INTRODUCTION

ORES and minerals are wasting assets which occur in nature and their use as raw materials in different mineral based industries needs very systematic and scientific approach in rationalising these resources for the development of the country's economy so that they can be utilized for as long a period as possible to the best of their advantage in terms of quality and quantity with minimum or no wastage whatsoever. This rational utilization of the mineral resources can very well be achieved by adopting the latest ore-beneficiation methods. These methods are well developed in recent times and can be very much applicable if the beneficiation problems of the low grade ores and minerals are properly understood. A proper understanding of these problems related to them can be achieved by knowing fully well their basic and most important characteristics like physical properties, chemical analysis and mineralogical nature which would actually reflect the nature of formation of these ores and mineral deposits in a broad manner.

An attempt has been made to group and elaborate these above characteristics of the various low grade ores and minerals which were actually studied for beneficiating them at the N.M.L. on bench scale as well as on pilot plant scale in most cases, on the basis of which flow sheets were designed and developed and plant-equipments recommended for the setting up commercial size ore treatment plants to the various industries in the country.

Chemical analysis and Mineralogy

Chemical analysis of any ore or mineral sample indicates its chemical nature in terms of the various elements and/or radicals present and its grade or tenor. Mineralogy indicates the actual mineral assemblage, their association and nature of interlocking, the amount of the valuables or economic minerals, the gangue minerals and the by-product minerals which all combined together, contribute to the chemical analysis of the ore.

In fact a preliminary assessment of the ore/mineral sample can be made in the first instance on studying its chemical analysis and mineralogy as to the nature of the beneficiation problems, its objective and probable limitations in attaining the stipulated grade coupled with maximum recovery possible, which are the two factors ultimately responsible to decide the economics of the beneficiation process.

As seen from the experience of studying hundreds of low grade ores and mineral samples it can be broadly said that no two ores are similar and each has to be tackled with on its own individual merits and demerits of the problem.

Chemical analysis is done by standard conventional wet methods as well as instrumental methods of analysis depending on the nature of the ore sample.

Mineralogical studies are done by conventional microscopic techniques, which are supplemented by X-ray diffraction and D.T.A. Studies.

D.T.A. Studies

Differential thermal analysis (D.T.A.) studies play a very important role in the identification of clays, bauxite group of minerals, carbonates of calcium and magnesium, iron ore minerals, manganese ore minerals etc. by virtue of their characteristic thermal behaviour of individual minerals when subjected to continuous heating at a controlled rate under normal or controlled atmosphere. Each of the clay and bauxite group of minerals give their characteristic D.T.A. curves.

In the study of carbonate minerals their calcination characteristics such as starting temperature of calcination, peak temperature and culmination of reaction can be very clearly indicated in their D.T.A.

curves. The amount of carbonates present can also be adjudged by the amplitude of the peak. Iron ore minerals and manganese ore minerals have got different characteristic D.T.A. curves of their own.

Instrumentation and test procedure

Deltatherm appartus made by Technical Equipment Corporation, U.S.A., is used. The sample to be tested is made —200 mesh powder. Sensitivity of the apparatus and rate of rise of temperature and the maximum temperature to which the sample is to be heated are pre-set. The test sample is compared against high grade alumina which is used as the inert material in the D.T.A. studies. A smooth curve is resulted on an electrically conducting paper.

The temperature is marked with a line for an interval of every 50°C, till it reaches the maximum adjusted temperature. The heating is accomplished by electric furnace. The temperature is read by chromel olumel thermocouple. The unit can be worked for recording the cooling characterstics also by putting off the electric furnace after reaching any desired temperature. This is particularly useful in observing any reversible reaction or inversion point of any particular mineral phase by cooling, which would actually record exactly in a reversed curve as compared to the heating curve.

Wide variety of ores and minerals are studied to find out their D.T.A. characteristics in solving their identification, calcination behaviour etc., as part of their beneficiation studies.

Physical characteristics of ores and minerals

The physical characteristics of ores and minerals include bulk density, angle of repose, crushing strength, (compressive strength), hardness, shatter size-stability, tumbling and abrasion indices, grindability, screenability, etc., actually dependant on the basic crystalline structure and texture, grain fineness as well as the hardness of the associated minerals and the nature of bonding of the mineral grains inherent in the oresample (chemical bonding or physical cementing), as well as banding, stratification, lamination, macro and micro-jointing, fracture, cleavage planes etc. A fine-grained ore will have more strength than a coarse grained one and a very fine crystalline structure is usually more resistant than a coarsely crystallized ore.

Bulk density, angle of repose and Microporosity

Bulk density is the very important factor needed to be calculated for the design of ore bins and haulage equipments that may be stationary or moving. Bulk density of any ore or mineral depends on its size distribution, and varies inversely with it. The larger the size of the sample the lesser is the bulk density.

It is determined using a cubical box of standard dimensions and is expressed as tonnes/cubic meter.

Angle of repose is another very important factor to be calculated for the purpose of estimating the floor area for stacking purposes of ores and minerals at various stages of processing and with different sizes. It is expressed as in degrees and varies with the size of the material-with the higher the lump size the lower be the angle of repose.

Microporosity

Microporosity indicates the compactness of the ore lumps and is determined by mercury balance method by finding out the apparent specific gravity and the true specific and expressed as percentage.

Some results of bulk density and angle of repose are given in Table 6.1, and of microporosity in Table 6.2.

TABLE 6.1—BULK DENSITY AND ANGLE OF REPOSE RESULTS

Sl. No.	Sample & Locality	Bulk density tonnes/ Cu.m.	Angle of Repose
(1)	(2)	(3)	(4)
1.			
	Kiriburu Iron Ore Washed products		
	$+\frac{3}{8}''$ Lumps	—	35.5°
	$-\frac{3}{8}$ Cl. sand		35°
2.			
	Laminated Iron Ore, Dalli Mines Washed Products		
	$+\frac{1}{2}''$ Lumps	1.54	36°
	$-\frac{1}{2}''$ sand	1.62	33° 15′
3.			
	Iron ore from Rajhara Mines		
	R.O.M. ore	2.28	
	Crushed 2″ ore	2.40	
	$+\frac{3}{8}''$ washed Lumps	1.79	38° at 3.5% moisture
	$-\frac{3}{8}''$ Cl. sand	1.90	37° at 10.7% moisture

TABLE 6.1—BULK DENSITY AND ANGLE OF REPOSE RESULTS *(Contd.)*

Sl. No.	Sample & Locality	Bulk density tonnes/ Cu.m.	Angle of Repose
(1)	(2)	(3)	(4)
4.	Bailadila Iron Ore Deposit 5 screened products		
	—6″ +$\frac{3}{8}$″ Lumps		38.6°
	—4″ +1/4″ Lumps		37.4°
	—$\frac{3}{8}$″ sand		32.3°
	—$\frac{1}{4}$″ sand		24.7°C
5.	Lateritic iron ore from Sesa mines, Goa		
	Washed Lumps		
	—30+18 mm	1.78	36.6°
	—18+ 6 mm	1.62	35.0°
	—6 mm sand	1.87	34.0°
6.	Iron ore (flaky) from Joda mines TISCO Ltd.		
	R.O.M. Ore	1.95	37°12 at 3.0% Moist.
	—50 mm washed lumps	2.04	37° at 5.6% Moist.
	Classifier sand	1.96	36°54′ at 15% Moist.
	—30 mm washed lumps	1.96	37°12′ at 5.1% Moist.
	Classifier sand	1.94	37° at 13.4% Moist.
7.	Iron ore from Bolani Mines		
	R.O.M. Ore	2.23	39° 48′
	Campaign I		
	Washed lumps	1.91	37°12′ at 3.5% Moist.
	Cl. sand	2.04	39°36′ at 19% Moist.
	Campaign II		
	Lumps	1.90	37° at 3.4% Moist.
	Sand	2.04	39°48′ at 19.1% Moist
	Campaign III		
	Lumps −50+25 mm	1.85	35°12′ Moist.
	Lumps −25+10 mm	2.10	38°18′ at 4.5% Moist.
	Cl. sand	2.00	39°12′ at 18.7% Moist
8.	Hard iron ore from Khondband Mines TISCO Ltd.		
	R.O.M. Ore	2.10	42° 48′ at 1.2% Moist.
	Washedlumps —50 mm	2.33	39° 18′ at 2.6% Moist.
	Cl. sand	1.95	38° at 11% Moist.
	Washed lumps —30 mm	2.27	37° at 3.0% Moist.
	Cl. sand	2.05	37° at 11.4% Moist.
9.	Hard iron ore from Naomundi mines TISCO		
	R.O.M. Ore	1.95	36°24′ at 3.0% Moist.
	Washed lumps —50 mm	2.12	35° at 4.47% Moist.
	Cl. sand	1.93	39°24′ at 13.72% Moist
	Lumps —30 mm	1.96	36° at 3.97% Moist.
	Cl. sand	1.90	39°48′ at 14.0% Moist
10.	Iron ore (hard) from Joda mines TISCO Ltd.		
	R.O.M. Ore	2.44	39°48′ at 0.72% Moist.
	Washed lumps —50 mm	2.55	38° at 1.8% Moist.
	Cl. Sand	2.55	38° at 9.2% Moist.
	Washed lumps —30 mm	2.40	39°12′ at 2.1% Moist.
	Cl. Sand	2.24	37°30′ at 9% Moist.
11.	Limestone from Tiruchy mines Tamil Nadu	1.334	—
12.	Copper slag from I.C.C. Ghatsila —12.5 mm feed to ball-mill	1.8	32°
13.	Low grade iron ores from Codli mines, Goa		
	Lumpy ore R.O.M.	2.07	
	Fines	2.21	
	Low grade washed lumps		
	—30+10 mm	1.65	35.5° at 6.7% Moist.
	—10 mm sand	—	41.1° at 12.9% Moist.
	Low grade fines		
	—30+10 mm	1.64	33.5° at 8.9% Moist.
	—10 mm sand	—	42.1° at 12.9% Moist.
14.	Lime stone samples from TRF (1) Yerrakuntla A.P.		
	At 1″ size	1.289	—
	3/8″ size	1.381	—
	(2) Neemuch		
	At 1″ size	1.247	—
	$\frac{3}{8}$″ size	1.356	—
	(3) Tamil Nadu		
	At $\frac{3}{8}$″ size	1.210	—
	(4) Akaltara cement plant		
	At 1″ size	1.088	—
	At $\frac{3}{8}$″ size	1.163	—
	(5) From Churk U.P.		
	At 1″ size	1.200	—
	At $\frac{3}{8}$″ size	1.305	—

