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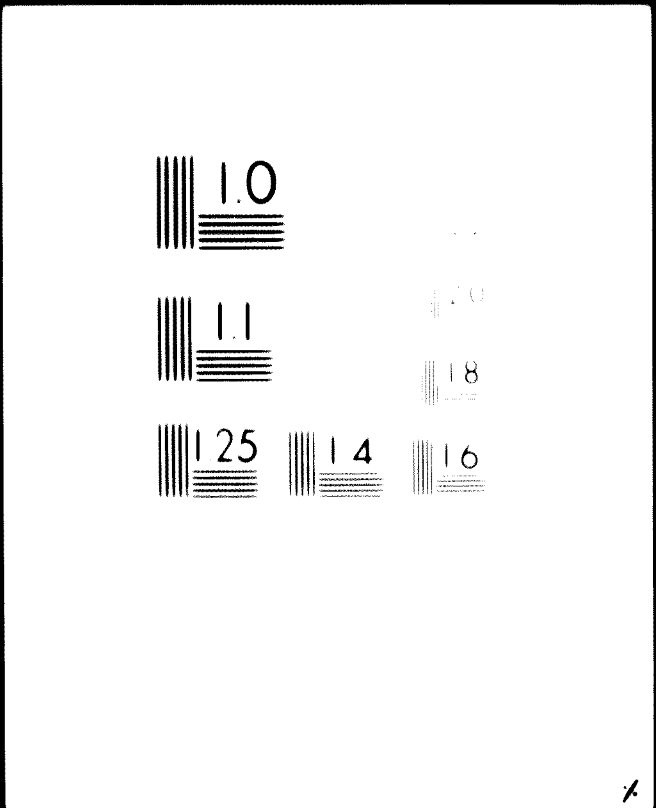

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LABORATORY AND PILOT PLANT SCALE STUDIES
ON
THE ASWAN IRON ORE
FROM
THE ARAB REPUBLIC OF EGYPT

NATIONAL METALLURGICAL LABORATORY
COUNCIL OF SCIENTIFIC & INDUSTRIAL RESEARCH
JAMSHEDPUR 7, INDIA

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JAMSHEDPUR, INDIA

MARCH 1972

A PROJECT

SPONSORED BY THE UNIDO

UNDER CONTRACT NO. 70/26

CONTENTS

PREAMBLE

The National Metallurgical Laboratory, Jamshedpur (NML), entered into a contract with the United Nations Industrial Development Organisation (UNIDO) (Contract No. 70/26) to undertake laboratory and pilot plant scale studies on the suitability of Aswan Iron Ores, from the Arab Republic of Egypt, for iron making. The terms of reference of the contract are given in Appendix I. Results of Laboratory scale studies on the ores were submitted to UNIDO in June 1971, in partial fulfilment of the contract, prior to taking up studies on bulk samples on a pilot plant scale.

The present final report, submitted as per terms of the contract, contains a detailed description of

(a) Petrological and DTA studies.

(b) Laboratory scale trials on the beneficiation of the ores, for the formulation of a flow sheet for optimum and economic upgrading to yield the highest possible iron recovery consistent with high quality concentrate.

(c) Extensive pilot plant scale trials to obtain requisite data for industrial scale implementation of the recommended beneficiation flow sheet, based on laboratory scale trials.

(d) Comprehensive tests on sintering and large scale pelletization of the concentrate, for industrial scale implementation.

(e) Pilot plant scale trials on pre-reduction, of the pellets.

(f) Smelting of pre-reduced pellets in an electric arc furnace for pig iron making.

(g) Techno-economic assessment of each stage of the process, including crushing, grinding, beneficiation, agglomeration, pre-reduction and smelting.

Samples

The following samples were received from Aswan on dates mentioned against each

1.	Lumpy iron ore (15-80 mm)	42 Kg.	on 22nd Sept. 1970
2.	Lumpy iron ore (below 15 mm)	50 Kg.	
3.	Run of mine ore (representative of the bulk sample for pilot plant study)	100 Kg.	on 21st Nov. 1970
4.	Run of mine ore (bulk sample)	194 tonnes	on 1st July 1971
5.	Aswan lime stone	20 tonnes	

Laboratory scale studies were carried out on both samples 1 and 3, for further pilot plant trials the results on sample 3 (run of mine ore), being representative of the bulk (sample 4), were taken into consideration.

Exhaustive petrological and D. T. A. studies, which *inter alia* include observations on megascopic and microscopic characteristics, mineralogical distribution, grain size, interlocking of constituent minerals, etc., were carried out. Details of the observations are furnished in Chapter I of the report.

Chapter II of the report contains details of the bench scale beneficiation studies carried out on samples 1 and 3 outlining data obtained on different methods of beneficiation so as to evolve an appropriate flow-sheet for the optimum recovery associated with maximum iron content in the concentrate. The studies included sieve and chemical analyses of the samples, dry magnetic separation, washing followed by reduction roast, wet magnetic separation, calcination followed by magnetic separation, and flotation tests. Pilot plant trials on beneficiation by roast reduction and wet magnetic separation (which were found to yield the best results during laboratory scale investigations) have also been included in this Chapter.

Results of studies carried out on the physical and physico-chemical properties, including reducibility, of the iron ores, and of the concentrate obtained by beneficiation, decrepitation behaviour of the iron ore and calcination of limestone are presented in Chapter III.

The details of agglomeration studies employing pelletizing and sintering methods on the iron ore concentrate prepared in the pilot plant are dealt with in Chapter IV.

Laboratory scale studies on static bed reduction of the Asswan iron ore, as well as tonnage scale pre reduction data obtained from green, heat-hardened, unfluxed pellets are covered in Chapter V

The data obtained from the smelting of the Asswan iron ore pellets, including pre reduced pellets, in an electric arc furnace, are presented in Chapter VI.

Techno-economic feasibility studies have been carried out for each of the processes, viz. beneficiation, pelletization, pre reduction and electric smelting are set out in Chapter VII.

A summary of the observations and our conclusions on the suitability of the Asswan iron ore for iron-making are given in the following paragraphs

SUMMARY

In order to evolve an economic flow-sheet and to select a process for optimum beneficiation of the oolitic iron ore, agglomeration of the beneficiated product by sintering or pelletization, pre-reduction of pellets and smelting of the pre-reduced charge in a submerged electric arc furnace for pig iron production based on samples of Asswan Iron Ore, and limestone, from the Arab Republic of Egypt, comprehensive laboratory scale and pilot plant scale studies were undertaken at the National Metallurgical Laboratory under a United Nations Industrial Development Organisation contract.

Petrological Studies

Detailed petrographic and D.T.A. studies with samples 1, 3 and 4 revealed that the samples were predominantly lean hematitic hydroxides of definite sedimentary origin formed by colloidal deposition and were siliceous in nature. The hydrated iron oxides were mostly goethite, laterite and limonite and were intimately associated with crypto-crystalline siliceous gangue mainly quartz chert. Bulk of the lumps in the sample were made up of aggregates of oolites of average size of 0.4 mm. They were all scattered and cemented in an intimately admixed matrix.

The oolites were mainly crypto-crystalline with typical oolitic texture showing concentric alternate layers of rings and deposition of hydrated iron oxides viz. goethite, limonite and laterite or their dehydrated product, mainly hematite and some magnetite with intercaletions of silica grains. The nuclei of some oolites were quarts grains and in the others crypto-crystalline siliceous grains. Samples from the run of mine ore also exhibited two distinct varieties of lumps i.e. lumps having reddish brown colour with lesser amounts of oolites and more of mega-crystalline siliceous gangue grains and lumps, pinkish-brown to steel grey in colour, with conchoidal fracture due to the presence of more oolites embedded in a loosely consolidated ferruginous matrix and generally termed as the oolitic variety. This formed the bulk of the sample with specific gravity of the lumps varying from 2.7 to 2.9. None of them were magnetic. It is confirmed that oolites formed about 65 to 80% of the matrix with a size ranging from 1.02 mm to 0.18 mm.

Grain Size and Mineralogical Studies

Detailed petrological studies of the different types of ore samples revealed varied degree of size variation of the siliceous and other non metallic minerals. It has been observed that only the interstitial quartz with an average size range of 0.06 mm to 0.08 mm was expected to be freed at a grind size of about 200 mesh. The other type namely "cherty" quartz which was present as very fine inclusions in the hematite goethite matrix as well as in and around the oolite ring structure was 0.004 mm to 0.01 mm in size and was not likely to be freed unless the whole material was ground to such fineness. These observations were confirmed by the mineral interlocking studies of the ore samples subsequently. These studies confirmed the presence of siliceous gangue as interstitial grains in the ferruginous matrix of the ferruginous sandstone variety as well as in the lumpy oolitic variety. The grains could be freed at about 60 to 80 microns size, which was the average size of the interstitial quartz. The very fine cherty quartz and other siliceous gangue grains inherent in the oolite ring structure were too finely interlocked with the ferruginous components of the oolites due to their colloidal nature of deposition. The separation of the interlocked ferruginous constituents would, therefore, need a very fine grind to free them. It is, therefore, evident that at such a fine size, any attempt of the ore-gangue separation may not be effective to the fullest extent by adopting any of the conventional ore dressing methods.

D.T.A. Studies

The differential thermal analysis studies of the samples carried out on prepared samples of -200 mesh fineness confirmed qualitatively the presence of hydroxides of iron, quartz and hematite. No quantitative estimates were possible due to the very complex nature of the ore resulting from its colloidal origin.

Beneficiation

The bulk sample, as received, analysed Fe - 41.20%, SiO_2 - 17.86%, Al_2O_3 - 7.54%, CaO - 4.04%, MgO - 1.58%, MnO - 0.04%, TiO_2 - 0.30%, S - 0.34%, P - 1.03%, and L.O.I. - 6.20%. The sample was stage crushed to about 18 mm size and subjected to magnetizing reduction roast using a mixture of oil and coal as reductants. The reduced ore was ground in a continuous ball mill to about 85% -

Prof. V. A. Altekar, Director, National Metallurgical Laboratory, inspecting Asswan iron ore drums received for pilot plant scale investigation.

200 mesh size. The ground product after thickening was passed through low intensity wet magnetic separators to yield a magnetic concentrate and a non magnetic tailing. The magnetic concentrate, constituting about 77% by weight, analysed Fe 51.7%, FeO 15.84%, TiO_2 0.34%, MnO Trace, SiO_2 11.3%, Al_2O_3 5.23%, CaO 4.65%, MgO 1.82%, P 1.03%, S 0.16%, L.O.I. 5.11%.

Agglomeration

(a) Pelletizing

The above magnetic concentrate, after thickening filtration and partial drying, was pelletized using 1% bentonite as binder. The green pellets (9-12 mm size) on an average had the compression strength of 1.5 to 1.7 kg per pellet. Controlled drying, pre heating and firing of the pellets yielded heat hardened pellets having strengths varying from 141 to 331 kg per pellet under different conditions. It may be mentioned that the rate of increase of temperature, particularly during drying and pre-heating, had to be controlled very carefully to produce good quality heat hardened pellets. A higher rate of pre-heating resulted in cracking and bursting of pellets or incipient fusion of pellets to form fused mass. Compression strength of heat hardened fluxed pellets (basicity ratio varying from 1.0 to 1.6) was 151 to 156 kg per pellet.

(b) Sintering

Sintering studies of the magnetic concentrate were also conducted. The sinter mix for producing unfluxed sinter of good strength contained 16% water, 20% return fines and 4% coke, whereas for fluxed sinter of basicity ratio 1.8, the mix contained 16% water, 3.25% coke and 20% return fines.

The Physico-chemical Characteristics of the Iron Ore and Limestone

Detailed studies were undertaken on the determination of Bond's work index in closed circuits for the magnetic concentrate from magnetising reduction roast tests. Settling rates of both the magnetic concentrate and the tailings were determined and the rates of filtration of both the magnetic and non-magnetic products were studied. The rate of formation of dry cake in the filtration test was determined and it was found that higher percentage of solids in the pulp, higher was the rate of dry cake formation.

Studies of some physico-chemical properties of the ore samples showed that they were highly porous and reducible, having a low specific gravity. The ores having the size -10+20 mm, -20+30 mm attained over 90% reduction in about 30 minutes. The reduction beyond about 97% was rather sluggish, possibly due to pockets of iron oxide being enveloped by silica.

Calcination of Asswan limestone was studied both at a constant temperature of 900°C and also during continuous heating from [illegible] to 930°C. The amount of CO_2 liberated due to calcination of the limestone by both these processes was found to be almost identical. Due to its high CaO content, the limestone was considered suitable for use as a flux.

Pre-reduction

After preliminary static bed reduction tests for obtaining the process parameters, green and fired pellets were subjected to pre-reduction in a 10.5 metre long and 0.4 metre inside diameter kiln using highly volatile non-coking coal as a solid reductant. All the operating parameters for pre-reduction such as the speed of rotation of the kiln and its inclination, raw material composition and feed rate, temperature profiles inside the kiln, temperature and composition of the exit gases, etc. were recorded. The kiln discharge was subjected to magnetic separation. Pre-reduced pellets of 85% average degree of metallization with 82.4 per cent of iron in the product equivalent to 87.5% Fe recovery with respect to the original sample were obtained.

Smelting in the Electric Arc Furnace

The heat hardened and pre-reduced pellets were smelted in a 500 KVA sub-merged electric arc furnace. The pre-reduced pellets obtained from the rotary kiln were suitably blended to obtain three different lots with average degree of metallization of 60%, 70% and 80%. In the base period, heat hardened pellets were employed and the results compared with those obtained on further trials with pre-reduced pellets. The operating data for the four types of pellets was obtained and the efficiency of smelting was determined. Fuel and power consumption data were determined and it was found that the pre-reduced pellets made from processed Asswan iron ore were quite suitable for electric smelting, a higher degree of pre-reduction being conducive to improved smelting practice, leading to higher yields and greater furnace productivity.

Techno-economic Feasibility

Techno economic assessment has been made for each stage of the process, viz. crushing, grinding, beneficiation, agglomeration (pelletisation), pre-reduction and smelting. The capital cost for equipment and the cost of raw materials, fuel, power and labour have been based on Indian conditions. The actual cost in the Arab Republic of Egypt will depend on these price factors.

CONCLUSIONS

The detailed studies carried out, as summarized above, lead to the following conclusions :-

1. The most economic and optimum method for beneficiation of the Asswan Iron Ore (41% Fe) has been found to be magnetizing reduction roast followed by fine grinding and magnetic separation. The iron content of the concentrate can be raised to only 51-52%, since the siliceous gangue is very intimately associated with the iron minerals and cannot be economically removed completely. However, the overall Fe recovery in the concentrate (constituting nearly 77% by weight of the ore) is as high as 94.7%.

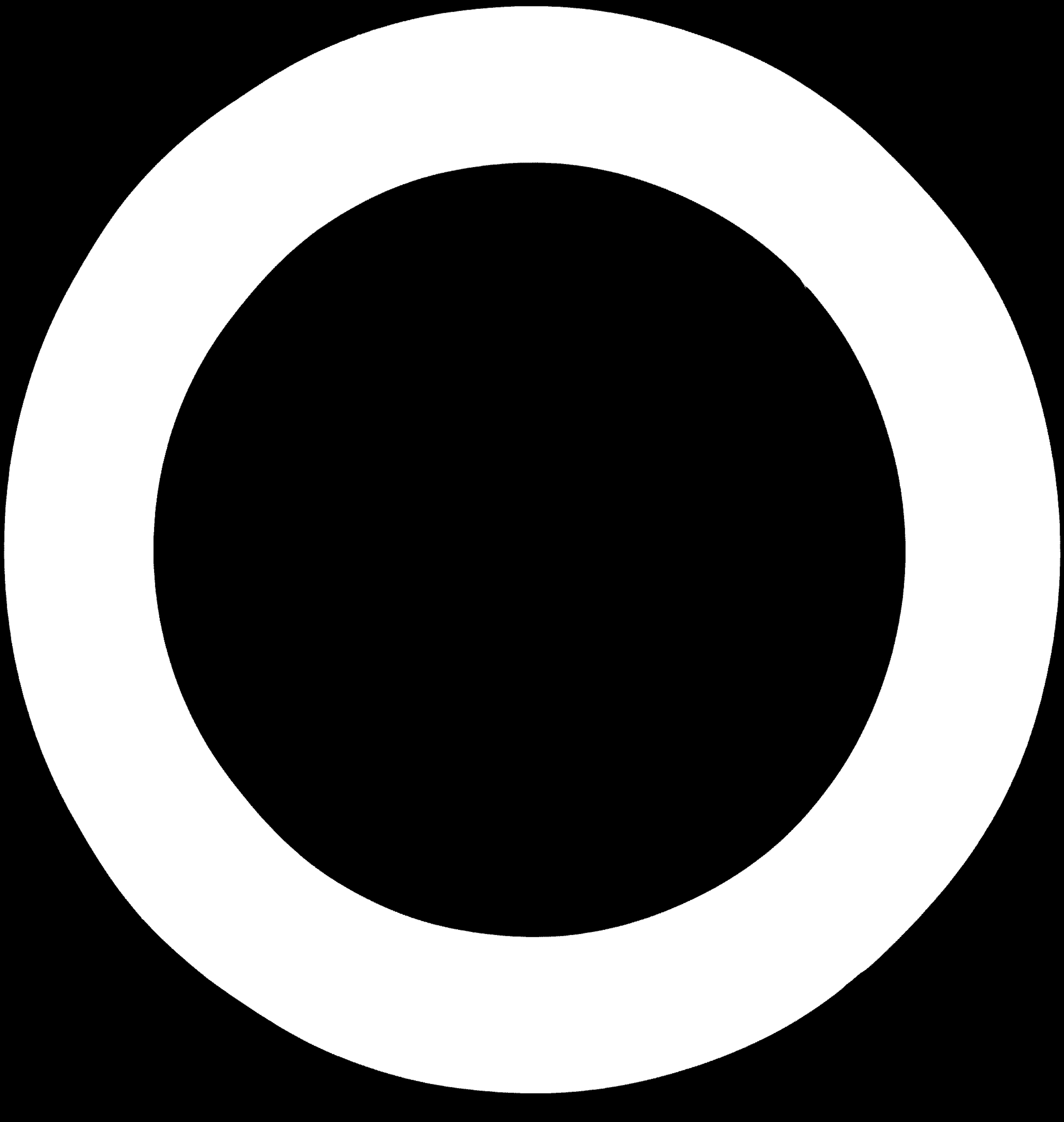
2. For purposes of agglomeration of the ore concentrate, pelletizing is to be preferred to sintering, the sintering rate of the ore being as low as 1.94 to 2.67 cm/min.

3. Pellets made from beneficiated Asswan iron ore, could be pre-reduced to a high degree of metallization in a rotary kiln employing solid reductant.

4. Pre-reduced pellets made from the upgraded Asswan Iron Ore are quite suitable for electric smelting, the higher the degree of pre-reduction, higher the yields of pig iron, lower the fuel and power consumption and higher the furnace productivity. A better quality product, with lower silicon and sulphur contents is obtained from the pre-reduced pellets.

V. A. ALTEKAR
Director
National Metallurgical Laboratory
Jamshedpur
India

March, 1972



CHAPTER 1

PETROLOGICAL AND D.T.A. STUDIES

Detailed petrological and D.T.A. studies were conducted on samples No.I (crushed ore), No.III (run of mine ore) and No.IV (bulk sample) which are lumpy. Sample No.II constituting the 'fines' was not investigated as detailed beneficiation tests were not programmed on this sample.

PETROLOGICAL STUDIES

Megascopic Observations

Sample No. I (crushed ore)

The as received lumps, varied from 50 mm to 100 mm in size. Two varieties could be easily distinguished, namely (a) lumps of brownish grey colour resembling ferruginous sand stone with lesser amount of oolites and comparatively coarse grained siliceous mineral constituents, and (b) lumps of pinkish brown colour with more oolites and lesser amount of coarse grained siliceous mineral constituents. The specific gravity of the different lumps varied from 2.8 to 3.5. Both the varieties were friable and loosely compacted, the former showing comparatively coarser grain size than the latter. They were of dull lustre with uneven fracture and non-magnetic. The oolites, in certain cases, showed exfoliation.

Sample No. III (run of mine ore and representative of 200 tonne gross sample)

The as received sample, consisted of lumps varying from 50 mm to 100 mm in size with some fines. Two distinct varieties of lumps could be distinguished namely, (a) lumps having reddish brown colour, dull lustre with occasional shining portions on the fresh surface, uneven fracture, massive in form, with lesser amounts of oolites and more of mega-crystalline siliceous gangue grains with occasional presence of calcite as observed by a binocular microscope and generally resembling a ferruginous sandstone in nature, and (b) lumps having pinkish-brown to steel grey in colour sub-metallic to metallic in lustre, conchoidal fracture due to the presence of more oolites embedded in a loosely consolidated ferruginous matrix and generally termed as the oolitic variety. The second variety formed the bulk of the sample. The specific gravity of the lumps varied from 2.7 to 2.9. None of them were magnetic.

Microscopic characteristics

Mineralogy and Textural Relationship

Sample No. 1 (Crushed Ore)

Petrological examination of thin as well as polished sections of selected lumps, and different sieve fractions of -10 mesh representative sample of the lumps, revealed that the ore consisted mostly of hydrated iron oxides (goethite, laterite, and limonite) and their dehydrated product, hematite with traces of magnetite and siderite, followed by quartz as the chief gangue mineral, feldspar alternating to kaolin (traces) and minor amounts of muscovite and biotite, calcite, apatite, pyrite, marcasite and chalcopyrite.

Ferruginous Sandstone Variety

Examination of the lumps of the first variety namely the ferruginous sandstone variety, showed under transmitted and reflected illumination that they mainly consisted of allocthonous and autocthonous quartz. The grains were sub-angular to sub-rounded with unequal sizes namely maximum 9.6 mm; minimum 0.001 mm and average 0.06 mm, and the quartz was normal type in association with pressure quartz and unit undulose quartz (Figs. 1.1, 1.6, 1.16 and 1.17). With some exceptions, the grain always showed varying amounts of ferruginous matrix, which can be considered as a gel-like chemically precipitated admixture of hydrated ferruginous oxide minerals some of which appeared to have been later dehydrated. Their chemical gel-like deposition could be inferred by the presence of scattered oolites (Figs. 1.11 and 1.16). These ferruginous constituents were again intimately associated with crypto-crystalline hematite and magnetite in the matrix were of average size 0.02 mm, with specks of pyrite finer than 0.001 mm. The size of cherty fragment varied from maximum 0.2 mm; minimum 0.08 mm and average 0.1 mm respectively. Traces of feldspar were present but due to thick coating, the species could not be identified.

Occasionally, leached out ferruginous materials were found to fill in the intergranular spaces. Common, though not persistent, inclusions of hematite were found in the quartz grains. Their size was maximum 0.03 mm, minimum 0.0002 mm. The size of apatite grains as inclusions mainly in quartz was maximum 0.02 mm, minimum 0.004 mm and average 0.006 mm. Traces of muscovite, biotite and siderite were also present.

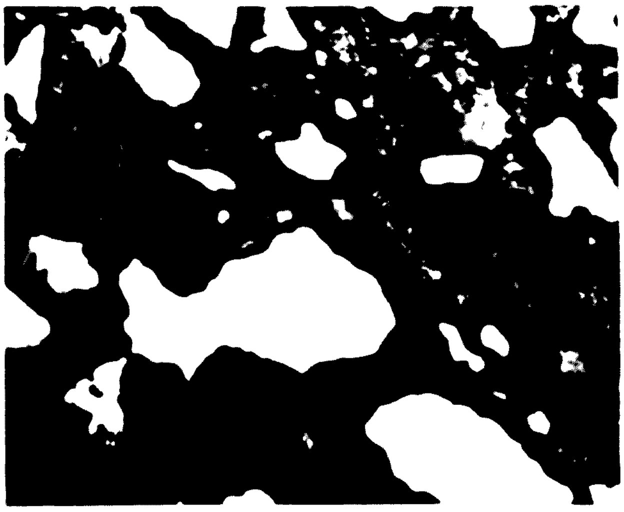


Fig. [illegible] crystalline quartz grains [illegible] (white) in ferruginous matrix (black) [illegible] incident polarised illumination [illegible]

Fig. [illegible] polarised [illegible]

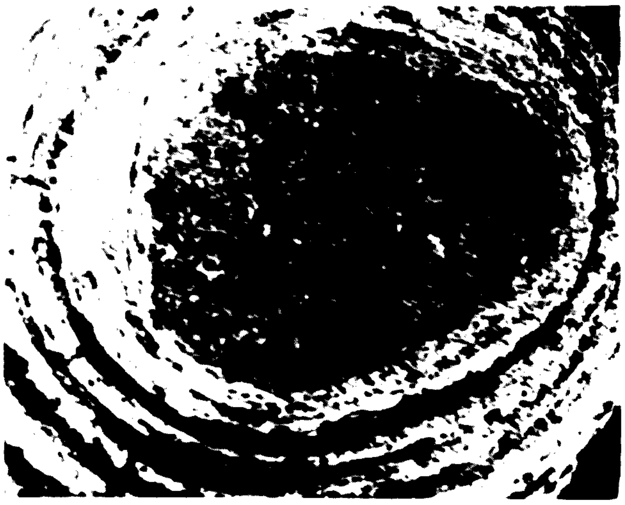


Fig. 1.[illegible] Typical individual oolite consisting mostly of cryptocrystalline fine grained hematite (white) in concentric layers resulted by dehydration of goethite etc. (black) with very fine siliceous grains (dark grey). Incident polarised illumination × 140

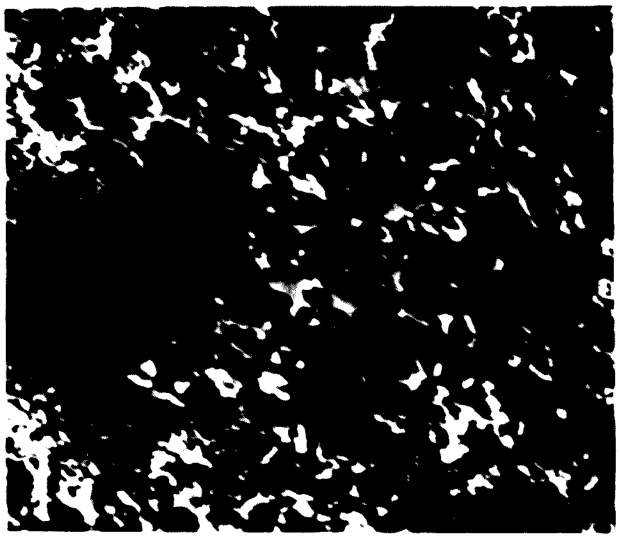


Fig. 1.4 Ferruginous matrix (dark areas) in intimate association with siliceous grains (grey) and hematite (white). Incident illumination × 140

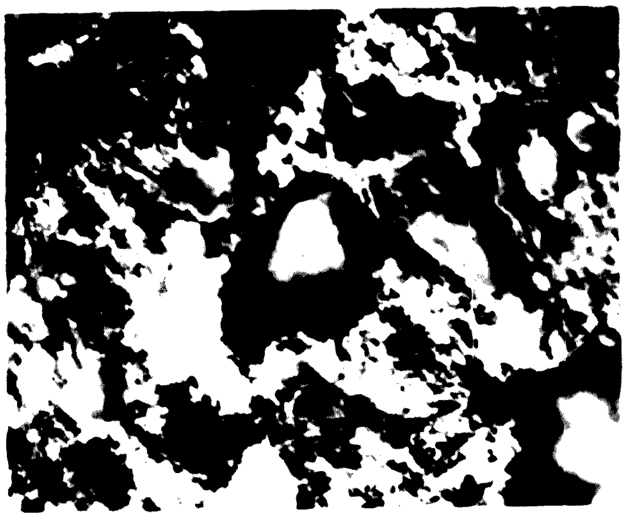


Fig 1.5 Very fine chalcopyrite specks and stringers (bright white) in quartz (grey) and goethite (dark areas). Incident polarised illumination ×420

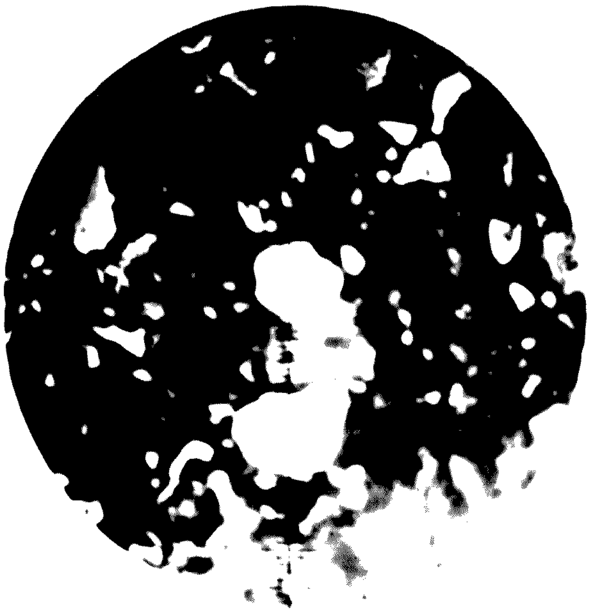


Fig 1.6 A [illegible] showing [illegible] matrix (black) [illegible]

Fig 1.7 A general view of oolites (dark areas) showing quartz grains (greyish white) forming nuclei of oolites, crystalline hematite (white) as interstitial matrix and fine grained silica gel (greyish white) as part of layers of oolites. Incident polarised illumination ×100

Fig 1.8 [illegible] grained [illegible] alternating with [illegible] polarised [illegible]

[illegible]

[illegible]

[illegible]

Fig. 1.13 Oolitic variety. A General field of rounded oolites with fine grained quartz grains (white). Transmitted polarised illumination. Crossed nicols × 26.

Fig. 1.14 Oolitic variety. General field of oolites and siliceous gangue minerals (white). Transmitted polarised illumination. Crossed nicols × 26.

Fig 1.15 Oolitic variety. Close up of oolite showing concentric rings and nucleus filled in with quartz grains (white). Transmitted polarised illumination. 335

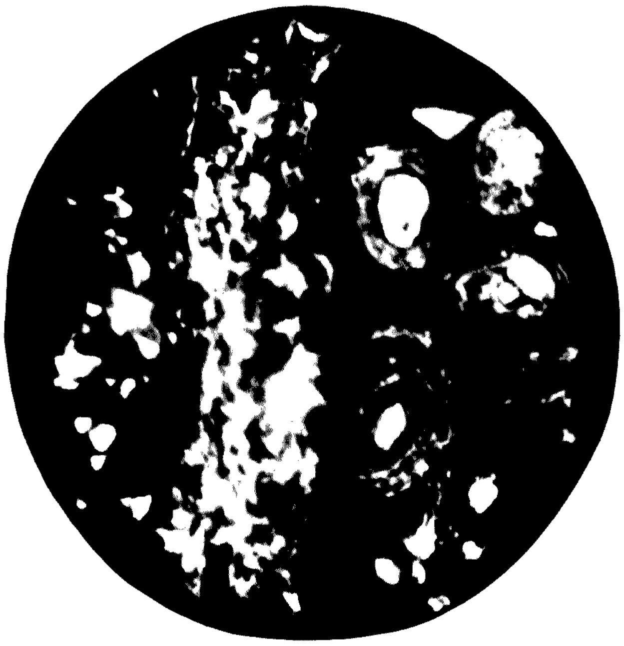


Fig 1.16 Ferruginous sandstone variety. Crystalline quartz grains (white) in a ferruginous matrix with oolites. Transmitted polarised illumination. Crossed nicols. 75

Fig 1.17 Ferruginous sandstone variety. Quartz grains (white) and oolites in a ferruginous matrix. Incident polarised illumination. Inclined nicols × [illegible]

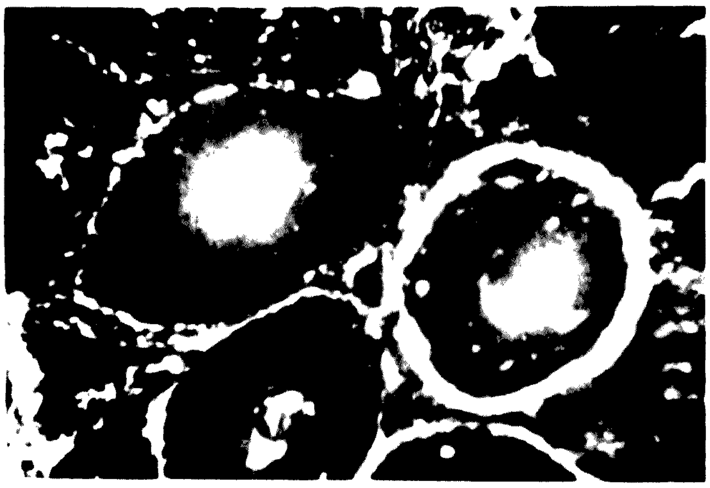


Fig 1.18 Oolitic variety. A view of oolites with quartz grains (white) as nuclei inside them. Transmitted polarised illumination. Crossed nicols × 100

Oolitic Variety

The second variety of the lumps could be considered as the oolitic iron ore. The sizes of the oolites were variable, ranging from maximum 1.01 mm to minimum 0.1 mm diamter with an average of 0.4 mm cemented in an intimately admixed matrix. The oolites and the cementing matrix chiefly composed of an admixture of hydrated and dehydrated ferruginous and siliceous mineral constituents. The siliceous gangue minerals were crypto-crystalline in nature and mostly cherty. Some amount of aluminous or clayey material was also observed intercalated therein.

The oolites were mainly crypto-crystalline with typical oolitic texture showing concentric alternate layering of rings and deposition of hydrated iron oxides, viz. goethite, limonite and laterite or their dehydrated product, mainly hematite and some magnetite with intercalations of silica grains (Figs. 1.3, 1.8 and 1.15). The proportion of the ferruginous and siliceous constituents in the oolites as well as in the cementing matrix could not be ascertained because of their crypto-crystalline nature and very fine interlocking due to colloidal precipitation (Fig. 1.4). The size of the siliceous grains present in the ferruginous cementing matrix ranged from 0.15 mm to submicroscopic range with an average size of 0.016 mm. The nuclei of some oolites were quartz grains and in the others crypto-crystalline siliceous grains (Figs. 1.7, 1.12 and 1.15). No fossil fragments as nuclei of the oolites could be detected. Minor amounts of mica, feldspar and apatite were noticed in the siliceous constituents associated with the cementing materials.

The pyrite and marcasite inclusions in the cementing base were quite easily detectable and their sizes were maximum 0.002 mm, minimum 0.0004 and average 0.001 mm.

Sample No. III (Run of Mine Ore)

Petrological studies were conducted under transmitted and reflected illuminations on thin sections and polished sections of different selected specimens, as well as various sieve fractions of -10 mesh representative sample for finding out the mineralogical assemblages, textural relationship, grain size and mineral locking nature in the sample. It was observed that the ore essentially consisted of oxides and hydroxides of iron namely hematite, goethite, laterite, limonite, trace amounts of magnetite and siderite and manganese hydroxides. These ferruginous minerals were followed in order of abundance by quartz (crystalline, cherty and colloidal), feldspars, kaolin and other clayey minerals, muscovite, biotite, chlorite, calcite, apatite, pyrite, marcasite and chalcopyrite.

Ferruginous sandstone variety

A few selected lumps of this variety showed that it mainly consisted of ferruginous minerals forming the matrix in which quartz and other siliceous gangue minerals, 15% - 20% on an average, were embedded along with some occasional oolites. The quartz was both allocthonous and autocthonous (detrial) in nature. The grain size was maximum, minimum and average 0.5 mm, 0.001 mm and 0.06 mm respectively. (Figs. 1.1, 1.6, 1.11, and 1.16).

Some dehydration of hydroxides of iron had resulted in crystalline aggregates of hematite (hydrohematite). The clayey minerals in certain fields appeared as patches, but otherwise dispersed throughout the ferruginous matrix. Calcite was present in minor amounts as grains and stringers (0.16 mm and 0.01 mm on average). Traces of muscovite and chlorite 0.06 mm in width were present. Apatite inclusions in quartz grains having maximum 0.13 mm, minimum 0.001 mm and average 0.009 mm in size were present. Inclusions of quartz in crystalline aggregates of goethite measured maximum 0.009 mm, minimum 0.001 mm and average 0.004 mm in size. Traces of chalcopyrite (Fig. 1.5) pyrite and marcasite (average 0.001 mm size) and feldspars were present.

The general texture as a whole was of a typical ferruginous sandstone type. Occasionally leached out ferruginous materials were found to fill in the intergranular spaces and could be considered as a gel-like chemically precipitated admixture of hydrated and later dehydrated oxides of iron with variable clay content. Their gel-like deposition could be inferred by their formation of scattered oolites along in the matrix (Figs. 1.4, 1.11 and 1.16).

Oolitic Variety

The second distant variety was the oolitic iron ore, where the whole matrix of the specimens consisted of oolites of varied size and shape cemented in an intimately admixed matrix of ferruginous, siliceous and clayey constituents (Figs. 1.2, 1.7, 1.9, 1.13, 1.14 and 1.18). The oolites formed about 65% to 80% of the matrix. Their sizes varied from 1.02 mm to 0.18 mm and consisted of an admixture of hydrated iron oxides (goethite, limonite) and dehydrated oxide of iron namely hematite with siliceous and clayey minerals. The siliceous gangue in both the matrix and the oolites proper was cryptocrystalline in nature and mostly present as chert (colloidal silica) with some amount of clayey materials intercalated with it. Traces of magnetite and chalcopyrite were also present.

A typical oolite showed a material of cryptocrystalline nature and oolitic texture indicating colloidal condition of deposition, showing concentric alternate layers of goethite or hematite mainly with intercalations of silica (Figs. 1.3 and 1.8). The nucleus of the oolite may be either quartz or crypto crystalline siliceous grains [illegible] 1.12 and 1.15). No fossil fragments could be detected as nuclei of the oolites. The proportion of the ferruginous, the siliceous and clayey constituents in the oolites as well as in the cementing matrix could not be assessed due to their crypto crystalline and very fine interlocking nature, as resulting from the colloidal gel like precipitation.

The amount of micro crystalline quartz and other siliceous minerals would be comparatively less than that in the ferruginous sandstone variety. The grain size of such quartz was maximum 0.25 mm, minimum 0.001 mm and average [illegible] mm. The size of cherty quartz present in the oolites ranged from 0.001 to 0.01 mm. Minor amounts of mica, feldspar and apatite were present in the matrix.

Mineralogical Distribution

Sample [illegible] (Crushed Ore)

A general mineralogical distribution in the different types of lumps examined is given in Table 1.1.

Table 1.1 Mineralogical distribution in the different types of lumps

Nature of the lumps	Ferruginous Sandstone type		Oolitic type		
Specimen No.	1	2	3	4	5
Specific Gravity	2.8	[illegible]	[illegible]	2.[illegible]	2.8
Mineral Association	Quartz, Pressure Quartz Unit undulose Quartz Muscovite Chlorite Biotite Apatite Goethite Laterite Limonite Hematite	Quartz, Pressure Quartz Unit undulose Quartz Biotite Chlorite Muscovite Apatite Siderite Goethite Laterite Limonite Hematite	Goethite Limonite Hematite Laterite Crypto crystalline Silica Mica Quartz Apatite	As specimen No. 3 No hematite	As specimen No. 4

Sample III (Run of Mine Ore)

A higher percentage of hematite was observed in the sample as a whole as compared to sample No. I. More crystalline quartz grains were present in the ferruginous sandstone variety than in the oolitic

Table 1.2 - Qualitative mineralogical distribution in different lumps of the ore

Specimen No.	1	2	3	4	5	6
	Ferruginous sandstone variety	Ferruginous sandstone variety	Oolitic type	Oolitic type	Oolitic type	Oolitic type
Specific Gravity	2.7	2.7	2.8	2.9	2.8	2.7
Mineral	Quartz. Pressure quartz. Unit undulose Quartz, Micas, Apatite Calcite Siderite Hematite Goethite Laterite Chalcopyrite, Pyrite	Quartz Chert Micas. Apatite Calcite Hematite Goethite etc. Pyrite Chalcopyrite	Quartz Chert Micas. Apatite Siderite Hematite Goethite etc. Pyrite Chalcopyrite	Same as Specimen No. 3	Same as Specimen No 4	Same as Specimen No 4

variety lumps. Some calcite grains and stringers were noticed. Apatite was present as inclusions in the quartz grains. Cholcopyrite, pyrite and micas were present as specks in the ferruginous matrix.

A general qualitative mineralogical distribution in the different varieties of lumps is given in Table 1.2.

Grain Size Study

Sample I (Crushed Ore)

A detailed study of thin sections as well as polished sections of different lumps revealed considerable size variation of the siliceous gangue minerals as given in Table 1.3.

Table 1.3

Grain size of siliceous gangue minerals in different lumps

	Specimen No.				
	1	2	3	4	5
Maximum	1.9350	0.4515	0.1000	0.1200	0.2000
Minimum	0.010	0.010	0.001	0.001	0.001
Average	0.100	0.100	0.020	0.030	0.020

Sample III (Run of Mine Ore)

A detailed study of thin sections as well as polished mounts of different types of lumps revealed the varied degree of size variation of the siliceous and other non-metallic minerals present as given in Table 1.4. It may be seen from these results that only the interstitial quartz, where average size range was 0.06 mm to 0.08 mm was expected to be freed at a grind size of about -200 mesh. The other type, namely, cherty quartz, which was present as very fine inclusions in the hematite-goethite matrix as well as in and around the oolite ring structure was 0.004 mm to 0.01 mm in size and was not likely to be freed unless the whole material was ground to such fineness. This observation was confirmed by the mineral interlocking studies of the samples that followed.

Interlocking of Constituent Minerals

Sample I (Crushed Ore)

Examination of the various sieve fractions of the -10 mesh representative sample of the lumps revealed the presence of siderite

Table 1.4 - Grain size in mm of siliceous and other non-metallic and trace metallic minerals

Specimen No.	1	2	3	4	5	6
Mineral Constituent						
1. Quartz (Interstitial)						
Maximum	0.112	0.50	0.090	0.16	-	0.250
Minimum	0.001	0.002	0.001	0.001	-	0.001
Average	0.080	0.060	0.010	0.010	0.010	0.018
2. Quartz as inclusions and nucleus						
Maximum	0.009	-	-	-	-	0.190
Minimum	0.001					0.004
Average	0.004					0.010
3. Calcite grains and stringers						
Average		0.16 to 0.01				
4. Apatite inclusions in quartz						
Maximum		0.130				
Minimum		0.001				
Average		0.009				
5. Chalcopyrite						
Maximum	0.002					
Minimum	0.0002					
Average	0.001					
6. Pyrite						
Average	0.01					

and other carbonates, pyrite, marcasite and apatite in association with siliceous minerals even at -200 + 270 mesh size. The observations on general liberation and interlocking of the ferruginous oxides with the siliceous minerals are given in Table 1.5.

Table 1.3 and Table 1.5 evidently show that most of the interstitial siliceous gangue present in the ferruginous matrix of the

Table 1.5 - Liberation and interlocking of mineral of -10 mesh sieve fractions

	-100+150	-150+200	-200+270	-270
Mineral Constituent	Vol %	Vol %	Vol %	Not examined
1. Free ferruginous	25.4	29.2	32.2	
2. Interlocked ferruginous (with fine colloidal silica)	68.8	63.4	58.3	
3. Free siliceous	5.8	7.4	9.5	
Total	100.0	100.0	100.0	

ferruginous sand stone variety would be free at about 60 to 100 micron size (about 200 mesh). The siliceous gangue present in the oolites was of colloidal nature and was too finely interlocked with the ferruginous minerals during the formation of oolites and its average size was 0.016 mm. As such, a very fine grind would be needed to free this material from the oolites.

A detailed study of the samples revealed that they consisted more of oolitic variety and the ferruginous sandstone variety was a minor constituent. This was confirmed from the results in Table 1.5, wherein the interlocked ferruginous constituents which formed the bulk, were derived only from the oolites and whatever free silica was present, it had resulted from the sandstone variety material of the sample.

Sample III (Run of mine ore)

The various sieve fractions of the -10 mesh representative sample of the ore were examined to find out the general interlocking nature of the various mineral constituents to indicate the optimum grind to be employed for a fair liberation of the gangue from the ore minerals. It was observed that calcite, siderite, pyrite, apatite etc. were in association with siliceous minerals even in the -200 +270 mesh fraction. The general liberation and locking results are given in Table 1.6.

