WINNING COAL & IRON ORE

VOL. I

BY

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PREFACE

The popularity of our first work "Winning and Working Coal in India" and the demand from a section of readers to include iron ore mining in our next edition, prompted us to undertake the present publication "Winning Coal and Iron Ore", a treatise on mining methods in our premier mining industries. The scope of the subject is very wide and pervasive and barring a few articles on topical issues in professional journals there is no published literature in India offering consolidated treatment of the subject. The need for this publication is, therefore, self-evident and no apology is attempted in explaining the origin of the work.

We do not claim perfection nor exhaustive treatment ; in fact we are conscious of the limitations in India that inhibit free flow of information and know-how from practising firms to technical writers, grateful though we are to these enlightened establishments and executives who did break through the traditional conservatism to share their knowledge with us.

We have devoted the first few chapters of this work to the treatment of the historical background of these major mineral industries, delineation of the present problems and perspective look into the future. We then passed on to geology of the two fields, exploration and prospecting and opening of the new mines. Considering the importance and prevalence of open cast mining in these fields, we have dealt at considerable depth and expanse with the face layouts, deployment of machinery, basic principles underlying their operation and maintenance, in this type of mining. Since breaking ground is the essence of all mining methods, we dilated on the subject of explosives and substitutes in subsequent chapters. We propose to deal with underground methods of work and preparation of the minerals, including beneficiation, in the second volume.

We are indeed very grateful to many of our colleagues and well-wishers who have overwhelmed us with their generous

co-operation and wise counsel, to manufacturers of mining equipment who readily made available their literature and catalogues and to numerous authors of publications on whose original ideas and experience we have profusely drawn. To name a few at this place would be drawing an invidious distinction and we, therefore, earmark a separate place for "acknowledgements", seeking the pardon of those whose names might have been inadvertently omitted. Special reference is, however, due to Dr. O. P. Varma, Professor of Geology, Indian School of Mines, who wrote practically the complete chapter on iron ore deposits, to Dr. J. Singh, Professor of Geophysics, Indian School of Mines, Dhanbad, who scrutinised the chapter on prospecting, and to Sri R. K. Pande, General Works Manager, Pandit Brothers, Asansol, who scrutinised the chapter on basic principles of heavy earthmoving machinery.

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Dhanbad
31st May, 1967.

R. T. Deshmukh
D. J. Deshmukh

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Dr. O. P. Varma, Professor of Geology, ISMAG.

In this book

Ton or tonne refers to metric tonne.

Present terse, wherever used, refers to late 1966 and early 1967.

Chief Inspector of Mines is the old designation for Director General of Mines Safety. Regional Inspector of Mines is the old designation for Joint Director of Mines Safety. The new designations have come into effect from 1st May 1967.

Abbreviations used :

g—gramme ; t—tonne ; km—kilometre ; h or hr—hour ; m—metre ; kg—kilogramme.

Note—no 's' is added for plural.

CONVERSION TABLE

Inches	×25.4 =Millimetres.	Millimetres	× .039 =Inches.
Feet	× .305 =Metres.	Metres	×3.28 =Feet.
Yards	×.914 =Metres.	Metres	×1.09 =Yards.
Miles	×1.609 =Kilometres.	Kilometres	× .621 =Miles.
Lbs. per Sq. inch	× .0703 =Kg/Cm².	Kg/Cm²	× 14.2 =Lbs. per Sq. inch.
Cft.	× .0283=Cubic metres	Cub. metres	× 35.31=Cubic feet.
Cyd.	× .764 =Cubic metres	Cub. metres	×1.307 =Cubic yards.
Pounds	× .453 =Kilograms.	Kilograms	× 2.2 =Pounds.
Ounces	× 28.35 =Grams.	Grams.	× .035 =Ounces.
Sq. ins.	× 6.451 =Sq. cms.	Sq. cms.	× .155 =Sq. inches.
Sq. ft.	× .092 =Sq. metres.	Sq. metres	×10.76 =Sq. feet.
Sq. yards	× .836 =Sq. metres.	Sq. metres	×1.195 =Sq. yards.
Sq. miles	×2.59 =Sq. kilometres.	Sq. kilometres	×.386 =Sq. miles.
Imp. gallons	×4.54 =Litres.	Litres	× .219 =Gallons (imp.)
Long ton (2,240 lbs.)	×1.016 =metric tonne.		

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+ Indicates iron ore field and black dot, coal or lignite field.
W. Tiruchi is also known as Tiruchirapalli.

CHAPTER 1

INDIAN COAL INDUSTRY

1. Introduction. 2. Early history. 3. Mining conditions. 4. Size of mines and pattern of mechanisation. 5. Specifications, commercial grades and markets of coal. 6. Coal washeries.

1. INTRODUCTION

Energy consumption of a country is taken as an index of its economic development. The primary sources of commercial energy are coal of all ranks, oil and natural gas, hydropower and nuclear fuels. According to United Nations' Statistical Year Book, 1964, 45% of the world's total production of energy came from coal, 34% from crude petroleum, 15% from natural gas and 6% from water power. Of course, oil and natural gas appear as chief competitors to coal and though the demand for coal is steadily falling in many countries, particularly Western Europe, U. S. A. and Japan, coal will perhaps remain the principal source of energy in India, backed by oil, natural gas and nuclear fuels. The Energy Survey Committee of India (1963-1965) reported that in 1960-61, out of the total energy in India of the tune of 242 million tons in coal replacement units, 22% came from coal, 19% from oil and about 4% from hydro-electricity, remaining 55% being the non-commercial energy from cowdung, fire-wood, vegetable waste, etc. On the assumption of national income growth of 7% and corresponding industrial growth of 10% annually, the total energy demand is likely to increase to 439 million tons of coal replacement units in 1970-71 and 857 million tons in 1980-81 ; out of this 256 and 648 million tons respectively will be the demand for commercial energy. To guard against excessive dependence on imported fuels, India must obtain this requirement from indigenous sources. In terms of heat value, present known resources of coal in India are estimated at a thousand times those of known reserves of oil and ten times still

larger than those of Uranium[1]; it is, therefore, expected that coal will play a predominant role in the energy pattern in the years to come.

2. EARLY HISTORY.

Historic names of localities as Angarpathra, Kalijora, Barakar, Kalipahari, etc. suggest that coal was known in India for a long time. The first recorded mining venture was started by Messrs Heatly and Sumner in 1774 in Raniganj coalfield of West Bengal.[2] The industry, however, did not progress till 1885, when the coal consuming industries, namely, railways, cotton, jute, paper mills appeared on the scene to give support. Jharia Coalfield was opened in 1890-93 when Grand Chord Railway line was constructed. Other fields entered the arena almost contemporaneously, Giridih in 1857, Madhya Pradesh in 1862, Singareni in 1886, Upper Assam in 1881, so that in 1900 the output of coal reached 6.12 million tons with the help of steam power for pumping and winding.

The first Mines Act was enacted in 1901, which brought into being the Department of Mines in 1902. To impart mining education to meet the requirement of statutory positions in mines, a mining department was started in 1905 in the Bengal Engineering College, Sibpur. Establishment of a colliery power station in 1906 and introduction of a bar coal-cutter in 1908 were important milestones in the technological progress in coal mining but further advancement was marred by a slump (fortunately, short-lived) in coal market in 1911. However, the starting of operations by the Tata Iron & Steel Co. in 1911 and the First World War in 1914 revived the industry. During the war a control machinery was set up to meet the demand of coal for the Admiralty and the Government. The output rose to 17.9 million tons by 1920.

Period from 1920 to 1945

During the period from 1920 to 1945, the industry passed through two slumps, one of minor nature in 1925 and another of major dimension during 1931-36. Before the onset of the major slump, coal prices, dictated by the law of demand and

supply, ruled firm (except in 1925) owing to the demand from newly set up factories at Kulti and Hirapur ; recovery after the slump was gradual. War years of 1941-45 gave a boost to the industry, so much that in April 1944 Government had to issue Coal Control Order to fix prices for different grades of coal, a system which is still operating with modifications. The price of selected A grade was fixed at Rs. 13.30 per tonne.

Developments on the technical side during the period under discussion were : the amendment of Mines Act in 1923, framing of Coal Mines Regulations in 1926, introduction of Workers' Compensation Act in 1923 and incorporation of mining diseases for compensation in 1938. As a sequel to a series of explosions and fires in 1935-36, Coal Mines Regulations were amended and Coal Mines Committee was formed, which reported in 1937. As a result, Coal Mines Safety (Stowing) Act was passed in 1939, under which Coal Mines Stowing Board was instituted, which was a forerunner of the present Coal Board. This Board raised funds by levying a cess on all coal despatches and granted loans and assistance for stowing and other protectives works. Further, Mines Rescue Rules were promulgated in 1939 bringing into being, in 1941, two Rescue Stations at Dhansar and Sitarampur.

In education, mining department was started in the Banaras Hindu University in 1923 and the Indian School of Mines, Dhanbad, was established in 1926, to impart degree level education in Mining Engineering.

On the labour front, employment of women underground was prohibited in 1937 ; this ban was lifted for a short period from December 1943 to February 1946 when paucity of labour in the coal industry threatened to thwart war effort. Other measures during war years to attract labour for mining were supply of free rations, production bonus scheme, increase in wages and a number of welfare measures financed by Coal Mines Labour Welfare Fund which was created by an ordinance in 1944 by a levy on coal despatches.

POST-WAR PERIOD (1945-1960)

Coal Control Order, 1944, was an important land-mark in the history of coal industry ; besides fixing prices of different

grades of coal, it created an organisation (Coal Commissioner's Organisation) for equitable distribution and rationalising the transport and industrial consumption of coal. Government interest in the coal industry in those days was reflected in the setting up of Coal Advisory Committee in 1945, Coal Transport Advisory Committee, 1947, and Indian Coalfields Committee, 1945 ; the last-named Committee submitted its report in 1946 on future economic and administrative measures to deal with problems of the coal industry. Immediate post-war period was characterised by labour unrest which continued unabated despite voluntary wage increase in 1946 and prompted the Government to appoint a Conciliation Board under the Chairmanship of Justice Puranik. The Board awarded higher wages for labour and introduced Provident Fund and Bonus Schemes, which found legal expression in Provident Fund and Bonus Scheme Act of 1948 (subsequently revised in 1951) ; under its impact the coal price had to be increased by Rs. 3.50 per ton. The peace generated by this award was, however, soon broken by numerous strikes in 1953 and in 1954, the Government appointed All-India Industrial Tribunal (Colliery Disputes) under the chairmanship of Justice J. N. Majumdar. The Award, published in 1956, created ten categories of workers, enhanced wages by 40% and evoked a price increase of Rs. 3/- per ton of coal. There was, however, widespread dissatisfaction with this award and the dispute went to Labour Appellate Tribunal, which awarded in January 1957 substantial increase in wages of some categories and provided for an automatic increase in dearness allowance with an increase in cost of living index. The coal prices have increased with every increase of wage rates and this trend is making coal so costly as to be virtually pushed out of use in distant southern and western India. Appellate Tribunal's award kept peace for considerable time ; on labour unions' demand a Wage Board has been constituted in 1962 for coal industry, whose award is expected shortly.

To pursue some of the recommendations of the Indian Coalfields Committee, 1946, Government appointed Metallurgical Coal Committee in 1949, a Working Party for Coal Industry in 1950 and with their recommendations set up, in

1951, Coal Board (replacing the earlier Coal Mines Stowing Board) charged with the responsibility for conservation of coal resources and safety of mines and invested with powers under the Coal Mines (Conservation and Safety) Act 1952. The Board gives assistance for stowing and for protective works, and grants subsidy to collieries handicapped by adverse mining conditions such as thin seams, gassy nature of seams, steep gradient, great depth of working, high overburden : coal ratio in opencast mines, high cost of transport from pit head to rail-head etc. To finance its extended activities the Board is empowered to levy cess upto Rs. 4/- per tonne on coal despatches under the Amending Act of 1961. The present (1966) rate of cess is Rs. 1.653 per tonne of coal and Rs. 2.48 per tonne of hard coke.

Among other developments in the post-war period, mention must be made of Mines and Minerals (Regulation and Development) Act of 1948 and Mineral Concession Rules framed under it to fix procedures for granting and operating mineral concessions and prescribing royalties payable to State. This Act was consolidated in 1957 to deal with minerals other than petroleum and is now operative as Mines and Minerals (Regulation and Development) Act, 1957 together with Mineral Concession Rules, 1960. Mines Act of 1923 was replaced by a more comprehensive and stringent Act of 1952 and new Coal Mines Regulations were formed in 1957. Rescue Rules of 1939 were amended in 1959 which, *inter-alia*, provided for more rescue stations in the outlying fields. There are eight rescue stations now in operation.

In 1954 Coal Washeries Committee published its report and focussed attention to beneficiation of coals for metallurgical purposes, recommending setting up of several washery units. In 1955, an Amalgamation Committee was set up under the chairmanship of Balwantrai Mehta, which recommended that small mines should be amalgamated to form sizeable units of production of more than 10,000 tons per month. In April 1956 in an Industrial Policy Resolution, coal was included in the list of industries earmarked for development in the public sector and a public corporation styled as National

Coal Development Corporation was created in 1956 to take care of coal mining in the public sector but not so as to interfere in the existing private sector; to help it acquire coal bearing land with compensation, Coal Bearing Areas (Acquisition and Development) Act was enacted in 1957. To work the mixed economy and co-ordinate efforts in the two sectors, Coal Council (subsequently renamed as Coal Advisory Council and recently restyled as Coal Development Council) was formed in August, 1956. Planning for coal production began with the Second Five Year Plan (1956-61) and production target for the year 1960-61 was fixed at 60 million tons. The country by and large acquired the planned production capacity by the year end, though actual production in 1961 was short of the target, at 56 million tons, owing to bottleneck in railway coal movement capacity.

Other events worthy of note during this period were the establishment in 1946 of the Central Fuel Research Institute at Jealgora, Dhanbad and in 1955, of the Central Mining Research Station, Dhanbad, two of the national laboratories under the Council of Scientific and Industrial Research. The former initiates and directs research into problems of utilization of fuels and their by-products and has, besides, several Coal Survey Stations in different parts of the country. The Central Mining Research Station conducts research into mining problems and provides facilities for testing and standardisation of mining equipment.

With a view to meet the requirement of machinery for mining projects, Coal Mining and Allied Machinery Project was set up at Durgapur in collaboration with Prommash-export of Russia and this public sector undertaking. together with a few other sizeable private sector machinery manufacturing units licensed so far, are expected to turn out adequate machinery required in the country in the immediate future.

In the manpower planning for the mining industry, several degree colleges and diploma polytechnics were started to impart education in Mining Engineering in a bid to supply adequate strength of supervisory and executive technical personnel.

For giving them the necessary practical in-plant experience, the Directorate of Practical Training in Mining was set up in Dhanbad in 1961.

PERIOD OF THE THIRD FIVE YEAR PLAN (1961-66).

The target of production at the end of the Third Five Year Plan period (1961-1966) was fixed at 97 million tons. To enable the private sector to attain their allotted share of production, a loan agreement was signed between the Government of India and the International Bank for Reconstruction and Development in August 1961 for a loan of Rs. 166.7 million to cover purchase and import of machinery and equipment to mechanise and modernise their mines. Against this, the private sector made an investment of Rs. 137.8 million by 30th September 1965, the target date for utilising the loan facilities.

Under the impact of various schemes, the production of coal caught up with the actual demand and soon exceeded it. The actual demand proved to be far less than originally envisaged. The chief causes are attributed to large-scale dieselisation of railways which were the main consumers of coal, change from coal to oil by certain cement and textile consumers, delay in setting up thermal power stations and in expansions of steel plants, and general set-back in industrial expansion for various reasons. In successive appraisals the Plan target was scaled down to a final figure of 76 million tons, against which actual production in 1965-66 was roughly 70 million tons which left no major demand unsatiated. Considering the satisfactory trend of production in relation to actual demand and improvement in rail transport, Government of India decided to remove controls on distribution of grade II and grade III non-coking coals and soft coke for domestic consumption with effect from 1st July 1964.

Following the stagnated domestic demand and in view of the surplus capacity recently created in the coal industry, the Govt. of India turned its attention to exploration of foreign market in coal and appointed in 1965 a Study Group under the chairman-

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ship of Dr. P. S. Loknathan to recommend an export policy for any of the grades of coal surplus to our requirement. The Group's main recommendations are :

(1) Non-coking coals in all grades should be allowed freely for exports. Export market exists mainly in selected grades and a margin of 2 million tons may be planned in the production of this grade of coal for export.

(2) Movement of coal for export should be exempted from railway rationalisation rules to ensure supply to foreign buyers.

(3) A rebate of 20% on rail freight should be made available to coal so moved.

(4) Calcutta Port Commissioner should be requested to consider reduction in shipping charges and river dues chargeable by them on coal.

Tangible result of the export promotion drive has been the issue of tax credit certificate at the rate of 10% of the sale proceeds of coal exported outside India in places other than Nepal, Bhutan and Sikkim, with effect from 1st October, 1965.

An important legislation which affected coal industry in 1965 was the enactment of payment of Bonus Act, 1965, which came into effect from 29 May 1965, whereby a workman becomes entitled to a minimum bonus of 4% of the wages earned in the year or Rs. 40/-. The workers in coal industry have also been given several wage increases and the controlled prices of coal were increased from time to time to neutralise costs due to these increases. The royalty payable to States in respect of coal was also increased to 5% of pithead price of coal and corresponding increase granted in the controlled price of coal.

On the labour front, the Government of India set up in 1963 National Council for Safety in Mines in the Ministry of Labour & Employment with the main objective of propagating safety ideas and promoting safety education in the mining industry in India.