TABLE 6.2—MICROPOROSITY

Sample :—Mixed iron ore from Kiriburu mines N.M.D.C.

Sample/Type	App. Sp. gr.	True. Sp. gr.	Microporo-sity. %
1. Massive	4.478	5.040	11.2
2. Massive porous with blue dust	3.919	4.465	12.2
3. Laminated with blue dust	3.553	4.440	20.0
4. Goethite type	2.781	4.232	34.3
5. Goethite with ochre	1.897	4.050	53.2
6. Red & Yellow ochery type	2.084	4.526	60.0

Sample :—Mixed iron ore from Rajhara mines

1	2	3	4
1. Compact laminated	3.39-3.65 Av. 3.51	3.73-4.25 Av. 4.02	7.5-19.3 Av. 12.69
2. Porous ,,	2.86-3.32 Av. 3.14	3.42-4.17 Av. 3.283	14.03-31.78 Av. 25.0
3. Friable ,,	3.26-3.51 Av. 3.42	3.68-4.51 Av. 4.11	5.45-25.2 Av. 16.74
4. Goethite	3.20-3.25 Av. 3.22	3.75-3.80 Av. 3.77	13.35-15.80 Av. 14.75
5. Laterite	1.98-2.22 Av. 2.1	2.54-3.12 Av. 2.83	22.40-28.80 Av. 25.6

Crushability studies

For collecting the basic data necessary for the design of the crushing section, crushing strength of the ores is to be determined. Since any ore has to undergo compression during crushing, the resistance offered by the ore to compression is to be measured by standard method used for rocks, ores and minerals.

Compressive test method

For determining compressive strength, big specimens from all the important varieties of the ore sample are selected. Cubes are prepared from these specimens as far as possible by using a diamond section cutter or any other device, followed by grinding the surfaces on fine carborundum powder to get very smooth surface. If due to practical difficulties, perfect cubes cannot be prepared conveniently, specimens very near to cubes (all sides being parallel) are prepared. This is expected not to effect the test results adversely, since the crushing load at which the specimens yield is not a function of the height so long it does not exceed the lateral dimensions.

Crushing strength of the test specimens is determined by using an Avery 25 Tonne self acting Universal Testing Machine. The load in tonnes at which the test specimen is yielded will be noted in each case and crushing strength given by calculation in Kg. per Sq. Cm. or Tonnes or Pounds per Sq. Inch.

Generally ore and mineral samples upto 30,000 PSI are considered to be hard. The rock compressive strength according to ASTM C170 specification are given as follows and can be applied for ores and minerals.

Rock compressive strength

5,000 to 10,000 PSI	————	Soft.
10,000 to 20,000 PSI	————	Medium.
20,000 to 30,000 PSI	————	Hard.
30,000 to 45,000 PSI	————	Very hard.
Over 45,000 PSI	————	Extremely hard.

Hardness of rocks and minerals

The hardness of rock, ore or mineral is measured by the resistance which a smooth surface offers to abrasion and is given in terms of a number of the standard Moh's scale. Hardness and related toughness provide an index for application of impact and grinding in mills. Both hardness and toughness indices also will be considered in pressure crushers in combination with compressive strength.

Moh's scale of hardness

1. Talc.
2. Gypsum.
3. Calcite.
4. Flourite.
5. Apatite.
6. Orthoclase.
7. Quartz.
8. Topaz.
9. Corundum.
10. Diamond.

TABLE 6.3—COMPRESSIVE OR CRUSHING STRENGTH TEST RESULTS

Sll. No.	Sample & Inv. No.	Compression strength (Tonnes per Sq. Cm.)	Average Com. Stg.
(1)	(2)	(3)	(4)
1.	Bailadila Iron ore. Deposit No. 5. (Inv. No. 486/68)	1.83-2.68	
2.	**Surajgharh iron ore samples:**		
	(a) Massive iron ore	0.572	
	(b) -do-	0.530	0.516
	(c) -do-	0.446	
	(a) Float iron ore	0.693	
	(b) -do-	0.906	
	(c) -do-	0.628 ⊥ to lami'n.	0.693
	(d) -do-	0.546// to -do-	
	(a) Laminated iron ore	0.630 ⊥ -do-	0.511
	(b) -do-	0.392 // -do-	

Remarks:—The power consumption for crushing the float ore sample is expected to be higher to the massive and laminated variety.

3.	**Lateritic iron ore from Sesa Goa**		
	(a) Massive	0.946	
	(b) Laminated	0.343	
	(c) Porous	0.329	
	(d) Lateritic	0.163	
4.	**Kiriburu iron ore**		
	(a) Massive	1.401	
	(b) Laminated ⊥ to plane	0.825	
	(c) Laminated// to plane	0.567	
5.	**Barajamda iron ore. (Inv. No. 640/71)**		
	(a) Massive	0.753	
	(b) Laminated ⊥ to plane	0.708	
	(c) Laminated// to plane	0.400	

TABLE 6.3—COMPRESSIVE OR CRUSHING STRENGTH TEST RESULTS *(Contd.)*

Sl. No.	Sample & Inv. No.	Compression strength (Tonnes per Sq. Cm.	Average Com. Stg.
(1)	(2)	(3)	(4)
6.	**Joda falky ore.**		
	(a) Massive & compact	1.250	
	(b) Laminated	3.375	
	(c) Friable (soft)	0.165	
7.	**Khondband iron ore.**		
	(a) Massive & compact	2.88-4.40	
	(b) Laminated	2.15-2.35	
	(c) Friable (soft)	1.285-1.930	
8.	**Noamundi hard iron ore.**		
	(a) Massive & compact	2.15-3.50	
	(b) Laminated	1.90-3.10	
	(c) Friable (soft)	0.95-1.40	
9.	**Joda hard iron ore.**		
	(a) Massive & compact	2.985-4.500	
	(b) Laminated	2.275-2.415	
	(c) Friable (soft)	1.083-1.790	
10.	**Dolomite from Tamil Nadu for Salem Steel Project.**		
	(a) ———	0.4228	
	(b) ———	0.395	
	(c) ———	0.563	
	(d) ———	0.728	
	(e) ———	0.658	
11.	**Limestone from Tiruchirapally mines.**		
	(a) ———	1.063	
	(b) ———	0.378	
	(c) ———	0.581	
	(d) ———	0.623	
	(e) ———	1.074	
	(f) ———	0.325	
	(g) ———	0.318	
12.	**Limestone samples**		
	(a) Yerrakuntala	2.66	
	(b) Yerrakuntala	1.55	
	(c) Neemach	5.68	
	(d) -do-	10.20	
	(e) Tamil Nadu lime stone	1.14	
	(f) Akaltara cement plant	1.46	
	(g) -do-	1.55	
	(h) Churk (U.P.)	4.85	
	(i) -do-	3.42	
	The limestone samples above can be grouped as medium to hard in terms of their compressive strength.		
13.	**Copper slag samples from I.C.C. Ghatsila.**		
	(a) Massive, heavy with numerous vesicles	1.187	
	(b) Massive, compact	0.517	
	(c) Massive, compact, layered (perpendicular to layers)	1.439	
	(d) Massive, compact with few vesicles	1.936	
	(e) Lighter, pores, fissures	0.306	
	(f) Massive, compact, layered (perpendicular to layers)	1.293	

Shatter size stability

Ores and minerals undergo degradation due to drops during handling and transit at different stages and likely to generate fines. The extent to which these fines thus produced is a factor directly related to the hardness and toughness of the samples and can be determined by drop shatter tests simulating handling and transit conditions.