These results show that most of the siliceous gangue present as interstitial grains in the ferruginous matrix material of the ferruginous sand stone variety lumps as well as the oolitic variety lumps would be free at about 60 to 80 microns size (about 200 mesh size) which was the average size of the interstitial quartz. The very fine cherty quartz and other siliceous gangue grains inherent in the oolite ring structure were too finely interlocked with the ferruginous components of the oolites due to their colloidal nature of deposition. The interlocked

Table 1.6 - Liberation and locking analysis

Mesh Size	-65+100	-100+150	-150+200	-200+270	-270
Mineral Constituents	Vol %	Vol %	Vol %	Vol %	Not examined
1. Free ferruginous	20.5	23.2	27.8	33.4	
2. Interlocked ferruginous (with clay and fine silica)	74.0	70.6	63.0	54.3	
3. Free siliceous gangue	5.5	6.2	9.2	12.2	
Total	100.0	100.0	100.0	100.0	

ferruginous constituents column of table 1.6 is formed of this material from the oolites for which a very fine grind would be, therefore, needed to free them from the ferruginous minerals.

On the basis of the above studies, it may be broadly stated that oolites constituted the bulk of the sample (55% to 65%) followed by ferruginous matrix (15% to 20%) and quartz and other siliceous and non-metallic minerals (20% to 30%). Among the non-metallic minerals crystalline quartz constituted the bulk of them, followed by micas, feldspars, calcite, apatite, etc.

Sample No. IV (194 tonne Bulk sample)

Of the 194 tonne bulk sample received for pilot plant investigations, some typical hand-picked lumps were chosen for further petrological studies which would be complimentary and confirmative of previous studies conducted on the earlier lots of the samples to have a more wider picture of the mineralogical nature of the ore which would be directly responsible for the behaviour of the sample as a whole in the various beneficiation and metallurgical treatment cycles to be followed on it.

The studes on the lumps included (1) Megascopic observations on visual examination, (2) Apparent specific-gravity determination, (3) X-ray fluorescence analysis for determining the Fe-content, (4) Examination of thin sections under petrological microscope, and (5) Examination of polished sections under ore-microscope. Besides, sieve fractions of -10 mesh representative sample were also examined for qualitative and quantitative mineralogical estimation of the bulk sample. The results are given in the following paragraphs:

Megascopic Observations

The bulk sample, as received, was found lumpy (150 cm to fines), massive in nature and invariably coated with the pinkish-red loamy

Fig 1.19 Different grades of lumps of Asswan iron ore and their powders

material. Washed and freshly broken specimens and their powders showed different colours varying from earthy white, brownish grey, brown, brownish, yellow to pinkish-red, with increasing iron content. A wide variation of texture depending on the grain size and denseness was noticed. The apparent specific gravity of different lumps and pieces varied from 2.2 to 3.4 as determined by Jolly's spring balance. A gradation from shale, siltstone, ferruginous sandstone and oolite iron ore with uneven admixtures with one to the other as expected from sedimentary deposits of oolitic origin was noticed. Their powders were subjected to X-ray fluorescence analysis for the determination of Fe % in them. The various physical properties of different lumps and their X-ray fluorescence analysis for Fe % are given in Table 1.7. As may be seen from the results of Table 1.7, there was an increase of iron content ranging from 9% to 48% in the different lumps with increase of their apparent specific gravity. This type of variation in the grade of lumps is expected in sendimentary deposits which must have experienced and resulted from different geological processes in ages long before. As such, a thorough mixing of the sample after mining operations is very much essential to get a homogenous and more or less uniform feed from such deposits of heterogenous nature. The different lumps and their powders are shown in Fig. 1.19.

Microscopic Studies

Thin sections and polished sections of the different lumps as well as sieve fractions of -10 mesh representative sample were examined under petrological and ore microscopes.

A. Mineralogy and Texture:

The ore essentially consisted of oxides and hydroxides of iron, namely hematite, goethite, hydrogoethite (limonite), laterite, traces of magnetic siderite and manganese hydroxides. The other minerals which formed the bulk of the gangue were quartz (crystalline, cherty and colloidal), followed by clay minerals and feldspars, micas, calcite, apatite, pyrite, marcasite and chalcopyrite. Depending upon the nature of the lumps, the mineral assemblage varied in them. Thus the microscopic observations of a silt-stone revealed an extremely fine grained and crypto-crystalline sub-microscopic texture constituting of silicate mineral grains chiefly quartz impregnated in a colloidally admixed clayey - hydrated iron oxide, and a ferruginous clayey shale with apparent fine grained nature revealed crypto-crystalline sub-microscopic texture constituted of micron size siliceous grains segregated in a colloidal admixture of hydrated iron oxide and alumino-silicate matrix (Fig. 1.20). In a ferruginous sandstone with its loosely consolidated medium grained structure, quartz (crystalline) was the chief siliceous mineral constituent along with minor to trace amounts of carbonates, micas and apatite, impregnated in a matrix of colloidally admixed ferruginous-clayey constituents composed of oxides and hydroxides of iron (Fig. 1.21). The oolitic iron ore, which formed the bulk

Table 1.7 - Physical properties and X-ray fluorescence analysis of lumps

Sl. No.	Lump category	Colour		Apparent specific gravity	Denseness	Lusture	Fe % (X-ray)
		Lump	Powder				
1.	**Clayey shale**	**Earthy white**	Earthy white	2.2	Compact	Dull Earthy	9.0
2.	**Ferruginous clayey siltstone**	**Light pink**	Light pink	2.2	Compact	Dull	13.0
3.	**Ferruginous sandstone**	**Earthy brown**	Earthy brown	2.4	Less Compact	Dull	18.0
4.	**Ferruginous sandstone**	**Brown**	Brown	2.5	Compact	Dull	34.1
5.	**Ferruginous sandstone**	**Dark Brown**	Dark Brown	2.6	Compact	Dull	37.0
6.	**Ferruginous sandstone**	**Brownish yellow**	Brownish yellow	2.7 to 2.8	Compact	Dull	40.2
7.	**Oolitic iron ore**	**Reddish brown**	Pinkish red	2.9 to 3.0	Compact	Sub-metallic	47.0
8.	**Oolitic**	**Reddish brown**	Pinkish red	3.4	Compact	Sub-metallic	48.0

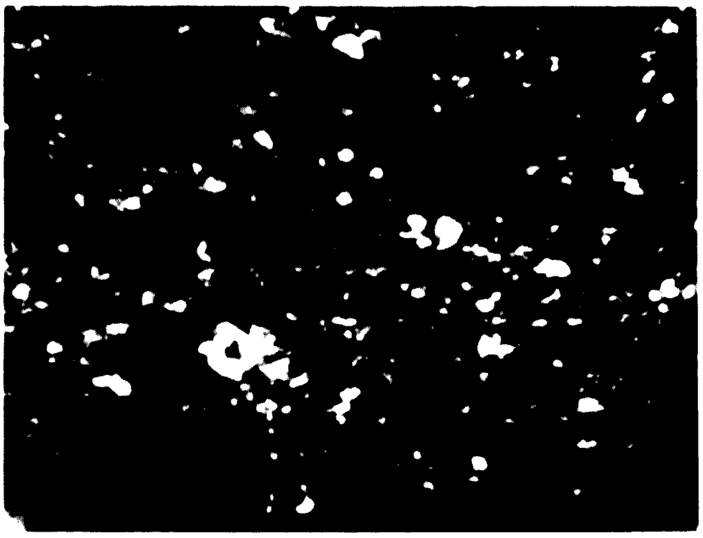


Fig. 1.20 Ferruginous shale showing siliceous gangue (white) in bands in an opaque ferruginous matrix. Transmitted light. Crossed nicols. × 100.

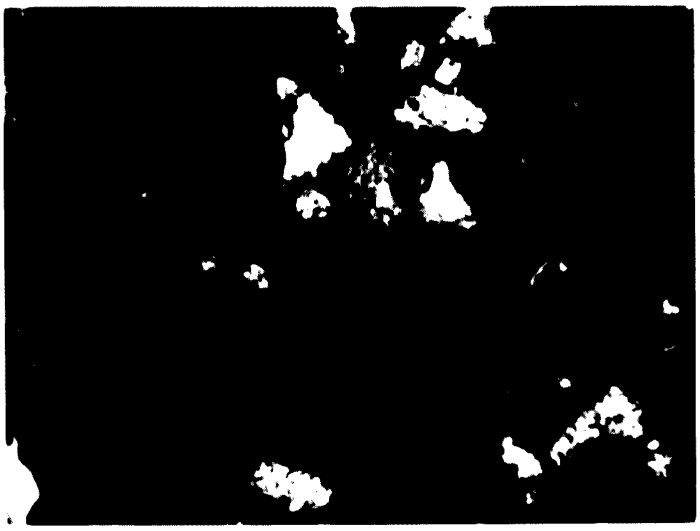


Fig. 1.21 Ferruginous sandstone variety showing coarse grained quartz (white-grey) in a ferruginous matrix (dark). Transmitted light. Crossed nicols. × 100.

Fig. 1.22 Oolitic variety low grade showing very fine siliceous gangue (white) in concentric rings of oolites. Transmitted light. Crossed nicols. × 100.

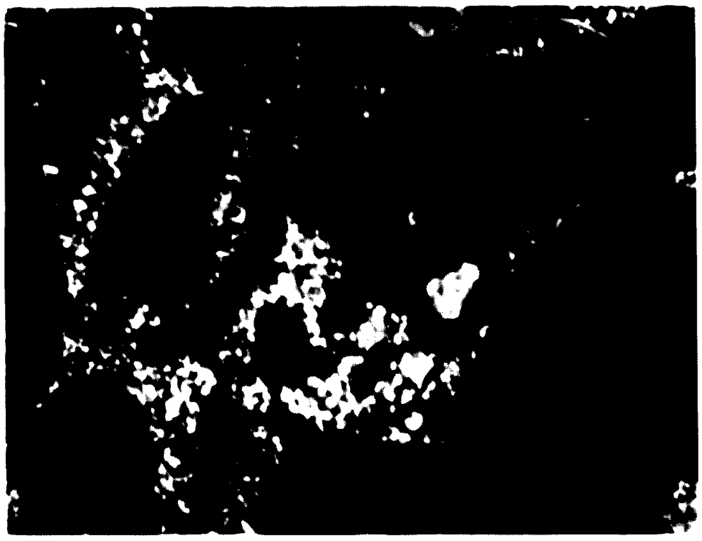


Fig. 1.23 Oolitic variety high grade showing very fine siliceous gangue (white) in the ferruginous matrix (black). Transmitted light. Crossed nicols. × 60.

of the sample, was actually a gradation of the ferruginous sandstone into the oolitic variety. Oolites of different shapes and sizes and entirely made up of colloidal precipitated hydrohematite in some cases were found in a ferruginous cementing matrix of amorphous nature. In others they constituted of hydrated iron oxides and alumino-silicates (Fig. 1.22, 1.23) as concentric bands with quartz grains as nuclei occasionally and were embedded in a fine grained ferruginous matrix. Thus the uneven and irregular mineral assemblages of the oolites and the cementing base material as a whole could be expected from the natural physico-chemical conditions of deposition and later metamorphism.

B. Grain size and liberation of the minerals:

The aspect of grain size and liberation of the various mineral constituents of the ore as a whole was already dealt with very elaborately with the samples I and III and is now presently confirmed once more that the average grain size of the siliceous minerals (mostly crystalline quartz present interstitial in the ferruginous matrix) ranged from 60 to 80 microns (about 200 mesh size). The colloidal silica which was present as concentric rings along and within oolites had a grain size ranging 0.001 mm to 0.01 mm (about 1 to 10 microns) and was very intimately associated with the matrix of the oolites and would not be effectively separated from them.

Discussion of Results and Conclusions

(i) Samples No. 1 and No. III were more or less alike and represented a lean hematite hydroxide iron ore of definite sedimentary origin by colloidal deposition and siliceous in nature;

(ii) It was predominantly of oolitic variety followed by ferruginous sandstone variety material as observed both in samples No. I and III;

(iii) The ferruginous minerals were hematite hydro-hematite (dehydrated), goethite, limonite, laterite and traces of magnetite and siderite;

(iv) Pyrite, marcasite and chalcopyrite were present in minor amounts and contributed the sulphur in the ore;

(v) The chief siliceous gangue mineral was quartz (both crystalline, cherty and colloidal) followed by calcite, feldspars micas, and clayey minerals;

(vi) The crystalline and cherty fragments of quartz present interstitial to the ferruginous matrix were medium to fine grained (tending more to the fine grained) in nature and had an average liberation size of [illegible] mm to [illegible] mm (about 200 mesh size). This constituted about nearly half of the total siliceous gangue in the sample with a grind of 200 mesh, the separation and rejection of this material from the ore by conventional ore-dressing methods should not be met with any difficulty;

(vii) The other form of siliceous gangue namely the colloidal silica which formed part and parcel of the oolites was in intimate association with the ferruginous material present as concentric rings in them, and had a grain size ranging from 0.001 to 0.01 mm. As such it would always be accompanying the oolite fragments unless an absolutely fine grind of a few microns size was employed. With such a fine size, any attempt of the ore-gangue separation by adopting any of the conventional ore-dressing methods may not be fully effective.

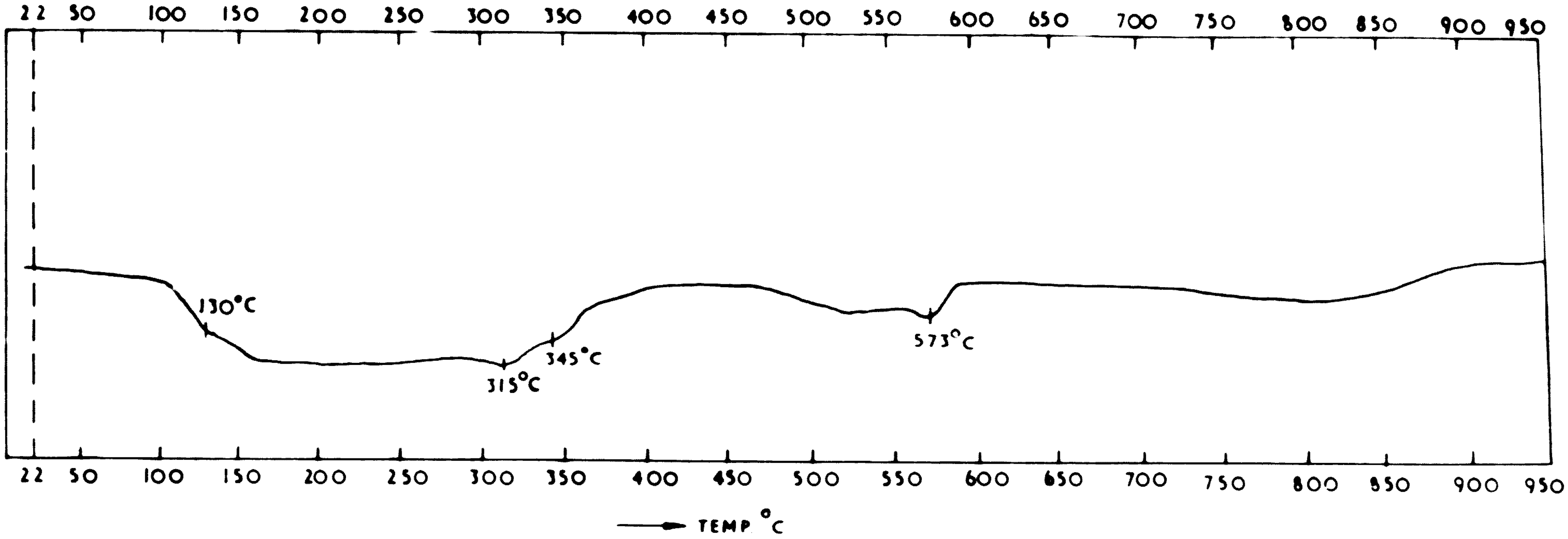
(viii) The thorough mineralogical study of the bulk sample confirmed the diverse mineralogical nature of the lumps ranging right from highly alumina rich siltstone to ferruginous shale and sandstone to iron-rich oolitic iron ore and thus stressed the complexity of the problem for the preferential removal of silica for the purpose of upgrading of the iron content of the sample by the conventional ore-dressing methods.

Differential Thermal Analysis (DTA) Studies

The differential thermal analysis studies on samples No. I, III and some typical lumps of the bulk sample (Sample IV) were carried out in the Deltatherm unit to have complimentary affirmative indications of the minerals present in them as far as possible in coroboration to their petrological findings. The samples were crushed to -200 mesh fineness and were heated in normal atmosphere to a differential temperature of 900°-950°C. The rate of increase of temperature was adjusted to 10°C/minute.

Observations, Results and Discussion

The differential temperature behaviour of the crushed and run of mine samples I and III was more or less identical. The D. T. A. curves revealed the regular endothermic peaks, characteristics of the ferric hydroxide minerals (goethite, etc.) between 300°C and 400°C, the presence of free quartz and its inversion point at 573°C, and more or less straight trend of the curve above this temperature was characteristic of hematite. On the whole, the presence of hydroxides of iron, quartz and hematite were the main qualitative indications, which were confirmed by petrological studies even though no quantitative estimates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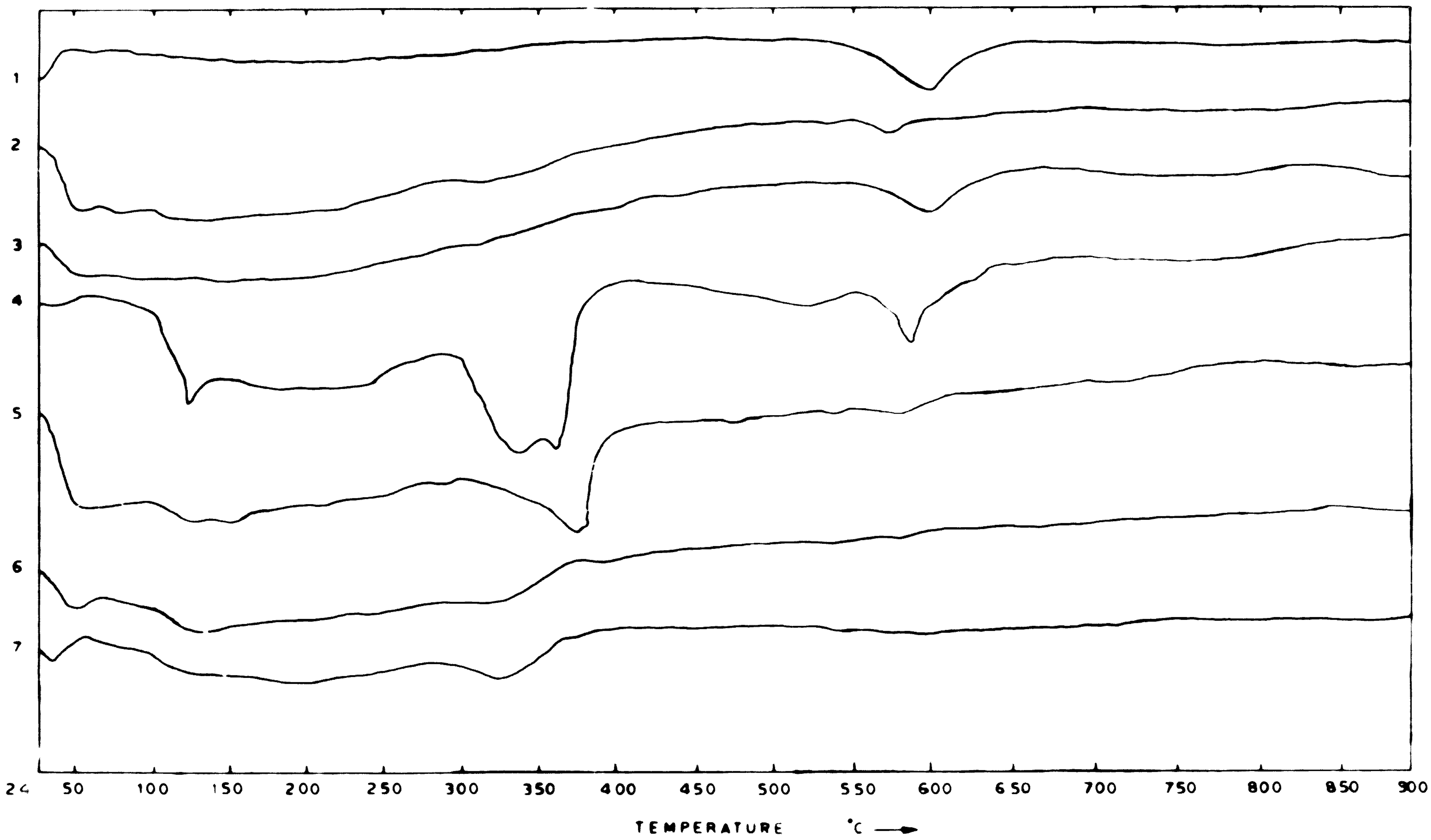
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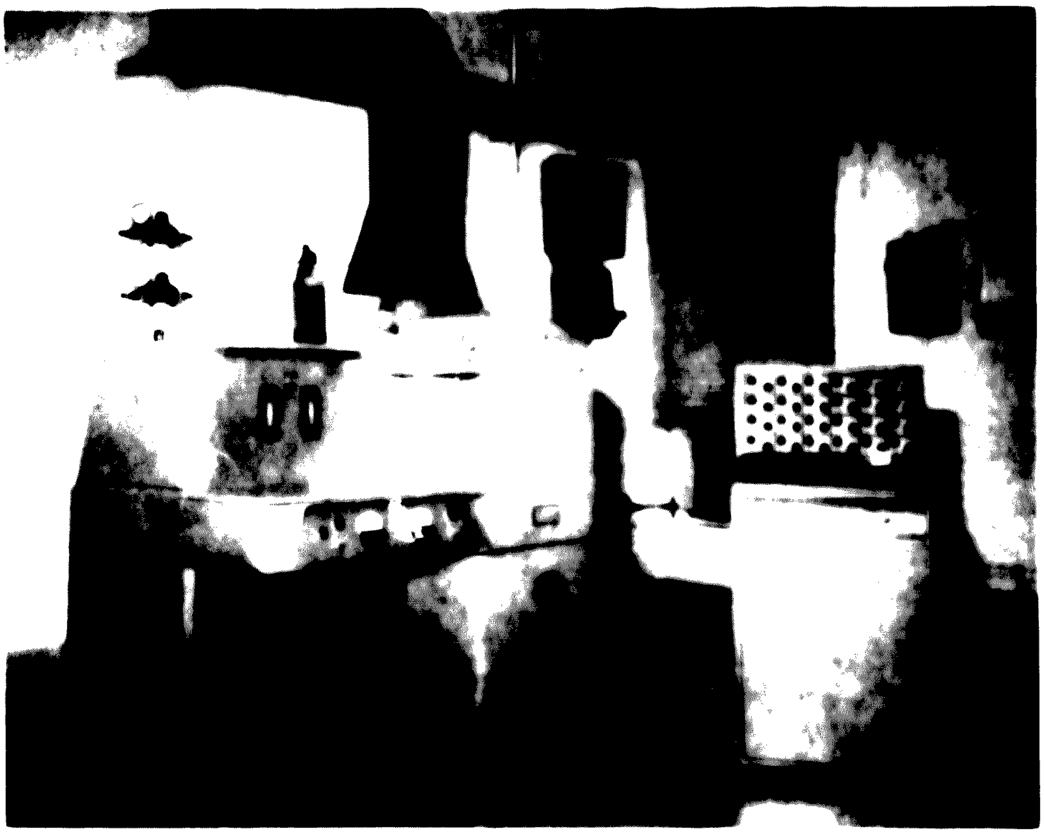
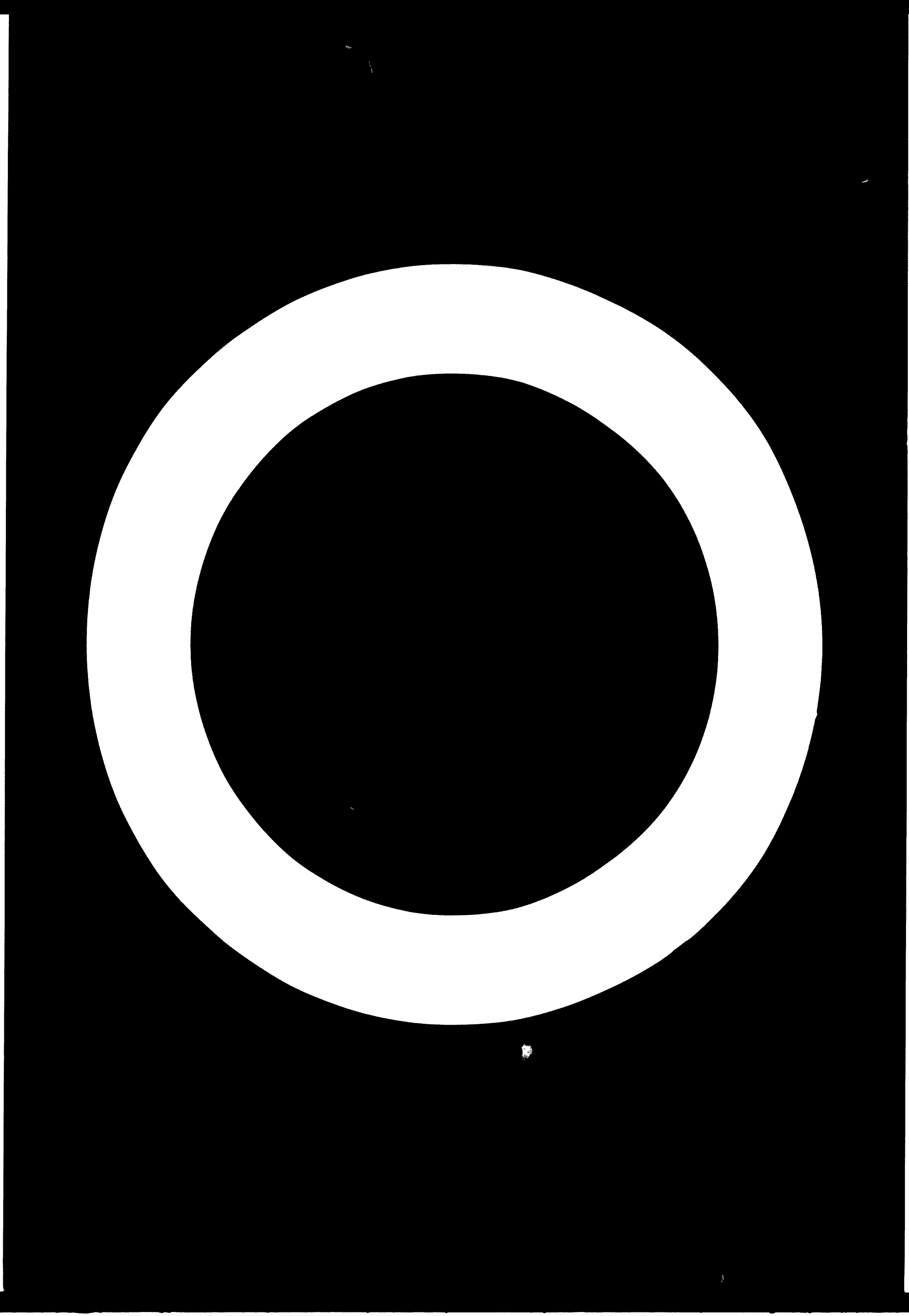


[illegible]

could be possible due to the very complex nature of the ore resulted from its colloidal origin. The typical D.T.A. curve of sample No. III is shown in Figure 1.24.

Regarding the bulk sample as seen from the D.T.A. curves (Fig. 1.25) of the ore lumps, curve No. 1 resulting from clayey shale lump whose Fe content was only [illegible] (X-ray analysis), the large endothermic peak from [illegible] was typical of any of the oxyhydroxides of aluminium (Kaolinite etc.). Curves No. 2 and 3 resulting from ferruginous clayey siltstone (Fe = 13 %) and ferruginous sandstone (Fe = 18 %) respectively were typical with the endothermic peak of oxyhydroxides of aluminium in major amounts from 550° – 600°C with minute indication of endothermic peak of hydroxides of iron at about 30[illegible] – [illegible]°C range. Curve No. 4 resulted from ferruginous sandstone (Fe = 34 %) was typical of hydroxides of iron (goethite, lepidocrocite) and free quartz, with the first endothermic peak due to hygroscopic moisture at about 325°C and the typical sharp exothermic hump at about 400°C. The typical quartz inversion point was shown with a peak of 5[illegible] – 580°C range with a straight run beyond 600°C. Curve No. 5 also resulted from ferruginous sandstone lump with higher Fe content (Fe = 40.2%) had given the usual endothermic peak at 325°C and 380°C and the slight exothermic hump at about 400°C indicating of hydroxides of iron (goethite, lepidocrocite), with the usual inversion point of quartz at 573°C and a straight course beyond 600°C. Curves 6 and 7 both representing the oolitic iron ore lumps (Fe = 47% - 48%) had the initial endothermic peak of goethite at about 325°C with a straight run indicating beyond indicating more of hematite content in them than the rest of the lumps.

On the whole the presence of oxyhydroxides of aluminium (Kaolinite, etc.), hydroxides of iron (goethite, lepidocrocite), limonite and hematite and free quartz was once more confirmed as the main mineral indications quantitatively present in the various ore lumps, which were already confirmed earlier by their detailed petrological studies. These studies had also indicated the complex nature of the ore as a whole so far as their mineralogical assemblage was concerned.



CHAPTER 2

BENEFICIATION STUDIES

(i) LABORATORY SCALE

All the three samples were separately crushed to about 25 mm in a laboratory jaw crusher. Half of the crushed product of each sample was staged crushed in a roll crusher and used for experimental pruposes. The sieve analyses of the crushed products of the three samples are given in Table 2. 1.

Table 2. 1 - Sieve analyses of the roll crushed products

Size in Mesh	Sample 1 Wt %	Sample 2 Wt %	Sample 3 Wt %
+6	2.0	-	-
-6+10	11.5	2.5	-
-10+14	10.5	6.5	14.5
-14+20	9.0	8.0	12.30
-20+28	8.0	7.5	11.20
-28+35	6.5	7.0	9.50
-35+48	6.0	6.5	7.20
-48+65	5.5	6.5	6.30
-60+100	5.5	7.0	6.00
-100+150	5.0	6.5	5.00
-150+200	3.0	3.5	36.00
-200	27.5	38.5	25.00
Head	100.00	100.00	100.00

Note: Screens of Tylers Sieve series were used for the tests.

The chemical analyses of the three samples is given in Table 2.2.

The chemical analyses of the three samples indicated that the Fe content varied from 38.5 to 42% and the samples were high in SiO_2, Al_2O_3 and P contents and the problem consisted in separation of these from iron minerals.

Table 2.2 - Chemical analyses of Asswan iron ore samples

	Sample 1 lumps	Sample 2 fines	Sample 3 (representative of 200 ton lot)
Fe%	41.17	38.67	41.55
FeO%	0.43	0.43	-
Al_2O_3%	4.94	4.63	5.21
SiO_2%	20.44	22.67	17.33
CaO%	4.26	5.41	4.69
MgO%	1.17	0.74	1.20
S%	0.04	0.21	0.08
P%	0.93	0.80	0.995
TiO_2%	0.41	0.39	0.38
LOI%	6.70	6.96	4.96
MnO%	0.80	0.85	0.78
Ni%	0.006	0.006	

Mineralogical examination showed that hematite, goethite, limonite and laterite constituted the iron minerals in the sample. High intensity dry magnetic separation tests were aimed at first for recovery of these feebly magnetic iron minerals from the samples after grinding.

Dry Magnetic Separation Tests

Sample 1 (Crushed Ore)

A representative fraction of the original 6 mesh sample was ground to 100 mesh, deslimed, dried and subjected to dry magnetic separation using a strean's electromagnetic separation fractors are recorded in Table 2.3.

The magnetic concentrate analysed only 44.2% Fe and represented a recovery of 76.5% Fe. Dry Magnetic tests at still further finer sizes (viz 100 and 200 mesh) did not improve the results.

Magnetic separation of reduced Asswan iron ore in low intensity wet magnetic separator

Fig. 4

Table 2.3 - Dry magnetic separation results on sample 1

Product	Wt%	Assay% Fe	Distribution% Fe
Magnetic Conc.	71.0	44.2	76.5
Non-Magnetic Tails	11.0	19.4	5.2
Slime	18.0	41.8	18.3
Head	100.0	41.04	

The magnetic concentrates analysed only 44.2% Fe and represented a recovery of 76.5% Fe. Dry Magnetic tests at still further finer sizes (viz 100 and 200 mesh) did not improve the results.

Samples 2 & 3 (Fines & Run of Mine Ore)

Straight high intensity dry magnetic separation tests after grinding the sample to 65 mesh did not yield encouraging results. High intensity dry magnetic separation employing 65 to 200 mesh deslimed feeds, also did not give satisfactory results.

Reduction Roast followed by Wet Magnetic Separation

The method based on low temperature reduction roast followed by low intensity wet magnetic separation was next attempted for rejection of siliceous minerals from the ore sample. Reduction roast was carried out separately on all the three samples. The tests were carried out in an electrically heated tubular furnace maintained at a temperature of 500-520°C. The reducing agent employed was locally available coke oven gas analysing by volume: CO_2 - 3.8%, saturated higher hydrocarbons C_nH_m - 2.2%, O_2 - 0.7%, CO - 9.7%, H_2 - 51.0%, CH_4 - 23.0% and N_2 - 9.6%.

A batch of 1.5 kg. of 12 mm in size was taken from each of the three samples for reduction. It was initially heated to about 400°C after which coke oven gas was passed at the rate of 1 litre/minute. Since the reaction was exothermic, no external heat was necessary during passing of the gas. The temperature of the furnace was maintained between 500 and 520°C. The reduction was complete in about 100 minutes. The reduced material was cooled in the furnace, wet ground to various sizes and subjected to wet magnetic separation using Ding's wet magnetic separator. The results are recorded in Tables 2.4, 2.5, 2.6. Since grinding to -200 mesh before magnetic separation gave best results in case of samples 1 & 2 (crushed and fine ore) wet magnetic separation test on sample 3 (run of mine ore) was carried out only at this size.

Table 2.4 - Results of wet magnetic separation (after reduction) with sample 1 crushed ore

Mesh Size	Product	Wt %	Assay % Fe	Distribution % Fe
-48 mesh	Magnetic conc.	83.0	4[illegible].1	[illegible]
	Non-magnetic tails	7.0	1[illegible].1	[illegible]
	L.O.R.*	10.0		
	Head (calc.)	100.0	40.22	100.0
-100 mesh	Magnetic conc.	74.0	49.3	90.5
	Non-magnetic tails	16.0	24.04	9.5
	L.O.R.	10.0		
	Head (calc.)	100.0	40.33	100.0
-200 mesh	Magnetic conc.	75.0	49.4	91.4
	Non-magnetic tails	15.0	21.62	8.6
	L.O.R.	10.0		
	Head (calc.)	100.0	40.3	100.0

*Loss on reduction

The results recorded in Tables 2.4, 2.5, 2.6 showed that optimum results were obtained when wet magnetic separation was carried out on 200 mesh size. In case of sample 1, the concentrate assayed 49.4% Fe, with a recovery of 91.4% Fe. Wet magnetic separation test at 200 mesh size on the reduced product of sample 2 yielded a concentrate assaying 54.68% Fe, 9.8% SiO_2, 0.79% P with a recovery of 89.3% Fe, that on sample 3 gave magnetic concentrate assaying 51.25% Fe, with a recovery of 94.5% Fe in it.

Washing of the Original Sample Followed by Reduction Roast and Wet Magnetic Separation

The original sample was first washed then subjected to reduction roast and magnetic separation under the conditions described earlier.

Table 2.5 Wet magnetic separation test results on sample 2 (fines)

Mesh Size	Production	Wt %	% Fe	Assay % SiO2	P	Dist % Fe
28 mesh	Magnetic conc	78.0	46.88		-	96.6
	Non magnetic tails	12.0	10.80	-		3.4
	L.O.R	10.0	-	-	-	-
	Head (calc.)	100.0	37.86	-	-	100.0
100 mesh	Magnetic conc	64.0	53.5	11.0	0.792	89.3
	Non magnetic tails	26.0	15.67	-	-	10.7
	L.O.R	10.0		-	-	-
	Head (calc.)	100.0	38.32	-	-	100.0
200 mesh	Magnetic conc	63.0	54.68	9.8	0.79	89.3
	Non magnetic tails	27.0	15.23	-	-	10.7
	L.O.R	10.0	-	-	-	-
	Head (calc.)	100.0	38.56	-	-	100.0
325 mesh	Magnetic conc	62.0	52.74	11.7	0.71	85.9
	Non magnetic tails	28.0	19.2	-	-	14.1
	L.O.R	10.0	-	-	-	-
	Head (calc.)	100.0	38.07	-	-	100.0

Table [illegible] Results of wet magnetic separation (after reduction) at 200 mesh size on sample 3 (run of mine ore)

Product	Wt. %	Assay % Fe	Distribution % Fe
Magnetic conc	77.4	51.25	94.5
Non magnetic tails	12.6	18.00	5.5
Loss on reduction	10.0		-
Head	100.0	41.89	100.0

Sample 1 & 2 (Crushed and Fine Cre)

No tests were done with the washed sample

Sample 3 (Run of mine Ore)

The results of the test are given in Table 2.7.

Table 2.7 - Wet magnetic separation of the washed sample after reduction roast (sample 3 - run of mine ore)

Product	Wt. %	Assay % Fe	Distribution % Fe
Magnetic conc.	71.7	51.3	90.2
Non-magnetic tails	11.8	16.1	4.6
Slime	5.0	42.1	5.2
Loss on reduction	11.5	-	-
Head	100.0	40.78	100.0

The results of roast reduction tests after washing did not yield concentrate of a better grade.

Heating followed by Magnetic Separation

Only sample 3 was subjected to this test. The original 12 mm sample was heated to 850-900°C for about two hours, cooled to room temperature, then subjected to wet magnetic separation at 100 mesh size. The results of the wet magnetic separation is given in Table 2.8.

Table 2.8 - Results of wet magnetic separation test after heating (with sample 3 - run of mine ore)

Product	Wt. %	Assay % Fe	Distribution % Fe
Magnetic conc.	50.0	49.0	59.6
Non-magnetic tails	44.5	37.3	40.4
Loss on heating	5.5	-	-
Head	100.0	100.0	100.0

Filtration of magnetic concentrate after grinding and thickening

[illegible]

From the test results, it is clear that a good concentrate with a satisfactory iron recovery could not be obtained by simple heating followed by magnetic separation.

Flotation Test

Sample 1 (Crushed Ore)

Anionic flotation tests were performed for recovery of iron minerals from the sample. The -6 mesh sample was wet ground to -100 mesh in a laboratory ball mill at 66% solids. Due to the extremely friable and soft nature of the ore, considerable amount of slime was generated during grinding and hence desliming was found to be absolutely essential before flotation. The following reagent combinations were attempted for flotation of iron minerals, but without success.

(i) Sodium silicate as gangue depressant and dispersant and oleic acid + fuel oil as collector for iron minerals.

(ii) Petroleum sulfonate + fuel oil as collector for iron minerals keeping the siliceous gangue depressed using H_2SO_4 at a pH of 3.5 - 4.0.

Sample 3 (Run of Mines Ore)

Cationic Flotation

A few flotation tests were also conducted after grinding the sample, using sodium silicate as dispersant, starch as depressant for iron bearing minerals, caustic soda as pH regulator and aeromine as collector for siliceous minerals. The results of the tests were not satisfactory.

Anionic Flotation of Siliceous Minerals Using $BaCl_2$ as Activator

A few flotation tests were undertaken using $BaCl_2$ as activator and oleic acid emulsion as collector for siliceous minerals, sodium silicate as dispersant and starch as depressant for iron bearing minerals. The conditions of a particular test is given in Table 2.9 and the results of the test are given in Table 2.10.

Summary and Conclusion of Investigations on Beneficiation

Three samples of iron ore were received from Arab Republic of Egypt for laboratory scale beneficiation studies. Sample 1 mainly consisted of lumps and assayed 41.17% Fe, 20.44% SiO_2, 4.94% Al_2O_3, 0.93% P; sample 2 assayed 38.67% Fe, 22.67% SiO_2, 5.63% Al_2O_3, 0.80% P; and sample 3 which is a representative sample of the 200 tonne lot analysed 41.55% Fe, 17.33% SiO_2, 4.69% Al_2O_3; and 0.955% P.

Table 2.9 - Reagents for flotation

Reagents	Reagents used		Kg/tonne of feed
	Rod Mill	Classifier	For Flotation
NaOH	0.5		
Starch		0.2	0.2
Sodium		0.4	0.4
$BaCl_2$			0.3
Oleic acid emulsion			0.4
	pH during flotation	=	10.4

Table 2.10 - Results of flotation test

Product	Wt. %	Assay % Fe	Distribution % Fe
Float 1	6.0	15.05	2.10
Float 2	5.2	20.70	2.5
Tails (Fe conc.)	52.2	49.0	58.5
Slime	36.6	44.2	36.9
Head (calc.)	100.0	43.73	100.0

Dry magnetic separation tests on the sample did not yield satisfactory results. Next all the three samples were subjected to low temperature (500 - 520°C) reduction roast followed by wet magnetic separation at different sizes.

Grinding to 200 mesh yielded the optimum results with all the samples. The wet magnetic concentrate from samples 1, 2 and 3 analysed 49.4% Fe, 54.68% Fe and 51.25% Fe with recoveries of 91.9%, 89.3% and 94.5% respectively.

A few roast reduction tests were conducted on sample 3 after washing the original sample. The results of the tests were no better than that obtained with straight reduction roasting.

Wet magnetic separation after heating the sample at 850 - 900°C for 2 hours also did not yield encouraging results.

Anionic flotation tests on sample 1 (crushed ore) using different reagent combinations did not give satisfactory results. Anionic flotation of silica on sample 3 (run of mine ore) using $BaCl_2$ as activator, oleic acid emulsion as collector, sodium silicate as dispersant and starch as depressant for iron bearing minerals yielded a Fe concentrate assaying 49.0% Fe with a recovery of 58.5% Fe.

(ii) INVESTIGATION ON PILOT PLANT

About 194 tonnes of run-of-mine ore from Asswan (Egypt-UAR, Egyptian Iron & Steel Co., Cairo) was received through UNIDO for comprehensive beneficiation studies on a pilot plant scale followed by pre-reduction and smelting in an electric arc furnace for the production of pig iron. The bulk sample was received at National Metallurgical Laboratory on 1st July, 1971. The sample consisted mostly of lumps upto 200 mm in size down to fines. The sieve and chemical analysis of the "as received" sample are recorded in Table 2.11 and 2.12 respectively.

Table 2.11 - Sieve analysis of ROM sample

Size	Percent
+ 8" (200 mm)	9.0
- 8" + 6" (-200 + 150 mm)	6.9
- 6" + 4" (-150 + 100 mm)	11.5
- 4" + 2" (-100 + 50 mm)	22.3
- 2" + 1" (-50 + 25 mm)	15.6
- 1" + 1/2" (-25 + 12.5 mm)	9.5
- 1/2" + 4 mesh (-125 mm + 4 mesh)	8.5
- 4 + 6 mesh	1.7
- 6 + 8 mesh	2.4
- 8 + 10 mesh	1.4
- 10 mesh	11.5
	100.0

Table 2.12 Chemical Analysis of ROM Sample

Constituents	Percentage
Fe	41.20
SiO_2	17.86
Al_2O_3	7.54
CaO	4.04
MgO	1.58
MnO	0.04
TiO_2	0.30
S	0.34
P	1.03
L.O.I	6.20

Additional data for R.O.M. sample

(a) Specific gravity - 2.9
(b) Moisture content as received - 1.82%
(c) Bulk density 1.244 tonne/m^3
(d) Angle of repose. 42°56'

Beneficiation Procedure

Based on the laboratory scale studies on the 3 samples received earlier, the major beneficiation steps for upgrading this ore involved are:

(a) Hand cobbing and subsequent primary crushing of the lumps to 38.0 mm size using jaw crusher, followed by crushing the same in a secondary crusher to about 18.5 mm size.

