

# RESEARCH COUNCIL OF ALBERTA

Information Series No. 40

## THE REDUCTION OF CLEAR HILLS IRON ORE BY THE R-N PROCESS

by

C.S. Samis and J. Gregory



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BY THE R-N PROCESS

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C. S. Samis and J. Gregory

Research Council of Alberta  
87th Avenue and 114th Street  
Edmonton, Alberta  
1962

## PREFACE

The program and report on the reduction of Clear Hills iron ore by the R-N process represents co-operation and contributions by many persons and organizations. The authors are extremely pleased at the fine relationship between Premier Steel Mills Ltd. and the Research Council of Alberta during the joint undertaking of the program. Mr. G. R. Heffernan, managing director of Premier Steel Mills Limited, provided the information on the background of the iron and steel industry in Western Canada. Dr. C. S. Samis, Professor of Metallurgical Engineering, University of British Columbia, represented Premier Steel Mills at the tests at Birmingham, Alabama, and is co-author of this report.

Sincere appreciation is extended to the R-N Corporation for their extremely willing co-operation in all aspects pertaining to the tests and for kindly providing the descriptions and photographs of the R-N process and plant, as well as the operating procedures and test result data. The authors gratefully acknowledge the assistance of Mr. G. G. Ridley and Dr. John Dartnell of Stewarts and Lloyds, England, in the interpretation and assessment of the R-N tests.

Thanks are extended to Mr. George Viens and Mr. Dan Pickett of the Department of Mines and Technical Surveys for their presence and valuable assessments during the tests at Birmingham, Alabama, and also to their Department for making their assistance possible. The co-operation of Research Council of Alberta personnel in this project is greatly appreciated. Dr. C. P. Gravenor prepared the section on the iron ore deposits. Dr. R. Green observed and described the mining and shipping of the iron ore. Mr. J. Groot provided the drafting services and Mrs. V. L. Stover and her staff edited and prepared the report for publication.

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# THE REDUCTION OF CLEAR HILLS IRON ORE BY THE R-N PROCESS

## I. INTRODUCTION

The development of an iron and steel industry is of prime consideration in the industrial growth of any nation or area. This statement is no less true for Alberta, and as a consequence the Research Council of Alberta has throughout its entire history been actively interested in the development of this vital basic industry. Indeed, the Research Council was established as a consequence of geological investigations on iron ore deposits in Alberta by Dr. J. A. Allan during the First World War. The interest and activity of the Research Council in this field has been particularly marked in recent years culminating in its participation in the semicommercial demonstration plant tests on the reduction of Peace River iron ore by the R-N process at Birmingham, Alabama.

This report traces the developments leading to the R-N testing program, and also offers a description of the tests and an analysis of the results.

## II. HISTORY OF THE DEVELOPMENT OF THE PEACE RIVER IRON ORE DEPOSIT

In 1953 geologists of the Phillips Petroleum Company and of the Oil and Gas Conservation Board noted the presence of iron during the examination of core from a Phillips Petroleum Company well (Phil C No. 1 well). Subsequently the iron-bearing materials were found in three other Phillips wells.

In May, 1954, D. B. McDougall obtained an iron prospecting permit covering 100,000 acres in the Clear Hills area. He drilled 11 holes in the Swift Creek deposit and a report was written on the results of this work. For this investigation the chemical analyses were performed by W. E. Harris and K. B. Newbound at the University of Alberta, and the mineralogical analysis by C. P. Gravenor of the Research Council of Alberta. A bulk sample was sent to the Mines Branch of the Department of Mines and Technical Surveys. Later in 1954 a surface geological party was sent in by McDougall and a report was prepared by H. S. McColl of his staff. This report carries the first mention of outcrops along Swift Creek.

In 1956 mineralogical and age dating reports were prepared by A. C. Lenz at the University of Alberta under the direction of C. P. Gravenor. This was the first comprehensive report on the constituents of the iron-bearing material. In the same year the Research Council of Alberta sent R. Green and D. J. Kidd up to examine the Swift Creek deposit. Also in 1956, Premier Steel Mills Ltd. obtained a report from McDougall on his findings and took out several permits on the Swift Creek deposit.

In 1957, Premier Steel Mills Ltd. continued their investigations, working southwards, and along with a local farmer (Anderson) located the southern Clear Hills deposit. Also in 1957, G. L. Colborne, graduate student at the University of Alberta, examined the Swift Creek occurrence for Cleveland Cliffs Iron Company and wrote a thesis on this material.

In 1958, D. J. Kidd of the Research Council mapped all the outcrops along the southern Clear Hills indicating the best areas for further prospecting. Kidd made an estimate of 130 million tons available for strip-mining operations. At the same time, Premier Steel Mills Ltd. continued their operation in an examination of the Worsely occurrence by stripping operations.

In 1959, Preliminary Report 59-3 entitled "Iron Occurrence in the Peace River Region", published by the Research Council, indicated reserves of 1-1/2 billion tons. During the same year, Premier Steel Mills Ltd. core drilled the southern Clear Hills deposit. Throughout this work, samples were sent to the Mines Branch at Ottawa, the Ontario Research Foundation, and several private companies for testing purposes.

A symposium on the subject of iron and steel in Western Canada was sponsored by the Research Council of Alberta and the Coal and Metallurgical Divisions of the Canadian Institute of Mining and Metallurgy in Edmonton in September, 1959. At this meeting, Dr. C. P. Gravenor presented data on the Peace River iron ore deposit. This data appears in Chapter III of this report. It was apparent from the proceedings that there is room for an integrated steel industry in Western Canada. In the interests of the industrialization of Alberta, it was obvious that maximum information on this Alberta iron deposit should be available at an early date.

### III. PEACE RIVER DEPOSITS

by

Dr. C. P. Gravenor, Chief,  
Earth Sciences Branch, Research Council of Alberta

#### Regional Geology

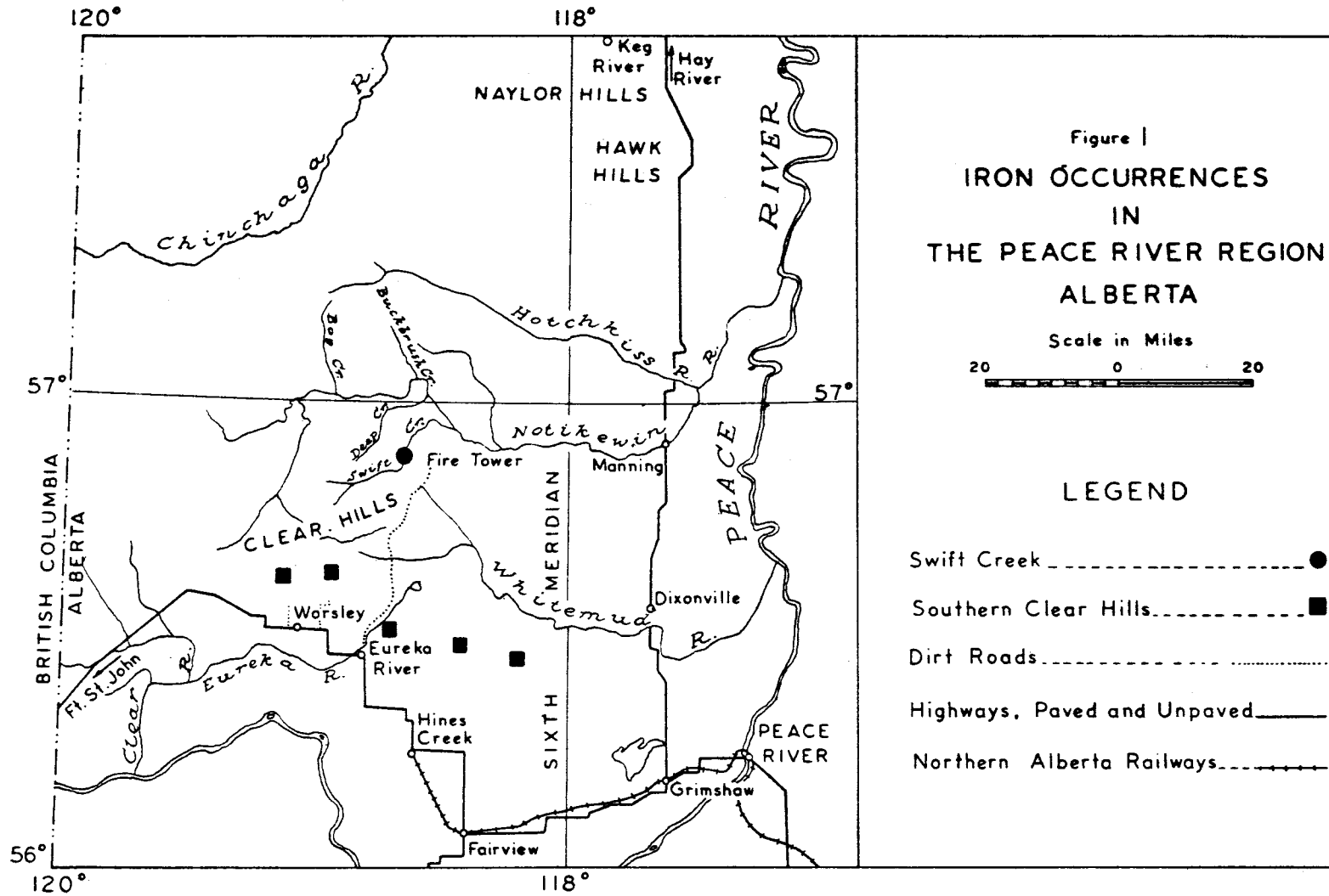
The main iron deposits of the Peace River region are found in the Clear Hills approximately 50 miles northwest of the Town of Peace River and 25 miles north of Hines Creek (figure 4). The Clear Hills may be reached from Worsley and Eureka River, on the Hines Creek-Fort St. John highway. For purposes of description the deposits are divided into two main groups, Swift Creek and the Southern Clear Hills occurrences.

The Clear Hills are underlain by Upper Cretaceous sandstone and shales of marine, brackish, fluvial and deltaic origin. The nature of the origin of the rocks makes it rather difficult to correlate the sandstone and shale members from one area to another and to give precise ages to the rocks. From the available data (Kidd, 1959) it would appear that the iron-bearing strata are Wapiabi in age and fossil evidence from strata above and below the iron-bearing bed suggests that there is an unconformity at the base of the iron deposit on Swift Creek and at Worsley.

The sedimentary strata in the Clear Hills area is nearly horizontal and hence the depth to the iron-bearing zone can be predicted with some certainty from a topographic map. Local folds are known to occur in the area but are apparently of minor importance.