Test Method

The test method involves in dropping a known weight of the sample from a fixed height on a mild steel plate for a fixed number of times in each lot and increasing the number of drops in the next lot, keeping the weight of the sample constant. Alternatively the height may be increased with different lots of the sample keeping the number of drops constant. Sieve analysis of the sample are determined before and after shatter. The shatter size stability is determined by the following formula, used in determining shatter size stability of coal in ASTM standard.

$$\text{Shatter size stability} = \frac{\text{Total of Av. Screen opening X Wt.\% X 100 after shatter}}{\text{Total of Av. Screen opening X Wt\% X 100 before shatter.}}$$

Shatter size stability test results:

(1) Laminated iron ore from Dalli mines. (338/65) Washed + 1 2.7 mm lumps—from 6′ height on a M.S. plate after 6 drops. . . . . .Av. = 6.9%

(2) Laminated iron ore from Rajhra mines. (366/66) Washed + 9.5 mm lumps—from 6′ height on a M.S. plate after 6 drops. . . . . . . .Av. = 5.09%

(3) Bailadila iron ore Deposit No. 5 (468/68)
Drop height . . . . .2 Mt, 6 Mt. 10 Mt.

 (a) Washed —150 + 9.5 mm lumps—
 % of —9.5 mm fines. . . .1.80 — 3.10—6.50

 (b) Washed —100 + 6.3 mm lumps—
 % of —6.3 mm fines. . . .1.20 — 3.40—4.30

TABLE 6.4—SCREEN ANALYSIS OF DROPPED —150 + 9.5 mm ORE.

Size (mm)	2 M Drop Wt. %	6 M Drop Wt. %	10 M Drop Wt. %
—150 + 125	5.3	—	—
—125 + 100	12.8	7.7	7.3
—100 + 75	29.4	25.1	17.0
—75 + 50	28.1	37.2	36.6
—50 + 25	13.1	15.4	17.0
—25 + 12.5	6.2	7.0	10.5
—12.5 + 9.5	3.3	4.5	5.1
— 9.5	1.8	3.1	6.5

TABLE 6.5—SCREEN ANALYSIS OF DROPPED —100 + 6.3 mm ORE

Size	2 M Drop	6 M Drop	10 M Drop
(mm)	Wt. %	Wt. %	Wt. %
—100 + 75	37.5	26.6	20.2
—75 + 50	26.9	30.6	29.4
—50 + 25	16.4	18.8	22.9
—25 + 12.5	11.2	12.7	13.5
—12.5 + 6.3	6.8	7.9	9.7
—6.3	1.2	3.4	4.3

The sample above is hard and compact and the amount of fines likely to be genarated during handling and transit would be well within limits.

TABLE 6.6—KIRIBURU AND BARAJAMDA IRON ORES FOR BOKARO.

Screen Size (mm)	Kiriburu iron ore Wt%			Barajamda iron ore Wt. %		
	Before Shater	After Shater	After Shater	Before Shater	After Shater	After Shater
(1)	(2)	(3)	(4)	(5)	(6)	(7)
+38	5.4	3.0	1.8	5.6	2.8	2.0
—38 + 25	34.0	30.0	30.2	43.3	43.2	41.8
—25 + 19	21.4	26.3	26.0	27.2	24.9	24.5
—19 + 12.5	13.7	15.3	15.0	11.5	11.0	13.7
—12.5 + 9.5	15.5	13.9	14.2	1.0	8.1	8.1
—9.5 + 7.3	1.9	2.0	2.5	2.1	2.2	2.4
—6.3 + 3.2	1.1	1.4	1.5	2.8	2.9	3.0
—3.2	7.0	8.1	8.8	0.5	2.9	4.5
Total	100.0	100.0	100.0	100.0	100.0	100.0

TABLE 6.7

(1)	(2)	(3)	(4)	(5)	(6)	(7)
Shater size stability		94.9	92.7		94.4	91.6
% of additional —9.5 mm fines produced		1.5	2.8		2.6	4.5
% of additional —6.3 mm fines produced		1.4	2.2		2.5	4.2

From the results given in the above table, it is seen that Kiriburu iron ore has slightly better size stability than the Barajamda ore.

TABLE 6.8—SHATTER TEST RESULTS OF IRON ORES:

Size (mm)	Wt% of —10 mm fines produced after shatter. (After 6 drops on a mild steel plate from 6′)			
	Joda flaky 707/72	Khnondband 721/73	Noamundi 739/73	Joda hard 748/73
—50 + 10 mm washed lumps	10.45	7.80	9.811	5.04
—30 + 10 mm washed lumps	10.89	8.54	11.576	5.26

TABLE 6.9—BOLANI IRON ORE SAMPLE NO. 1 SHATTER RESULTS:

Washed + 10 mm Lumps—Six drops on M.S.Plate from 6 ft height

	% —10 mm fines before shatter	% —10 mm fines after shatter
Campaign I	10.8	20.3
Campaign II	11.5	19.2
Campaign III	8.4	17.2

The above results indicate that the nature of the ore is rather soft and results in more fines.

Limestone samples from T.R.F.

Tests done in "as received" size dropped three times on a M.S. plate from a height of 6 feet (1.83 Metres). Samples D & E are hand-cobbed before tests.

TABLE 6.10—TEST RESULTS (DROP TEST) ON LIMESTONES FROM T.R.F.

Size in mm	A WT % B/S	A WT % A/S	B WT % B/S	B WT % A/S	C WT % B/S	C WT % A/S	D Wt % B/S	D Wt % A/S	E Wt % B/S	E Wt % A/S
+125	100.0	99.6	100.0	98.4	—	—	—	—	100.0	99.1
+100	—	—	—	—	—	—	—	—	—	
+75	—	—	—	—	—	—	41.9	40.4		
+50	—	—	—	—	62.5	53.6	35.0	30.1		
+37	—	0.4	—	1.6	37.5	34.0	14.7	14.0		
+25	—	—	—	—	—	3.2	8.4	9.4		
+19	—	—	—	—	—	2.3	—	1.3		0.9
+15	—	—	—	—	—	1.6	—	0.4		
+12.5	—	—	—	—	—	1.3	—	0.6		
+12.5	—	—	—	—	—	4.0	—	3.8		
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Note: A—Yerrakuntla; B—Neemuch; C—Tamilnadu; D—Akaltara cement plant; E—Churk-U.P.; A/S—After Shatter; B/S—Before Shatter. According to the above results the amount of —1/2″ fines produced are too low (less than 4%).

Tumbling and Abrasion Indices

Tumbling and abrasion test data are needed in the consideration of impact or grinding type crushers and relate to the maintanence cost data. A higher abrasion index indicated more crusher wear.

Test method

The usual ASTM method is employed to determine tumbling and abrasion indices for coking coal, iron ores etc. The tumbling drum is of 91.4 cm. diameter and 45.7 cm height and rotates at 25 r.p.m. It is fitted with two 50.5 mm lifters inside. Known quantity of material is charged and tumbled for 200 revolutions. The tumbled material is screened on 6.3 mm and 28 mesh (0.595 mm) screens. The percent weight of +6.3 mm material is taken as tumbler index and the percent weight of —28 mesh material as abrasion index.

In the case of soft ores and minerals low tumbler index and correspondingly high abrasion index would result whereas the reverse would be the case with hard and tough ores.

TABLE 6.11—TUMBLER AND ABRASION INDEX RESULTS:

(1) Kiriburu and Barajamda iron ores

Test sample : —40+10 mm ore

Products after Test	Kiriburu Ores Wt. %	Barajamda Ore Wt. %
+ 6.3 mm	75.4	75.0
— 6.3 mm + 28 mesh	9.7	11.6
— 28 mesh	14.7	13.4
	100.0	100.0
Tumbling Index (+6.3 mm)	75.4	75.0
Abrasion Index (—28 mesh)	14.7	13.4

(2) Iron ore from Bolani Mines :

Sample Lumps	Tumbling Index	Abrasion Index
Campaign I	66.8	16.6
Campaign II	70.1	15.2
Campaign III	70.0	15.0

(3) **Iron ores from TISCO**

Products	Joda Flaky		Khondaband		Noamundi		Joda hard	
	T.I.	A.I.	T.I.	A.I.	T.I.	A.I.	T.I.	A.I.
—50+10 mm washed lumps	63.0	12.6	83.5	6.75	75.44	11.47	85.55	6.25
—30+10 mm washed lumps	59.6	17.6	83.45	7.15	73.67	13.29	85.45	6.45

T.I.—Tumbler Index +6.3 mm%
A.I.—Abrasion Index —28 mesh %

The low tumbling index and corresponding high abrasion index indicated the sample is of soft nature.