(b) Magnetising reduction roast of the crushed ore at an optimum size followed by grinding the reduced material to the desired fineness and wet magnetic separation of the ground product. The concentrate from the magnetic separation constituted the feed material for pelletization.

[illegible]

[illegible]

[illegible]	[illegible] discharge
	Wt [illegible]
[illegible]	[illegible]
[illegible]	[illegible]
[illegible]	[illegible]
[illegible]	[illegible]
[illegible]	[illegible] 4
[illegible]	2 [illegible]
[illegible]	2.0
[illegible]	1.5
[illegible]	17.8
	100.0

Secondary crushing. The jaw crusher discharge was subsequently crushed in a gyratory crusher ([illegible] type) with a setting of 18 mm discharge size for the first [illegible] tests. The sieve analysis of the product is recorded in Table 2.14.

Three batches of the representative original sample were crushed to 25 mm, 12 mm and [illegible] mm sizes for magnetizing roast tests for determining the optimum size for roasting. The sieve analysis of these samples are recorded in Table 2.15.

Table 2.14 - Sieve analysis of the secondary crusher product

Size	Wt. %
+ 18 mm	7.0
- 18 mm + 12.5 mm	26.0
- 12.5 mm + 4 mesh	20.5
- 4 + 6 mesh	9.8
- 6 + 8 mesh	4.7
- 8 + 10 mesh	6.6
- 10 mesh	25.4
	100.0

Magnetising Reduction Roasting

Description of the Rotary Kiln - These tests were carried out in a pilot plant rotary kiln (Fig.2.1) having a length of 10.67 m with 0.914 m inside diameter. The kiln is lined with firebricks and is driven by a variable speed motor of 3 HP. The inclination of the kiln was kept at 6° from the horizontal position and the r.p.m. of the kiln could be varied from 0.5 to 1.0. Three chromel-alumel thermocouples are evenly spaced along the kiln shell and are connected via strip rings to a multi-channel temperature recorder. All the thermocouples are projected beyond the brick lining inside the kiln so that the readings indicate the temperature of the gaseous zone inside the kiln. For heating purpose, a gas burner is fitted in the discharge side of kiln which works on a mixture of producer gas and air in proper proportion to give the optimum temperature required for the process. The process of heating is counter-current so as to give the maximum thermal efficiency to the furnace. A 50.8 mm additional pipe is inserted upto about 3 meters inside the furnace from the discharge end through which reducing gas can be supplied to the reduction zone of the furnace.

Reduction Roasting Process

A low temperature magnetising roast process for beneficiation of ferruginous manganese ores of India was developed by NML. The process briefly consisted of converting the oxides of iron to magnetite by solid or gaseous reduction in the temperature range of 500-650°C. A temperature higher than 600°C can be tolerated provided there is

Fig. 2.1 A view of the rotary kiln installed at NML for conducting reduction roasting of Asswan iron ore

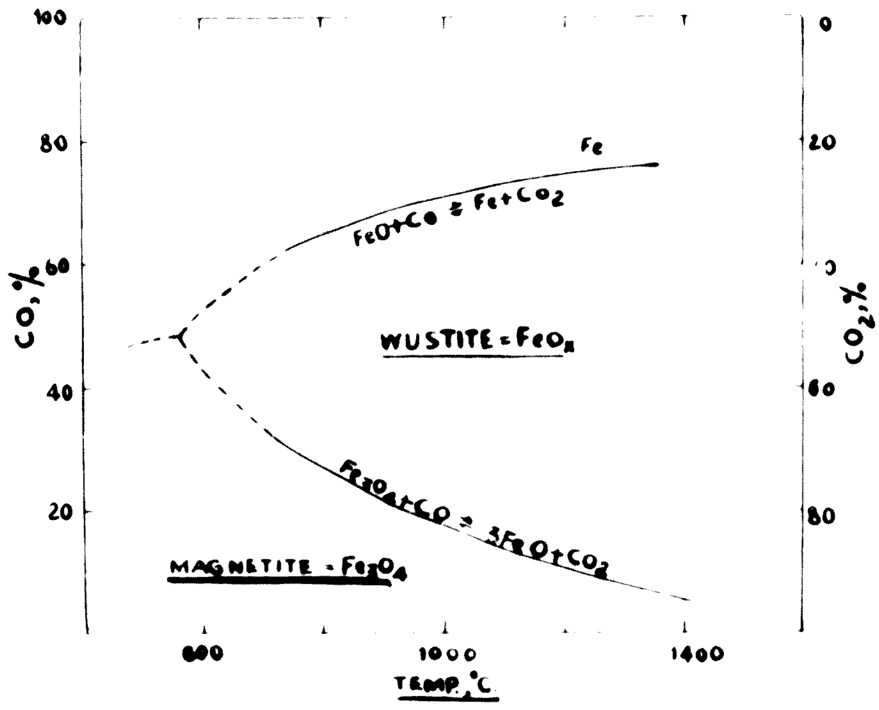


Fig 2.2 Iron - Oxygen - Carbon reduction equilibrium diagram

Table 2.15 - Sieve analysis of secondary crushed product for 25 mm, 12.5 mm and 9.5 mm

Size	Wt. % Products 25 mm	12.5 mm	9.5 mm
+ 18.5 mm	5.6	-	-
- 25.0 + 12.5 mm	28.7	2.6	-
- 12.5 + 9.5 mm	21.8	20.7	7.3
- 9.5 + 4 mesh	9.0	18.2	18.3
- 4 + 6 mesh	6.8	7.8	8.0
- 6 + 8 mesh	3.9	5.5	5.9
- 8 + 10 mesh	3.7	5.2	5.3
- 10 + 14	20.5	4.1	4.4
- 14 + 20		4.4	4.9
- 20 + 28		4.7	4.8
- 28 + 35		3.7	6.7
- 35 + 48		4.6	6.6
- 48 + 65		4.1	6.3
- 65 + 100		3.8	6.7
- 100		10.6	14.8
	100.0	100.0	100.0

a close control over the reducing gases as indicated by equilibrium diagram (Fig. 2.2). The reduced ore is softer than the unreduced ore, making it easier to comminute to the size necessary for economic liberation. Magnetic separation of the ground reduced ore removes the iron rich magnetic fraction, leaving the gangue in the non-magnetic fraction. The process of reduction roast is exothermal and the heat generated can be effectively utilised in minimising the fuel requirement for continuous operation.

Solid, liquid or gaseous reductants could be used for the magnetising roasting process. Blast furnace or coke oven gas can be used for reduction depending upon their availability. The quantity of gas required for the reduction depends on the carbon monoxide and and hydrogen content in the gas, as well as on the amount of iron to be reduced in the ore. The ore is roasted at a temperature of 600 650°C.

Experimental Results: Producer gas which was available at the NML pilot plant was used as a fuel. It has the following composition

Table 2.16 - Analysis of the producer gas

Constituent	Percentage
CO_2	3.5
CO	27.8
H_2	7.5
CH_4	3.8
O_2	0.2
N_2	54.3
Calorific value	130.5 B.t.u./Cuft.

A coke oven gas which was used for reduction during batch scale tests, was not available at the pilot plant site. The reducing agents were coal and L.D. Oil the analyses of which are recorded in table 2.17

Table 2.17 Analysis of coal and L.D. oil used as reductants

Coal		L. D. Oil	
Fixed C	27.11%	C	86.0%
Ash	36.58%	H	13.0%
V.M.	31.24%	S	1.0
Moisture	5.07%	Sp. Gr.	0.88
		B.P.	180°C
Calorific value	390 Cal/kg	C.V.	21,380
		Flash point	- 74°C

The reductants were throughly mixed up with the crushed ore before feeding into the rotary kiln with vibratory feeders. The waste gas is let out of the kiln at the feed end. The volumes of flow gas and air were registered on the flow-meters.

The reduction of Fe_2O_3 to Fe_3O_4 took about 30 minutes at reaction temperature. However, the time required for drying and preheating the dry ore increased the retention time of the ore in the kiln to about 100-120 minutes. The ore mixed with the reducing agents was continuously fed by a vibrating feeder to a belt conveyor which in turn, fed a hopper in the feed end of the kiln. From the feed hopper, the ore flowed by gravity through a pipe extended well upto the inside of the kiln. To avoid re-oxidation of the reduced ore, the material was discharged through a dip-pipe into a water quenched tank from where it was picked up by a conveyor belt.

Altogether 15 tests were carried out in the rotary kiln to find out the optimum conditions for commercial operation. The variables studied were feed size, quantities of reductants and the feed rate. The results of these tests are recorded in Table 2.18. The performance data of these tests are recorded in Table 2.19. In Test 11, the raw material was carried on a 12.5 mm screen and the reductants were separately mixed with the screened fractions before charging.

The results indicated that either coal or oil alone as reductant did not yield good recovery of iron content of the sample as the reduction in almost all the cases was partial. A combination of 25% coal and 5% oil gave the best results. Optimum results were obtained with a feed size of 18.5 mm. For larger sizes the retention time inside the kiln was not sufficient for the penetration of reducing gas inside the core of the ore pieces to reduce the same to magnetite. Good results were also obtained with crushed ores of 12.5 mm and 9.5 mm sizes but the extra amount of dust produced interfered with the operation of the kiln and subsequently the yield obtained in both the case were comparatively low. Mineralogical examination of the products of each test was performed under ore and petrological microscopes and the finding of the same are recorded below:-

Test 1:

The size of the sample pieces as received were in two groups, coarse +18.5 mm and fine -18.5 mm. These pieces experienced a temperature range round about 600 to 650°C. Physically these appeared brown to brownish black, but the freshly broken pieces were black or an admixture of brown and black. In some of the pieces, specially in the coarse variety, black highly lustrous rim with brown central material was noticed. The lustre of the pieces ranged from satiny to dull from occasionally segregated metallic lustrous body. The hardness of the sample was found variable trending from hard to friable. The degree of affinity of the various pieces for magnetic

Table 2.18 Results of the tests carried in the rotary kiln

Test No.	Feed Size m.m.	Producer gas m^3/hr	Air m^3/hr	Temp °C	Feed rate kg/hr	Percent reductants Coal	Percent reductants Oil	Wt. % Magnetic Fraction
1.	18.5	110	12	640	240	50	-	57.8
2.	18.5	110	12	640	240	40	-	49.0
3.	18.5	110	12	640	240	25	-	40.0
4.	18.5	112	16	640	240	15	-	32.5
5.	18.5	110	16	630	240	-	10	43.0
6.	18.5	110	16	630	240	-	-	36.0
7.	18.5	92	16	650	240	-	4	33.0
8.	18.5	110	12	650	240	25	5	76.0
9.	18.5	108	12	640	240	20	5	72.1
10.	18.5	112	12	640	240	25	4	71.5
11.	+ 12.5 mm 79.2% - 12.5 mm 20.8%	110	12	640	240	25	5	76.5
12.	18.5	110	12	600	360	25	5	65.8
13.	12.7	110	12	700	240	25	5	75.6
14.	9.5	110	12	650	240	25	5	74.8
15.	38.0	110	12	720	240	25	5	31.2

Table 2.19 - Magnetic Concentrate

Test No.	Wt. %	Assay % Fe	Dist % Fe	Remarks
1.	57.8	49.8	68.9	
2.	49.0	50.9	59.8	
3.	40.0	50.5	48.3	
4.	32.5	47.8	37.3	
5.	43.0	49.5	51.2	
6.	36.0	49.8	43.0	
7.	33.0	49.5	39.4	
8.	76.0	52.0	94.0	
9.	72.1	51.8	89.4	
10.	71.5	51.5	88.5	
11.	76.5	50.6	91.9	
12.	65.8	51.5	83.3	
13.	75.6	50.1	90.6	
14.	74.8	50.8	89.8	
15.	31.2	49.6	36.9	

field was variable in both the groups but the number of attractable pieces in the finer group was found to be more. Around some of the pieces highly magnetic fillings were present adherent to their surfaces.

The study of different polished sections of the two groups under reflected polarised light revealed that the effect of reduction no doubt existed in all cases but uneven and variable from piece to piece. The prevailed temperature effect could not disturb, as expected, in any of the prices of the original textural or the structural features. Thus the oolites present in coarse or in fine pieces either in the ochreous-argelorceous or rargelaceous-arenaceous fine grained ground mass were found insignificantly affected and did never approach to the magnetic stage. The ferruginous constituent admixed in the ground mass of oolites also in these coarse pieces was found to behave similarlv, while in the fine size the ferruginuous constituent in basement of oolites found appreciably affected and gave rise to innumerable anhedral patches of magnetite. The ferruginous constituents with less oolite content or oolite free argilaceous-arenaceous pieces were found to reach upto magnetite stage in the coarse fraction but relatively much advanced in the fine size, where recrystallisation gave rise to coarsegrained magnetite quite in appreciable quantity. In some cases, the dendrites of magnetite (not of primary texture) with replacing dendrites of hematite (also not primary) was noticed. The octahedral magnetite and the magnetite formed at the expense of hematite exhibiting the texture of solid diffusion effect was also noticed. It is worthy of mentioning that reduction process no doubt proceeded and the textural evidence also conforms the transition stage with the uneven effect, as appreciable amount of crystallite hematite, hydro-hematite remained and lack of complete magnetite stage was predominant both in the case of finer and coarser pieces. But the reduction effect could be enhanced by employing a finer feed was clear. It might be anticipated the complete stage of reduction to the magnetite stage by increasing the retention time. The varied nature of mineralogical assemblage was also one of the features of the uneven effect of reduction even in a single piece.

Test 2

The physical and microscopic observations exhibited, on the whole, the same criteria of the pieces due to reduction as that of the coarse and fine material of the test 1. The difference worthy of any mention was that excepting oolites the ocherous argelaceous/ocherous arenaceous fine grained ground mass was found to experience the initial stage of magnetite formation in coarse size lumps, whereas in finer size, the magnetite was quantitatively appreciable. Regarding the transition stage, ferruginous constituents adherent to the ground mass were quite appreciable in both the groups. The mesh like stringe occasionally gave rise to patchy magnetite and hematite body. In the fine size, like the earlier test pieces, recrystallised magnetite was found. In this test, the uneven effect of reduction, uneven mineral assemblages, and the incomplete stage of magnetite were also noticeable. The magnetic susceptibility of the reduced pieces either in Test 1 or Test 2 depended not only on the development stage of magnetite but also on the enrichment of the metallic minerals as such in them.

Test 3

Physically, these pieces were found alike those of the earlier test and microscopic observations revealed the same varied reduction effect in different pieces but the magnetite stage was nevertheless approached though not appreciable excepting in the sandstone type variety. It is noticed well in this test magnetite patches formed at the expense of hematite with intimately mixed non-metallics.

Test 4

In this test the physical property of the coarse size was like the coarse pieces of the earlier tests but the finer fraction was comparatively black. The magnetic property of coarse fraction was as that of the earlier tests, but the finer which was somewhat powdery in form was a bit highly magnetic.

Microscopically the reduction was found to approach to the magnetite stage. But the intensity of magnetite stage was in the start. The nature of reduction was, as earlier, with respect to its unevenness, incomplete. But in the finer fraction, some coarsegrained magnetite was noticed along with goethite and hematite associates. It is interesting to note here that quite a good amount of magnetite was found embedded in a non metallic anisotropic finegrained body. These trapped and intimately gangue-associated magnetite grains were of irregular micro to crypto-crystalline in form.

Test 5

In all respects, it resembled sample of test No. 4.

Test 6

Not much marked difference to the coarse pieces and fines of this test was found in comparison to those of test No. 4.

Test 7

The physical property of the sample more or less resembled that of the test pieces of 1 and 2. Microscopic observations revealed effect of reduction and mineral association. But it is interesting to note that the magnetite stage was no doubt approached but found the least occurrence of magnetite relative to all the previous stages of reduction in the earlier tests.

Test 8

Physically the sample of this test resembled to samples of other test but a good number of metallic lusterous pieces were noticed in the samples of this test. Microscopic observations as usual, revealed the uneven effect of reduction and mineral associations. But the most interesting features was in some of the pieces the 90% of the oolites in the siliceous finegrained ground mass was reduced to magnetite and the parent metallic mineral portion of each of the oolites reduced to magnetite stage upto 90-100%. But in other pieces magnetite occurrence was found more dominant in the metalliferous ground mass of Oolites and only partial reduction to magnetite stage in the metalliferous oolites themselves. There wa3 a sharp boundary of patchy magnetite between the ground mass and unreduced hematite oolites. This revealed a faster reduction effect in the metallics of the ground mass than that of the oolites. In the fine product, coarse grained magnetite was noticed along with all other parent metallic bodies. In this case, the variable metalliferous argalaceous-arenaceous pieces found to be reduced to magnetite stage. Studying the nature of reduction, in general, it might be inferred that if the retention time is prolonged in this test probably the magnetite stage would have been enhanced further. Due to the reduction effect like the earlier tests, no parental structural as well as textural change had taken place. The finer sample exhibited coarse grained magnetite and some magnetite grains intimately associated with sub-microscopic non-metallic grains. It was evident from the textural pattern that some state of difference took place and the coagulation of the segregated grain gave rise to coarse grain texture.

Test 9

Similar to the test 8.

Test 10

Similar to the test 8 but the coarsenss of the magnetite in finer group material was a bit high. In some of the coarse pieces pre-reduced brownish rim around the partially reduced black core was noticed.

Test 11

Not much marked difference from sample of Test 2.

Tests 12, 13, 14 & 15

Reduction quite advanced in all the tests with more magnetite.

The Asswan ore sample, with all its multi-mineral ore-gangue assemblage and varied and mixed texture such as coarse to medium grained sandstone to medium to fine grained argalaceous-arenaceous-ferruginous (silty-shaly) and oolitic lumps, had not behaved very much identical when compared to any of the conventional hematitic iron ores of India so far as the low temperature reduction process was concerned. Some sluggishness was felt in the advancing of the reduction stage particularly with more oolitic variety in association with the silty and shaly material. This may be attributed to the mineral assemblage and the natural texture in the oolites which were made up of concentric layers of very fine grained material by colloidal deposition, wherein the access to the reductant gases would be the minimum due to the general impervious nature of the matrix in which the oolites are embedded.

As was expected a higher degree of reduction was noticed with the finer material than withof coarser lumps. It was observed that besides the ferruginous matrix of the lumps, the oolites also had been fairly reduced to magnetite stage (as in test 8, 12, 13 & 14) where their association was indicative of the easy behaviour of the sample reduction under a controlled atmosphere and temperature and proper retention time besides mineral association.

On the whole, it can be surmised that the sample behaved with less sluggishness to the low temperature reduction process, and a proper control of temperature, feed size, temperature and retention time could result in a reduced product with a fairly moderate magnetic susceptibility although a complete reduction stage may be not

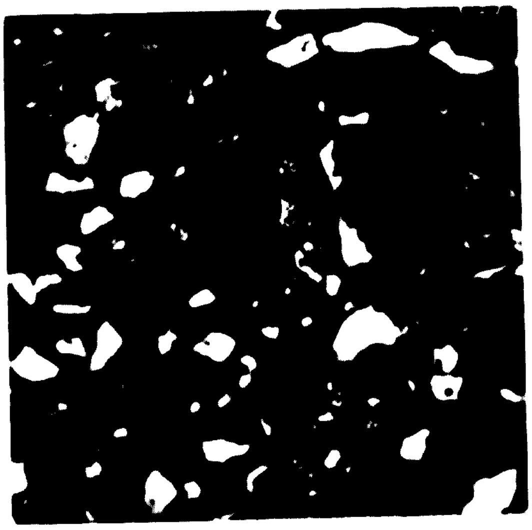


Fig. 2.3 Showing very and scanty response of reduction of a ferruginous shale sample. Dark grey grains are quartz. Reflected light. Test No. 14 100

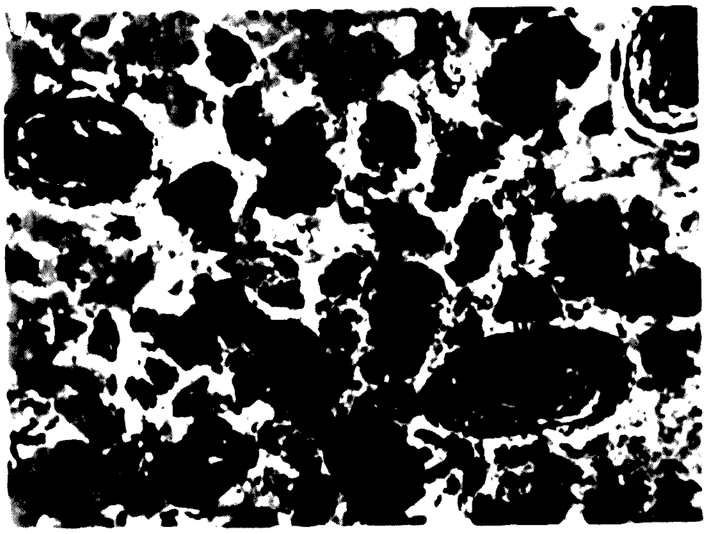


Fig. 2.4 Showing advanced stage of reduction of ferruginous [illegible] with some oolite [illegible] the matrix. Dark grey grains are quartz. Light grey grains are magnetite. Reflected light. Test No. 12 100

Fig. 2.5 Showing a typical oolite partially reduced with hematite and lightgrey magnetite. Dark grey grain in the centre is quartz. Reflected light. Test 8 100

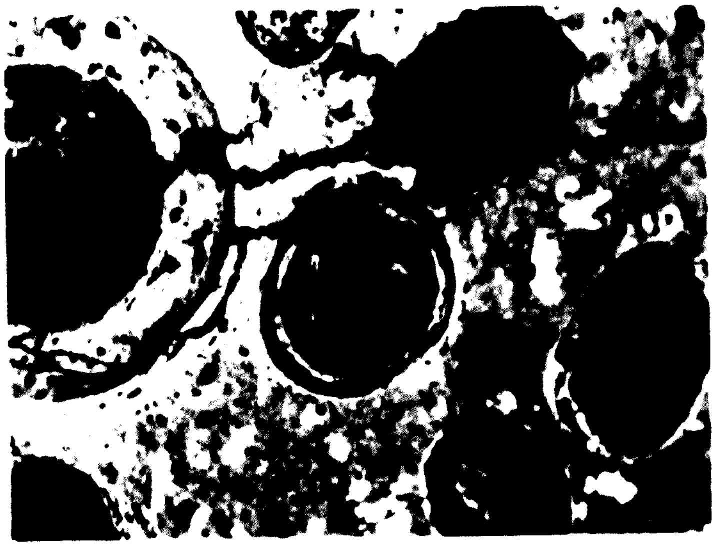


Fig. 2.6 A typical oolitic variety showing advanced stage of reduction of both the matrix and oolites. Light grey material is mostly magnetite. Dark grey grain inside oolites are quartz grains. Reflected light. Test 15 100 possible due to the [illegible] micrographs [illegible] 2.[illegible]

Continuous Pilot Plant Operation [illegible] Reduction Roast [illegible]

Having determined [illegible] magnetising roast [illegible] for an optimum degree of reduction [illegible] rotary kiln was employed [illegible] ore. Prior to magnetising roasting, [illegible] crushed to 18.5 mm size using primary and secondary [illegible]. The log data obtained during the crushing [illegible] Table 2.20 and the sieve analysis of the [illegible] given in Table 2.21.

Table 2.20 Log data obtained from primary and secondary [illegible]

(A) Primary Crushing

Type	Jaw crusher [illegible] type
Size of the Crusher	[illegible] cm x 40 [illegible] cm
Capacity	[illegible] tonnes/hr
Power consumption	4.[illegible] kWh/tonne

(B) Secondary Crushing

Type	TY type Gyratory Crusher
Size	38 cm
Capacity	[illegible] tonnes/hr
Power consumption	4.[illegible] kWh/tonne

Non-metallurgical coal from Maharashtra area, as used [illegible] previous tests, was also crushed to 18.5 mm for use as reductant. The sieve analysis of the coal is recorded in Table [illegible].

The rotary kiln was run continuously for magnetising roasting of the ore. The feed rate was kept constant at 150 kg. of ore per hour. The reductants used were 25% of coal and 5% oil. The temperature of the kiln was kept at 600-680°C in the reaction [illegible] requirement was kept constant at 110 m^3 and primary air [illegible]

Table 2.21 Sieve analysis of the crushed ore

Size	Wt %
+18.5 mm	6.5
18.5 + 12.5 mm	26.5
12.5 + 10 mm	19.8
10 mm + 4 mesh	9.7
4 + 6 mesh	5.0
6 + 8 mesh	3.9
8 + 10 mesh	6.5
10 mesh	22.1
	100.0

Table 2.22 Sieve analysis of coal

Size	Wt.%
+18.5 mm	3.5
18.5 mm + 12.5 mm	23.7
12.5 mm + 10 mm	18.2
10 mm + 4 mesh	9.1
4 + 6 mesh	8.7
6 + 8 mesh	7.2
8 + 10 mesh	13.1
10 mesh	16.5
	100.00

per hour. The discharge material was conveyed by a belt conveyor through a water seal trough for grinding followed by wet magnetic separation to recover the magnetite content from the product. The day-to-day operational data is recorded in Table 2.23.

The operation of rotary kiln was smooth throughout the campaign. The assay of the discharge product indicated that there was slight

Table 2.23 - Operational data of the rotary kiln for magnetising roasting

Date of operation	Hr. run.	Tonnage treated	Feed		% Reductants		Fuel consumed		Temperature Gradient °C			Assay % of discharge	
			size in mm	kg/hr.	Coal	Oil	Gas m³/hr.	Air m³/hr.	T1	T2	T3	Fe	SiO₂
23.7.71	5	0.75	18.0	150	25	5	110	12	560	300	150	41.4	22.1
24.7.71	24	3.60	18.0	151	25	5	110	12	600	320	160	43.0	20.6
25.7.71	24	3.62	18.0	151	25	5	112	13	630	325	155	43.1	20.4
26.7.71	24	3.60	18.0	152	25	5	112	12	590	320	155	42.0	21.1
27.7.71	24	3.66	18.0	140	25	5	110	12	650	340	170	41.6	21.5
28.7.71	24	3.65	18.0	151	25	5	110	12	660	350	160	42.7	21.5
29.7.71	24	3.63	18.0	149	25	5	112	12	650	350	170	42.1	21.1
30.7.71	24	3.58	18.0	148	25	5	110	12	630	340	150	42.0	21.2
31.7.71	24	3.56	18.0	152	25	5	110	12	650	345	155	41.2	22.7
1.8.71	24	3.61	18.0	153	25	5	110	12	640	350	160	41.3	22.2
2.8.71	24	3.62	18.0	150	25	5	110	12	630	340	140	41.2	22.2
3.8.71	24	3.65	18.0	155	25	5	110	12	640	320	140	41.7	21.8
4.8.71	24	3.59	18.0	151	25	5	110	12	660	350	160	41.6	22.4
5.8.71	24	3.62	18.0	148	25	5	110	12	660	360	160	41.7	22.7
6.8.71	24	3.65	18.0	151	25	5	112	12	580	325	130	42.0	21.3
7.8.71	24	3.67	18.0	154	25	5	110	12	650	350	170	42.9	22.4
8.8.71	24	3.60	18.0	148	25	5	110	12	640	340	160	41.6	22.5
9.8.71	24	3.62	18.0	149	25	5	110	12	630	315	150	41.8	22.6
10.8.71	24	3.58	'8.0	152	25	5	110	12	640	320	160	41.7	22.3
11.8.71	24	3.59	18.0	151	25	5	110	12	650	350	150	41.3	22.8
12.8.71	24	3.60	18.0	150	25	5	110	12	640	325	160	41.4	22.7
13.8.71	24	3.62	18.0	152	25	5	110	12	650	340	170	42.6	21.5
14.8.71	24	3.65	18.0	153	25	5	110	12	650	350	150	42.1	22.3
15.8.71	24	3.64	18.0	154	25	5	110	12	660	350	160	42.6	21.6
24 days	557	**82.8**											

T1 = Discharge Zone
T2 = Middle Zone
T3 = Feed Zone

variation in the mineralogical composition of the ore. The discharge products were occasionally sampled out and magnetically separated so as to see the percentage of char and ash coming out along with the reduced material. An analysis of the char indicated that it contained only about 1.25 - 1.50 percent of fixed carbon indicating that the furnace atmosphere was sufficiently reducing so as to have consistently good grade of reduced material. The sieve analysis of the reduced material after magnetic separation (for removal of char and ash) is recorded in Table 2.24.

Table 2.24 - Sieve analysis of the reduced material

Size	Wt. %
+ 18.5 mm	1.8
- 18.5 mm + 12.5 mm	18.7
- 12.5 mm + 10 mm	19.5
- 10 mm + 4 mesh	12.8
- 4 + 6 mesh	7.8
- 6 + 8 mesh	4.9
- 8 + 10 mesh	4.6
- 10 mesh	29.9
	100.0

As expected, the reduced ore became more soft and friable and disintegrated during magnetising roasting.

Additional Data on Rotary Kiln Operation

a) The volume and analysis of the waste gas:-

The waste gas volume was occasionally measured during the continuous run of the rotary kiln and was found to vary from 181.5 to 183.0 m^3/hour. The analysis of the same is recorded in Table 2.25.

Table 2.25 - Analysis of the waste gas

CO	3.4 %
CO_2	17.7 %
O_2	1.2 %
H_2	0.6 %

Material Balance Around the Kiln

For calculation of the material balance, the following parameters were studied during continuous operation at frequent intervals:

i) Ore input,
ii) Gas input, and
iii) Air input

In the output size, the material discharged per hour, the quantity of flue gases evolved, the flue dust escaping through the chimney were also measured several times, the generation of steam inside the kiln which was carried out by flue gas was theoretically calculated. The quantities of combustibles lost as tar, soot etc. was determined by difference and the complete material balance chart is recorded in Table 2.26.

Table 2.26 - Material Balance Around the Kiln

Basis: 1 hour

A. Input	Wt. in kg.	B Output	Wt. in kg.
1) Ore	150	Ore plus unreacted coal and ash	145.0
2) Coal	37.5	Steam	18.75
3) Oil	7.5	Flue gas & dust	171.0
4) Primary air 12 m^3	14.5	Combustibles lost as tar, soot etc. in smoke	17.75
5) Secondary air 15 m^3	18.0		
6) Producer gas 100 m^3	125.0		
	352.50 kg		352.50 kg

C) Capacity of the Rotary Kiln

The capacity of any specific rotary kiln depends on the physical and chemical properties of the charge material; its time of passage, temperature and heat requirements and on the temperature and the amount of hot gases supplied for heating. Thus a specific kiln will have different capacity for different materials. In this particular case, the standard formula postulated by Gibbs was utilised to determine the capacity factor.

$$\text{Tonnes of product/day} = \frac{KLD^2}{100}$$

This equation relates product capacity to kiln volume, where L is kiln length in feet, D the shell diameter in feet, the K is the factor whose magnitude depends on the charge material and other variables. For smallest kiln like ours whose length is 10.6 meter and inside diameter 0.76 meter, the shell volume to capacity ratio become very high due to the large percentage effect of ring build up in small diameter. The shell volume/capacity ratio Cu. m/ton/day is taken to be 2.9 in this particular case according to standard practice and the value of K as 0.76. The theoretical capacity comes to 1.66 tonnes/day. But in the trial, feed rate was quite high and about 3.6 tonnes of reduced material per day was obtained. As such the furnace was operated upto 2.2 times of the rated capacity of the kiln and 1.26 cu. m. of the shell volume was utilised for treating one tonne of material. According to 'Worthwhile' graphs, this rating of the kiln is considered as satisfactory.

D) Heat Requirement and Thermal Efficiency

The thermal efficiency of rotary kiln vary widely, depending on the extent of heat utilization or recovery facilities and on the economy of operating practices. Kiln length is a major factor and kilns with high ratio of length to diameter have a greater thermal efficiency than those with a low ratio.

The rotary kiln was operated without any attempt to recover the heat from the waste gases escaping through the chimney. The hot discharge was collected in the water-seal trough and the sensible heat in the hot discharge product was also lost. Without these facilities for heat recovery, it is obvious that the thermal efficiency of the kiln will be comparatively low, but attempt was made to see that the heat input inside the kiln through burning of gases, coal and fuel oil was not unnecessarily high.

Heat requirement was theoretically calculated for heating the ore and incoming gas to the reaction temperature, and the amount of gas required for reduction purpose as follows:

a) Heat requirement for heating up the material upto 550°C

Fe_2O	43.76	Kcal
SiO_2	17.10	"
Al_2O_3	8.46	"
CaO	4.20	"
H_2O	4.90	"
	78.42	Kcal/kg

Heat required per tonne of ore = 48420 Kcal

b) Heat required to heat air upto 550°C for use with producer gas = 9110 Kcal

c) Coal

C	26.97 Kcal
Ash	36.38 "
Volatile matter	37.45 "
Moisture	4.70
	105.50 Kcal/kg

Kcal required for 250 kg of coal = 26375 Kcal

Total Kcal required = 104, 795 Kcal

Heat obtained from exothermic reaction for 1 kg of ore for converting hematite to magnetite at 550°C = 21.9 Kcal/kg

Heat obtained after reduction of 1 tonne of ore = 11,826 Kcal

Net heat required = 92,969 Kcal

Heat Balance in the Rotary Kiln

a) Heat input per hour

i) Heat produced by producer gas	=	127,000 Kcal
ii) Heat produced due to LD oil	=	38,000 Kcal
iii) Heat produced due to coal	=	34,000 Kcal
Total heat produced inside the kiln	=	199,000 Kcal

b) Heat output per hour

i) For heating up the kiln and the incoming material upto 550°C	=	45,800 Kcal
ii) Heat lost with steam drawn at 160°C	=	3,200 Kcal
iii) Heat lost with flue gas etc. at 160°C	=	41,500 Kcal
iv) Heat lost with discharge material at 650°C	=	82,800 Kcal
v) Heat lost by radiation through the furnace wall	=	25,700 Kcal
Total	=	199,000 Kcal

Grinding Thickening and Magnetic Separation of Material Obtained by Roast Reduction

In order to establish the optimum particle size to which the reduced ore should be ground before wet magnetic separation, a series of batch tests were conducted employing feeds of different fineness, for wet magnetic separation. As revealed from the mineralogical studies, the ore minerals were finely disseminated with the siliceous gangue, which denoted that a grind as fine as 100 microns or below is necessary to cause effective separation. The concentrate obtained by wet magnetic separation may be of the desired particle size suitable for subsequent pelletisation studies. Since grinding is a costly process, the magnetic values must be recovered at as coarse a size as possible. Too coarse a grind would not yield satisfactory results as hoped for. On the other hand, over grinding will generate considerable amount of slimes which ultimately make the separation ineffective. Keeping these factors in view, a number of wet magnetic separation tests were conducted to arrive at the optimum particle size and the results are recorded in Table 2.27

The results indicated that 85% - 200 mesh grind gave the best results. The concentrate assayed 52.5% Fe and represented a recovery of 94.7% Fe. The tests employing still finer grinds indicated some loss of Fe in the tails due to generation of excessive slime in them. Hence it was concluded a grind of 85% - 200 mesh is optimum for this ore.

The optimum pulp density for the continuous magnetic separation was established. Samples of 500 kg. each of the reduced ore ground to 85% - 200 mesh were subjected to wet magnetic separation at 25, 30, 40, 50 and 60% solids employing drum type wet magnetic separator. The results are recorded in Table 2.28.

The results indicated that magnetic separation with 40% pulp density gave the best performance. Hence for large scale operation 40% pulp was kept as the optimum value.

Large Scale Operation - The magnetising reduction roast product was continuously ground in a ball mill and the pulp was passed through the three drum type wet magnetic separators kept in series. The log data of the grinding performance is recorded in Table 2.29

The sieve analysis of ball mill discharge, classifier overflow and classifier underflow are recorded in Table 2.30

The ball mill discharge was collected in a 4.88 m x 2.44 m thickener and the thickener underflow containing about 40% solids was pumped to the first magnetic separator, the tailing from the same went to the second magnetic separator and the tailing from the second went to the third. The concentrates from each of the mag-

[illegible]

Table 2.27 - Magnetic separation employing different grinds

Test No.	Grind % -200 mesh	Product	Wt. %	Assay %		Dist. %
				Fe	SiO$_2$	Fe
1.	45.0	Magnetic	35.8	49.8	14.5	42.9
		Non mag.	64.2	37.0	-	57.1
		Head (Reduced ore)	100.0	41.57		100.0
2.	60.0	Magnetic	43.5	50.1	14.2	52.1
		Non-mag.	56.5	35.4		47.9
		Head (Reduced ore)	100.0	41.8		100.0
3.	70.0	Magnetic	63.8	50.9	13.3	66.8
		Non-mag.	36.2	25.0		33.2
		Head (Reduced ore)	100.0		21.1	100.0
4.	85.0	Magnetic	77.4	52.5	11.1	94.7
		Non-mag.	22.6	10.5		5.3
		Head (Reduced ore)	100.0	42.9	17.94	100.0
5.	90.8	Magnetic	73.2	53.1	10.7	91.0
		Non-mag.	26.8	14.3		9.0
		Head (Reduced ore)	100.0	42.7		100.0
6.	96.0	Magnetic	72.8	53.3	10.5	90.5
		Non-mag.	27.2	14.9		9.5
		Head (Reduced ore)	100.0	42.8		100.0
7.	98.5	Magnetic	73.1	52.8	11.0	90.4
		Non-mag.	26.9	15.1		9.6
		Head (Reduced ore)	100.0	92.7		100.0

Table 2.28 - Magnetic separation at different pulp densities

Test No.	Grind % - 200 mesh	% Solids in the pulp	Product	Wt. %	Assay %		Dist. %	
					Fe	SiO_2	Fe	SiO_2
1.	85.0	25	Magnetic	46.3	48.5	16.6	52.8	39.5
			Non-mag.	53.7	37.2	22.6	47.2	60.5
			Head (Reduced ore)	100.0	42.5	19.9	100.0	100.0
2.	"	30	Magnetic	53.8	49.7	15.1	63.8	40.7
			Non-mag.	46.2	32.8	25.8	36.2	59.3
			Head (Reduced ore)	100.0	41.9	20.0	100.0	100.0
3.	"	40	Magnetic	75.8	51.8	12.7	93.3	55.0
			Non-mag.	24.2	11.7	35.3	6.7	45.0
			Head (Reduced ore)	100.0	42.10	18.0	100.0	100.0
4.	"	50	Magnetic	70.3	52.3	12.1	88.7	46.6
			Non-mag.	29.7	15.8	32.8	11.3	53.4
			Head (Reduced ore)	100.0	41.5	18.2	100.0	100.0
5.	"	60	Magnetic	64.6	49.7	15.3	75.7	51.4
			Non-mag.	35.4	29.0	26.4	24.3	48.6
			Head (Reduced ore)	100.0	42.4	19.2	100.0	100.0

Table 2.29 - Log data for grinding

Ball mill size	1.829 m x 0.914 m
Ball charge	1400 kg
Feed rate (through continuous weight feedometer)	500 kg/hr
Power consumption	11 KWH/tonne
Ball consumption	0.35 kg/tonne
Pulp density	22.0% solids
Circulating load	165%

Table 2.30 - Sieve analyses of different products

Size in mesh	Ball Mill discharge	Classifier overflow	Classifier underflow
+48	18.3	-	30.0
-48+65	10.8	0.6	18.0
-65+100	12.1	2.4	10.0
-100+150	9.3	3.7	15.0
-150+200	9.2	7.4	19.0
-200	40.3	85.9	8.0
	100.0	100.0	100.0

netic separator were collected separately, assayed and finally mixed up and was sent to the dryer. The final tailing was collected, assayed and was finally disposed off. The material balance around the magnetic separator is recorded in Table 2.31.

Table 2.31 - Material balance around the magnetic separators

Basis = 15 seconds
Thickener underflow (40% solid)
9.18 kg pulp
(3.67 kg solid
5.51 kg water)

Water 9.0 kg

1st Mag. Separator
Drum type, size 10.

1st Mag. tails
12.88 kg pulp (8.57% solid)

1st Mag. Conc. 5.3 kg (50% solids)
(2.65 kg solid
2.65 kg water)

(1.02 kg solid
11.86 kg water)

Water 5.58 kg

2nd Mag. Separator
Drum type size 9.14 x 5.5 m

2nd Mag. tails
15.46 kg pulp (5.76% solid)

2nd Mag. Conc.
3.0 kg pulp (6.0% solid)

(0.84 kg solid
14.62 kg water)

3.73 kg water

3rd Mag. Separator
Drum type size 9.14 x 5.5 m

Final tails
17.88 kg pulp (4.1% solid)

3rd Mag. Conc.
1.31 kg pulp (4.6% solid)

(0.78 kg solid
17.10 kg water)

(0.06 kg solid
1.25 kg water)

Product	Wt. in kg.	Wt. %	Assay % Fe	Dist. Fe
1st Mag. Conc.	2.65	72.3	52.0	87.3
2nd Mag. Conc.	0.18	4.9	51.4	5.8
3rd Mag. Conc.	0.06	1.6	50.0	1.9
Final tailing	0.78	21.2	10.3	5.0
	3.67	100.0	43.1	100.0

The results indicated that 78.8% by weight of the iron values could be taken out from the reduced product by magnetic separation in three stages with a recovery of 95.0% Fe.

The daily operational data of the grinding followed by magnetic separation are recorded in Table 2.32.

Table 2.32 - Operational data on grinding and magnetic separation

Day	Feed rate kg/hr	Grind % -200 mesh	Magnetic Product	Separation Assay % Fe	Separation Assay % SiO_2
	500	85.6	Mag 1	51.5	12.1
			Mag 2	50.6	13.2
			Mag 3	49.8	13.7
			Non-mag	13.2	35.6
	505	85.3	Mag 1	52.1	11.7
			Mag 2	51.7	12.3
			Mag 3	51.3	12.6
			Non-mag	14.1	36.1
	510	87.2	Mag 1	52.0	12.0
			Mag 2	57.6	12.4
			Mag 3	51.1	12.7
			Non-mag	12.7	34.8
	506	85.6	Mag 1	51.3	13.1
			Mag 2	50.9	14.6
			Mag 3	50.3	14.7
			Non-mag	14.0	32.3
	505	85.8	Mag 1	52.6	11.3
			Mag 2	51.8	12.3
			Mag 3	51.1	12.7
			Non-mag	11.2	36.6
	507	86.2	Mag 1	52.8	11.1
			Mag 2	51.4	12.4
			Mag 3	50.9	12.6
			Non-mag	10.8	37.7
	508	85.7	Mag 1	52.5	12.0
			Mag 2	51.6	12.6
			Mag 3	51.5	13.0
			Non-mag	11.4	35.7
	508	85.8	Mag 1	52.1	12.2
			Mag 2	51.7	13.1
			Mag 3	50.9	13.4
			Non-mag	13.1	31.6

Day	Feed rate kg/hr	Grind % -200 mesh	Magnetic Product	Separation Assay %	
				Fe	SiO_2
	505	85.0	Mag 1	51.8	13.1
			Mag 2	51.2	12.8
			Mag 3	50.8	12.9
			Non-mag	12.2	33.2
	508	85.7	Mag 1	52.3	11.1
			Mag 2	51.9	11.7
			Mag 3	51.3	12.4
			Non-mag	10.7	36.5
	507	86.2	Mag 1	52.7	11.8
			Mag 2	51.9	12.0
			Mag 3	51.3	12.4
			Non-mag	11.2	38.1
	510	86.5	Mag 1	52.8	11.0
			Mag 2	51.8	12.1
			Mag 3	51.0	12.5
			Non-mag	10.4	36.2
	502	86.3	Mag 1	53.0	10.9
			Mag 2	51.9	12.2
			Mag 3	51.7	12.4
			Non-mag	10.3	37.1
	505	85.6	Mag 1	52.8	11.3
			Mag 2	52.1	12.0
			Mag 3	51.8	13.2
			Non-mag	10.4	37.0
	503	85.8	Mag 1	52.0	11.7
			Mag 2	51.5	12.0
			Mag 3	50.9	12.1
			Non-mag	14.2	33.3
	500	85.7	Mag 1	52.7	10.9
			Mag 2	51.9	12.0
			Mag 3	51.3	12.7
			Non-mag	11.5	32.9
	505	86.0	Mag 1	51.9	12.0
			Mag 2	51.4	12.4
			Mag 3	51.0	12.7
			Non-mag	11.0	32.6
	510	86.5	Mag 1	52.4	11.2
			Mag 2	52.0	11.9
			Mag 3	51.6	12.4
			Non-mag	10.3	36.3

The results obtained in the continuous run of the plant indicated that the magnetic separation consistently gave reasonable yield of iron values. The pulp density throughout the entire operation was kept between 38 to 42% solid and care was taken to keep same constant as nearly as possible. The concentrate was collected in tanks from which it was fed to the dryer to remove the moisture content and to render the same suitable for pelletization.