#### Swift Creek Deposit

Oolitic iron sandstone outcrops as a weathered seam along both sides of Swift Creek for a distance of three-quarters of a mile. Southeast of the Swift Creek outcrop the bed is approximately 20 feet thick (McDougall, 1954). Towards the southwest the bed thins and is absent 9 miles south-southwest of the Swift Creek outcrop. The northern limit of the seam is 6 miles south of the Notikewin River (McDougall, 1954) and is erosional. From core hole data, McDougall (1954) estimated that the deposit underlies an area of 42.9 square miles and that the beds dip south-southwest at 10 feet to the mile.





### Southern Clear Hills Deposit

Friable red-weathering sandstone outcrops are found over a distance of 37 miles along the southern slopes of the Clear Hills. Generally the individual outcrops are less than 100 feet in length and there is some uncertainty on the continuity of the beds between outcrops. Exposures of oolitic sandstone reach thicknesses of 30 feet, and average more than 7 feet.

The northern extent and dip of the iron-bearing zone - into the Clear Hills - is not known and hence it is difficult to make an accurate assessment of potential strippable reserves.

### Petrography, Chemistry, and Reserves

The iron-bearing sandstone is composed of a friable aggregate of brown oolites about 0.5 mm. across. They form from 60 to 70 per cent of the rock in the upper part of the seams and decrease in amount toward the bottom where the sandstone may contain up to 85 per cent detrital quartz. The oolites consist of goethite and quartz about nuclei of quartz fragments. A small proportion have nuclei composed of feldspar fragments or a green chamosite-like mineral. The oolites are closely packed and lie in a matrix of quartz, goethite, siderite, calcite, and green chamosite-like mineral. Core samples generally contain more siderite than the weathered surface samples.

Four analyses of samples from the Swift Creek and 5 from the southern Clear Hills deposits are given in Tables 1 and 2. Analyses of 24 samples from the Swift Creek deposit show an overall grade of 31.55 per cent iron and a range of from 37.17 per cent to 11.0 per cent (Kidd, 1959).

Table 1. Some chemical analyses of iron-rich samples from the Swift Creek deposit (after Kidd, 1959)

	Weight per cent			
	3	4	5	6
Total Fe	30.15	34.26	37.17	32.98
SiO <sub>2</sub>	25.36	23.88	20.43	29.26
Al <sub>2</sub> O <sub>3</sub>	6.24	5.94	5.72	5.47
MgO	-	1.17	0.78	1.60
L.O.I.	-	12.00	9.69	11.91
P	0.58	0.62	0.45	0.45
S	0 to 0.70	tr.	-	-

Table 2. Chemical analyses of iron-rich samples, southern Clear Hills  
(after Kidd, 1959)

	Weight per cent				
	13	14	15	16	17
Total Fe	27.33	41.47	38.10	20.90	18.49
Mn	0.25	0.29	0.29	0.18	0.16
SiO <sub>2</sub>	41.74	18.84	25.32	39.46	55.68
Al <sub>2</sub> O <sub>3</sub>	5.94	6.63	6.02	3.97	5.10
CaO	0.63	0.52	0.95	9.19	3.40
MgO	0.83	0.82	0.48	1.05	0.90
TiO <sub>2</sub>	0.24	0.34	0.23	0.17	0.27
P	0.377	0.353	0.536	0.278	0.120
S	0.007	0.012	0.008	0.032	0.015
C	0.65	0.66	0.58	2.55	1.10
Combined water	6.84	8.79	7.60	5.37	3.27

Sulphur content ranges from zero to 0.03 per cent (Kidd, 1959). The five analyses of the samples from the southern Clear Hills show an average of 29.2 per cent iron.

The silica and phosphorus content would appear to be high for long-established North American smelting methods but apparently a concentrate can be prepared which is suitable for blast furnace operation (Kidd, 1959). The percentages of other constituents such as sulphur are within metallurgically acceptable limits.

In the absence of a detailed drilling program it is difficult to give an accurate assessment of reserves. Based on available data, Kidd (1959) has suggested 1.5 billion tons for the Swift Creek deposit and 136 million for the southern Clear Hills deposit. In terms of strippable reserves the Swift Creek estimate might have to be reduced due to the heavy overburden on certain parts of the deposit.

Other ferruginous Cretaceous sandstone deposits in the Peace River region have been reported from the Dunvegan area, Spirit River Town, and the Smoky River (Kidd, 1959). Some of these deposits are similar petrographically and chemically to those described from Swift Creek but very little is known about their extent.

## IV. IRON AND STEEL INDUSTRY IN WESTERN CANADA

Early operations in the iron and steel industry in Western Canada consisted of fagot-mills in Alberta, Manitoba, and Saskatchewan before and during the First World War. These mills supplied some of the local needs for bars and small shapes and were based on local scrap supplies. In time, these mills largely closed down and it was not until after the Second World War that a substantial steel industry became established in the West utilizing electric furnaces and local scrap.

Alberta

In 1955, Premier Steel Mills Ltd. erected a bar mill at Edmonton using an electric furnace and a scrap feed. A second furnace was installed in 1957, bringing the capacity up to 100,000 tons of ingots per year. The mill consisted of a 16-inch roughing mill with 5 stands of 12-inch mill for finishing.

In 1956, Alberta Phoenix Tube and Pipe Ltd. built a plant having a capacity of 200,000 tons per year to produce pipe from 2 1/2 inches to 16 inches in diameter. In 1960, Camrose Tubes Limited opened a pipe mill of 350,000 tons per year capacity to produce pipe from 18 inches to 42 inches in diameter. During the same year, Big Inch Pipe Corporation Ltd. constructed a 300,000 ton-per-year mill in Calgary to produce 18- to 36-inch pipe. Page-Hersey Tubes Ltd. have announced plans for the construction of a plant in Camrose to produce pipe in the 2- to 16-inch size.

British Columbia

In Vancouver, a rolling mill based on scrap is operated by Vancouver Rolling Mills Ltd. Its steel furnace has a capacity of 70,000 tons per year. Canadian Western Pipe Mills Ltd. established in Vancouver in 1954 a pipe mill of 120,000 ton per year capacity to produce pipe from 1/2 inch to 4 1/2 inches in diameter. While the British Columbia Steel Industry is based on scrap, substantial quantities of iron ore are being shipped from the coastal inlands to British Columbia.

In 1960, Consolidated Mining and Smelting Company of Canada Ltd. opened at Kimberley a 100-ton-per-day electric smelter for pig iron, based on the iron residue of their mill tailings from the Sullivan mine. The company has plans for expansion to 300 tons of pig iron per day as well as an entry into steel making.

Manitoba

Manitoba Rolling Mills Ltd., a subsidiary of Dominion Bridge Co. Ltd., operate a rolling mill producing bars and light structurals utilizing electric furnaces and local scrap. The capacity is 120,000 tons of ingots per year.

Saskatchewan

In 1957, Prairie Pipe Manufacturing Co. Ltd. was established in Regina. An associated company, Interprovincial Steel Co. Ltd. with a capacity of 110,000 tons per year, in 1960 started production of ingots to be rolled into skelp for the former company as well as to supply in part the requirements of the Big Inch plant in Calgary.

The rapid development of industry and construction in Western Canada since the Second World War has presented a growing demand for bars and light structurals with the result

that steel plants totalling a capacity of 400,000 tons per year have been established in the four western provinces. In addition, the centrifugal casting plants of Anthes-Imperial at Calgary and Griffin Wheel Company in Winnipeg require an additional 40,000 tons of iron per year. Recent years have witnessed the amazing growth of pipe mills and their combined capacity is in excess of 1,000,000 tons per year. With the exception of the Saskatchewan plant the skelp for remaining plants is imported into Western Canada.

While the iron needs of the present rolling mills have been met by scrap supplies, the future expansion of the iron and steel industry together with the requirements for the pipe mills is raising the question of a basic source of supply of iron. The Consolidated Mining and Smelting Company of Canada Ltd. are already producing pig iron from their iron-rich tailings. Interest is also being shown by various groups in the Kelsey Lake deposits of Saskatchewan, the Peace River deposits, the Burmis magnetic sands, and the magnetites of Dillon, Montana.

## V. STEPS LEADING TO R-N TESTS

### Metallurgical Investigations

It was early apparent to Premier Steel Mills Ltd. that the Peace River iron ore was unsuitable as a blast furnace feed and the company began investigating various methods of processing low-grade ores.

At Watenstedt, Germany, a plant was in operation on a large scale utilizing low grade ores. The process in use was the Krupp-Renn process, a rotary kiln method using low-grade coal for reduction and oil or powdered coal for fuel. In 1956, samples of the ore and of Alberta coal were sent to Germany for pilot plant tests. Following satisfactory results on the test, Premier Steel Mills Ltd. sent a technical team to Germany to observe the commercial operation at Watenstedt. The basic conclusion arrived at by the team was that the process was satisfactory but that the product was unsuitable for anything but blast furnace charge material.

It was then recommended by Dr. Colclough of the British Iron and Steel Research Association, W. Voice of the British Iron and Steel Research Association, and P. E. Cavanagh of the Ontario Research Foundation that further work be done toward concentration of the ore. As a result, Cavanagh of the Ontario Research Foundation was commissioned to carry out a program of investigation involving flash roasting and magnetic concentration. Preliminary tests indicated that a concentrate running 54 - 56 per cent iron which was suitable charging material for a blast furnace could be produced.

Following these laboratory test results which indicated that the Peace River ore could be concentrated up to a grade of 54 - 56 per cent iron, it was decided that further work should be done on the geological and mining end of the project, and in 1958 a program of development was initiated calling for the drilling of closely spaced holes to definitely prove tonnage, grade, and overburden. Unfortunately, great difficulty was encountered in recovering reliable samples by standard drilling techniques and the program was abandoned until a suitable technique could be developed. In the meantime, some large bulldozer cuts were made into the deposit and bulk samples were taken for further metallurgical work. In late 1959, a dry drilling technique using an air blast to bring up the sample was tested and found to work successfully, and by March 1960 25,750,000 tons averaging 32.65 per cent iron had been proven.

In the meantime, metallurgical work was going along four different approaches as follows:

- (1) An attempt to verify the laboratory work of Cavanagh on a pilot plant basis;
- (2) Evaluation of the material for direct reduction processes by the Mines Branch in Ottawa;
- (3) Laboratory work on heavy media separation by Professor E. O. Lilje of the University of Alberta;
- (4) Laboratory pilot plant testing by the R-N Corporation.