Screenability and Moisture content

Considerable difficulties are experienced during monsoon seasons for handling particularly Indian iron ores in cases where they contain a fair amount of fines. Presence of little moisture in the ore has a tendency to blind the screen, chute, bunker etc. thereby causing tremendous difficulties in handling the ore. It is therefore, necessary to determine the effects of different moisture content on the screenability or effective screening. The test procedure includes of preparing a few batches of representative —50 mm and —30 mm crushed samples of equal weights and increasing water contents from 0 to 15% by mixing up thoroughly with the samples. The material is screened over a 10 mm screen. The oversize obtained in each case is dried and then weighed to determine the percentage of oversize as well as the undersize. The weight percent of —10 mm material is taken as a measure of screenability. The critical moisture content can thus be found out for the sample which gives the best screenability results.

Screenability and Moisture Results

TABLE 6.12—(1) KIRIBURU IRON ORE

Feed sample from screenability tests —50mm ore Weight taken=25 Kg.

% Water in 50 mm ore	% Weight of —9.5 mm fraction during screening
0	30.3
5	38.0
7.5	27.5
10.0	28.5
12.5	30.0
15.0	30.2

The results indicate that screenability decreased with increase in the moisture contents, reach a minimum when it is at 7.5% and then increased. The screenability tests results are indicative of the general screening trends rather than qualitatively defining the screening characteristics of the ore to any degree of precision.

(2) Laminated iron ore from Dalli Mines

TABLE 6.13—FEEDSAMPLE—50 mm ORE FOR THE TESTS

% of water added to —50 mm ore	% Wt. of —12.5 mm fraction during screening
0	39.7
2.0	38.3
5.0	36.1
7.5	38.0
10.0	39.7

As per the results minimum screenability is with 5% moisture contents.

(3) Laminated iron ore from Rajhara Mines, M.P.

TABLE 6.14—ORE FEED SIZE FOR SCREENING TESTS = —50 mm

% water in the —50 mm ore	% Wt. of —9.5 mm fraction after screening
0	26.85
2.5	26.49
5.0	25.75
7.5	24.18
10.0	21.95
12.5	24.67
15.0	26.78

The screenability reached minimum with 10% moisture.

TABLE 6.15—(4) LATERITIC IRON ORE WITH CLAYEY MATRIX FROM GOA SCREENING SIZE = R.O.M ORE

Added % water in the —30 mm ore	% Wt. of +30 mm	Wt. % of —30 mm
0	28.00	72.00
5.0	28.20	71.80
7.5	28.28	71.72
10.0	30.20	69.80
12.0	30.04	69.96
15.0	28.84	71.16
20.0	28.40	71.60

TABLE 6.16—(5) IRON ORE FROM BOLANI MINES SCREENING ORE SIZE —50 mm

% Water added	Wt. % of —10 mm
0	66.6
5.0	65.1
7.5	64.4
10.0	55.4
12.5	61.5
15.0	65.7

The results show that at 10% moisture content stage 11.2% of —10 mm fines are adhered with the + 10 mm product. As such, except in summer, the screening would have to be done wet with about 20% moisture with the sample.

TABLE 6.17—TISCO IRON ORES

Feed samples —50 mm
—30 mm

%	Joda Flaky		Khondaband		Noamundi		Joda Hard	
	—50 mm —10 mm Wt %	—30 mm —10 mm Wt %	—50 mm —10 mm Wt %	—30 mm —10 mm Wt %	—50 mm —10 mm Wt %	—30 mm —10 mm Wt %	—50 mm —10 mm Wt %	—30 mm —10 mm Wt %
0	53.2	55.2	19.00	27.40	37.08	46.1	21.1	27.9
2	—	—	18.70	26.70	—	—	—	—
3	—	—	16.65	25.35	—	—	20.1	25.7
4	—	—	17.25	25.00	—	—	—	—
5	50.0	49.5	20.00	27.15	29.8	43.5	21.7	27.0
6	—	—	21.15	28.30	—	—	—	—
7.5	7.7	6.0	—	—	33.0	41.8	22.0	27.3
10.0	14.8	14.4	—	—	34.66	42.4	22.2	27.5
12.5	30.9	29.1	—	—	34.7	44.4	—	—
15.0	53.0	52.0	—	—	37.5	46.0	—	—

Grindability and Bond's work Index

Grindability and work index are the two parameters needed for the design and selection of the crushing and grinding equipment and estimation of the power consumption.

Grindability indicates degree of amenability of the sample for grinding and according to the grinding characteristics, the ores and minerals are classified as soft, medium-soft, medium, medium-hard, hard and very hard. Once the nature of the ore and size of the test sample and the desired product are known, suitable grinding equipment can be selected.

On the other hand, work index is useful for the estimation of the power consumed due to the size reduction from a given feed size to a given product size, which would be useful in the selection of the proper motor and ball mill sizes and the power requirements for grinding the sample under investigation. In fact, the work index is defined as the power in KWH/tonne required to reduce the test sample from infinite size to a product size, 80% of which passes through 100 microns sieve, from which, the actual power input can be calculated. Naturally, the power consumption is dependent on the feed and product sizes, the reduction ratio, closed and open circuit grinding, wet or dry grinding, etc.

The actual power required to grind a desired size of feed to a desired product can be calculated from the equation.

$$W = 10\,Wi\,(P^{-0.5} - F^{-0.5})$$

Where W = Power input required in KWH/Tonne to reduce the test sample from Size "F" to Size "P".

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

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

Grinding Tests

Sample preparation

The sample ranging in size from 25 mm down to fines is crushed to 10 mesh size in a roll crusher in closed circuit with a 10 mesh screen, and used for grindability tests.

The tests are carried out with representative samples of 10 mesh size, adopting the procedure followed by Denver Equipment Co., U.S.A. for the determination of grinding characteristics of the ores. The method broadly consists in grinding 2000 gms. of the representative batch sample of —10 mesh size with 1000 cc of water for different lengths of time (3 min, interval) in a batch ball mill (30.5 × 12.7 mm) revolving at 54 R.P.M., with a ball charge of 18.2 kg. After grinding, the material is removed and screened to find out the percentage of fines passing a 200 mesh screen produced due to that particular grinding.

In this way, all the grinding tests are done at intervals of three minutes till 80% of the material passes through 200 mesh screen, to a maximum of 45 minutes of grinding time. The data thus obtained is compared with the standard curves designated from soft to hard and the test sample can thus be categorised as soft or medium soft or medium or hard etc. according to where its curve fits in.

Bond's work Index Determination

Sample

For the determination of Bond's work index a representative portion of the sample is crushed to —6 mesh size in a roll crusher in closed circuit with a 6 mesh screen.

Test Procedure

The test procedure developed by F.C. bond of Allis chalmers with some modifications is adopted for determination of work index which consists of the following stages.

(1) Measurement of the size in micron through which 80% of the feed sample (F) passes.

(2) 700 C.C. of the —6 mesh representative test sample is charged into a ball mill (30.5 cm—30.5 cm). fed uniformly rotating at 64 R.P.M. with a ball charge of 20.12 kg. and ground for a known number of revolutions.

(3) The ground sample is removed and screened through a specified sieve of P_1 microns.

(4) The screen undersize is weighed from which the quantity finer than P_1 size originally present in the —6 mesh test sample is deducted so as to get the net undersize produced due to grinding for 'N' revolutions. From this, the net ground material produced per revolution is calculated.

(5) A second feed sample of the ball mill ~~is~~ prepared by combining the screen oversize with a fresh —6 mesh test sample of the same quantity as the screen undersize.

(6) The grinding and screening operations are repeated until the desired circulating load (i.e. 250%) is achieved.

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

(8) The final product obtained during the last three grinds is screen analysed from which the size in micron through which 80% of the product (P) passes is determined. (by plotting)

(9) The work index of the sample is calculated by using Bond's revised equation (1971).

$$\text{Work Index } W_i = \frac{44.5 \times 1.1}{(P)^{0.23} \times (Gbp)^{0.82} \times \left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)} = \text{KWH/T.}$$

Where Wi = Work index in KWH/Ton. required to reduce the test sample from infinite size to the product 80% of which passes thro' 100 microns.