On 1st March 1965, the Directorate General of Employment and Training established two Mines Mechanisation Institutes—one at Kurasia (Madhya Pradesh) and another at Bhurkunda (Bihar)—for training maintenance personnel of special mining machinery and equipment (Electricians, mine mechanics, Diesel mechanics, etc.) and entrusted them to the executive charge of the National Coal Development Corporation. More such institutions will be set up, if the pilot scheme is found successful. Under the Mines Act, Mines Vocational Rules, 1966 have been enforced with effect from 1st August, 1966 in Bengal and Bihar requiring mine managements to arrange a course of vocation training for new entrants and refresher courses for certain categories of mine employees.

Table 1 depicts the trend of the production of coal and lignite since 1900, chiefly based on the reports of the Chief Inspector of Mines.

TABLE 1

Output of coal and lignite in India from 1900 to 1965.

Year	Output Mt	Year	Output Mt
1900	6.1	1950	32.3
1905	7.5	1955	38.2
1910	16.3	1960	52.2
1920	17.0	1961	55.7
1925	20.0	1962	61.4
1930	22.7	1963	66.8
1935	21.0	1964	64.0
1940	26.1	1965	69.5
1945	26.3	1966	70 (Estimated)

An output of 67.2 Mt of coal (which includes 17 Mt of coking type) and 2.3 Mt of lignite with an aggregate pithead value of Rs. 1755 million raised during 1965 constitutes 75%

of the total value of mineral production (excluding petroleum, atomic mineral and minor minerals) in India excluding Goa. In this coal production, Bihar's share is 46%, West Bengal's 30%, Madhya Pradesh's 14% and Andhra Pradesh's 6%.

Future prospects

Further expansion of the coal industry will depend on and be influenced by the change in pattern of demand. Railways would progressively reduce their steam power and consequently, off-take of coal ; thermal power stations in Bihar-Bengal region would operate on middlings released from the washeries being set up to cater to the demand of washed coking coal for steel plants thus making a very marginal claim on additional off-take. The expansions in the steel industry would perhaps impose an additional demand for 20 Mt/year of coking and blendable coal and the industrial growth and domestic consumption may require additional 15-20 Mt/year of non-coking coal by 1970-71. An interesting feature of attempts at diversification is a bid to harness chemical potentialities of coal and this finds expression in the Feasibility Project Report of Ramgarh Complex, which includes a mining project, washeries, coke ovens and chemical bye-product plants.

3. MINING CONDITIONS

Majority of the coal seams in India are thick, presenting difficulties in recovery except when they can be economically quarried out. The share of coal from seams thicker than 4.5 m in the total reserves of all types above a thickness of 0.5 m, to a depth of 600 m, is 57% in the case of Jharia coalfield, 56% in Ranigunj coalfield and 93% in East Bokaro coalfield. Quarriable reserves in these three fields in the limiting overburden ratio of 5 : 1 are estimated at 1,187 million tons.

Coal measure strata in India consist mainly of sandstone, shale and coal with an alluvial cap at the surface and rarely has fireclay as overclay or underclay. In the Jharia and Ranigunj coalfields the proportion of sandstone in the coal measure strata

varies from 62 to 89, of shale from 4 to 34 and of coal from 4 to 31.[4] This may be contrasted with the percentages in the North of England given by Redmayne,[5] *viz.* sandstone 66.6%, shale 36.3% and coal 3.1%. Raniganj Measures sandstones comprise 50-61% of quartz, 28-43% of felspar & rock fragments, and 11-16.5% of matrix, while Barakar sandstones give an average composition of 60 to 65% of quartz, 13 to 19% of felspar and rock fragments and 20 to 23% of matrix. Sandstones in Raniganj Measures have been known to have ultimate crushing strength (uniaxial) of 598 to 914 kg/cm^2 across the bedding plane and 436 to 500 kg/cm^2 along the bedding plane.[4] Shale in the same area gave a crushing strength of 429 kg/cm^2 at right angles to the bedding plane.

Indian coals consist of bands of thickness from a few centimetres to several metres, thicker seams containing more numerous bands. Majority of coal is rich in Durain (upto 60-70% of the whole) and may contain 3 to 4% of Fusain, bright constituents making up the rest. Cleats are not well-defined and may occur at intervals of 50 mm to 500 mm. Impact Strength Index[6] lies between 66 and 84, mean ultimate strength across bedding planes varies between 140 and 330 kg/cm^2 and friability[7], between 10.6 and 37.91%. Hardgrove grindability[3] index of most coals is round 55 to 65, with a few variations upto 45 and 75.

The thickness of seams varies from 0.3 m to over 45 m and recently a thick seam of over 131 m has been located in Singrauli coalfield of Madhya Pradesh.

General gradient in Jharia coalfield varies from 5° to 15°, with a maximum of 60° near southern fault and 40-45° in the Sudamdih-Chasnala area. In the Raniganj coalfield, prevailing gradient is within a range of 2° and 35°, with a maximum gradient of 45° near southern boundary fault. In the East Bokaro Coalfield, the general gradient is within a range of 5° and 25°, with exceptional grades of 0° and 60° at a few places. The seams of Makum coalfield in Assam, of syncline-anticlinal structure, have gradients varying from 25° to 35° in northern limb and steeper inclinations (almost to vertical in Tipong and Nam-

dang valley) in the southern limb. In other areas in India, only a very gentle dip is encountered.

The average depth of working is perhaps 200 m, the deepest workings being at 700 m depth in Chinakuri mine in the Raniganj field. Several workable seams elsewhere have, however, been located at greater depths and projects are under way to mine coal at depths of 600 to 1,000 m from surface.

Water is invariably met in all mines, some of the mines are known to pump up as much as 80 m^3 of water per ton of coal raised.

Gas bearance of Indian mines is small compared to the mines in Europe and U. S. S. R. Only 152 mines were declared gassy in 1965 and a very few mines have a gas content of over 15 m^3 per tonne of coal raised. No attempts are made so far for gas drainage and its utilisation.

4. SIZE OF MINES AND PATTERN OF MECHANISATION

In comparison to other advanced countries of the world, average mine unit in India is very small ; however, the mines projected in the future would be fashioned to take advantages of concentration in space and time. Thus some new deep mine projects of the N. C. D. C. would be planned to produce over 2.2 million tons per year. There are schemes of amalgamation and reconstruction of existing mines in the private sector to form better profitable units producing over 1.6 million tons/year. Table 2 gives average size of our mines for the past few years.

Out of 819 total mines operating in December, 1965, 88 produced over 20,000 tonnes per month, 120 produced between 10,000 and 20,000 t p.m., 116 mines produced between 5,001 and 10,000 t p.m., 94 between 2,501 and 5,000 t p.m., 143 between 1,001 and 2,500 t p.m. and 258 lower than 1,000 t p.m.

TABLE 2

Average Size of Mines in India.

Year	No. of Mines.	Average No. of persons employed per mine.	Average annual output per miner 1,000 t	Output per man/year. t
1950 ..	891	393	36	94
1955 ..	853	403	45	112
1960 ..	848	469	62	131
1961 ..	848	485	66	135
1962 ..	851	509	72	142
1963 ..	879	513	76	148
1964 ..	860	501	74	149
1965 ..	841	505	83	164

Table 3 gives an idea of employment in our mines.[8] About 8% of the total labour force consist of women who work on the surface during restricted hours. The average weekly earning of a worker in November, 1965 was Rs. 31.18, absenteism being about 13.06%. Output per manshift in December, 1965 was 1·40 for miner-and-loader and 0.57 for all persons.

TABLE 3

Employment and output of coal mines in India

	1961	1962	1963	1964	1965
1. Number of coal mines operating	848	851	879	860	841
2. Number of mines with underground workings	657	654	684	681	602
3. Number of mines declared "gassy" ..	111	119	126	131	152
4. Total employment in all mines	411,265	432,909	450,679	430,757	424,509
5. Total employment below ground ..	229,547	244,977	261,000	253,300	252,319

	1961	1962	1963	1964	1965
6. Employment below ground in 'gassy' mines, 1,000 men ..	81.4	85.1	87.5	85.4	93.5
7. Employment in opencast workings, 1,000 men ..	60.5	63.9	63.2	52.8	47.3
8. Output from all mines, Mt	55.71	61.37	66.76	63.98	69.51
9. Output from gassy mines, Mt ..	16.0	17.5	18.5	18.06	21.48
10. Output from opencast workings, Mt ..	10.82	11.35	13.4	11.7	12.97
11. Output per man year in all mines, t ..	135	142	148	149	164
12. Output per man year in opencast, t ..	179	178	211	222	270
13. Output from gassy mines as per-cent of the total output from under-ground workings	34.82	34.97	34.67	34.55	38.0
14. Output from opencast as per cent of total output from all mines ..	19.42	18.50	20.07	18.29	18.66
15. Fatal injuries ..	268	266	267	182	469
16. Serious injuries ..	3,605	3,207	2,533	2,080	2,015
17. Death rate per 1,000 workers employed	0.66	0.61	0.59	0.42	1.10
18. Death rate per Mt of output	4.81	4.33	4.00	2.84	6.70
19. Serious injuries per 1,000 employed ..	7.79	7.41	5.61	4.83	4.73
20. Serious injuries per Mt of output	64.71	52.25	37.97	32.51	28.90

The pace of mechanisation has been halting and slow but it is now realised that new projects will have to adopt mechanisation to improve output per manshift, to rationalise the industry and to improve working conditions. The present use of electricity, coal cut by coalcutting machines, loaded by machines,

transported by conveyors and the use of explosives are indicated in Table 4 which highlights the scope for more widespread application of cutting, loading and transporting machinery in our mines.

TABLE 4

	1961	1962	1963	1964	1965
Output of coal to a kg of explosive used in mines ..	8.41	7.56	7.93	6.36	7.06
Installed H.P. of electrical machinery in 1,000 H.P.	438	441	509	522	590
Output of electrified mines as percentage of total output	94	96	92	98	98.5
Coal cut by machines as % of u.g. output ..	33.36	34.31	36.18	36.04	39.04
Coal power-loaded u.g. as % of u.g. output ..	2.04	1.46	1.24	0.93	2.23
Coal transported by conveyors as % of u.g. output	9.42	9.79	11.34	11.82	13.70

The bord-and-pillar system dominates the mining methods in India, owing to its relatively inexpensive initial capital involvement and ease of adoption in the field abounding in unskilled labour. The popular mechanisation in bord-and-pillar makes use of mobile coal-cutters, gathering-arm loaders and shuttle cars ; only two mines at the moment have taken to continuous mining in conjunction with shuttle cars.

However, in the recent days advantages of longwall mining, particularly in the realm of concentration of work, efficient ventilation, better recovery and possible elimination of dangers from bumps at depth, have been realised and about 20 longwall faces were in operation in 1965, contributing just over 1% of the total production. The longwall practice in India may be broadly classified in the following categories :

(1) Mechanised longwall with cutter-loaders, armoured chain conveyors and yielding props.

(2) Conventional longwall with cutting by longwall coal-cutter, blasting and manual loading on the face conveyor.

(3 Barry face or modified longwall on previously developed bord-and-pillar workings.

No satisfactory method has yet been evolved for working thick seams for satisfactory recovery, though sandstowing of the goaf is widespread.

5. SPECIFICATIONS, COMMERCIAL GRADES AND MARKETS OF COAL

Specifications for coal mentioned in contracts between buyers and sellers usually consist of (a) proximate analysis, (b) sulphur content, (c) calorific value, (d) coking properties and (e) size. These are standardised in Indian Standards Specifications IS : 1350-1959, 1353-1959 and 437-1965. Analysis of coal expressed in terms of percentages by weight of moisture, volatile matter, ash and fixed carbon is known as proximate analysis. The results of proximate analysis are generally recorded as percentage of the air-dried material ; however it is sometimes necessary to know the nature of the coal substance, that is, the pure combustible or organic part of coal. Coal can be treated and beneficiated to reduce its impurities of moisture and mineral matter. In order to predict behaviour of such treated coal, volatile matter and fixed carbon may be computed on the basis of *unit Coal* or on *dry ash-free* or *dry mineral-free* sample (that is, they are expressed as percentage by weight of the sample of coal less moisture and mineral matter).

Moisture content of air-dried coal samples varies depending on the size of coal, temperature and relative humidity of air in which it is dried ; and for purposes of comparison, such as in commercial grading, it is usual to specify these conditions. In commercial grading, coal sample is ground to pass IS sieve 20 and equilibrated in an atmosphere of 40°C and 60% relative humidity. One method of determining moisture consists in spreading a carefully weighed 10 g of air-dried material of

—20 IS Sieve on a 10 cm diameter aluminium dish and heating it in an oven at a temperature of 105°C to 110°C until there is no further loss in weight. This may take 1.5 to 3 hours. Loss of weight gives moisture content and it is calculated as per cent of the air-dried sample.

Volatile matter is determined by heating out of contact with air a weighted quantity of air-dried sample of coal at a temperature of 900°±15°C for a period of 7 minutes in a vessel of specified dimensions. Loss of weight as percentage of the weight of the material gives the percentage of moisture and volatile matter and after deducting moisture percentage in the sample, determined separately, v.m. percentage is obtained. For coals the tolerance limit is 1% in the same laboratory.

Ash is the inorganic residue left when the powdered laboratory sample of coal or coke has been incinerated in an open dish until it no longer changes in weight, at 815±10°C. Mineral matter in the coal from which ash is derived, is slightly greater than the ash determined above, largely because of loss, during incineration, of the water of hydration and CO_2 of carbonates. In Indian coals, percentage of mineral matter is very nearly 1.1 × percentage of ash.

Percentage of fixed carbon in the sample is obtained by subtracting from 100, the sum of the percentages of moisture, volatile matter and ash. It may be noted that the fixed carbon as determined above is generally (particularly in bituminous and lignite coals) far less than the total carbon content of the coal, as some carbon is lost in volatile matter which contains carbon in combination as hydrocarbon, phenols, oxides of carbon, etc.

Calorific value of coal is the heat units evolved by the complete combustion of a unit weight ; in coal technology, it is usual to determine gross calorific value at constant volume, which is the number of heat units liberated when unit weight of coal is burnt at constant volume in oxygen saturated with water vapour, the original materials and final products being at 25°C. This assumes that all the heat produced is available, including heat of condensation of steam to water at room temperature ; but boiler efficiencies are often expressed on the *net*

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calorific value, which assumes that heat of condensation of steam below 100°C is not available. To convert gross calorific value to net calorific value, the formula :

$N_c = G_c - 53H$ is used, where N_c = net calorific value, Kcal/kg
G_c = gross calorific value, Kcal/kg
H = % of Hydrogen in coal.

Coking Coal

Coal, which on heating in the absence of air, between 300°C and 400°C, becomes plastic, or even fluid, and on further heating, leaves a porous solid residue, called coke, is known as *caking* coal. The term *caking* is usually employed to describe this general phenomenon and the term *coking* is reserved for such coals as yield a strong hard coke suitable for blast furnace use. If the coal *by itself* is capable of producing coke which is sound, strong and suitable for metallurgical purposes, it is said to have good coking power and is known as *prime coking* coal or simply, *coking coal*. Coal, which on carbonisation yields coke that falls short of the specifications of metallurgical coke, is known as *medium coking* coal. Semi-coking coal is one which gives a coherent coke but weaker than the one given by medium coking coal[10]. Only some of the bituminous coals possess caking properties. Anthracites, when heated under the conditions specified above, yield a powder not differing much in appearance from the original (powdered) anthracite.

The quality of coke produced from a coal depends on a number of factors, *viz.* rank, petrographic composition, mineral matter content, rate of heating in the ovens, preparation of the coal charge to the required degree of fineness, oven width, etc. Laboratory methods used to measure caking properties of a coal are Caking Index Test (Agglutinating Index), Crucible Swelling Index Test, Gray-King Low-temperature Assay, Plastometric Test, etc. However it has not been found possible to assess the actual quality of coke produced from Indian coal on the basis of any of these tests alone.

For general use, caking property of a coal is assessed by a study of the type of the residue obtained in Gray King Carbonisation Assay at 600°C. 20 grams of a laboratory sample previously dried at 108° ± 2°C is taken in a silica tube and is heated at a specified rate starting from 300°C and terminating at 600°C. The residue is compared with a photograph of profiles (Fig. 1) Types A to G indicate the degree of caking property of coal. Type A represents completely non-caking coal, type G, strongly coking which does not swell on heating and type G_1, G_2 etc. represent coking coals which swell on heating, the subscript number being the number of parts of electrode carbon required to be blended to give a standard G coke. The coke type is affected if the ash percentage in coal exceeds 17-18%; therefore, for determining the coke type by Gray-King Low Temperature Assay, the sample should be beneficiated to the ash level of 17%.

Agglutinating Index (or Caking Index) Test

The Caking Index or Agglutinating Index is the figure which represents the maximum ratio of sand to coal in a mixture which, after carbonisation, gives a coherent mass capable of supporting a 500 g weight. It is essential that the loose powder produced by the weight should not exceed 5% of the weight of sand and coal. The Index obtained gives only a relative idea of the coking property of a coal. The sand used should be of uniform quality with regard to size, purity, sharpness, etc.

For this test the weight of coal and sand as prescribed by the British Standard Institution is 25 g. In India the same method adopted by Tata Iron & Steel Company uses only 5 g of mixture of sand and coal instead of the 25 g. The result obtained by B. S. I. procedure and TISCO procedure for the same coal has approximately the following relation :

B. S. I. Caking Index = TISCO Caking Index × 1.5

Coal which gives Caking Index of 15 and above (TISCO procedure) is suitable for making metallurgical coke.

The specifications of metallurgical (hard) coke as specified in ISI standard IS : 439-1965 are given in Table 5.