Of the various methods being tested, the R-N process gave the best results. Laboratory tests on the Clear Hills area of the Peace River deposit at the R-N laboratory offered a product running 90 - 92 per cent iron with an 87 per cent recovery of iron from the ore. This was considered an excellent result. An analysis of costs indicated that R-N briquettes for electric furnace melting stock on a scale of 300,000 tons per year could be laid down in Edmonton for direct costs of \$25.00 per ton, and that electric furnace ingots could be made for \$55.00 per ton. These costs compared favorably with current integrated steel-making costs in Canada.

The next step to be taken in the development of the Clear Hills ore was to ship 3,000 dry tons of ore to the large R-N pilot plant at Birmingham, Alabama, for production scale testing of the ore. This would involve engaging the R-N plant for a period of 30 days and actually producing 1,000 tons of briquettes. From this test, full information would be obtained regarding operating costs and final feasibility of the process as applied to the Clear Hills ore. Total cost of the pilot plant operation was estimated at about \$300,000, including ore, freight, etc.

#### Arrangements of Participants

Up to this time, both the Research Council of Alberta and Premier Steel Mills had undertaken considerable studies and investigations in the development of the Clear Hills iron ore. The establishment of a basic steel industry in Alberta would constitute development of the most significant proportions. It was only appropriate, therefore, that the Province of Alberta through the Research Council of Alberta agreed to participate with Premier Steel Mills in undertaking the pilot plant testing of the feasibility of the R-N process on Clear Hills ore.

#### Mining and Shipping of Ore

The ore was obtained from a strip pit located in Lsd. 16, Sec. 33, Tp. 87, R. 7, W. 6th Mer. The bed, about 10 feet thick in the pit, was 100 feet by 150 feet. Ore was loaded onto trucks, transferred to Hines Creek, and there transferred to ore cars by means of a conveyor belt.

The pit was opened during the middle of June, 1960, but due to unfavorable weather conditions, trucking operations did not begin until June 27th. About six days were required to load 348.6 tons of ore into 45 ore cars.

A second shipment of 1800 tons was made from the same pit during August.

## VI. R-N PILOT PLANT TESTS

### Description of the R-N Process

The R-N process is a process, subject to Canadian and other patents, for producing metallic iron briquettes or metallic powder from iron ores or iron concentrates by metallization of the iron-containing ores or concentrates by means of solid carbonaceous matter in a rotary kiln

below the melting point of the materials, and with subsequent concentration and briquetting of the metallic iron. The process had been successfully demonstrated during five years' operation of the R-N pilot plant, Birmingham, Alabama, on a great variety of ores and concentrates with iron contents from about 30 per cent iron to 70 per cent iron, and with ore particle sizes ranging from about 20 mesh to about 1 inch. Even very fine-grained ores and concentrates with particle size less than 200 mesh have been successfully treated after pelletizing the material before processing. As the process is carried out below the melting temperature of the materials, the process is suitable to basic as well as acidic ores. Thus the versatility of the process as regards ore grade and ore particle size has been thoroughly established.

A similar versatility has been established with respect to the fuel requirements of the process. The fuel requirements consist of:

#### 1. Solid carbon for reducing agent

The solid carbonaceous matter is mixed with the ore and acts as reducing agent for the iron oxides, forming CO which is combusted above the ore bed in the rotary kiln. A wide variety of carbonaceous matter as, for instance, coke breeze, anthracite and char with low as well as high sulfur and ash contents, have been used successfully for this purpose.

#### 2. Additional gaseous fuel for heating the kiln load

For this purpose, it has been established that natural gas, mineral oils (heavy as well as light), coal char gases, and other industrial gases of similar composition can be used. An extremely important development in this respect is the development of special processes for charring coking as well as noncoking coal - processes which can be integrated with the R-N reduction kiln, producing char for reducing agent and hot char gas as fuel for the kiln. Thus, coal, coking or noncoking, can be used as the sole source of fuel required for the process. The possibilities of the R-N process are thereby greatly enhanced and expanded, especially for locations and countries which lack coking coals for conventional steel production or where gaseous fuel is costly.

### Description of the R-N Pilot Plant

The R-N plant consists primarily of a large, rotary kiln similar to a cement kiln. Ore mixed with a solid reductant such as coke breeze or coal char is fed into the kiln which is fired by oil or hot combustible gases as, for instance, char gas or partially combusted natural gas. The material travels through the kiln, is heated up and reduced to metallic iron progressively in counter-current to the kiln gases. The discharge from the kiln is cooled, separated from excess coke, ground and magnetically separated for removal of gangue, leaving a high-grade metallic iron concentrate which is pressed into briquettes suitable for steel melting stock.

The rotary kiln is 150 feet long with 9 feet OD and 7 1/2 feet ID. Spaced along the length of the kiln and penetrating the kiln wall are 6-foot-long, open-end air inlet tubes, extending to the axis of the kiln. Through these tubes regulated amounts of air are introduced for progressive combustion of the combustible gases in the kiln. Each tube acts as an inverted burner, admitting air into a stream of combustible gas. Instead of one strong, high-temperature flame, as in the case of only one burner at the end of the kiln, there are as many smaller flames as there are air tubes. This system of heating provides very precise control of the temperature profile over a major portion of the kiln, with corresponding increase of capacity and with relatively low flame temperatures in the kiln, minimizing the danger of clinking.

For temperature control, ten thermocouples are mounted in the kiln. Thermocouple No. 1 measures the temperature in the stationary precombustion chamber at the discharge end of

the kiln. Thermocouple Nos. 2 - 9 are mounted in the rotary kiln itself, protruding through the kiln wall. Thermocouple No. 2 extends to the kiln axis and gives the temperature of the kiln gases at this point. Thermocouples Nos. 3 - 9 extend about eight inches inside the kiln lining and read the temperatures partly in the gas phase and partly in the kiln load as the kiln rotates. Thermocouple No. 10 shows the temperature of the gases leaving the kiln.

The charge travels through the kiln towards the discharge end at a predetermined rate and a desired bed depth. The discharge material, consisting of reduced ore, excess coke, lime, and ashes, is cooled in a rotary cooler and subsequently separated by screening and magnetic methods. The excess coke is recovered and recycled in the process. The reduced ore is subjected to stagewise grinding in ball mills with intermediate magnetic separation for successive removal of gangue and progressive increase of the iron in the product. The recovery and cleaning of the excess coke as well as the grinding and magnetic concentration of the metallic iron values can be done commercially by either wet or dry methods. The R-N demonstration plant, however, is equipped only with wet processing equipment for these steps.

The final iron concentrate is dewatered, dried, and briquetted. A number of photographs of the R-N plant at Birmingham, Alabama, are appended.

### Raw Materials

#### Iron Ore

A total of 5,285.6 tons of Clear Hills ore was received for R-N processing in the pilot plant at Birmingham, Alabama. The ore was received in two shipments as follows:

First shipment	3,486.05 natural net tons
Second shipment	1,799.55 natural net tons

The approximate analysis of the ore on a dry basis was as follows:

Total Iron	37.0 %
Silica	20.0 %
Alumina	8.0 %
Lime	4.0 %
Phosphorus	0.7 %
Sulfur	0.02%

Preliminary laboratory tests had shown that the reducibility of the ore was satisfactory for one-half inch particles at a reduction temperature of 1970°F. The "as received" ore was, therefore, crushed and screened to minus one-half inch by zero in the pilot plant crushing plant before processing in the kiln.

#### Coke

The coke used in the test was delivered from the Republic Steel Corporation plant at Gadsden. The particle size of the coke was minus 3/16 inch, its carbon content was 78.26 per cent and its calorific value 12,500 B.t.u. per lb.

### Natural Gas

The heat value of the gas was 1,042 B.t.u. per cu. ft. Its analysis was:

CH <sub>4</sub>	93.8 %
C <sub>2</sub> H <sub>6</sub>	2.6 %
C <sub>3</sub> H <sub>8</sub>	0.6 %
C <sub>4</sub> H <sub>10</sub>	0.25%
C <sub>n</sub> H <sub>n</sub>	0.25%

In the combustion chamber of the kiln, the gas is partially combusted and the resulting hot combustible gases are passed into the kiln.

### Limestone

Limestone is normally mixed with the feed to effectively minimize the sulfur content of the R-N product which usually contains less than 0.05 per cent sulfur, even when relatively high sulfur content ores and fuels are used. In the first part of the test, a locally available limestone with particle size minus 3/16 was used. Later the addition of limestone was discontinued as the amount of lime in the ore itself seemed to be sufficient for sulfur removal.

### Water

Water is used for cooling the kiln bearings, for indirect cooling of the kiln discharge, and for wet grinding, classification, and separation.

### Operation

During the period from August 1 to October 9, 1960, two tests were made in the R-N pilot plant on 5,285.60 net tons of Clear Hills ore, supplied by Premier Steel Mill Ltd., Edmonton, Alberta, Canada. The first test was made on 3,486.05 net tons of ore from August 1 through September 2, and the second test on 1779.55 net tons of ore from September 17 through October 9.

The purpose of the tests was to demonstrate the amenability of the Peace River ore to treatment by the R-N process under conditions to allow realistic, economic projection to commercial scale. For any ore tested in the pilot plant, a "run-in" period is required to reach the optimum conditions, before a so-called "steady state" period can be established, during which complete operational data and heat and material balances are obtained as a basis for the commercial projection.

In the first test run, the ore "as received" was crushed to 5/8 inch prior to feeding to the kiln. The amount of fines in the ore "as received" and as created by crushing and handling was considerable - at times, the ore, as fed to the kiln, contained as much as 70 per cent minus 6 mesh. As the amount of fines varied, it was difficult to maintain an even load in the kiln. Besides, the presence of the fines necessitated a very close temperature control on the kiln in order to avoid sintering when operating so close to the softening temperature of the ore fines. Although sintering deposits on the walls were not formed, considerable difficulties were experienced with loose sinter deposits of fine particles on the thermocouple protection tubes requiring constant exchange and cleaning of thermocouples. In the latter part of the test run, some of the ore fines



were added in form of pellets and the situation improved considerably. Thus, during the last three days of the run, no thermocouple had to be removed for cleaning or exchange.