Drying of Magnetic Concentrates The magnetic concentrates collected separately from the three magnetic separators working in series were mixed up. The final moisture content of the same when collected in the settling tanks varied from 18 to 25%. This material had to be dried before pelletisation upto 4-5% water. The mixed magnetic concentrate was demagnetised and thickned to 50 to 60% solids, in a thickener. The thickened conc. was dired to 5 to 6% in rotary driers and used for pelletisation tests.

Description of the Rotary Dryer The rotary dryer used for the purpose was of direct type, the heating gas was passed counter-current with the flow of the material. It was the simplest and most commonly used single shell rotary dryer. There was no brick lining inside the shell but was equipped with flights on the interior surface of the shell for lifting and showering the wet material through the hot gases during passage through the cylinder. The flight extended continuously the entire length of the dryer and the shape of the flights was radial with a 90 degree top. The gas burner was fitted at the discharge end of the burner working on producer gas and air in proper proportion. A blower was fitted at the feed end so as to give a positive draft on the flue gases through the chimney. The wet material was fed through the feed hopper which discharged the material at the feed end of the dryer through a launder. The log data of the dryer is recorded in Table 2. 33.

The dryer was operated continuously with an average product of 1. 57 tonnes of dry product per day of 24 hours. The dried material due to effect in the drier produced balls of irregular size and shape and the same had to be passed through a flupper before pelletization. The concentrate was thereafter stored in a storage bin for further treatment for pelletizing. The average chemical analysis of the dry product is recorded in Table 2. 34.

Table 2.33 - **Log** data of the dryer

Moisture in the feed	Average 23%
Product moisture	Average 6%
Feed rate	80 kg/hr (wet material)
Retention time	20 minutes
R.P.M.	7.5
Inclination	4^{o}
Length	4.57 m
Dia	0.46 m
Temperature	
a) at the discharge end	350^{o}C
b) of the flue gas	110^{o}C
Gas consumption	71 m^3/hr
a) Primary	10.8 m^3/hr
b) Secondary	35.4 m^3/hr
Peripheral speed.	10.6 meter/minute.

Table 2.34 - Chemical **Analysis** of the concentrate

Constituent	Percent
Fe	51.7
FeO	15.84
TiO_2	0.34
MnO	Trace
SiO_2	11.30
Al_2O_3	5.23
CaO	4.65
MgO	1.82
P	1.03
S	0.16
L.O.I.	5.11

CHAPTER 3

STUDIES ON PHYSICAL & PHYSICO-CHEMICAL PROPERTIES

In addition to normal working data for the study of physical & physico-chemical properties, the following tests have been carried out.

1. Determination of Bond's work index in closed circuit for magnetic concentrate from magnetising reduction roast tests.
2. Determination of the settling rates of (a) magnetic concentrate, (b) non-magnetic tails.
3. Determination of the rate of filtration of the magnetic and non-magnetic products.

1. *Determination of Bond's Work Index*

The magnetising reduction roast product of the Asswan iron ore constituted the pelletisation feed. The Bond's work index for grinding the same to 85% -200 mesh in closed circuit at a circulating load of 250% was determined, as this was found to be the optimum grind for subsequent magnetic separation treatment to yield the maximum Fe recovery.

The results indicated that the Bond's work index value of the sample was 11.05 KWH/tonne for dry grinding and 9.19 KWH/tonne for wet grinding.

The object of determining the work index value is to obtain the data useful for determining the size of the ball mill and the power requirements for grinding the sample to the desired fineness. The work index value represents the power in KWH/tonne required to reduce the test sample from infinite size to a product 80% of which passes through 100 microns. The actual power input required to grind from a given feed size to the desired product size can be calculated from Bond's equation (1961).

$$W = 10\ W_i\ (p^{-0.5} - F^{-0.5})$$

where

W - Power in KWH tonne required to reduce the test sample from size 'F' to 'P'.

W_1 = Work index in KWH tonne required to reduce the sample from infinite size to a product 80% of which passes through 100 microns.

P = Size in microns through which 80% of the desired product passes.

F = Size in microns through which 80% of the feed for crushing and grinding passes.

Evidently, the power consumption is dependent on the feed and product sizes, wet or dry grinding, close or open circuit grinding etc. Wet grinding normally consumes about 25 to 30% less power than dry grinding carried out under similar conditions.

For the purpose of determining Bond's work index, the -3/4" magnetising reduction roast product was crushed to -6 mesh in closed circuit using a roll crusher.

Test procedure: The test procedure developed by F. C. Bond of Allis-Chalmers was adopted for the determination of work index which consisted of the following stages.

i) Measurement of the size in microns through which 80% of the feed sample 'F' passes.

ii) 700 cc of the representative test sample was weighed and ground in a ball mill of size 30.5 cm x 30.5 cm rotating at 64 r.p.m. with a ball charge of 20.187 kg.

iii) The sample is ground for a known number of revolution 'N'.

iv) The ground sample was removed and screened through a specified size P_1 microns (270 mesh screen was used in the test).

v) The screen undersize was weighed from which the quantity finer than P_1 size originally present in the sample, was deducted to get the net undersize produced by grinding for 'N' revolution. From this, the net grams produce per revolution. From this, the net grams produce per revolution was calculated.

vi) A second feed sample from the ball mill was prepared by combining the screen undersize with a fresh test sample of the same quantity as the screen undersize.

vii) The grinding and screening operation were repeated until the desired circulating load of 250% was achieved.

viii) This was repeated twice or thrice at the desired circulating load and the amount of net product finer than P_1 size produced due to grinding was determined from which the average quantity of fines produced per revolution (Gbp) was calculated from the last two or three grinds.

ix) The final products obtained during the last three grinds were screen analysed from which the size in microns through which 80% of the product 'P' passed was determined.

The work index of the sample was calculated by using Bonds' revised equation (1961):-

$$\text{Work Index (Wi)} = \frac{44.1 \times 1.1}{(P_1)^{0.23} \times (Gbp)^{0.82} \times \left(\frac{10}{P^{1/2}} - \frac{10}{F^{1/2}}\right)}$$

where,

W_i = Work index in KWH/metric tonne of ore.

P_1 = Screen size in microns, used for screening the ground product.

Gbp = Grindability in net grams produced due to grinding per revolution of the mill.

P = Size in microns through which 80% of the ground product passes.

F = Size in microns through which 80% of the feed passes.

Work index of the sample: The test was conducted with the procedure mentioned above and the results are tabulated in Tables 3.1 and 3.2.

Results

Grindability (gm/revolution), Gbp Gbp = 1.83

Feed size in microns (F) = 1780

Product size in microns (P) = 65

Bond's work index in KWH/tonne for grinding = 11.05

Work index value for wet grinding (25% less) = 9.19

Table 3.1 - Determination of Gbp at 250% circulating load on 200 mesh screen

Wt. of 700 ml of -6 mesh feed = 975 gms.
Feed size = 1780 microns.
Wt. % of -200 mesh in the feed = 8.5

Time in Secs.	**No. of mill revolution**	**Gms. of return + 200 mesh**	Gms. of -10 mesh fresh feed	Gms. of -200 mesh in the product	Gms. of -200 mesh in the feed	Net gms. -200 mesh produced	Gbp Net gms. of -200 mesh produced per revolution	% Circulating load
60	**64**	-	475	190	83	107	1.67	413.2
90	**96**	785	190	195	16	179	1.89	400.0
120	**128**	**780**	195	255	17	238	1.86	282.3
130	138.7	720	255	275	22	253	1.82	254.5
130	138.7	700	275	217	23	254	1.83	258.0
130	138.7	**698**	277	**280**	**24**	256	1.84	248.2

Table 3.2 - Screen analysis of the ball mill feed and ground product

Mesh size	Ball mill feed Wt. % retained	 Cumulative Wt. % passing	200 mesh ground product at 250% circulating load Size	 Wt %	 Cumulative passing
+8 mesh	17.1	82.9	+250 mesh	26.5	73.5
+10 mesh	16.2	66.7	+270 "	1.5	72.0
+14 "	12.0	54.7	+325 "	14.0	58.0
+20 "	9.4	45.3	-325 "	58.0	
				100.0	
+28 mesh	8.5	36.8			
+35 "	7.7	29.1			
+48 "	5.6	23.5			
+65 "	4.3	19.2			
+100 "	4.7	14.5			
+150 "	4.7	9.8			
+200 "	1.3	8.5			
-200 "	8.5				
	100.0				

Size in microns at which 80% passes.

Ball mill feed = 1780 microns
Ground product = 65

2. *Determination of the Settling Rates of Ground Products*

Settling rate tests were performed with the classifier overflow for magnetic separation, magnetic concentrate which would be utilised for pelletisation and non-magnetic tailings which would be rejected after reclamation of water. The tests were carried out separately with different percentages of solids viz., 5, 10, 15, 20, 25 and 30% solids without using any flocculant.

The effect of these tests are to obtain data useful for calculating thickener, size, capacity etc. required to collect the ground ore.

The tests were conducted separately in 500 ml. graduated measuring cylinder. After bringing the solid in suspension the settling rate was recorded every minute for the first few minutes so long as there was free settling.

The ultimate pulp density after allowing the solids to settle for 19 hours was also recorded in each case, which would indicate the maximum per cent solid in the underflow that could normally be expected from a thickener of standard depth.

Table 3.3 - **Results** of rate of settling tests with magnetic fraction after reduction roast and grinding

% Solids in the feed	5	10	15	20	25	30
Liquid to solid ratio in the feed (F)	19.0	9.0	5.67	4.0	3.0	2.33
Average rate of settling cm/min.	6.02	3.05	2.50	2.36	2.06	1.85
Average rate of settling Fe/hr (R)	11.85	6.04	4.92	4.64	4.06	3.05
Ultimate pulp density %Solid (after 19 hours)	46.3	51.0	51.4	54.4	56.5	60.6
Liquid to solid ratio at ultimate pulp density (D)	1.16	0.96	0.95	0.78	0.77	0.65
Settling rate per tonne in sq. ft. = $1.33 \frac{(F-D)}{R}$	1.71	1.51	1.09	0.79	0.62	0.62
Settling area required at 125% of above in sq. ft. /ton	2.14	2.0	1.36	0.99	0.78	0.78
Settling area required in Sq. meter/tonne	0.20	0.19	0.12	0.09	0.07	0.07

The results of settling tests are recorded in Tables 3.3, 3.4, and 3.5.

From the results it could be seen that the rate of settling was quite fast in the case of the magnetic concentrate and the use of any flocculant may not be necessary.

3. *Determination of Filtration Rates*

Filtration tests with the magnetic concentrate and non-magnetic tailings were carried out with a view to investigate the filtering characteristics of the products. The data obtained by these tests will be useful for designing a filter suitable for dewatering the thickener underflow.

Test procedure: Filtration tests were carried out using a standard EIMCO test filter disc, having an effective grid face area of .0093 sq. metre. The filter was secured in position on the leaf with an adjustable clamp. The filter leaf was connected to a filtrate receiver through a flexible hose pipe which in turn was connected to a vacuum pump.

Table 3.4 - Results of rate of settling with non-magnetic fraction obtained after reduction roast and grinding

% Solid in the feed	5	10	15	20	25	30
Liquid to solid ratio in the feed (F)	19.0	9.0	5.67	4.0	3.0	2.33
Average rate of settling cm/min.	3.0	1.85	1.64	1.16	0.76	0.49
Average rate of settling ft/hr (R)	5.9	3.64	3.26	2.28	1.50	0.96
Ultimate pulp density %Solid (after 19 hrs)	42.8	43.85	48.4	52.1	53.1	54.8
Liquid to solid ratio of ultimate pulp density (D)	1.34	1.18	1.07	0.92	0.90	0.82
Settling area per ton in sq. ft. = $1.33 \frac{(F-D)}{R}$	3.40	2.43	1.60	1.53	2.19	1.78
Settling area required at 125% of above in sq. ft./ton	4.25	3.01	2.0	1.91	2.74	2.22
Settling area required in sq. meter/tonne	0.40	0.28	0.19	1.18	0.25	0.21

The filter disc. under vacuum (560 mm of Hg) was sub-merged for a pre-determined time in a well agitated pulp of the sample for cake formation and afterwards, was taken out of the pulp to allow the wet cake to dewater itself for the same time under the same vacuum. The cake was then blown out and moisture content, thickness of the cake and wet weight of the filter cake was determined. The quantity of the filtrate collected was also measured to determine the rate of filtrate collection. Thus the cycle of operation comprised of submergence, drying and blowing. The duration of submergence, and drying was 37.5% each of the total cycle time and the remaining 25% was kept as blowing time.

The tests were conducted with solids percentage of 40, 50 and 60 without any flocculant. The cycle time was also varied in each case i.e. 40, 80, and 120 seconds. The results obtained are recorded in Tables 3.6 and 3.7.

Table 3.5 - Results of rate of settling with classifier overflow after reduction roast and grinding

% Solids in the feed	20	25	30
Liquid to solid ratio in the feed (F)	4.0	3.0	2.33
Average settling rate cm/min.	1.8	1.5	1.2
Average rate of settling ft./hr. (R)	3.54	2.95	2.36
Ultimate pulp density % solid (after 19 hrs)	64.0	63.4	61.0
Liquid to solid ratio at ultimate pulp density (D)	0.56	0.58	0.64
Settling area per ton in sq. ft. $= 1.33 \frac{(F-D)}{R}$	2.62	1.17	0.81
Settling area required at 125% of above in sq. ft./ton	3.28	1.46	1.01
Settling area required in sq. meter/tonne	0.31	0.14	0.094

Decrepitation Characteristics

Since iron ore used in Blast Furnace is normally of size -50 to + 12 mm, the lumpy ores (-150 + 50 mm) were crushed and screened to get -40 to + 12 mm size fraction which was used for the test.

During the descent in the stack of the blast furnace, the burden is subjected to progressive increasing temperature, due to the exchange of heat from the ascending gases. Iron ores decrepitate within a certain temperature zone and bigger lumps disintegrate into smaller ones and thereby reduce the voids in the burden. During actual smelting of ore in the furnace, decrepitation occurs in different atmospheres of gas composition and under the load of the burden.

Table 3.6 - Filtration test with magnetic concentrate obtained after reduction roast and grinding

% solids in the pulp	Total cycle time in secs.	Thickness of the cake in cms.	% moisture in cake	Filtrate Qty. in lit/hr/sq.m.	Dry cake formation rate in kgs/hr/sq.m.
40	40	0.20	27.5	1119.0	483.9
	80	0.40	27.0	1138.0	353.2
	120	0.60	26.0	910.0	300.0
50	40	0.80	27.0	1208.0	1239.0
	80	1.20	26.5	822.4	1152.0
	120	1.50	25.5	629.0	850.0
60	40	1.10	26.0	503.1	2322.0
	80	1.50	25.0	324.4	1611.0
	120	2.00	24.0	322.5	1648.0

Table 3.7 - Filtration test with non-magnetic tailing obtained from reduction roast and grinding

% solids in the pulp	Total cycle time in secs.	Thickness of cake in cms.	% moisture in cake	Filtration Qty. in lit/hr/sq.m.	Dry cake formation rate in kgs/hr/sq.m
60	40	2.0	24.5	967.7	4074.0
	80	2.7	27.5	653.1	2724.0
	120	3.1	23.0	548.3	2329.0
50	40	0.9	24.0	890.2	1809.0
	80	1.5	24.0	696.8	1341.0
	120	2.0	24.0	580.6	1090.0
40	40	0.50	24.0	1161.0	871.0
	80	0.65	24.5	735.4	542.0
	120	0.90	26.0	613.0	527.8

The decrepitation test simulates the effect of time, temperature and furnace atmosphere on iron-bearing materials during their descent in the stack of the blast furnace, and may probably indicate the degradation due to these factors.

In the laboratory test, the decrepitation characteristics are determined separately under stagnant atmospheres, and under atmospheres of CO, CO_2, N_2 but without any superimposed load.

Procedure

The iron ore sample was placed inside a stainless steel pot having a perforated bottom, which was suspended inside another stainless steel tube and was heated in an electrically operated pot furnace. The temperature of the ore sample was continuously increased at a constant rate of heating from room temperature to 1000^oC. (In case of test under CO, CO_2, N_2 atmospheres, the gas was passed through the sample and continued upto 1000°C). During the rise of temperature, an audible clicking sound was heard. This represented the temperature of decrepitation, which was noted in each test. Three tests were conducted for each sample to take account of any possible lump to lump variation in the characteristics of the ore. After the test, the sample was screened though various sieves to determine the extent of size degradation due to decrepitation and the material thereafter was subjected to shatter test in order to determine its physical strength.

Discussion of the Test Results

The test results of Asswan iron ore under stagnant atmosphere and under CO, CO_2, N_2 atmosphere (CO.CO_2 ratio in the gas being approx. 2.0) along with its shatter strength data are recorded in Tables 3.8 and 3.9.

Decrepitation temperature varied from 380° to 400°C and the shatter strength on + 12 mm varied from 92.21% to 93.60% in case of test carried out in stagnant atmosphere (Table 3.8). Whereas in case of test conducted under CO, CO_2, N_2 atmosphere, the decrepitation temperature varied from 350-360°C and the shatter strength on +12 mm varied from 70.55% to 72.32% (Table 3.9).

Conclusions

1. The decrepitation temperature of the ore is lower in CO, CO_2, N_2 atmospheres than in stagnant atmospheres.

2. Shatter strength of decrepitated sample is also lower in CO, CO_2, N_2 atmospheres.

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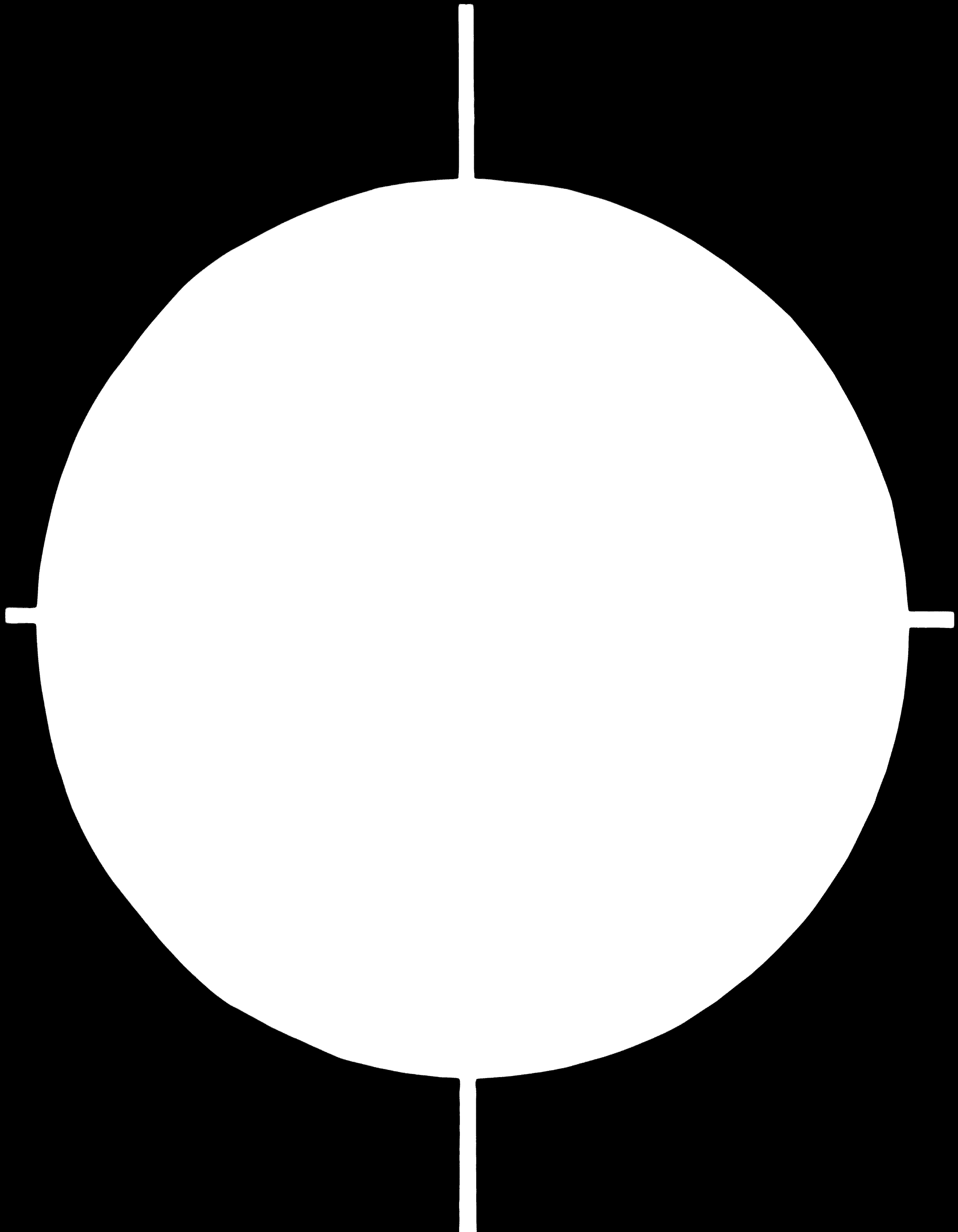

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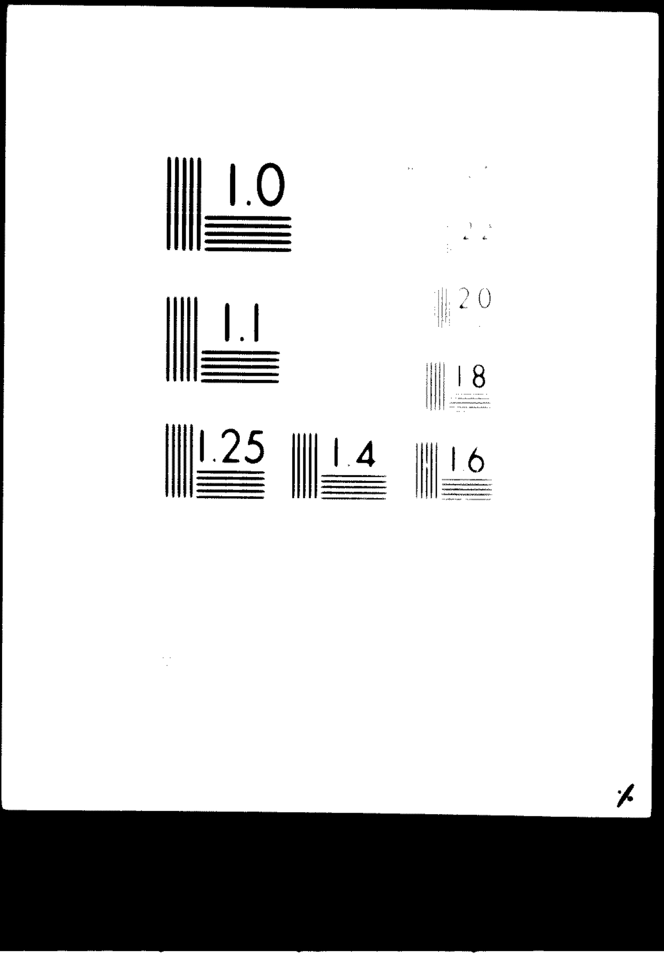

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Table 3.8 - Decrepitation test results under stagnant atmospheres and shatter strength after the test

Expt. No.	Decrepitation Temp. °C	Screen analysis after test mm %					Screen analysis after shatter test mm %				
		-40 +25	-25 +12	-12 + 6	-6 +3	-3	-40 +25	-25 +12	-12 + 6	-6 +3	-3
1	380	63.32	32.74	3.32	0.41	0.21	51.70	40.51	6.00	1.05	0.74
2	400	55.69	41.62	1.66	0.62	0.41	51.83	41.77	4.05	1.50	0.85
3	390	66.80	30.70	1.80	0.40	0.30	49.72	43.28	5.00	1.30	0.70

Table 3.9 - Decrepitation test results under CO, CO_2, N_2 atmosphere and shatter strength after the test

Expt. No.	Decrepitation Temp. °C	Screen analysis after the test mm %					Screen analysis after shatter test mm %				
		-40 +25	-25 +12	-12 + 6	-6 +3	-3	-40 +25	-25 +12	-12 + 6	-6 +3	-3
1	350	47.6	44.9	4.2	1.5	1.8	34.60	36.87	21.43	4.60	2.50
2	360	56.6	36.3	6.4	0.4	0.3	30.00	40.55	18.43	6.83	4.19
3	*N.A.	53.1	39.6	5.0	0.85	1.45	32.70	39.62	18.60	5.52	3.56

*N.A. = Not audible

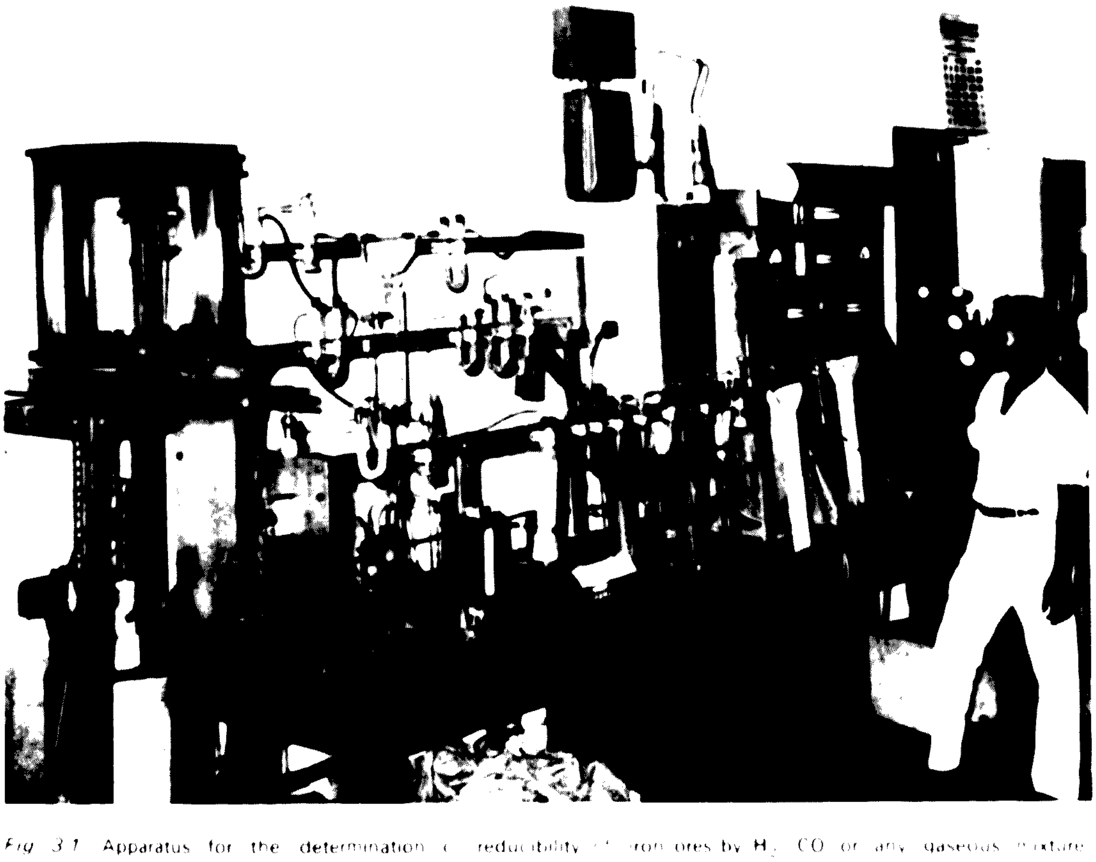


Fig. 3.1 Apparatus for the determination of reducibility of iron ores by H_2, CO or any gaseous mixture

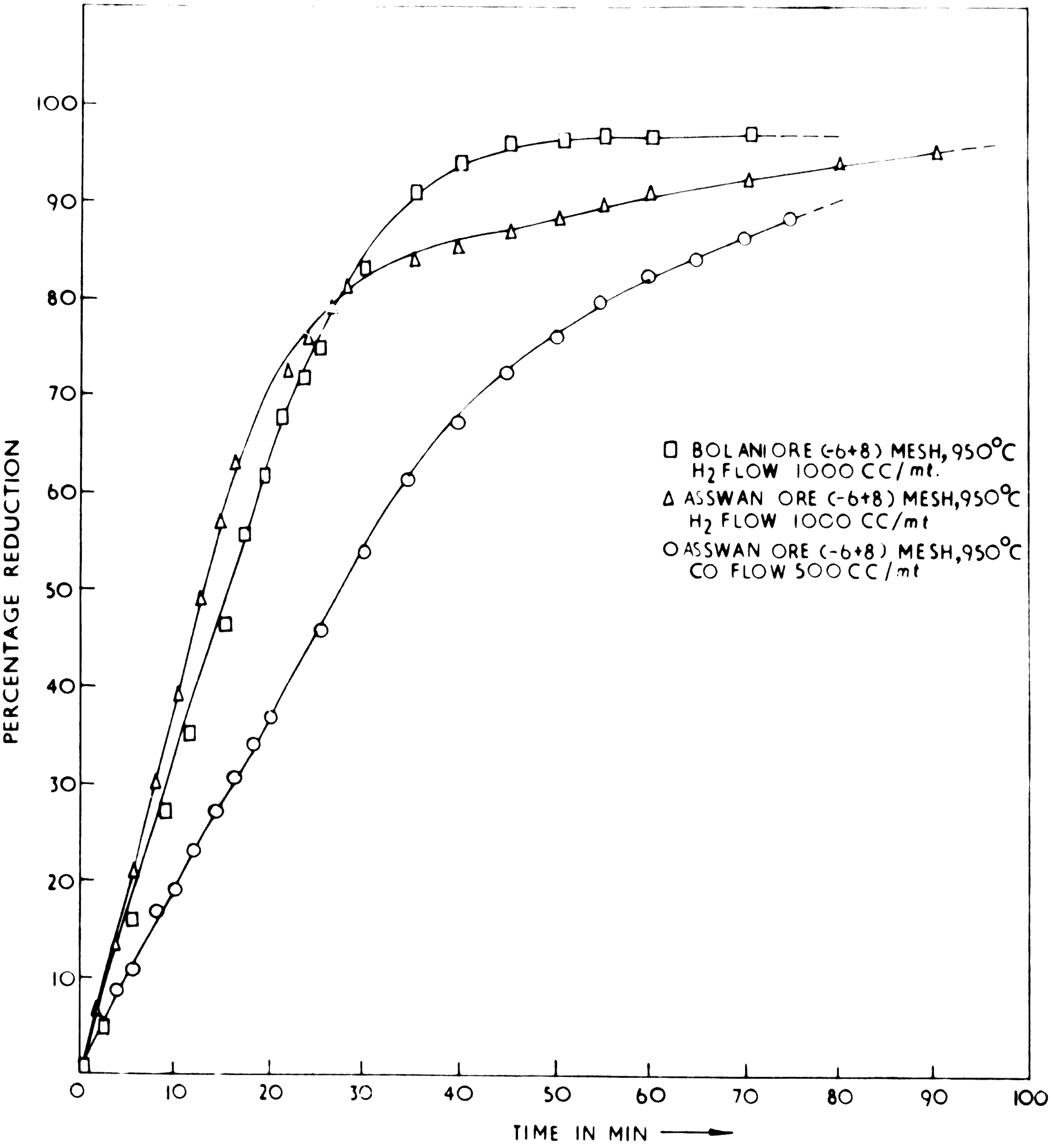


Fig. 3.2 Comparison of Asswan and Bolani iron ore reducibilities at 950 C with H_2 and CO

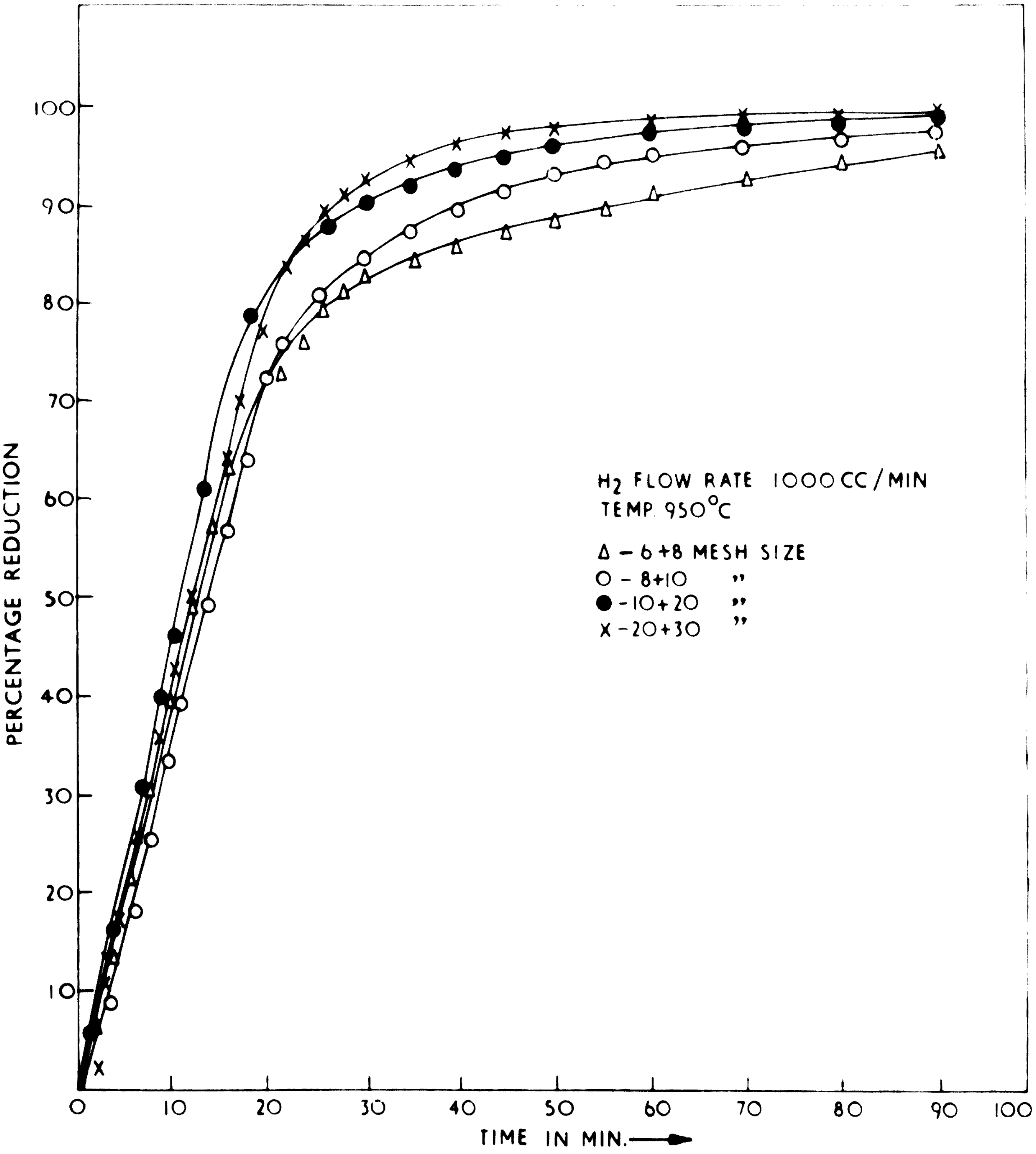


Fig. 3.3 Reduction behaviour of Asswan iron ore with various mesh sizes at 950 C and 1000cc mt H_2 flow rate

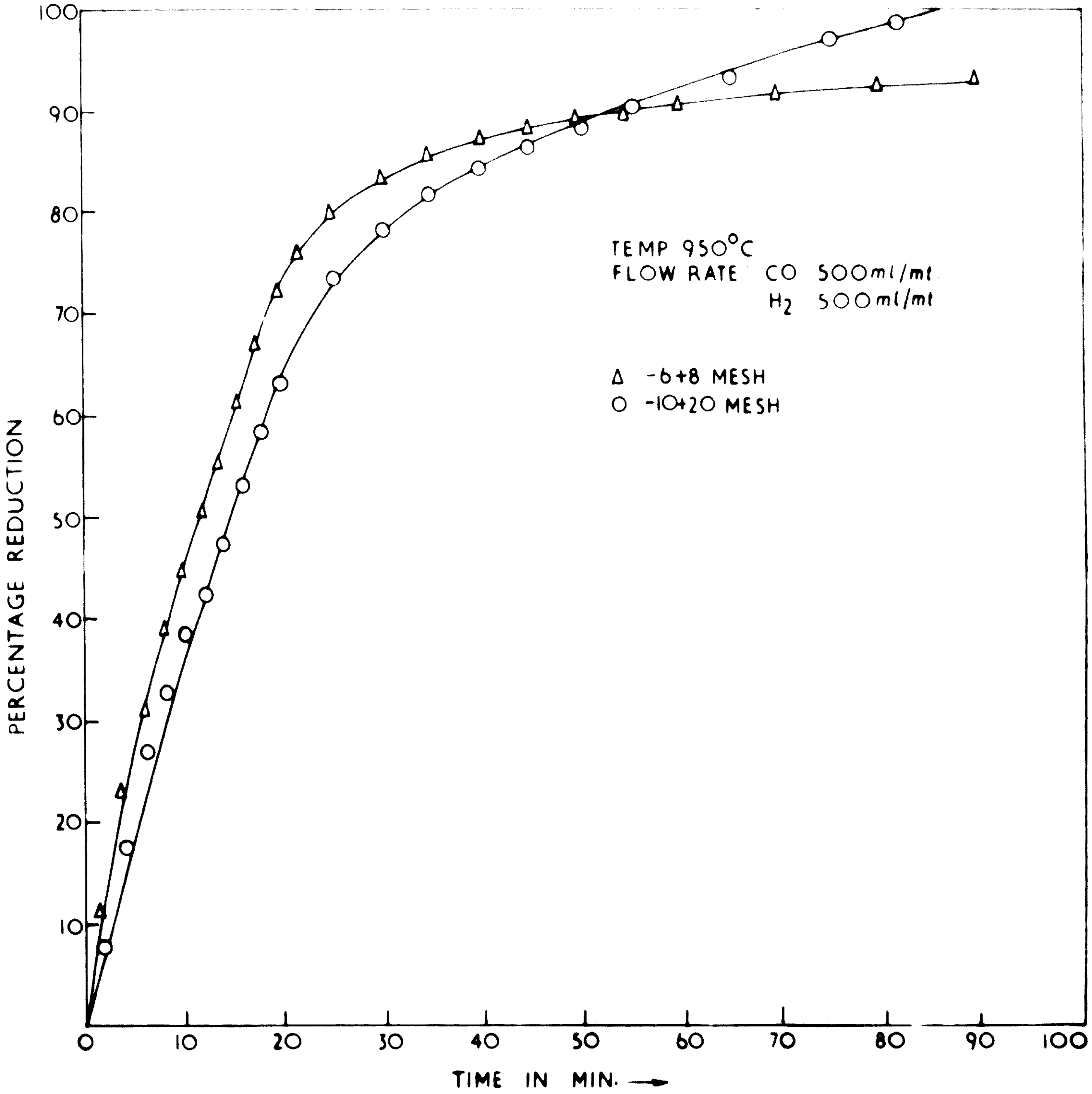


Fig. 3.4 Asswan ore reduction by H_2, CO mixture

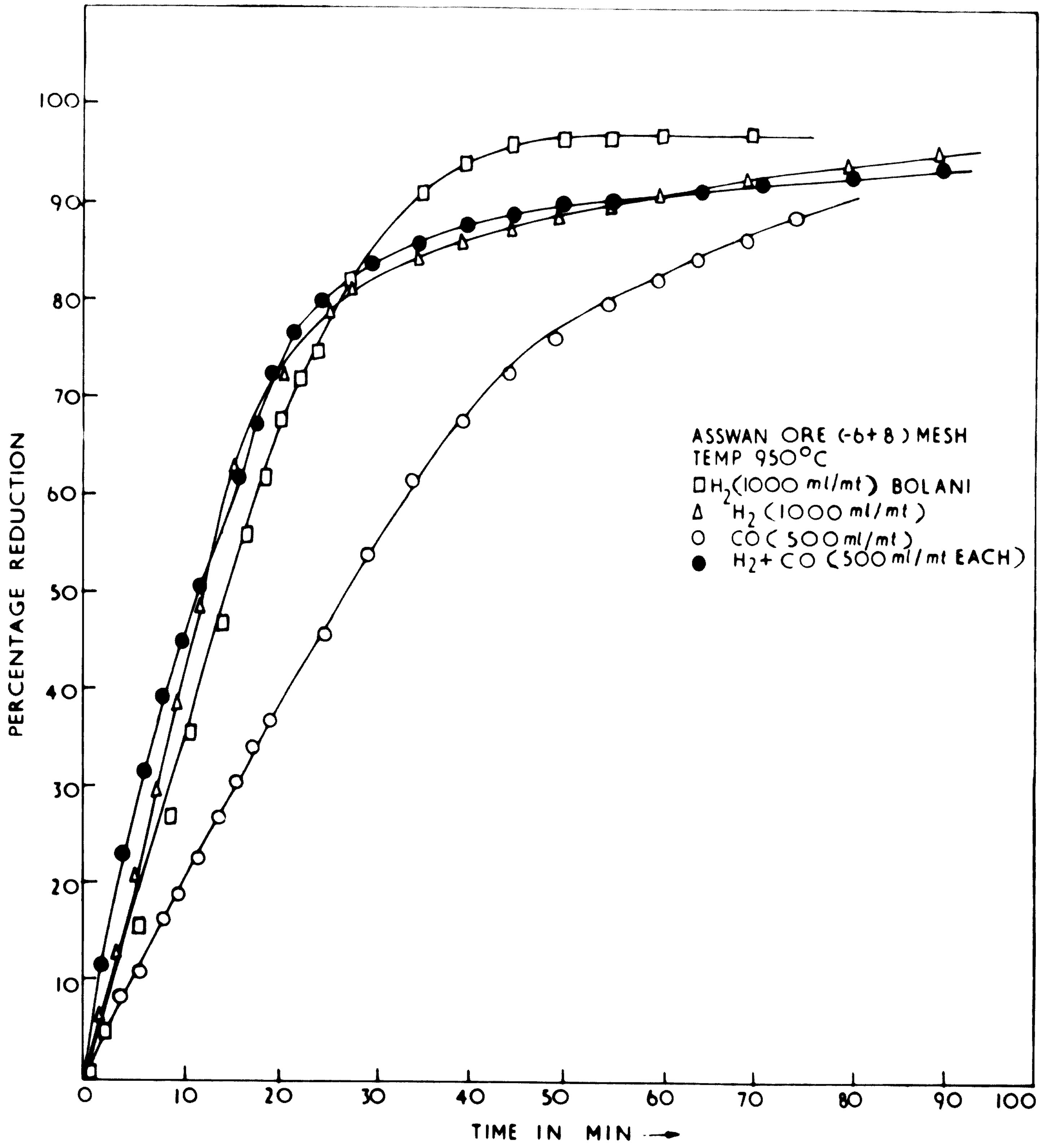


Fig 3.5 Reduction behaviour of Asswan ore (-6 +8) mesh size at 950°C with various reductants compared with Bolani ore

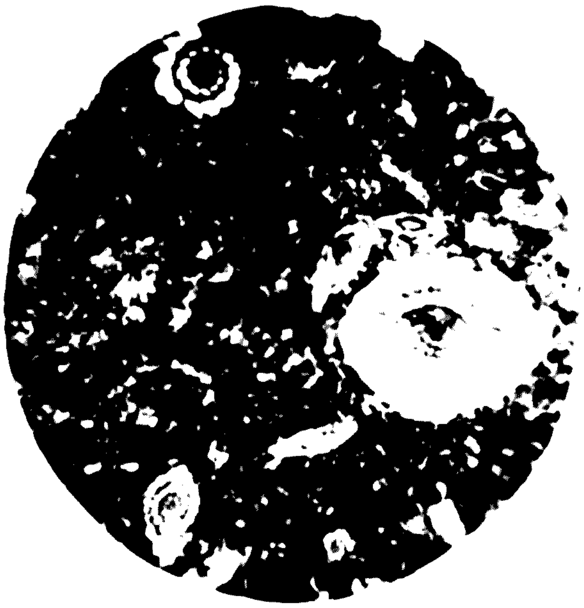


Fig. 3.6. Thin section of Asswan limestone showing microfossils in finegrained calcite matrix. Transmitted light 19