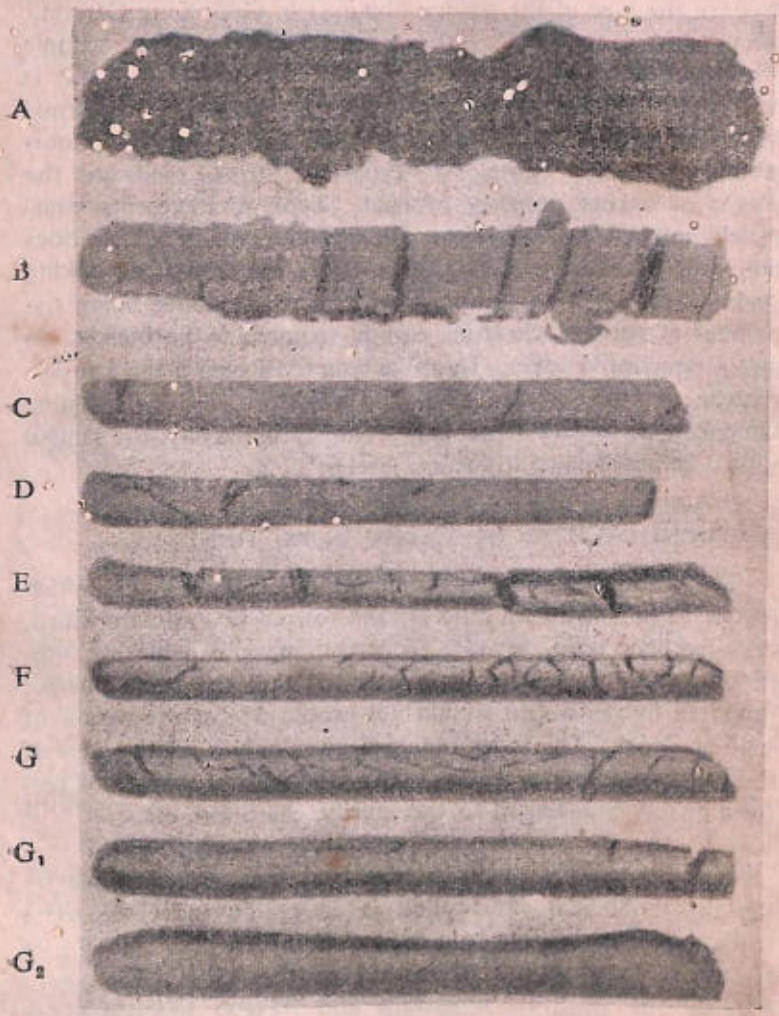

Fig. 1. Types of coke from GRAY KING assay.

TABLE 5

Specifications of Metallurgical Coke

	Grade 1	Grade 2	Grade 3
(i) Proximate analysis on dry basis :			
(a) Ash content, percent by weight, Max.	20	24	28
(b) Volatile matter, per cent by weight, Max.	2	2	2
(c) Fixed carbon (by difference) ..	to be reported		
(ii) Sulphur, on dry basis, per cent by weight, Max.	0.70	0.70	0.70
(iii) Phosphorus, on dry basis, percent by weight, Max.	0.30	0.30	0.30
(iv) Shatter index :			
(a) Over 38 mm, percent, Min. ..	85	85	..
(b) Over 13 mm, percent, Min. ..	97	97	..
(v) Micum Index :			
(a) Total on 40 mm, per cent by weight, Min.	75	75	..
(b) Through 10 mm, percent by weight, Max.	14	14	..
(vi) Haven stability factor, total on 25 mm, percent by weight, Min. ..	40	40	..
(vii) Porosity, percent	35 to 48	35 to 48	

Note—Unless otherwise agreed to, the moisture content of the coke shall not be more than 4% and size not less than 25 mm (ex coke plant).

Note

Shatter test : It is designed to simulate the effect of dropping coke on loading, dropping into bunkers and to the charge level of a blast furnace. It measures, therefore, the resistance to shattering, of the impact hardness, as distinct from the surface hardness which is measured in abrasion test. The shatter test treats a representative sample of coke, over 2" (5 cm) in size, in a standard dropping test, the resistance to breakage being measured by the percentage of the coke which remains on different sieve plates after the test.

Micum test : It is a relative measure of the resistance of coke to degradation by abrasion.

Haven test : The abrasion index of the coke may be determined by the Haven test which is also a relative measure of the resistance of the coke to degradation by abrasion. The results obtained are influenced by the effect of impact.

Prime coking coal reserves are limited, present reserves, mainly confined to seams No. X to XVIII of the Jharia and upper and lower Karharbari seams of the Giridih coalfields, being assessed at about 4,608 million tonnes.[3] The problems of their production are their localisation in restricted areas, occurrence at depth involving high capital and operating costs for development and exploitation, and their high ash content with high degree of dispersion, making washing indispensable.

Our low reserves of prime coking coal can be made to last longer for the use of iron and steel industry if coke ovens were charged with blends of primary coking coal with medium-coking or semi-coking variety; such blends yield cokes of metallurgical quality. Central Fuel Research Institute found that a ternary blend of coal consisting of 40-50% prime coking coal, 30-45% of medium-coking coal and 10-15% semi-coking

coal can make coke suitable for metallurgical purposes, by correct methods of preparation of blends, sizing of coal charge, bulk density control and right carbonisation conditions.[10]

Commercial Grades

Certain plants require coal of specific sizes to operate at high efficiency and the following size grading of coal for marketing (Table 6) has been standardised in ISI Standard IS : 437 : 1965 (second revision).

TABLE 6

Size Grading of Coal for Marketing

	Nomenclature.	Trade name.	Nominal size.	Screening at the colliery (Range of Round Hole Diameter) Upper limit	Lower limit
			mm	mm	mm
(i)	Coal, large	(Steam)	200 to 50	250	44
(ii)	Coal, Medium	(Rubble)	50 " 25	75	19
(iii)	Coal, Small	(Smithy)	25 " 13	32	10
(iv)	Coal, Slack (100)	..	100 " 0	150	..
(v)	Coal, Slack (50)	..	50 " 0	75	..
(vi)	Coal, Slack (25)	..	25 " 0	32	..
(vii)	Coal, Slack (13)	..	13 " 0	16	..
(viii)	Run-of-Mine	R.O.M.	Unscreened coal of all sizes.		

A tolerance of 5% shall be allowed in the results of sieve analysis.

For marketing and pricing, bituminous coal in Bengal and Bihar and in the outlying fields is graded by the Coal Controller on the basis of ash content in the case of low moisture coal and on the basis of ash and moisture content in the case of high moisture coal. Low moisture coal is one with moisture not exceed-

ing 2%. If the moisture exceeds 2%, it is considered as high moisture coal. The present grades are as presented in Table 7.

TABLE 7

Grades of Coal

The following grades are in force from 13th June, 1962,

For grading coal analysis is carried out on —72 mesh samples after equilibrating under the conditions given below for 48 hours.

Atmospheric temperature = 40° ± 2°C.
Relative humidity = 60% ± 2%

A. For coals from Bengal and Bihar coalfields.

	Non-coking		Coking	
Grade	High moisture coal Ash & moisture content%	Low Moisture coal Ash content%	Grade	Ash content%
Selected A	< 17.5	<15	A	< 13
Selected B	17.5-19	15-17	B	13-14
Grade I	19-24	17-20	C	14-15
Grade II	24-28	20-24	D	15-16
Grade IIIA	..	24-28	E	16-17
Grade IIIB	..	28-35	F	17-18
			G	18-19
			H	19-20
			HH	20-24

B. For coal from outlying fields (other than fields in Andhra Pradesh and Assam).

Grade	Ash & moisture content within limits of%
Selected Grade	<19
Grade I	19-24
Grade II	24-28
Grade III	28-35

C. Coals in Andhra Pradesh & Assam are not graded but prices are statutorily fixed on the size of coal in Andhra Pradesh and from colliery to colliery in Assam.

Present commercial grading of coals of the seams of the Barakar and Raniganj stages on the basis of ash or ash and moisture leaves much to be desired. Study group appointed by the Coal Board to draw up a concrete scheme of fixation of prices, therefore, considered that grading should be based on useful heat units in a coal. Useful heat units in a coal are given by the formula :

$$H_u = \frac{(100-1.5A_d)\times B_u}{100}$$

where, H_u —useful units,

A_d —ash percentage in dry coal=

$$\frac{\text{Ash \% found in sample}}{(100-M)} \times 100$$

B_u —Calorific value of pure dry coal *(unit coal)*

$$= \frac{\textit{Calorific value of the sample}}{100-(M+1.1A)} \times 100$$

M —Percentage of moisture found in the sample,

A —Percentage of ash in the sample.

The grading on this basis would apply to both high and low moisture coking and non-coking coal, but the indications are that this method of grading will not be adopted in the near future.

Markets

Despatches of coal and lignite of all fields in India totalled 62.54 M tonnes in 1965. Railways continuned to be the chief consumers accounting for 29% of the despatches. Next in order of consumption were iron and steel industry (19%), power generating units (11%), cement (5%), textiles (3%), etc.

The norms for consumption of coal in some leading industries are as follows :

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The norms for consumption of coal in some leading industries are as follows :

(i) Iron & Steel	per tonne of pig iron produced 1 tonne of hard coke equivalent to 1.3 tonnes of washed coking coal (17% ash) or two tonnes of raw coking coal.
(ii) Power plants	Inferior coal 3 tonnes per annum per kw of installed capacity.
(iii) Cement manufacture	1 tonne of coal for 3 tonnes of cement.
(iv) Paper	2.25 tonnes of coal per tonne of paper and 1.3 tonnes of coal per tonne of pulp (in new plants).
(v) Textiles	0.38 tonne of coal per 1,000 m of cloth.

Distribution of coking coal and non-coking coal of grade I and superior quality to consumers is controlled by Coal Controller; grade higher than that fixed by him is not allowed to be used by the consumer concerned though a lower grade can be used. Gr. II and inferior coals are not controlled but their ceiling prices are fixed by the Government. Consumers are classified in priority classes on the basis of the broad division of industry to which they belong and these classes are further split up on the basis of the use to which coal is ultimately put. For example, in priority class E of Inland Transport, Railways are allotted Selected A non-coking steam coal for mail and passenger services, Selected B non-coking steam for passenger and through goods services, and Grade I non-coking steam or Grade II coking steam for other services, while steamer services are allotted Grade I non-coking steam. In allotting coal for generation of power at power stations or continuous processes in other industries, the general principle followed is that hand-fired boilers get selected B non-coking steam; chain-grate stokers with natural draught are given Selected B non-coking rubble or slack ; chain-grate stokers with mechanical draught are entitled to Grade I slack with surplus slack from Selected A & B grades ; spreader stokers or sprinkler stokers can use

Grade II non-coking slack ; and pulverised coal-fired boilers are expected to burn Grade IIIA, Grade IIIB slack or washery middlings.

6. COAL WASHERIES

A coal washery may be of 2 product type, yielding washed coal and rejects, or of 3 product type, yielding washed coal, middlings, and rejects. The yield of washed coal in the 3 product washeries at present in operation, varies from 65% to 75% of the input. In the Kargali washery, which is of 3 product design, washed coal yield is nearly 70% of the raw coal.

Coal is treated in a coal washery to reduce its ash content. Such treatment is particularly important for the iron and steel industry as each 1% increase in ash in the coal increases consumption of such coal in the blast furnace by 7 to 8%. With the exception of Nowrozabad coal washery set up in 1959-60 by A. C. C. (rated capacity 122 tonnes per hour, raw coal), all the washeries in India treat coking coals for the ultimate manufacture of hard coke required for blast furnaces and foundries. Table 8 gives the present position of washeries.

TABLE 8

Annual installed capacities of washeries for coking and blendable caking coal

	Washery Project	Type of Washer & raw coal capacity t.p.h.	Raw coal input, Mt/year	Expected washed output, Mt/year	Remarks
1.	West Bokaro (TISCO)	Unit washery ; Chance ; 137	0.70	0.50	In operation ; treats medium coking coal.
2.	Jamadoba (TISCO)	Unit washery ; Chance ; 305	1.44	1.00	In operation ; treats prime coking coal.
3.	Lodna (Turner Morrison)	Unit washery ; Feldspar jig ; 70	0.40	0.23	In operation ; treats prime coking coal.
4.	Dugda I (H.S.L.)	Central washery ; Baum jig prewasher, McNally Tromp heavy medium bath ; 600	2.40	1.32	In operation ; treats prime coking coal.

(i) Iron & Steel	per tonne of pig iron produced 1 tonne of hard coke equivalent to 1.3 tonnes of washed coking coal (17% ash) or two tonnes of raw coking coal.
(ii) Power plants	Inferior coal 3 tonnes per annum per kw of installed capacity.
(iii) Cement manufacture	1 tonne of coal for 3 tonnes of cement.
(iv) Paper	2.25 tonnes of coal per tonne of paper and 1.3 tonnes of coal per tonne of pulp (in new plants).
(v) Textiles	0.38 tonne of coal per 1,000 m of cloth.

Distribution of coking coal and non-coking coal of grade I and superior quality to consumers is controlled by Coal Controller; grade higher than that fixed by him is not allowed to be used by the consumer concerned though a lower grade can be used. Gr. II and inferior coals are not controlled but their ceiling prices are fixed by the Government. Consumers are classified in priority classes on the basis of the broad division of industry to which they belong and these classes are further split up on the basis of the use to which coal is ultimately put. For example, in priority class E of Inland Transport, Railways are allotted Selected A non-coking steam coal for mail and passenger services, Selected B non-coking steam for passenger and through goods services, and Grade I non-coking steam or Grade II coking steam for other services, while steamer services are allotted Grade I non-coking steam. In allotting coal for generation of power at power stations or continuous processes in other industries, the general principle followed is that hand-fired boilers get selected B non-coking steam; chain-grate stokers with natural draught are given Selected B non-coking rubble or slack ; chain-grate stokers with mechanical draught are entitled to Grade I slack with surplus slack from Selected A & B grades ; spreader stokers or sprinkler stokers can use

Grade II non-coking slack ; and pulverised coal-fired boilers are expected to burn Grade IIIA, Grade IIIB slack or washery middlings.

6. COAL WASHERIES

A coal washery may be of 2 product type, yielding washed coal and rejects, or of 3 product type, yielding washed coal, middlings, and rejects. The yield of washed coal in the 3 product washeries at present in operation, varies from 65% to 75% of the input. In the Kargali washery, which is of 3 product design, washed coal yield is nearly 70% of the raw coal.

Coal is treated in a coal washery to reduce its ash content. Such treatment is particularly important for the iron and steel industry as each 1% increase in ash in the coal increases consumption of such coal in the blast furnace by 7 to 8%. With the exception of Nowrozabad coal washery set up in 1959-60 by A. C. C. (rated capacity 122 tonnes per hour, raw coal), all the washeries in India treat coking coals for the ultimate manufacture of hard coke required for blast furnaces and foundries. Table 8 gives the present position of washeries.

TABLE 8

Annual installed capacities of washeries for coking and blendable caking coal

	Washery Project	Type of Washer & raw coal capacity t.p.h.	Raw coal input, Mt/year	Expected washed output, Mt/year	Remarks
1.	West Bokaro (TISCO)	Unit washery ; Chance ; 137	0.70	0.50	In operation ; treats medium coking coal.
2.	Jamadoba (TISCO)	Unit washery ; Chance ; 305	1.44	1.00	In operation ; treats prime coking coal.
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4.	Dugda I (H.S.L.)	Central washery ; Baum jig prewasher, McNally Tromp heavy medium bath ; 600	2.40	1.32	In operation ; treats prime coking coal.

	Washery Project	Type of Washer and raw coal capacity t.p.h.	Raw coal input, Mt/year	Expected washed output Mt/year.	Remarks
5.	Dugda II (H.S.L.)	Central washery ; heavy medium cyclone ; 600	2.40	1.32	Under construction.
6.	Bhojudih (expanded) (H.S.L.)	Central washery ; Baum jig, Heavy medium & cyclone rewasher ; 500	2.00	1.40	To be ready by April 1967, for prime coking coal.
7.	Patherdih (H.S.L.)	Central washery, Baum jig prewasher, heavy medium (D. S. M.) bath, cyclone ; 500	2.00	1.30	In operation.
8.	Durgapur (H.S.L.)	Central washery ; Drewbouy bath and Feldspar jig ; 350	1.50	0.90	In operation; for prime coking coal.
9.	Kargali (expanded) (N.C.D.C.)	Composite washery ; WEMCO baths & Baum jigs ; expansion in cyclones ; 640	2.72	1.90	In operation; treats medium coking coal.
10.	Kathara (N.C.D.C.)	Unit washery, heavy medium (H.M.S.) prewasher ; heavy medium cyclone.	3.00	1.50	Under construction, to treat medium coking coal.
11.	Gidi (N.C.D.C.)	Heavy medium (Disa) baths and Baum jigs.	2.84	2.20	Under construction, to treat medium coking coal.
12.	Sawang (N.C.D.C.)	Heavy medium cyclone for +0.5 *mm* and autogeneous cyclone for −0.5 *mm*	0.75	0.50	Under construction.
13.	Durgapur West Bengal (Govt.)	Heavy medium ; 300	1.35	0.67	Under construction, to treat prime and medium coking coal.
			23.60	14.74	

REFERENCES

1. GUHA, A. B. "Coal as a major source of energy in India," *Coal & Steel, January*, 1966.

2. GEE, E. R. "Bulletin of economic minerals—Coal", *"Records of Geological Survey of India*, 1945".

3. MAJUMDAR, S. AND GUPTA, S., "Resources and qualities of Coals in India", *Journal of Mines, Metals and Fuels, Special issue*, 1964.

4. SINGH, R. D. AND AGARWAL, Y. K., "Characteristics of coal Measure strata with particular reference to their effect on mining operation" *Metals and Minerals Review*, April, 1966.

5. REDMAYNE "Modern Practice in Mining" Vol. III.

6. GHOSE, A. K., BARAT, D. AND BAGCHI, S., "Some preliminary studies on the impact strength of Indian coals", *Journal of Mines, Metals and Fuels, May*, 1964.

7. SINGH, R. D. "Comprehensive strength of some Indian coals" *Trans., Mining, Geological and Metallurgical Institute of India*, 1962.

