.2	2.6	13.5
Minus 10 mesh	3.3	—	2.9	.8

Table 28.—Screen analysis of powdered iron

Size, mesh	Weight-percent	Size, mesh	Weight-percent
Plus 10	0.8	Minus 80 plus 100	5.7
Minus 10 plus 20	8.3	Minus 100 plus 120	3.5
Minus 20 plus 40	43.6	Minus 120 plus 200	6.1
Minus 40 plus 60	15.2	Minus 200	4.7
Minus 60 plus 80	12.1		

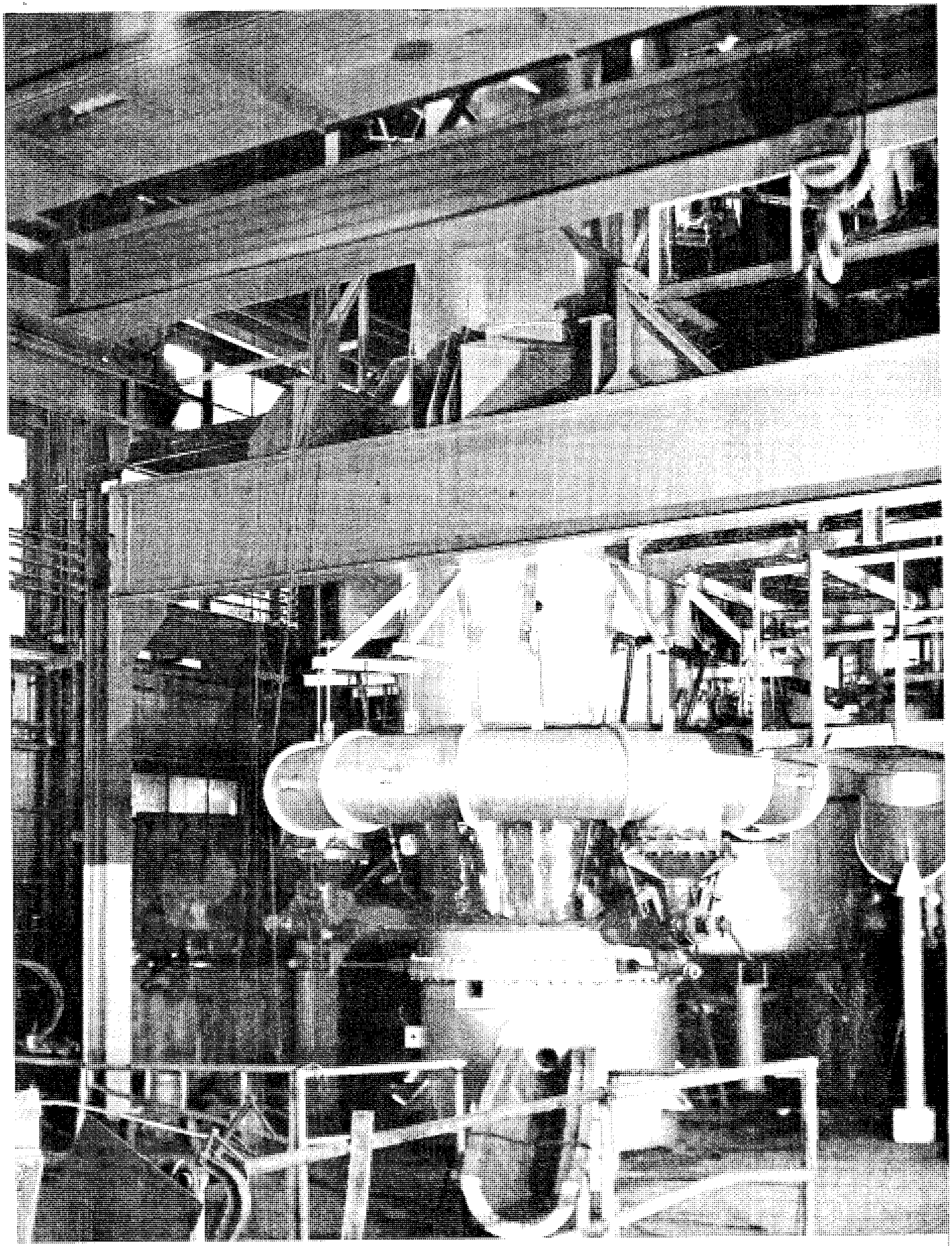

Figure 21.—Bureau of Mines experimental blast furnace.

Facilities

The experimental blast furnace used for the 1962 tests with prereduced pellets was replaced by a new furnace which was blown-in in April 1964. The new furnace has similar lines to its predecessor and is designed, along with its auxiliaries, to permit operation at wind rates up to 2,500 scfm with top pressures up to 40 psig. The working height is 20 feet, which is about one-fourth that of a large commercial furnace. The furnace diameter at tuyere level is 4 feet, which is only one-seventh of the corresponding dimension of a modern furnace. Thus, the experiment furnace is not a true scaled-down model, although it does conform generally to commercial furnaces. The furnace hearth diameter is the minimum necessary to yield good experimental data. The shaft dimensions were designed to duplicate either solids or gas retention times that are encountered in full-scale operation.

The hearth is bolted to the lower bosh, which is detachable for ease in cleaning the furnace at the end of a campaign. Three water-cooled cast-copper tuyeres are equally spaced around the furnace just above the hearth-bosh juncture. The inner lining of the furnace shaft is of hard-burned, super-duty fireclay brick. The bosh and hearth are lined with carbon brick to minimize sticking of the charge materials.

The blast air is heated by two pebble stoves which are fired by natural gas-air burners. On the blast heating cycle, air enters a stove through the same duct used to discharge combustion products when the stove is being fired. The air passes up through the heated pebbles and is discharged through an alumina-brick-lined hot blast main for delivery to the tuyeres.

The furnace has a three-bell top and a distribution hopper. The silicone seals, which are mounted in the two upper bell hoppers, provide excellent insulation. Two totally enclosed rods are used to follow the level of the stock. Figure 21 is a photograph of the furnace, figure 22 shows a cross-section of the furnace and one of the pebble stoves. The injection system used for the iron powder was that used previously for coal.[4] The iron powder was injected through water-cooled lances inserted through the face plates and extending to the nose of the tuyeres.

Experimental Procedure

Three series of tests were conducted. The first series comprised a reference period using Carol Lake pellets and three tests substituting 30, 60, and 85 percent SL/RN pellets for Carol Lake pellets. Because the SL/RN pellets were made from a lower gangue concentrate than Carol Lake pellets, gravel was added with the SL/RN pellets to simulate the chemical composition of prereduced Carol Lake pellets. The wind rate was 1,500 scfm, the blast temperature was 2,200° F, and the furnace was controlled by varying blast moisture within the range 20 to 24 grains per cubic foot. The aim was to produce basic iron with 0.85 to 1.15 percent silicon and a maximum sulfur content of 0.04 percent. Burden ratio was adjusted to maintain blast moisture within the stated range.

The reference burden for the second series consisted of 40 percent Carol Lake pellets and 60 percent Wabana ore. The blast conditions were changed to 1,200 scfm wind, 2,500° F blast temperature, and 24 to 28 grains per cubic foot of air moisture for control. After the base period was established, the raw Wabana ore was replaced by prereduced Wabana ore and a test period was obtained. The Carol Lake pellets were then replaced by prereduced Wabana ore; thus the burden consisted entirely of prereduced ore. After a 3-day test the prereduced ore was completely replaced by the prereduced briquets and a comparison test was obtained.

The prereduced ore and pellets were charged in the top of the furnace in the normal manner. Prereduced materials may also be injected into the tuyeres of the furnace. The third series of tests was a study of powdered-iron injection. The blast conditions of the prereduced-pellet tests were adopted for these tests. Two levels of iron injection were tested, 20 and 40 percent of the total iron. Metal chemistry was controlled by varying blast moisture. Burden ratio was adjusted to keep the blast moisture between 22 and 25 grains per cubic foot.

RESULTS AND DISCUSSION

Prereduced Pellets

Three successful test periods were obtained with 30-, 60-, and 85-percent prereduced pellets. A brief run was made with 100-percent prereduced pellets; although the operation was good, there was not enough time to obtain reliable data. The complete operating data for the base and three additional tests are given in table 29. Furnace operation was very good throughout. Metal-sulfur

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[4] Woolf, P. L., and H. H. Lowry. Coal Injection in an Experimental Blast Furnace. Presented at 23d Ironmaking Conf., Pittsburgh, Pa., April 13–15, 1964. American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, v. 25, 1967, pp. 217–220.

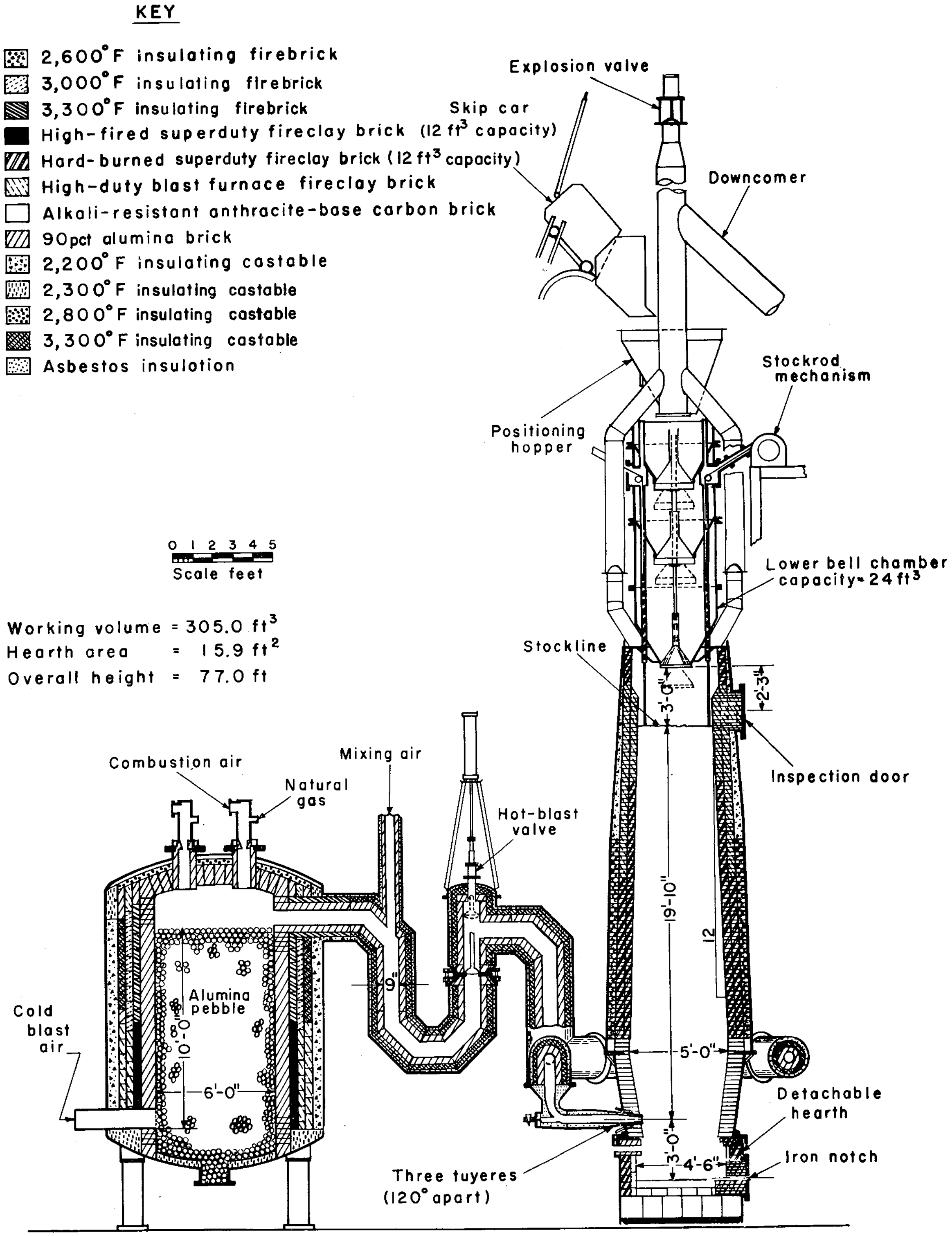

Figure 22.—Cross section of Bureau of Mines experimental blast furnace.

content was high for the first day of the 60-percent reduced-pellet test, but was brought under control for the second and third days of the test period.

Table 30 shows the principal effects of increasing proportions of prereduced pellets on coke and production rates. The coke savings were large, exceeding 500 lb per ton with 85-percent prereduced pellets. The effect on productivity was also very pronounced; with 85 percent SL/RN pellets output increased from 40.9 to 71.6 tons per day. Slag volumes decreased moderately with increased percentages of prereduced pellets.