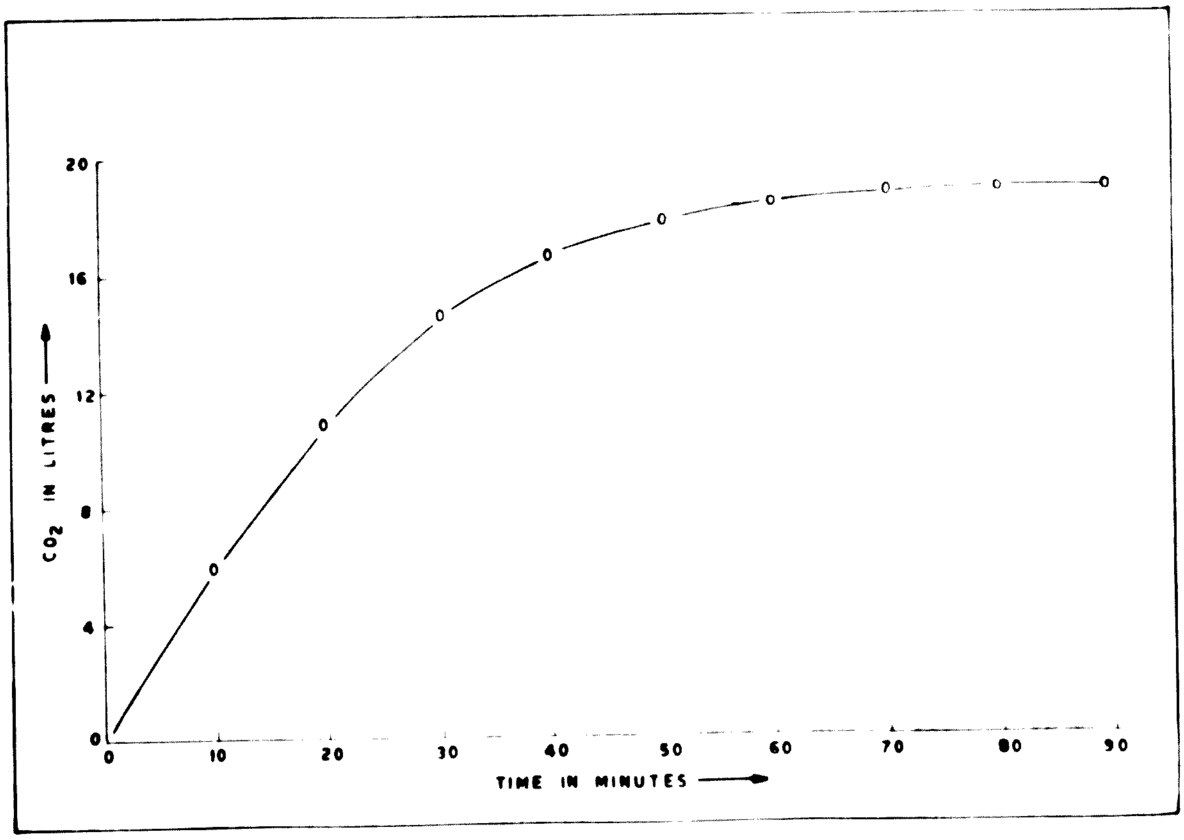


Fig. 37 CO_2 evolution at 930°C against time for size 25 - 12 mm Asswan lime stone

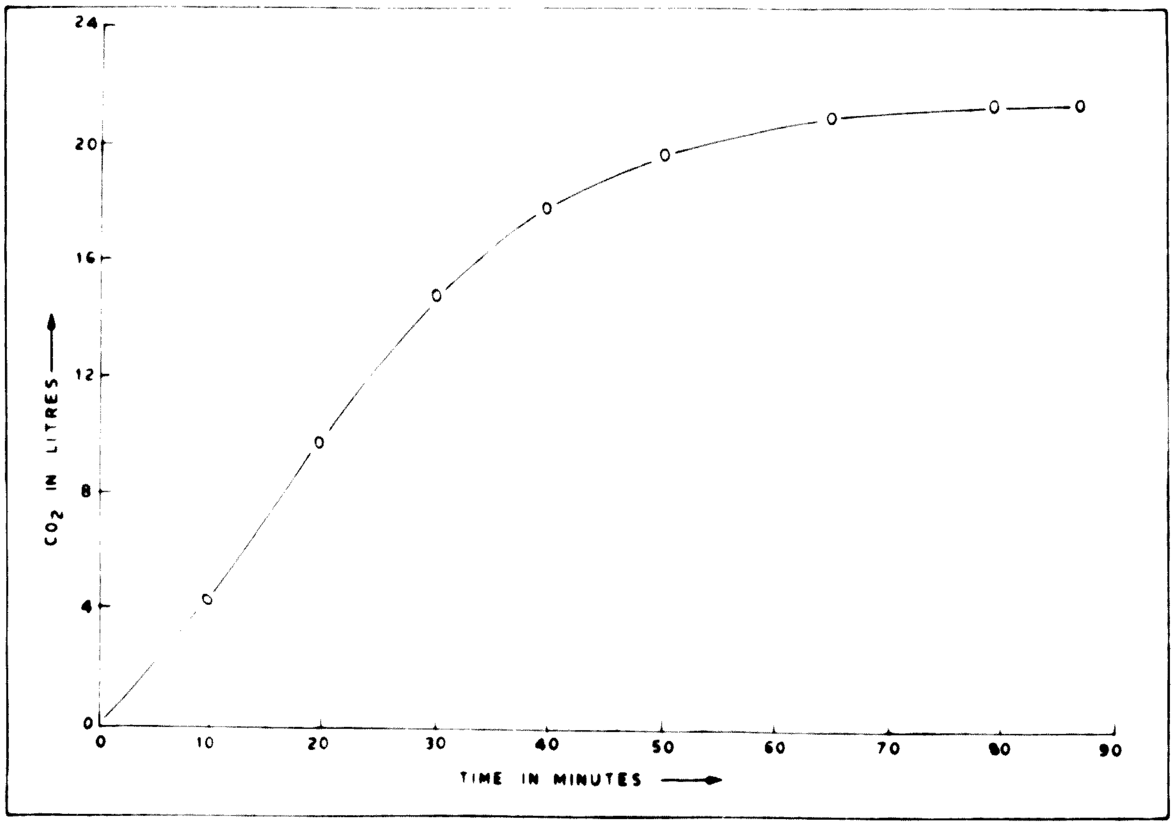


Fig. 3.8 CO_2 evolution at 930 C against time for size 12 6 mm Asswan lime stone

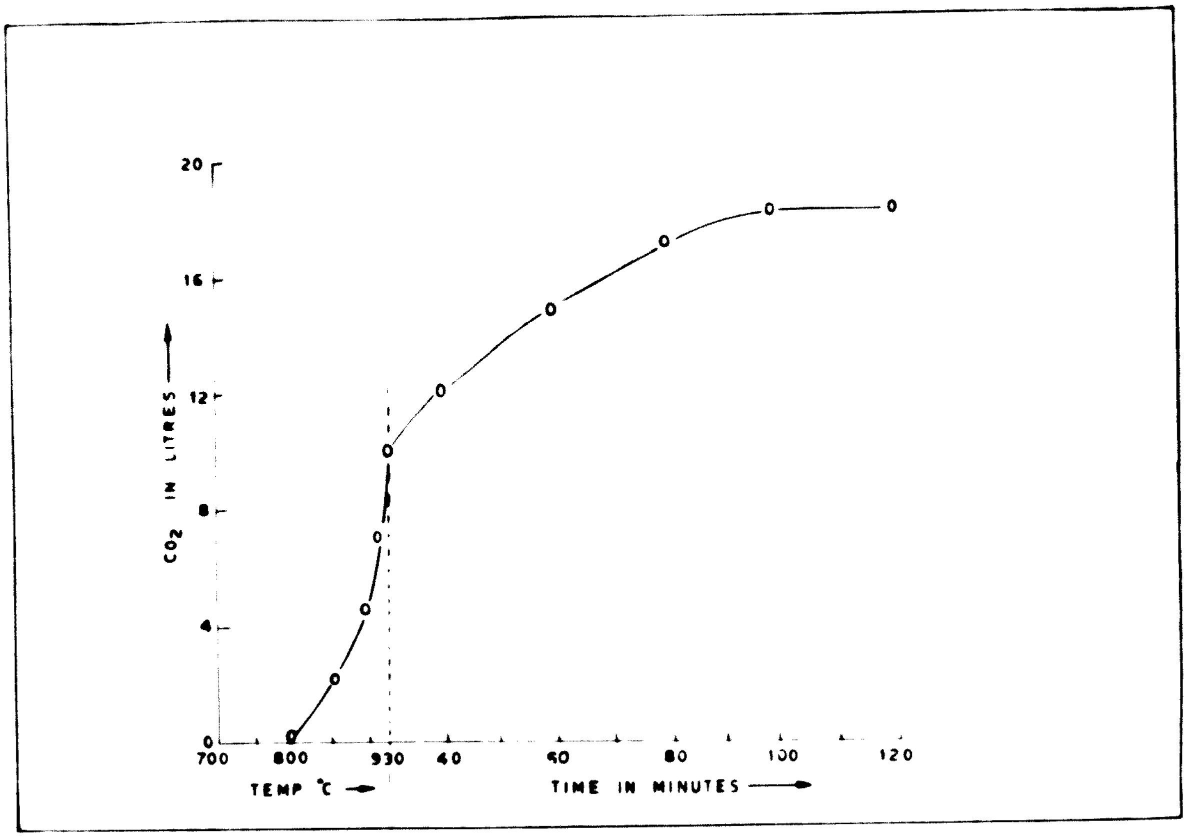


Fig 3.9 CO_2 evolution at constant rate of heating and against time at 930 C 25 12 mm fraction

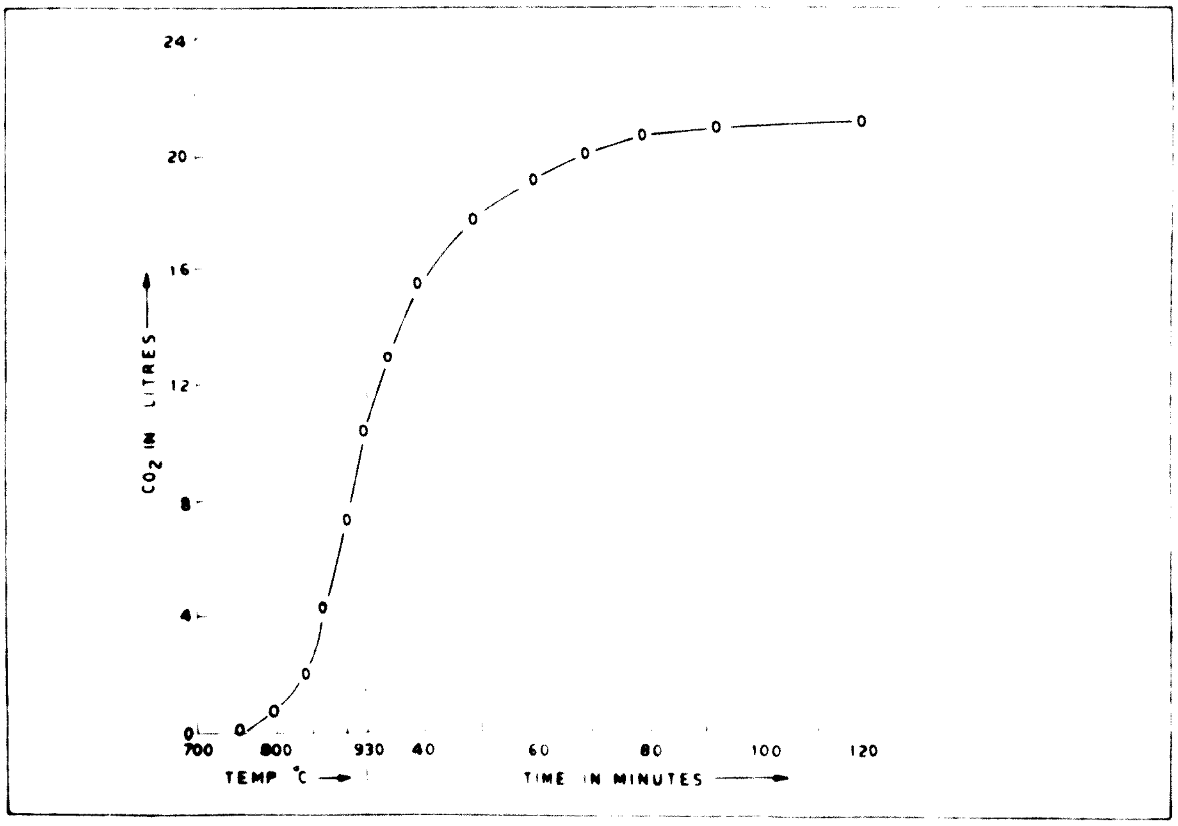


Fig 3.10 CO$_2$ evolution at constant rate of heating and against time at 930 C for 12 6 mm fraction

3. Degradation is comparatively [illegible] ore under reducing condition, i.e. under CO, CO_2, H_2 atmospheres but in actual practice, there will be tremendous load on the ore and hence there is likely to be much more degradation than as shown in Table 3.9.

Reducibility Tests

(i) With CO or H_2

With CO or H_2 reducibility or rate of reduction of iron ores determines the probable behaviour of iron ores during smelting in blast furnaces. Several standard methods, have been used by various workers. In most methods either CO or H_2 has been used as reductant and experiments have been performed at 900 - 1000°C at varying flow rate of reductants and the measure of reducibility was determined by calculating the rate of oxygen removal from the ore. Reducibility indicates the rate of removal of oxygen from the ore under the test conditions. In view of the complications of conducting the reducibility test under non-isothermal conditions and variation in gas composition, for example in a manner identical to blast furnace stack condition, the routine isothermal test at 900°C at a definite flow rate of the reducing gas discloses comparative rate of removal of oxygen from the ore.

In these tests, the ore was kept in a vertical tube as shown in Fig. 3.1 at a temperature of 950°C and the hydrogen was passed at the rate of 1000 cc/min. The water collected was noted every two minutes till no further appreciable amount of water was collected. Similarly in the case of CO reduction, the CO produced of the reduction was collected and noted every 2 minutes for first 40 minutes and every 4 minutes for subsequent 80 minutes. The experiments were performed with various mesh sizes of Asswan ore, which was obtained by crushing and screening. The results have been furnished in Figs. 3.2 and 3.3.

(ii) With CO-H_2 Mixture

The reducibility of Asswan iron ore was also determined at 950°C, keeping pCO and pH_2 as 0.5 for 90 minutes. Two mesh sizes -6+8 and -10+20 were tested keeping temperature, flow rate and time constant. It was observed that reduction rate of -6+8 mesh size ore was faster upto 50-55 min. but after about 55 minutes the reaction with -10+20 mesh size ore proceeded much faster and about 98% reducibility was obtained in 75 minutes. This behaviour has been shown in Fig. 3.4.

Another plot, as shown in Fig. 3.5 was made for comparing the reduction behaviour of -6+8 mesh size ore with H_2, CO +H_2 mix. and the rate of reduction is also much faster than the preceeding two. The

reduction by CO **is** much **slower in** **comparison** with hydrogen. The **curve for** CO + H_2 reduction lies in between. The observation appears **to be obvious** in view of the faster reaction rate by hydrogen at 950°C **in comparison** to carbon monoxide.

Calcination of Asswan Limestone

Calcination characteristics of the Asswan limestone were determined at isothermal temperatures and at constant rate of heating with two different sizes.

Limestone Sample

Representative samples of Asswan limestone were crushed so as to obtain -25 +12 mm and -12 +6 mm size fractions.

The temperature at which a limestone decomposes and the rate of its calcination are important from the point of view of solution loss reaction, which is nothing but solid carbon consuming reaction. If a limestone starts decomposing at lower temperatures, the evolution of CO_2 takes place a little higher up on the stack than if it decomposed at higher temperature. Limestone decomposing in the higher temperature zone (say 1000°C or above) evolves CO_2 which reacts with solid carbon to give rise to the solution loss reaction ($CO_2 + C = 2CO$). In order to predetermine the behaviour of limestone in a blast furnace or in an electric smelting furnace, it is worthwhile to investigate its calcination characteristic in addition to its chemical analysis and petrological study.

Calcination Test

100 **gm.** of limestone sample, dried at 100 °C was placed in a **stainless steel** pot, open at one end and connected to a flow meter for measuring CO_2. This stainless steel pot was then kept in an electrically heated furnace maintained at a temperature of 930 °C ± 5°C. The CO_2 liberated due to the dissociation of limestone at that temperature was continuously recorded at interval of 5 min. until CO_2 evolution

Table 3.10 - Chemical analysis, calcination **test results at isothermal**

Limestone	Chemical analysis %						**Isothermal** -25 + 12 mm	
	CaO	SiO_2	Al_2O_3	S	P	LOI	Time in min.	CO_2 in lit.
Asswan	54.86	1.3	0.96	0.27	tr.	40.14	76	18.94

ceased. In a continuous heating test, the stainless steel pot containing the 100 gm limestone sample was kept in the furnace maintained at 700°C, and then the temperature of the furnace was raised at the rate of 7-8°C, to 930°C. The CO_2 evolved in this case was also recorded at interval of 5 min. Three tests were conducted on each size fraction to take account of any possible lump to lump variation in the characteristics of limestone. The quantity of CO_2 obtained was plotted against time in case of isothermal test and against temperature in case of continuous heating rate test upto 930°C ± 5°C and then against time as in the case of isothermal test.

Discussion

The limestone sample was petrologically studied. The sample had a light yellowish white colour on the surface and white on fresh surfaces. The sample was dense and compact.

The thin section study of various lumps revealed that the sample consisted of fine grained calcite matrix and on the whole, the sample could be graded as a high grade variety with minimum of insolubles in it as shown in Fig. 3.6.

The calcination characteristics of limestone at both isothermal and continuous heating rate was determined with two sizes. The test results and the chemical analyses of the limestone are given in Table 3.10.

From Table 3.10 it can be seen that the CO_2% is quite high whereas SiO_2% is low, hence due to its high available base, the limestone may be used as flux. The sulphur content of the limestone is somewhat high and, therefore, it is likely to increase the sulphur input in the burden. This may, however, be decreased by blending with low 'S' limestone.

Figs. 3.7 and 3.8 show the relationship between CO_2 evolution with time in isothermal test with the two sizes. From the curves it can be observed that there is not much variation in the total CO_2 evolved with either of the two sizes. In 60 min. time, the CO_2 evolved was 18.31 lit.

temperature and at constant rate of heating

test 930°C		Constant rate of heating upto 930°C				Calcination temp. °C
-12 + 6 mm		-25 + 12 mm		-12 + 6 mm		
Time in min.	CO_2 in lit.	Time in min.	CO_2 in lit.	Time in min.	CO_2 in lit.	(-25 + 12 mm)
73	21.21	30	10.10	30	10.42	800

and 20.7 lit. with -25 +12 mm and -12 + 6 mm sizes respectively. Figs. 3.9 and 3.10 show the relationship between CO_2 evolution and temperature at constant rate of heating upto 930°C together with CO_2 evolved at 930°C (constant temperature) with time till evolution ceases. The total CO_2 evolved in both the sizes in this case is also more or less the same as in the isothermal test.

Conclusions

1. Asswan limestone may be graded as of a high grade variety with minimum insolubles in it.
2. Available base in the limestone seems to be satisfactory.
3. The calcination characteristics of limestone may be considered as satisfactory.
4. The limestone can be used as flux.

CHAPTER 4

AGGLOMERATION OF IRON ORE CONCENTRATES

PELLETISATION STUDIES FOR THE PRODUCTION OF GREEN, HEAT-HARDENED UNFLUXED AND FLUXEDPELLETS

Tests were conducted to determine the optimum conditions for large scale production of pellets for subsequent treatment in the rotary kiln for making sponge iron. Magnetic concentrate assaying 51.7% Fe, 11.3% SiO_2, 5.23% Al_2O_3 was used for pelletising tests. Sieve, sub-sieve analysis and surface area of magnetic concentrate are recorded in Table 4.1.

TABLE 4.1: Sieve & Sub-sieve analyses of the concentrate for pelletization

Mesh	Micron	Wt. %
+ 65	210	0.4
- 65 + 100	149	3.0
-100 + 150	105	3.9
-150 + 200	74	7.0
- 200	< 53	85.7
		100.0

Sub-sieve analysis (Feed percentage - 325 mesh = 68.4%)

+ 41.5	43.0
29.6	15.0
20.8	11.0
14.2	8.0
10.4	10.0
7.4	3.0
< 7.4	10.0
Surface area in sq. cm/gm = 2922	100.0

Initially, a number of laboratory scale pelletization tests were conducted in a balling drum 0.610 m dia x 0.305 mm width rotating at 11 rpm. The pelletizer feed was prepared by thoroughly mixing 50 kgs of sample with desired amounts of water and other additives. The moisture content of the mix was kept at about 6 per cent. In the balling drum additional amount of water was sprayed to initiate balling up and these were allowed to grow to about 10 to 15 mm size by addition of fresh ore-binder mixture with the addition of requisite amount of water. The entire operation took about 25 minutes to form 10 to 15 mm pellets and 10 minutes for rolling afterwards to get a shining polished surface. The following tests were performed to assess the quality of pellets produced.

Physical Tests on Pellets

(a) Green pellets:

(i) Moisture content: The moisture content of freshly rolled green pellets was determined and expressed in terms of per cent moisture in green pellets on wet basis.

(ii) Drop strength: The resistance to withstand shock as would be experienced by green pellets during handling, drying, and heat-hardening in a pelletizing plant, was simulated in the laboratory by dropping the pellets from a height of 500 mm and recording the number of drops at which the first crack on a pellet appeared.

(iii) Green compression strength: Green compression strength of pellets was determined by keeping a freshly prepared sized pellet as a short column between a steel plate and a hollow brass tube 26.4 cm long closed at the bottom and held vertically between two rings with sharp inner edge. Mercury was dropped continuously in the brass tube till the pellet yielded by breaking and the total weight of mercury and the tube was expressed as the green compression strength in kg/pellet.

(b) Air dried pellets

(i) Dry compression strength: Dry compression strength of air dired pellets was determined by the same method. The weight at which the pellets broke was taken as dry compression strength in kg/pellet.

(c) Heat-hardened pellets

(i) Sieve analysis: The fired pellets were sized using punched hole screens, having round apertures of 12.5 mm, 9.3 mm and 6.3 mm and expressed in terms of wt% retained.

(ii) Compression strength: Compression strength of fired pellets and pre-heated pellets was determined by using Amsler Universal Testing machine and expressed in terms of kg/pellet.

(iii) Microporosity: The apparent specific gravity of pellets was determined by mercury hydrometer and the true specific gravity of powder by specific gravity bottle and the microporosity was expressed as:

$$\text{Microporosity \%} = \frac{\text{True sp. gr.} - \text{App. Sp. gr.} \times 100}{\text{True sp. gr.}}$$

(iv) Reducibility: The object of the test was to determine reducibility characteristics of iron ore pellets. The method adopted in the present investigation was to determine and compare time taken by individual test sample for 90% reduction under similar experimental conditions. The comparison of rates of reduction of various samples of iron ore pellets under identical conditions are recorded in Table 4.6.

Test procedure:

The reduction of iron ore pellets was carried out at 850°C (± 10°C) with commercially pure hydrogen as passed through concentrated H_2SO_4, anhydrous $CaCl_2$, ascarite and copper turnings heated to 500°-550°C. Dry pellets weighing 200 gms was placed inside a steel reduction tube and its position was adjusted so as to keep the test sample at the centre of the tubular reduction furnace. The sample was heated to 850°C in N_2 atmosphere. The reduction of sample was carried out and the flow of H_2 was maintained at about 1700 cc/min with precalibrated flow meter. The quantity of water vapour produced during the reduction was condensed and measured after each interval of 15 minutes. The experiment was continued till there was no condensation of water during a period of 30 minutes. The reduced sample was cooled to room temperature in inert atmosphere and analysed. The percentage reduction of the sample was calculated as follows:

$$\text{Percentage reduction} = \frac{\text{Total water condensed upto any instant due to reduction} \times 100}{\text{Total water collected by condensation during the entire experiment due to reduction.}}$$

(v) Swelling index: The test procedure consisted of measuring the percentage increase in volume of the pellets due to reduction in a reducing atmosphere at a particular temperature and for a given time.

Sample: The sample of each test consisted of pieces of pellets of -12 +10 mm size. The pellets were oven-dried at 105° ± 5°C for 60 minutes in order to remove the moisture.

Test procedure

a) The volume of the pellets was measured first by mercury hydrometer.

b) The pellets were placed in a reduction tube which was connected to a gas circuit.

c) The sample was heated upto 850°C in an inert atmosphere using purified N_2 gas.

d) After maintaining the temperature of the sample at 850°C for 30 minutes in the inert atmosphere, the inert gas was replaced by purified H_2 gas for reducing the pellets. The sample was reduced for 100 minutes at 850° ± 10°C.

e) At the end of 100 minutes reduction, the reducing gas was replaced again by the N_2 gas and the sample was allowed to cool to room temperature.

f) The volume of each of the pellets was measured.

g) The swelling index was calculated from the values given below:

$$SW = \frac{(V_2 - V_1) \times 100}{V_1}$$

Where SW = % swelling index
V_1 = volume in cc before reduction
V_2 = volume in cc after reduction.

Bulk density: The bulk density of the fired pellets and the true specific gravity were determined.

Additives

Limestone and bentonite were used as additives for pelletization. The limestone sample was supplied by UAR, whereas the locally available bentonite ground to -100 mesh was used as binder. Chemical analyses of limestone and bentonite samples are given in Table 4.2. Sieve analysis of limestone powder and physical properties of bentonite are given in Table 4.3.

Table 4.2 - Chemical analysis of additives %

Constituents	Limestone	Bentonite
Fe_2O_3	0.80	2.7
CaO	50.40	3.4
MgO	2.61	2.8
Al_2O_3	1.12	26.4
SiO_2	2.06	45.6
Na_2O	-	0.4
CO_2	40.94	-
SO_3	0.45	-
L.O.I.	41.89	12.3
P	0.035	-

Table 4.3 - Sieve analysis of limestone and physical characteristics of bentonite

Size in mesh	Wt. %
+ 48	0.7
- 48 + 65	3.2
- 65 + 100	4.5
-100 + 150	8.6
-150 + 200	12.1
-200	70.9
	100.0

Physical Characteristics of Bentonite

a)	sp. gr.	1.87
b)	Swelling index	5.7 gm/gm for 24 hrs.
c)	pH	9.0

Laboratory Scale Pelletisation Tests

A) Unfluxed pellets:

Laboratory scale pelletisation tests were carried out with magnetic concentrate under varying conditions to determine the optimum conditions for pilot plant production of pellets for subsequent pre-reduction. The tests were carried out in a balling drum (0.61 m dia x 0.305 m) rotating at 11 r.p.m. using 50.0 kg of material per batch. Bentonite addition was varied from 0 to 1.5% Limestone powder was also used as additive and in some cases a combination of limestone and bentonite was also employed. The results are recorded in Table 4.4.

The results indicated that green compression strength and no. of drops withstood bv a pellet from a height of 500 mm did not vary considerably and addition of 1% bentonite gave satisfactory results. Combination of bentonite and limestone did not improve the green compression strength of pellets. Moreover, it was observed that in the case of 0 to 0.5% bentonite addition and 0.5 to 2.0 limestone addition, the green pellets had a tendency to crumble and generate more fines during handling than those made with 1% bentonite addition.

B) Fluxed pellets

Laboratory scale tests for the preparation of fluxed pllets were also conducted using the same pelletizing drum.

$$\frac{(CaO + MgO)}{(SiO_2 + Al_2O_3)}$$

was varied from 1.0 to 1.6 by the addition of requisite amount of limestone to the beneficiated iron ore. No bentonite addition was found to be necessary for these tests. The results are recorded in Table 4.5.

With the alteration of the basicity degree of the fluxed pellets, the physical properties did not deteriorate. The green compression strength and the resistance to knocking were also satisfactory. Production of fines (-9.5 mm) during handling of green pellets was of the order of 5-6%.

Drying and Heat Hardening of Green Pellets

Pot Grate Furnace

Laboratory scale drying and heat-hardening was carried out in a pot grate furnace unit. The unit had arrangements for both downdraft and updraft drying utilising L. D. oil and low calorific value producer gas at regulated temperatures.

Pot grate furnace fabricated at NML for heat hardening of pellets

[illegible]

Table 4.4 - Result of laboratory scale pelletisation tests:

Test No.	Surface area sq. cm/gm	Grind % -200 mesh	Additives			Green pellets		Comp. strength after 24 hr's. kg/pellets
			Lime-stone %	Bento-nite %	H_2O	Comp. strength kg/pellet	No. of drops with stood from 500 mm ht.	
1	2922	85.7	-	-	13.5	1.2	0.94	1.1
2	2922	85.7	-	0.5	13.6	1.5	2.3	2.0
3	2922	85.7	-	1.0	13.8	1.7	3.2	3.8
4	2922	85.7	-	1.5	13.9	1.65	3.0	2.9
5	2922	85.7	0.5	0.5	13.7	1.53	2.4	2.1
6	2922	85.7	0.5	1.0	13.8	1.59	2.5	2.4
7	2922	85.7	1.0	0.5	13.8	1.63	2.5	2.5
8	2922	85.7	0.5	-	13.4	1.41	1.92	2.0
9	2922	85.7	1.0	-	13.5	1.51	2.0	2.1
10	2922	85.7	1.5	-	13.7	1.56	2.2	2.3
11	2922	85.7	2.0	-	13.7	1.57	2.2	2.4
12	2922	85.7	1.0	1.0	13.8	1.63	2.85	2.9

Table 4.5 - Results of pelletisation tests using limestone as a flux

Test No.	Surface area sq. cm/gm	Grind % -200 mesh	Basicity ratio $\frac{(CaO + MgO)}{(SiO_2 + Al_2O_3)}$	% H_2O	Green pellets		
					Comp. strength kg/pellet	No. of drops with-stood form a heat of 500 mm.	Comp. strength after 24 hrs. kg/pellet
1	2922	85.7	1.0	14.5	1.16	2	1.20
2	2922	85.7	1.2	14.6	1.15	1.8	1.6
3	2922	85.7	1.4	14.8	1.18	2.1	2.3
4	2922	85.7	1.6	14.8	1.21	2.2	2.4

The pot grate furnace consisted of a central movable pot grate section hinged on two bearings and mounted on a frame. The frame was movable on rails for easy handling and unloading of pellets. The top section of the unit consisted of a hood chamber connected on two sides to two burner chambers, one for low temperature and the other for high temperature firing. Both the burner chambers had bypass stacks having stack caps and refractory valves in front to control the flue which could be passed through the bed of let out to atmosphere, as desired. The bottom chamber of the unit consisted of the wind box on one end of which was connected to the suction line while the other chamber was connected to a low temperature burning chamber to produce flue for updraft drying. This burner chamber also had a refractory control valve in front and a bypass stack to which a special annular air preheater was also built in to provide preheated combustion air for burners. The entire pot grate section, top hood, burner chambers and stacks were lined with a special castable refractory of high alumina cement. The low temperature burners were designed and fabricated at NML, to be used for producer gas which could develop a temperature upto 1000 - 1100°C. The burner could operate with producer gas upto a pressure of 0.211 kg/sq.cm. The air required for combustion was kept upto 60.96 cm.

A low pressure oil burner was used for the high temperature side, burning preheated L.D. oil with primary and secondary air. Excess air for controlling furnace atmosphere was provided at ambient temperature. The suction line from the two stage exhaust blower was connected to the bottom wind box through a bypass suction control valve which also allowed diluent air from outside to cool the exhaust gases. One branch from the suction line was also connected to the top of hood chamber above the pot through a control valve to provide alternatively updraft gas flow through the bed, when the bottom line was closed. Arrangement of external water spray was provided on the suction line to lower the temperature of exhaust flue gases to about 150°-160°C, to protect the blower. The blower used was a two-stage centrifugal type, each stage driven by a 20 HP motor, capable of developing a suction of 1524 mm W.G. with 85 cubic metre per minute free air flow. Low pressure air for combustion for all 3 burners was obtained from a single stage centrifugal blower connected to the preheater with a bypass control valve and driven by a 15 HP motor capable of delivering 1250 CFM (36.5 CuM/Min) free air flow at 610 mm W.G.

The central section of the pot consisted of movable shell divided into two portions. The top pot with annular corrugated refractory lining to reduce the chances of gas channeling and the bottom suction carrying the special high chromium cast iron grate. Thermocouple ports were provided at various heights of the pot, like top, middle, bottom and at a point just below the grate to measure the temperature at different zones and also of the grate and exit gases passing through the gate. For high temperature measurements, Pt-Rh thermocouples were used while for low temperature, chromel-alumel thermocouples were utilised in the pot section and other places. Temperature of the combustion chamber, hood, wind box, the suction line, preheated air and oil were also measured and recorded in two 6-point recorders. Pressure drop across the bed, hood and wind box blower etc. were measured by water manometers. Flow rate of product gases was measured by differential pressure measurements across a straight length of G.I. pipe on the suction line preceding the blower. Fuel oil consumption was recorded by volume measurements and gas consumption by a orifice plate on the gas line.

Procedure: Both the gas burners were ignited with stack caps open and stabilised at the desired temperature. Oil burner was also lighted simultaneously and kept running to obtain uniform combustion. The pot grate was slided out and 76.2 mm layer of previously fired pellets put as the layer on the grate. Freshly prepared green pellets was charged into the pot upto the desired height. The pot was then put in position and sealed by means of the ring clamps. Drying of the green pellets was then started by closing the low temperature stack cap and opening the refractory valve to allow the flue gases to pass through the bed. Pressure drop was controlled by means of by-pass valve to the desired extent. For testing updraft drying, the bottom burner and the top suction line through the hood was utilised. After drying was completed, the pot was pulled out momentarily to observe the effect of drying i.e. decriptitation, sagging etc. and also to draw a sample of dried pellets. Preheat gases were then passed from the high temperature burner by oepning the refractory valve on the high temperature burner chamber and closing the valve on low temperature side. The stack cap on the low temperature side was correspondingly closed and the low temperature side opened to blend the low temperature flue gas to the atmosphere. From the exhaust gas of the blower, it would be observed whether any ducting was taking place.

The time taken to reach the desired preheating temperature was noted and after the bottom attained the preheating temperature or after the given time interval, the flue gases were bypassed. The pot was again taken out to collect samples from the top, middle and bottom layers, to see the preheating effect. For final firing cycle, the pot was once again shifted to its position and high temperature flue gases were passed thereby raising the bed temperature to the desired limit. After allowing the required soaking time if any, the firing was stopped and the pellets cooled by shutting off the low temperature burner and opening low temperature bypass stack and refractory valve when cold air was drawn in from outside through the bed. The time taken from the different layers of bed to cool down to about 200-250°C was also recorded before taking out the pot. In the laboratory set up, no attempt was made to utilise the waste heat from the exit flue gases, but from the temperature data and gas volume the available sensible heat of the flue gases can be ascertained. Fuel

Discharging heat hardened pellets after cooling

Table 4.6 - Results of heat-hardening tests on fluxed and unfluxed pellets

Test No.	Temp. °C	Unfluxed Pellets					Reducibility rate for 90% reduction
		Cycle time in mts.	Comp. strength kg/pellets	Tumbling index +6.5 mm	Abrasion index % -28 mesh	Micro porosity %	Minutes
1	1150	42.0	141	65.3	19.4	20.0	101
2	1300	45.0	186	86.1	18.3	19.5	103
3	1300	45.3	331	94.1	3.9	29.9	105
4	1300	45.9	223	88.4	6.0	26.1	104
5	1300	45.1	195	76.3	12.1	26.0	101
6	1300	43.5	315	84.7	9.3	25.4	105
7	1300	43.5	228	71.7	15.4	29.2	103
8	1300	45.3	218	74.3	13.6	24.2	102
9	1300	45.0	165	67.1	18.4	24.6	105
10	1300	45.1	168	68.0	18.1	24.5	105
11	1300	45.3	172	68.1	18.0	24.9	105
12	1300	45.0	197	74.3	12.1	25.8	106
				Fluxed Pellets			
1	1180	58.0	156	59.4	21.3	27.3	102
2	1220	58.5	151	58.3	22.1	28.2	102
3	1230	58.5	150	58.0	22.5	29.1	101
4	1230	58.0	153	57.8	22.6	29.0	101
5	1230	58.3	153	58.5	21.7	27.5	101

consumption in terms of productivity, therefore had no actual or industrial significance except to the extent to which a comparative study could be made for different rates and time of heating.

After carrying out a number of drying and preheating tests under different conditions, varying flue gases, temperature, bed height, suction, etc. the optimum conditions for heat-hardening were established for further experiments.

Results: The laboratory scale heat hardening test results fluxed and unfluxed pellets are record in Table 4.6.

It was observed during the heat hardening of both fluxed and unfluxed pellets that meticulus care had to be taken in the initial stage to drive off the moisture, followed by gradual heating to 1300°C and soaking at that temperature for 5-6 minutes. Particularly in the case of fluxed pellets, a number of test had to be conducted for satisfactory hardening of pellets without fusion. The results indicated that compression strength of fluxed pellets did not vary appreciably with the alteration of the basicity ratio, although it was much less than the pellets made with addition of 1% bentonite alone. The cycle time taken by the fluxed pellets was also higher than that taken by the unfluxed pellets.

Sieve analysis of the fired pellets is record in Table 4.7.

Table 4.7 - Sieve analysis of fired pellets

	Wt. %
+ 12 mm	51.3
- 12 + 9.5 mm	35.7
- 9.5 mm	13.0
	100.0

The results indicated that the quantity of -9.5 mm pellets produced after firing was under reasonable limits which could be during plant practice.

Large Scale Preparation of Pellets for Pre-reduction

After assessing the behaviour of the pellets with 1% bentonite during heat hardening cycle, attempts were made to produce bulk quantity of unfluxed pellets for pre-reduction. For this purpose, a disc pelletizer of 91.44 cm inclined at 42° was used. The data for pelletizer operation is recorded in Table 4.8

Table 4.8 - Pelletizing data

Moisture content of the mix	5-6 %
Feed rate	950-1000 kg/hr
Inclination of the disc.	42°
Rotation of the disc.	12 r.p.m.
Retention time of balling	6-7 minutes.

7-8% water was sprayed on the disc. The green pellets were tested for moisture content, suceptibility towards degradation during transportation, compression strength etc. A certain quantity of pellets was heat-hardened in a rotary kiln and the balance was charged in the green state for pre-reduction in the rotary kiln. It afforded the possibility of ascertaining the degradation of both the varieties of pellets and for comparable degradation the costly step of heat hardening can be omitted in industrial scale. The heat hardening data obtained during rotary kiln operation are recorded in Table 4.8a.

Table 4.8a - Results of the large scale heat-hardening of unfluxed pellets in the rotary kiln

Temperature in the discharge end	1150°C.
Retention time inside the kiln	90 to 110 minutes.
Compression strength kg/pellet	226
Tumbling index % + 6.5 mm	28.8
Abràsion Index % -28 mesh	5.8
Microporosity %	27.8
Reducibility rate for 90% reduction	102
Swelling Index gm/gm	3.7
Feed rate/hr	180 kg.

Chemical analysis of the heat-hardened pellets are given in Table 4.9.

TABLE 4.9: Chemical analysis of heat hardened pellets

No.	Basicity	Fe	FeO	Al_2O_3	SiO_2	CaO	MgO
1	1	50.36	2.35	4.57	9.82	11.97	1.94
2	1.2	49.12	4.49	4.40	9.45	14.52	1.97
3	1.4	48.3	2.04	4.20	8.99	16.1	2.01
4	1.6	47.5	3.74	4.04	8.62	17.92	2.05

PETROLOGICAL STUDIES ON HEAT-HARDENED PELLETS

The pellets were made of concentrates obtained after magnetic separation after reduction roasting of the sample. They were heat-hardened at different temperatures to study the temperature effect on the heat hardening property in relation to their physical and mineralogical changes. Petrological studies were carried out on the polished cross sections of different pellets under ore-microscope besides observing their physical and surface properties. Limestone was added to make the mix basic and bentonite was used as the binder. The results are given as follows:-

(1) Sample I (Heat-hardened at 750-800°C)

A. *Physical properties:*

The pellets were all free, spherical in shape, reddish brown to brown and brownish black in colour, feebly magnetic in some cases and earthy lustre. No superficial cracks were observed. Visual examination of the cross section of the pellets exhibited general dense structure with occasional porous core in some cases.

B. *Microscopic characteristics*

The minerals present were goethite, hematite, magnetite, hydrogoethite, hydrohematite, traces of copper pyrite and iron pyrite and siliceous gangue. Goethite was present as crypto-crystalline aggregates with scattered and patchy occurrence. Hematite was present as crypto-crystalline segregated globular grains. Hydrogoethite formed the amorphous ground mass with admixing of siliceous gangue grains. The general texture was porous and less compacted in the central core.

The mineral assemblage varied from pellet to pellet. The transition phase of magnetite to hematite was noticed in some pellets. Original colloform texture present in the oolites of the ROM ore was still persisting. In some pellets the ferruginous constituents other than magnetite were found as disseminated patches while in others as seggregated sub-microscopic hematitic body with patchy occurence together with magnetite.

(2) Sample No. 2 (Heat hardened at 1000°C)

A. *Physical properties*

The pellets were all free, spherical in shape, blue black to pitch-black in colour and adamantine lustre. They were strongly magnetic when tested with a hand magnet. Some of them were cracked. Visual examination of the cross section of the pellets exhibited dense nature of the periphery and rim with pitty and porous central core. The width of the rim and the diameter of the core varied from pellet to pellet.

B. *Microscopic characteristics*

Hematite was present on the peripheral zone with magnetite as the core was approached and minor amounts of goethite, hydrohematite and traces of pyrite. Hematite was microcrystalline finegrained and present as locally dense aggregate patches. Magnetite was present as microcrystalline finegrained aggregate and globules admixed with scattered and locally enriched octahedral grains with very fine siliceous slag bonding. Some microlites and crystallites were also noticed. Some radial cracks towards the periphery with more voids in the central zone was relatively more dense in texture with a central porous field. The width of the marginal rim from the periphery (consisting mostly of hematite) was found more or less uniform in all the pellets and extended upto half the distance towards the core beyond which magnetite was present.

3. Sample - 3 (Heat hardened at 1050°C)

A. *Physical properties*

The pellets were found to be mostly free but some fused together on surface and could be freed by applying some force. The colour, shape and lustre and magnetic property were same as sample 2. But visual examination of the cross section of the pellets exhibited no separate distinct zonal rim from the periphery but a generally scattered orientation of voids, which were present more in the core in some cases. External cracks were also noticed on some pellets.

B. *Microscopic characteristics*

The mineralogical and textural features of this group of pellets were found more or less identical with sample 2. The special feature which was noticed was the presence of more hematite with an admixture of hematite and minimum magnetite all around towards the core. The porosity and denseness varied from pellet to pellet. Cracks were less but vesciles and crevaces were more.

Conclusions

1. As seen from the mineralogical assemblages and the texture of the pellets the heat-hardening temperature range of 750 to 800°C may be too low for recrystallisation and phase change followed by necessary grain growth and siliceous bonding.

2. Even the temperature of 1000°C may not be enough for the present magnetic concentrate pellets since the complete oxidation phase was not present in any of the pellets of this temperature range. The central core of the pellets is still constituted of magnetite, which should all be changed to hematite in an ideal heat-hardened pellet.

3. A higher temperature of 1050°C with a complete oxidising atmosphere had given better heat-hardening property to the pellets. Typical photomicrographs of heat hardened pellets are shown in Figs. 4.1 to 4.3.

(ii) PRODUCTION OF SINTER FROM MAGNETIC CONCENTRATES

The magnetic concentrate of Asswan iron ore, which was used for pelletization work, was taken up for the study of sintering behaviour along with limestone from U.A.R.