P_1 = Screen size used for screening the ground products (microns).

Gbp = Grindability in net gms 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.

TABLE 6.18—TEST RESULTS OF GRINDABILITY AND BOND'S WORK INDEX

Sl. No. 1	Sample & Inv. No. 2	Grin'ty	P1 microns 3	GbP gms 4	P microns 5	F microns 6	% C.L. 7	Wi KWH/T 8	Remarks 9
1.	Rakha Copper Ore								
	(a) R.O.M. Ore	M-M.S	149	2.786	113	4550	250	8.44	
	(b) R.Moly. Conc.	—	74	0.2135	32	74	—	99.80	Open circuit
2.	Saladipura Pyrite-Pyrhotite								
	(a) Level I	—	208	2.526	162	2380	252.40	5.030	
	(b) Level II	—	208	2.830	166	2580	251.41	4.581	
3.	Bandalamottu Lead Ore	M-M.S	210	2.136	168	2450	252.10	5.789	
4.	Malanjkhand Copper Ore	MH to M	125	1.305	105	2362	250.0	12.97	
5.	I.C.C. copper slag	—	104	0.779	89	3327	249.9	23.93	
6.	Kundremukh Iron Ore	—	62	1.214	49	260	250.1	16.16	
7.	Kundremukh Iron Ore Conc	—	63	1.105	50	425	249.8	18.7	17.37 for dry grinding calculated
8.	Phosphate ore from F.C.I. Sindri								
	(a) 65 mesh size	—	208	3.032	155	2000	249.7	5.56	For dry grinding
	(b) 80 mesh size	—	175	2.719	125	2000	248.1	6.57	
9.	Phosphate rock from Rajasthan	MS-S	210	3.079	154	1735	249.8	10.08	
10.	Maton Block Phosphate—Udaipur	MS-S	74	1.960	44	2380	249.0	10.70	

Note: H=Hard, MH=Medium Hard, M=Medium, MS=Medium Soft, S=Soft.

Settling Rate and Terminal Density

The handling of different process products of finer sizes like flotation concentrates, tailings, cyclone slimes etc. which would be having varied pulp densities etc. different stages of the running plant need specific studies on their settling behaviour. For thickening of these products as well as for reclamation of water to be secured in the plant if possible, it is necessary to determine their settling rate for the calculation of the thickener size and capacity.

The settling tests are conducted in 1,000 c.c. graduated cylinders by measuring the settling rates of the products at different pulp densities like 5, 10, 15, 20, 25% solids with/without addition of floculant. The settling rate is determined at very close intervals initially (say every 15 seconds/30 seconds/one minute) for the first five minutes and the average settling rate is calculated. The ultimate pulp density after allowing the solids to settle for 19 hours is also determined which would indicate the maximum percent solids in the underflow that could normally be expected from a standard depth thickener.

The size of the thickener required can be calculated using the following formula :

$$A = \frac{1.333\ (F - D)}{R}$$

Where A = thickener area in Sq.ft./ton. of dry solids thickened in 24 hours.

F = Initial density (Parts water to solids by Wt.).

D = Final density to which pulp will settle at which pulp is wanted to be discharged from the thickener.

R = Settling rate ft./hr.

It can be generally observed during the settling tests that the water separating from the settling pulp is quite clear and the thickener overflow can, therefore, be reused in the plant, if there is no objectionable reagent in it.

Terminal density

The thickened pulp (thickener underflow) may go next to filtration or to recycling in the plant etc. The terminal density of the thickened pulp after 19 hours of settling can be calculated by the formula.

$$\text{Terminal density} = \frac{\text{Mass of the thickened pulp}}{\text{Total volume of the pulp}}$$

The settling rate results etc. of some Ores are given in Table No. 6.19.

TABLE 6.19—RESULTS OF SETTLING RATE AND ULTIMATE PULP DENSITY OF SOME ORES

Sample	% Solids in feed	Liquid—Solid ratio in feed (F)	Settling rate ft./hr. (R)	Ultimate pulp density % Solids	Liquid-Solid ratio at ultimate pulp density (D)	Area reqd. for settling sft/tonne	125% of above	Area reqd. Sq. m/tonne
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
1. Laminated Iron ore from Rajhara	3	—	5	58.1	—	—	—	—
Mines 366/66	5	—	4	61.3	—	—	—	—
2. Fluorspar concentrate (acid grade	15	—	9.42	63.9	—	—	—	—
Dungarpur Rajasthan	20	—	6.28	64.0	—	—	—	—
464/68 without flocculant)	25	—	4.329	65.0	—	—	—	—
with floculant	5	—	32.2	48.0	—	—	—	—
Separan 0.01 kg/t.	10	—	46.25	56.1	—	—	—	—
after 1/2 mt.	15	—	29.75	62.7	—	—	—	—
	20	—	18.82	62.9	—	—	—	—
	25	—	9.42	63.9	—	—	—	—
3. Kiriburu iron ore NMDC 511/69								
After $6\frac{1}{2}$ mts.	7	—	5.832	60.9	—	—	—	—
7 mts.	10	—	4.168	62.0	—	—	—	—
9 mts.	15	—	3.332	62.5	—	—	—	—
10 mts.	20	—	1.666	62.9	—	—	—	—
14 mts.	25	—	0.833	63.8	—	—	—	—
20 mts.	33.3	—	0.417	65.4	—	—	—	—

TABLE 6.19—RESULTS OF SETTLING RATE AND ULTIMATE PULP DENSITY OF SOME ORES *(Contd)*

Sample	% Solids in feed	Liquid—Solid ratio in feed (F)	Settling rate ft./hr. (R)	Ultimate pulp density % Solids	Liquid-Solid ratio at ultimate pulp density (D)	Area reqd. for settling sft/tonne	125% of above	Area reqd. Sq. m/tonne
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
4. Lateritic iron ore from Goa	3	32.33	9.0	49.5	1.02	4.65	5.81	—
	5	19.0	8.0	46.9	1.09	3.02	3.78	—
	10	9.0	6.0	43.5	1.30	1.83	2.29	—
5. Kudermukh iron ore (a) Concentrate	10	0	25.24	63.77	0.568	0.445	0.556	0.057
	15	5.565	17.76	64.5	0.55	0.384	0.48	0.049
	20	4	16.79	65.1	0.536	0.275	0.344	0.035
	25	3.0	13.8	66.8	0.497	0.242	0.303	0.031
	30	2.333	10.5	68.0	0.471	0.236	0.295	0.03
(b) Tailings	7	13.286	4.0	37.94	1.636	3.886	4.857	0.497
	9	10.111	2.5	40.3	1.467	4.61	5.762	0.589
	12	7.333	1.7	41.7	1.398	4.656	5.82	0.595
6. Bolani iron ore	16	5.25	8.0	79.2	0.263	0.832	1.04	0.1
	18	4.55	7.4	79.4	0.26	0.724	0.905	0.087
	20	4.0	6.64	79.6	0.256	0.751	0.937	0.09
7. Kohndband Iron ore from Tisco	2	49.0	4.35	47.1	1.12	14.60	18.2	1.7
Slime from —50 mm washing	3	32.3	3.3	50.0	1.0	12.6	15.6	1.46
	4	24.0	2.53	52.5	0.91	12.1	15.1	1.4
Slime from —30 mm washing	4.5	21.2	2.88	55.2	0.81	9.4	11.75	1.09
	5.5	17.1	2.30	57.5	0.74	9.1	11.4	1.07
	6.5	14.4	2.08	59.0	0.70	8.8	11.0	1.02
8. Joda Iron ore Tisco	2.0	49.0	4.95	45.2	1.21	12.8	16.0	1.49
Slime from —50 mm washing	3.0	32.3	4.23	46.0	1.17	9.8	12.25	1.14
	4.0	24.0	3.44	46.8	1.14	9.35	11.7	1.09
Slime from —30 mm washing	3.0	32.3	3.64	45.0	1.22	11.35	14.2	1.32
	4.0	24.0	3.0	46.0	1.17	10.1	12.6	1.17
	5.0	19.0	2.4	47.1	1.12	9.82	12.3	1.14
9. Copper slag from I.C.C.	5.0	19.0	9.76	70.4	0.375	2.5437	3.1797	0.296
	7.5	12.33	8.64	70.2	0.425	1.8367	2.2959	0.214
Final concentrate	10.0	9.0	6.5	70.8	0.412	1.7612	2.2015	0.205
Final Tails	30.0	2.333	2.84	75.2	0.33	0.9434	1.1796	0.11
	34.6	1.89	2.24	74.2	0.348	0.9176	1.147	0.107
	40.0	1.5	1.69	72.9	0.372	0.8897	1.1121	0.103
10. Rakha Mines Copper Ore	5	19	7.0	63.7	0.571	3.53	4.411	26.226
Final Copper Nickel concentrate	10	9	4.4	63.7	0.571	2.55	4.411	26.226
Final Tails	20	4	3.5	62.3	0.603	1.3	2.59	225.217
	25	3	1.7	62.3	0.603	1.88	0.59	225.217
	33.3	2	0.9	62.3	0 603	2.07	0.59	225.217
Final Moly concentrate	5	19	59.0	63.3	0.579	0.416	3.613	0.336
	10	9	10.7	63.3	0.579	2.0	3.613	0.336
	12.5	7	8.1	63.3	0.579	2.89	3.613	0.336
11. Maton Bolck Phosphate Final concentrate	5	19.0	11.98	65.9	0.517	2.093	2.626	—
	10	9.0	10.2	65.9	0.517	1.128	2.626	—
	15	5.67	7.66	65.9	0.517	0.916	2.626	—
Middling	5	19.0	12.2	65.8	0.519	1.989	2.736	—
	10	9.0	8.4	65.8	0.519	1.430	2.736	—
	15	5.67	7.02	65.8	0.519	0.954	2.736	—
Final Tails	5	19.0	12.6	65.7	0.520	1.987	2.484	—
	10	9.0	7.2	65.7	0.520	1.663	2.484	—
	15	5.67	6.46	65.7	0.520	1.081	2.484	—