Table 29.—Blast furnace operating data from prereduced-pellet tests

	Base	Test 1	Test 2	Test 3
Burden composition, percent				
Carol Lake pellets	100	70	40	15
SL/RN pellets	—	30	60	85
Burden ratio	2.50	2.96	3.39	3.61
Calcite, lb/ton	227	210	256	221
Dolomite, lb/ton	122	89	19	—
Gravel, lb/ton	—	35	72	95
Wind rate, scfm	1,513	1,500	1,500	1,500
Blast temperature, ° F	2,200	2,200	2,200	2,200
Blast moisture, gr/cu ft	21	23	20	22
Wind required, cu ft/ton	50,750	39,460	32,570	28,720
Coke rate, lb/ton	1,147	881	705	613
Production rate, tpd	40.9	52.7	63.7	71.6
Metal silicon, pct	1.10	1.04	0.99	0.99
Metal sulfur, pct	0.035	0.039	0.045	0.033
Slag volume, lb/ton	470	428	420	371
Slag composition, percent:				
SiO_2	36.6	37.2	38.6	38.2
Al_2O_3	10.4	10.0	11.7	10.1
CaO	40.1	38.2	38.6	37.6
MgO	10.5	11.8	11.2	12.0
Fe	0.34	0.32	0.26	0.20
S	1.59	1.49	1.14	1.20
Mn	0.27	0.18	0.16	0.10
Basicity: $CaO + MgO/SiO_2 + Al_2O_3$	1.08	1.06	0.99	1.02
Top gas temperature, ° F	340	260	200	250
Top gas composition, volume-percent:				
CO	22.5	23.3	26.1	29.3
CO_2	19.7	17.9	13.6	8.8
H_2	2.3	2.7	3.0	3.7
N_2	55.5	56.1	57.3	58.2
Top pressure, psi	2.5	1.9	1.8	2.0
Flue dust, lb/ton	2	1	1	1
Tuyere pressure, psi	7.1	6.6	6.5	6.8
Prereduction, pct	0	33.1	59.9	78.8

Table 30.—Effects of prereduction on furnace operation at 1,500 scfm wind rate and 2,200° F blast temperature

	Base	Test 1	Test 2	Test 3
Burden, percent:				
Carol Lake pellets	100	70	40	15
SL/RN pellets	0	30	60	85
Blast moisture, gr/cu ft	21	23	20	22
Production rate:				
Net tons per day	40.9	52.7	63.7	71.6
Increase, pct	—	28.9	55.7	75.1
Coke rate:				
Pounds per ton hot metal	1,147	881	705	613
Decrease, pct	—	23.2	38.5	46.6
Slag volume, lb/THM	470	428	420	384

Figure 23 shows the effect of prereduction on coke rate. The relationship shows slightly decreasing benefits with increasing reduction. The savings drops from 8.0 lb for 30 percent SL/RN to 6.8 lb for 85 percent SL/RN for each percent prereduction. The coke rate of 613 lb with 85 percent SL/RN pellets is the lowest ever achieved in the experimental furnace for a test period.

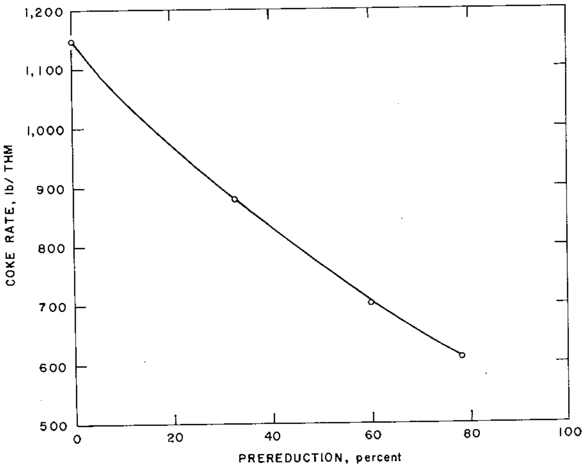

Figure 23.—Effect of prereduction on coke rates.

Figure 24 shows that the effect of prereduction on production rate was almost linear. Each percent of prereduction resulted in productivity increase of approximately 1.0 percent.

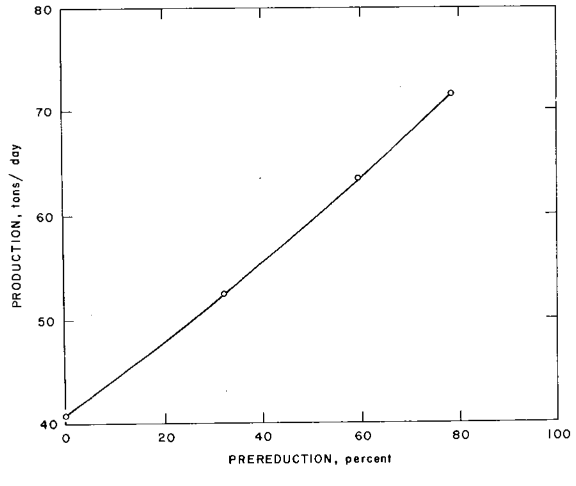

Figure 24.—Effect of prereduction on productivity.

Top gas CO-to-CO_2 ratio increased with increased prereduction. This was not a result of decreased shaft efficiency; there was simply less oxygen to remove. The effect of prereduction on CO-to-CO_2 ratio and solution loss is shown in figure 25. With 85 percent SL/RN pellets, the solution loss was zero. In the 1962 tests, at lower wind (800 scfm) and with natural gas enrichment, negative solution loss was obtained with prereduced pellets. The limited data with 100 percent SL/RN pellets indicated a continuation of the curve to a negative value for solution loss.

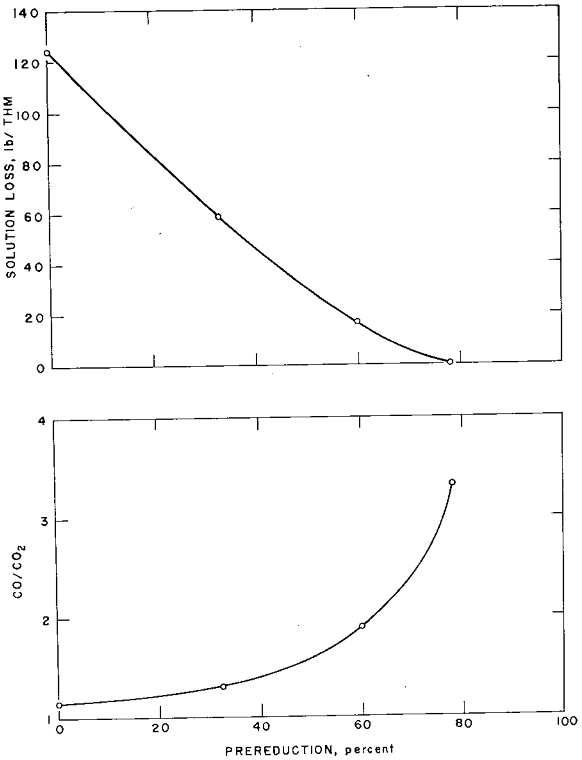

Figure 25.—Effect of prereduction on top gas CO-to-CO_2 ratio and solution loss.

In 1964, The Steel Co. of Canada performed tests with a prereduced burden in a commercial blast furnace with a hearth diameter of 18 feet. The raw materials include highly metallized Hilton pellets prepared by the SL/RN process, and Hilton and Erie oxide pellets. During the test periods (6 to 8 days each), 70 percent of the furnace charge was oxide pellets; the balance was made up of metallized pellets. The metallic iron content of the mixture was 34.7 percent. This netted a 7-percent productivity increase and a 7.4-percent coke rate decrease for each 10 percent metallic iron. These

Table 31.—Blast furnace operating data for prereduced ore and briquets

	Base	Test 1	Test 2	Test 3
Burden composition, percent:				
Carol Lake pellets	40	40	—	—
Raw Wabana ore	60	—	—	—
Prereduced Wabana ore	—	60	100	—
Prereduced Wabana briquets	—	—	—	100
Burden ratio	2.12	2.57	2.13	2.17
Calcite, lb/ton	320	269	290	292
Dolomite, lb/ton	492	448	632	429
Wind rate, scfm	1,200	1,200	1,200	1,200
Blast temperature, ° F	2,500	2,500	2,500	2,500
Blast moisture, gr/cu ft	27	27	25	27
Wind required, cu ft/ton	70,750	50,250	59,560	56,730
Coke rate, lb/ton	1,577	1,087	1,236	1,248
Production rate, tpd	23.3	33.2	28.0	29.6
Metal silicon, pct	1.09	1.13	1.08	0.86
Metal sulfur, pct	0.032	0.034	0.035	0.058
Slag volume, lb/ton	1,016	882	1,170	975
Slag composition, percent:				
SiO_2	34.3	33.2	32.5	32.2
Al_2O_3	14.6	14.8	16.3	17.3
CaO	37.8	37.8	37.0	37.7
MgO	11.6	12.3	12.2	10.3
Fe	0.33	0.31	0.33	0.37
S	1.13	1.35	1.37	1.64
Mn	0.13	0.10	0.06	0.14
Basicity, $CaO + MgO/SiO_2 + Al_2O_3$	1.01	1.04	1.01	0.97
Top gas temperature, ° F	400	370	460	470
Top gas composition, volume-percent:				
CO	28.9	29.5	36.0	34.8
CO_2	13.8	12.6	4.8	7.6
H_2	4.0	3.9	5.9	4.8
N_2	53.3	54.0	53.3	52.8
Top pressure, psi	2.5	2.2	2.3	2.6
Flue dust, lb/ton	1	1	2	2
Tuyere pressure, psi	6.4	6.4	6.5	6.3

gains were achieved without fuel injection.

"A comparison of the results with those of the Bureau's experimental furnace shows reasonably good agreement. Prereduction of the industrial burden proved to be slightly less rewarding in both productivity and coke rate, but this may be explained by differences in burden composition, furnace heat losses and possibly to furnace stack efficiency."[5]

Prereduced Ore and Briquets

Full operating data for the tests with the Wabana ore and briquets are given in table 31. Initially, an attempt was made to operate at the same blast conditions as the pellet tests. However, erratic burden movement, including hanging and large swings in metal chemistry, necessitated a blowing-rate reduction from 1,500 to 1,200 scfm. The blast temperature was raised to 2,500° F to obtain a coke rate comparable to what might be experienced in a large furnace with the base burden of 40 percent Carol Lake and 60 percent Wabana ore. A satisfactory base period was obtained. A good test period was achieved when the raw Wabana was replaced by the reduced Wabana. Table 32 shows principal data from these two periods, together with the data from two other tests, one with 100 percent reduced Wabana ore and one with briquets.

The base and the first test show that the benefits of prereducing ore are similar to those obtained by prereducing pellets. The degree of reduction was approximately 45 percent for the first test. The coke rate was lowered 490 lb from the base value of 1,577 lb. Productivity increased from a base of 23.3 to 33.2 tons per day, an increase of 42.5 percent.

There was a complication in calculating the percent prereduction of the Wabana ore. During prereduction, considerable beneficiation was achieved as evidenced by the change in the iron-to-silica ratio from 4.1 to 5.2. Wabana ore contains free quartz, some of which, as well as other gangue material, was removed during preparation of the

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[5] Peart, J. A., and F. J. Pearce. The Operation of a Commercial Blast Furnace With a Prereduced Burden. J. Metals, v. 17, No. 12, December 1965, pp. 1396–1400.

Table 32.—Principal operating data at 1,200 scfm wind rate and 2,500° F blast temperature showing effects of prereduction and briquetting

	Base	Test 1	Test 2	Test 3
Burden, percent:				
Carol Lake pellets	40	40	—	—
Raw Wabana ore	60	—	—	—
Prereduced Wabana ore	—	60	100	—
Prereduced Wabana briquets	—	—	—	100
Blast moisture, gr/cu ft	27	27	25	27
Production rate:				
Net tons per day	23.3	33.2	28.0	29.6
Increase, pct	—	42.5	20.2	27.0
Coke rate:				
Pounds per ton hot metal	1,577	1,087	1,236	1,248
Decrease, pct	—	31.1	21.6	20.9
Slag volume, lb/THM	1,016	882	1,170	975

reduced ore. Prior to prereduction, there was an ignition loss of 2.8 percent. There was no ignition loss with the prereduced ore, which indicated that combined water and/or CO_2 from carbonates was driven off during prereduction. Only an approximate precent prereduction may be obtained because of the inability to precisely relate the final product to the original through a change in oxygen content.

No direct comparison can be made with the 100 percent reduced Wabana ore since there was no test with 100 percent raw ore. Coke and production rates were not as good as with the mixed burden of test 1; thus it may be concluded that reduced

Table 33.—Blast furnace operating data for iron powder injection tests

	Base	Test 1	Test 2
Carol Lake pellets, pct	100	100	100
Burden ratio, pellets/coke	2.50	2.23	1.82
Calcite, lb/ton	227	179	141
Dolomite, lb/ton	122	119	91
Iron powder, lb/min	0	14.8	28.2
Iron powder, lb/ton	0	432	788
Iron powder/total iron	0	0.23	0.42
Wind rate, scfm	1,500	1,500	1,500
Blast temperature, ° F	2,200	2,300[1]	2,300[1]
Blast moisture, gr/cu ft	21	24	25
Wind required, cu ft/ton	50,750	42,960	40,990
Coke rate, lb/ton	1,147	987	930
Production rate, ton/day	40.9	48.7	50.8
Metal silicon, pct	1.10	0.83	0.75
Metal sulfur, pct	0.035	0.039	0.033
Slag volume, lb/ton	470	370	297
Slag composition, percent:			
SiO_2	36.6	35.9	35.5
Al_2O_3	10.4	10.6	12.0
CaO	40.1	39.8	38.8
MgO	10.5	11.0	10.9
Fe	0.34	0.32	0.38
S	1.59	1.67	1.80
Mn	0.27	0.26	0.28
Basicity, $CaO + MgO/SiO_2 + Al_2O_3$	1.08	1.09	1.04
Top gas temperature, ° F	340	380	520
Top gas composition, volume-percent:			
CO	22.5	23.6	25.9
CO_2	19.7	18.1	14.8
H_2	2.3	3.2	3.4
N_2	55.5	55.1	55.9
Top pressure, psi	2.5	2.3	2.2
Flue dust, lb/ton	2	5	8
Tuyere pressure, psi	7.1	7.6	7.9

[1] Equivalent to 2,200° F when corrected for cooling water and carrier air.