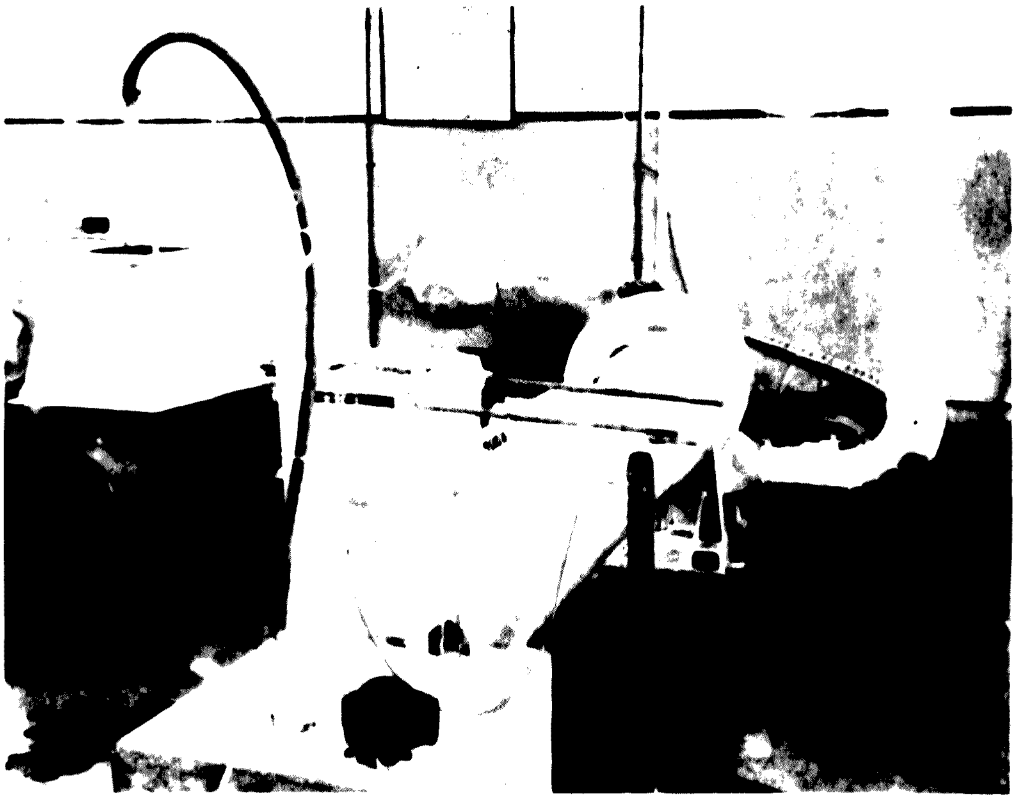
Experimental Sintering Unit:

A batch sintering unit, designed and fabricated at the NML with a grate area of 16 cm square was used for this work. This unit consisted of a combustion chamber fitted with a hood and gas burner on the top for initial ignition of sinter mix and also for mixed-firing afterwards, a sinter box of 16 cm square, 28 cm height with a special C.I. grate and a wind box. The wind box was connected by a tapered cylindrical pipe to an exhaust fan having a suction capacity of 1300 cft (36.81 cu m) of air per minute at NTP driven by a 10 HP motor and capable of operating at 20" (50.8 cm) w.g. suction. A chromel-alumel thermocouple was provided in the wind box for measuring the temperature of the exit gases.

Raw Materials Used

(1) Magnetic conc.: The fine magnetic concentrate(used for pelletization) was employed for sintering studies.

(2) Limestone: The limestone supplied by U.A.R. was crushed to -6 mesh and used for the tests.



A view of the batch sinterer and fabricated at NML

[illegible]

(3) Coke: Coke breeze available locally was used for the test after crushing to -6 mesh.

(4) Coke Oven gas: Supplied from M/s Tata Iron & Steel Co. Ltd., Jamshedpur was used for the tests.

The sieve analysis of limestone, coke and the pre-balled mix (of magnetic concentrate, limestone and coke) are shown in Table 4.10 and their chemical analysis in Table 4.11.

Table 4.10 - Sieve analysis

Mesh size	Coke %	Limestone	Sieve size	Preballed product of mix %
-6+8	15.1	21.5	9.6 mm	13.0
-8+10	13.3	12.0	9.6+6.5 mm	37.5
-10+14	13.6	9.0	-6.5 mm + 4 mesh	13.3
-14+20	10.4	7.6	-4+6 mesh	18.0
-20+28	9.9	6.8	-6+8 mesh	9.0
-28+35	8.6	6.3	-8+10 mesh	5.3
-35+48	7.3	5.9	-10+14 mesh	2.4
-48+65	5.7	5.8	-14 mesh	1.5
-65+100	6.2	5.8		
-100+150	3.1	5.0		
-150+200	1.6	3.3		
-200	5.2	11.0		
	100.0	100.0		100.0

Mixing Procedure: The magnetic concentrate, limestone, coke, in correct proportions for a particular basicity, were mixed for a period of 2 minutes in a mixer fitted with lifters and then subjected to for preballing in a disc pelletizer where the requisite amount of water was added in the form of spray to form fine balls. The time taken for preballing was kept at 8 minutes. 2 kgs of -2.54 cm + 16 cm was used as a bedding layer of nearly 4 cm thickness. The sinter box was hand filled with mix. The effective height of the bed was nearly 24 cm.

Table 4.11 - Chemical analysis of raw materials (%)

		Fe	Fe_2O_3	FeO	SiO_2	Al_2O_3	CaO	MgO	P	S	LOI	Others
1.	**Magnetic conc.**	51.7	-	15.84	11.30	5.23	4.65	1.82	1.03	0.16	5.11	-
2.	**Limeston**	-	0.80	-	2.06	1.12	50.40	2.61	0.035	-	41.89	(CO_2 = 40.94 SO_3 = 0.45)
3.	Coke	-	-	-	14.35	6.82	0.89	-	-	-	-	(Fc 68.90, Moisture 0.30, Ash 25.70, VM 3.10)
4.	Coke oven gas	CO_2	CnHm	CO	H_2	CH_4						
		3.8	2.2	9.7	51	23						

Ignition: Coke oven gas was used for initial ignition. The time of ignition was kept at 2 minutes at a gas pressure of 30 mm Hg. The suction during ignition was kept at its maximum, i.e. 50.8 cm Wg. as measured by a water manometer connected to wind box. After ignition the burner was swung inside. The temperature of the exit gases gradually increased, reached a maximum and then started falling after the sintering was completed. The sinter was allowed to cool in sinter box with full suction on, till the temperature of exit gases dropped to 100OC.

The suction during sintering was found initially to decrease but was found to increase during the cooling period by about 7.6 mm Wg, from the minimum value reached at the end of sintering period. The sinter was then removed from the box and broken to 5.08 cm.

Physical Tests for Sinter

The broken sinter is allowed to sieve through 25.4, 16.0, 9.6 and 6.5 mm and 4 mesh and the amount of +96 mm expressed as T/h m^2 was taken as the productivity +16 mm -25.4 mm and is used as bedding layer and 9.6 mm is taken as return fines for subsequent tests.

Shatter test:

-50.8 + 25.4 mm fraction is allowed to drop from a height of 2 metres thrice on to a m.s. plate and then sieve analysed in the above manner and the shatter size stability and % -4 mesh after shatter were determined, % + 10 mm fraction after shatter gives the shatter strength.

Experimental Procedure: The main object in undertaking these tests was to determine the optimum conditions to produce a fluxed sinter of good quality and strength. For this, some initial tests were conducted to establish the base range of sinter variables like coke, return fines. However, the water variation could not be studied since the magnetic concentrate was too fine that it had to be preballed before sintering, for which a certain minimum quantity of water (16%) was required. After determining the optimum range for base variables, subsequent tests were made with different basicity

$$b = \frac{CaO + MgO}{SiO_2 + Al_2O_3}$$

For each test, rate of sintering % 9.6 mm in the sinter product, productivity, % -4 mesh fines after shatter, shatter size stability and shatter strength were found out. The parameters of importance in sintering are good productivity and high shatter strength (or shatter size stability).

Results

Effect of variation of coke (Table No. 4.12)

A trail sinter produced with 5% coke was found to be fused. Hence the coke was lowered down gradually in steps (by 1%) to 2% when more return fines were produced at 3% coke, higher productivity favoured this as an optimum value, although the shatter size stability was somewhat low and the % -4 mesh after shatter was a little high.

Effect of variation of return fines (Table No. 4.13)

Except the case of 20% return fines, the sintering rate decreased in all other cases but not to a very great extent and the % + 9.6 mm and shatter size stability increased and %-4 mesh after shatter decreased. But 20% return fines was chosen to be optimum in view of its high productivity and good shatter size stability (although decreased somewhat compared to higher range of return fines used).

Effect of variation of basicity (Table No. 4.14)

Taking water, coke, return fines at 16%, 3% and 20% respectively as their optimum values, the basicity was varied from 1.3 to 2.5. However test 8 conducted at 3% coke was found to contain lot of fines and % 9.6 mm was thus very low. Hence coke was increased to 3.5%. However at increased basicity, i.e. at b = 1.8, sinter was all fused, hence the coke was decreased to 3.25%. Further tests at b = 1.8, 2, 2.2, 2.5 showed that at increased basicity (b 2) the return fines were more. Hence b = 1.8 was taken as the optimum in view of higher productivity, good shatter size stability.

Mixed firing studies (Table No. 4.15)

The purpose here was to decrease the amount of coke input to the sinter mix and thus avoid troubles attendant with increased ash content of coke in the mix.

The coke was decreased gradually from its optimum value of 3.25% at stages by 0.25% and the deficit in heat input due to lesser coke employed in each case was made up by passing coke oven gas of equal heat value.

As before, the ignition was made for 2 minutes and then instead of stopping the gas flow at this stage, it was continued but at a lesser rate, so that there was no fusion as would have happened with higher flow rate.

The tests were conducted at a constant basicity of 1.8, the optimum value. With increase in replacement of coke by the gas, the

Table 4.12 - Effect of variation of coke

Table 4.12 - Effect of variation of coke

Test No	Total wt. of mix. kg.	Coke %	Water %	Return fines %	Sintering time (mts.)	Sintering rate (cm/min)	% [illegible] product	Productivity t/m² day	% +4 mesh after shatter	Shatter size stability	Shatter strength
1	15.15	2	16	20	11		more return fines produced.				
2	14.05	3	16	20	12.5	1.42	73.87	24.14	9.16	57.35	75.21
3	15.60	4	16	20	16	1.5	65.79	20.21	6.90	62.93	83.43
4	15.1	5	16	20	16	1.5	66.86	19.84	6.72	61.84	82.36

Table 4.13 - Effect of variation of return fines

5	15.55	3	16	25	17.5	1.372	63.25	17.96	7.241	64.36	78.65
6	15.65	3	16	30	15	1.548	72.71	21.04	7.087	61.60	79.31
7	15.65	3	16	35	16.5	1.454	75.25	21.33	7.605	63.42	66.51

Table 4.14 - Effect of variation of basicity

Test No.	Basicity	Total wt. of mix. kg.	Coke %	Water %	Return fines %	Sintering time (mts.)	Sintering rate cm/min.	% +9.6 mm produced	Productivity t/m²/day	% -4 mesh after shatter	Shater size stability	Shatter strength
8	1.3	15.50	3	16	20	15	1.6	Too much fines.				
9	1.3	14.5	3.5	16	20	12.5	1.92	84.94	23.71	9.29	54.43	75.23
10	1.5	14.8	3.5	16	20	12	2	75.67	24.53	13.54	55.11	72.23
11	1.8	15.75	3.25	16	20	9.5	2	73.19	30.49	7.26	53.19	75.75
12	2	15.15	3.25	16	20	8.5	2.82	66.80	29.94	8.73	53.51	75.34
13	[illegible]	15.40	3.25	16	20	9	2.67	65.54	[illegible]	10.06	[illegible]	74.41
14	2.5	15.15	3.25	16	20	10	2.4	[illegible]	[illegible]	[illegible]	[illegible]	72.06

Table 4.15 - Effect of mixed firing

Test No.	Wt. of sinter mix.	% coke	Vol. of coke oven gas cft	% coke less than optimum	Time	Rate of sintering	Max. temp. in wind box	Wt % of +9.6 mm	Productivity t/m² day	% -6.5 mm after shatter	Shater size stability	Shatter strength
15	15.45	3	2.35	7.7	11.5	2.086	2.086	80.21	[illegible]	8.10	53.75	78.93
16	15.7	2.75	4.7	15.39	[illegible]	[illegible]	1.846	79.64	22.68	[illegible]	63.33	82.39
17	15.6	2.5	7.05	23.09	12.5	[illegible]	[illegible]	69.74	[illegible]	[illegible]	[illegible]	78.63
18	15.4	2	11.75	38.47	12.5	1.92	1.92	[illegible]	[illegible]	[illegible]	[illegible]	74.69

Fig. [illegible]

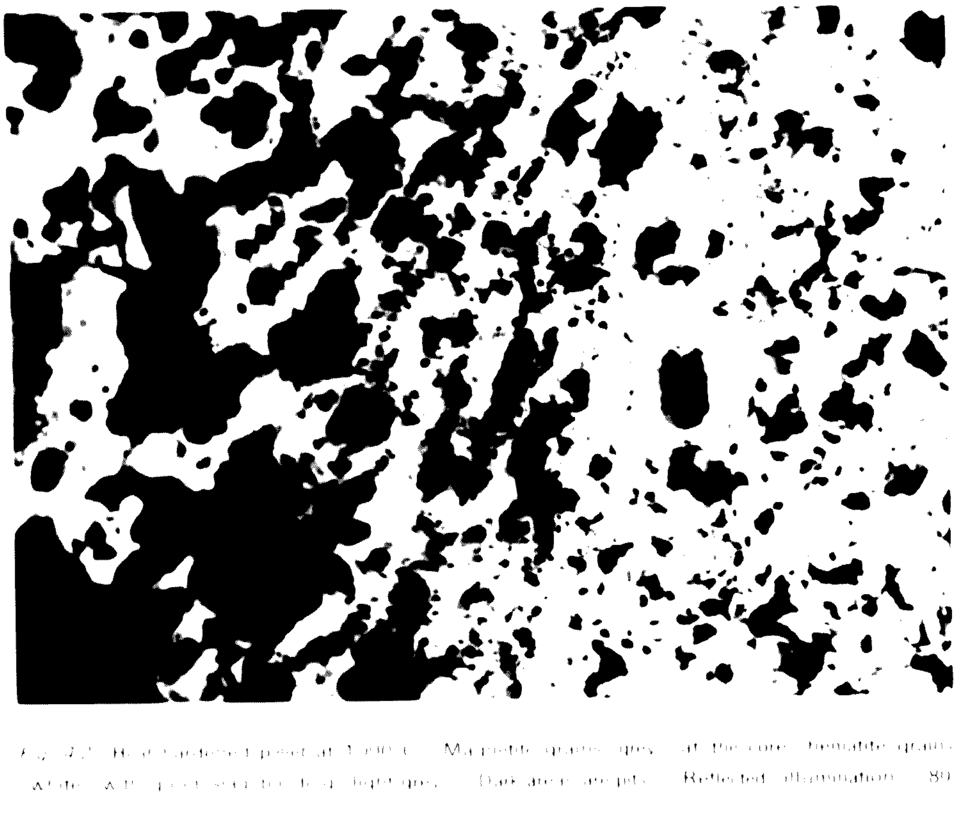


Fig. [illegible] Heat-hardened pellet at [illegible] C. Magnetite grains (grey) at the core; hematite grains (white) with [illegible] (light grey). Dark areas are pits. Reflected illumination ×80

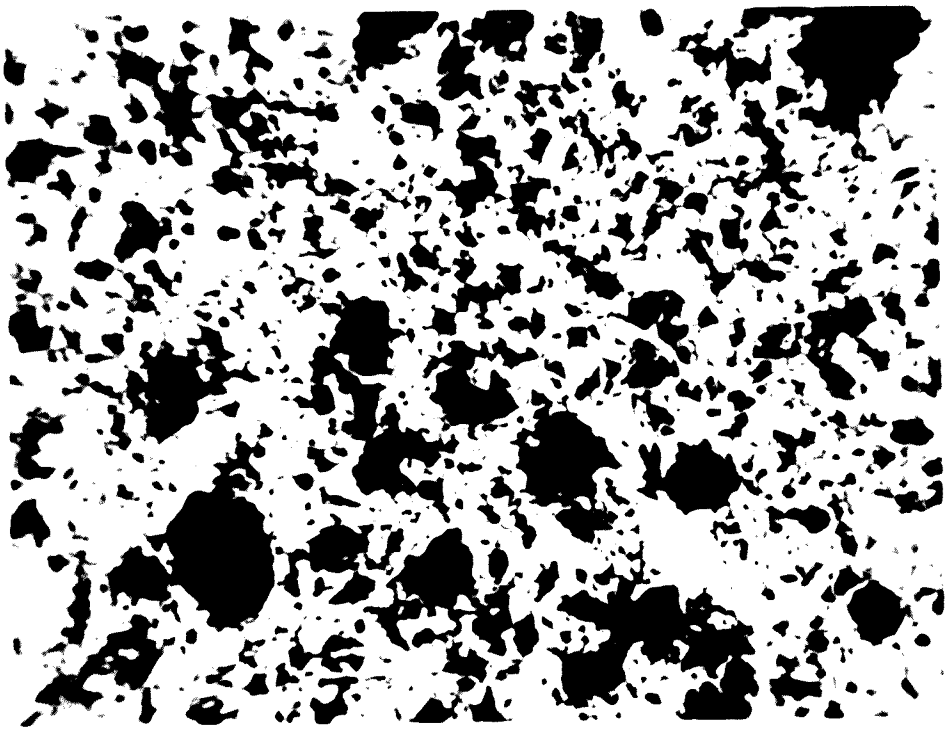


Fig. 4.3 Heat treated pellet at 1[illegible] showing [illegible] [illegible] [illegible] [illegible] and dendrites [illegible] spinel (grey). Reflected illumination ×[illegible]

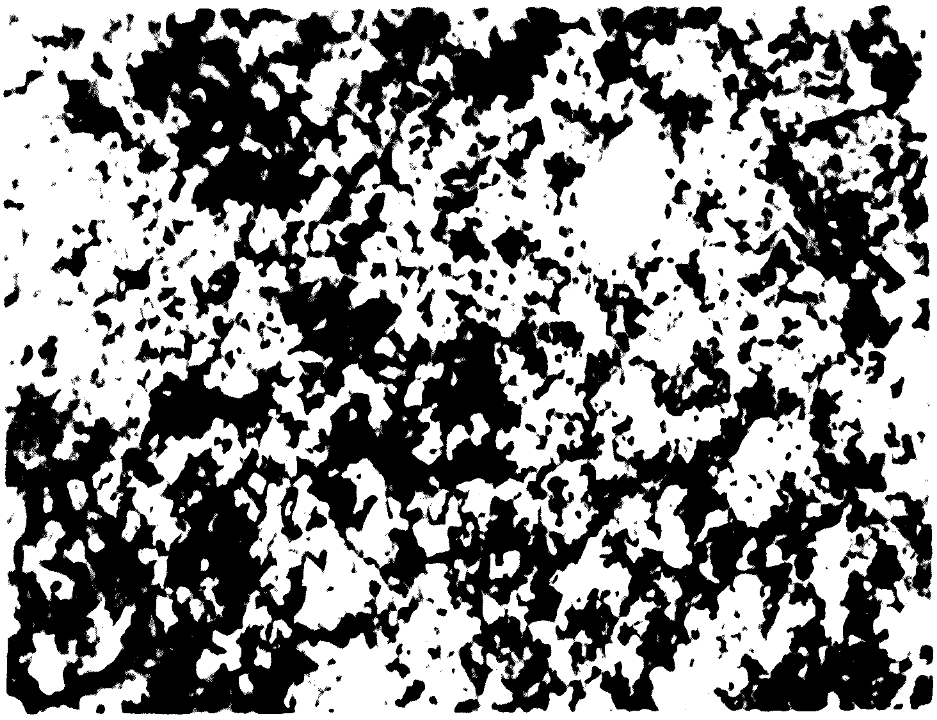


Fig. 4.4 Showing re-crystallised magnetic grains (white) in a calcic olivines and ferrite matrix (grey). Reflected illumination ×120

productivity decreased. But however, it was seen that coke can be replaced to an extent of 15.39% where a good strength of sinter is obtained at a little sacrifices of productivity.

Weathering test

A portion of + 9.6 mm sample (basicity = 1.8) was allowed to stay in atmosphere and it was screened at regular intervals (4 hrs) to find -9.6 mm produced. However, it was found that no-9.6 mm size was produced even after 24 hrs, showing that there is no excess free CaO to slake.

Table 4.16: Weathering test (basicity ratio = 1.8)

	50.8 + 9.6 mm	3.440 kg	% -9.6 mm
48 hrs.	-9.6 mm	0.029 kg	0.8429
72 hrs.	-9.6 mm	0.040 kg	1.162

Chemical analysis:

Chemical analysis of some typical sinters are recorded in Table 4.17.

PETROLOGICAL STUDIES ON SINTERS

Megascopic Observations

The self-fluxed sinters formed from the preballing of the concentrate obtained from the magnetic separation treatment after low temperature reduction roasting of the R.O.M. ore were black in colour with less glaze, feebly magnetic, comparatively less porous and friable having good physical strength. Occasional white inclusions on the surface of unreacted lime were present around the sinter masses.

Microscopic Characteristics

Microscopic examination of the various polished specimens of the sinters of different basicities and their powder mounts revealed that they consisted essentially of magnetite crystals (Fig. 4.1) which were formed by re-crystallization while in plastic state of the earlier formed magnetite during reduction roasting operation of the ore. Minor amounts of maghemite and hematite were present due to oxidation of magnetite. The large amount of magnetite in the pellets facilitated its reaction with lime and silica during sintering resulting in

Table 4.17 - Chemical analysis of typical sinters (%)

Test No.	Basicity	Fe	FeO	SiO_2	Al_2O_3	CaO	MgO
11	1.8	36.58	14.27	14.54	5.90	25.63	1.56
12	2	36.52	10.10	13.74	4.37	27.50	1.44
14	2.5	35.96	8.95	12.77	5.92	28.45	0.90

calcium olivines and lesser calcium ferrites which formed the matrix and act as bonding. Some minor amounts of hercynite and fayalite were also observed.

Conclusion

Thus it is seen that sintering of preballed fine magnetic concentrate (used for pelletizing work) with limestone and coke gives a productivity of 20 - 22 T/m^2/day. Optimum conditions found are water 16%, R/F 20%, Coke 3.25%, b = 1.8 and coke replaced by coke oven gas to an extent of 15.39%. But, however, it is to be noted that because of fineness of concentrate, greater quantity of water is required than it is normally practised (i.e. 6 - 8% water). The sinter is found to be independent of weathering action.

CHAPTER 5

PRE-REDUCTION

STATIC BED REDUCTION TESTS

In order to get an idea of the retention time, temperature at which and the extent to which the ore can be reduced and the amount of reduction under the given experimental conditions, a few static bed reduction tests were carried out with Aswan Iron Ore, green and heat-hardened pellets, with non-metallurgical coals.

The experiments were conducted by heating the pellets and coal (layer charged) in a furnace upto the requisite temperature (900-1150°C) and maintaining it at that temperature for a pre-determined period (45-150 minutes). It was observed that at 1000°C, 94% of the removable oxygen in the pellets could be removed, in the first 45 minutes of heating and the balance in the subsequent 15-30 minutes.

Some experiments were also conducted with the addition of limestone to the fluxed pellets and coal. It was observed the addition of limestone causes a decrease in the rate of reduction. A more important feature, however, was the considerable agglomeration of the pellets during the reduction.

The results of the above experiments are shown in Table 5.1 from which it can be seen that:

(i) above 96% reduction could be obtained in 60 minutes without limestone addition.

(ii) in the presence of limestone, the reduced pellets showed agglomeration, the degree of which increased when the temp. reduction was raised from 1000°C to 1150°C.

From the results it may be inferred that if the rotary kiln is operated at temperatures above 1000°C, agglomeration of the reduced pellets would cause considerable difficulty in the kiln operation.

Pre-reduction of the green pellets was conducted in the rotary kiln described in Chapter 4 using Wardha Valley coal (Ghugus colliery) as solid reductant. Unfluxed pellets (Chapter 4) made from Aswan

Table 5.1 - Static bed reduction tests on Aswan iron ore fluxed pellets with non-coking coals

	Sl. No.	Temp. °C	Duration of expt. in mts.	Charge			Percentage reduction	Remarks
				Coal	Pellet	L. S.		
Experiments with Heat Hardened Pellets.	1.	1000	45	1	1	0	94.48	-
	2.	1000	60	1	1	0	97.58	-
	3.	1000	90	1	1	0	99.13	-
	4.	900	150	1	1	0	88.18	-
	5.	1100	150	1	1	0	99.13	-
	6.	1150	60	1	1	0	99.9	-
	7.	1000	90	1	1	0.25	92.92	Some agglomeration of reduced pellets was observed
Experiments with Green Fluxed Pellets	8.	1000	60	1	1	0.25	91.38	Agglomeration of the reduced pellets observed.
	9.	1100	60	0.5	1	0.20	97.5	Agglomeration of the reduced pellets observed.
	10.	1150	60	1	1	0.25	99.13	About 50% of the pellets were in agglomerated mass.
	11.	1100	60	0.5	1	0	99.13	-

ore magnetic concentrate with 1% bentonite constituted the feed to the rotary kiln. The chemical and sieve analyses of the pellets are recorded in Tables 5.2 and 5.3 respectively.

Table 5.2 - Chemical analysis of pellets

Constituent	Percentage
Fe	51.2
SiO_2	10.2
Al_2O_3	4.3
CaO	3.6
MgO	1.2
MnO	0.78
TiO_2	0.38
P	0.09
S	0.08
L.O.I.	4.96

Table 5.3 - Sieve analysis of pellets

Size	Wt. %
+12 mm	6.3
-12 mm + 9 mm	23.7
-9 mm + 6 mm	56.2
-6 mm	13.3
	100.0

The chemical and sieve analysis of the Wardha valley coal are recorded in Tables 5.4 and 5.5 respectively. The coal was crushed to 18 mm and the -6mm fraction was screened out before charging into the furnace.

The process:

The process consisted of forming green pellets in a conventional manner using disc pelletizer and charging green pellets into a rotary kiln along with coal. The rotary kiln was heated at the discharge end

Table 5.4 - Chemical analysis of coal

Fixed C	42.6%
Volatile matter	32.46%
Ash	20.37%
Moisture	4.57%

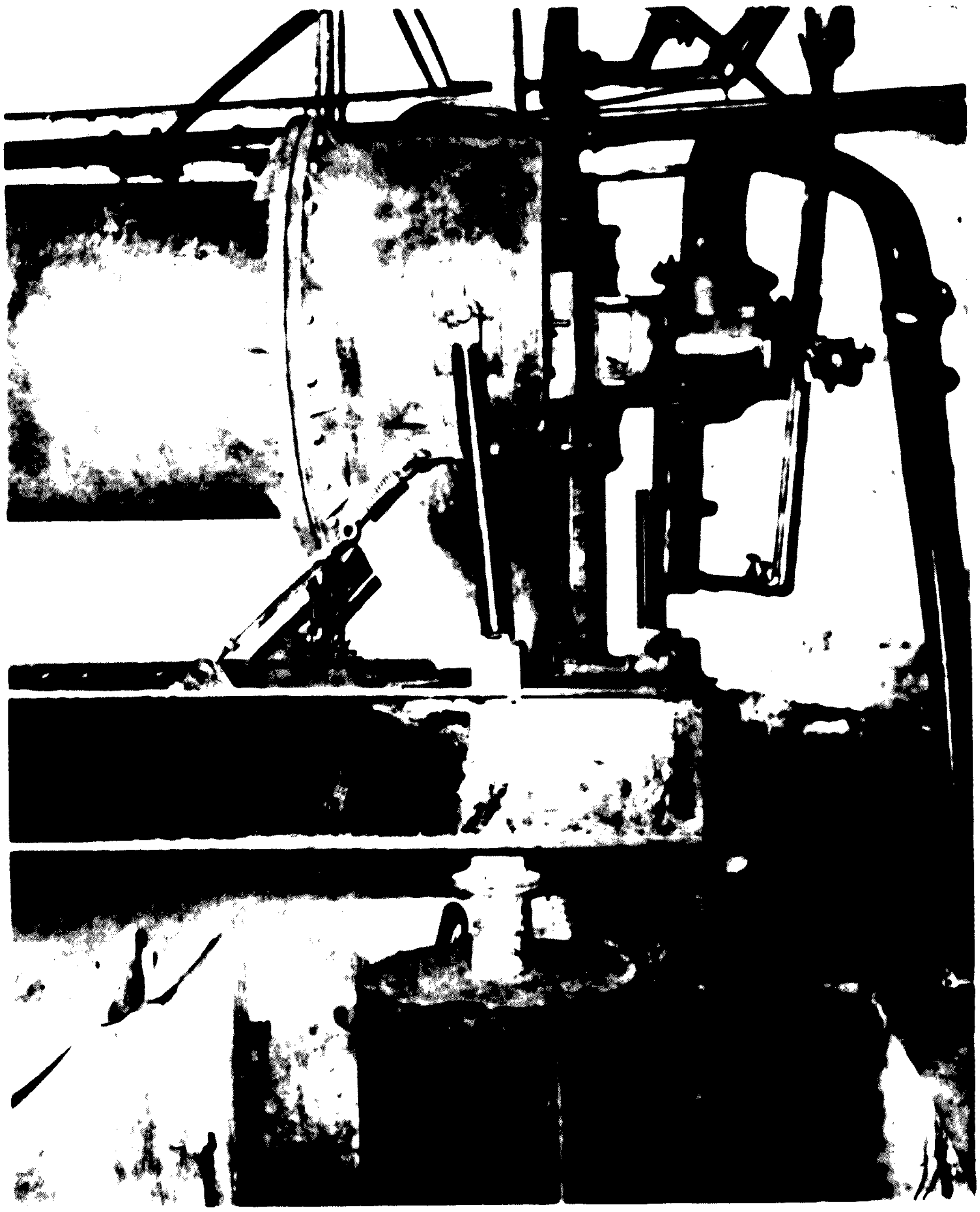
Table 5.5 - Sieve analysis of the coal

Size	Wt. %
+18 mm	10.2
-18+12 mm	32.9
-12+6 mm	46.2
-6 mm	10.7
	100.0

with a single inspirating type oil burner. The inclination of the kiln was 6° from the horizontal position, the r.p.m. was kept at 0.5. Four thermocouples, T1, T2, T3 and T4 were fitted on the kiln at various distances from the burner end for continuous measurement of temperature inside the kiln. The secondary air to the furnace was kept at the minimum so that only slight positive pressure existed within the kiln at all times. A retaining ring was used at the discharge end to increase bed depth and retention time for reduction. The kiln discharge was by gravity through an air-tight chute in air-tight water-jacketed 200 litre drums held tightly against the chute bottom so as to prevent air ingress into the kiln. The drums were partially cooled in a small water bath while being filled and when filled, transferred to a larger bath for further cooling. Physical tests of pre-reduced pellets consisted of determining compression strength and tumbler index. The kiln discharge, when cool, was subjected to magnetic separation for the removal of char, ash etc. as a non-magnetic product and then screened over a 6 mm screen to remove any fines produced during the reduction.

Experimental Results

The rotary kiln was operated continuously during the campaign; the operational data for the entire campaign is recorded in Table 5.6. The ratio of pellets to coal was maintained at 1:1 to compensate the losses of reductants as char, flue dust, exertion to the furnace wall etc. The raw material feed rate was kept at 150 kg of pellets and 150 kg of coal per hour.



Discharge end of the rotary kiln for sponge making

[illegible]

Table 5.6 - Operational data with Aswan ore pellets

Oil consumed lit/operation	Qty. of pellets charged tonnes/day of operation	Hours run	Temperature °C				Assay %		
			T1	T2	T3	T4	Fe (met)	FeO	Fe (Total)
125	1.8	12	1040	860	530	280	n. d.	n. d.	n. d.
172	3.6	24	1030	860	560	270	59.5	n. d.	63.1
240	3.6	24	1060	870	580	280	53.2	n. d.	61.9
204	3.6	24	1050	860	570	240	51.1	6.3	61.0
147	3.6	24	1050	850	560	250	44.7	11.0	59.1
191	3.6	24	1060	860	550	250	51.1	5.2	62.0
33	1.2	8	1030	850	560	270	40.6	22.0	58.8
180	2.7	18	1040	860	580	280	51.8	5.0	54.4
190	3.6	24	1030	850	570	280	49.7	12.8	60.2
237	3.6	24	1050	860	570	290	53.2	7.2	59.1
186	3.6	24	1060	850	580	280	47.6	11.9	57.9
136	3.6	24	1070	850	590	290	55.3	9.9	64.7

Average oil consumption

$\frac{2041}{254}$ = 8 litres/hr

or 60 litres/tonnes of pellet

Average degree of metallisation = $\frac{\text{Metallic Fe x 100}}{\text{Total Fe}}$ = about 85%.

The results indicated that the pre-reduction attained throughout the entire operation of the kiln was satisfactory. The waste gas analysis of the rotary kiln is recorded in Table 5.7.

The compression strength is the ability of a pellet to withstand a compressive load before cracking and was determined with a Amsler Universal testing machine. The tumbler test consisted of tumbler test consisted of tumbling 5 kg. of +4 mesh pellet in a standard ASTM drum for 200 revolutions at 24 r.p.m. The drum, 914 mm dia by 437.2 mm long was fitted with two diametrically opposite 50.8 mm lifters to exaggerate cascading of the contents. +8 and -28 mesh fractions obtained after screening defined as tumble and abrasion indices respectively. Metallisation, the percentage fraction of metallic to total iron, was determined from chemical analyses.

Table 5.7 - Exit gas analysis from the rotary kiln

	Vol. %		
	CO_2	O_2	CO
1.	12.6	1.2	3.6
2.	13.8	1.8	2.0
3.	14.22	1.18	2.75
4.	13.77	2.36	3.15
5.	14.44	2.2	3.8

Average CO_2/CO ratio of exit gases = approx. 4

Table 5.8 - Additional data

(a) Chemical analysis of flue dust.

(i)		Per cent
	Fixed C	3.76
	Ash	91.2
	Volatile matter	4.28
(ii)	Analysis of ash	Per cent
	Fe	49.28
	SiO_2	14.80
	Al_2O_3	7.40
	CaO	3.93
	MgO	2.01

Table 5.9 - **Sieve analysis of flue dust**

Size	Per cent
+100 mesh	2.0
-100+150 mesh	1.0
-150+200 mesh	1.0
-200+250 mesh	1.5
-250 mesh	94.5
	100.0

Table 5.10 - **Chemical analysis of char**

	Constituent	Per cent
(i)	Fixed C	54.73
	Ash	38.41
	Volatile matter	5.56
	Moisture	1.30
(ii)	Analysis of Ash	
	Fe	11.20
	SiO_2	53.76
	Al_2O_3	26.30
	CaO	1.68
	MgO	1.02
	S	0.48
	P	0.28

Table 5.11 - **Sieve analysis of char**

Size	Wt. %
+18 mm	0.2
-18+12 mm	24.1
-12+6 mm	42.7
-6 mm	33.0
	100.0

The operation was evaluated on the following factors: (a) average compression strength of the pre-reduced pellets, (b) size indices from tumbling test, (c) abrasion index, (d) extent of metallization, (e) efficiency of iron recovery and the material balance, (f) heat balance and efficiency of the furnace, and (g) the quantity of accretions formed on the kiln wall.

The recovery was calculated from the proportion of iron reporting in the pre-reduced pellets as compared to the total amount of accounted iron. This included, apart from the pellets, -6 mesh material produced after screening of the magnetic product, accretions removed from the kiln walls after the end of operation, flue dust and the unreduced iron oxide associated with the char.

The ideal operational efficiency would be to produce pellets with high compression strength and tumble indices and optimum metallisation during pre-reduction. The results are recorded in Table 5.12. No universal standards exist for stipulating physical properties of indurated pellets but a tumbler abrasion index of 97 per cent and a 100 kg minimum compression strength were considered as being satisfactory.

Table 5.12 - Test results and physical properties of pre-reduced pellets

No. of test conduced	Crushing strength kg/pellet	Tumbler Test		Total Fe	Metallic Fe	% Degree of metallisation
		+8 mesh	-28 mesh			
1	185	98.5	0.7	61.6	55.3	89.0
2	202	98.3	0.8	63.24	54.6	87.0
3	205	97.9	0.9	54.64	54.6	91.0
4	192	97.8	0.8	59.36	47.6	80.0
5	195	97.5	0.9	59.08	49.7	84.0
6	198	98.1	0.5	57.40	44.1	78.0

The iron recoveries, recorded in Table 5.13 were determined after the end of the operation and treatment of the discharge material by magnetic separation and screening, collection of stack dust and accretion formed in the wall of the furnace.

Table 5.13 - Iron recovery in pre-reduced pellets

Product	Wt. %	Assay % Fe	Distribution % Fe
Pre-reduced pellets of 85% metallization	76.5	62.2	92.4
Flue dust	1.4	49.3	1.3
-6 mesh fines	1.2	31.3	0.7
Accretion	1.5	48.0	1.4
Char	19.4	11.2	4.2
Heac (calc.)	100.0	51.54	100.0

The iron recovery could be improved by reclaiming the char and re-circulation of stack dust and -6 mesh screened product.

Material balance		Basis one hour Total wt.
Input	Iron ore pellets containing 14% moisture	150 kg.
	Coal containing 32.46% V.M. 4.57% moisture	150 kg.
		300 kg.
Output	Pre reduced pellets	98.7 kg.
	Flue dust	3.1 "
	-6 mesh fines	2.7 "
	Accretions	3.4 "
	Char	43.4 "
	Other losses (e.g. flue gases etc.)	148.7 "
		300.0 kg.

Heat Balance

Basis 1 hour.

The calorific value of the fuel oil is 11,700 Kcal/kg and for the volatile matter, it has been estimated at 9,700 Kcal/kg. The calorific value of carbon is 8,120 Kcal/kg.

(a) **Heat input**

	Kcal	%
(i) Combustion of 53.6 kg. carbon	434,000	78.0
(ii) Combustion of 48.7 kg. of volatiles	48,600	8.5
(iii) Combustion of 6.4 kg. of fuel oil	75,000	13.5
	557,600	100.0

(b) **Heat output**

	Kcal	%
(i) Reduction of oxides	122,700	22.0
(ii) Sensible heat of storage upto 1030°C	69,000	12.4
(iii) Evaporation of water	16,500	3.0
(iv) Sensible heat of flue gases at 280°C and chemical heat of flue gases	195,400	35.0
(v) Radiation loss from kiln shell	154,000	27.6
	557,600	100.0

Thermal efficiency of kiln = 22.0 + 12.4 + 0.7 = 35.1%

PETROLOGICAL STUDIES ON PRE-REDUCED PELLETS

Pellets from the kiln discharge end:

Megascopic Observations

The discharged pellets from the kiln were found magnetic when tested with a hand magnet. They had brownish grey colour. The pellet shape was generally distorted in some cases, perhaps, during their movement through the kiln. Their surface was not very smooth in all cases with some adherent material sticking around resulting in a rough surface. The pellets were hard with no surface cracks.

Microscopic Characteristics

Microscopic examination of the pellets revealed that in cases where reduction was advanced, they consisted of metallic iron grains followed by wustite and magnetite and traces of hematite in a homogeneous non-metallic bond consisting of calcium ferrites, fayalite and iron cordierite (Fig. 5.1).

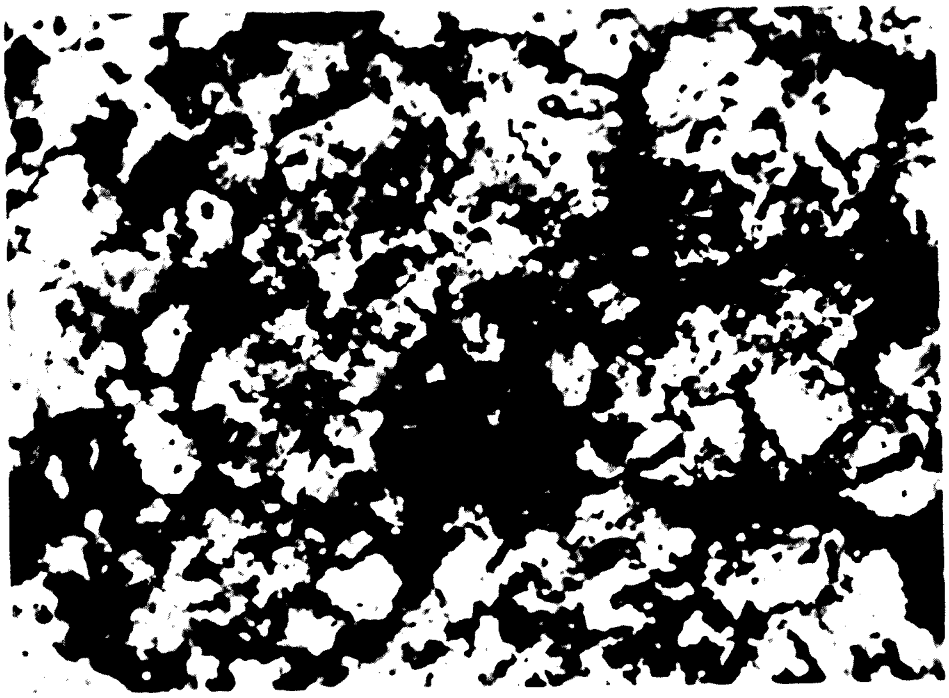


Fig. 5.1 Showing advanced stage of reduction with metallic iron (white), [illegible] [illegible] [illegible] dark grey). Reflected illumination ×120.

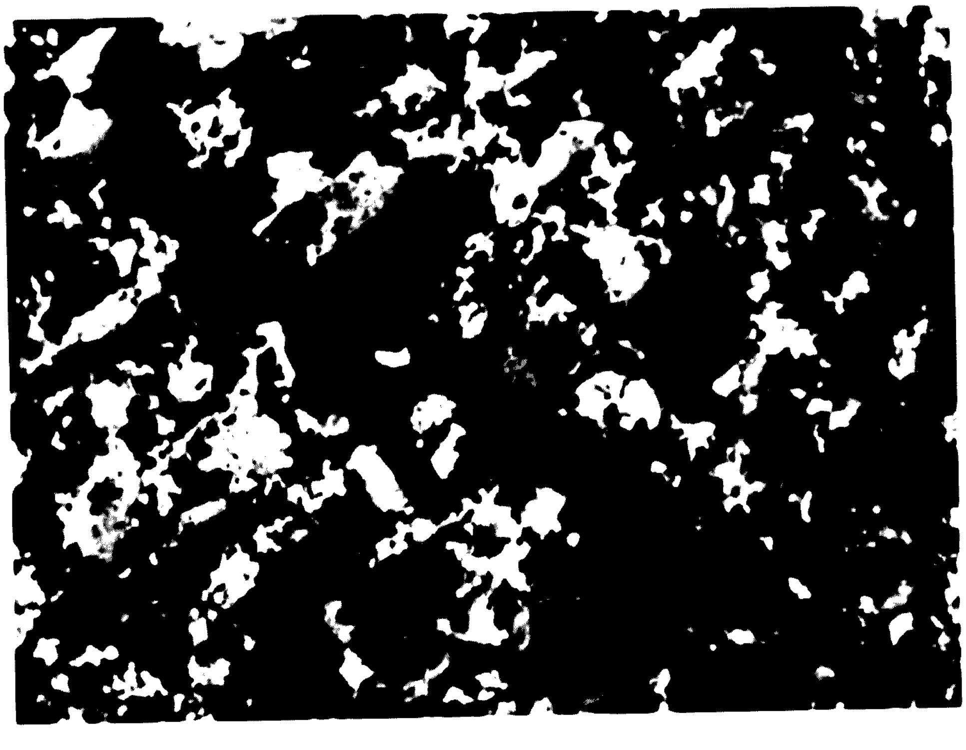


Fig. 5.2 Showing partial reduction of pellet with sporadic iron grains (white), magnetite, wüstite (light grey) in a slag matrix (dark grey). Reflected illumination ×120.