The various terms employed in the process are defined as follows:

% Metallization	$\frac{\% \text{ Metallic iron in material} \times 100}{\% \text{ Total iron in material}}$
Actual iron recovery	$\frac{\text{Total iron in concentrates} \times 100}{\text{Total iron into kiln}}$ (less iron in coke, recycle coke, recycle concentrate, collector dust and limestone).
Theoretical iron recovery	$\frac{\text{Total iron out in concentrate} \times 100}{\text{Total iron out less spillage and recycle}}$
Recycle coke ratio	$\frac{\text{Input rate of recycle coke (dry basis)}}{\text{Input rate of fresh coke (dry basis)}}$
Kiln tailings	Overflow material from quench tank, and filtrate from recycle coke centrifuge. See diagram on page 23
1st Stage tailing	Reject washings from 1st stage magnetic separators and filter effluent. See diagram on page 24
2nd Stage tailings	Reject washing from 2nd stage magnetic separator. See diagram on page 25

The average degree of metallization of the kiln discharge was approximately 90 per cent. No attempts were made to obtain a higher degree of metallization than 92 per cent, due to the limited time and ore available for the test. Occasional low metallization values during the test are attributed mainly to uneven feed and uneven sizing of the feed, causing variations in the kiln bed level, retention time, etc. In the latter part of the run, the pellets were fed together with either the fine ore or the coarse ore. The latter alternative seemed to be the better. Feeding the pellets separately from individual bins would have given still better control of the feed and the kiln, and it is expected that a commercial unit would operate in this manner.

In the initial test run, the metallized ore was concentrated by grinding and magnetic separation using all available milling capacity in the concentrating plant. By classification in cyclones, different grades of concentrates could be made; the total iron of the concentrates varied from 85 - 93.8 per cent iron, and iron recoveries varied inversely. However, the overflow of one of the cyclones (No. 3), which went to waste, contained as much as 66 per cent Fe. This material was too fine for further beneficiation by grinding and separation and represented a major loss when put to waste. In the latter part of the test, this product was pelletized together with the dust from the kiln dust collector and refeed to the kiln. By the second passage of this material through the kiln, additional grain growth of the iron was obtained which made the iron separable. Later, as much as possible of the ore fines were incorporated in the pellets, thereby reducing the amount of fines fed as such to the kiln. The sulfur content of the ore and the resultant product were low, indicating the possibility of withdrawing the limestone feed. The phosphorous content of the product was about 0.3 - 0.4 per cent which was considered acceptable for this particular ore.

As further research on the concentration circuit seemed to be needed, four one-thousand-pound samples of concentrate were shipped to the following laboratories for study:

Department of Mines and Technical Surveys, Ottawa  
 Ontario Research Foundation  
 Department of Mining and Metallurgy, University of Alberta  
 Wright Engineering, Vancouver, B.C.

Test II was carried out from September 16 to October 9, 1960. An additional 1800 tons were received and treated in this period, with the procedures evolved towards the close of the previous period. The ore for Test II had been exposed to heavy rains in transit and on the stock pile and presented more feeding problems due to sticking in the bins and passages. However, the last fourteen days of operation were satisfactory and gave an average metallization of 90 per cent. The "steady state" period ran from October 3 through October 7 and is presented as basis for commercial projection.

### Observation

The R-N Corporation welcomes observers representing their clients. The supervisory personnel of the corporation were most cooperative and willing to discuss all aspects of the operation and to answer any questions. Photostatic copies of daily data sheets were made available to all observers. All observers were impressed with the complete openness and objectivity with which the R-N personnel discussed all matters. The following observers were present during various phases of the tests:

Dr. C. S. Samis, Professor of Metallurgy, University of British Columbia, representing Premier Steel Mills, Ltd., Edmonton.  
 Mr. L. R. Cunningham, Premier Steel Mills, Ltd., Edmonton.  
 Mr. John Gregory, Research Council of Alberta, Edmonton.  
 Mr. George Sibakin, Metallurgist, Steel Company of Canada, Ltd. Hamilton.  
 Mr. George Viens, Head, Pyro-metallurgical Section, Department of Mines and Technical Surveys, Ottawa.  
 Dr. John Dartnell, representing Stewarts and Lloyds, England.  
 Mr. G. G. Ridley, representing Stewarts and Lloyds, England.  
 Mr. P. Audette, Steel Company of Canada Ltd., Hamilton.  
 Mr. D. K. Pickett, Department of Mines and Technical Surveys, Ottawa.

## VII. R-N CORPORATION TEST RESULTS

The test results were compiled by the R-N Corporation and presented to Premier Steel Mills in reports entitled:

- (1) Spaulding Pilot Plant Test Report, Premier Steel Mill Ltd., Canada, August 1 to October 9, 1960
- (2) R-N Report No. 179, R-N Commercial Facility Projection of Premier Steel Mills Ltd. Peace River Ore, Cases 1 & 2

### Summary

The "steady state" period from October 3 through October 7 is representative of stable, controlled operations of the pilot plant on this ore. The results and data obtained during

this period will serve as a conservative basis for the commercial scale projection. The detail results and data for this period are presented as follows:

Table 3. Operating Data Summary

Dates	"Steady State" Period Oct. 3 - Oct. 7
Number of days	5
Actual ore feed rate to kiln, including pellets, NTPH dry	3.79
Actual ore feed rate to kiln, new ore only, NTPH dry	3.12
Degree of metallization of ore and pellets discharged from kiln, %	90.86
Total Fe in concentrate (unbriquetted metallic iron product) produced, %	90.00
Recovery of iron to concentrate, (spillage free basis) %	83.17
Gross Fuel (corrected) see page 26	
Million B.t.u./ net natural ton of R-N concentrate	28.250
<u>Analyses of Metallic Concentrate before Briquetting</u>	
Total iron, %	90.00
Metallic iron, %	84.69
SiO <sub>2</sub> , %	3.56
Al <sub>2</sub> O <sub>3</sub> , %	2.31
CaO, %	0.76
P, %	0.29
S, %	0.026
C, %	0.25

### VIII. GENERAL CONCLUSIONS

The Clear Hills iron ore is amenable to reduction and recovery by the R-N process. The R-N kiln reduction of ores of this type is practical and the results on Clear Hills ore conform closely to results obtained during an extended trial with Big Seam ore at Birmingham, Alabama.

The engineering principles involved are sound, and this technique appears to represent the best available process for the treatment of this particular ore under existing economic circumstances.

1. A high quality metallic iron concentrate was produced during the "steady state" period, with the following average quality:

Total iron, %	90.00
Metallic iron, %	84.69
SiO <sub>2</sub> , %	3.56
Al <sub>2</sub> O <sub>3</sub> , %	2.31
CaO, %	0.76
P, %	0.29
S, %	0.026
C, %	0.25

Table 4.

R-N SPAULDING PILOT PLANT

Peace River Ore

2nd Test - Steady State

Oct. 3 - Oct. 7, 1960

Metallization 90.86% Recovery 74.22% actual, 83.17% corrected

<u>Item</u>	<u>Natural Tons</u>	<u>% Water</u>	<u>Dry Tons</u>	<u>% Iron</u>	<u>Tons Iron</u>	<u>% Silica</u>	<u>Tons Silica</u>	<u>% Carbon</u>	<u>Tons Carbon</u>	<u>% Ash</u>	<u>% Met.</u>	<u>% Al<sub>2</sub>O<sub>3</sub></u>	<u>% CaO</u>	<u>% P</u>
<u>RAW MATERIALS</u>														
Ore (plus Pellets)	527.89	14.51	451.28	40.17	185.28	18.50	83.48	1.65	7.43	-	-	7.45*	3.66*	0.69*
Fresh Coke	110.35	13.45	95.51	3.53	3.37	9.23	8.82	76.34	72.91	21.04	-	-	-	-
Recycle Coke	177.91	19.53	143.17	5.35	7.66	9.79	14.01	73.53	105.27	25.52	34.77	-	-	-
Limestone	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL IN	816.15	15.46	689.96	28.45	196.31	15.41	106.31	26.90	185.61	-	-	-	-	-
<u>CONCENTRATE</u>														
Final Concentrate	121.21	7.84	111.71	90.00	100.54	3.56	3.98	0.25	0.28	-	94.10	2.31	0.76	0.29
Special Sample	4.05	37.45	2.53	50.43	1.28	26.28	0.66	2.19	0.06	-	-	-	-	-
Filter Cake to Pellets	92.52	15.38	78.29	62.00	48.54	14.65	11.47	3.46	2.71	-	76.74	-	-	-
<u>WASTES</u>														
First Stage Tailings	5420.00	97.01	162.12	21.18	34.34	39.68	64.32	7.78	12.62	-	46.51	-	-	-
Second Stage Tailings	5001.00	99.43	28.35	18.10	5.13	39.26	11.13	0.53	0.15	-	28.86	-	-	-
Kiln Tailings	6329.00	99.73	17.09	19.78	3.38	19.02	3.25	38.15	6.52	-	36.13	-	-	-
Stack Loss (Est.)	9.55	-	9.55	23.88	2.30	15.71	1.50	23.12	2.20	-	-	-	-	-
Spillage	1.22	13.94	1.05	50.48	0.53	10.48	0.11	4.76	0.05	-	-	-	-	-
Total Waste	-	-	218.16	20.94	45.68	36.81	80.31	9.87	21.54	-	-	-	-	-
Recycle Coke	177.91	19.53	143.17	5.35	7.66	9.79	14.01	73.53	105.27	25.52	34.77	-	-	-
TOTAL OUT	-	-	553.86	-	203.70	-	110.43	-	129.86	-	-	-	-	-

<u>Gas MCF</u>	<u>Power KWH</u>	<u>Recycle Coke Ratio</u>	<u>Kiln Gas - Average Analysis - % By Volume</u>				
			<u>CO</u>	<u>CO<sub>2</sub></u>	<u>CH<sub>4</sub></u>	<u>O<sub>2</sub></u>	<u>H<sub>2</sub></u>
2119	116,000	1.50	3.0	15.5	1.9	0.7	1.2

KILN TEMPERATURES, DEGREES F.

	<u>Kiln Speed MPR</u>											<u>F.E.H.</u>	<u>Stack</u>	<u>Kiln Disc.</u>	<u>Cooler Disc.</u>	<u>WEATHER</u>	
		<u>#1</u>	<u>#2</u>	<u>#3</u>	<u>#3A</u>	<u>#4</u>	<u>#5</u>	<u>#6</u>	<u>#7</u>	<u>#8</u>	<u>#9</u>					<u>Avg. Temp. °F</u>	<u>Avg. Hum. %</u>
Avg. Gas	5:39	1885	1964	1976	1962	1960	1943	1936	1915	1910	-	904	507	1169	-	73	82
Avg. Bed	-	-	1907	1895	1886	1880	1869	1864	1817	1568	-	-	-	-	288		

\*For ore included only - for composition of ore plus pellet feed, see following page.

Table 5.

R-N SPAULDING PILOT PLANTFeed Ore CompositionSteady State PeriodOct. 3 - Oct. 7, 1960

The composition of the ore plus pellets shown on the summary for the steady-state period is given on the following table.