Wabana ore is not as good as Carol Lake pellets in the blast furnace, despite containing 5 percent more iron. The test with the briquets was conducted to determine whether a competent briquet could be made from the reduced fines. The furnace operation was good, but metal sulfur content was high. The supply of briquets was exhausted before sulfur could be controlled below the specification of 0.04 percent. The coke rate was about the same for the ore and briquets; however, the production rate was higher with the briquets.

Iron Powder Injection

The base raw material for the iron injection test was Carol Lake pellets. Following the Wabana ore tests, the furnace was burdened with Carol Lake and the reference period of the SL/RN pellet tests was repreated. A good check was obtained within 10 lb of coke and 500 lb of hot metal per day at equal delay times. The complete results of two tests with powdered iron injection are given in table 33. The principal operating data, together with the base data, are shown in table 34.

Table 34.—Operating data with iron powder injection at 1,500 scfm wind rate

	Base	Test 1	Test 2
Blast temperature ° F.	2,200	[1]2,300	[1]2,300
Blast moisture gr/cu ft.	21	24	25
Iron powder lb/ton.	—	432	788
Production rate:			
Net tons per day	40.9	48.7	50.8
Increase pct.	—	19.3	24.2
Coke rate:			
Pounds per ton hot metal	1,147	987	930
Decrease pct.	—	13.9	18.9
Slag volume lb/THM.	470	389	297
Metal silicon pct.	1.10	0.83	0.75
Metal sulfur pct.	0.035	.038	.033

[1] Equivalent to 2,200° F when corrected for cooling water and carrier air

There were two tests to determine the feasibility of iron powder injection. Test 1 was successful; the operation proceeded smoothly with substantial benefits in coke and productivity. Test 2 was conducted at double the injection rate of test 1 and was also a success, although the unit benefits were much less. Values of 22.6 and 41.1 percent prereduction are obtained when the iron powder is considered as a prereduced burden material and the degree of reduction for the overall burden is calculated. The coke savings of 7.1 and 5.3 lb per percent reduction for tests 1 and 2, respectively, were less than with prereduced materials charged in the top of the furnace. Another approach to the evaluation of powdered iron injection is to compute the portion of the coke rate needed to smelt the oxide pellets; the balance of the coke would be that necessary to melt and carburize the iron powder. For the first test, 77.4 percent of the iron was from Carol Lake pellets and would have required 888 lb of coke, leaving 99 lb of coke for processing 432 lb of powder. For the second test, the 58.9 percent of the iron from Carol Lake would have required 676 lb of coke, leaving 254 lb of coke for 788 lb of iron powder. Expressed on a ton-of-hot-metal basis, the coke rates were 438 and 618 lb for the iron powder in tests 1 and 2, respectively.

General

Despite the different materials and blast conditions, there was a strong similarity in the effect of prereduction on the high-grade pellets and the low-grade ore. The first test with the reduced Wabana ore was at approximately 45-percent prereduction. The coke savings of 490 lb was greater than the 442 lb saved with the pellet burden at the 58.8 percent prereduced level. However, considering the additional effect of the beneficiation as well as the possible influence of a higher initial coke rate the agreement appears fairly good.

The production rate increase with the reduced Wabana ore was 42.5 percent, or 0.94 percent for each percent prereduction, compared with 0.96 percent with the SL/RN pellets.

Effects on solution loss and CO-to-CO_2 ratio obtained with Wabana ore were similar to those with the pellet burden. The base solution loss of 138 lb was decreased to 26 lb with the 45-percent prereduced burden. Top gas CO-to-CO_2 ratio increased from the base value of 2.09 to 2.34.

Owing to different blast conditions, materials, and initial base coke and production rates, the effect of prereduction may perhaps be best expressed by relating relative changes in initial rates to the degree of reduction. Figure 26 illustrates the relationship between the precent decrease in coke rate and percent prereduction for the SL/RN pellets and Wabana ore used in this investigation. Also in 1962 a test was conducted with SL/RN pellets at a lower wind rate and lower blast temperature. There is very little deviation from a smooth curve, which shows a slight decrease in coke savings with increased prereduction.

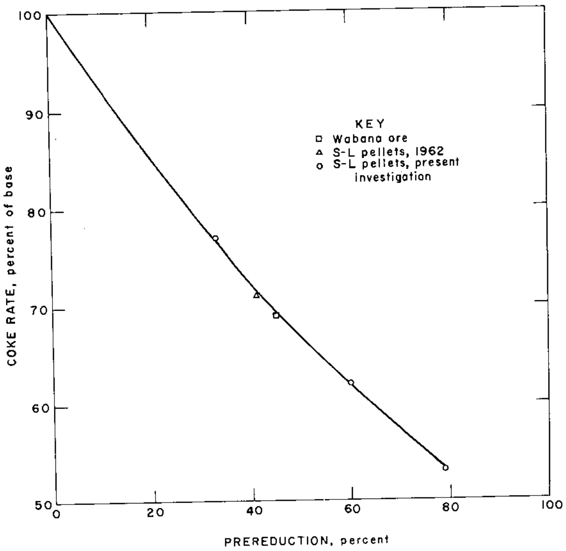

Figure 26.—Effect of prereduction on coke rate decrease expressed as a percent of original coke rate.

The effect of prereduction on productivity is shown in figure 27. Again, all five points may be well represented by a curve which shows a slight increase in the production gain per unit of prereduction with increased prereduction.

Despite differences in operating conditions and burden materials, the percent changes in coke and production rates are a function of the percent prereduction. Figures such as these can probably be used to predict the blast furnace performance with prereduced materials accurately enough to permit an economic evaluation of the potential benefit of prereduction.

The foregoing is applicable to operation with high wind rates and no auxiliary fuel injection. The benefits obtained in the 1962 tests in which natural gas was injected at a low wind rate were less rewarding. Evidently, when the conditions are very favorable for reduction in the blast furnace stack (for example, low wind and enriched bosh gas from hydrocarbon injection) part of the benefit of prereduction is lost.

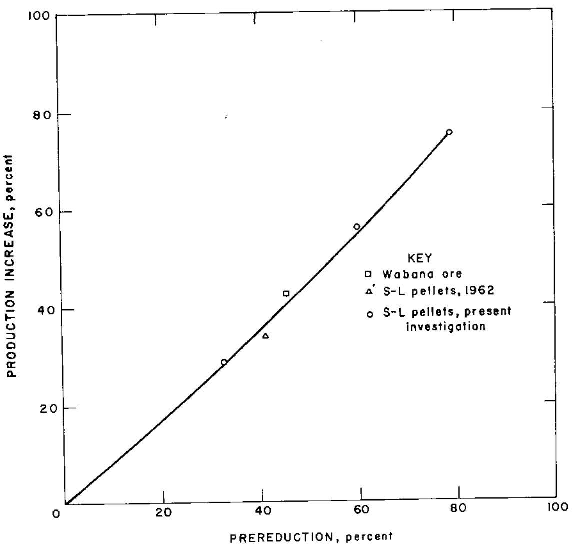

Figure 27.—Effect of prereduction on production increase expressed as a percent of the original production rate.

The coke savings with iron powder injection of 7.1 and 5.3 lb for each percent prereduction were less than with the prereduced burdens charged in the top of the furnance. With iron powder injection, there was a decrease in high-temperature heat, or flame temperature; blast moisture was maintained at a high level. It is possible that reduced powders could be injected at high blast temperature with low blast moisture, in which case the results could compare favorably with those achieved by charging reduced materials in the top of the furnace. The significance of these tests is the technical and mechanical feasibility of injecting reduced powders into the tuyeres to obtain benefits. They also prepare for additional test work with reduced concentrates, flue dust, or fines generated from the manufacture of prereduced burdens.

PART 5.—AN ECONOMIC EVALUATION OF THE PRODUCTION AND BLAST FURNACE SMELTING OF PREREDUCED IRON ORE PELLETS

by

By N. Bernstein, J. L. Reuss,[1] and P. L. Woolf

The evidence is clear that structural or chemical improvement in raw materials will net corollary gains in smelting. What is less clear, perhaps, is whether certain demonstrable gains are worth the added effort. For example fluxed sinter is both technologically and economically viable as a blast furnace raw material. However, consider fluxed pellets as another case in point. Here is a commodity which could effect a 10-percent improvement in coke economy and productivity as compared with acid pellets, but it has failed to achieve commercial status; evidently, the deterrents outweigh the potential 10-percent gain.

With prereduced pellets the potential betterment in smelting data is truly spectacular, 75-percent-higher productivity and 45-percent-lower coke rate as indicated earlier. This is an attractive goal, but is it economical? An appraisal of hot-metal costs from various raw materials concluded that prereduced pellets may have financial advantages over the others. Graff and Bouwer[2] considered natural ore, coarse ore, sinter, regular pellets, fluxed pellets, and prereduced pellets and calculated that not only was total investment lower but the percentage return on the investment was higher for hot metal made from prereduced pellets. The projection was based on all new facilities for mining, preparation, and smelting to produce 1,000,000 net THM per year with an assumed value of $50 per ton. For example, the installation centered around prereduced pellets was figured at $86,030,000, upon which an after-tax profit of 11.5 percent was prognosticated. Corresponding figures based on regular pellets were $99,380,000 and 8.5 percent, respectively. However, if the comparison is based on new pelletizing facilities only, profit as a percent of the additional investment required for prereduced pellets would be 7.9 percent. Hot-metal data included a cost of $28.27 per ton when derived from regular pellets and $26.67 per ton when originating from the reduced variety, both values exclusive of depreciation.

The Bureau of Mines also appraised the economics of prereduction and the following projection includes cost estimates of iron derived from prereduced pellets, regular iron ore pellets, and a mixture of the two. Full commercial-size prereduced-pellet production and smelting facilities were extrapolated, in part, from experimental data, the pellet plant being designed to yield 2 million long tons of product annually. The blast furnaces were existing facilities that would yield approximately 1 million annual net tons of hot metal each from an oxide pellet burden. Estimates and assumptions were considered reasonable for the Mesabi range and lower lake steel centers; however, in practice each situation would differ as to raw materials, transportation, desired production capacities, and individual operational features.

The estimates and computations are based on 1963–64 manufacturers' prices. Although inflation in the national economy has escalated most costs since then, it may nevertheless be assumed that the relative proportion of one operation to another, costwise, has been maintained.

GENERAL PREREDUCED-PELLET-PLANT DESCRIPTION

Figure 28 is a simplified flow diagram of the proposed prereduced pellet plant. Iron ore concentrate is balled into green pellets and charged to a rotary kiln along with lignite or barley-size anthracite and limestone or dolomite. The size of the anthracite is selected so that it can be separated from the product pellets by simple screening. Lignite char also may be screened out because it decrepitates drastically upon partial combustion.

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[1] Metallurgist, Salt Lake City Metallurgy Research Center, Bureau of Mines, Salt Lake City, Utah; formerly with Twin Cities Metallurgy Research Center, Bureau of Mines, Minneapolis, Minn.

[2] Graff, Howard M., and Sidney C. Bouwer. Economics of Raw Materials Preparation. Min. Eng., v. 17, No. 8, August 1965, pp. 73–77.

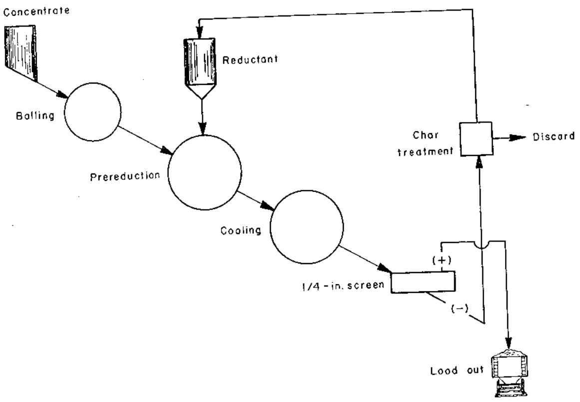

Figure 28.—Simplified flow diagram of proposed prereduced-pellet plant.

Assumptions were made on the basis of Bureau experimental data[3] and from a direct reduction operation at The Steel Co. of Canada (Stelco).[4] Rotary kilns were selected because the Bureau's experience indicated that this vessel was capable of simultaneously indurating and metallizing iron ore pellets, yielding a product suitable for blast furnace consumption. The selection is not intended to exclude the possibility of using other reactors.

Figure 29 depicts the plant layout. The flowsheet begins with a tripper belt system (1) to the concentrate bins (2). Table feeders deposit concentrate on conveyors (3) passing under reel mixers (4). Prior to the mixers, belt feeders deposit bentonite on the conveyors (3). A conveyor (5) feeds the pellet mix via weightometer (not shown) into balling drums (6) equipped with trommels, and recycle belts (7, 8, 9) which return undersized green spheres to the drums.