Filtration

The thickened pulp has to be filtered before going to the next stage. The selection of a proper size vaccum filter for dewatering the thickener underflow can be done by conducting filtration tests.

The rates of filtration of the thickened pulp are determined using standard EIMCO Test filter disc. EMICO filter is circular in shape with a grid face area of 0.00929 sq. mtrs. (0.1 sq. ft.). The filter cloth is secured in position on the leaf with an adjustable clamp. The filter leaf is connected to a filtrate receiver through a flexible hose pipe, which, in turn is connected to a vaccum pump.

The filter disc under vaccum (56 cm. of Hg.) is submerged for a pre-determined time in a well agitated pulp containing 50% solids, for cake formation and afterwards it is taken out of the pulp to allow the wet cake dewater itself for the same time under same vaccum. The filter cake is then removed from the leaf by blowing air through it. The thickness of filter cake, its moisture content, its weight and the quantity of filtrate are measured to determine the rate of filtrate collection. Thus one cycle of operation of the filtration test comprised of (1) submergence (2) dewatering (drying) and (3) blowing. The blowing time is kept at 1/4 of the entire cycle time and the rest is divided equally between the submergence and the dewatering time.

The tests are repeated with different pulp densities and also varying the cycle time.

TABLE 6.20—FILTRATION TEST RESULTS:—

Sample	Floculant	% Solids in Pulp	Total cycle time in Secs.	Thickness of the cake in cm.	% moisture	Filtrate Quantity Litre/hr/sq.m.	Dry cake formation Kg/hr/sq.m.	Wet wt. of filter cake in gm.	Dry wt. of filter cake in gms.
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)
1. Laminated iron ore from Rajhara Mines	No Floculant	60	480	—	22.4	—	255.8	16.16	13.2
	Alum. 0.3 kg/ton.	60	480	—	21.0	—	378.7	23.66	19.55
	Lime 1.5 kg/ton.	60	480	—	19.6	—	538.7	33.25	27.8
	0.5 kg/ton.	60	480	—	20.0	—	556.0	34.44	28.7
2. Fluorspar concentrate	—	60	40	0.6	17.1	762.27	1082.4	—	—
acid grade from	—	60	80	0.9	15.3	476.6	685.44	—	—
Rajasthan	—	60	120	1.1	12.8	360.9	493.45	—	—
	—	50	80	0.3	16.3	346.5	170.16	—	—
	—	50	120	0.4	14.8	277.2	130.51	—	—
	—	50	160	0.4	14.3	268.5	124.30	—	—
	Flocal	60	40	0.85	15.0	953.0	1363.1	—	—
	T-214	60	80	0.90	15.4	506.9	744.4	—	—
	0.025 Kg/ton.	60	120	1.0	14.0	369.7	547.0	—	—
		50	80	0.50	14.9	476.5	332.1	—	—
		50	120	0.70	14.6	447.6	296.3	—	—
		50	160	0.72	14.5	368.2	293.6	—	—

TABLE 6.20—FILTRATION TEST RESULTS:—*(Contd.)*

Sample	Floculant	% Solids in Pulp	Total cycle time in Secs.	Thickness of the cake in cm.	% moisture	Filtrate Quantity Litre/hr/ sq.m.	Dry cake formation Kg/hr/ sq.m.	Wet wt. of filter cake in gm.	Dry wt. of filter cake in gms.
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)
3. Copper slag from	—	70	80	1.75	22.39	571.58	2010	—	—
IC.C.	—	70	120	2.1	22.63	458.56	1518	—	—
		70	160	2.35	22.72	392.36	1245	—	—
		60	200	1.35	22.76	479.46	864	—	—
		60	240	1.65	22.65	371.37	651	—	—
		60	320	2.0	23.18	333.02	525	—	—
		50	160	0.4	25.02	402.05	300	—	—
		50	240	0.55	25.78	339.07	231	—	—
		50	320	0.7	25.39	326.96	242	—	—
4. Maton Block Phosphate Rack from		40	40	0.25	—	775.0	0.36	—	—
Rajasthan.			80	0.35	—	460.0	0.25	—	—
			120	0.40	—	410.0	0.20	—	—
			160	0.45	—	376.0	0.19	—	—
		50	40	0.55	—	349.0	0.96	—	—
			80	0.65	—	557.0	0.54	—	—
			120	0.70	—	394.0	0.39	—	—
			160	0.85	—	336.0	0.37	—	—
		60	40	0.60	—	630.0	1.10	—	—
			80	0.75	—	446.0	0.76	—	—
			120	1.10	—	381.0	0.64	—	—
			160	1.40	—	339.0	0.53	—	—
5. Copper Ore from		60	120	0.7	20.3	252.0	0.543	—	—
Rakha Mines. IR		60	160	0.8	20.2	190.5	0.407	—	—
		60	240	1.0	20.1	133.0	0.375	—	—
		50	160	0.35	18.0	195.9	0.230	—	—
		50	240	0.5	17.6	161.5	0.218	—	—
		50	320	0.7	17.2	130.0	0.183	—	—

Reducibility

Reducibility may be defined as a measure of the rate of de-oxidation in case of iron ores. Various methods are employed to assess the same.

One of the standard methods adopted is to determine the time taken for 90% reduction of the sample and compare it with a standard or known sample. This comparison of the rates of reduction of different iron ore samples under similar conditions can be made from the reduction curves drawn between time versus percentage reduction at a given time.

The reduction of iron oxide to metallic iron above 570°C takes place in three steps.

$$3\,Fe_2O_3 + H_2 = 2\,Fe_3O_4 + H_2O$$
$$Fe_3O_4 + H_2 = 3\,FeO + H_2O$$
$$FeO + H_2 = Fe + H_2O$$

Below 570°C the reduction proceeds from Fe_3O_4 directly to Fe since FeO (wustite) is unstable below 570°C. The gaseous reduction of dense iron ore is of topochemical reaction that proceeds from the outer surface inwards, forming reaction zones which are approximately parallel to the original surface of the specimens. However, such topochemical mode of reduction is affected by cracks, pores, impurities and residual oxide phases developed during the reduction process.

For reducibility studies representative sample is crushed to —50 mm size and screened to closely sized fractions viz. —38.1 + 25.4 mm, —25.4 + 12.7 mm, —12.7+9.5 mm and —9.5 mm + 4.7 mm. 200 gms of such closely sized fraction is reduced by hydrogen gas flowing at 1.7 litres per minute at a temp. of 850°C ± 10°C in a horizontal tubular electric furnace.

Test Procedure

The reduction test is carried out with commercial hydrogen. It is dried and purified by passing through conc. H_2So_4, anhydrous cal, chloride, ascertite and through copper turnings maintained at a temp, of 500°—550°C. The test sample is pre-dried and after weighing, it is placed inside a stainless steel reduction tube and its positions is so adjusted as to keep the sample at the centre of the furnace. The sample is heated to 850°C and nitrogen flushed through to prevent oxidation before reduction commences with hydrogen. The temp, of the sample is measured with a chromel-alumel thermo-couple placed near the centre of the sample and is controlled by a dimmerstat. Extreme care is taken that there is no leakage in the experimental setup. Rate of flow of hydrogen is measured by a pre-calibrated flow meter. The excess hydrogen gas and the water vapour produced during reduction process is passed through a spiral condenser which is kept in a freezing mixture, and through which cold water is circulated. The condensed water is collected in a separating flask connected to the bottom of the condenser and measured. The reducing gas is further passed through two U tubes full of anhydrous calcium chloride to pick up residual water vapour. The exit gases from these U tubes containing chiefly hydrogen are burnt away. The quantities of condensed water collected in separating flask as well as that picked up by U tubes are noted after each interval of 15 minutes. The experiment is continued till there is no further condensation of water during a period of 30 minutes. At the end of the experiment the system is cooled to room temperature under nitrogen atmosphere and the weight of the sample is again noted. The percentage reduction of the sample under test is calculated as follows :

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

Swelling Index

There will be increase of volume of iron ore when reduced in a reducing atmosphere. This increase in volume is measured and termed as swelling index.