<u>Item</u>	<u>Natural Tons</u>	<u>% Water</u>	<u>Dry Tons</u>	<u>% Iron</u>	<u>Tons Iron</u>	<u>% Silica</u>	<u>Tons Silica</u>	<u>% Carbon</u>	<u>Tons Carbon</u>	<u>% Ash</u>	<u>% Met.</u>	<u>% Al<sub>2</sub>O<sub>3</sub></u>	<u>% CaO</u>	<u>% P</u>
1. Filter Cake Pellets	92.52	15.38	78.29	62.00	48.54	14.65	11.47	3.46	2.71	-	76.74	-	-	-
2. Fine Ore Pellets	21.49	17.05	17.83	37.50	6.69	19.02	3.39	1.10	0.20	-	-	-	-	-
3. Bentonite Pellets	1.45	-	1.45	1.14	0.02	52.08	0.76	2.50	0.04	-	-	-	-	-
4. Pellets	99.70	2.14	97.57	56.63	55.25	16.01	15.62	3.02	2.95	-	63.67	-	-	-
5. Fine Ore to Kiln	226.52	17.00	188.02	37.27	70.08	19.01	35.74	1.30	2.44	-	-	7.47	3.41	0.59
6. Coarse Ore to Kiln	201.67	17.84	165.69	36.18	59.95	19.39	32.12	1.23	2.04	-	-	7.42	3.96	0.80
7. Total Ore Feed including Pellets	527.89	14.51	451.28	40.17	185.28	18.50	83.48	1.65	7.43	-	-	-	-	-
8. Ore Feed to Kiln	-	-	371.54	36.80	136.72	19.18	71.25	1.26	4.68	-	-	7.45	3.66	0.69

## NOTES:

Note A. Pellets (4) equals the sum of Items 1, 2 and 3. Most of the moisture contained in the filter cake and the fine ore to pellets have been removed when drying the pellets.

Note B. Total Ore Feed (7) equals the sum of Items 4, 5 and 6. This is the value shown on the steady state summary.

Note C. Ore Feed (8) equals the sum of Items 2, 5 and 6 and is therefore, the total new ore fed to the kiln. The natural tons of new ore used is 449.68 tons at an average moisture content of 17.38%. The natural tons of ore and average moisture of the new ore to the kiln is somewhat less because of the pellet drying.

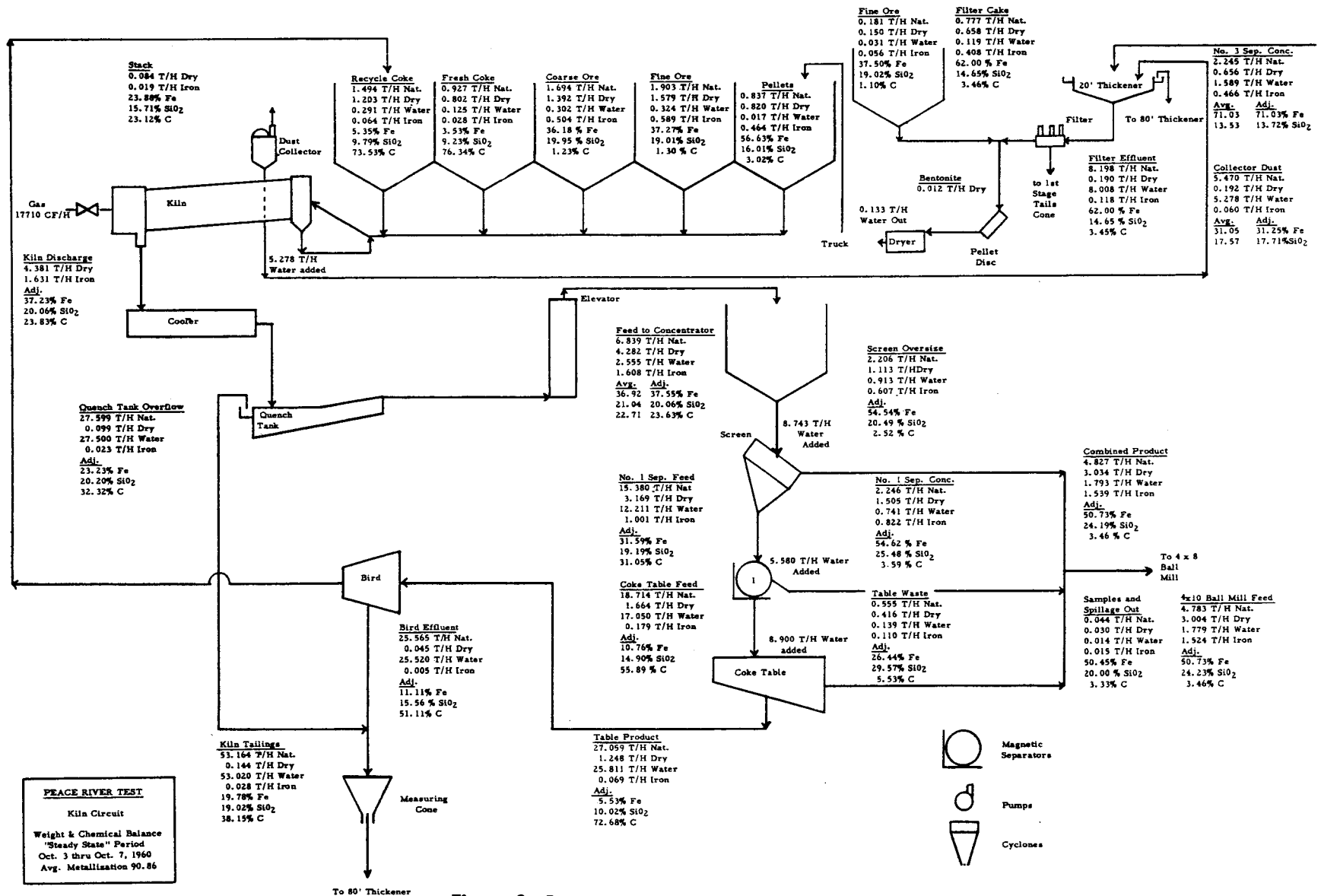


Figure 2. Peace River test - kiln circuit

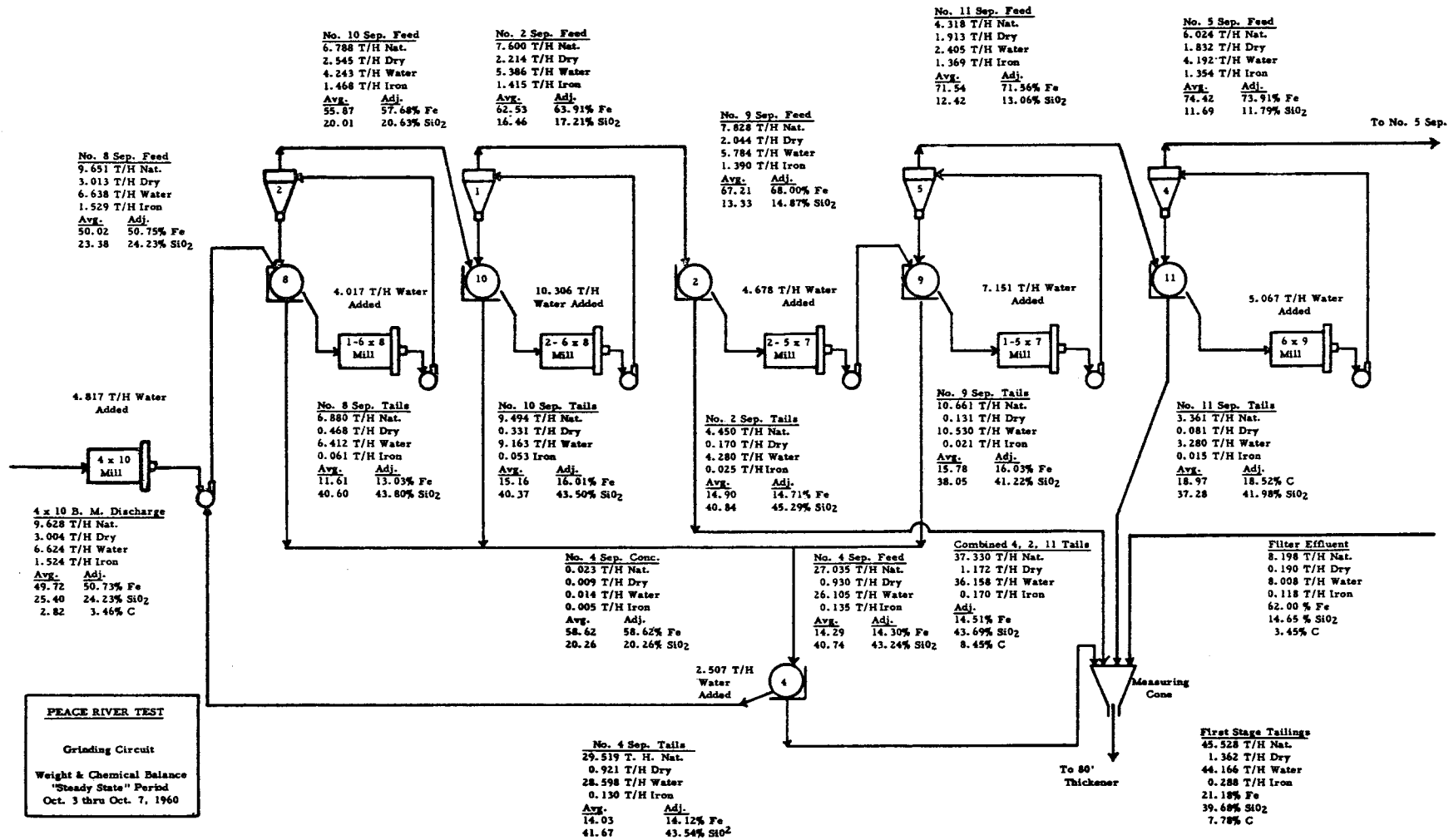


Figure 3. Peace River test - graining circuit

No. 5 Sep. Feed  
 6.024 T/H Nat.  
 1.832 T/H Dry  
 4.192 T/H Water  
 1.354 T/H Iron  
Avg. Adj.  
 74.42 73.91% Fe  
 11.69 11.79% SiO<sub>2</sub>

Cyclone Feed  
 3.338 T/H Nat.  
 1.706 T/H Dry  
 1.632 T/H Water  
 1.336 T/H Iron  
Avg. Adj.  
 78.64 78.31% Fe  
 9.27 9.49% SiO<sub>2</sub>

Cyclone Effluent  
 4.289 T/H Nat.  
 1.352 T/H Dry  
 2.937 T/H Water  
 1.019 T/H Iron  
Avg. Adj.  
 75.02 75.37% Fe  
 10.04 11.02% SiO<sub>2</sub>