8. CHIEF INSPECTOR OF MINES, Annual reports of 1961, 1962, 1963 and 1964.

9. ENGINEER, B. H. "Coal for Steel", Presidential address, M.M.G.I. March, 1966.

10. BANERJEE, N. G., SARKAR, G. G., AND LAHIRI, A.— "Requirement of coking coal at present and in future". *India in Industries, Vol. 7 No. 2.*

11. COAL CONTROLLER & CHAIRMAN COAL BOARD, "Monthly review and coal production and distribution"—June, 1966.

CHAPTER 2

INDIAN IRON ORE INDUSTRY

1. Introduction. 2. Basic concepts of metallury of iron and steel. 3. Requirement of iron ore in 1970-71. 4. Mining and other legislation. 5. Grades and prices of iron ore and the export market.

1. INTRODUCTION

Iron and steel industry is of basic importance to the economy of any nation aspiring for industrial development. The last decade saw a planned growth of the iron and steel industry in this country under the impetus of the Five Year Plans. The Fourth and the subsequent plans aim at developing the iron and steel industry not only to meet the internal demand but also to export the finished products to other developing countries.

Iron is an element which seldom occurs in a pure form in its natural state ; it is rarely found as a native metal in the form of meteoric iron. Iron in its various compounds, mixed with compounds of other metals, forms the earth's crust in such abundance that it is second only to aluminium.

The principal minerals of iron and their uses will be described, later in Chapter IV of this book. Suffice it to state here that hematite is the ore most abundant in India and is used for the manufacture of iron and steel. In coal washeries employing principles of heavy media separation, *e.g.* Kargali Coal washery, magnetite is used to form the heavy medium. Micaceous hematite is used in paint manufacture and coating for electric welding rods. Mineral paints industry uses impure varieties of hematite (red ochre) and limonite (yellow ochre). Pyrites, large deposits of which have been found in Amjhor (Bihar), will be used for the manufacture of sulphuric acid.

The *Rig Veda* and other Vedas, written several centuries B.C. have references to iron indicating that the metal was known to

Indians before the Vedic times. The Delhi iron pillar, standing near the Kutab Minar in Delhi, is an outstanding example of the skill in iron and steel making in earlier days. Constructed sometime in the Fourth century A.D., this pillar has properties to resist rusting. The inscriptions on it refer to the victories of Chandragupta II. The total length of the pillar is 7.3 m and the weight over 6 tons. This rust-free pillar, constructed about 1,500 years ago when heavy foundry practice was unknown is a remarkable achievement of Indian iron makers whose knowledge of metallurgy is to be marvelled at.

Sir Robert Hadfield obtained a small piece of iron from this pillar and had it analysed with the following result :

	Per cent (by Wt.)
Carbon	0.080
Silicon	0.046
Sulphur	0.006
Phosphorus	0.114
Nitrogen	0.032
Manganese	Nil
Copper	0.034
Iron	99.72
Total ..	100.032
Sp. Gr. ..	7.81

The pillar is evidently of pure iron.[1]

The development of the iron ore industry is linked with that of the iron and steel manufacturing industry ; the latter did not make any technological and quantitative progress till the fag end of the 19th century. In 1778, Messrs. Motte and Farquhar had started smelting operations in Birbhum district, but they were soon abandoned. In 1874 Bengal Iron Works Company was formed. It had two blast furnaces at Kulti in 1878. The State Government acquired the Company in 1882 and it was then known as Barakar Iron Works.

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In 1889 the Government transferred the undertaking to a private firm, the Bengal Iron & Steel Co. (Mg. Agents—Messrs Martin & Co.). By 1920, the output of pig iron in this company was about 500 tons per day. In 1936, this company was amalgamated with the Indian Iron & Steel Co. A company, known as Steel Corporation of Bengal (Mg. Agents—Messrs Burn & Co.) floated in 1937 for producing steel from pig iron manufactured in the works of the Indian Iron & Steel Co., merged with the latter in 1952.

The Tata Iron & Steel Co. commenced pig iron manufacture at Jamshedpur in December, 1911 when the first blast furnace was blown. The Mysore Iron and Steel Works had its first blast furnace blown in 1923, with charcoal used in place of coke.

With the erection of the blast furnaces, iron ore mines were opened in the Singhbhum-Keonjhar iron ore belt of Bihar and Orissa. These mines carried on ore extraction on a large scale by manual labour. The first mechanised mine came into operation sometime in 1958 at Noamundi (TISCO), followed by more mechanised mines, namely Joda and Gua, in 1960.

During the Second Five Year Plan, the Hindusthan Steel Ltd., a Government of India undertaking, started three steel plants at Bhilai, Rourkela and Durgapur, each with a capacity of 1 million tons of steel ingots per annum. The fourth steel plant, Bokaro Steel Plant, under the public sector was initiated in the Third Five Year Plan and the spade work is continuing in the Fourth Five Year Plan. Location of the fifth steel plant in the public sector is still undecided. As already stated, the development and expansion of the iron ore mining industry closely followed the development of the local iron and steel industry as the export was only about 25% of the iron ore production. Since 1962 the picture has, however, changed. Goa became part of India in December, 1961. The iron ore production from that territory, about 6.5 million tons per annum and forming nearly 33% of India's production, was and is exported in full, there being no local consumption. Two

large mining projects, one at Bailadila in M.P. and the other at Kiruburu in Orissa, have been opened in the public sector to export iron ore to Japan to earn oreign exchange. Kiruburu is being developed to export 2 million tons of sized ore per year and Bailadila, 6 to 8 million tons. The need for export and the expansion of the existing steel plants in the Fourth Five Year Plan would give a boost to the iron ore production. Table 1 gives a projected picture of the raw materials for the various steel plants, and Table 2, the trend of iron ore production.

In 1963, there were 269 iron ore mines in India, excluding Goa.[2] Out of these annual production of :

26	mines was	50,000 to 100,000	tonnes per mine
17	-do-	100,000 to 500,000	-do-
5	-do-	500,000	tonnes and above.

The rest of the mines had outputs smaller than 50,000 tons per year.

The Singhbhum district of Bihar and the adjoining Keonjhar and Mayurbhanj districts of Orissa contain very rich iron ores (60 to 65% Fe) in abundant quantities and are located within 250-350 km of the Jharia and Raniganj coalfields, the only sources of coking and blendable coal. Limestone and dolomite, used as fluxes, are also available within 80 to 150 km of the iron ore fields. The country is thus very fortunate in having the principal raw materials, required for iron and steel industries within close proximity and enjoys a unique position in the world in this respect. The Lake Superior deposits in America, considered to be one of the largest in the world, are situated at a distance of over 1,600 km from the iron smelting centre at Pittsburg, U. S. A. Their transport involves railways, waterways and again railways. India's envious position can thus be well appreciated. Our ore reserves can meet the domestic requirements of steel plants for a few centuries and also earn valuable foreign exchange if exported at competitive prices.

TABLE 1

The installed capacities and production of steel and sources of raw material for Steel plants by the end of 1970-1971, the last year of the Fourth Five Year Plan.

	In 1970-71	Source of main raw materials.			
Plant.	Ingot Steel million tons.	Coal (Washery).	Iron ore.	Lime stone.	Dolomite.
Bhilai	3.2	Kargali Dugda.	Rajhara, Dalli.	Nandini	Hirri.
Durgapur	3.4	Durgapur Washery.	Barajamda Barbil, Banspani, Bolani etc.	Birmitrapur Nandini Satna.	Birmitrapur, Hirri.
Rourkela	2.5	Kargali, Dugda.	Barsua, Barajamda.	Purnapani Satna.	Hirri.
Indian Iron & Steel, Burnpur.	1.3	Chasnalla Patherdih.	Gua, Manoharpur.	Birmitrapur	Birmitrapur.
Tata Iron & Steel, Jamshedpur.	2	West Bokaro, Jamadoba, Bhojudih.	Noamundi Joda, Badampahar, etc.	Hathibari	Panposh.
Mysore Iron & Steel Works, Bhadrawati.	0.1	Coke, and electric furnace used.	Kemmangundi.	Bhadigund	Shankargudda.
Bokaro	1.7 (first stage)	Kargali, Dugda.	Kiruburu, Meghataburu.	Bhavanathpur, Koteshwar.	
Neyveli Salem Project (under consideration).	0.25	Lignite coke from Neyveli.	Salem	Sankaridurg.	

TABLE 2

Production of iron ore in India and the world (in tonnes)

Year	Indian production	World production
1955 ..	4,753,000	373,888,000
1956 ..	4,976,000	399,288,000
1957 ..	5,167,000	433,832,000
1958 ..	6,130,000	405,100,000
1959 ..	7,982,000	438,300,000
1960 ..	10,683,000	520,500,000
(Goa) ..	5,856,000	
1961 ..	12,310,000	511,800,000
(Goa) ..	6,310,000	
1962 ..	13,510,000	512,100,000
(Plus Goa) ..	6,130,000	
1963 ..	14,995,000	534,416,000
(Plus Goa) ..	5,470,000	
1964 ..	14,881,000	....
(Plus Goa) ..	6,482,000	
1965 ..	16,900,000	....
(Plus Goa) ..	6,490,000	

2. BASIC CONCEPTS OF METALLURGY OF IRON

To assess the requirement of iron ore in respect of quality and quantity it is essential to have a fundamental idea of the processes where it is used in the iron and steel plants and the following digression would therefore be of interest.

Ordinary iron, as we understand it for engineering and construction uses, contains not only the element Fe, but other non-metallic materials, chiefly carbon, silicon, phosphorus and sulphur and also some metals such as manganese etc. in small quantities. Some of these are undesirable constituents, but some are added intentionally depending on the use to which iron or its products have to be put. The carbon content has a decisive influence on the properties of iron and steel.

Pig iron is the first product available from iron ore in the process of making iron or steel and contains carbon, phosphorus, silicon, sulphur and other impurities which have to be removed by further processes.

Cast iron, by definition, does not contain less than 2% carbon. It is practically pig iron cast into different shapes with addition of silicon or manganese to suit the specifications required for different castings. In the process of increasing silicon or manganese content the carbon content of pig iron goes down so that cast iron, as a general rule, contains 2 to 4% carbon, 0.3 to 3% silicon and 0.2 to 1.2% manganese. It is brittle and possesses no malleability at any temperature.

Steel is the term used for iron which is malleable at some range of temperatures and is capable of becoming considerably harder by sudden cooling. It is essentially an alloy of iron and carbon with carbon less than 1.8%. Presence of other metals like nickel, chromium, manganese, tungsten, copper, etc., in small quantities confer upon steel distinct properties which make it suitable for special types of application, as indicated in Table 3.

Wrought iron is practically pure iron, highly malleable, with a carbon percentage of 0.02 to 0.05. It is too soft and ductile for many purposes *e.g.* for making tools. If wrought iron is heated with carbon at red heat for a fairly long time, it becomes steel.

Smelting and reduction in a blast furnace is still the cheapest process for large-scale iron making and the largest blast furnace in Asia is at Bhilai with a capacity of 1,750 tons of pig iron in 24 hours though the one proposed at Bokaro will be of 2,600 tons, the largest in the world. The burden fed to a blast furnace consists of iron ore, hard coke and limestone. The hard coke performs the following functions :

(1) It generates the heat necessary to bring about the chemical reactions in the furnace and melts the iron. The impurities in iron ore, coke and limestone also melt and form slag.

(2) It supplies the carbon required to reduce the ore.

TABLE 3

Common alloy elements and their effects on steel.

Alloy element	Character of steel	Uses
Low carbon % (upto 0.15% C).	Soft and ductile	Wire, tubing, boiler plate, rivets, nails.
Medium carbon % (0.15 to 0.4% C).	Harder	Axles, forgings and machinery.
High carbon % (0.4 to 1% C).	Very hard	Chisels, drills, files, dies, machine tools.
Chromium	Hard, temperature and corrosion resistant, high tensile strength.	Dies, ball bearings, safes, stainless steel.
Copper	Much the same as for small amount of nickel.	Corrosion resistant tubes and sheet metal.
Manganese	Tough and abrasion resistant.	Rock crushing and digging equipment, curved rails.
Molybdenum	Shock resistant, strong, tough, hard.	Automobile parts, tools, machinery.
Nickel	High elasticity, ductility, resistance to corrosion, hardness.	Armour plate, structural steel for bridges.
Silicon	High electrical resistance, large magnetic permeability.	Electrical equipment.
Tungsten	Hard and tough even at dull red heat.	Tool steel, high speed cutting tools, magnets, armour-piercing shells.
Vanadium	Like Molybdenum steel but also fatigue resistant.	Shafts, axles and steel springs.
Wrought iron (low carbon with small amount of slag "Kneaded" in).	Tough, corrosion resistant, malleable.	Ornamental metal work, pipes for wet ground.

(3) It gives permeability in the furnace for better contact of gases with the burden.

Chemically pure iron melts at 1,530°C. The presence of 0.1% carbon in the iron reduces the melting point to 1,520°C.

In countries where cheap hydro-electric power is available the electric power can be utilised to bring the furnace charge to a reduction temperature of about 1,200°C, so that 2/3 of the coke requirement can be saved. The coal from which coke is made should not contain more than 15% ash for economic blast furnace operation but due to the limited resources of coking coal, and government pressure, steel works have agreed to use coal containing 17% ash. Coal consumption in the blast furnace goes up by 7 to 8% with every 1% increase in ash. In the Mysore Iron and Steel Works at Bhadrawati the smelting of iron ore takes place in electrically heated furnaces and coke is used for reduction of the ore.

Limestone fed to the blast furnace acts as flux, that is, it causes the impurities like silica, alumina, etc. (having high melting point) in the iron ore to melt at a comparatively low temperature. This is possible because limestone combines with the impurities at a high temperature to form compounds which are more easily fusible. The limestone used for flux purposes in a blast furnace should be of the following specifications :

$CaCO_3$ content should be not less than 90%, silica not more than 7%, MgO to be less than 4%, sulphur and phosphorus to be as low as possible. The material should be in blocks of 5 to 10 cm size.

In the limestone used for steel melting shop the insolubles should not exceed 4%.

Impurities in iron ore and their effects

The value of an iron ore is to be judged not only by its iron content, but also by the mineral form in which the iron appears and the slag forming impurities. The non-metallic impurities which form the slag are termed "gangue". The commercial analysis of iron ore gives the percentage of Fe, SiO_2, Al_2O_3 and, if so desired by the seller or buyer, percentage of moisture and/or

other impurities like sulphur and phosphorus. It is the usual practice to report the result of analysis on dried samples (laboratory samples dried at 105°C to 110°C to constant weight) and express the results of moisture, iron, silica, alumina, titanium, vanadium, CaO, ferrous oxide and combined water to second place of decimal and the results of phosphorus, sulphur, lead, copper, zinc and arsenic to third place of decimal (ISS 1493-1959). A complete analysis of iron ore gives the analysis on the following lines and is carried out only for special purposes.

Complete analysis of iron ore—Sanguem area, Goa.

Fe_2O_3	..	80.31	.. Fe :	58.00 %
FeO	..	2.37		
SiO_2	..	3.94		
Al_2O_3	..	5.47		
TiO_2	..	0.20	.. Ti :	0.12
MnO	..	1.05	.. Mn :	0.81
CaO	..	Trace		
MgO	..	Trace		
SO_3	..	0.05	.. S :	0.020
P_2O_5	..	0.09	.. P :	0.039
Combined water		6.50		

The effects of various impurities in iron ore[3] are as follows :—

SILICA

Silica and other clay matter has a high temperature of fusion and such impurities cannot be fused or melted at the working temperature of a blast furnace. To render them easily fusible a flux, generally in the form of limestone, is added to the burden of a blast furnace.

High grade iron ore (containing 60% Fe and above) with silica percentage of more than 3 is generally not accepted by blast furnace operators. Silica renders the pig iron more fluid.

Its presence is therefore desirable if pig iron is to be used for foundry practice.

ALUMINA

Its melting point is 2,000°C. Higher the percentage of alumina in the ore, more viscous is the slag and this causes operational difficulties as the slag has to be tapped from time to time. Alumina percentage of more than 5 in the ore is normally not tolerated by blast furnace operators. Indian iron ores are generally high in alumina content. A Steel Plant would consider unfavourably an ore in which silica and alumina percentage together exceeds 8.

SULPHUR

This causes red shortness *i.e.* the iron is brittle when it is red hot. Part of the sulphur from the ore can be removed by calcining the latter before smelting. Sulphur upto about 1% is generally present in the hard coke. Assam coals, though forming good coke, are not suitable for metallurgical purposes due to their high sulphur content. The presence of sulphur in the iron ore may be 0.01 to 0.4%, in the form of FeS_2 or $CaSO_4$. Sulphur upto 1% is removed in the form of slag if sufficient lime or manganese ore is added to the burden of blast furnace.

PHOSPHORUS

Phosphorus in iron causes cold shortness i.e. the iron is brittle when cold. Its presence in iron or steel is therefore harmful. So undesirable is the presence of phosphorus in iron ore that the latter are classified as *acid ores* with Phosphorus not more than 0.05%, and *basic ores* with phosphorus more than 0.05%. Acid ores lend themselves to manufacture into steel by a simpler and cheaper "acid" process but the basic ores need a more costly "basic process" for steel making.

Phosphorus may be present in the hard coke or iron ore in the form of phosphates, $Ca_3(PO_4)_2$ or $Fe_3(PO_4)_2$. It cannot be removed in the blast furnace by way of slag, but the pig iron has to undergo further treatment during steel making for phosphorus elimination, at an extra cost of flux and

fuel. A good steel must not normally contain more than 0.05% of sulphur and 0.05% of phosphorus.

MANGANESE

Normally manganese is present as an impurity upto about 1.5% in iron ores. In the blast furnace, the manganese combines with sulphur to form MnS which is less harmful than FeS. In the molten pig iron, some MnS rises to the surface and a little sulphur is oxidised escaping in the air as SO_2. Manganese confers on steel the property of toughness and abrasion resistance.

If the iron ore contains impurities like $CaCO_3$ or $MgCO_3$ the cost of flux to be added to the burden is proportionally reduced. Such ore is partially self-fluxing. Sometimes the proportion of silica, alumina and limestone in the gangue may be such as to render the ore self-fluxing, eliminating the need for adding external flux. The famous Minette ore from Minette district mines on the border of France, Germany and Belgium, is nearly self-fluxing and is valued as an iron ore though the Fe content is as low as 36%. Generally the Fe content of iron ores used in our blast furnaces is 57 to 60%.