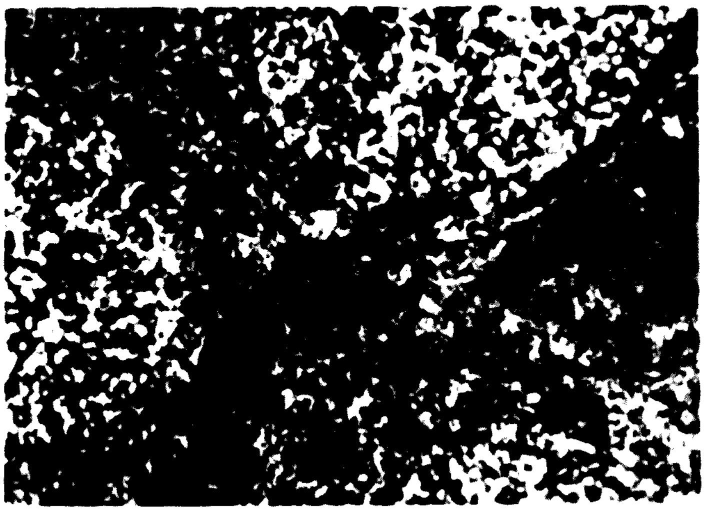


Fig. 5.3 Oolitic iron ore variety light grade [illegible] of [illegible] oolites showing [illegible] uniformity of metallization and [illegible] in the oolite bodies [illegible] the interstitial matrix. Reflected light × [illegible]

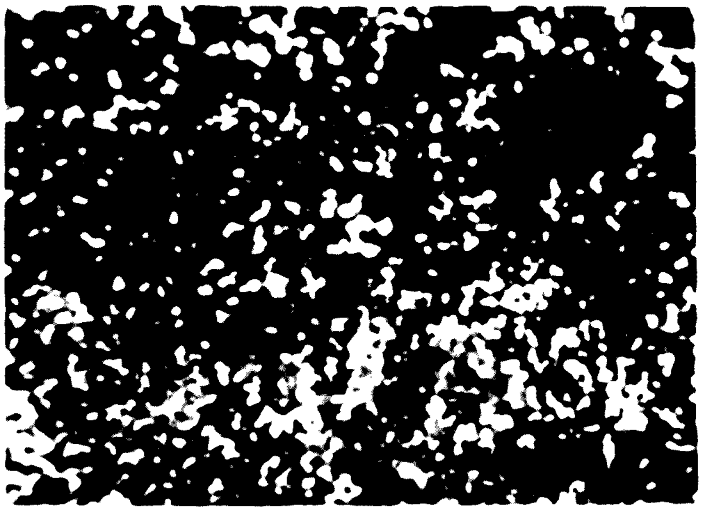


Fig. [illegible] Oolitic [illegible] ore after reduction to [illegible] under progress. White [illegible] are metallic iron and [illegible] are [illegible] magnetite [illegible] complex. Reflected light × [illegible]

Fig. 5.5 Oolitic [illegible] grade [illegible] showing [illegible] of reduction both [illegible] as well as [illegible] very fine metallic [illegible] Reflected light × [illegible]

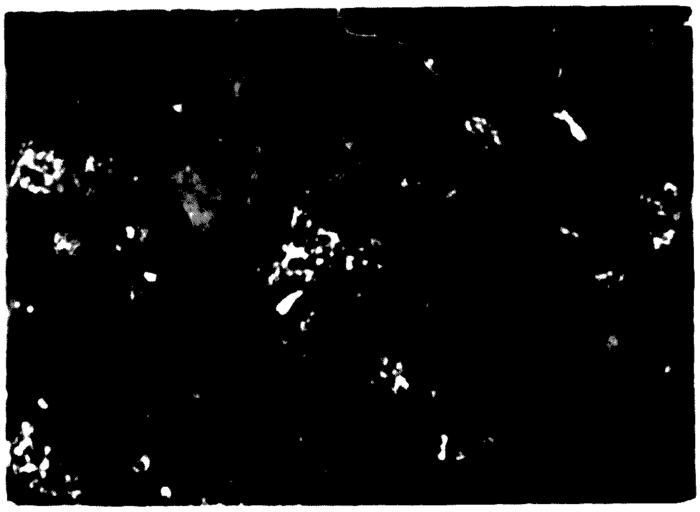


Fig. [illegible] Ferruginous sandstone [illegible] showing [illegible] degree of reduction with [illegible] metallic iron [illegible] interstitial matrix. Dark grey [illegible] are quartz [illegible] light grey [illegible] are magnetite. Reflected light × 300

The reduction stage had not been much advanced in some cases with the result that only scanty appearance of metallic iron grains had resulted (Fig. 5.2) with some amount of magnetite followed by wustite in a matrix consisting mostly of faylite and iron cordierite. The magnetite was seen recrystallized resulting as very fine cubic grains along with wustite and fine metallic iron grains. Some re-oxidation of magnetite to hematite was also observed in the form of very thin rims around big holes.

PETROLOGICAL STUDIES ON DIRECT REDUCTION OF LUMPS

Megascopic Observations

The colour of the reduced lumps varied from black, dark grey to whitish grey and different from lump to lump depending on its inherent mineralogical nature. Some of them were quite strong and offered some resistance and attained metallic lustre when ground against grinding wheel, whereas some other were absolutely weak and powdery and crumpled. The former were highly magnetic while the latter were very feebly magnetic when tested with a hand magnet.

Microscopic Characteristics

Microscopic examination of the polished sections of the reduced lumps indicated that direct reduction into metallic iron stage was there. But the extent and degree of reduction varied from lump to lump depending on their original mineralogical nature. Thus the high grade ollitic variety had showed the highest extent of metallisation both in the oolite body and the interstitial matrix even though non-uniformity of grain size of the metallic iron grains ranging from 0.05 mm to 0.002 mm and averaging 0.024 mm was noticed (Fig. 5.3). Further the core of some of the oolites showed progressive reduction stage wherein the oxides of iron (magnetite-wustite complex) and metallic iron grains were present side by side (Fig. 5.4). In another lump of lów grade oolitic variety sluggishness of reduction both in the oolite bodies as well as in the matrix resulting in very fine metallic iron grains of 0.001 to 0.008 mm, averaging 0.004 mm distributed at random was noticed (Fig. 5.5). Very poor degree of reduction was observed in the ferruginous sandstone variety lumps wherein minimum number of metallic iron grains of very fine size ranging 0.012 mm to 0.002 mm, averaging 0.002 mm were found in the interstitial matrix (Fig. 5.6). In other lumps, which were very feebly magnetic, the presence of the oxides of iron or metallic iron grains wae very meagre and insignificant.

Conclusions

From the above studies it was once more aecertained the varied mineralogical nature of the ore which had directly resulted in varying degree of metallization in the different lumps on direct reduction. Further the grain size of the metallic iron grains ranged from 0.05 mm to 0.001 mm which further indicated the absolutely fine grained interlocking nature of them with the associated siliceous gangue matrix. The very high degree of reducibility to metallic iron stage was a welcome sign of the behaviour of the ore to direct reduction but the very fine grained nature of the resulted metallic iron grains would lead into much difficulty for their effective separation from the associated gangue by magnetic separation. As such obtaining of a much cleaner iron-rich magnetic concentrate of very high grade with minimum of siliceous gangue may be a difficult proposition.

CHAPTER 6

ELECTRIC SMELTING OF ASSWAN IRON ORE PELLETS

Trials were conducted on heat hardened and pre-reduced pellets of Asswan Iron Ores, prepared as described earlier, in a 500 KVA submerged arc furnace, to study their behaviour during electric smelting. Parameters to be particularly evaluated were the slag metal ratio, production rate, coke rate, and power consumption as functions of the degree of pre-reduction.

Plant and Equipment Used

A sub-merged arc furnace of BIRLEFCO make having a capacity of 500 KVA was used for the smelting trials of the processed Asswan Ores. This sub-merged arc furnace is energized through a three phase transformer rated at 500 KVA with an operating range of 55 volts to 100 volts in steps of 7.5 volts. The furnace is open type and equipped with arrangements for three 200 mm pre-baked graphite electrodes, a suitable fume extraction and dust collection system. Water cooling arrangements are also provided.

Furnace shell diameter is 2.40 meters (internal) with a lining thickness of 425 mm. Suitable temperature measurement devices linked up with a central control panel are also installed at appropriate points.

The facility is fully equipped with all the necessary ancillaries such as conveying and handling systems, weighing machines, tapping and casting arrangements, etc.

Raw Materials

1. **Aswan Iron Ore**

Processed Asswan iron ore in the form of heat hardened pellets and pre-reduced pellets of varying degree of pre-reduction formed the basis of the blended burden fed into the submerged arc furnace.

The pre-reduced pellets were received from the kiln in a large number of batches of varying degrees of pre-reduction. These were blended to form three lots, one each of 80%, 70% and 50% degree of metallization.

The heat hardened lot of pellets, and the three graded lots of pre-reduced pellets were mixed with calculated quantities of screened pearl coke, limestone and dolomite to form a furnace burden of acceptable porosity and material content.

The chemical analysis of the heat hardened pellets and the various pre-reduced lots are presented in Table 6.1 followed by the screen analysis of these materials in Table 6.2.

Table 6.1 - Chemical analysis of heat hardened and pre-reduced pellets

%	Heat hardened pellets	Pre-reduced pellets		
		80%	70%	50%
Fe Total	51.00	-	-	-
Fe Metal	-	49.40	39.15	30.66
FeO	-	23.72	34.66	39.41
SiO_2	12.00	13.43	13.09	14.42
Al_2O_3	6.80	5.53	5.39	8.14
CaO	3.42	4.74	4.62	4.11
MgO	1.30	1.73	1.69	1.56
S	0.14	0.11	0.13	0.2
Phos	0.90	1.34	1.27	1.10
MnO	Trace	-	-	-
TiO_2	0.34	-	-	-
LO_I	2.00	-	-	-

Table 6.2 - Screen analysis of heat hardened and pre-reduced pellets

Size	Heat hardened pellets	Pre-reduced pellets		
		80%	70%	50%
+12.7 mm	42.32	18.18	19.82	2.23
-12.7 + 9.5 mm	17.84	40.91	44.24	58.72
- 9.5 + 6.3 mm	19.21	27.27	22.10	24.47
- 6.3 + 3.2 mm	20.63	13.64	13.84	11.58

Reducing Agents

The reducing agent utilized in these electric smelting trials was pearl coke supplied by Hindustan Steel Limited, India.

The chemical analysis of the pearl coke is presented in Table 6.3 and the screen analysis in Table 6.4.

Table 6.3 - Chemical analysis of pearl coke

	Percent
Fixed carbon	69.34
Volatile matter	1.96
Moisture	0.72
Ash	27.98
Ash analysis	
SiO_2	52.06
Al_2O_3	29.70
MgO	1.91
CaO	2.00
S	0.13
Phos	0.40

Table 6.4 - Screen analysis of pearl coke

Size		Percent
-19.05 + 12.7 mm		25.53
-12.7 + 9.52 mm		27.65
- 9.52 + 6.35 mm		17.03
- 6.35 + 3.18 mm		29.79
	Total	100.00

Fluxes

Two fluxing agents were utilized (i) limestone from Aswan; and (ii) dolomite - to neutralize the effect of the high alumina in the pellets.

The chemical analyses of the Asswan limestone and the Indian dolomite utilized are given in Table 6.5 and their sieve analysis in Table 6.6.

Table 6.5 - Chemical analysis of Asswan limestone & Bihar dolomite

Oxide	Limestone (%)	Dolomite (%)
CaO	50.4	28.67
SiO_2	2.06	2.70
MgO	2.61	21.00
Al_2O_3	1.12	
Fe_2O_3	0.80	
CO_2	40.94	
SO_3	0.45	
Phos	0.035	
LOI	41.89	
Others		2.00

Table 6.6 Screen analysis of fluxes

	Limestone	Dolomite
-50.8 + 25.4 mm	7.69%	0.92%
-25.4 + 19.0 mm	19.20%	47.84%
-19.0 + 12.7 mm	49.20%	23.29%
-12.7 + 6.3 mm	23.88%	27.95%

Furnace Burden

Burden Preparation

The furnace burden was prepared using calculated quantities of raw materials. The basic iron source material consisted of the heat hardened pellets and the three lots of pre-reduced pellets of varying degrees of pre-reduction described earlier.

This source material was blended with pearl coke, limestone and dolomite to form individual charging lots which were fed into the furnace called for by process conditions.

The pellets were normally screened to remove fines (-3 mm) which were present upto 15%. This step was taken to ensure good bed porosity and to facilitate burden descent in the furnace.

Burden Calculations

For calculating the burden, the following chemical reactions for the reduction of various elements have been considered.

$$Fe_2O_3 + 3C = 2Fe + 3CO$$

$$FeO + C = Fe + CO$$

$$SiO_2 + 2C = Si + 2CO$$

$$MnO + C = Mn + CO$$

$$P_2O_5 + 5C = 2P + 5CO$$

The following assumptions have been made while calculating the burden on the basis of previous operating experience for the production of pig iron by electro-thermic smelting in the 500 KVA furnace:

a) The distribution of elements in the metal and slag is shown in Table 6.7.

Table 6.7 - % Distribution of elements to the metal, slag and fumes

Element	To metal	To slag	To fumes
Iron	99.5	0.5	-
Silicon	20.0	70.0	10.0
Manganese	80.0	10.0	10.0
Phosphorous	70.0	-	30.0
Sulphur	5.0	25.0	70.0

b) The pig iron to contain 3.5% carbon.

c) The basicity of the slag to be maintained at 1.2 not taking MgO into consideration.

d) MgO in the slag to account for 5% by weight of slag.

e) Approximately 25% of excess carbon to account for oxidation losses in the open top smelting furnace (including the carbon necessary for the reduction of SiO_2 and iron oxides arising out of the coke ash).

f) About 10% of the iron oxides reduced by CO while the furnace is in operation with iron ore or its agglomerates while no such reduction to take place when the material being charged in pre reduced condition.

Calculations are as follows

a) The coke requirement for reduction of 100 kg of Asswan Iron Ore (heat hardened pellets) allowing for 25% excess coke and 68.204% active carbon in the coke has been calculated as 33.46 kg.

b) 5 kg of dolomite to be added to each charge for maintaining MgO content around 5% in the slag.

c) The basicity (CaO/SiO_2) of the slag to be 1.2 to reduce the sulphur during the melting process, since the limestone and the ore contain high sulphur viz. 0.20 and 0.14 respectively. The limestone needed to be added to the charge will be

	From 100 kg ore	From 33.46 kg coke	From 5.0 kg dolomite	Total
CaO	3.42	0.187	1.434	5.041
SiO_2	12.00	4.868	0.135	17.003

SiO_2 goin to slag will be 17.003 x 0.7 i.e. 11.902 kg.

Net CaO required at a basicity 1.2 will be 11.902 x 1.2 5.041 i.e. 9.421 kg.

Available lime in Asswan limestone will be 50.4 - 2.06 x 1.2 x 0.7 i.e. 48.670 kg per 100 kg of limestone.

Limestone needed per charge will therefore be approx. 19.0 kg. Hence the average charge composition with the Asswan iron ore agglomerate (heat hardened) is as follows -

Agglomerate	100 kg
Durgapur pearl coke	33 kg
Asswan limestone	19 kg
Dolomite	5 kg

The slag: metal ratio under these conditions may be expected to be 0.72 and the basicity as CaO/SiO_2 ratio will be 1.204. The material balance for the specific composition of charge while smelting will be as shown in Table 6.8.

Table 6.8 – Material balance

Input	kg.	Output	kg.
Agglomerate	100.00	Metal	55.22
Pearl coke	33.00	Slag	39.85
Limestone	19.00	Losses as fumes, etc., (by difference)	61.93
Dolomite	5.00		
	157.00		157.00

SMELTING TRIALS

General Procedure

The cleaned out refractory lined hearth was preheated gradually by means of small coke heaps placed below each of the three electrodes. The furnace load was gradually increased to 200 KW over a period of 72 hours to attain the required process temperature.

The process conditions were then stabilized by using a known iron ore of Indian origin before commencing the charging of Asswan material.

Single tapping of metal and slag was done at intervals of 2.5 to 3 hours. Metal slag separation was achieved by permitting the slag to overflow into an adjacent slag pot.

The metal was poured into moulds and the cast pigs were sampled and analysis done. Slag analysis was also obtained.

Campaign Sequence

The furnace was stabilized with a typical Indian ore and heat-hardened pellets prepared from the Asswan iron-ore were then charged into it. This was followed by pre-reduced Asswan iron ore pellets starting with the 80% pre-reduction grade, and followed by the 70% and 50% pre-reduction grades.

Smelting of Heat Hardened Asswan Iron Ore Pellets

A typical charge consisted of

Heat hardened pellets	100 kg
Pearl coke	33 kg
Limestone	19 kg
Dolomite	5 kg

Charging was regulated to maintain a constant burden height. These heat hardened pellets were run for a total of 18 hours, involving 25 charges and 5 tappings. A total of 1291 kilos of metal and 617 kilos of slag were tapped, with a power consumption of 4248 KWH during this phase.

The slag to metal ratio was 0.82

The metal analysed as

C	1.34 to 1.48%
Si	6.[illegible] to [illegible].86%
Mn	1.2[illegible]% (carried over from previous phase)
P	1.3[illegible] to 1.42%
S	0.0[illegible]%

The slag analysed as

CaO	40.20 to 46.37%
SiO_2	26.[illegible]0 to 27.70%
MgO	2.[illegible]%
Al_2O_3	22.50%

The secondary voltage maintained was [illegible] Volts with an average input of 4500 amps to the electrodes.

The electrode penetration and burden penetration was poor due to the lower burden permeability causing occasional explosive sparking at the top due to gas pocket formations in spite of constant pushing or crush breaking.

(b) Smelting of 80% Pre-reduced Asswan Iron Ore (AIO) Pellets

A typical charge consisted of:

80% Pre-reduced AIO pellets	100 kg
Pearl coke	11 kg
Limestone	17 kg
Dolomite	3 kg

These 80% pre-reduced AIO pellets were run for a total of 16.25 hours involving 32 charges and 5 tappings. A total of 2607 kilos of metal was tapped with the slag/metal ratio being 0.58. The power consumption was 2674 KWH during this phase.

The metal analysed as:

C	2.70 to 3.29%
Si	2.76 to 3.95%
P	1.05 to 1.19%
S	0.010 to 0.012%

The slag analysed as:

CaO	39.76 to 44.18%
SiO_2	26.60 to 33.80%
MgO	2.48 to 4.51%
Al_2O_3	20.80 to 23.84%

The average CaO/SiO_2 ratio in the slag was 1.30. Slag fluidity was excellent.

The secondary voltage maintained was 62.5 volts with an average input of 4000 amps to the electrodes.

The electrode penetration (in the order of 45 to 50 cms) and the burden descent were both good. Burden permeability was also acceptable.

(c) Smelting of 70% Pre-reduced AIO Pellets

A typical charge consisted of:

70% Pre-reduced AIO pellets	100 kg
Pearl coke	13 kg
Limestone	17 kg
Dolomite	3 kg

These 70% pre-reduced AIO pellets were run for a total of 10.25 hours involving 23 charges and 3 tappings. A total of 1221 kilos of metal was tapped, the slag/metal ratio being 0.65. The power consumption was 3317 KWH during this phase.

The metal analysed as:

C	2.43 to 3.02%
Si	2.88 to 4.44%
P	1.31 to 1.50%
S	0.040 to 0.044%

The slag analysed as:

CaO	33.76 to 35.96%
SiO_2	33.00 to 34.90%
MgO	5.00 to 6.54%
Al_2O_3	21.53 to 23.28%

The average CaO/SiO_2 ratio in the slag was 1.08. Slag fluidity was good. The secondary voltage maintained was 62.5 volts with an average input of 4000 amps to the electrodes.

The electrode penetration, burden descent and permeability were good.

(d) Smelting of 50% Pre-reduced AIO Pellets

A typical charge consisted of:

50% Pre-reduced AIO pellets	100 kg
Pearl coke	15 kg
Limestone	17 kg
Dolomite	3 kg

These 50% pre-reduced AIO pellets were run for a total of 15.5 hours involving 27 charges and 4 tappings. A total of 1770 kilos of metal was tapped with the slag/metal ratio being 0.68. The power consumption was 3400 KWH during this phase.

The metal analysed as:

C	2.16 to 2.65%
Si	4.16 to 5.60
P	1.23 to 1.26%
S	0.01 to 0.03%

The slag analysed as:

CaO	35.34 to 37.86%
SiO_2	29.60 to 30.45%
MgO	6.54 to 6.87%
Al_2O_3	21.80 to 27.33%

The average CaO/SiO_2 ratio in the slag was 1.21. Slag fluidity was good.

The secondary voltage maintained was 62.5 volts with an average input of 4000 amps to the electrodes.

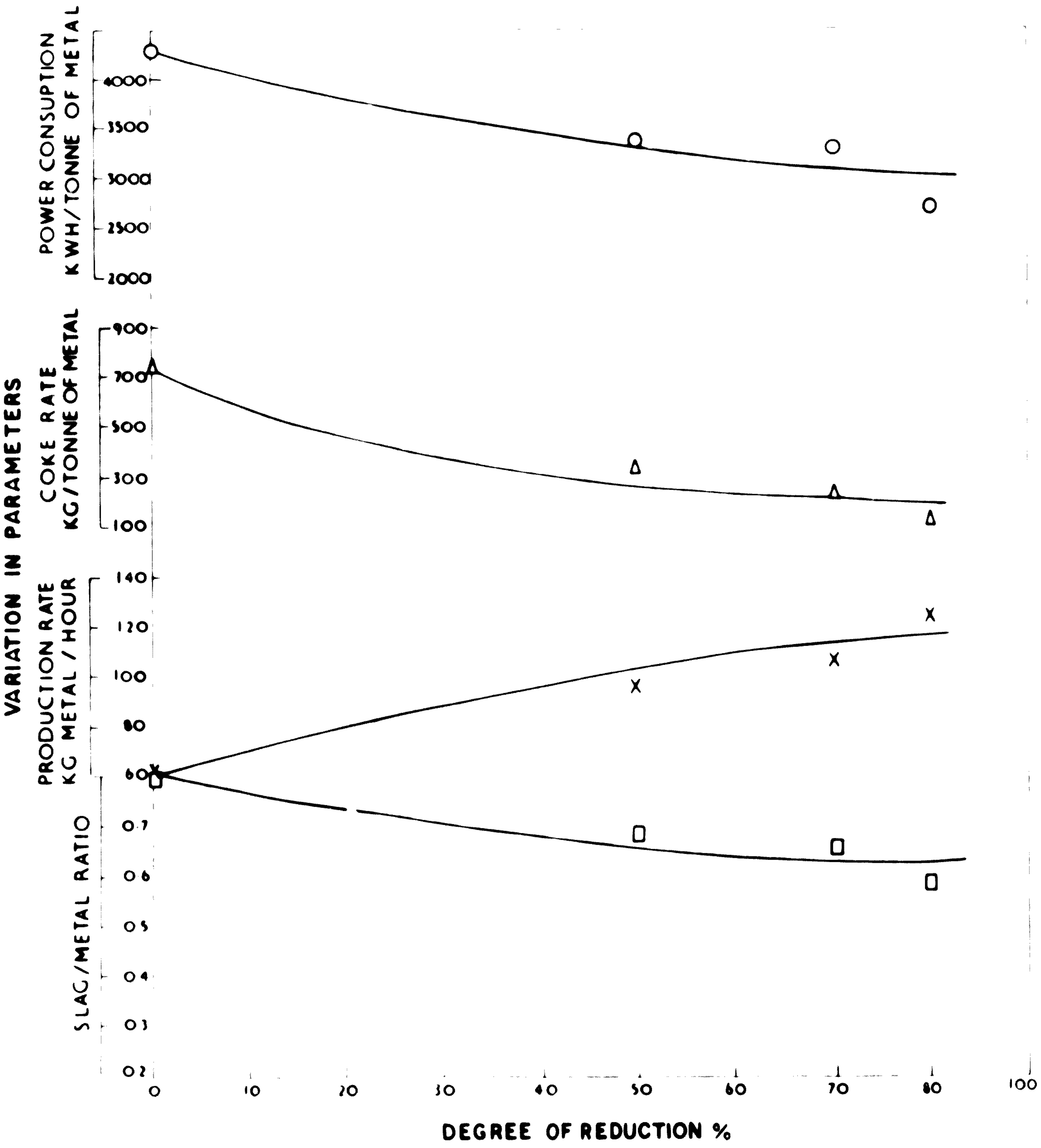


Fig 6.1 Results obtained from the electric smelting trials of Asswan iron ore

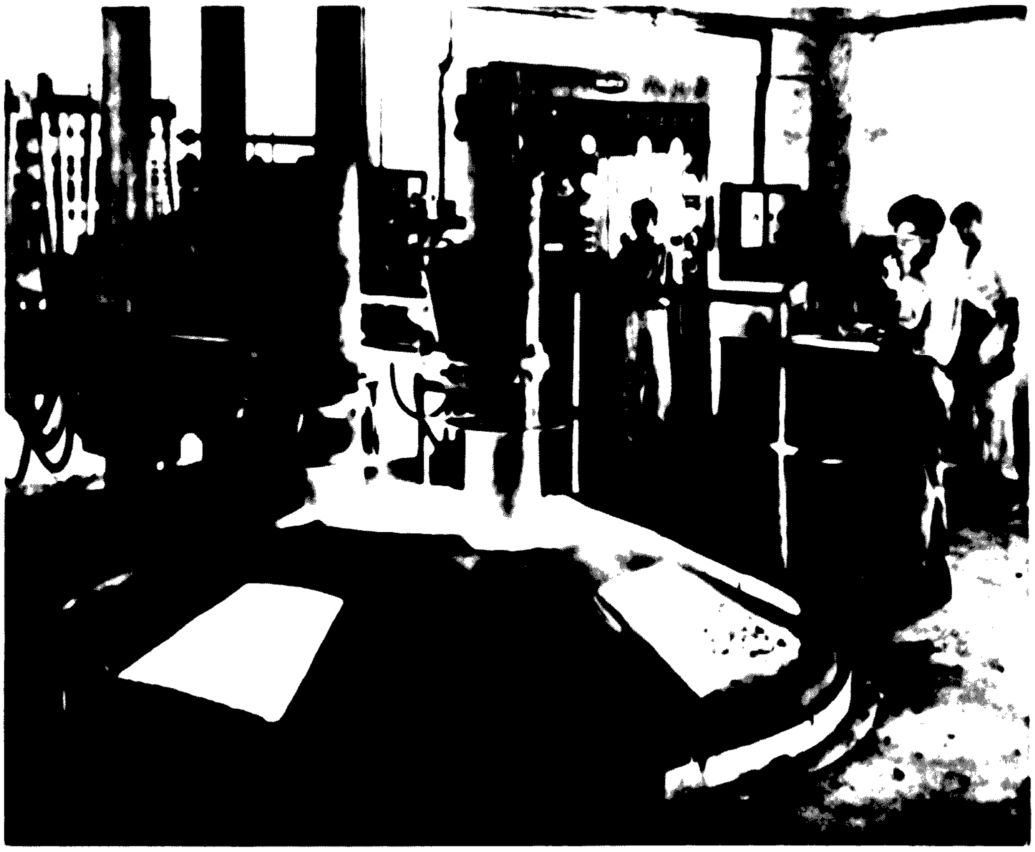


Fig 6.2 Top view of the sub merged arc electric smelting furnace showing the smelting operation of pre reduced Asswan iron ore pellets

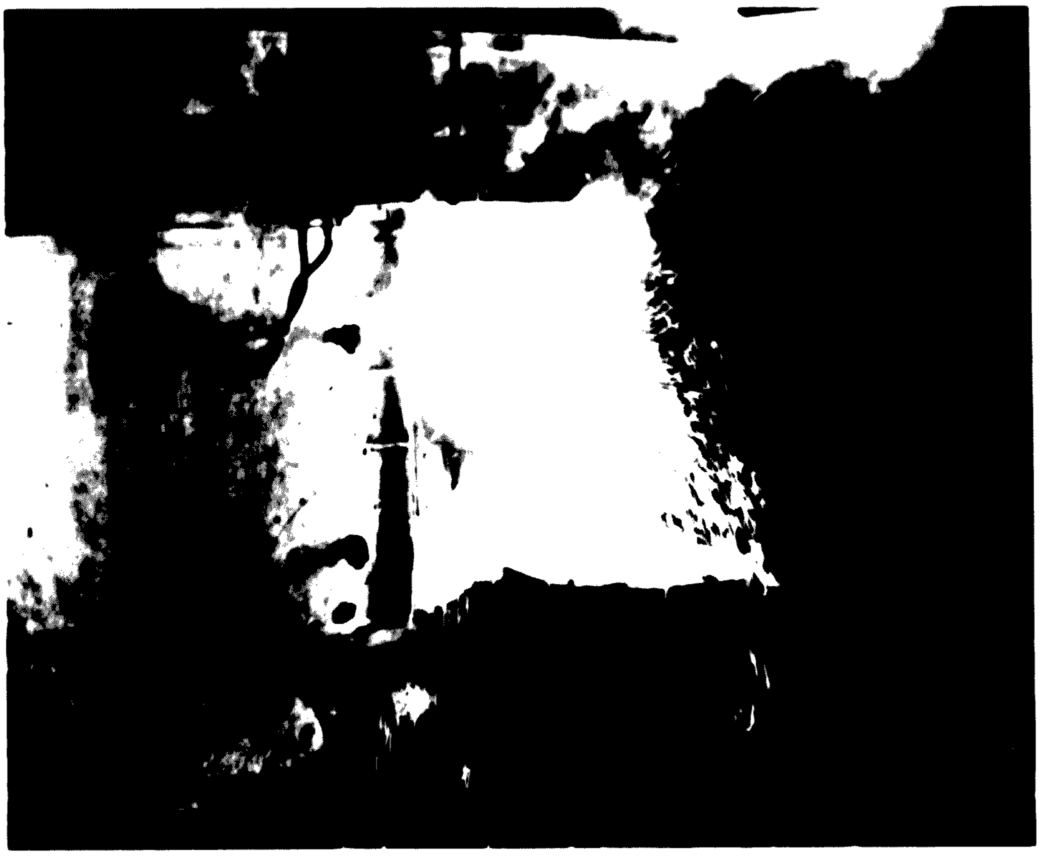


Fig. 6.3 Tapping of pig iron from Asswan iron ore smelted in submerged arc electric furnace

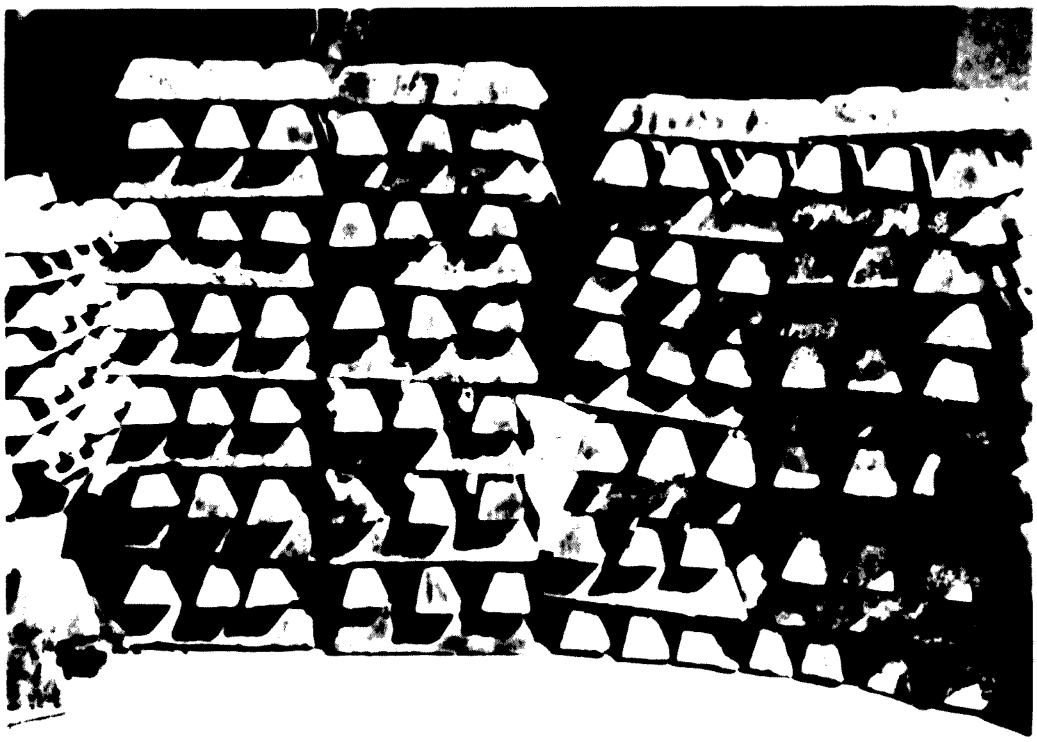


Fig. 6.4 A stack of pig iron ingots produced from pre-reduced, upgraded Asswan iron ore pellets

The electrode penetration and the burden permeability remained good. The burden descent was however comparatively slower.

Results and Conclusions

The results obtained from the electric smelting trials of Asswan iron ores are graphically presented in Figure 6.1. Table 6.9 gives a perspective of the comparative evaluation of various parameters observed during trials with Indian iron ore and with the various types of Asswan iron ore pellets.

Discussion

The submerged arc furnace was brought to stabilised operating conditions by using a good grade of lumpy Indian ore in order to conserve the processed Asswan iron ore pellets. The Indian iron ore required an average of 3109 KWH of power per tonne of metal produced at a basicity of 1.21 yielding a silicon content of 2.8%. The higher power requirements were a direct result of the high ash content of Indian cokes and the additional limestone required to be added to the charge.

Heat hardened pellets were charged in the second phase of furnace operation. It was observed that the specific power consumption was higher at 4298 KWH. The slag/metal ratio was found to be lower, slag viscosity higher, electrode penetration and burden descent were poor. These drawbacks may be ascribed to the varying oxidation levels in the agglomerated ore ranging from magnetite to haematite. The physical characteristics were also detrimental to process efficiency due to pellet disintegration and inadequate pellet size which resulted in low permeability and poor burden descent characteristics.

Phase three of the furnace operation followed with the charging of reduced Asswan iron ore pellets in the order of 80% pre-reduction, 70% pre-reduction and finally 50% pre-reduction. The trend observed has been graphically shown in Figure 6.1. Power consumption showed a noticeable drop with increase of the degree of pre-reduction. The coke rate and slag/metal ratio also dropped with the increasing degree of pre-reduction, while the smelting rate increased perceptably. The overall advantages of higher degrees of pre-reduction would increase quantitatively in larger furnaces as the specific coke losses due to surface oxidation and other disadvantages of a small scale pilot plant operation would be considerably suppressed.

Phosphorus contents were 1 to 1.5% increasing with coke rates for lower degrees of pre-reduction. Silicon was also lower in the case of higher degrees of pre-reduction.

Table 6.9 - Comparative evaluation of parameters (Operation data for the electric smelting of Asswan iron ore)

Sl. No.	Parameter		Indian ore (61% Fe)	Evaluation			
				Asswan Iron Ore Pellets			
				Heat hardened (51% Fe)	80% Pre-reduced (68% Fe)	70% Pre-reduced (67% Fe)	50% Pre-reduced (66% Fe)
1.	**Burden composition kg.**	Ore	100	100	100	100	100
		Coke	36	33	11	13	15
		Limestone	17	19	17	17	17
		Dolomite	5	5	3	3	3
2.	**Power consumption in KWH**		3109	4298	2674	3317	3406
3.	**Coke rate kg/tonne of metal**		714	744	124	235	326
4.	**Flux rate kg/tonne of metal**		337	368	223	292	259
5.	**Slag/metal ratio**		0.5	0.8	0.58	0.65	0.68
6.	**Basicity**		1.21	1.59	1.3	1.08	1.21
7.	**Slag fluidity**		Good	Viscous	Good	Good	Good
8.	**Electrode penetration**		48 cm (good)	23 cm	46 cm (good)	48 cm (good)	46 cm (good)
9.	**Burden permeability**		Good	In proper	Good	Good	Good
10.	**Metal**	Carbon	3.3	1.68	2.89	2.72	2.40
		Silicon	3.11	6.80	3.16	3.56	4.88
		Sulphur	0.03	0.03	0.01	0.04	0.02
		Phosphorus	0.20	1.38	1.12	1.31	1.24
11.	**Productivity (kg. metal per hour)**		122	62	126	106	95

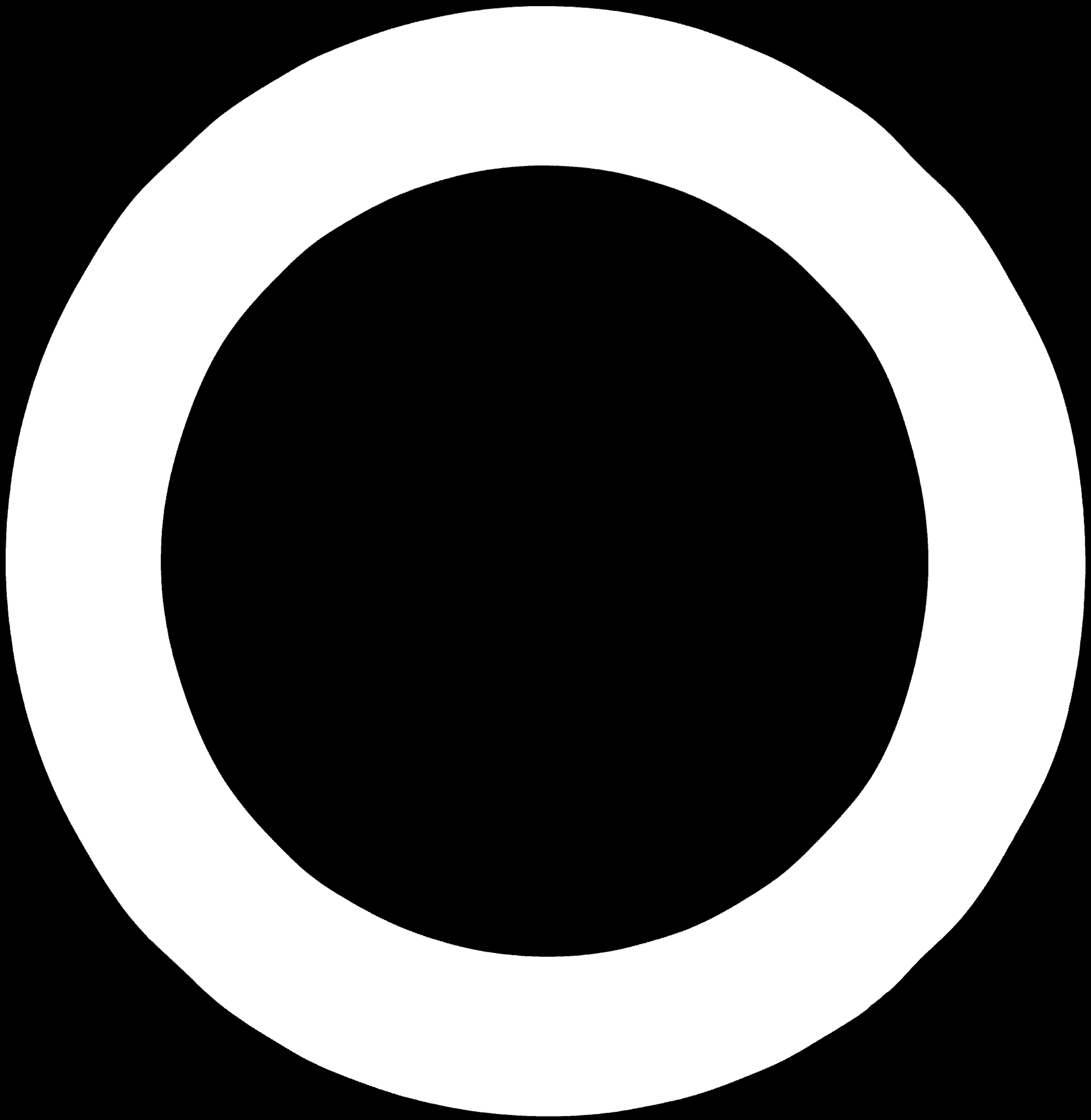
Conclusions

a) It has been clearly established that pre-reduced pellets made from processed Asswan iron ore are quite amenable to electric smelting. The results obtained in these pilot scale trials can be further improved upon when the smelting is done on a bigger scale using pellets of controlled quality and size.

b) A higher degree of pre-reduction is conductive to improve smelting practice leading to higher yields and greater furnace productivity, better quality product with low silicon and sulphur, lower power consumption, lower coke rates, and lower slag/metal ratios.

c) A basicity of 1.2 was found to be the most desirable.

d) The performance of heat hardened pellets was not encouraging due to their poor physical characteristics, higher power requirements, lower furnace productivity and other inherent drawbacks such as higher coke rates, flux rates, slag volume, etc.



CHAPTER 7

TECHNO-ECONOMIC FEASIBILITY

The results detailed in the preceeding chapters on the pilot plant trials on the beneficiation, pelletization, pre-reduction and electric smelting, have been taken as a basis for scaling up in order to study the techno-economic feasibility of utilizing the Asswan Iron ores and Asswan lime-stone for establishing a one-million tonne per year pig iron making facility, especially making use of the cheap electric power available from the Asswan High Dam in the Arab Republic of Egypt. The present analysis is, however based largly upon conditions currently prevailing in India, and is, therefore, subject to extrapolation to conditions in Egypt.

Flow-sheet and Material Balance

Flow-sheets drawn up on the basis of the pilot-plant trials, are shown in Figs. 7.1 to 7.6. The major process operations taken into consideration, and their overall operational basis, are as follows:-

Crushing	16 hrs/day
Mixing	24 hrs/day
Roasting	-do-
Grinding	-do-
Thickening	-do-
Magnetic Separation	-do-
Drying	-do-
Pelletizing	-do-
Coal Crushing/Preparation	16 hours
Pre-reduction	24 hrs/day
Smelting in Electric Arc furnace	-do-

The requirements of the run of mine (ROM) ore have been estimated at 10,000 tonnes/per day on the following basis:

Fe content of ROM ore	41.2%
Handling losses	8% Fe
Tailings losses	5% Fe
Pre-reduction losses	3% Fe
Smelting losses	2% Fe
Fe content of Pig iron	93.5%

The capital investment for plant, equipment and installation for the crushing of ore, mixing, roasting, grinding, and thickening of the ore fines so produced, upgrading by wet magnetic separation, filtration and drying and pelletization processes, is estimated at about 150 million Indian rupees (equivalent to about 20 million US dollars) based on the current prices of the required equipment in India.

The processing cost for the production of green pellets from the beneficiated ore, at the cost level of labour and raw materials required, in India, works out to approximately Rs [illegible] (equivalent to 8 US dollars). In arriving at this cost figure, the costs of raw materials (except that of the run of mine ore), process water, labour and supervision, electric power, depreciation, consumable stores, running and preventive maintenance, insurance and interest on investment have been taken into consideration. In India, the major factor contributing to this cost of beneficiation and pelletization is the cost of I. D. oil, added to the ore fines at the rate of 4% of its weight, for the process of reduction roasting, which has been estimated at Rs 30 (4.1 US dollars) per tonne of dry pellets. Since the cost of fuel oil is very much lower than this price in the Arab Republic of Egypt, cost of beneficiation and pelletisation will be proportionately lower.

The capital investment on plant and equipment, including engineering and installation for the pre-reduction and electric smelting operations has been estimated at 340 million Indian rupees (approximately 47 million US dollars). This includes requirements of coal crushing and handling; plant including ancilliaries, rotary kilns for pre-reduction, coolers, and electric arc furnaces for smelting etc. as well as on such ancillaries as water storage and distribution, compressed air, transport, electrical distribution system, service workshop and control laboratory, and erection.

The operational cost of pre-reduction and electric smelting of the pellets produced from beneficiated iron ore has been estimated at 363 Indian rupees (equivalent to 50 US dollars) per tonne of pig iron under Indian conditions. This includes the costs of power required for the operation of the pre-reduction rotary kilns, and for the electric arc furnace, fuel oil, coal, coke, limestone, dolomite, electrode paste and ramming mixes, refractories, oxygen and lancing tubes, labour and supervision, interest on investment, maintenance etc.