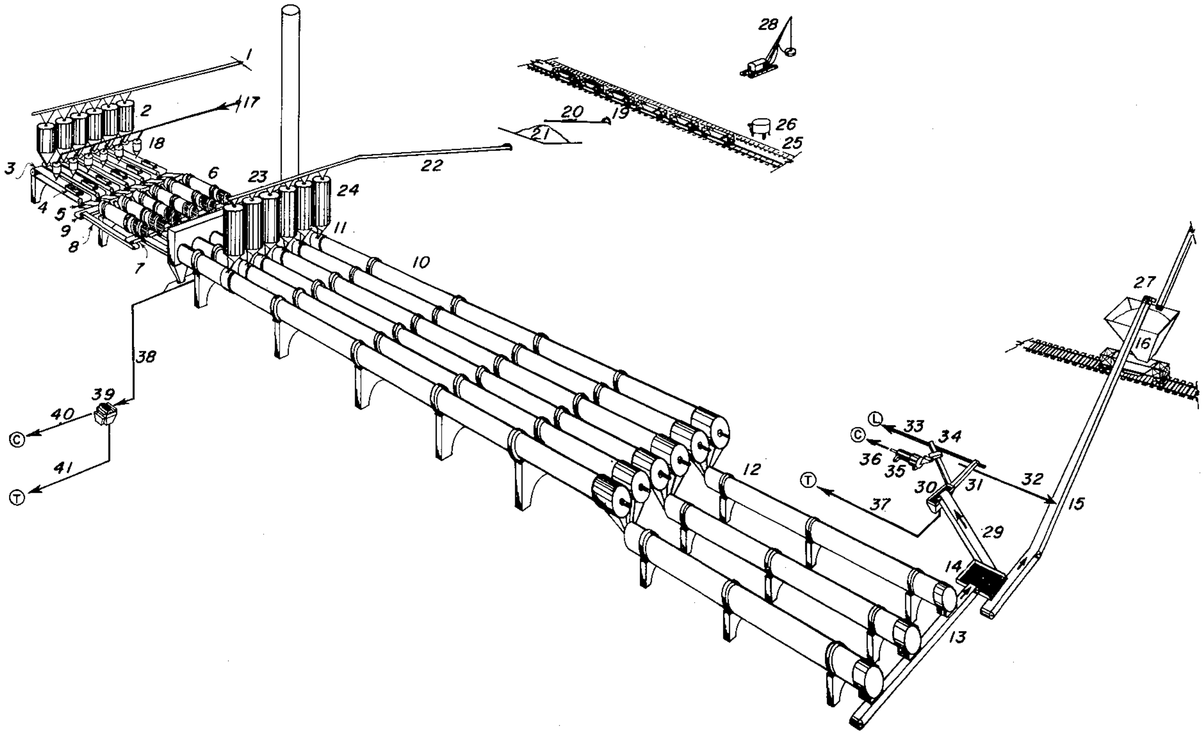

Figure 29.—Prereduced-pellet plant.

Properly sized green pellets are charged via feeders to rotary kilns (10). Lignite feeders (11) feed lignite to the kilns. The kiln charge is heated to approximately 2,150° F and empties into rotary coolers (12). Cooler discharge is conveyed (13) to a ¼-inch screen (14) where the oversize is lifted (15) and diverted into a surge hopper (16) or onto a stacking tripper (not shown) which passes over the storage area.

A pneumatic conveyor (17) unloads the benton-

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[3] Fine, M. M., P. L. Woolf, and N. Bernstein. Experimental Production and Smelting of Prereduced Iron Ore Pellets. BuMines Rept. of Inv. 6523, 1964, 28 pp.

[4] Sibakin, J. G. Development of the SL Direct Reduction Process. Blast Furnace and Steel Plant, v. 50, No. 10, October 1962, pp. 977–989.

ite from railroad cars to feed bins (18). Lignite drops from railroad cars into an unloading pit (19) and is lifted by conveyor (20) to the surge pile (21). Conveyors (22) move the lignite from the surge bin to a tripper system (23) and into lignite bins (24). Limestone likewise is placed in the unloading pit (19) and lifted by conveyor (20), but the stone then goes directly to conveyor (22), and is diverted via another conveyor (not shown) to a belt tripper (not shown) and into limestone bins (not shown). Oil (25) pumped from railroad cars to a storage tank (26) is sprayed (27) onto the pellets as they are loaded into railroad cars or dropped onto the storage pile. Stored pellets are later loaded into cars with a magnet-equipped crane (28).

Minus ¼-inch material passing the cooler screen (14) is conveyed (29) to a double-deck screen (30). The 6-mesh by ¼-inch size is passed over a magnetic pulley (31,) and the magnetics are conveyed (32) to a belt (15) as product. Nonmagnetics are directed onto another belt (33) and conveyed to the lignite surge pile (21). Material 6- by 28-mesh is magnetically separated (39), and the nonmagnetics are returned to the lignite surge pile via a belt (34), while the magnetics are ground (35) and pumped (36) to the concentrating plant. Minus 28-mesh fines are pumped (37) to the tailing pond. Dust from the kiln stacks is pumped (38) to a wet magnetic separator (39); and the magnetics are pumped (40) to the concentrate thickener, while the nonmagnetics are pumped (41) to the tailings pond.

KILN AND COOLER SELECTION

The dimensions of a rotary kiln may be extrapolated with the equation $C = K D^2 L$, where C is capacity in long tons per day, D is the kiln diameter in feet, L is length in feet, and K is a production factor determined by experience. This equation is similar to that quoted by Perry.[5] Many variables might influence the production factor, such as the use of shell burners, dimensions of damming rings, rotation speed, slope, raw materials, and burner and draft considerations, as well as details that might not be apparent until operation was commenced.

The Bureau kiln, 34 inches by 36 feet, was successfully operated with lignite at a rate of 4.7 long tons of product pellets per day. Solving the equation for K gives 0.016. Stelco's kiln, 7½ by 115 feet, was operated at 100 long tons of sponge product per day with anthracite, which gives a production factor of 0.015. A commercial-size kiln operating on lignite would have a greater production factor than the small Bureau vessel. Tests at Minneapolis demonstrated that anthracite is not as reactive as lignite;[6] therefore equivalent metallization would require a longer residence time with the hard coal. The selected production factors were slightly in excess of 0.017 for the lignite and 0.015 for the anthracite.

Production rates in other processes utilizing rotary kilns determine the plausibility of these factors. The grate kiln at Humboldt, rated at 1,000 tons per day of conventional iron oxide pellets, utilizes a 10- by 120-foot rotary kiln for the final heating.[7] The dimensions and capacity of this last stage kiln, receiving pellets at 2,000° F, are such that the production factor is 0.08, approximately five times that of the prereduction kilns. The 1.2-million-annual-ton unit at Empire uses a 17- by 115-foot rotary kiln for a production factor of 0.11.[8] The Freeman sponge iron process anticipated a 350-ton-per-day production is a 10- by 100-foot rotary kiln using coke fines,[9] giving a production factor of 0.035. Production rates from oxide practice should be higher owing to the absence of solid reductant, but this may be partially offset in that an iron-prereduction kiln would be operated approximately 400° F lower than oxide indurating furnaces and possibly with a deeper bed.

The vessels selected for use with lignite were six 12- by 400-foot rotary kilns; those used with anthracite would be 50 feet longer. Burners were designed to produce approximately 2.5 million Btu per long ton of product. The balance of the heat would be supplied by the solid fuel reductant.

The coolers were extrapolated with the equation $C = K D^2 L$ from the 5- by 60-foot Stelco cooler. This indicated that three 12- by 200-foot coolers would suffice for the prereduced-pellet plant.

RAW MATERIALS AND UTILITIES

Green pellets were made from magnetic taconite concentrate containing 65.3 percent iron. As calculated from table 35, this concentrate would make conventional oxide pellets containing 62.8 percent iron or metallized pellets with 82.0 per-

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[5] McCormick, P. Y., R. L. Lucas, and D. F. Wells. Gas Solid Systems. Ch. in Perry's Chemical Engineers Handbook. McGraw-Hill Book Co., Inc., New York, 4th ed., 1963, p. 23.

[6] Work cited in footnote 3.

[7] Berkhaln, R. W., and D. M. Urich. Grate Kiln Pelletizing at Humboldt. Proc. 23d Annual Univ. of Minn. Min. Symp. and Ann. Meeting, Minn. Section, AIME, Duluth, Minn., Jan. 15–17, 1962, Duluth, Minn., 1962, pp. 25–35.

[8] Skillings, David N. Cliffs New Empire Mine, Marquette Iron Range. Skillings' Mining Review, v. 53, No. 13, Mar. 28, 1964, p. 12.

[9] Freeman, Horace. Reasons for the Co-Current Kiln in Direct Reduction. Ch. in Iron Ore Reduction, ed. by R. R. Rogers. The Macmillan Co., New York, 1962, pp. 154–162.

cent iron. Table 35 outlines a calculated material balance of the concentrate and pellets. It was assumed that the prereduced pellets are 80 percent metallized with the remaining iron as FeO. Iron in conventional pellets is present as hematite. Whereas conventional pellets made from magnetite gain weight owing to oxidation, metallized spheres undergo a considerable weight loss.

Table 35.—Contents of concentrate and resulting pellets, lb/100 lb of concentrate, dry basis

	Concentrate	Metallized pellets	Oxide pellets
Fe	65.3	65.3	65.3
Oxygen[1]	24.9	3.7	28.1
Silica	8.4	8.4	8.4
Bentonite	—	.8	.8
Gangue[2]	1.4	1.4	1.4
Total	100.0	79.6	104.0

[1] Iron oxide.
[2] Other than silica and bentonite.

Table 36 includes the ratios of raw materials and power required per gross ton of reduced product. As indicated in table 35, a weight loss of 20 percent is sustained in reducing dried pellets to the metallized product. This increases raw material input over what is typical of a conventional pellet flowsheet. Bentonite, for example, at 17 lb per ton of dry concentrate, is 21 lb per ton of product. Obviously, the ratios of raw materials and power might differ in practice, but they are considered reasonable. The power consumption, though less than that given by English,[10] is considered valid because the metallizing plant would use less power for fans than a shaft furnace or grate system. Bennett,[11] for example, revealed a power consumption of 16.2 kwhr per ton of nodules for the Extaca nodulizing kiln.

Table 36.—Raw material and power costs required for making prereduced pellets[1]

Item	Amount required per long ton of product	Estimated cost per unit	Cost per long ton of pellets
Concentrate	1.250 long tons	$ 8.00	$10.00
Lignite	.7 short ton	4.50	3.15
Anthracite	.3 short ton	14.60	4.38
Bentonite	21 pounds	.0134	.28
Limestone	75 pounds	.002	.15
Power, with lignite	17.0 kwhr	.010	.17
Power, with anthracite	19.0 kwhr	.010	.19
Burner fuel	2.5 million Btu	.50	1.25
Protecting oil	2.0 gallons	.09	.18

[1] Raw material and power cost per ton of product was $15.18 using lignite and $16.43 using anthracite.

The assumed unit costs in table 36 are not necessarily applicable to blast furnace operation owing to the difference in location. For the Minnesota taconite area, English and coworkers[12] give the costs of bentonite, electricity, and burner fuel as $30 per gross ton, $0.01 per kwhr, and $0.50 per million Btu, respectively. If natural gas were available it would probably be less expensive. The cost of concentrate varies considerably, depending on individual situations. Subtracting assumed pellet-processing costs of $2.50, transportation of $3.70, investment return of $.80, and income tax

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[10] English, Alan, and Melvin J. Greaves. Construction and Operating Costs of a Modern Pelletizing Plant. Trans. AIME, v. 226, September 1963, p. 316.
[11] Bennett, R. L., R. E. Hagen, and M. V. Mielke. Nodulizing Iron Ores and Concentrates at Extaca. Min. Eng., v. 6, No. 1, January 1954, p. 36.
[12] Work cited in footnote 10.

Table 37.—Total costs for 2-million-annual-ton prereduced-pellet plant

	With lignite	With anthracite
Cost of erected equipment:		
Raw material handling	$333,200	$333,200
Balling circuit	632,900	632,900
Indurating section	8,321,400	8,821,400
Product handling	535,800	535,800
Total	9,823,300	10,323,300
Installed equipment on site	14,822,700	15,502,100
Fixed costs	21,939,400	22,938,000
Working capital	7,000,000	7,000,000
Total capital investment	28,939,400	29,938,000
Depreciation per ton of product, 20-yr. straight line . . . cents	55	57

of $0.80 from the listed conventional pellet selling price of 25.2 cents per iron unit indicates that $8 per long ton is appropriate. The higher shipping cost of anthracite makes lignite a more desirable economic selection on the Mesabi Range; however, anthracite provides cleaner operation and may be more competitive in other locations.

CAPITAL AND OPERATING COSTS

The cost of the complete pellet plant using lignite is estimated at $21,939,400, or approximately $11 per annual ton. This is considerably higher than the $7.70 to $8.80 for conventional pellet plants reported by English;[13] however, the higher cost is justified because there are more iron units in the metallized product. The cost of the complete plant includes all the equipment installed in adequate buildings, as well as 20,000 feet of railroad trackage. Allowance is made for contractor and engineering fees, contingencies, and interest during construction. The cost of the plant using anthracite is $29,938,000, owing to the use of larger kilns.