For 1 ton of pig iron produced, 1% increase in gangue increases coke consumption by 25 to 30 kg, flux consumption by 30 kg, and the slag formation goes up by 40 kg. The loss of production is also nearly 4%.

Specifications of iron ore laid down by MMTC are given below.

Type	Grade % Fe	Size
(a) BF grade ..	58—60	½" to 4" (10% allowed undersize and above)
(b) Export grade ..	63—65	½" to 8"
(c) SMS grade (domestic)	64 and above	2" to 8" (only lumpy ore)

The grade of iron ore utilized in the blast furnaces of the different steel plants in the country varies within certain limits depending upon the chemical composition of other batch materials. Following is the general specification of iron ores used in blast furnaces[2] :

Fe—57 to 63%
SiO_2—2 to 6% Maximum ratio of SiO_2 : Al_2O_3=1 : 1.5
Al_2O_3—5 to 6%
P—0.03 to 0.15% and maxm. combined SiO_2+Al_2O_3=10%
S—below 0.02%
Size=12 to 100 mm. Tolerance limit on minimum and maximum is 10% in each case.

In the steel melting shop, high grade iron ore with the following specifications, in general, is used :

Fe—63 to 67%
Al_2O_3—2 to 2.5% P—0.05% max.
SiO_2—2 to 2.5% S—0.02% max.
SiO_2+Al_2O_3=7% max.
Size=12 to 150 mm.

Against this may be compared the actual analysis of the materials used in one of the steel plants (Durgapur Steel Plant) as given below—

(1) Average Iron ore analysis (%) Bolani ore :
SiO_2—2.05 ; Al_2O_3—5.70 ; Fe—60.17 ;
Fines of minus 12 mm—28.18%

(2) Analysis of dolomite (%). (Birmitrapur)
SiO_2—6.70 ; Al_2O_3—2.60 ; Fe_2O_3—0.84 ; CaO—27.55 ; Mgo—19.86 Loss on ignition—41.90

(3) Limestone analysis (%)

Al_2O_3	Fe_2O_3	CaO	MgO	Loss on ignition
1.33	0.35	45.80	2.16	13.42

(4) Coke (%)

Ash—24.60 Vol. matter—1.20 Carbon—74.20, Moisture—3.36

Average Micum index over 40 mm—78.80

(5) Manganese ore from Bansapani (%)

Mn—30.72, Fe—20.39, Insolubles—9.35

SLAG

The molten mass formed by the reaction between gangue of the ore and the flux is known as slag. Slag from blast furnace consists mainly of calcium, magnesium and aluminium silicates, *e.g.* (percentages of one sample at Durgapur Steel Plant)

SiO_2—30.85, Al_2O_3—25.69, CaO—34.99, MgO—6.52

Sulphur in the ore is removed in the form of CaS or MnS which dissolves in the slag.

The slag has various uses. Calcareous slag is mainly used for cement production whereas slag with a low lime content is processed to give road construction materials, ballast, etc. Slag wool is a high grade heat insulating material. It is produced by subjecting a shallow stream of suitable slag to the action of a steam jet. Slag wool is mainly used for lagging steam pipe ranges, but it can also be used for sound proofing purposes. Powdered slag is used as an agricultural lime fertilizer. If the slag is properly granulated, it provides a good material for packing the goaf of underground mines by hydraulic stowing and is a matter of interest to the mining ndustry.

Cement and fertiliser factories, which can get an important raw material in the slag of blast furnaces, normally flourish near iron and steel plants.

SIZE OF IRON ORE

Iron ore fed to blast furnace should be within certain size limits. The suitable size of iron ore to be fed to blast furnace is 10 mm and above. With smaller sizes or fines, resistance to flow of gas in the blast furnace increases at a fast rate from about a relative index of 2 to 3 for 12 mm size, to 4 for 10 mm,

7 for 6 mm and 15 for 3 mm. This apart, the iron ore fines from the charge fed to blast furnace are likely to be carried away in the flue dust. The upper limit of the size varies from plant to plant. Bhilai uses—50 mm lumps, Rourkela—80 mm and some others—100 mm.

The Indian iron ores produces nearly 33% fines during mining operations and transport. Until a few years ago many mining companies used to discard them in rejection dumps. The blast furnace flue gases also contain 100 to 200 kg of dust particles per ton of pig iron produced in the form of iron ore fines which are wasted if not recovered and utilised. Most of the steel plants are now equipped with sintering plants where the coarser fines recovered from flue dust (and in some cases, fines received from the mine, as at Jamshedpur and Bhilai) are agglomerated to form sinter or lumps suitable for feed to blast furnace. Fines produced in the mines can also be formed into pellets of nearly 15 mm size in a pelletising plant. At present the only pelletising plant in India (and also in Asia, excepting Japan), is at Chowgule & Co.'s Pale mine in Goa. Pellets may be formed of low, medium or high grade ore fines, but the expenses involved justify a plant only if the ore fines contain not less than 60% Fe for domestic use and not less than 65% Fe if pellets are to be exported.

Fines suitable for sintering should contain at least 90% particles of +0.1 mm size, whereas fines suitable for pelletising should contain at least 75% particles of −0.06 mm.

3. REQUIREMENTS OF IRON ORE IN 1970-71

The requirements of iron ore in 1970-71, the last year of the Fourth Five Year Plan would consist of (a) requirements of the domestic steel plants and (b) requirements of export market.

(a) For production of 1 ton of pig iron, the important raw materials to be fed in the blast furnace are :

Hard Coke—0.9 to 1 t
Limestone (flux)—0.5 to 0.7 t
Iron ore (Lumpy)—1.5 to 1.7 t

To produce 1 ton of ingot steel, generally 0.8 ton of pig iron, nearly 20% of steel scrap and 10 to 12% iron ore are required. The scrap is always available from the steel plant's rolling mill and steel melting shops. In short, for 1 ton of steel ingot 0.8 ton of pig iron produced from 1.2 to 1.4 tons of iron ore is required. It is reasonable to assess, for purposes of planning, the requirement of iron ore as 1.5 tons for 1 ton of ingot steel production.

The construction delays and production trends in the steel plants indicate that in 1970-71, the total production in steel plants will be about 12 million tons ingot steel requiring about 18 million tons of iron ore.

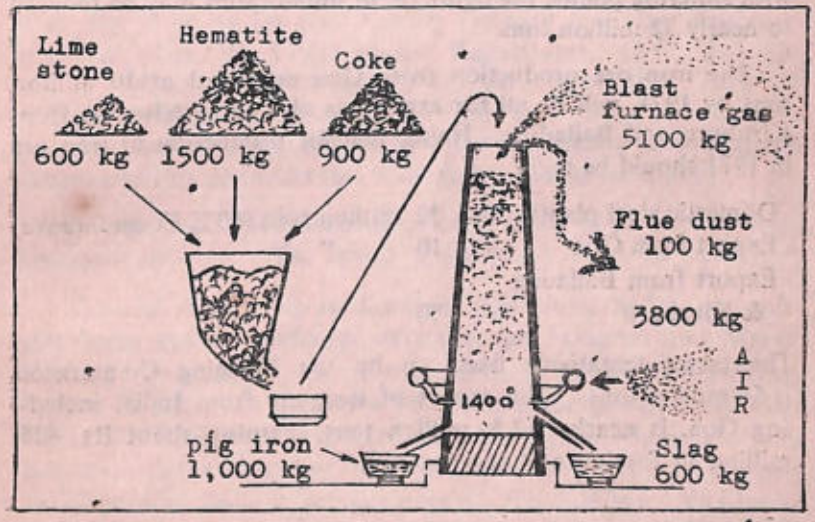

Fig. 1. Raw materials and end products in a blast furnace.

The demand for pig iron for foundry purposes is estimated at 3 to 4 million tons, equivalent to nearly 6 million tons of iron ore. Demand of iron ore would thus be,

For 12 million tons steel ingots	—18 million tons iron ore.
For 4 million tons pig iron in foundry practice	— 6 million tons iron ore
Total	24 million tons iron ore.

Other raw materials required for smelting 24 million tons iron ore would be—

Hard coke —16 million tons, equivalent to 22 million tons of washed coking coal or 32 million tons of raw coking coal.

Lime-stone, —nearly 10 million tons.

As the Indian iron ores produce nearly 33% fines during mining operations and transport, to get 24 million tons of lumpy ore the production of ore with 60% Fe and above should be nearly 36 million tons. This quantity may be reduced partly if the fines are used in the form of pellets or sinter after agglomeration, and since most of the steel works are equipped with sintering plants, the figure of 36 million tons may be reduced to nearly 32 million tons.

The iron ore production from Goa estimated at 10 million tons by 1971, will be all for export, as also the production from Kiruburu and Bailadila. Hence realistic production of iron ore in 1971 should be :

Domestic steel plants	—	32 million tons (60% Fe and above)
Export from Goa	—	10 "
Export from Bailadila & Kiruburu	—	10 "

The target tentatively fixed up by the Planning Commission is 54 million tons. The export of iron ore from India, including Goa, is nearly 12.5 million tons, earning about Rs, 425 million in foreign exchange*

PUBLIC SECTOR COMPANIES PRODUCING IRON ORE.

The following Companies in the Public sector run their own iron ore mines.

1. Hindusthan Steel, Ltd.
2. National Mineral Development Corporation, Ltd.
3. Andhra Pradesh Mining Corporation.

*Estimated figures of 1966.

4. Board of Mineral Development, Mysore.
5. Mysore Iron & Steel, Ltd.
6. Board of Mineral Development & Bolani ores, Ltd.
7. Orissa Mining Corporation.

The reserves of iron ore of workable grade in the country are nearly 28,000 million tons and are the largest in the world, constituting nearly 25% of the world's resources. The reserves of high grade ore (58 to 60% Fe) alone are estimated at 15,000 million tons.

MINING AND OTHER LEGISLATION

Mining legislation applicable to the iron ore industry is practically the same as for coal mining industry with the difference that in place of Coal Mines Regulations, 1957, Metalliferous Mines Regulations, 1961, are applicable. Officials holding managerial and supervisory posts in mines, should have passed statutory examinations held by the Mining Examinations Board and this provision has become applicable since 1964.

A wage board had been constituted in 1963 for iron ore and limestone quarries. The award of the board is not yet published.

The sale of iron ore to domestic steel plants and buyers outside India can be effected only through Minerals and Metals Trading Corporation, a Government of India undertaking, formed in 1963. An exception has been made in respect of Goa where the established exporters of Portuguese days are still permitted direct export of iron ore, till the existing contracts between them and the foreign buyers are fulfilled. Thereafter the sale will be through M.M.T.C.

5. GRADES & PRICES OF IRON ORE AND THE EXPORT MARKET

For the purpose of sampling, iron ore available in this country has been classified into the following three categories on the basis of its physical properties (IS 1405-1960).

(a) Hard ore —ore of sizes not exceeding 100 mm and of bulk density 2,500 kg/m³ or more.

(b) Soft ore —ore of sizes not exceeding 100 mm and of bulk density below 2 500 kg/m³.

(c) Screen ore —ore of sizes not exceeding 10 mm.

The hematite ore, by commercial terminology, is divided into 3 grades as follows based on Fe content.

54 to 57%	Fe content	— low grade.
57 to 60%	-do-	— normal grade.
above 60%	-do-	— high grade.

Even amongst the low grade ores, ores with Fe content of 55% and less are not easily accepted by the buyers and they can only be blended with high grade ores.

Depending upon the size, ores of —10 mm size are considered as fines, +10 mm —40 mm as nuts and +40 mm as lumps. The maximum size of lumps generally accepted by buyers is 200 mm. The smaller the size of lumps, the more is the possibility of a uniform composition of the cargo.

In the mines of Goa, Redi and a few areas on the West coast, where the iron ore in some pockets is mixed with a sizeable percentage of manganese, certain commercial terms are widely used. The term Ferromanganese (technically it should be called ferruginous manganese) contains minimum 28% Mn and minimum 22% Fe, the total percentage of manganese and iron not exceeding 55. With more manganese percentage the ore is called manganese ore with the following grades.

Mn%	Grade
38 to 41.9	— Low or No. 3
42 to 45.9	— Medium or No. 2
46 to 48	— High or No. 1
above 48	— Special.

Another term used in the same areas is *black iron* (technically it should be called manganiferous iron). It contains manganese from 5 to 10% and iron from 50 to 45% the total percentage of iron and manganese not exceeding 55.

The sale of iron ore to the steel plants of Hindustan Steel Ltd. is effected only through the Minerals and Metals Trad-

ing Corporation (M.M.T.C.) which has fixed up the following prices f.o.r. loading station of the mine (May 1966).

For Rourkela, price of ore of export grade with iron content of 65% has been fixed at Rs. 16/- per ton with provision for rejection below 63% content.

For blast furnace grade supply to Durgapur, price is fixed at Rs. 13/- per ton for 58% Fe content ; ore below 58% Fe is to be rejected.

For export purposes, a price of Rs. 14.50 per ton (based on 65% Fe content and rejection below 63%) has been fixed ex-loading stations in the Barajamda sector. For Units situated at a distance of more than 13 km an additional load charge of 40 paise per ton-mile subject to a ceiling of Rs. 16/- per ton is fixed. The export price is to be in force for a period of three years ending March, 1969.

The prices of iron ore from Goa which is all exported were as follows in 1966 f.o.b. Marmagoa harbour.

Lumpy ore	per tonne
57% Fe	dollars 5.20
60% Fe	" 6.78
62% Fe	" 8.49
Powdery ore	
60% Fe	" 4.50

Japan and Germany are the main importers of Indian iron ore. How long we shall be able to attract foreign market is however doubtful. North-West Australia, having huge reserves of iron ore, is opening new projects, and has entered into agreement with Japan to supply 100 million tons of ore in 10 years' time. Liberia in Africa has alsc opened large mechanised Mines. Germany and other European countries will naturally be interested in Liberia because of low freights. Swaziland in Africa is likely to be another formidable competitor to India. On the north-east coast of Baffin Island in Canadian Arctic, very high grade iron ore has been discovered in recent years. The ore has purity within a fraction of one percent of the theoretical purity. Although shipping from the

island can take place only for 2 months in the year due to the coast remaining frozen for major part of the year, the land would appear to be a good competitor.

The normal cost of ore produced in semi-mechanised mines, transport and ore loading could be generalised as below :

Mining cost F.O.R. mine	Rs. 11.50 to 14.00 per ton
Railway freight (about 400 km distance)	Rs. 17.00 per ton
Port handling charges	Rs. 10.00 "
Overheads	Rs. 3.00 "
Total	Rs. 41.00 to 44.00 per ton.

The sale price of ore f.o.r. port of loading on East coast is nearly Rs. 38.66 per ton for iron ore with 60-62% Fe. It would thus appear that in exporting the ore, there is a net loss of Rs. 2.50 to 5.50 per ton unless efforts are made to curtail the costs at various stages. Table 4 indicates the scheme for export of iron ore by 1971 through various ports.

TABLE 4

Probable export trade in iron ore by 1971

Port	Mines/areas to be served (distances in km inside bracket)	Export in 1970-71 in million tonnes
1. Vizag ..	Bailadila (480), Kiruburu and Bonai range.	6.0
2. Madras ..	Hospet (550)	3.0
3. Marmagoa ..	Bellary-Hospet (375) and Goa (50).	10.0
4. Paradeep ..	Tumka-Daiteri (150) and Nayagarh (300).	2.0
5. Haldia ..	Dirishimburu (380)	2.0
6. Mangalore ..	Kudremukh (65), Hassan	2.0
7. Cuddalore ..	South Mysore	0.5
8. Karwar ..	Hospet (300)	0.5
		26.0

The charges per ton for shipping iron ore from East coast Indian ports to Japanese ports are nearly as follows (pre 1966 devaluation).

10,000 tonner ship	—	Rs. 33/-
30,000 " "	—	Rs. 23/-
60,000 " "	—	Rs. 15.50

There is no port in India as yet which can load even a 30,000 ton ship at a high rate by mechanised loading. Vishakhapatnam and Paradeep may have high capacity loading facilities in the near future. Small ships for transport of iron ore are generally of 10,000 tons capacity and special ore carriers are of nearly 35,000 tons capacity. The trend is for high capacity carriers. Between Peru and Japan carriers of 65,000-70,000 tons are already in use and three 100,000 tonner carriers are being used between South Africa and Japan.

REFERENCES.

1. KRISHNAN M. S. "Iron Ores of India", 1955.
2. INDIAN BUREAU OF MINES "Indian Minerals Year Book, 1963."
3. SWARUP D. "Elements of Metallurgy."
4. ITALAB (GOA), LTD. "Analysis Reports on Goa Iron Ores."
5. SHARMA N. L. & RAM K. S. V. "Introduction to India's Economic Minerals"
6. SEHGAL K. L. & SARIN G. K. "Iron Ore Export", paper read at C.M.R.S., Dhanbad, March, 1966.
7. NATIONAL COUNCIL OF APPLIED ECONOMIC RESEARCH "Cost price structure of iron ore."

CHAPTER 3

GEOLOGY OF INDIAN COALFIELDS

1. Origin of coal. 2. Petrography of coal. 3. Classification of coals. 4, Geographical distribution of coalfields. 5. Gondwana coals. 6. Tertiary coals. 7. Reserves. 8. Principal coalfields : Jharia, Raniganj, East Bokaro, South Karanpura, Pench-Kanhan valley, Pranhita-Godavari Valley.