No. 3 Sep. Feed  
 6.904 T/H Nat.  
 0.756 T/H Dry  
 6.148 T/H Water  
 0.486 T/H Iron  
Avg. Adj.  
 64.58 64.29% Fe  
 15.41 16.66% SiO<sub>2</sub>

No. 3 Sep. Conc.  
 2.245 T/H Nat.  
 0.656 T/H Dry  
 1.589 T/H Water  
 0.466 T/H Iron  
Avg. Adj.  
 71.03 71.03% Fe  
 13.53 13.72% SiO<sub>2</sub>

No. 5 Sep. Tails  
 9.130 T/H Nat.  
 0.126 T/H Dry  
 9.004 T/H Water  
 0.018 T/H Iron  
Avg. Adj.  
 19.17 14.28% Fe  
 36.91 42.86% SiO<sub>2</sub>

No. 6 Sep. Feed  
 1.760 T/H Nat.  
 0.950 T/H Dry  
 0.810 T/H Water  
 0.850 T/H Iron  
Avg. Adj.  
 89.05 89.47% Fe  
 3.72 3.79% SiO<sub>2</sub>

Combined 6 & 7 Sep. Tails  
 8.571 T/H Nat.  
 0.012 T/H Dry  
 8.557 T/H Water  
 0.005 T/H Iron  
Avg. Adj.  
 39.81 39.81% Fe  
 24.31 24.31 SiO<sub>2</sub>

Cyclone Conc.  
 0.760 T/H Nat.  
 0.354 T/H Dry  
 0.406 T/H Water  
 0.317 T/H Iron  
Avg. Adj.  
 89.11 89.54% Fe  
 3.20 3.67% SiO<sub>2</sub>

No. 3 Cyclone Conc.  
 1.000 T/H Nat.  
 0.596 T/H Dry  
 0.404 T/H Water  
 0.533 T/H Iron  
Avg. Adj.  
 89.00 89.43% Fe  
 3.95 3.86% SiO<sub>2</sub>

No. 3 Sep. Tails  
 13.514 T/H Nat.  
 0.100 T/H Dry  
 13.44 T/H Water  
 0.020 T/H Iron  
Avg. Adj.  
 20.03 20.03% Fe  
 36.10 36.10% SiO<sub>2</sub>

No. 7 Sep. Conc.  
 2.626 T/H Nat.  
 0.938 T/H Dry  
 1.688 T/H Water  
 0.845 T/H Iron

Concentrate  
 1.018 T/H Nat.  
 0.938 T/H Dry  
 0.080 T/H Water  
 0.845 T/H Iron  
 90.00 % Fe  
 3.56 % SiO<sub>2</sub>  
 0.25 % C

**PEACE RIVER TEST**  
 Finishing Circuit  
 Weight & Chemical Balance  
 "Steady State" Period  
 Oct. 3 thru Oct. 7, 1960

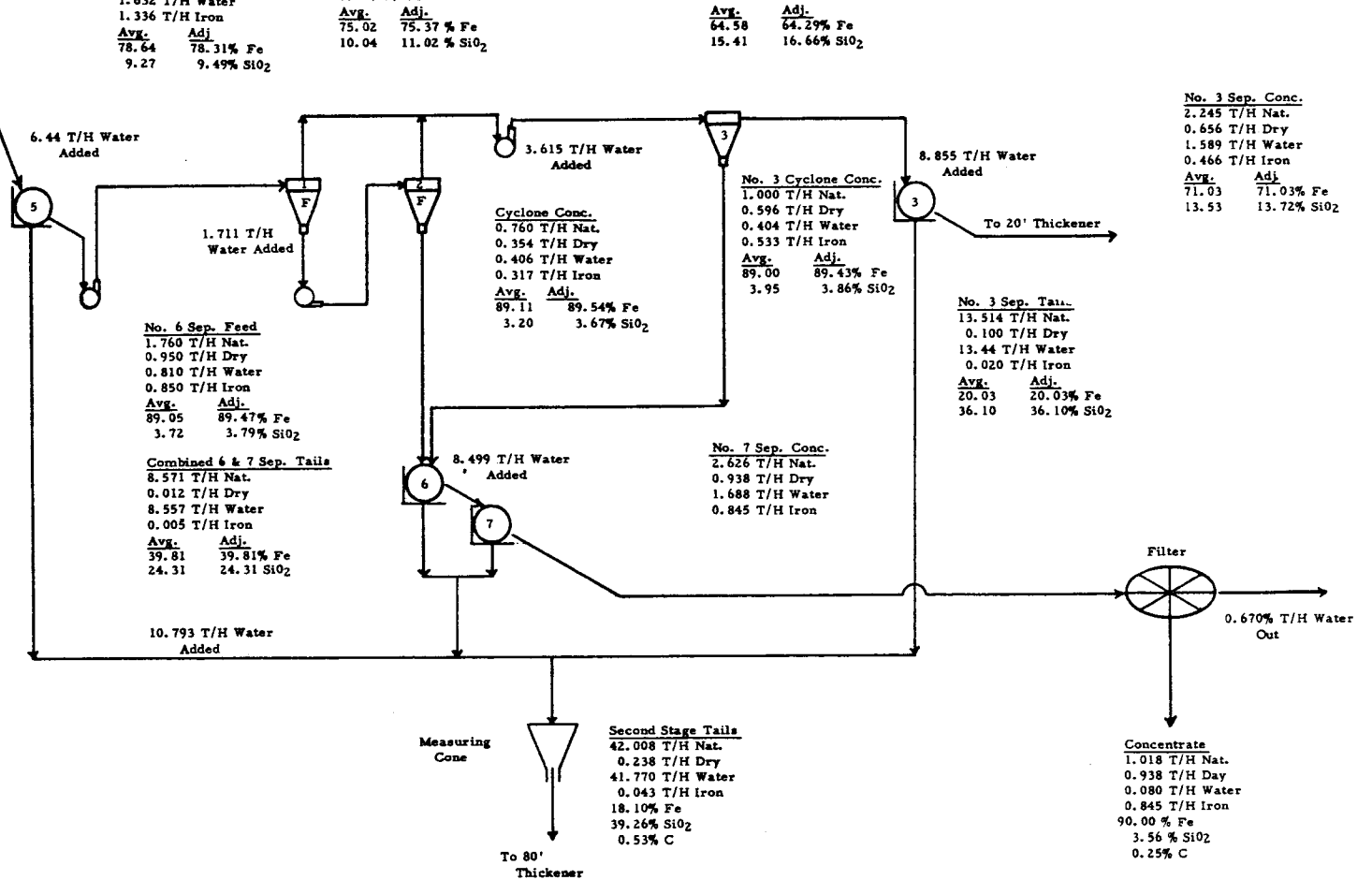


Figure 4. Peace River test - finishing circuit



During the first phase of the test, it was shown that the grade can be increased above 92 per cent total iron at some sacrifice in recovery. Test results indicate that the product may be further improved by the substitution of hindered settling equipment in the place of cyclones in the concentrating circuit.

2. The actual recovery of iron in the process during the five-day "steady state" was 74.22 per cent. The recovery figure adjusted for recoverable measured spillage samples and filtrate loss is 83.17 per cent. Due to defective equipment, the pilot plant experienced a loss of iron values of approximately 8 per cent at the filter ahead of the balling step during the "steady state" period. This loss was measured, and, while the cause could not be corrected at the time, it is wholly avoidable by normal metallurgical procedures. The average degree of metallization of the ore discharged from the kiln during the same period was 90.86 per cent.

In order to consistently obtain the highest possible degree of metallization, it is recommended that essentially all of the fines associated with the crushed ore be removed and balled. These balls should then be heat-hardened on a grate ahead of the R-N kiln by the use of the kiln waste gases. This should permit some increase in the average metallization. The improvement in recovery from this factor is expected to at least equal the increase in metallization. Other benefits are expected from pelletizing the fine ore including: (a) less sensitive operation due to the absence of fines in the kiln, (b) some expected iron grain growth, and (c) decreased concentrating circuit reoxidation of the metallic iron particles. A short test with the hindered settling equipment indicated that the iron recovery can be expected to be substantially improved by substituting this type of equipment for cyclones. These projected recovery improvements expected over the 83.17 per cent realized indicate a figure of 85 per cent to be a logical basis for commercial projection.

3. A heat balance is presented below for the pilot plant "steady state" period projected to correct for certain known deficiencies in equipment and operating procedures shown in the footnotes following table 6. A second heat balance is included to show these data projected to commercial practice.

Table 6. Heat balance

	Projected "steady state" Mil. B.t.u. per Nat. Ton R-N Concentrate	Commercial Projection Mil. B.t.u. per Nat. Ton R-N Concentrate
Reduction of iron	7.542	7.077
Sensible heat, kiln discharge	4.186	3.721
Heat to water in exhaust	4.040 (1)	3.592
Heat available in unburned gases	3.456 (2)	3.072
Radiation and unaccounted for	<u>3.061</u>	<u>2.721</u>
Total out (net)	26.250	23.708
Physical carbon losses	<u>2.000 (3)</u>	<u>1.778</u>
Total (gross)	28.250	25.486
Fuel gas	15.066 (4)	13.608 (4)
Coke burned	11.167 (4)	10.085 (4)
Sensible heats	<u>0.017</u>	<u>0.015</u>
	26.250	23.708

Footnotes:

- (1) The moisture content of the ore, as processed during the test, was higher than expected for commercial conditions.
- (2) This lower value of heat in the unburned gases is approximately one million B.t.u. per ton of concentrate more than was actually obtained from the heat balance calculations of the period August 3 to August 10 and is equivalent to 3.5 per cent combustibles in the exhaust gas. The per cent combustibles for this period was just over 2 per cent as compared to over 6 per cent for the "steady state" period. During the "steady state" period no particular effort was made to reduce the combustibles in the exhaust gas since the reduction of iron was of prime importance.
- (3) During the latter part of the test it was necessary to borrow a magnetic separator from the location in the circuit where the initial separation is made. Single separation instead of the usual double separation was thus employed. Consequently, more carbon was carried into the milling circuit and thus lost in the first stage tailings. These units of carbon would have normally been reclaimed with the recycle coke. This condition increased the physical carbon loss. It is estimated that the pelletizing of all the fine ore will further reduce the physical carbon loss to approximately 2.0 million B.t.u. per ton of concentrate.
- (4) The heat input needed to balance with the reduced heat requirements (output) is made up with coke and gas at the same ratio as actually used during the "steady state" period.