Detailed cost estimates are developed in appendix tables A–1 to A–4, which include an approximate material balance and equipment accordingly sized and priced. In determining the material balance, it was assumed that the plant would operate 330 days per year. Table 37 summarizes (1) the cost of erected equipment, including installation labor for both lignite and anthracite plants, and (2) the capital costs in addition to the equipment necessary for a complete plant. Working capital was taken as approximately 2½ times the total monthly operating cost.

Owing to the recognized lack of precision in estimating without a knowledge of terrain and other local conditions, most of the items were estimated as proportions of the capital equipment.[14] Buildings were considered as being constructed of simple steel siding. They would be insulated and would have side windows only in the change-house, part of the maintenance shop, the office, and the laboratory.

Table 38 shows the operating costs. As indicated in table A–4 (appendix), it would require 14 men per shift to operate the plant. Accordingly, a direct labor force of 65 men is in order, which with supervision and payroll overhead would cost approximately $600,000 per year, or 30 cents per pellet ton. Annual maintenance was estimated at 8 percent of the installed equipment cost, and operating supplies at 15 percent of the maintenance cost. Indirect costs were assumed to be 50 percent of the combined labor and maintenance costs. Local taxes and insurance were taken as 2 percent of the total plant cost. The total operating costs, per long ton of product pellets, would be $17.38 for the lignite plant and $18.71 for the anthracite plant.

Operating costs for conventional pellets might be estimated by subtracting $3.70 transportation and $1.60 for income tax and investment return, from the $15.83 Lake Erie price of a 62.8-percent-iron pellet. The remaining $10.53 was $6.85 less than the calculated operating cost of making prereduced pellets from lignite. This indicates that the additional $6 necessary to produce prereduced pellets assumed by Graff[15] is in fairly good agreement.

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[13] Work cited in footnote 10.

[14] Peters, M. S. Plant Design and Economics for Chemical Engineers. McGraw-Hill Book Co., Inc., New York, 1958, pp. 91–104.

[15] Work cited in footnote 2.

Table 38.—Total costs per long ton of prereduced pellets

	With lignite	With anthracite
Operating costs:		
Raw materials and power (table 36)	$15.18	$16.43
Labor	.30	.30
Plant maintenance	.59	.62
Operating supplies	.09	.10
Indirect costs	.45	.46
Taxes and insurance	.22	.23
Depreciation (table 37)	.55	.57
Total	17.38	18.71
Net return on investment	1.45	1.50
Federal income tax	1.40	1.45
Transportation	4.50	4.50
Grand total	24.73	26.16
Price per Fe unit[1] . . . cents	30.2	31.9

[1] Delivered price in lower lake ports.

PRICE OF PREREDUCED PELLETS

Table 38 provides for a 10-percent net return on the plant investment (after Federal income tax) and transportation. Transportation rates for metallized pellets have not been established, but with a Mesabi-to-lower lake ports rate of approximately $3.70 for conventional oxide pellets, $4.50 per ton for metallized pellets is generous. Prereduced pellets should be easier to load and unload than regular pellets, owing to the use of magnet-equipped cranes.

On the assumptions made, the cost of prereduced pellets at a lower lake blast furnace would be $24.73 per long ton with lignite as a reductant. The use of anthracite would increase the figure by $1.43. If a metallized pellet of 82 percent iron is produced, the price per long ton iron unit would be slightly over 30 cents.

BLAST FURNACE SMELTING

Since commercial cost data are not yet available for blast furnace smelting of prereduced pellets, the economics must be judged by extrapolations of experimental data. In the following examples a standard furnace is presumed to operate on the same blast volume, temperature, and moisture but with different burdens. The base example utilizes a burden of conventional oxide pellets only, the second is comprised of 40 parts by weight prereduced pellets and 60 parts conventional oxide pellets, and the third consists of prereduced pellets only. Raw material analyses are presented in table 39.

Table 39.—Blast furnace raw material analyses, percent [1]

	Fe	SiO_2	Al_2O_3	CaO	MgO
Oxide pellets	62.8	8.5	0.50	0.60	0.40
Prereduced pellets	82.0	11.1	.65	.78	.52
Limestone	.7	3.5	.60	52.3	.60
Dolomite	.7	.9	—	29.7	21.6
Coke ash	8.0	51.0	30.0	3.2	1.3

[1] Chemical analysis of coke in percent: 0.8 volatile matter; 8.5 ash; 90.0 ultimate carbon; 0.6 sulfur, and 0.3 hydrogen.

Table 40.—Estimated blast furnace operating data

	Example 1	Example 2	Example 3
Burden, percent:			
Oxide pellets	100	60	—
Prereduced pellets	—	40	100
Production rate . net tons/day	2,755	3,725	4,850
Coke rate (dry) . . . . lb/THM	1,200	865	550
Stone rate . . . . lb/THM	540	510	470
Slag volume . . . . lb/THM	655	620	575
Top gas temperature . . . . . ° F	350	300	250
Wind rate . . . . scfm	115,000	115,000	115,000
Blast temperature . . . . ° F	1,500	1,500	1,500
Blast moisture . . . . . . gr/cu ft	12	12	12
Prereduction, pct oxygen removal	0	40	87

The principal estimated operating data for the three conditions are shown in table 40. Data for the first example are considered typical of blast furnace operations with burdens comprised entirely of conventional pellets. The examples utilizing prereduced burdens are predicted on the basis of recent experimental blast furnace tests with such materials.[16] By coincidence, the replacement of 40 percent of the oxide pellets by the prereduced variety yields a burden with 40-percent prereduction. The prereduction with all prereduced pellets of the analysis given in table 39 is 87 percent.

There are several combinations of available hot metal capacity and projected requirements that affect the economic evaluation, but only the following two will be considered:

1. Hot metal cost has been calculated assuming existing furnaces have sufficient capacity to meet increased production demands.
2. Hot metal cost has been calculated for the case where existing facilities have insufficient capacity with conventional pellets but could produce adequate tonnages by smelting prereduced pellets.

Table 41 lists the costs of blast furnace raw materials and processing, as well as fixed costs and top-gas credit. Note that pellet costs are given in short tons. Fixed costs per ton of hot metal consist of those that do not vary with production rates, such as metal handling, runners, refractory and fund allowance, and slag disposal. Labor, maintenance, utilities, and overhead are examples of variable costs, since they decrease per ton of metal as furnace output increases. Blast compression is listed separately. Normally it is virtually fixed, but with prereduced burdens it becomes variable because the blast volume remains constant while production is increased. Excess top gas is that remaining after being used to heat the blast air, assuming 80-percent stove efficiency. Hot metal costs for the base example using only oxide pellets and the two cases using partial or complete prereduced burdens are shown in table 42.

Table 41.—Unit prices used in hot metal cost analyses

Raw materials, per net short ton:	
Coke (dry)	$15.00
Limestone and dolomite	3.60
Pellets	14.13
Prereduced pellets	22.08
Credit for top gas excess, per 10^6 Btu	.25
Fixed cost, per net THM	2.75
Cost variable with productivity (base case)	2.00
Blast compression, per 100,000 cu ft	2.00

BLAST FURNACES HAVING AMPLE CAPACITY

Despite the increase in pellet cost per iron unit, the total cost of raw materials decreases with prereduced pellets owing to the coke savings. Increased productivity also accounts for savings in operating costs. Decreases in hot metal costs are calculated at $1.16 and $1.94 per ton of hot metal for the second and third prereduced pellet burden examples, respectively. The savings for each percent of reduction approximate 2.9 cents with the 40-percent prereduced-pellet burden but only 2.2 cents with 87-percent prereduction in the burden. With a multifurnace plant it would be more economical to share a limited supply of prereduced pellets than to use them entirely on a single furnace. Any savings by using prereduced pellets may be considered an advantage, since a return on the

Table 42.—Comparative hot metal costs per THM, showing effect of prereduced pellets

	Example 1	Example 2	Example 3
Raw materials:			
Conventional oxide pellets [1]	$21.58	$11.51	—
Prereduced pellets [2]	—	11.99	$25.70
Coke	9.00	6.49	4.13
Stone	.97	.92	.85
Total	31.55	30.91	30.68
Fixed cost per THM	2.75	2.75	2.75
Costs variable with productivity	2.00	1.48	1.14
Blast compression	1.20	.89	.68
Total	5.95	5.12	4.57
Credit for excess top gas	.96	.65	.65
Grand total	36.54	35.38	34.60
Savings per THM over example 1	—	1.16	1.94

[1] Assumed 2-percent loss to dust and slag.
[2] Assumed 1.5-percent loss to dust and slag.

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[16] Woolf, P. L. Blast Furnace Operation With Prereduced Burdens. J. Metals, v. 18, No. 2, February 1966, pp. 243–247.

pellet plant investment has already been included in the pellet cost. Capitalization was excluded from the foregoing analysis because the plant is assumed to have ample capacity and the blast furnace is assumed to be fully depreciated. Any capitalization included in the iron cost would elevate the economic advantage of prereduced pellets, since depreciation per ton decreases as production rises.

BLAST FURNACES HAVING INSUFFICIENT CAPACITY

With a projected need for production which exceeds blast furnace capacity, a choice between a new furnace and the use of prereduced pellets may be considered. For example, if a plant consisting of three standard blast furnaces produced 8,000 to 9,000 THM daily from conventional oxide pellets but 11,000 tons were required, there would be two alternatives: (1) an additional blast furnace might be built, or (2) the burden to the three existing furnaces could be modified to use 40-percent prereduced pellets. A new blast furnace would not only require large capitalization, but might necessitate the construction of an additional coke battery to supply the increased coke requirement (1,650 tons per day). Coke may be purchased, but this would probably cost more than $15 per ton. The use of prereduced pellets to meet the increased production demand would, conversely, require less coke. For simplicity, it has been assumed that additional coke is obtainable at $15 per ton.

Taking the cost of a new blast furnace at $30 million and using the straight line method over 20 years, depreciation would be $1.56 per THM, assuming 350 operating days per year. The cost of the increased metal production in a new furnace with conventional pellets is $38.10 per THM, $36.54 from table 42 and $1.56 depreciation.

The effects of this depreciation on total plant hot metal costs may be seen in table 43, under operations A and B. The four furnaces of operation B have a daily output of 11,020 tons at a cost of $406,968. Depreciation on the new furnace is diluted so that the cost of all hot metal is increased only from $36.54 to $36.93.

If the production increase were obtained by operation C using the prereduced burden, a daily production of 11,175 tons could be obtained at a cost of $395,371, or $35.38 per ton. This represents a savings per ton of $1.55 over the new furnace choice, besides freeing the capital for other investments.

It is informative to compare the incremental costs of the additional hot metal. Table 44 shows a production gain of 2,190 tons in the three furnaces by using the prereduced pellets (operation C). This is achieved at a cost of $93,368 over operation A, or $32.09 per ton. On the other hand, a similar increase in hot metal, 2,755 tons, may be obtained with a new furnace at an extra daily expenditure of $104,965, or $38.10 per ton. The incremental savings by using the prereduced pellets is $6.01 per ton.

Table 43.—Daily costs and production of blast furnaces operating with conventional pellets and on a partial prereduced burden

Operation	No. of furnaces	Burden	Daily production THM	Daily cost	Cost per ton
A	3	All oxide	8,265	$302,003	$36.54
B	4	. . do . .	11,020	406,968	36.93
C	3	40-pct prereduced pellets....	11,175	395,371	35.38

Table 44.—Incremental costs of increased blast furnace production

Operation	Increased daily tonnage over operation A	Increased daily cost over operation A	Incremental cost per THM over operation A
B	2,755	$104,965	$38.10
C	2,910	93,368	32.09

CONCLUSION

The major capital expenditure in a plant to make prereduced pellets is for kilns and coolers. Other items beyond those required for conventional pellets, such as reductant bins and conveyors, add little to the depreciation charge per ton of product. Prereduced pellet plants cost more per product ton than those for oxide spheres, largely because there is more iron in the metallized pellets. The estimated plant cost is $11 per annual ton.

The major operating cost in making prereduced pellets is the reductant. This is a highly variable expenditure, depending on location, since transportation is a substantial portion of the delivered price.

Based on assumptions in this paper, prereduced pellets can be made on the Mesabi Range with lignite and smelted in lower lake blast furnaces at overall savings of almost $2 per THM. This is gaged against conventional iron ore pellets smelted in fully depreciated facilities.

The use of a partial prereduced burden in lieu of building (and depreciating) an additional blast furnace for increased hot metal could save $6 per ton on the added production.

PART 6. — PREREDUCED IRON ORE PELLETS — STATE OF THE ART

By N. B. Melcher and M. M. Fine

PROCESSES FOR PREREDUCED PELLETS

Rolling a fine iron ore or concentrate into a ball and then reducing it partially to metal is a routine operation, although it can be affected by a number of variables including (1) gaseous or solid reductants, (2) choice of solid reductant, (3) internal or external solid reductant, (4) moderate or large excess of internal solid reductant, (5) limestone, or none, within the pellet, (6) green balls or fired oxide pellets as the iron raw material, (7) rotary kiln, shaft furnace, traveling grate or specialized reaction vessels, and (8) degree of reduction.