The test procedure consists of measuring the present increase in volume of the ore due to reduction at a particular temperature and for a given time.

Procedure

The test sample will be of —12+10 mm ore. It is oven dried at 105° + 5°C for 60 minutes in order to remove moisture initially.

The volume of the test piece is measured first by mercury hydrometer. It is placed in the reduction tube which is connected to a gas circuit, and heated upto 850°C in an inert atmosphere using purified Nitrogen gas. After maintaining the temperature of the sample at 850°C for 30 minutes in the inert atmosphere, purified hydrogen gas is passed in place of nitorgen at the rate of 100 cc/minute for reducing the ore. It is reduced for 100 minutes at 850°C ± 10°C. At the end of 100 minutes nitrogen is passed and the sample is cooled to room temparature. The volume of the ore piece after reduction is measured as before.

The swelling index is calculated using the formula:

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

Where SW = % Swelling index

V_1 = Volume in cc before reduction of the sample

V_2 = Volume in cc after reduction of the sample

Some test results on reducibility of some ores are given in Table 6.21, and Table 6.22 and swelling index results in Table 6.23.

TABLE 6.21—REDUCIBILITY TEST RESULTS

Sample & Inv No.	Ore/Sinter size	Temp. °C	Flow rate of Hydrogen. (litres/Min)	Time for 90% Reduc'n. in Mins.
(1)	(2)	(3)	(4)	(5)
1. Bolani iron ores and sinters.				
Ore— (1)	—0.263+0.185″	600	1.7	261
(2)	-do-	700	1.7	155
(3)	-do-	800	1.7	115
(4)	-do-	900	1.7	110
Opti-tests (5)	-do-	800	1.2	260
(6)	-do-	-do-	1.7	120
(7)	-do-	-do-	1.8	115
(8)	-do-	-do-	2.1	92
(9)	-do-	-do-	2.3	85
(10)	-do-	-do-	4.0	72
(11)	-do-	-do-	5.3	62 Opt'm
(12)	-do-	-do-	7.5	55
(13)	−3+2½″	800	5.3	260
(14)	−2½+2″	-do-	-do-	200
(15)	−2″+1½″	-do-	-do-	160
(16)	−1½+1″	-do-	-do-	185
(17)	−1″+¾″	-do-	-do-	150
(18)	−¾″+⅜″	-do-	-do-	160
HMS Products. (19)	−3″+2½″	800	5.3	260
(20)	−2½+2″	-do-	-do-	200
(21)	−2″+1½″	-do-	-do-	160
(22)	−1½+1″	-do-	-do-	185
(23)	−1″+¾″	-do-	-do-	150
(24)	−¾″+⅜″	-do-	-do-	160
Sinters.	+3/8″ washed cl. sand non fluxed.			
(1)	−0.263″+0.185″ size	800	5.3	151
(2)	-do-	-do-	-do-	130
(3)	Fluxed sinter	-do-	-do-	56
(4)	-do-	-do-	-do-	62
(5)	-do-	-do-	-do-	55
(6)	-do-	-do-	-do-	49
(7)	−⅜″+6 mesh HMS + −6 mesh Jig conc. non-fluxing	-do-	-do-	75
(8)	-do- fluxed	-do-	-do-	55
(9)	Washed ore	-do-	-do-	65

The Bolani iron ore may be classed amongst the more easily reducible ores. It has been evident from the results that the washed products and HMS product had the maximum reducibility. Beneficiation not only serves to secure the removal of silica and alumina but also affects distinct improvement in the reducibility. —2″ + 3/8″ fraction should be a preferable charge for blast furnaces.

TABLE 6.22

2. Specular hematite Iron ore samples of Nathara-Ki-Pal Deposits, Rajashthan 324/65. (Darlamata & Amsivali Deposits)

Samples & Size	Temp. °C	Flow rate of Hydrogen Litres/mt.	Time for 90% reduction in minutes
(1)	(2)	(3)	(4)
1. Table Conc. Briquettes 50 gm.	800	3.0	35
2. -do-	800	3.8	34
3. -do-	800	4.5	30
4. -do-	800	6.0	25
5. -do-	800	7.0	24
6. -do-	500	4.5	Very slow; not completed
7. -do-	600	4.5	44
8. -do-	700	4.5	35
9. -do-	800	4.5	30
10. -do-	900	4.5	25
11. Darlamta lumpy ore 50 gm.	800	4.5	50 (70% reduction only)
12. Amsivali lumpy ore 50 gm.	800	4.5	42 (80% reduction only)
13. Amsivali −36+60 mesh Briquettes	800	4.5	30
14. -do- −60+200 mesh briquettes	800	4.5	28
15. Darlamata table conc. briquettes	800	4.5	28
16. -do- Spiral + table conc. briquettes	800	4.5	28
17. Amsivali table conc.	800	4.5	30
18. ,, Flotation conc.	800	4.5	31
19. Darlamata flotation conc.	800	4.5	31
20. Amsivali Table Conc. briquette (−60+200 mesh)	800	4.5	30
21. Kiriburu iron ore −60+200 mesh briquette	800	4.5	35
22. Bolani (washed)	800	4.5	40
23. Bolani washed lumpy	800	4.5	46

Optimum tests:

From the above results it is seen that the lumpy ores are reduced with difficulty and the reduction did not proceed bəyond 72% and 90% respectively in the Darlamata and Amsivali ores. The crushed samples were found to be highly reducible even with the presence of gangue. The concentrates were highly reducible. When compared with Bolani ores the reduction rate of the table concentrates was found to be faster by 1 1/2 times as that of the lumpy washed Bolani ore. The bri- quetted samples of the concentrates were disintegrated under reducing conditions and this faciliated the faster reduction of the ores by improvingt he diffusion of the reaction gases.

TABLE 6.22 *(Contd.)*—**3 Reducibility of Kiriburu Iron ores and Sinters**

	Sample	Temp. °C	Hydrogen gas flow rate litres/mt.	Time taken for 90% reducibility in minutes	
	(1)	(2)	(3)	(4)	
(A)	U. Portion+Bench 3				
1.	Original ore (—0.263" + 0.185") size	800	3.5	105	
2.	Washed + scrubbed + wet screen at 50 mm	800	3.5	103	
3.	-do- at 100 mm	800	3.5	103	
4.	Wet screened at 100 mm	800	3.5	104	
5.	Hard massive —50+45 mm	800	3.5	162	—182
6.	Massive laminated	800	3.5	148	
7.	Massive porous laimnated	800	3.5	130	
8.	Laminated blue dust	800	3.5	100	
9.	Goethite	800	3.5	87	
10.	Goethite with ochre (Red)	800	3.5	74	
11.	Laterite (very porous) Massive laminated	800	3.5	62	
12.	Lump —100+70 mm (500 gm.)	800	3.5	270	
13.	—100+70	800	3.5	150	Disintegrated
14.	—70+50	800	3.5	250	cracked
15.	—70+50	800	3.5	220	Disintegrated
16.	—70+50	800	3.5	155	,,
17.	—50+40	800	3.5	160	cracked
18.	—40+25	800	3.5	145	,,
B.	**Hill 1 Bench 2 + Hill 2 Bench 2**				
19.	—50+25 mm lumps	800	3.5	140	
20.	—75+50 mm ,,	800	3.5	208	
21.	—100+50 mm ,,	800	3.5	212	
22.	—100+75mm "	800	3.5	260	
23.	—75+50 mm ,,	800	3.5	204	
24.	—50+38 mm washed Kiriburu	800	3.5	100	
25.	,, Bolani washed	800	3.5	225	
C.	**Sinters**				
	U Portion + Bench 3 Hill 1 Ore				Shatter
26.	—3/8" washed ore	800	3.5	115	57.9
27.	—3/8" washed + beneficiated ore	800	3.5	95	49.9
	Fluxed sinters with limestone				
28.	0.8 Basicity	800	3.5	85	59.4
29.	1.0 ,,	800	3.5	85	52.4
30.	1.4 ,,	800	3.5	80	56.5
31.	1.6 ,,	800	3.5	80	56.3
	with dolomite				
32.	1.2	800	3.5	85	63.1
33.	1.4	800	3.5	125	56.5
34.	1.6	800	3.5	106	66.1
35.	9:1 Fines:Blue dust ratio	800	3.5	119	53.85
36.	7:3 ,,	800	3.5	95	61.9
37.	10:0 ,,	800	3.5	106	66.1
38.	Mixed Firing coke 2.1%	800	3.5	100	64.6
39.	2.8%	800	3.5	69	61.9
40.	3.5%	800	3.5	106	66.1
D.	**Hill No. 1 Bench 2 + Hill Hill No. 2 Bench 2**				
41.	Unfluxed sinters —1/2" washed ore	800	3.5	195	—
42.	—1/2" washed + beneficiated ore	800	,,	172	—
	Fluxed sinters with limestone				
43.	Basicity 0.8	800	,,	135	55.0
44.	1.2	800	,,	90	57.9
45.	1.4	800	,,	110	61.2
46.	1.6	800	,,	117	60.8
	with dolomite				
47.	Basicity 1.2	800	,,	100	54.9
	1.4	800	,,	115	52.3
	1.6	800	,,	135	66.9
	With blue dust				
	ore : Blue dust				
48.	9:1	,,	,,	107	56.8
49.	8:2	,,	,,	120	61.9
50.	7:3	,,	,,	125	54.8
51.	10:0	,,	,,	135	66.9
	Mixed Firing		,,		
52.	Coke 2.8% (30% decrease)	,,	,,	110	74.7