1. ORIGIN OF COAL

It is almost universally accepted that coal is formed from plant remains preserved from complete destruction under propitious circumstances and later altered by various chemical and physical agencies. There appear two distinct phases in its formation : (1) biochemical or peat phase and (2) Dynamo-chemical or metamorphic phase ; the former represents the period of accumulation and preservation of plant substances (peat deposits) and the latter, the period of transformation of peat deposits to coal. During peat formation, environments must have been favourable for plant growth and for preservation of their remains. The different types of coal are accounted for by different environmental conditions under which peat accumulated in successive time periods in the same locality or in differing environments at different localities in the same time period. The factors which characterised these environments may be the nature of topographic depression which determined the size and form of peat swamp, nature of underlying soil, the climate, the latitude and altitude, inlets and outlets to the swamp and the nature of swamp water.[1] Long time interval that must have elapsed in the peat formation, gave rise to variation of plant material and this resulted in vertical variation in the normal banded coal. In this stage biochemical forces transformed the organic material of peat into fusain and

vitrain, the former rather more rapidly than the latter. The latter is nearly absent in waxy resinous plant substances of peat.

The next (second) stage in coalification was the lignite stage. Under acid conditions and with no covering, the peat tended to form ulmified material approximating to the molecular formula $C_{65}H_4O_{30}$. The covering over peat gave rise to alkaline conditions resulting from a base exchange with sodium chloride and this in turn resulted in anaerobic bacterial activity and decomposition. CO_2 was evolved and hydrogen and carbon contents increased and a low-rank bituminous coal approximating to a formula $C_{79}H_{5\cdot5}O_{14\cdot1}$ resulted.

In the next (third) stage low-rank bituminous coal changed to medium-rank bituminous coal ; in this hydrogen content remained constant but oxygen fell. In the fourth stage, that is, from medium-rank to high-rank bituminous coal, hydrogen content fell and oxygen fell very slowly, by-products being CH_4, CO_2 and probably H_2O. In the last (fifth) stage, bituminous coal was converted into anthracite, oxygen remaining constant and hydrogen falling rapidly, by-products being CH_4 and little H_2O and, no CO_2. In all these stages, carbon content increased steadily, till in anthracite it is over 93% on unit coal basis.

This normal sequence of coalification under which coal underwent gradual and progressive alteration from lower rank to higher rank as a result of prolonged and sustained application of heat and pressure of sedimentary cover overlying peat during the dynamo-chemical period, is summarised in next table.

There are, however, instances of relatively immature coals passing to higher rank by heat treatment under pressure under the influence of hot intrusive igneous rock, the alteration depending on the distance of the seam from the igneous mass. This is known as *thermal metamorphism*. Another source of heat may be rock disturbances such as folding, faulting and fracturing that accompany mountain-building activity. The coalfields of Assam and of Jammu are the examples in question. The upgrading of coal to a rank above that reached in normal coalification is called an *extraordinary process of coalification*.

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SEQUENCE OF COALIFICATION

Rank designation	Stage	Phase
Peat	Peat forming stage	Biochemical phase.
Brown coal	Lignite Stage	Dynamo-chemical phase.
Lignite		
Sub-bituminous		
High volatile Medium volatile Low volatile	Bituminous stage	
Semi-anthracite Anthracite Meta-anthracite	Anthracite stage	

Rank of coal :—Rank of coal is now defined in terms of carbon content of coal ulmins. It is reasonable to assume that in the maturing process with uniform temperature and pressure over a long period of time coal ulmins have undergone a uniform progressive change in carbon content, perhaps in accordance with the natural law of geometric progression. "Increase of rank" connotes a natural process in which carbon content of coal is increased at the expense of hydrogen and oxygen. Carbon content in coal can also be increased by the process of carbonisation but such increase by a process which is not natural, does not connote increase in rank.

In—Situ and Drift Theories

There are two broad concepts of the origin of the coal deposits. Some and the most important world deposits of Carboniferous age, are *in situ* deposits, that is, those laid down in basins subjected to frequent subsidence ; others comparatively thick, (these include some fields of Carboniferous age and many

TABLE 1

Classification of Indian Coals by Rank (IS : 770-1964)

Percentages in Unit Coal, that is, in Dry and Mineral-Free Coal.

Type		Carbon	Hydrogen	Nitrogen	Oxygen
Anthracite	A 1	Over 93	3 to 4	1	1 to 2
	A 2	93 to 92	3—4	1	1— 2
Bituminous	B 1	92—91	4.4—4.6	1.5	1— 2
	B 2	91—87	4.5—5.3	1.5	2— 6
	B 3	87—84	5.0—5.8	2	5— 8
	B 4	84—82	5.0—5.5	2.5	8—11
	B 5	82—80	4.5—5.5	2.5	10—15
Sub-Bituminous	B 6	80—76	4.5—5.0	2	15—17
Lignites or Brown Coal	L 1	75—65	4.5—5.5	1	20—30
Cannel (rare in India)	L 2	75—65	5—6	1	20—30
Peat		60—50	5.5—6.5	1 to 3	30—40

of later age) may have been due to drift of the accumulation of vegetable matter into an enclosed basin (drift deposits).

Fox who did most of the early work on Indian coals, believed that coals in India are almost entirely of drift origin. The two main periods of coal formation in India were the Gondwana period (corresponding to Permian and Permo-Carboniferous age) and the Tertiary period. The Gondwana coal deposits of Raniganj, Jharia and other fields are of fresh water origin, accumulations having taken place in great basins of lakes, while the Tertiary coals, such as in Rajasthan, are of marine origin. The Indian coal deposits have the following characteristics[2] :

1. Coal is formed largely from terrestrial plant material growing on dry land and not in swamps.
2. The plant material must have been carried by water and deposited into seas or lakes. Indian coal seams are not found lying unconformably over older strata nor directly on granites or gneisses. There is no evidence of erect stems with roots in underclays and there

is no soil found under coal seams ; the coal and lignite seams are very intimately intermixed with sedimentary deposits.

3. The transported materials settled down in regular succession according to their densities and velocities of carrying currents.

4. Minerals in carrying water replaced the organic constituents of tree trunks and have given rise to fossil wood or tree trunks in both fresh water and marine formations.

FACTORS AFFECTING THE COMPOSITION OF COAL

1. The nature of the coal forming plants, particularly nature that affects chemical composition or resistance to decay.

2. Mode of accumulation and burial of the plant debris forming the deposits.

3. The chemical composition of the coal forming debris and its resistance to decay.

4. The nature and intensity of the plant decaying agencies.

5. The age of the deposits and their geographical distribution.

6. The subsequent geological disturbances.

2. PETROGRAPHY OF COAL

As plant debris gets converted to lignite, outward form of the main structure of wood or cortex is retained and is generally distinguishable from the remains of the other miscellaneous debris consisting of cellular tissues, leaves, spores, pollen grains, etc.

In the higher ranks, wood or cortex appear as bright bands and the other debris as dull bands. Thiessen[3] called the bright components "anthraxylon", a term derived from Greek *anthrax* (coal) and *xylon* (wood), and the dull components, "attritus".

Stopes[4] recognised and separated by hand picking four bands in bituminous coals and named them as under :

1. *Fusain* or "mother of coal" or "mineral charcoal"—charcoal-like layers appearing chiefly as patches and

wedges, flattened parallel to bedding plane and often with square-cut ends, powdery, very friable, fibrous, dull and opaque.

2. *Vitrain*—which, together with "Clarain" is "bright coal", a term used by other scientists, occurring as definite narrow horizontal bands sometimes straight and flat and sometimes lenticular. A single brilliant vitrain band is coherent and uniform without layering, vitreous in texture, with conchoidal fracture, friable and clean to touch.
3. *Clarain*—which occurs as thin or very thick bands, stratified parallel with bedding plane, with silky or satin gloss, less friable than vitrain, with no conchoidal fracture.
4. *Durain*—a dull component occurring as a band of variable thickness and with lenticular shape. It is hard, firm, granular, and breaks with a bumpy or matte surface.

This earlier identification of components of coal based on megascopic study of visually recognisable bands (known as rock-types or litho-types) has in recent days been replaced by one based on microscopic examination of optically identifiable constituents known as *maceral constituents*, viz., vitrinite, exinite, inertinite.

As this new petrographic investigation progresses, we hope to know the fundamental characteristics of coal so that we may predict its behaviour in different technological processes.

Microscopic studies of coal have found useful application in determining rank, improving coke quality, correlating coal seams, briquetting, predicting nature of coal seams at depth from the study of weathered outcrop and in many other practical fields.[5]

Four principal methods of microscopic investigation of coal are :

1. Maceration method
2. Thin section method.
3. Polished section method
4. Polished thin section method.

In maceration method, coal is oxidised by a suitable oxidising agent over an appropriate period which may vary from 15 minutes in the case of lignites to over a week in bituminous coals; after digestion, coal is washed with distilled water, then alkali and again distilled water. The residue which consists of more resistant spores, cuticles and other plant remains, is dried and mounted on glass slides for microscopic examination of the microfossils. The study may reveal the nature of the original vegetation, indicate the palaeo-climate at the time of deposition and help in correlating coal seams.

Thin sections of coal are prepared for examination under transmitted light. In Indian coals, however, owing to high mineral matter content, it is difficult to prepare them. When it is possible to obtain a thin section, micro-structure of transparent and semi-transparent constituents is clearly seen.

Polished section is used for examination in the reflected light and is easily prepared. A 2 to 5 cm cube is cut out from the sample and a surface perpendicular to the bedding plane is first ground with carborundum powder in water or glass plates to be followed with polishing with alumina powder. Opaque constituents of coal are better seen on this section. Polished thin section permits both techniques of transmitted and reflected light to be used for thorough investigation of micro-constituents of coal.

The constituents which can thus be examined under the microscope are known as **macerals**, an analogy with minerals. According to the nomenclature adopted by the International Committee for Coal Petrology (1957) macerals are grouped in three groups : exinite, vitrinite and inertinite. These macerals groups corresponding to rock types or litho types, described earlier are given in Table 2. Coal is examined for (i) its petrological composition, that is, the amounts of various constituents present, and (ii) its degree of maturity or rank.

Petrological composition of coal is determined with the integrating stage or point counter and rank, by measuring reflectivity, that is, the percentage of light that is reflected from its vitrinite surface. The reflectance values lie between 0.3% as in the case of low rank coal, and 4.3% in anthracite.

TABLE 2

Classification of ingredients of Coal

Rock type	Macerals
Vitrain	.. Vitrinite with or without structure : (i) Collinite—without structure. (ii) Tellinite—with structure : (a) Xylinite—formed out of wood. (b) Periblinite—formed from cortex. (c) Suberinite—formed from cork.
Fusain	.. Fusinite—opaque cell walls with empty or mineral-matter-filled cavities.
Clarain	.. Consists mainly of— Vitrinite, Resinite, Exinite : cutinite, sporinite, with small small quantity of Micrinite—granular opaque matter, and Fusinite.
Durain	.. Consists of— Micrinite, Exinite—both varieties, Resinite with traces of Vitrinite.

With the help of reflectance and maceral composition of the types of coals examined, it is possible to calculate in a blend of different ranks of coals, the *index of leanness* and *coking capacity*. When these are known the strength of coke that can be produced on carbonisation of the blend can be predicted. This method of assessing the coking quality of the coal from the petrographic data will be of great use to us as small drilling cores can be used to predict coking qualities and also to decide on the blend of different coals to give the coke tolerated in the present blast furnace practice.

By measuring maximum and minimum reflectances in oil, the extent to which carbonisation or thermal metamorphism has proceeded can be assessed and this knowledge helps in preventing further spread of fire in a seam under spontaneous heating and in ascertaining, in an industrial carbonisation, the efficacy of the heat treatment.[5]

In briquette manufacture, reflectance measurements on the heated briquets indicate whether correct temperature and/or pressure and uniform heat treatment have been used.

The chemical composition of weathered outcrop coal differs from that of unweathered coal at depth but reflectance remains the same. The property can be used to predict the quality of coal seam at depth without the necessity of costly drilling further to the dip.

Reflectance and maceral composition of coal sections can be used to define the structure of the coal seam and these data help in correlating seams in a local area. They can form useful adjunct to correlation work with the help of "Tonstein" and other bands in coal or close to coal seams.

3. CLASSIFICATION OF COAL

Among the many attempts at classification of coal on the basis of its chemical and physical properties, may be mentioned the Seyler's & Mott's systems. In Mott's system, a chart is prepared with volatile matter and calorific value used as principal rectangular co-ordinates and hydrogen and oxygen are plotted on subsidiary coordinates inclined at an angle of 40° to the main coordinates. The majority of coals when plotted on this chart, show their distribution along a broad path known as **coal band** (though a few of Indian coals fall just outside the band). Seyler's system is also on similar lines. Though excellent in their own way, these systems could not find widespread use owing to their complicated nature.

After the World War II, with international trade in coal there arose need for international system of classification of coal based on parameters which can independently give some information bearing on its utilization. Internally, such classification can help in national planning of production of the types of coal needed to serve specific demands in the country, assists a consumer in choosing the correct type for his requirement and enables the industrial user to obtain a substitute for an old source of supply, since exhausted.

The coals are broadly divided into Hard coal and Brown Coal. Hard Coal, after the usage in Europe is defined as coal with gross calorific value of 5,700 kcal/kg $\pm$ 60 kcal/kg on moist ash-free basis and includes anthracitic, bituminous

and high-rank sub-bituminous coals. Brown Coals are coals which give gross calorific value below 5,700 kcal/kg ± 60 kcal/kg.

International classification of hard coal by types, published as United Nations document in 1956, gives each type of coal a code number of three digits ; the first figure in the code number indicates the class of the coal, determined by volatile matter content upto 33% V.M. (dry ash free basis) and by gross calorific value on moist, ash free basis, above 33% V.M. ; the second figure indicates the group of coal, determined by caking properties and the third figure indicates the sub-group determined by coking properties. The types of coal are then grouped in statistical groups for simplification.

The international scheme of classification for brown coals is based on two physical characteristics which indicate its value as fuel and as chemical raw material. The code number consists of 4 digits, the first two indicate the class number : (10, 11, 12, 13, 14, 15), determined by total moisture % (ash-free) and the next two digits indicate the group number (00, 10, 20, 30, 40) determined on tar yield percentage (dry, ash-free basis).

In India, recently a classification based on similar principles has been proposed ; but parameters used are gross calorific value (dmf), volatile matter (dmf) and Gray-King coke type. In the code number given to a type of coal, the first digit indicates the group of coal, determined by gross calorific value (dmf) ; the second figure represents the class of volatile matter and third figure indicates the type of coke obtained in Gray-King L.T. assay on sample beneficiated to contain less than 17% ash. The classification is given in Appendix III.

4. GEOGRAPHICAL DISTRIBUTION OF COALFIELDS

The main coalfields of India occur in the eastern and mid-eastern region roughly between latitudes of 17° and 24° and longitudes of 78° and 87° east. Starting from a point approximately 185 km north-west of Calcutta and travelling westwards, we meet successively the coalfields of (1) Raniganj (2) Jharia, (3) Giridih to the north, (4) East & West Bokaro and Ramgarh (5) South and North Karanpura. To the south of these are coalfields of (6) Rewa, Korea, Korba and

Orissa ; to the west are (7) Kanhan Valley, Pench Valley and Kamptee coalfields ; further south is a (8) coalfield of Kothagudium. These coalfields belong to lower Gondwana age and coal seams are found in Raniganj Measures and the Barakar Measures (the latter is a lower horizon) separated by Barren Measures or Ironstone Shales. Coal seams of Raniganj Measures are of less mature type than those of Barakar ; both are of bituminous variety. The latter, best developed in the Jharia Coalfield, are the chief source of prime coking coal in India. 99% of the present output in India is obtained from these Gondwana coalfields.

Apart from these, but of lesser importance, are the coalfields of Assam in the north-east and of Jammu in the north-west of India ; these belong to the Tertiary age. Though young, the coals in these regions have matured to lignitous to ligno-bituminous stage and even to anthracitic stage in sub-Himalayan areas of Kashmir under tectonic pressure.

In addition, there are lignite deposits of Miocene age in South Arcot (known as Neyveli lignites) and of Eocene age in Palana (Rajasthan).

The age of coal formations in India is indicated in Table 3.

5. GONDWANA COALS.

Gondwana coal measures and associated strata are fresh water deposits which were laid down in wide river valleys and lakes that traversed the continent of Gondwana land during Permian and later times. It appears that in these days, the land-mass of southern hemisphere united South Africa, India, Australia and perhaps South America. Relative to Africa, India drifted north-eastwards. Gondwanaland was exposed to sub-aerial denudation for a very long time and the land mass was cut up into a series of wide valleys and lowland tracks and in these the fresh water strata were deposited. In the later part of the carboniferous period, climate was particularly cold and high-up regions were covered by ice caps. They seem to have given rise to glacial and flucioglacial conglomerates of Talchir series. Subsequently there was a change to a warmer climate, resulting in melting of ice and growth of vegetation of the type of Glossopteris, Gangamopteris, Dadoxylon

TABLE 5

Coal Formation periods in India

Broad divisions	Period	Principal coal formations
Tertiary coalfields	Early Pleistocene to upper Pliocene.	Lignites in the Karewa formations of Kashmir valley.
	Miocene	Lignites in the Cuddalore series of South Arcot (Madras) and Varkala and Quilon in Kerala.
	Oligocene to upper Eocene.	Lignites in Barail series in Jaipur, Nazira, Namchik and Makum coalfields of Upper Assam.
	Middle Eocene	Lignites of Palana (Rajasthan) and Kutch.
	Lower Eocene	Coal seams in Jaintia series of Cherrapunji, Mowlong and Shillong, Khasi and Jaintia hills, Garo and Mikir hills, Assam. Coalfields of Western Assam : Daranggiri, Rongrenggiri. Coalfields of Kalakot, Metka, Mahogala, Chakar, Dhanswal-Sawalkot in Jammu.
Upper Gondwana coalfields.	Upper Jurassic	Coal seams in Chikia and Kota in Kota stage in Maharashtra, Satpura region of Madhya Pradesh, Ghuneri in Kutch.
Lower Gondwana coalfields.	Upper Permian	Parts of Raniganj, Jharia, Bokaro and Karanpura coalfields of the Damodar valley in West Bengal and Bihar.
	Lower Permian	-do- -do- Mahanadi valley, Brahmani valley, Sone valley, Pench valley, Pranhita-Godwavari valley, Coalfields of Darjeeling (West-Bengal), Ranjit valley (Sikkim), Abhor, Dalphia and Aka hills (NEFA).