A listing of all the individuals and organizations who have contributed to the technology of reduced agglomerates would be a very lengthy one. Research along these lines continues throughout the world and it is not our intent to summarize all the investigations. This chapter describes a limited number of current and recent developments, particularly those conducted on a pilot plant or commercial basis. However, the authors wish to acknowledge the work of Barrett and Wood[1] who were among the early pioneers involved with reducing pelletized iron ore. Using Minnesota taconite concentrates and fairly pure iron oxides, they succeeded in making almost completely reduced "glomerules" by gaseous reduction with hydrogen and water gas in an externally heated shaft furnace.

In March 1965, Midland-Ross Corp. announced development of an iron ore pelletizing process purported to increase the production of existing blast furnaces using conventional pellets.[2] The company entered into an agreement with The Hanna Mining Co. and National Steel Corp. to build a pilot plant near Nashwauk, Minn., for testing on a near-commercial scale. The process was based on U.S. Patent 2,793,109, filed April 1954 and granted 1957.[3] According to the patent, a method is provided for reducing oxidized iron in an agglomerated mass, calculated as Fe_2O_3 and C, in a weight ratio of from about 8:1 to about 1:1. The agglomerates are heated in an atmosphere substantially devoid of free oxygen at a rate so that the mass is at a temperature between 1,600° F and 2,200° F from 1 to 20 minutes. This heating rate is accomplished by hot body radiation. The distinctive features of this method are rapid heating and reaction and the use of radiant energy. The patent describes a direct-fired furnace with a refractory radiating surface that is generally semicircular in cross section.

The Hanna-Surface Combustion pilot plant was equipped with facilities for autogenous dry grinding, dry magnetic cobbing followed by wet grinding, and several stages of wet magnetic separation. The concentrate was then pelletized and reduction-indurated. The pellets are reduced by incorporating coal in the pellet mix. The flowsheet was essentially a four-step operation starting with low-temperature (300°) F) drying. The dry pellets were transferred to a rotary hearth furnace, approximately one layer deep, and heated by radiation to 2,300° F on the revolving annular turntable. Cooling took place in a vertical shaft. A description of this "Heat Fast" process was presented at the Evian symposium.[4]

Another variant is the ACAR (Allis-Chalmers Agglomeration-Reduction)[5] process, which is distinguished by solid fuel within the pellet, gaseous reductant external thereto, and a rather unique and complex reactor. The ACAR is an offshoot of the company's well-known grate-kiln system for agglomerating taconite concentrate, and some features have been retained or modified. Fine concentrate is compounded with the usual dosage of bentonite plus 3 to 9 percent of coke and formed into green balls. These are deposited on a traveling grate for drying and partial hardening, after which

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[1] Barrett, E. P., and C. E. Wood. Production of Sponge Iron: Gaseous Reduction of Iron Oxide Glomerules in a Shaft Furnace. BuMines Rept. of Inv. 4305, 1948, 19 pp.

[2] Skillings Mining Review, New Pelletizing Process Increases Output. V. 54, No. 10, Mar. 6, 1965, p. 10.

[3] Huebler, J., and D. Beggs. Induration Process for Powdered Iron Oxide Containing Material. U.S. Patent 2,793,109, May 21, 1957.

[4] Smith, L. W., and D. N. Vedensky. Heat Fast-Process for Prereduced Pellets. Proc. Internat. Cong. on the Production and Utilization of Reduced Ores, Evian, France, May 1967, pp. 57–65.

[5] Kirkland, T. G., C. L. Sollenberger, J. L. Platner, and W. W. Edens. Allis-Chalmers Agglomeration-Reduction Process. Ch. in Iron Ore Reduction, ed. by R. R. Rogers. The Macmillan Co., New York, 1962, pp. 255–270.

they enter the reduction kiln. The kiln consists of two concentric clyinders, the outer of which is lined with insulating refractory brick; the inner is of stainless steel only. The pellets enter the inner cylinder through a sealed feed pipe. Also introduced at this point are propane (other gaseous or liquid fuels may be used) and oxygen, air, and steam or combinations of these. The gases and materials to be reduced move through the metal cylinder, the temperature of which (1,920° F) is high enough to reform the original gas mixture into an atmosphere consisting primarily of hydrogen and carbon monoxide. Reduced pellets, after traversing the kiln, fall through a sealed pipe into a quench tank. Unused gases flow from the inner kiln back into the annulus where secondary air enters to complete the combustion that supplies heat for the process. Operation of the kiln on minus ¾ plus ½-inch taconite concentrate pellets containing 9 percent coke yielded about 95-percent reduction with a retention time of 2½ hours. The reduced pellets, analyzing 84.1 percent total Fe, 80.8 percent metallic Fe, and 10.5 percent SiO_2, were processed successfully to cast iron in a hot-blast cupola.

Research is continuing on direct reduction at Allis-Chalmers to develop a process which will (1) enable the economic conversion of existing grate-kiln pellets plants and (2) be a means of producing prereduced pellets in new plants. Development work has proceeded along several paths. The ACAR process is still under study; development work is concerned primarily with improved seals, mechanical construction, and variations in process conditions with different ores. This process, according to the company, appears to have merit where small tonnages of highly reduced products are required.

Exploratory work with other equipment has resulted in the design of a new Controlled Atmosphere Furnace. This unit will allow drying and preheating of stationary beds of material under conditions of controlled gas flow, temperature, and composition. The treated material then can be transferred into a rotating device without losing control of the gas composition or temperature, and treatment can be continued in a tumbling bed to which solid and gaseous fuels may be introduced. The product can be cooled in either a stationary or a tumbling bed, again under controlled or inert atmosphere. The company is planning a pilot plant for a continuous partial-reduction system.

The Arthur G. McKee Co. has engaged in development effort on partially reduced pellets since 1963. A laboratory furnace at Cleveland is designed to produce either oxidized or reduced pellets. For reducing conditions, temperature control is obtained either with excess or variable gas flow if predetermined proportions are required for partial reduction. Utilizing gas composition and flow controls, operation of a commercial traveling grate for preparation of partially reduced pellets is closely simulated. The company states that any desired range of pellet reduction can be obtained by control of atmosphere, temperature, retention time, and type and amount of internal reductant. Partially reduced pellets of about 80 percent total Fe and 60 percent metallized Fe have been obtained at reasonable rates. A conditioning method has been developed to render the metallized pellets suitable for outdoor storage. Further development is underway including testing preparatory to design of a traveling grate specifically suited to reduction systems and production of partially reduced pellets.

The agglomerates described previously are composite pellets manufactured from a mix of concentrate, binder, coal, and, if desired, limestone. The coal is virtually consumed during agglomeration so that the end product contains, at most, only a few percent carbon. There is another class in which the carbon content is sufficiently high to comprise a new approach to the continuous carbonization of coal. In brief, a coking (or caking) coal is pelletized with a high percentage of noncoking coal, iron ore, flux, or combinations thereof and coked rapidly.

The two best known carbonizing processes, at least in the United States, are the Orcarb[6] process and the Dwight-Lloyd McWane (D–LM)[7] process. In the Orcarb scheme, fine ore and high-volatile coking coal are pelletized while being heated to low-temperature carbonization of the coal. Ore and flux are fed into a refractory-lined, direct-fired rotary kiln for drying and preheating at 1,100° to 1,200° F. As the material discharges, it is blended rapidly with coal and charged into the Orcarb retort, an unlined, externally heated rotating steel cylinder maintained at 850° to 905° F. In this vessel the coal softens, becomes plastic, picks up the ore, and is rolled into pellets. The irregularly shaped agglomerates then pass to a third kiln, of conventional design, heated between 1,800° and 2,000° F, where prereduction to about 80 percent metallic iron takes place.

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[6] Lesher, C. E. The Orcarb-Electric Furnace Process. Ch. in Iron Ore Reduction, ed. by R. R. Rogers. The Macmillan Co., New York, 1962, pp. 271–278.

[7] Ban, T. E., and B. W. Worthington. Technology of Dwight-Lloyd McWane Ironmaking. J. Metals, v. 12, No. 12, December 1960, pp. 937–939.

The process was developed by Lesher and Associates at Verona, Pa., and operated for a time in the pilot plant of Swindell-Dressler Co., Indianola, Pa. Hot iron was first tapped in December 1958, and operation of the electric furnace was concluded in 1960 after 35 runs. A total of 146 tons of ore and concentrates from several locations was smelted to pig iron.

The D–LM process uses a different method to attain a carbonized product. Coal, ore, and limestone are proportioned, ground to minus 20-mesh, and balled into ½- to ¾-inch spheres containing about 15 percent moisture. The green balls are fed to a down draft sintering machine for drying and carbonization of the coal to coke, which takes place in less than 12 minutes. An analysis of product pellets showed 49.8 percent total iron, 43.5 percent FeO, 13.8 percent metallic iron, 10.7 percent SiO_2 and 9.1 fixed carbon. These also have been smelted satisfactorily in an electric furnace. A plant incorporating these features, with a 600-ton-per-day capacity is being built for McWane Cast Iron Pipe Co., Mobile, Ala.

Another major grouping of prereduced pellets comprises those in which the reductant is completely external to the iron raw material, which may be either a green ball or fully indurated oxide agglomerate.

The Freeman process,[8] which makes pellets of this type, placed particular emphasis on hardware; the central feature of the process was a cocurrent rotary kiln. The flow starts with the introduction of indurated oxide pellets (manufactured from high-grade natural lump or concentrates) into the combustion zone accompanied by coke or noncoking coal and limestone. Both gases and solids move toward the cooler stack end of the kiln, reduction taking place in transit. In less than 2 hours, 97.5 percent of the oxygen is removed from pellets with an average ¾-inch diameter. Solids are removed with a metal scoop which intrudes through the stack into the bed. The material moves into a cooler and over a trommel screen. Data given from a test period show that ½- to 1-inch pellets at 68 percent Fe were reduced to a product containing 91.5 percent metallic Fe and 3.5 percent ferrous Fe. Particular claims made for the cocurrent system are the absence of sticking to kiln walls and the use of a completely oxidizing flame for heating.

The Freeman Corp. has discontinued development work to perfect a process for pelleted sponge iron, although it is claimed the cocurrent kiln can reduce any form of pellet, including suitably prepared green pellets. The company is now directing its efforts toward a less costly method than the blast furnace for production of hot metal through the use of fine unagglomerated concentrates.

In some respects, the Stelco-Lurgi (SL)[9] process developed by The Steel Co. of Canada is similar to the Freeman process. The one significant departure is the use of a Lurgi rotary kiln as the reduction vessel. The Lurgi is countercurrent but differs from conventional kilns in that it is equipped with floating seals at each end and air, gas, or both may be introduced along its length as well as axially. Fully fired high-grade iron ore pellets, dolomite, and anthracite, which is the reductant, are charged to the kiln. Natural gas heats the vessel to 2,050° to 2,200° F. The pilot kiln is 7.5 feet in internal diameter by 115 feet long; the charging rate ranged from 80 to 140 net tons of ore pellets per day. The quality of the product can be controlled, and as high as 98-percent reduction has been attained. Operating on regular pellets from the Hilton Mine (Canada), the well-reduced sponge contained 91.8 percent total Fe and 85.0 percent metallic Fe. Figure 30 shows the Stelco reduction kiln and cooler.

It has been reported that green balls and lump ores may also be fed into the kiln.[10] A wide range of coals has been used, from lignite to anthracites. The ash-softening temperature of the particular carbonaceous fuel involved controls the operating temperature of the process. This temperature should be above 2,300° F, but coals with a value of approximately 2,100° F have been used successfully. The sulfur content of the fuel should be low, but sulfurs of 1.5 percent can be tolerated without pickup to the sponge iron.

In 1965, Stelco and Lurgi were joined by Republic Steel Co. and National Lead Co. to market a direct iron-ore-reduction process to be known as the SL/RN process which combines the features of the former SL and RN processes. Lurgi acts as the worldwide sales agent for SL/RN; both Pickands-Mather and Hains Engineering Co., Canada, represent the process for Lurgi. Several commercial installations are under construction at present. There is one in New Zealand (titaniferous iron sands, green ball technique)[11] and another in South

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[8] Freeman, Horace. Reasons for the Co-Current Kiln in Direct Reduction Ch. in Iron Ore Reduction, ed. by R. R. Rogers. The Macmillan Co., New York, 1962, pp. 154–162.

[9] Sibakin, J. G. Development of the SL Direct Reduction Process. Blast Furnace and Steel Plant, v. 50, No. 10, October 1962, pp. 977–989.

[10] Fraser, M. J., and C. R. Grigg. The SL/RN Process—Its Place in the Iron and Steel Industry. Eng. J., v. 48, No. 6, June 1965, pp. 29–34.

[11] Simmons, John J. Direct Reduction-Electric Furnace Plant To Process Iron Sands in New Zealand. Skillings' Mining Review, v. 56, No. 31, Aug. 5, 1967, pp. 1, 8–9.