Iron Recovery CalculationsActual Iron Recovery

The actual iron recovery, as determined for the daily and period summaries in this report, are based on the amount of iron in the final concentrate yielded from the amount of iron in the ore only. The calculation is made as follows:

$$\frac{\text{Tons of Fe in Concentrate}}{\text{Tons Fe in Ore Including Pellets*} - \text{Tons Fe in Filter Cake*}} \times 100 = \% \text{ Fe Recovery}$$

\*Pellets included ore and filter cake. The filter cake is composed of kiln collector dust and recirculated milling circuit material, and therefore does not contribute new units of iron.

For this "steady state" period this method of calculation gives:

$$\frac{100.54}{185.28 - 48.54 - 1.28*} \times 100 = 74.22 \%$$

\*Units of iron removed from circuit for special 4 x 10 ball feed sample.

Spillage Free Recovery

Another method for expressing iron recovery is by assuming a spillage free operation. This value is practical in commercial projection since large scale continuous operations would have

little, if any, spillage removed from the circuit. Calculation for spill-free recovery is the same as the actual recovery with the units of iron in the spillage also subtracted from the tons of iron in the ore. The iron units loss at the disc filter in the pelletizing circuit is an avoidable loss and is therefore considered a spillage. The calculation is made as follows:

Tons of Fe in Concentrate				x 100 = % Fe Recovery
Tons Fe in Ore Including Pellets	— in — in Filter Cake	Tons Fe in in Special Sample	Sum of Tons Fe in all Spillage	

For the "steady state" period the calculations would be:

$$\frac{100.54}{185.28 - 48.54 - 1.28 - (0.53 + 14.04^*)} = 83.17 \%$$

\*The tons of iron loss at the disc filter are obtained from the kiln circuit material balance labelled filter effluent. The value for these iron units loss by way of the first stage tailings is 0.118 tons of iron per hour. The balance is based on a period of 119 hours. The 119 hours multiplied by 0.118 tons of iron per hour gives the 14.04 tons of iron loss.