Note—Anthracites are found only in some pockets in the tertiary coalfields of Jammu-Kashmir, Darjeeling and upper Assam resulting from tectonic activities in the sub-Himalayan zone.

which flourished with passage of time. During early Permian, immense volume of decayed vegetation along with sand, gravel and clay eroded from higher places deposited on the Talchir strata in low-lying areas. These sediments represent the Barakar coal measures of to-day. Later, in greater part of this area, conditions favourable for large accumulation of vegetable debris ceased and Barakars were covered by strata of sandstone, shales and sandstones known as Barren Measures. In the eastern part of the present Damodar Valley region, these barren measures were succeeded by a second series of coal bearing strata, represented by Raniganj coal measures. Classification of Gondwana strata in the Peninsula is given in Table 4.

The Gondwana system is divided in two divisions, upper and lower, mainly on the basis of the type of fossils found in the strata. The upper Gondwana system contains only few isolated thin coal seams and is of no importance for coal mining. Lower Gondwanas contain Glossopteris, Gangamopteris and Vertebraria fossils.

Lower Gondwana System

Talchir Series

The Talchir series which forms the base of Gondwana system has at the base a boulder bed consisting of glaciated boulders and pebbles, which has developed to a thickness of 15-60 m. A group of fine khaki green unfossiliferous sandstones and shales containing undecomposed felspar overlie the boulder bed and is succeeded by Rikba plant bed containing fossils of Glossopteris and Gangamopteris. The last deposit is indicative of the change to warmer climate from the earlier frigid conditions.

Damuda Series

Birth place of the most important coal seams of India, this series is divided into four stages, the oldest being Karharbari stage, typically developed in the Giridih coalfield, where it consists of 60-120 m of pebble beds, grits, sandstones and shales. Though barren elsewhere, this stage in Giridih contains two oldest seams of good coking variety. Overlying Kar-

TABLE 4

Correlation of the Lower Gondwana Strata in India

		Gondwana Divisions	Damodar Valley	Son & Mahanadi Valley	Satpura	Wardha Valley & Godavari Valley	Age
GONDWANA SYSTEM (Age—Carboniferous to Jurassic)	Upper Gondwana System consisting of Mahadeva Series, Pachmarhi Stage, Jabalpur Stage, etc., but practically no coal.						
	Lower Gondwana System	Panchet Series	Panchet	—	Almod	Mangli	Lower Triassic.
		Damuda Series	Raniganj Measures or Stage; also called Upper Coal Measures	Himgir Pali	Bijori 1,000 m.	Kamthi	Upper Permian.
			Barren Measures or Iron stone shales	—	Motur		Middle Permian.
			Barakar Measures or Stage; also called Lower Coal Measures	Barakar	Barakar Umaria	Barakar	Lower Permian.
			Karharbari				
		Talchir Series	Talchir	Talchir	Talchir	Talchir	Upper Carboniferous.

METAMORPHICS

METAMORPHICS

harbari are the Barakar coal measures which are well developed along the present Barakar river, a tributary of the Damodar river, to a maximum thickness of 760 m and consist of false-bedded sandstones, conglomerates, grits, carbonaceous and micaceous shales and coal seams. In the Jharia coalfield this formation yields 25 workable seams making up a total thickness of 76 m. Wherever this stage occurs in the Peninsula, it contains workable coal seams, indicative of the widespread coal forming process.

Overlying the Barakars are a group of fine barren sandstones and shales which attain a thickness of 600 m in the Jharia coalfield and which are known as Barren Measures in the Jharia coalfield, Ironstone Shales in the Raniganj coalfield and Motur beds in the Satpura region. During their deposition, the environments must have been dry and arid incapable of supporting sufficient vegetation for coal seam formation.

Next in succession to Barren Measures are the Raniganj coal measures which are best developed in Raniganj coalfield to a maximum thickness of 1,000 m. They consist of grey to greenish felspathic sandstones, shales and coal seams, workable coal seams being found only in Raniganj, Jharia, and Karanpura coalfields. This stage is called Himgir beds in Mahanadi valley, Bijori beds in the Satpura region, Kamthi beds in Wardha-Godavari valley and Pali beds in Rewa.

Succeeding Raniganj measures are the Panchet series (named after their development in Panchet hill) of 450—900 m of greenish buff to brown sandstones and shales, devoid of coal seams.

"After their deposition the Gondwana deposits in the Indian Peninsula were subjected to numerous normal faults during Mesozoic times. Faults of at least 2,700 m and 1,500 m throw have limited the Raniganj & Jharia coalfields respectively in the south. These faults have dropped the Gondwana masses within older Archaean landmass and have prevented them from subaerial erosion during subsequent periods.

In addition to faulting, deposits of eastern part of Damodar valley were intruded by a series of ultrabasic dyke and silts during the late Mesozoic. Another series of basic intrusions of

dolerite and basalt have also visited the strata in the peninsular India and have appeared on the surface in the form of Deccan trap during late Mesozoic and early Tertiary periods. Intrusions occurring as sills have damaged coal seams and turned some of them into *jhama*; Deccan traps have covered wide areas of coal bearing measures, rendering them inaccessible for economic exploitation.

Most of the Gondwana coalfields are not affected by folding, a limited activity being confined near the faults, but the sub-Himalayan zone of Kashmir, Bhutan, Sikkim and Assam, was caught up in the intense mountain building movements and the Gondwana strata in this region have been severely crushed and folded.

6. TERTIARY COALS

Tertiary coals account for just over 1% of the total coal output of India at present; nevertheless they are important source of fuel in the areas where they occur. Tertiary coal seams are found in the Eocene and the Oligocene strata in Assam and Kashmir and a thick lignite deposit occurs in the Miocene strata of Madras. Some lignite seams are known to occur in Himalayan region of Kashmir, but are of less economic significance.

In contrast to Gondwana coal seams, tertiary deposits were the result of estuarine and marine conditions close to the sea coast. The north-western and north-eastern parts of Indian Gondwana land were encroached by Tethys sea or its branch, depositing marine sediments during the latter part of the Cretaceous. Then large parts were up-lifted above sea level giving rise to sub-aerial weathering and formation of bauxitic and lateritic deposits on the exposed land mass. During the Ranikot stage and the succeeding Laki stage coal forming conditions broke into this marine phase and gave rise to several seams of coal and lignite.

These young deposits could naturally mature to lignites only but in sub-Himalayan region of Kashmir, under high tectonic pressure, some of them have matured to bituminous and even to anthracitic stage.

Principal occurrences of Tertiary coals are in Assam, Jammu and Kashmir, Rajasthan, Madras and Gujrat, the first being the most important from the point of view of production.

Assam coals are high in sulphur content, the usual percentage being between 2 and 8, part of which is organic and part pyritic. The main 9 m seam worked in Margherita and Ledo collieries contains 2—3% of sulphur, 1% moisture and 2—3% ash. On pure coal basis (d.m.f.), this coal has a volatile content of 45%, calorific value of 8,000 kcal/kg and is highly caking. Sulphur content makes the coal highly susceptible to spontaneous heating, readily distintegrating and highly corrosive.

High caking property of these coals have prompted experiments by Central Fuel Research Institute to develop a process of desulphurisation and to use them as blends with primary coking coals of Jharia coalfield and medium coking coals of Raniganj coalfield. It appears from these experiments that the use of this coal in blends for manufacture of coke for metallurgical purposes will depend on acceptance of high sulphur (at least 1%) in coke in steel manufacture practice after the practice in European countries.

Attempts are also being made to recover high sulphur in Assam coals as a by-product. These coals have yielded 180—225 litres of low temperature tar per ton of coal (in contrast to a yield of 36—45 litres per ton of coal from other fields), 50% of which can be converted to diesel oil and kerosene fractions. For every litre of purified kerosene nearly a litre of lomex is produced. By de-alkalisation, 60% of lomex can be converted to benzene and 40% to napthalene, whose demands are increasing in India.

Ash in Assam coal is rich in rare elements of gallium and germanium, elements used in electronic industry.

7. RESERVES

For estimating reserves a standard procedure has been prescribed by the Committee on Assessment of Resources of the Coal Council of India for guidance of all agencies engaged in the work.[9] The following are the broad bases.

1. (a) Quarriable reserves of each seam are estimated separately for coal to overburden ratios of 1 : 1, 1 : 2, 1 : 3, 1 : 4 and 1 : 5.

   (b) Reserves of all coal in each seam are calculated separately within depth from surface of 0—150 m, 151 m—300 m, 301 m—600 m, 601 m—900 m and 901 m—1,200 m, due regard being paid to dislocation by faults.

2. Reserves are categorised into three classes, depending on the relative reliability of data :

   (a) Proved reserves—These are reserves estimated from dimensions revealed in outcrops, trenches, mine workings and bore-holes by assuming their continuity on geological evidence for a distance not greater than 200 m. Proved reserves are further sub-divided into :

      (i) those in working mines, standing in pillars and in partings, roofs and floors and in virgin ground,
      (ii) those in closed mines,
      (iii) those in unworked mining leases, and
      (iv) in other areas.

   (b) Indicated reserves—These are determined by taking an area of 1,000 metres square for each point of observation. This may be extended to 2,000 metres square for known geological continuity.

   (c) Inferred reserves—This is an estimate of coal based largely on broad knowledge of geological character of beds but for which there is no measurement.

3. Thickness of coal seam is taken after excluding dirt bands of over 50 mm, and portions of seam that have been burnt out or turned into *jhama* and reserves are calculated separately into following thickness ranges : 0.5 m to 1.5 m, 1.5 m to 3.2 m, 3.2 m to 4.5 m, 4.5 m to 9 m and over 9 m.

4. Suitable correction is applied for seams inclined at angles of more than 5°.

5. Reserves are classified into following ranks :

   I. Anthracite

II. Bituminous—

A. Low to medium volatile coals or coking coal — Coals upto 2% moisture in air-dried sample and upto 35% V.M. (dmf) and sulphur less than 1%.

B. High volatile or high moisture coals :
   (a) semi-coking
   (b) weakly to non-caking

C. High sulphur coals : coal containing more than 1% of sulphur.

III. Lignites.

Bituminous coals are further classified according to the quality of the seam samples and reserves are calculated on the basis of the average specific gravity of the class as indicated below. The term "class" of coal is used only for giving or calculating reserves.

TABLE 5

Specific gravity of coal of different classes for calculation of reserves

	Low to medium volatile coals or coking coals		High volatile or high moisture coal		High sulphur coal	
Class ..	Ash%	sp. gr.	Ash+ Moisture %	Sp. gr.	Ash %	sp. gr.
Class I ..	<17	1.42	<19	1.40	0— 5	1.30
Class II ..	17—24	1.47	19—28	1.45	5—10	1.34
Class III ..	24—35	1.57	28—40	1.55	10—15	1.38
Class IV ..	35—50	1.70	40—55	1.70		
Unclassified ..		1.50		1.50		

Coals with moisture upto 2% are termed low moisture coals and, above 2%, high moisture coals. Coals of volatile

matter (dmf) from 26 to 35% are called medium volatile coals ; those with lower and higher values, are low and high volatile respectively.

In situ reserves of workable seams as known so far, to the depth of 600 m, in different coalfields in India are classified in Table 6 on the basis of caking categories used by Central Fuel Research Institute.[12]

TABLE 6

Reserves of workable coal seams upto 600 m depth

Coal fields (State)	Prime coking	Caking Blendable	Non-caking
	Mt	Mt	Mt
A. BITUMINOUS COALS			
1. Assam* ..		200	3,300
2. Rajmahal (Bihar) ..			215
3. Deoghar (Bihar) ..			55
4. Barjora ..			25
5. Raniganj (West Bengal)		7,783	10,805
6. Jharia (Bihar) ..	4,600	8,000	
7. Giridih (Bihar) ..	8	30	
8. East Bokaro (Bihar)		4,307	
9. West Bokaro (Bihar)		3,000	
10. Ramgarh (Bihar) ..		1,100	
11. North Karanpura (Bihar)		300	4,280
12. South Karanpura (Bihar)		600	1,825
13. Chope-Itkhori (Bihari)			2
14. Palamu (Bihar) ..			170
15. Sone Valley (Madhya Pradesh) ..		590	33,600
16. Mahanadi Valley (Orissa)			32,685
17. Pench-Kanhan Valley (Madhya Pradesh) ..		100	1,570

*Of Tertiary age. Other coalfields belong to Permian age.

	Coal fields (State)	Prime caking	Caking Blendable	Non-caking
		Mt	Mt	Mt
18.	Jathakera (Madhya Pradesh) ..		170	
19.	Satpura basin (Madhya Pradesh) ..			9
20.	Nagpur Group (Maharastra) ..			515
21.	Wardha Valley (Maharastra) ..			3,610
22.	Godavari Valley (Andhra Pradesh) ..			8,015
		4,608	26,180	100,681
	Total		131,469	

B. Anthracite Fields

		Reserves Mt
23.	Darjeeling ..	100
24.	Jammu ..	105
		205

C. Lignites

25.	Nichoma (Kashmir)	100
26.	Palana (Rajasthan)	20
27.	Umarsar (Kutch)	10
28.	South Arcot (Neyveli Madras)	2,030
29.	Kerala	100
		2,260

Grand total ..4,608+26,180+100,681+205+2,260=133,934 Mt

Notes—1. Estimates are based on findings of Geological Survey of India categorised by Central Fuel Research Institute and compiled in respect of Jharia, Raniganj and East Bokaro coalfields by Committee on Assessment of Resources, Coal Council of India.

2. They include in situ reserves of proved, indicated and inferred reserves of seams of thickness of over 0.5 m in respect of Jharia, Raniganj and East Bokaro coalfields and of over 1.2 m in other fields.

Thus the total of all proved, indicated and inferred reserves of bituminous coal in India of seams of thickness above 1.2 m (except in the case of the major coalfields of Jharia, Raniganj and East Bokaro where estimates include reserves of seams above 0.5 m) to a depth of 600 m would appear to be of the order of 131,000 million tons. Out of this about 38% are located in coalfields of Bihar and West Bengal, about 28% in Madhya Pradesh, 25% in Orissa, 6% in Andhra Pradesh. Only about 4% of these reserves (about 4,608 Mt) belong to prime coking variety, which occurs mainly in the Jharia and Giridih fields. About 26,000 million tons are considered to be blendable coking inasmuch as by special techniques of washing, size preparation and carbonisation by blending with prime caking coals, they can yield metallurgical coke. This estimate of total reserves would be further increased if we were to consider reserves upto a depth of 1,200 m ; evidence in Jharia coalfield indicates that good quality coking coal seams have been damaged by igneous intrusions or altered in respect of quality and thickness, at depth. Drilling in other parts is still in progress, which may reveal further deposits, adding to our present estimate of minable coal.

In addition there are about 205 million tons of anthracitic coal in Darjeeling (West Bengal) and in Jammu State, and about 2,260 million tons of lignites, 90% of which lies in South Arcot (Neyveli).

Indian coals are usually dull in appearance in contrast to their counterparts in Europe or America. Coking variety coals give strong coke on high temperature carbonisation, though tar yields are approximately half of those obtained abroad. Coking coals have a moisture content of less than 2%, other varieties, within a range of 3 to 10%. High moisture coal tends to slack on storage and is prone to spontaneous combustion.

Compared to Europe and America, Indian coals contain high mineral matter (range being from 12 to 40%), very intimately disseminated throughout the coal substance ; only in some Assam coals, the ash percentage is within 2 to 8%. Ash in most coals is refractory in nature with a fusion temperature of over 1,400°C.

Most Indian coals have low sulphur content, well below 1%; only in Assam coal, it ranges from 2 to 8%. Phosphorus content is usually below the harmful limit of 0.15%. Calorific value on dry-mineral matter-free basis in coking coals ranges between 8,450 and 8,780 kcal/kg. Semi-coking varieties have calorific value between 8,100 and 8,900 and non-caking type, between 7,000 and 8,000 kcal/kg.

Indian Anthracites contain as much as 13—17% of volatiles on dry-mineral-matter-free basis, indicating that they just passed beyond the bituminous stage in maturity.

Lignites generally give high moisture content of 10—37% on air-dried samples, high volatiles of 45—65% on dry-mineral, matter-free basis. South Arcot and Palana lignites are low in ash (about 4%) but Kashmir lignites contain as much as 50%. Ash fuses at comparatively low temperature of 1,100°C.

7. PRINCIPAL COALFIELDS

Jharia Coalfield

Named after the town of Jharia in Dhanbad district, the Jharia coalfield in the Damodar valley lies to the north of the Damodar river, about 260 km north-west of Calcutta. It is the most important coalfield in the country, which produced in 1965, 36% of the country's output, is the stockhouse of the coking coal and has immense potentialities for development. (Fig. 1)

Roughly elliptical rather sickle-shaped, with its larger axis in northwest—southeast direction, it occupies an area of 450 sq km, extending roughly 38 km along east—west and 20 km along north—south and lying between latitudes 23° 37′ and 23°52′ and longitudes 86° 6′ and 86° 30′. It is served by excellent rail and road communications; Howrah-Delhi Grand Trunk road passes within 13 km to the north, with many feeder roads through the coalfield, and Grand Chord line of Eastern Railway and Gomoh-Howrah line of South Eastern Railway connect the coalfield with the rest of India.

The detailed geological survey of the coalfield was done by Fox in 1926-28; thereafter P. K. Ghosh (1947), K. Balu (1948), A. B. Dutt (1949) investigated certain parts and D. R. S. Mehta and B. R. Narayan Murthy re-examined the coalfield in 1952-53.