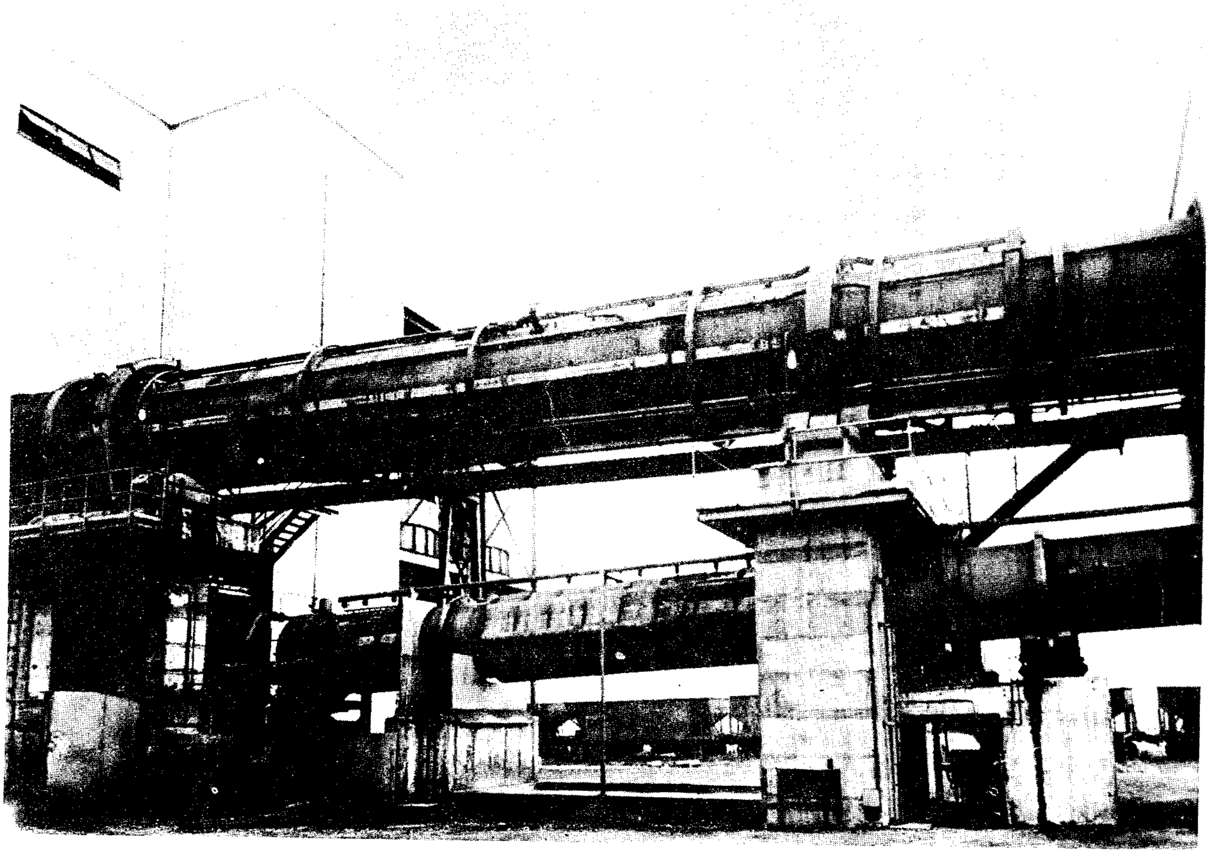

Figure 30.—Stelco reduction kiln and cooler.

Korea (high-grade lump ore); others are being considered.

The third system in this category is that of the Bureau of Mines, which pioneered the simultaneous reduction-induration concept.[12] The procedures outlined in a fairly simple flowsheet (fig. 13), have been tested on a small pilot scale. The absence of unconventional equipment and the use of green pellets are distinctive features of the process. In brief, the ¼- to ⅝-inch green balls, composed of moist concentrate plus bentonite, join a stream of reductant (lignite or anthracite) and a trickle of limestone; all of these are fed into a countercurrent rotary kiln at a maximum temperature of 2,100° F. The hardened and reduced pellets are discharged, cooled, and screened to separate excess char and recover the product. Considerable detailed data resulting from the Bureau's work were presented in parts 1 and 2 of this bulletin.

In November 1965, it was reported that Armco Steel was planning a direct-reduction steelmaking plant in Houston, Tex. The plant would take pellets as raw material and use natural gas for the reduction. The company is developing its own process at Kansas City and is within sight of success.[13] The authors have since learned that reduction occurs in a shaft furnace and that product pellets containing over 90 percent iron have been made. The procedure is apparently an adaptation of the classical Wilberg process.[14]

Late in 1967, Midland-Ross Corp. announced a

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[12] Fine, M. M., P. L. Woolf, and N. Bernstein. Experimental Production and Smelting of Prereduced Iron Ore Pellets. BuMines Rept. of Inv. 6523, 1964, 28 pp.

Fine, M. M., and N. Bernstein. Experimental Production of Prereduced Pellets From Natural and Synthetic Magnetites. Proc. 23d AIME Ironmaking Conference, Pittsburgh, Pa., Apr. 13, 1964. American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, 1965, pp. 229–245.

[13] Chemical Week. V. 97, No. 19, Nov. 6, 1965, p. 70.

[14] Dormsjo, T. Olaf. Present Direct Reduction Research at Stora Kopparberg, Sweden. Ch. in Iron Ore Reduction, ed. by R. R. Rogers. The Macmillan Co., New York, 1962, pp. 279–297.

completely new process for the manufacture of reduced iron ore pellets.[15] The company will build and operate a 400,000-ton-per-year plant in Portland, Oreg., to supply an adjacent electric furnace steel mill. The metallizing procedure, which comprises reduction of fully fired iron oxide pellets with reformed natural gas in a shaft furnace, is called the M–R Process. The flow of gas and pellets is countercurrent; the overall heat demand in the metallizing operation is approximately 13 million Btu per long ton of product.[16]

The M–R process is apparently similar to the Purofer direct reduction technique developed in Germany, which is also approaching industrial application. The Purofer reduction equipment is a shaft furnace in which ore or fired pellets are reduced by a counterflow of reformed natural gas or coke oven gas. Optimum results are claimed to be possible at relatively high temperatures limited only by the softening point of the raw material. A pilot plant to test this technique has been in operation and a commercial venture (500 tpd) was scheduled to start up towards the end of 1968.[17]

Finally, there is the HyL process of M. W. Kellogg Div., Pullman, Inc., which has been a commercial success in Mexico since 1967. This procedure converts iron ore into sponge through the action of a mixture of hydrogen and carbon monoxide resulting from steam-reforming of natural gas. The reaction occurs in vertical fixed-bed chambers where gas flows downward through sized lump ore. The reactors pass through a four-step cycle including removal of product and loading of ore, secondary reduction, primary reduction, and cooling. In the usual plant design, four reactors are employed, each spending 3 hours in each stage. The hottest gas temperature reaches 1,800° to 2,250° F and an average of 85 percent metallization is achieved. Total installed HyL capacity will soon reach 1,200 tons per day, all of it dedicated to reduction of lump ore. However, the process is adaptable to reduction of fired pellets and pilot tests along these lines have been conducted.[18]

PRO'S, CON'S, AND PROPERTIES

At this stage of development, it is impossible to select any one process as superior to the others. It is apparent that several share a basic similarity and that, within the groups listed, the reduction vessel is sometimes the only significant variable. Claims as to the virtues of one machine over another are difficult to substantiate; one may be rapid, another thermally frugal.

Processes based on carbonized pellets appear to be the most applicable to foundry iron at low production rates. Adaptation to large-scale manufacture of blast furnace hot metal or to agglomeration of fine concentrates at the mine site is much less likely.

The introduction of lesser quantities of coke or noncoking coal into the pellet mix to serve as both fuel and reductant has one advantage, thermal economy. There is no doubt that intimate contact of carbon and iron oxide through the pellet accelerates the reduction and lowers the quantity of coal required. It is also possible to control the residual carbon content of the product pellets, which may be beneficial in subsequent smelting. The principal deterrent to the use of internal coal is that the ash dilutes the grade and adversely affects the iron-to-gangue ratio. Sulfur from coal could be a particularly troublesome contaminant owing to its affinity for metallic iron. The addition of internal limestone is not a completely satisfactory remedy. It may "fix" some or most of the sulfur in the slag constituents, but it also attracts and holds more sulfur within the pellet. All other variables being equal, the addition of higher percentages of coal and almost any quantity of limestone yields a weaker structure and lower crushing strength. If this results in a marginal product, higher temperatures may be required to compensate.

Processes in which all the reductant is external to the pellet seem to be most compatible with existing pelletization flowsheets because nothing foreign enters the product and the original iron-to-gangue ratio is preserved. As indicated earlier, the sulfur content of the product is readily controlled by limestone or dolomite. The external-carbon technique may require expenditure of more Btu, but the difference probably would not be very great. There is one detrimental feature connected with the use of fully fired iron oxide pellets: They are subject to the mysterious malady known as "reduction disintegration," which causes spalling and

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[15] American Metal Market. Midland-Ross' Pellet Unit Called 'World's First' Commercial Plant. V. 74, No. 208, Oct. 27, 1967, p. 6.

[16] Sturgeon, James H. The Commercial Production of Prereduced Pellets by the Midland-Ross Process. Proc. AIME Ironmaking Conf., Atlantic City, N.J., v. 27, April 1968, American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, 1969, pp. 140–143.

[17] Pantke, Heinz-Dieter, and Peter Puhlman, Einsatzbereich des Purofer-Verfarhrens (Initial Operation of the Purofer Process). Proc. Internat. Cong. on the Production and Utilization of Reduced Ores, Evian, France, May 1967, pp. 151–166.

[18] Skelly, Joseph F. The Effect of Iron Ore Characteristics on the Operation of the HyL Process. Proc. Internat. Cong. on the Production and Utilization of Reduced Ores, Evian, France, May 1967, pp. 141–148.

size degradation upon exposure to hot reducing atmospheres. This phenomenon varies from one raw material to another so the extent of the damage would be impossible to predict and could range from little to great.

The authors would like to dispel the implication that highly reduced pellets are weak and friable, and require special handling to deliver them into the blast furnace without excessive breakage.[19] Testing and observation of reduced pellets made in Bureau of Mines laboratories and SL/RN pellets smelted in the experimental blast furnace prove just the reverse. This experience has been confirmed by The Steel Co. of Canada, which found in large-scale work on simultaneous reduction and induration that reduced pellets can withstand the roughest treatment without the creation of fines.[20]

In terms of the usual parameters by which physical competence is evaluated, prereduced pellets can be made fully as durable as their oxidized counterparts. For example, in recent Bureau pilot plant experiments on magnetic taconite concentrate, prereduced pellets (⅜- to ½-diameter) made with lignite and anthracite had crushing strengths of 300 to 520 lb, respectively. The tumble and abrasion indices, from tests in a standard ASTM coke drum, were well under 1 percent. Similarly, the amount of breakage (minus 8-mesh) from dropping 1,000 grams of pellets 15 times from a height of 15 feet to a smooth concrete floor was also much less than 1 percent.

Just as iron rusts, so do prereduced pellets. Hence, prereduced pellets should be consumed expeditiously with a minimum of open storage. If necessary, weather resistance can be imparted by a thin coating of heavy oil, applied hot. Pellets sprayed with either Bunker "C" oil or petroleum wax were stored outdoors for 9 months with virtually no reoxidation. On a commercial scale, 1 to 2 gallons of oil or wax per ton of pellets should suffice.

Prereduced iron ore pellets can be adapted to a number of diverse uses, and this market potential has been demonstrated in sizable trial runs. Steelmaking tests in electric arc, converter, and open hearth furnaces yielded satisfactory quality and recovery of metal. The pellets were a satisfactory scrap substitute but created somewhat more slag and required more energy because of the ferrous oxide and gangue contents. As indicated earlier, foundry cast irons are readily made in cupolas or electric furnaces. The pellets can also function as copper precipitants.[21]

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[19] Chase, P. W., and D. L. McBride. Present and Future of Direct Reduction Processes in Latin America. Blast Furnace and Steel Plant, v. 51, No. 10, October 1963, pp. 868–897.

[20] Pearce, F. J. Prepared Discussion, presented at 23d AIME Ironmaking Conf., Pittsburgh, Pa., v. 23, Apr. 13, 1964. American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, 1965, pp. 244–245.

[21] Fraser, M. J., and C. R. Grigg. The SL/RN Process—Its Place in the Iron and Steel Industry. Eng. J., v. 48, No. 6, June 1965, pp. 29–34.

PART 7. — CONCLUSION

By M. M. Fine and N. B. Melcher

The Bureau of Mines is credited for developing the tremendous interest in prereduced agglomerates.[1] Pioneering research by the Bureau proved that finely ground iron ore concentrates could be simultaneously reduced and indurated to yield well-metallized pellets with physical properties equal to or better than commercial iron oxide pellets. This is only one of many avenues open to industry for the manufacture of prereduced pellets; however, the multiple choices may be narrowed down to the following four broad categories:

1. Manufacturing processes utilizing substantial quantities of coking coal in the pellet mix, so that carbonization, reduction, and agglomeration take place.
2. Those in which a lesser quantity of solid, but noncoking, fuel is incorporated in the pellet mix and is largely consumed thereafter. Reducing gas may also be supplied from an external source to supplement the solid fuel.
3. Processes employing solid, noncoking reductant completely external to the iron raw material. The latter may be fully fired oxide pellets or green balls; in the latter case reduction and induration are effected simultaneously.
4. This group comprises practices in which ore or fired pellets are converted to metal by gaseous reduction in shaft furnaces or vertical reactors.

Metallized pellets are a potentially versatile raw material and have been tested successfully in steelmaking, as a copper precipitant and in the blast furnace. In the latter capacity, it has been demonstrated by the Bureau of Mines and others that well-reduced pellets can greatly improve the productivity of an existing furnace while cutting the usage of coke.