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Plate 1. R-N briquettes

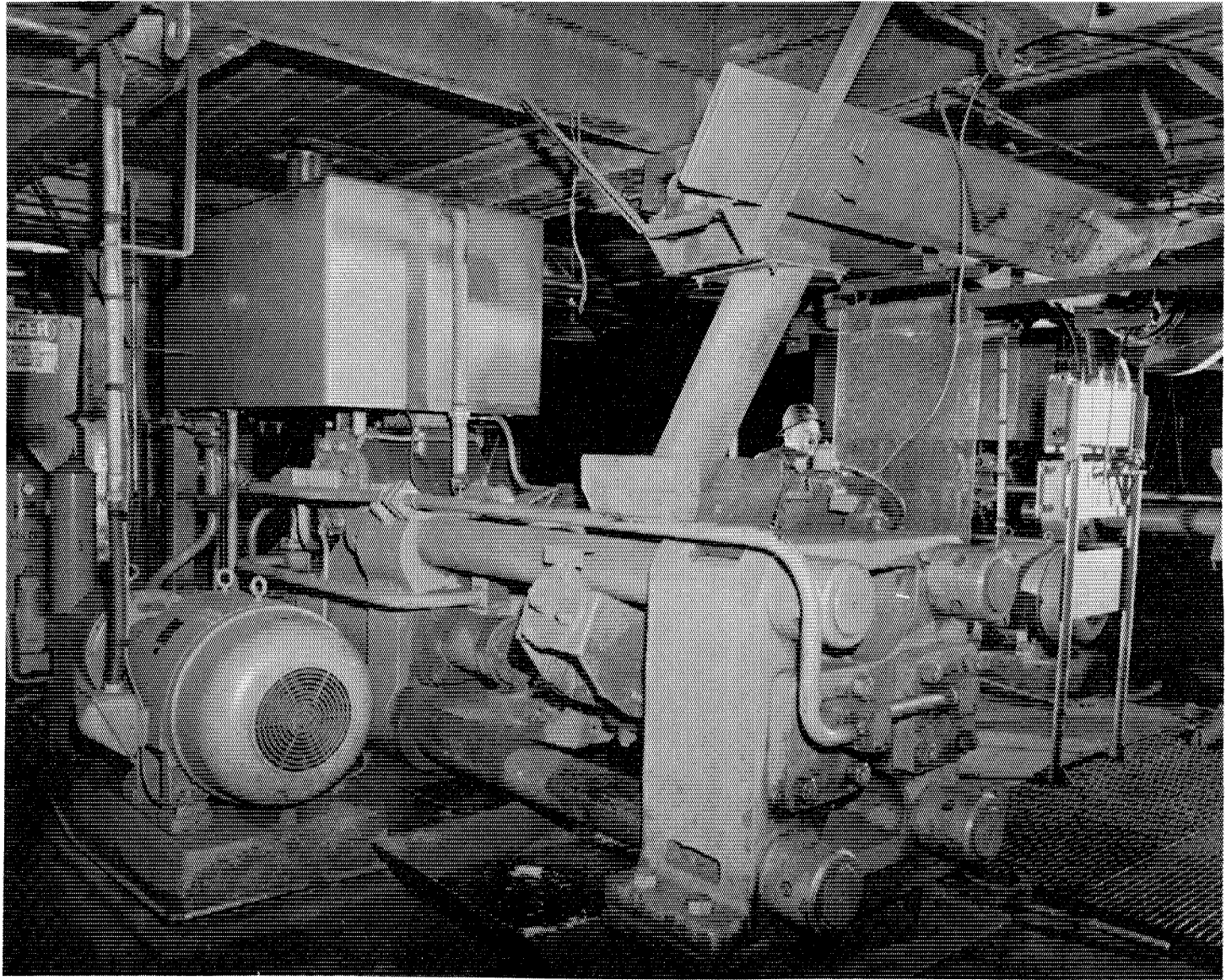


Plate 2. R-N concentrate briquetting press

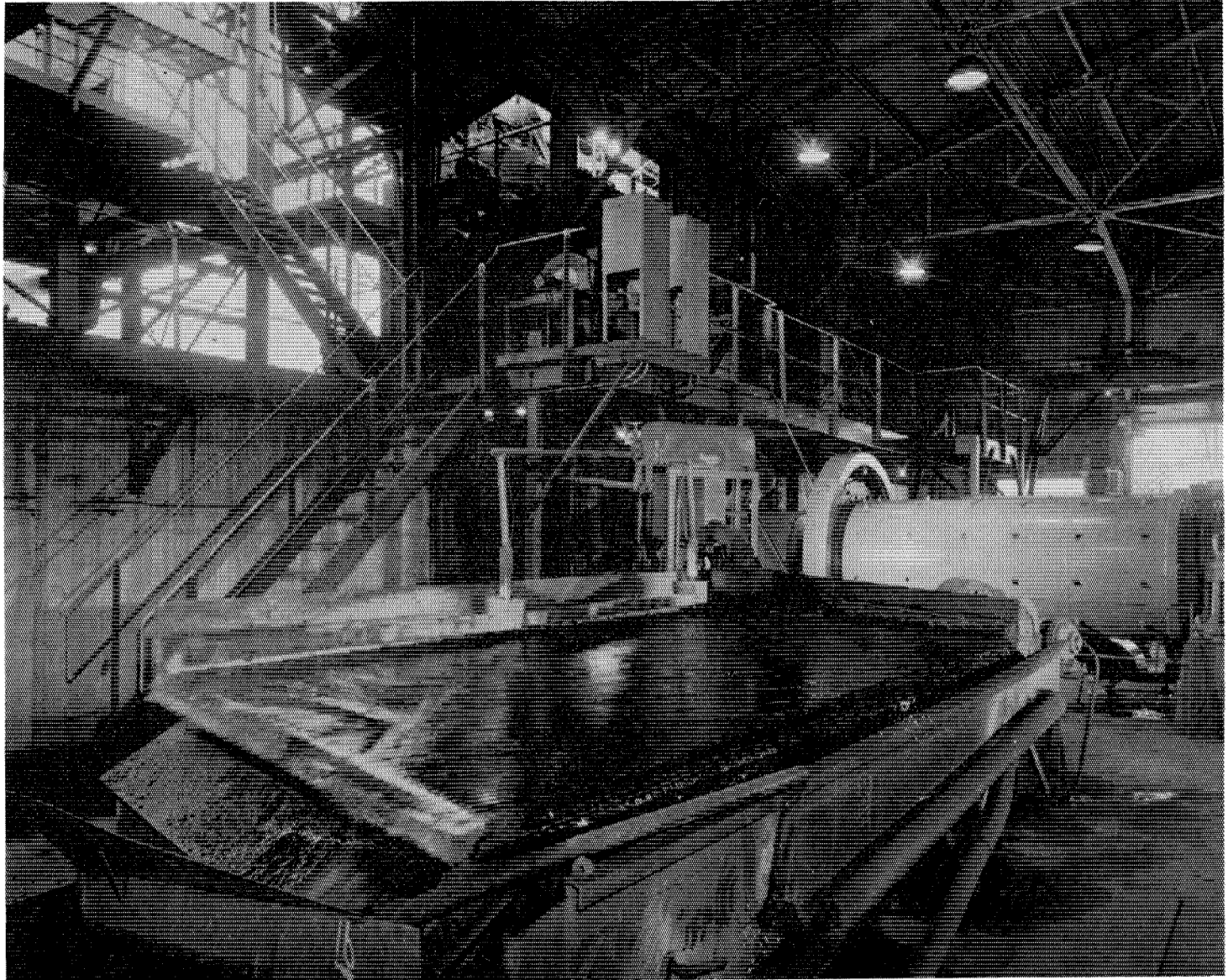


Plate 3. Excess coke recovery table



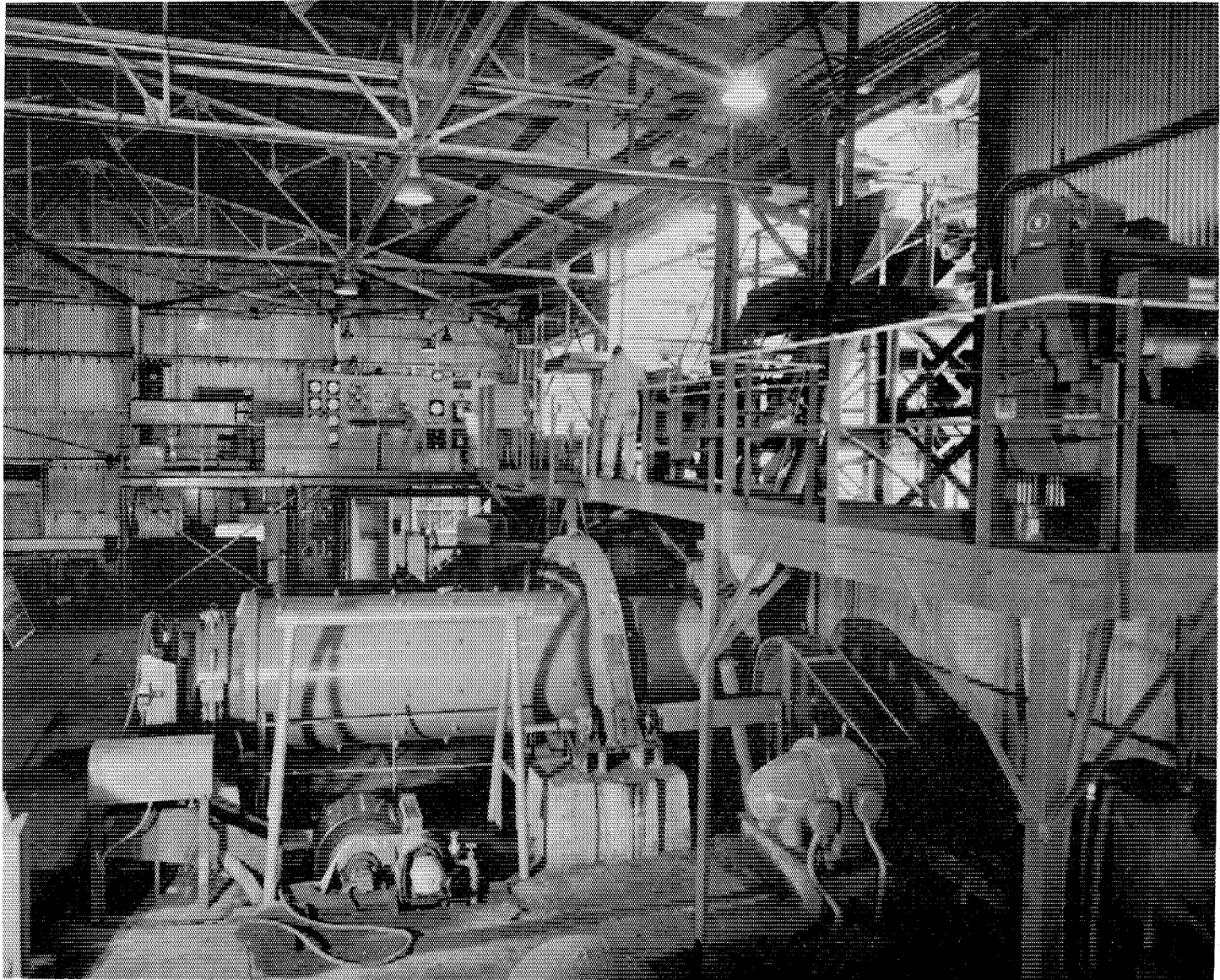


Plate 4. Process control panel - ball mill - magnetic separators



Plate 5. Recycle coke, ore, coke and limestone kiln feed hoppers



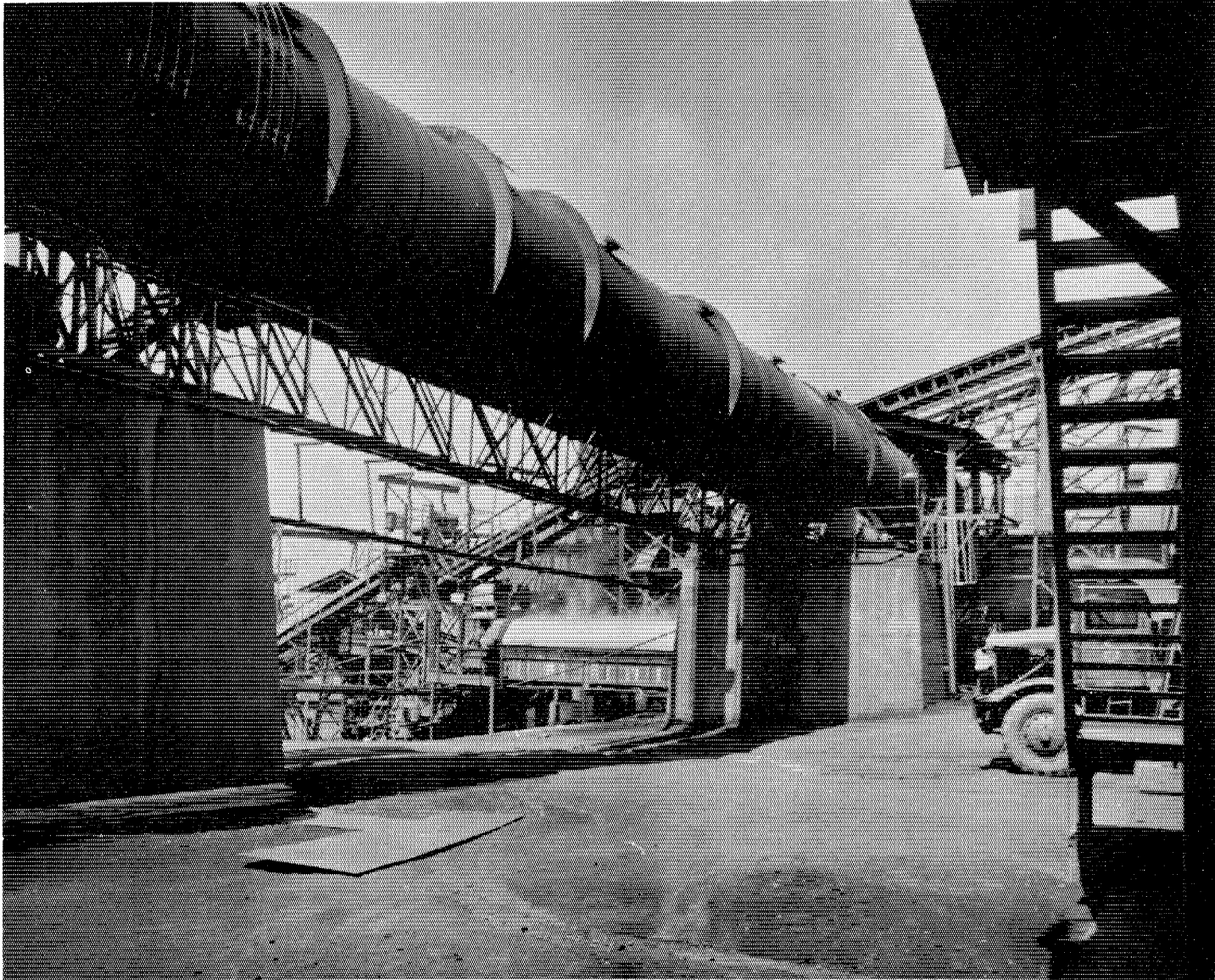


Plate 6. Rotary kiln looking toward discharge end of rotary cooler

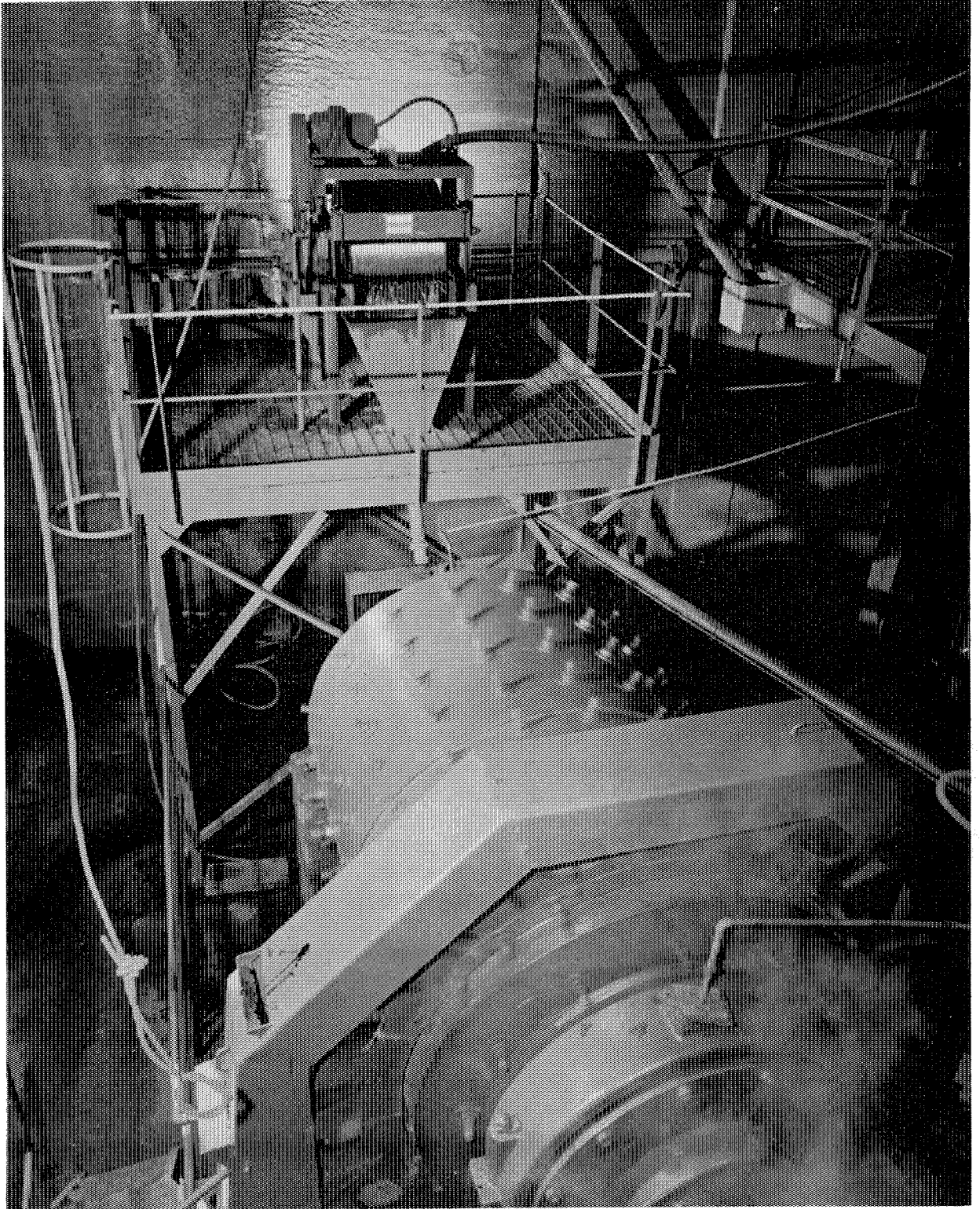


Plate 7. Ball mill and magnetic separator