The cost of production of pig iron (from the crushing of the run of mine ore to electric smelting) can, thus be worked out to approximately Rs. 490 (equivalent to approximately 67.5 US dollars) as under:

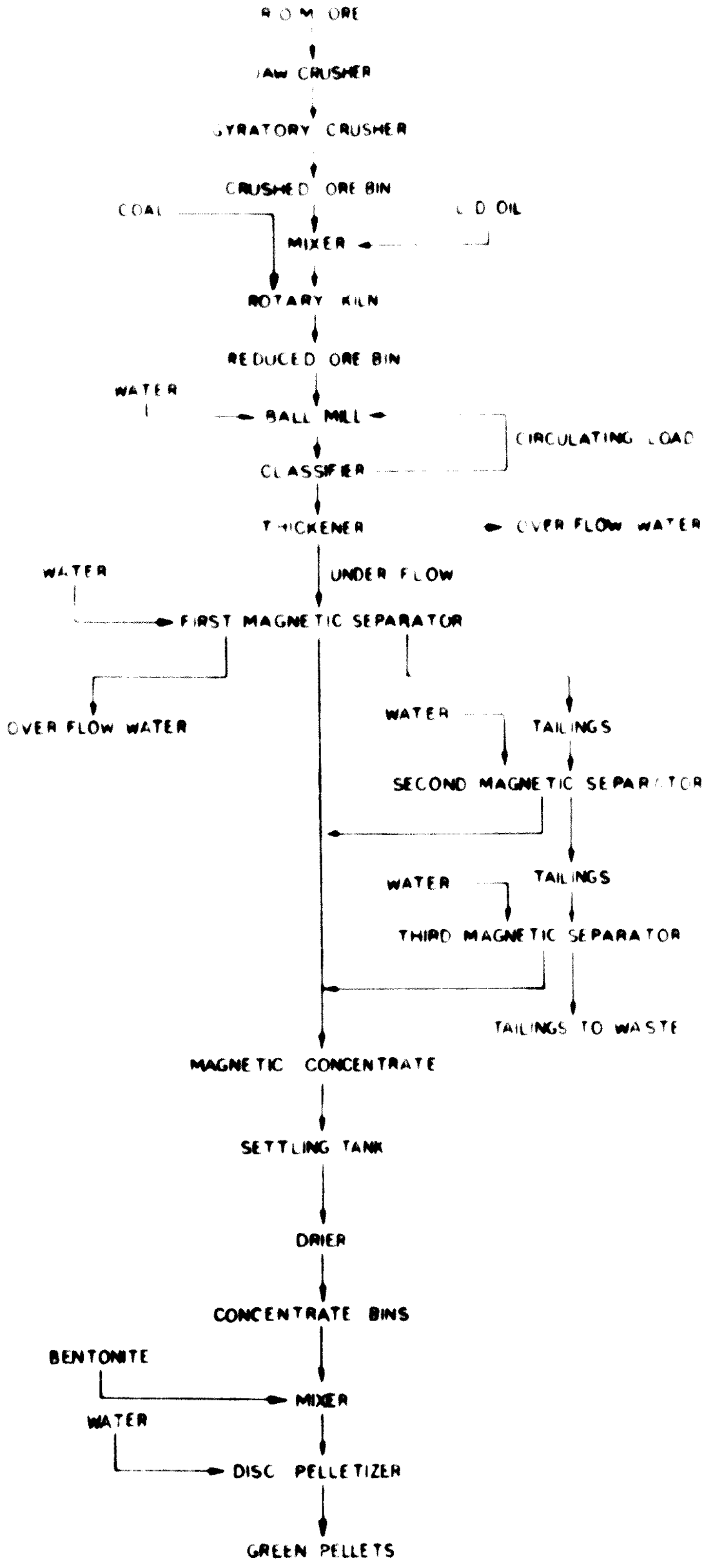
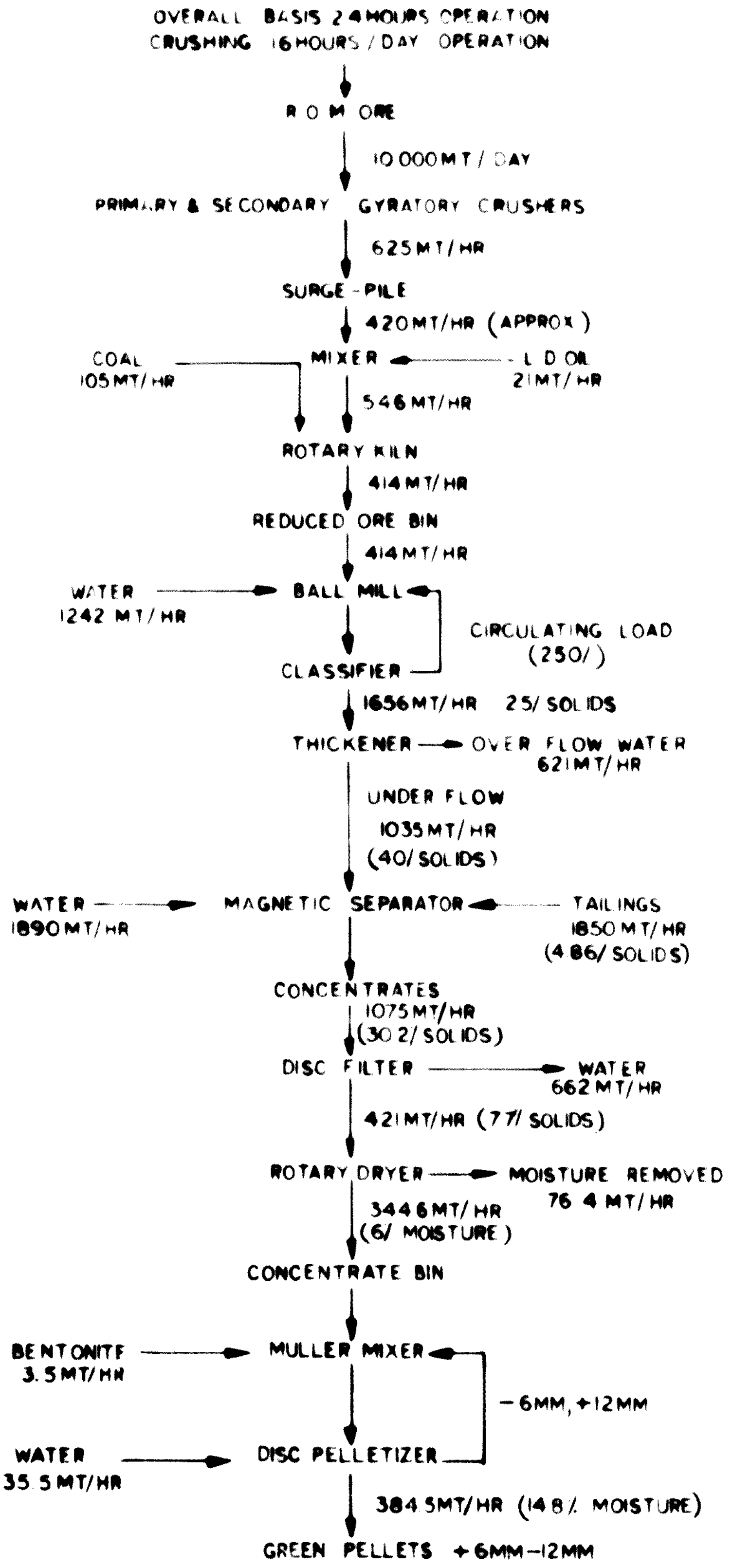


Fig. 7.1 Flow sheet for pilot plant scale beneficiation & pelletization of Aswan [illegible]


OVERALL BASIS 24 HOURS OPERATION
CRUSHING 16 HOURS / DAY OPERATION
R O M ORE
10 000 MT / DAY
PRIMARY & SECONDARY GYRATORY CRUSHERS
625 MT/HR
SURGE-PILE
420 MT/HR (APPROX)
COAL
105 MT/HR
MIXER
L D OIL
21 MT/HR
546 MT/HR
ROTARY KILN
414 MT/HR
REDUCED ORE BIN
414 MT/HR
WATER
1242 MT/HR
BALL MILL
CIRCULATING LOAD
(250%)
CLASSIFIER
1656 MT/HR 25% SOLIDS
THICKENER
OVER FLOW WATER
621 MT/HR
UNDER FLOW
1035 MT/HR
(40% SOLIDS)
WATER
1890 MT/HR
MAGNETIC SEPARATOR
TAILINGS
1850 MT/HR
(4 86% SOLIDS)
CONCENTRATES
1075 MT/HR
(30 2% SOLIDS)
DISC FILTER
WATER
662 MT/HR
421 MT/HR (77% SOLIDS)
ROTARY DRYER
MOISTURE REMOVED
76 4 MT/HR
344 6 MT/HR
(6% MOISTURE)
CONCENTRATE BIN
BENTONITE
3.5 MT/HR
MULLER MIXER
-6MM, +12MM
WATER
35.5 MT/HR
DISC PELLETIZER
384.5 MT/HR (14 8% MOISTURE)
GREEN PELLETS +6MM-12MM

[illegible]

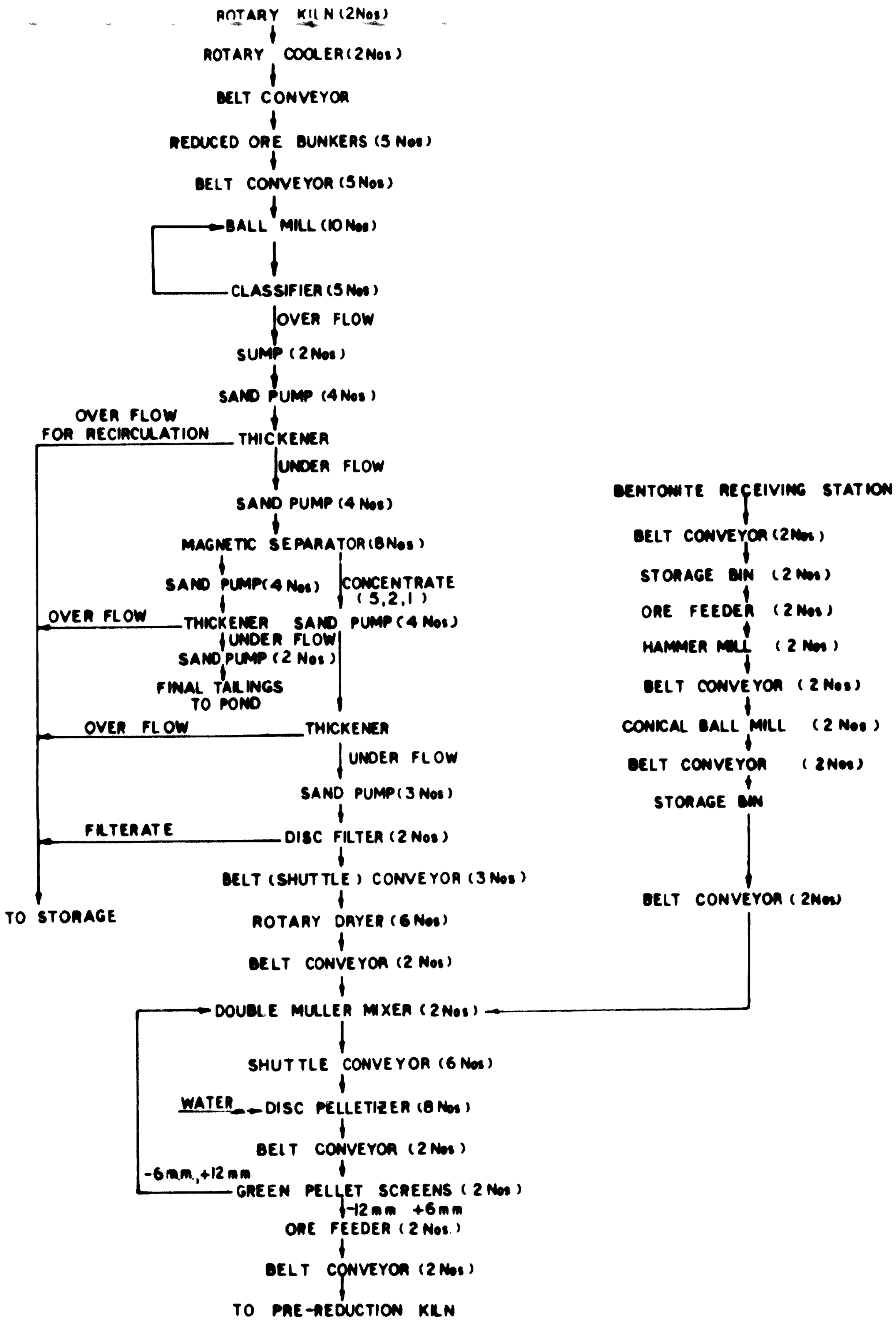
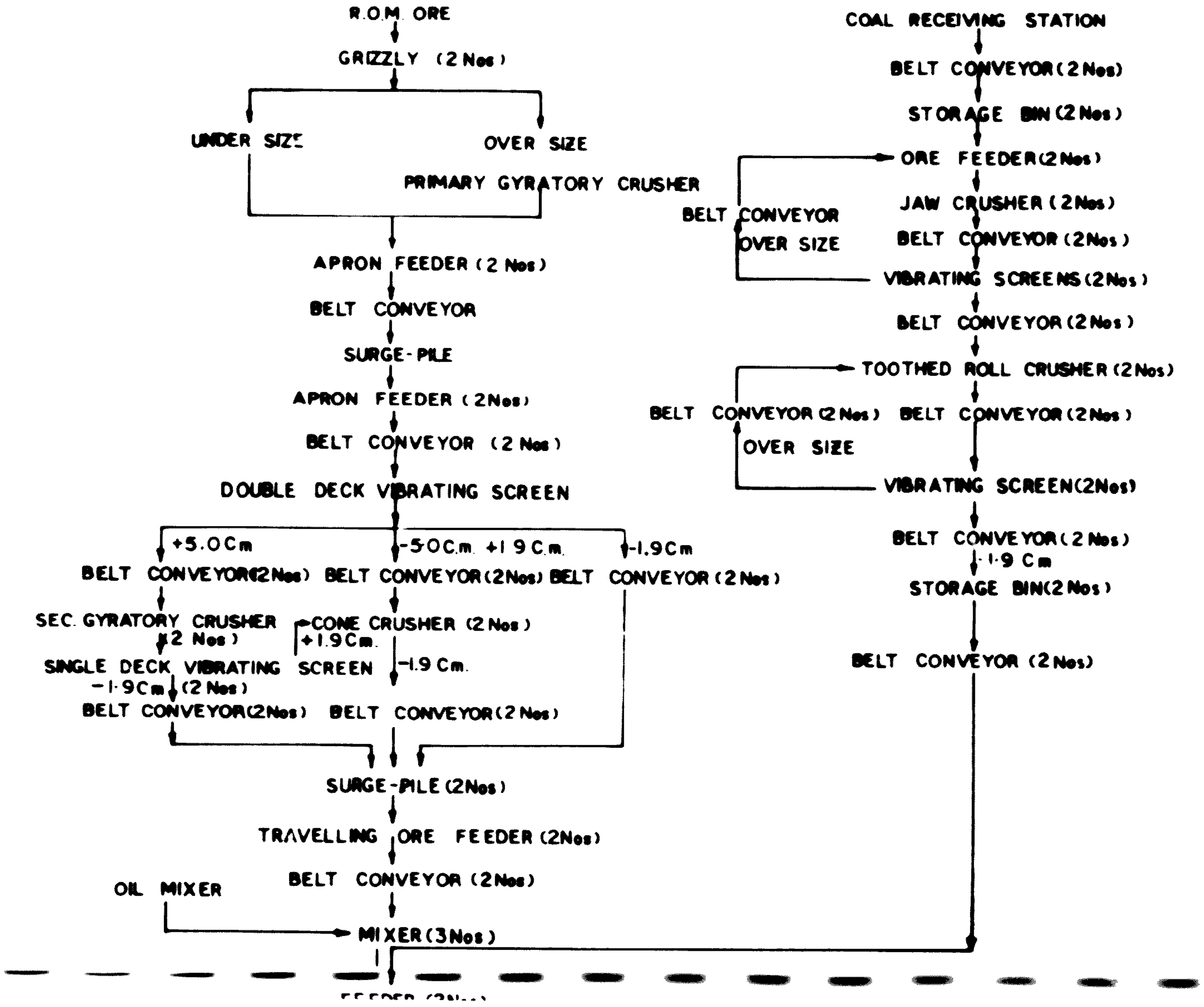
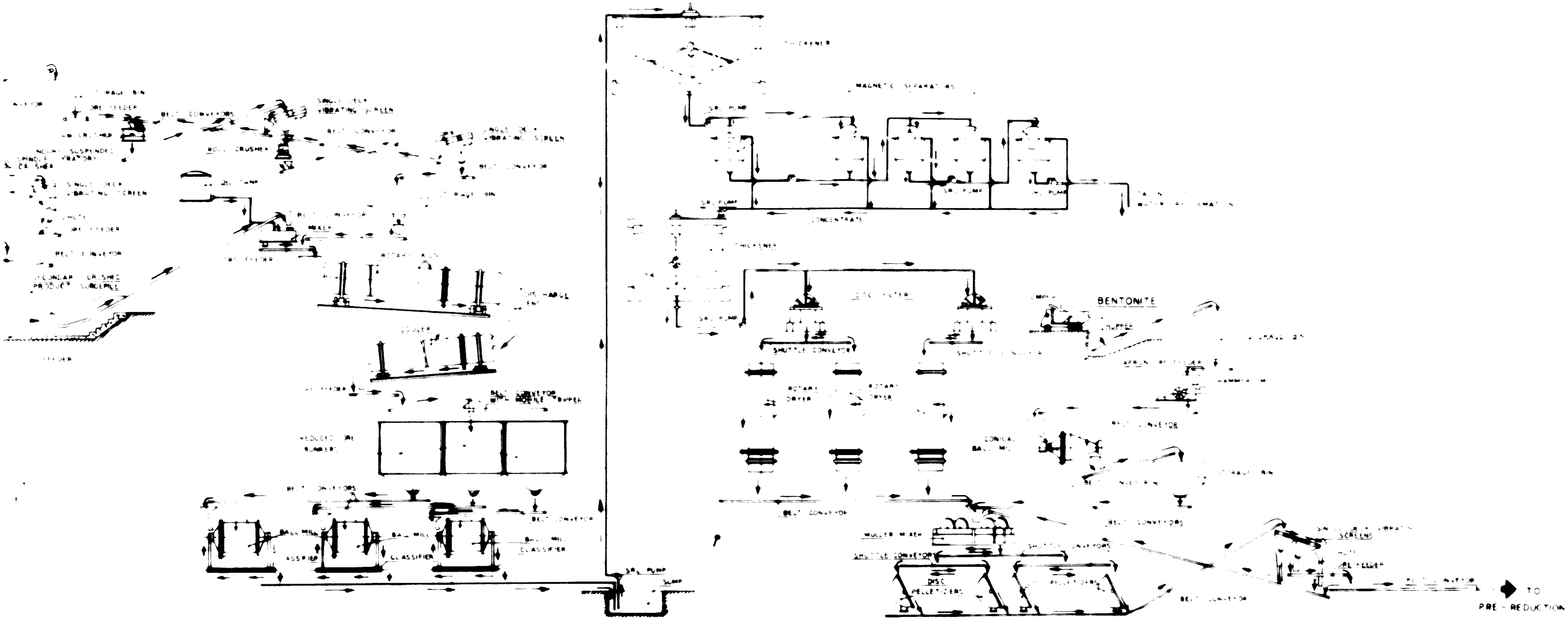


Fig 7.3 Proposed block diagram for beneficiation and pelletization of Asswan iron ore for one million tonnes iron making plant.


R.O.M. ORE
GRIZZLY (2 Nos)
UNDER SIZE
OVER SIZE
PRIMARY GYRATORY CRUSHER
APRON FEEDER (2 Nos)
BELT CONVEYOR
SURGE-PILE
APRON FEEDER (2Nos)
BELT CONVEYOR (2 Nos)
DOUBLE DECK VIBRATING SCREEN
+5.0Cm
-5.0Cm. +1.9Cm.
-1.9Cm
BELT CONVEYOR(2Nos)
BELT CONVEYOR(2Nos)
BELT CONVEYOR (2 Nos)
SEC. GYRATORY CRUSHER (2 Nos)
CONE CRUSHER (2 Nos)
+1.9Cm.
SINGLE DECK VIBRATING SCREEN (2 Nos)
-1.9Cm
-1.9 Cm.
BELT CONVEYOR(2Nos)
BELT CONVEYOR(2 Nos)
SURGE-PILE (2Nos)
TRAVELLING ORE FEEDER (2Nos)
BELT CONVEYOR (2Nos)
OIL MIXER
MIXER(3Nos)
COAL RECEIVING STATION
BELT CONVEYOR(2Nos)
STORAGE BIN (2Nos)
ORE FEEDER(2Nos)
JAW CRUSHER (2Nos)
BELT CONVEYOR (2Nos)
BELT CONVEYOR
OVER SIZE
VIBRATING SCREENS (2Nos)
BELT CONVEYOR (2Nos)
TOOTHED ROLL CRUSHER (2Nos)
BELT CONVEYOR (2Nos)
BELT CONVEYOR (2Nos)
OVER SIZE
VIBRATING SCREEN(2Nos)
BELT CONVEYOR (2 Nos)
-1.9 Cm
STORAGE BIN(2Nos)
BELT CONVEYOR (2Nos)


CRUSHER
SINGLE DECK VIBRATING SCREEN
BELT CONVEYORS
ROLL CRUSHER
BELT CONVEYOR
SINGLE DECK VIBRATING SCREEN
BELT CONVEYOR
SINGLE DECK VIBRATING SCREEN
ORE FEEDER
BELT CONVEYOR
SECONDARY CRUSHED PRODUCT SURGEPILE
MIXER
ROTARY KILN
DISCHARGE END
COOLER
BELT CONVEYOR WITH MOBILE TRIPPER
REDUCED ORE BUNKERS
BELT CONVEYORS
BALL MILL
CLASSIFIER
BALL MILL
CLASSIFIER
BALL MILL
CLASSIFIER
BELT CONVEYOR
SR. PUMP
SUMP
THICKENER
MAGNETIC SEPARATORS
SR. PUMP
SR. PUMP
SR. PUMP
SR. PUMP
CONCENTRATE
THICKENER
SR. PUMP
DISC FILTERS
SHUTTLE CONVEYOR
SHUTTLE CONVEYOR
ROTARY DRYER
ROTARY DRYER
BELT CONVEYOR
BENTONITE
DUMPER
HOPPER
APRON FEEDER
HAMMER MILL
BELT CONVEYOR
CONICAL BALL MILL
STORAGE BIN
BELT CONVEYORS
MULLER MIXER
SHUTTLE CONVEYOR
SHUTTLE CONVEYORS
DISC PELLETIZERS
PELLETIZERS
BELT CONVEYOR
SCREENS
CHUTE
ORE FEEDER
BELT CONVEYOR
TO PRE-REDUCTION

[illegible]

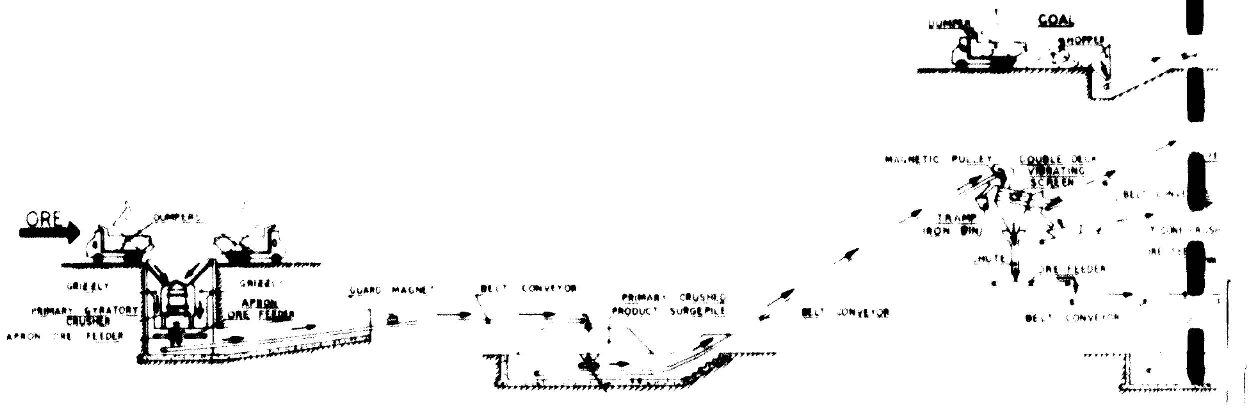

ORE
DUMPERS
GRIZZLY
GRIZZLY
PRIMARY GYRATORY CRUSHER
APRON ORE FEEDER
APRON ORE FEEDER
GUARD MAGNET
BELT CONVEYOR
PRIMARY CRUSHED PRODUCT SURGEPILE
APRON ORE FEEDER
BELT CONVEYOR
DUMPER
COAL
HOPPER
MAGNETIC PULLEY
DOUBLE DECK VIBRATING SCREEN
TRAMP IRON BIN
CHUTE
ORE FEEDER
BELT CONVEYOR

Fig.

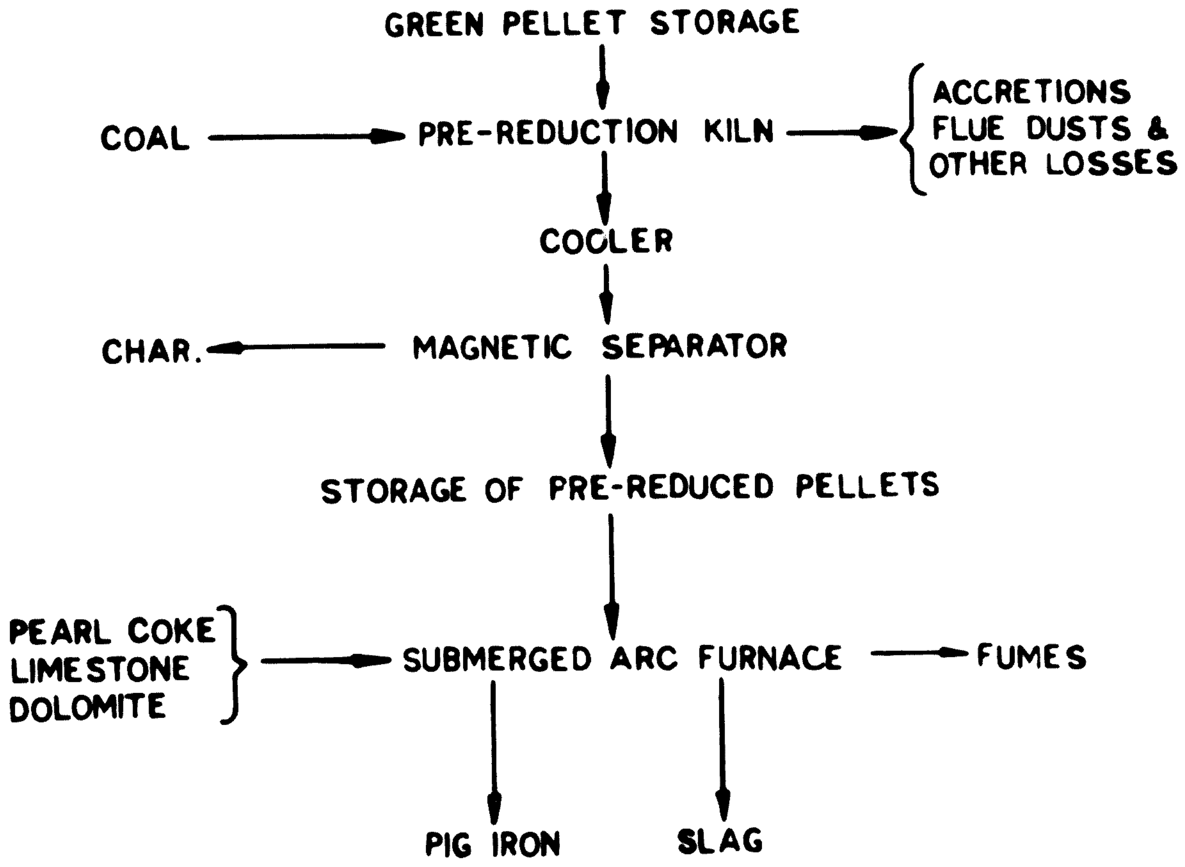


Fig. 7.6 Pre-reduction & electric smelting flow sheet for pilot plant trials for Aswan ore

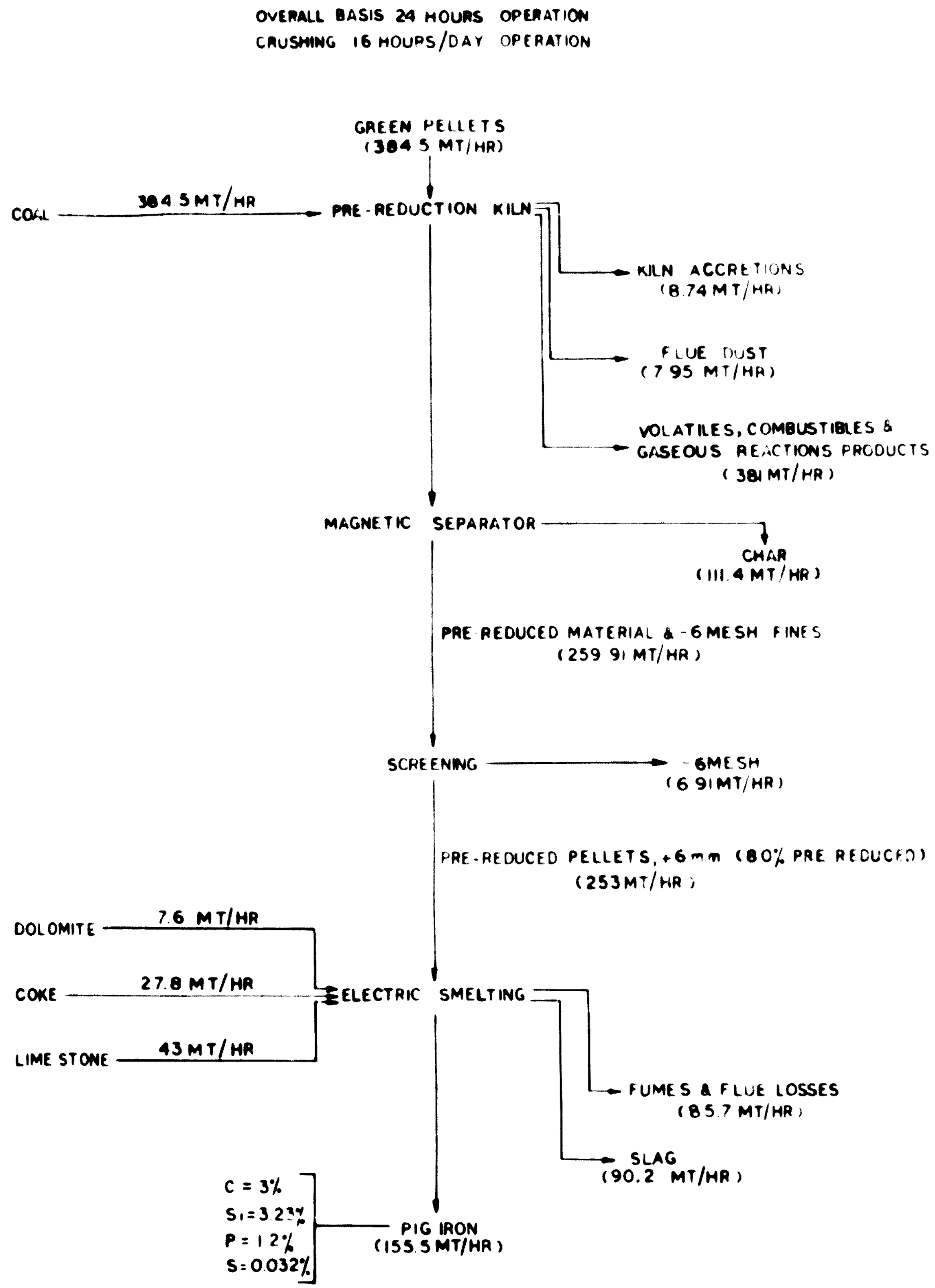
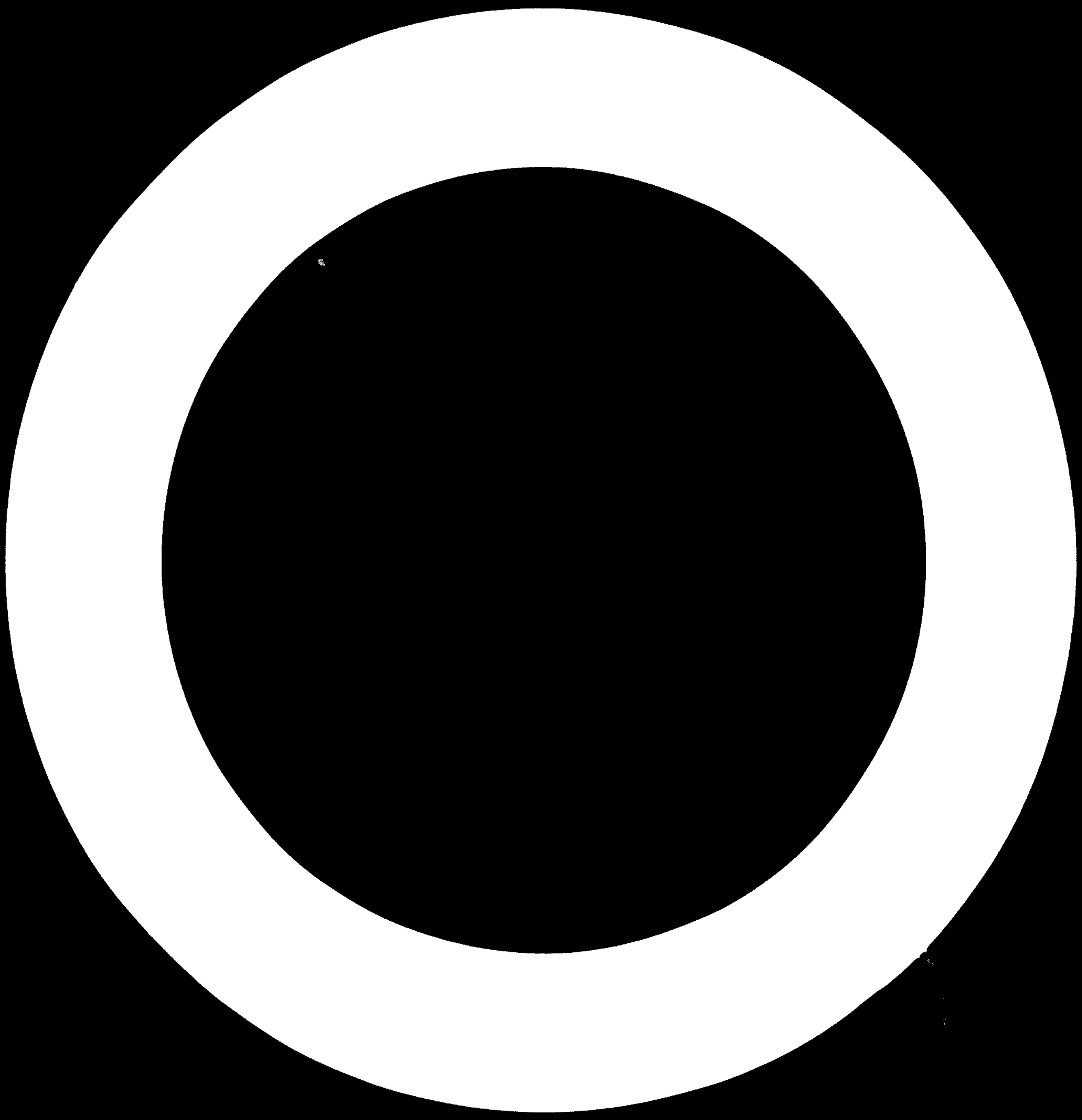


Fig. [illegible] Pre-reduction & electric smelting proposed material flow sheet for [illegible] iron making plant from Aswan iron ore

Cost of production per tonne of dry pellets	: Rs. 60.00	(US $ 8.00)
Converting this to per tonne of pig iron (on the basis of 327.62 tonne of dry pellets producing 155.50 tonnes of pig iron)	: Rs.127	(US $ 17.50)
Cost of pre-reduction and smelting per tonne of pig iron	: Rs.363	(US $ 50.00)
	Rs.490	$67.50

This cost does not include the cost of the run of mine ore delivered to the Steel plant. The actual cost of production of pig iron, may, however, be as low as 350 Indian rupees (or 48 US dollars) per tonne of pig iron produced, depending on the actual cost of power, fuel oil, coal and coke being commensurately lower in the Arab Republic of Egypt. The possible lower requirements of dolomite will also bring down the cost, if the ash content of the coal/coke to be used at Asswan, has a lower ash content than the high ash Indian coals used in the pilot plant trials

It may thus be seen that the capital investment required for the establishment of a one-million tonne per year pig iron plant based on the Asswan Iron Ore supplied for the pilot-plant trials would be approximately 490 million Indian rupees (or 67.5 million US dollars) which is quite comparable with the investment requirements for a conventional Integrated One million tonne per year pig iron plant in India. The higher operational cost (Rs.490 per tonne of pig iron, compared to approximately Rs.375-400/- per tonne in India) can be accounted for by the additional cost involved in beneficiating the very low grade (41.2% Fe) ore, as against the high grade ores (62 - 67% Fe) used in India.



APPENDIX

Terms of Reference of UNIDO Contract

On 2nd July, 1970, the United Nations Industrial Development Organisation (UNIDO) entrusted the National Metallurgical Laboratory, Jamshedpur, India, with the responsibility of undertaking the Laboratory and Pilot Plant studies on the suitability of Aswan Iron Ores in United Arab Republic for iron making. The contract No. 70/26 signed between the two organisations, namely the UNIDO and the NML provides the following terms of reference.

(i) Laboratory scale trials on the beneficiation of the iron ores (lumps and ore fines) for the formulation of the optimum and economic upgrading flow-sheet yielding the highest possible iron recovery consistant with high quality concentrate.

(ii) Extensive pilot plant scale trials to obtain requisite data for industrial scale implementation of the beneficiation flow sheet.

(iii) Assessment of techno-economic aspects and cost factors for the ore beneficiation.

(iv) Trials on the agglomeration of the ore concentrate sufficiently comprehensive to formulate requisite data for industrial scale implementation of the optimum agglomeration cycles based on:

  (a) Production of sinter (self-fluxing and super-basic sinters) of high quality;

  (b) Pelletizing of the ore fine (un-fluxed and self-fluxing pellets to yield high grade pellets);

  (c) Formulation of the optimum burden-mix for sintering and pelletizing test trials; and

  (d) Assessment of the techno-economic aspects of sintering and pelletizing and the operational and production costs.

(v) Trials on pre-reduction of the agglomerated (sinter and/or pellets) burden in rotary reduction kiln using solid, liquid or gaseous fuels or their optimum combination to obtain a highly metallized product.

  (a) Calculation of the raw materials burden to be aligned to yield data for industrial scale implementation of the pre-reduction treatment of the agglomeration burden;

  (b) Assessment of operational and final production costs of pre-reduction of the burden including their relationship with industrial scale operations; and

  (c) Preparation of a full scale flow-sheet showing the techno-economics of the industrial scale operations of pre-reduction treatment of agglomerated burdens.

(vi) Trials on the electric smelting of the pre-reduced product hot-charged continuously into the submerged-arc electric furnace on a sufficiently comprehensive scale of operations to yield to the following data:

  (a) Power consumption for electric smelting of the pig iron based on hot pre-reduced charge.

  (b) Overall economics of the smelting operation and its co-relation with industrial scale operational and final production costs; and

  (c) Relative advantages of the hot pre-reduced charges in relation to cold charge operations.

(vii) The final report is to be submitted in 15 copies in English based on the results of trials outlined above, integrating and dovetailing one operation with the other with a view to highlight the following:

  (a) Material balance for each stage of operation;

  (b) Operational and production costs for each stage and their interpretation to yield the overall costs from the raw materials to the liquid metal. These costs will have to be up-scaled to provide requisite data for industrial scale operation both in relation to capital costs and operational costs and should also include full data about the several trial results as relating to consumption of raw materials, thermal balance, chemical analysis of raw materials used and of the products of each stage, power consumption and their co-relation with the overall economics of integrated operations as well as the final recommendations for the choice and implementation or otherwise of the technological processes as outlined above.

(viii) The final report is to be submitted in 8 months after the receipt of the bulk iron ore samples.

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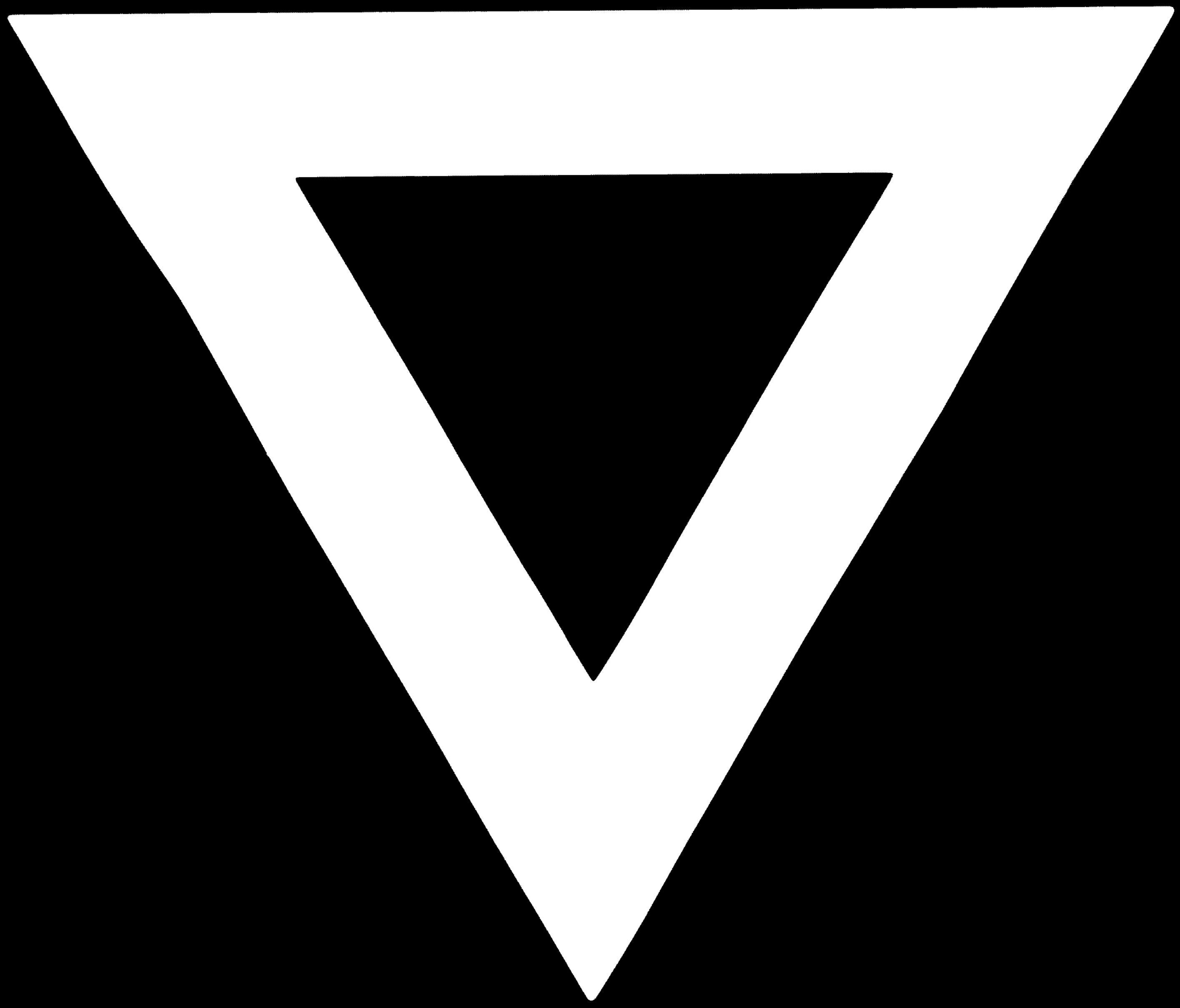

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