Interest in prereduction remains keen. In November 1966, a 1-day symposium on the subject was held at Michigan Technological University, Houghton, Mich. In May 1967, a 3-day international meeting on prereduced products took place at Evian, France. The latter meeting was organized by Institut de Recherches de la Siderurgie Française (IRSID) under the patronage of the European Coal and Steel Community (ECSC) and with the participation of the metallurgical technical societies of Europe. Many of the processes described at that meeting have been discussed in this bulletin.

At the Evian meeting, it was prognosticated that prereduced materials would begin to challenge conventional burden materials by 1975. In that year, production is expected to reach 10 million tons and to be consumed equally by electric-furnace steelmaking and blast furnace ironmaking. A worldwide growth to 29 million metric tons by 1980 was foreseen[2] It now appears that the new industry has been launched at a pace which exceeds the forecasts. The DL–M plant at Mobile, Ala., the M–R plant at Portland, Oreg., and the New Zealand venture are typical of the new entries. The latter is one of seven Lurgi rotary kilns for iron ore reduction installed or being constructed.[3] Among these is the 300,000-ton-per-year plant of Falconbridge Nickel Co., Sudbury, Ontario, Canada, which will offer customers metallized nickeliferous iron ore pellets. The pellets will contain 1.5 percent nickel and more than 90 percent iron. Plans have been made to double the capacity to meet anticipated market demands.[4] Other developments worthy of note include that of Hamersley Iron Propriety, Ltd., Australia, which plans to produce 1.2 million tons annually of metallized agglomerates containing about 93 percent iron. If Hamersley succeeds in developing world markets, the productive capacity will be escalated to 3.0 million tons by the end of 1980.[5] Finally, Midland-Ross Corp. and Korp Industrie Und Handel GmbH &

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[1] Stukel, John E. Blast Furnace Burdens—Present and Future. Min. Cong. J., v. 52, No. 7, July 1966, pp. 57–61.

[2] Miller, Jack R., and Clarence H. Lorig. Some Observations on the Magnitude of Iron-Ore Prereduction. Internat. Cong. on the Production and Utilization of Reduced Ores, Evian, France, May 1967, pp. 446–450.

[3] Dailey, W. H. Steelmaking With Metallized Pellets. Proc. AIME Ironmaking Conf., Atlantic City, N.J., v. 27, April 1968, American Institute of Mining, Metallurgical, and Petroleum Engineers, New York, 1969, pp. 132–139.

[4] Skillings Mining Review. Falconbridge to Produce Iron-Nickel Pellets. V. 58, No. 10, Mar. 8, 1969, p. 8.

[5] Innes, J. A. Why Hamersley Investigated Metallizing. Eng. & Min. J., v. 170, No. 5, May 1969, pp. 92–93.

Co., recently announced a joint venture to produce and sell metallized pellets and specialized steel products in the European Market. New facilities in the Netherlands and West Germany will have an initial combined capacity of 1.0 million tons of pellets by 1970.[6]

Although the prereduced-pellet industry is much too young to be committed as to market, most of the product is tailored for electric furnace steel. Furthermore, the pellets are being sold to specialty steel producers and fledgling steelmakers, the so-called "mini-mills."[7] These small operators in the United States, and around the world, are capturing a share of the steel market by melting scrap in electric furnaces. Although the number of products is now limited, it can be extended in the future by using metallized pellets, alone or in combination with scrap.

The hesitancy on the part of large, integrated steelmakers regarding prereduced pellets for the blast furnace results in part from the still unresolved question of economic feasibility. Significant steps have been taken, however. Two independent analyses of the cost of making hot metal indicate that reduced pellets could be more profitable than other iron raw materials in both investment capital and daily cost of operation. Extended trials on a plant scale would resolve any lingering doubts that well-made prereduced pellets are indeed a superior blast furnace raw material. It is inevitable that such tests will take place when metallized pellets are available worldwide in the next few years.

Although there are other deterrents to the speedy adoption of reduced burdens for ironmaking in the next 10 years, the oxide pellet-making capacity of the United States will double, to the point that about 75 percent of domestic ore production will originate from this source. This will engage the energies of ironmakers as blast furnaces turn to pellets and away from natural ore and concentrates, perhaps even from sinter. However, many aged blast furnaces in the United States are too poorly equipped to realize the maximum benefits from regular pellets; they would be even less suited to a diet of metallic pellets. It has also been claimed that the existing and expanding pellet industry would require major, costly plant modifications to manufacture a reduced product. This is not as discouraging as it seems. Additional storage and handling facilities for solid reductants would be required but the critical tools, the indurating furnaces, should be adaptable. Reduction-induration of green balls with internal coal could be accommodated by traveling grates or shaft furnaces, particularly the latter. The grate-kiln system could be adapted to reduction of green balls with either internal or external coal. Shaft furnaces can also accommodate gaseous reductants with little modification.

The global steel industry outside the United States will not be idle during the next decade. Although the rest of the world has lagged in manufacture and utilization of iron ore pellets, a considerable amount of "catching-up" will take place by 1975. A record amount of foreign steel is coming into this country; the yearly rate of increase far exceeds that of many other imports. If this is the situation at present while most overseas furnaces are on ore and sinter, what will the future bring?

The U.S. steel industry will need to use every cost cutting tactic to remain competitive, not only with foreign steel but with substitute materials such as plastics and nonferrous metals. In this connection, prereduced pellets rank high as a device to improve plant output. Although it may take a few years, use of prereduced burdens in both steel furnaces and blast furnaces will eventually become routine. As H. S. Harrison stated recently . . . "It would be a mistake to think that/oxide/pellets are the final answer in the development of iron ore products."[8]

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[6] Skillings Mining Review. Midland-Ross Corp. Venture for Metallized Pellets and Steel. V. 58, No. 7, Feb. 15, 1969, p. 25.

[7] Business Week. The Mini-mill: Steel on a Budget. Mar. 29, 1969, pp. 66–70.

[8] Harrison, H. S. The Changing Iron Ore Industry. Blast Furnace and Steel Plant, v. 54, No. 1, January 1966, pp. 40–43.

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APPENDIX

Table A-1.—Estimated materials balance using lignite to produce 2 million annual long tons of metallized pellets

Item	Hourly rate, long tons
Input:	
Concentrate, dry basis	330.0
Bentonite	2.4
Lignite	177.0
Lignite char	27.0
Limestone	8.5
Discharged:	
Pellets, plus ¼-inch	246.0
Pellets, 6-mesh by ¼-inch	6.5
Recycle magnetics, 6-by 28-mesh[1]	3.0
Lignite char, plus 6-mesh	6.0
Lignite char, minus 6-mesh	21.0
Minus 28-mesh	14.0
Stack dust, magnetic[1]	10.0
Stack dust, nonmagnetic	10.0

[1] Credited as concentrate.

Table A-2.—Equipment list for prereduced-pellet plant utilizing lignite (Item numbers refer to figure 29)

Item	Item description	Quantity	Cost (in $1,000)		
			Equipment	Labor	Total
	RAW MATERIAL HANDLING				
19	Pneumatic conveyor, 10 TPH bentonite	1	20.0	5.0	25.0
21	Railroad unloading pit	1	2.5	.5	3.0
22	Railroad pit conveyor, 3 by 130 feet	1	20.5	4.5	25.0
23	Lignite surge pit	1	25.0	25.0	50.0
24	Conveyor to lignite tripper, 3 by 200 feet	2	50.0	11.0	61.0
25	Lignite conveyor and tripper, 3 by 200 feet	1	32.0	7.0	39.0
26	Lignite bin	6	33.0	3.3	36.3
27	Conveyor to limestone tripper, 2½ by 200 feet	1	26.4	5.7	32.1
28	Limestone conveyor with tripper, 2½ by 200 feet	1	29.0	6.4	35.4
29	Linestone bin	6	16.0	1.6	17.6
30	Oil pump, railroad unloading, 1000 gph	1	4.0	1.0	5.0
31	Oil tank, storage, 15,000 gals	1	3.5	.3	3.8
	Total of raw material handling		261.9	71.3	333.2
	BALLING CIRCUIT				
1	Concentrate conveyor with tripper, 3 by 200 feet	1	34.0	7.5	41.5
2	Concentrate bins and feeders, 350 tons	6	42.0	4.2	46.2
3	Mixing conveyors, 2½ by 50 feet	6	42.0	9.2	51.2
4	Reel mixers	6	50.0	2.5	52.5
5	Conveyors to balling drums, 2½ by 15 feet	6	15.0	3.3	18.3
6	Weightometers	6	26.0	7.0	33.0
7	Balling drums, 10 by 40 feet	6	240.0	60.0	300.0
8	Conveyor, recycle, 2 by 15 feet	6	14.0	3.1	17.1
9	Conveyor, recycle, 2 by 60 feet	6	41.4	9.1	50.5
10	Conveyor, recycle, 2 by 15 feet	6	14.0	3.1	17.1
20	Bentonite bins, 15 tons	6	5.0	.5	5.5
	Total of balling circuit		523.4	109.5	632.9

Table A-2.—Equipment list for prereduced-pellet plant utilizing lignite—Continued

Item	Item description	Quantity	Cost (in $1,000)		
			Equipment	Labor	Total
	INDURATING SECTION				
11	Rotary kilns, 12 by 400 feet	6	4,800.0	1,200.0	6,000.0
12	Lignite feeders, 250 TPH	6	43.0	11.0	54.0
13	Rotary coolers, 10 by 200 feet	3	1,800.0	450.0	2,250.0
43	Pump for stack dust, 25 TPH	1	3.5	.3	3.8
44	Wet magnetic separator, 25 TPH	1	5.5	1.5	7.0
45	Pump for magnetic stack dust, 15 TPH	1	3.0	.3	3.3
46	Pump for nonmagnetic stack dust, 15 TPH	1	3.0	.3	3.3
	Total of indurating circuit		6,658.0	1,663.4	8,321.4
	PRODUCT HANDLING				
14	Conveyor from coolers, 3 by 120 feet	1	16.0	3.5	19.5
15	Screen, cooler products	1	10.0	1.0	11.0
16	Conveyor, to storage or loadout, 3 by 200 feet	2	64.0	14.1	78.1
17	Hopper, pellet loadout, 250 tons	1	6.3	.6	6.9
18	Pellet conveyor with tripper, 3 by 1,500 feet	1	240.0	52.8	292.8
34	Conveyor, 1½ by 120 feet	1	13.8	3.0	16.8
35	Double-deck screen	1	10.0	1.0	11.0
36	Magnetic pulley	1	5.0	1.2	6.2
37	Conveyor, 1¼ by 60 feet	1	5.0	1.1	6.1
38	Conveyor to lignite pile, 1½ by 450 feet	1	42.5	9.4	51.9
39	Magnetic cross belt	1	5.0	1.3	6.3
40	Grinding mill	1	12.5	1.2	13.7
41	Pump, magnetic fines	1	2.0	.2	2.2
42	Pump, minus 28-mesh material	1	3.0	.3	3.3
32	Oil spray system, 300 gal per hr	2	8.0	2.0	10.0
	Total of product handling		443.1	92.7	535.8

Table A-3.—Capital costs of prereduced-pellet plant

Item		Utilizing lignite	Utilizing anthracite
1	Labor to install equipment	$ 1,936,900	$ 2,036,900
2	Equipment, delivered cost	7,886,400	8,286,400
3	Buildings[1]	400,000	400,000
4	Structures (8 pct of item 2)	630,900	662,300
5	Foundations (6 pct of item 2)	473,200	497,200
6	Instrumentation (3 pct of item 2)	236,600	236,600
7	Electrical (4 pct of item 2)	315,500	331,500
8	Piping (5 pct of item 2)	394,300	414,300
9	Painting (2 pct of item 2)	157,700	165,700
10	Miscellaneous (10 pct of item 2)	788,600	828,600
11	Land, site preparation, rail trackage, improvements	814,000	814,000
12	Utilities portion (10 pct of item 2)[2]	788,600	828,600
13	Installed equipment on site (summary, item 1-12)	14,822,700	15,502,100
14	Indirect costs[3] (40 pct of item 13)	5,929,100	6,200,800
15	Total plant costs (summary of items 13–14)	20,751,800	21,702,900
16	Interest during construction (5 pct of item 15)	1,037,600	1,085,100
17	Mobile equipment (crane, loader, etc.)	150,000	150,000
18	Fixed capital costs (summary of items 15–17)	21,939,400	22,938,000
19	Depreciation per pellet ton[4]	$0.548	$0.573

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[1] 60,000 sq ft at $5 per sq ft, and 5,000 sq ft at $20 per sq ft.

[2] Only 10 pct is used, since most of the utilities would be available through the mine and concentrating plant.

[3] Engineering, construction, and contractor's fees and contingencies.

[4] Fixed capital costs divided by 20 years' production, or 40 million tons of pellets.

Table A-4.—Estimated labor requirements for prereduced pellet plant

Job description	Men per shift
Loads concentrate bins	1
Operates balling drums	2
Operates kilns and coolers	1
Assists kiln operator	2
Attends cooler discharge, samplers, and screens	1
Attends sorting section	1
Unloads raw materials	3
Loads pellets out or diverts to storage	1
Miscellaneous duties as required	2
Total	14

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