From the results it can be seen that the fluxed sinters of washed and beneficiated ore are more easily reducible than the unfluxed sinters of washed ore due to their increased Fe Content and decreased gangue.

Self fluxed sinters with limestone showed better reducibility than unfluxed sinters. Self fluxed sinters with limestone-dolomite showed lower reducibility characteristics than the fluxed sinters with limestone alone.

Blue dust addition upto 30% can increase the reduction rate in the fluxing sinters.

TABLE 6.22 *(Contd.)*—**Reducibility of Surajgarh Iron ores**

Size	Time in mts. for 90% reduction				
	known samples		Test samples Surajgarh		
	Rajhara iron ore	Dalli massive iron ore	Laminated ore	Float ore	Massive ore
—38+25 mm	92	88	62	70.5	45.5
—25+12.5 mm	82.5	79	48	74.5	44.5
—12.5+9.5 mm	78	43.5	47	81	41
—9.5+4.7 mm	88.5	48	40	82.5	43

From the above results it is seen that the rates of reduction of laminated and massive ores of Surajagarh deposits are found to be faster as compared to Rajhara iron ore. Among the three test samples the massive ore seems to be the most easily reducible ore followed by laminated and float ores except in size range —9.5+4.7 mm where the rate of reduction of laminated ore is higher than the rest.

TABLE 6.23—SWELLING INDEX RESULTS:

(1) Kiriburu and Barajamda Iron ore samples

	Sample	Swelling Index
1.	Kiriburu Iron ore	2.544%
2.	Barajamda Iron ore	2.91%

Decrepitation and Weathering studies:

In the case of iron ore sinters the effect of weathering during storage resulting in the generation of fines (—12.7 mm) is to be studied.

The tests involved in taking a representative sample of known weight is screened over a 12.7 mm screen (1/2") once in every 4 hours for 24 hours and the over-size kept everytime for weathering. The amount of —12.7 mm fines produced indicates the quality of the sinters and its behaviour of decrepitation due to weathering under storage conditions.

Some test results of weathering on sinters are given in Table 6.24.

TABLE 6.24—TEST RESULTS OF WEATHERING OF SINTERS

Sinter sample basicity	Time of exposure in hrs.	% —$\frac{1}{2}$" produced after screening	Sinter Basicity	Exposure period hours	Wt. of +$\frac{3}{8}$" sinter before test	% Wt. of —$\frac{3}{8}$" produced after test
(1)	(2)	(3)	(4)	(5)	(6)	(7)
1. Laminated iron ore from Dalli Mines			2. Mixed laminated iron ore from Dalli —$\frac{3}{8}$" washed fines			
1.4	4	2.5	1.8	4	12.5 kg.	0.26
	8	3.0		16	12.5	0.56
	12	3.5		20	12.5	0.71
	24	3.9		24	12.5	0.93
1.6	4	2.4	2.2	4	12.0	0.29
	8	2.8		16	12.0	0.97
	12	3.4		20	12.0	1.13
	24	3.8		24	12.0	1.57
2.0	4	2.1	2.4	4	11.5	0.67
	8	2.5		16	11.5	1.48
	12	3.5		20	11.5	1.93
	24	3.8		24	11.5	2.62
			—$\frac{3}{8}$" washed and beneficiated fines			
			1.8	4	10.9	0.23
				16	11	0.50
				20	11	0.60
				24	11	0.87
			2.2	4	10.5	0.24
				16	11	0.80
				20	11	1.00
				24	11	1.40
			2.4	4	11.0	0.64
				16	11	1.61
				20	11	1.90
				24	11	2.40

The results show that the fines produced due to storage and weathering are very negligible in quantity and are within limits.

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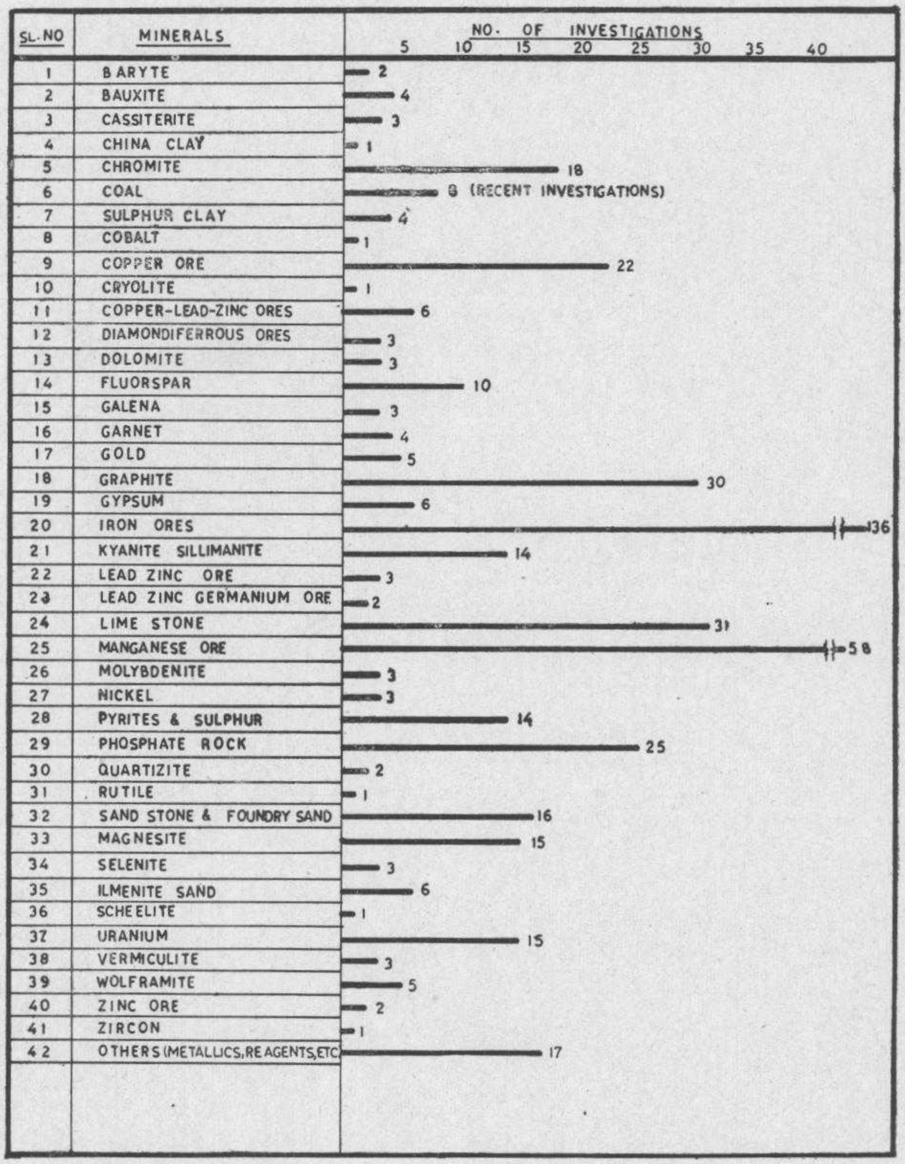

Fig. 6.1—Number of Investigations Conducted on Various Ores, and Minerals (1978)