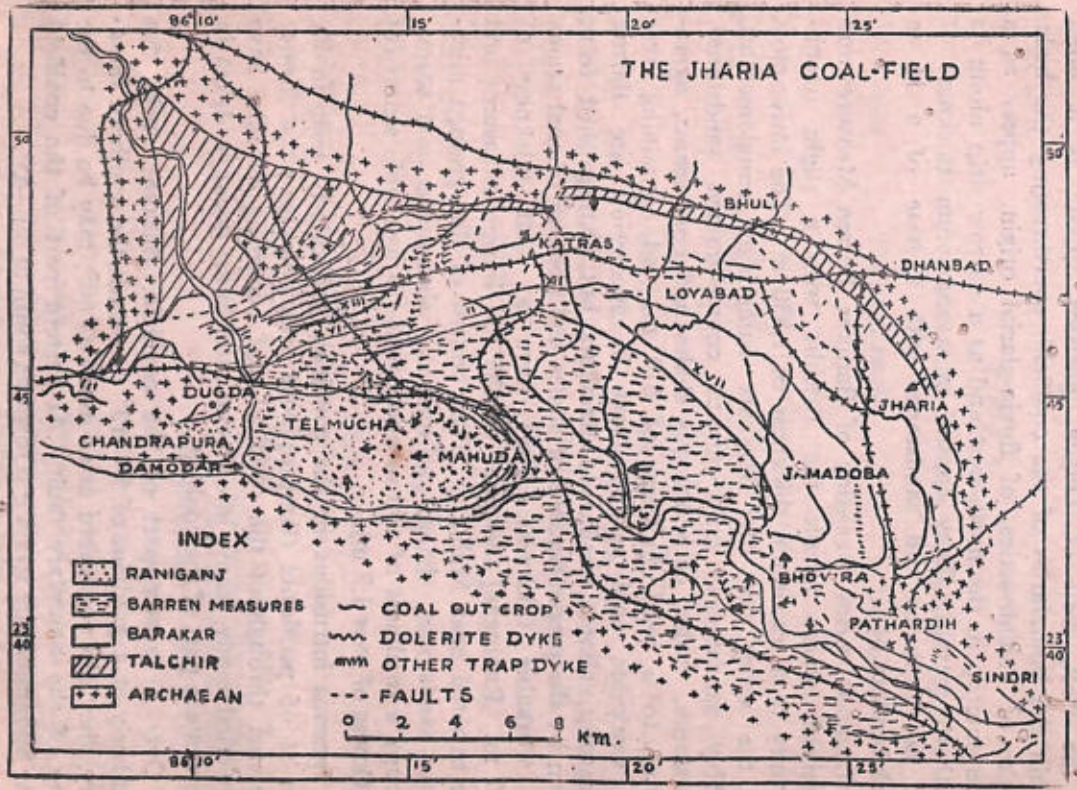

Fig. 1. Geological map of Jharia coalfield.

The field is surrounded on all sides by Archaean metamorphics with a prominent southern boundary fault estimated to have a throw of over 1,524 m. In the east, near Patherdih and in the west, west of Dugda, gneisses appear as a horst within the boundaries of the coalfield, overthrusting the beds. Rocks of Talchir-series, of fluvio-glacial origin, appear as a wide outcrop in the north-west and as a narrow strip along its northern margin. They consist of a maximum thickness of 240 m. of shales and sandstones with a base of a 15 m. thick boulder bed.

Damuda series consisting of Barakar, Barren Measures and Raniganj stages overlie the Talchirs with a slight unconformity to a maximum thickness of 1,830 m. The lower-most of the series are the Barakars comprising conglomerates, pebbly sandstones, grits, fine-to course-grained sandstones, silt-stones, shales, carbonaceous shales and coal seams, aggregating to a total thickness of 610 m, and containing over 25 workable coal seams. Next in succession are Barren Measures, almost similar to Barakars but distinguished from them by the preponderence of shales and absence of coal seams of workable thickness. They have a total thickness of 610 m. Raninganj measures overlie the Barren Measures and are exposed near Mahuda. They consist of fine grained micaceous sandstones, shales, carbonaceous shales and coal seams, occupy a thickness of 560 m and contain 9 workable seams (of thickness of over 1.2 m).

Igneous intrusions in the coalfield, particularly near faults, consist of peridotite (lamprophyre) and dolerites ; the former are met throughout the coalfield and occurring as sills, have converted many seams into *jhama* ; the latter occur as dykes in the western part of the coalfield.

Over a greater part of the coalfield, Barakar rocks dip southerly at a gradient of 5°—15° ; towards the southern boundary, they dip northward at as much as 60°, may be due to the drag of the boundary fault. At eastern part of the coalfield near Sudamdih the strata dip towards south at 40—45°.

There are nearly 25 coal seams of thickness above 1.2 m in the Barakar Measures ; 18 seams which are extensive in occurrence are numbered I to XVIII from bottom to top, and the

others, less persistent in extent, are identified by affixing A, B, etc. to the number of the coal seams, immediately below them. The general sequence, thickness and quality are given in Table 7.

TABLE 7

Coal seams in the Barakar Measures of Jharia coalfield

Seam	Thickness range m	Quality
XVIII (Topmost)	1.8— 5.2	Coking, Class I to II.
XVII	1.2— 5.8	Coking, Class I to II.
XVIA	1.8— 3.0	Coking, Class II.
XVI	1.2— 4.9	Coking, Class I to II.
XVA	1.1— 2.7	Coking, Class II.
XV	2.7—11.9	Coking, Class I to II.
XIVA	1.5— 3.7	Coking, Class I.
XIV	1.6—10.7	Coking, mostly Class I, deteriorating to II and some places III.
XIIIB	1.5— 4.0	Coking, Class I.
XIIIA	2.4— 4.3	Coking, Class I.
XIII	2.1— 9.1	Coking, Class I.
XI-XII (combined)	2.7—24	Coking, Class II.
XII	1.5— 5.5	
XI	1.2— 7.6	Coking, Class II.
X (and IX and X combined)	3.1—18.3	Coking, Class II.
IX	1.2—11.6	Coking in most areas ; in deep boreholes below Barren Measures it is semi-anthracite to low volatile bituminous ; Class II to III.
VIIIA	1.8— 6.0	Medium Coking, Class III.
VIII	2.1— 8.8	Medium coking, Class III.
V/VI/VII	11.9—22.6	Medium coking, Class III.
IV	2.1—12.0	Weakly caking, Class III to IV.
III	2.1— 7.3	Weakly caking ; Class IV.
II	2.1—18.3	Weakly caking ; Class III.
I (Bottom)	2.1— 4.3	Weakly caking, Class I to II.

The field is surrounded on all sides by Archaean metamorphics with a prominent southern boundary fault estimated to have a throw of over 1,524 m. In the east, near Patherdih and in the west, west of Dugda, gneisses appear as a horst within the boundaries of the coalfield, overthrusting the beds. Rocks of Talchir-series, of fluvio-glacial origin, appear as a wide outcrop in the north-west and as a narrow strip along its northern margin. They consist of a maximum thickness of 240 m. of shales and sandstones with a base of a 15 m thick boulder bed.

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V/VI/VII	11.9—22.6	Medium coking, Class III.
IV	2.1—12.0	Weakly caking, Class III to IV.
III	2.1— 7.3	Weakly caking ; Class IV.
II	2.1—18.3	Weakly caking ; Class III.
I (Bottom)	2.1— 4.3	Weakly caking, Class I to II.

Seams of Raniganj Measures in Jharia Coalfield

In the younger measures of Raniganj, there are nine seams of thickness of 1.2 m and above and their sequence in descending order is given below. Thickness is stated in the bracket.

Three seams of Lohapiti area	(1.2 m each) Class III.
Two seams of Kachara area	(0.9—1.3 m) unclassified.
Bhurangiya	(1.2 m) Class II, semicoking.
Mohuda Top	(2.7—3.7 m) Class II, semi-coking.
Mohuda middle	(1.2 m) Class II, semi coking.
Mohuda bottom	(1.8—2.4 m) Class II, semi-coking.

Reserves

The total of proved, indicated and inferred reserves of coal in the Jharia coalfield, on the basis of assessment in 1963 by the committee on Assessment of Resources of the Coal Council of India, rounded to a million ton are summarised in Table 8.

TABLE 8

Reserves of coal in Jharia coalfield

Depth	Total reserves of workable seams	Reserves of thick seams. Thickness range	
		4.5—9 m	Over 9.0 m
m	Mt	Mt	Mt
0—150	4,816	1,395	1,410
150—300	3,462	1,067	862
300—600	4,403	1,590	958
600—900	2,930	1,124	453
900—1,200	3,006	1,115	505
Total ..	18,617	6,291	4,188
		10,479	

(i) Percentage of reserves in thick seams to total reserves is 56.3.

(ii) Quarriable reserves within coal/overburden ratio of 1 : 5 are 535 Mt.

Quality of Coal

Quality of coal in seams occurring in Jharia coalfield as defined by the proximate analysis of air-dried samples and by other tests is summarised in Table 9.

TABLE 9

Quality of coal seams (Jharia field)

		Barakar Measures Seams I to IX	Seams X to XVIII	Raniganj measures Mahuda, Lohpiti Seams
1.	Proximate Analysis (%)			
	Moisture ..	1—2	1—2	2.0—2.50
	Ash ..	18—40	10—25	14—19
	Volatile matter ..	15—25	20—30	28—34
	Fixed Carbon ..	50—60	55—65	50—56
2.	Clorific Value (Kcal/Kg)	6,500—7,200	6,500—7,500	6,500—7,500
3.	Gray King (L.T.) Coke type ..	C—F	G—G5	E—F
4.	Hardgrove grindability index ..	65—75	65—70	60—65

Raniganj Coalfield

Named after the town of Raniganj in West Bengal round which the field has developed, the Raniganj coalfield is about 200 km northwest of Calcutta and is the eastern-most member of Damodar valley coalfields. The first coal mining venture (1774) started in this coalfield and it produced 30% of India's total coal output in 1965.

Located within latitudes 23° 30'N—23° 55'N and longitudes 86° 35'E—87° 20' E the field occupies an area of nearly 1,550 km² spread mostly in Burdwan District and partly in Bankura, Birbhum and Dhanbad Districts. It is bounded by Archaeans on the north, northwest and south ; eastern boundary extends beneath alluvium. To the south there is a

boundary fault of nearly 3,000 m throw. The field is well served by rail and road communications, the main and Grand Chord line of the Eastern Railway, and the Grand Trunk road passing right through the coalfield. (Fig. 2)

The coalfield was surveyed by Williams, Blanford and Wilson, Walker and Simpson and mapped on toposheets by Gee. D. R. S. Mehta and others revised it in 1951-52.

The Archaean metamorphics surround the field on all sides except east. In the east Gondwana strata extend beneath alluvium and recently a 6 m thick seam has been located in Bankura district. The rocks of Talchir, Damuda and Panchet series of Upper Gondwanas outcrop.

The Gondwana formations here occupy roughly a rectangular tract and show a full succession of Lower Gondwana and younger rocks attaining a maximum thickness of more than 3,000 m as follows :

Formation	Maximum thickness
Supra Panchets	.. 300 m.
Panchets	.. 600 m.
Raniganj	.. 1,000 m.
Ironstone shales	.. 360 m.
Barakars	.. 600 m.
Talchirs	.. 275 m.

Talchirs of fluvioglacial origin comprising boulder bed, sandstones, siltstones and shales are exposed along the northern edge of the western part. Barakars consisting of about 600 m thickness of fresh water sediments of sandstone, shales, carbonaceous shales and coal seams occupy an irregular area of 150 sq. km along the northern part of the coalfields. As we proceed southwards, we come across Ironstone Shales and then a 8—16 km width of Raniganj Measures with the greater width towards the east. These measures attain a thickness of 1,000 m consisting of greyish fine sandstone, carbonaceous shales and coal seams. Next to this tract, in the west, is the wide outcrop of Panchet series comprising micaceous sandstones and shales without any coal. Supra-Panchets overlying Panchets form Panchet hills in south-western part of the field.

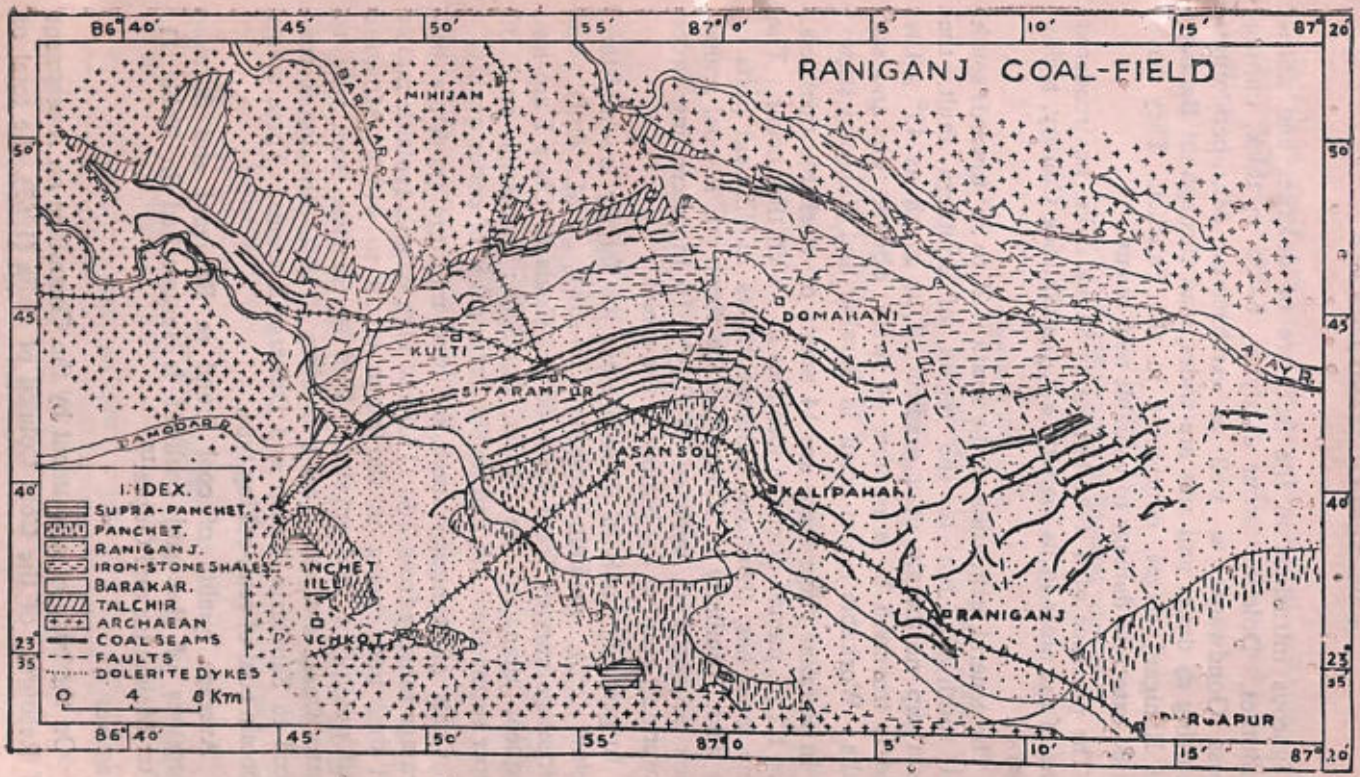

Fig 2. Geological map of Raniganj coalfield.

Igneous intrusions in the field are of dolerites and mica-peridotites. Dolerites occur as dykes, 6—36 m thick, cutting across Gondwanas as well as Archaeans. Mica-peridotites, occurring as dykes and sills are widespread throughout Barakar and Raniganj stages, continuing in the Panchet. They have caused extensive damage to valuable coal seams.

The presence of a dyke known as Salma dyke, has rendered some of the areas in its vicinity, *e.g.* the Ghusick Colliery, highly gassy.

The general inclination of the strata is 2° to 35° towards south and southwest; but in the region of southern fault, the strata dip northernly at a gradient of as much as 45°. The field is traversed by a number of strike, oblique, dip and sag faults. Workable seams occur in Barakar and Raniganj stages.

In Barakar stage there are 14 seams of workable thickness, over 1.2 m, named after the localities in which they occur. The same seam has, therefore, different names in different parts of the field. Similarly in Raniganj stage there are 12 coal seams. Their sequence is depicted in Table 10 in descending order (lowermost seam at the bottom of the table).

In general, coals of Barakar stage are largely low moisture (below 2%), medium volatile (26—36% V.M. dmf), ash percentage varying from 15 to 38%. Coals of Class I are less frequent than in the Jharia Coalfield; generally coal of upper seams (Laikdih and higher horizons) is of better quality.

Coals of Raniganj stage are high moisture (moisture percentage ranging from 2 to 11%) and high volatile (volatile matter on dmf basis being in the range of 39 to 44%). Moisture and quality of coal substance tend to increase from west to east. Coal of Class I (ash+moisture < 19%) are almost confined to Sanctoria-Poniati and Dishergarh seams. These seams contain coal of semi-coking type.

Average quality of coal found in Raniganj and Barakar measures of Raniganj coalfield is summarised in Table 11, after Majumdar and Gupta.[10]

Reserves

On the basis of assessment by the Committee on Assessment of Resources of the Coal Council of India (1963), the total of

proved, inferred and indicated reserves of coal in Raniganj coalfield would work out as in Table 12 (figures rounded to a million ton).

TABLE 10

Sequence of coal horizons in Raniganj coalfield

Index No.	Coal seam	Thickness(m)	Quality
	Raniganj Measures		
R X	Hirakhum Bharatchak-Narsamuda—Kalla.	1.5—2.5	Non coking, Class II to III.
R IX	Gopalpur-Upper Dhadka-Satpukhuria-Ghusick-Siarsol-Upper Kajora-Khandra.	2—6	Non-caking, Class II
R IXA	Ghusick A ..	2.7—3.0	Non-caking, Class II.
	Kushadanga ..	1.8—2.1	Non-caking, Class II.
R VIII	Barachak-Nega-Jameri-Raniganj-Lower Kajora-Jambad-Bowla-Bankola.	5—14	Weakly coking, Class II.
R VII	Lower Dhadka-Narainkuri-Bansra-Sonachora-Banbahal.	2—3.5	Non-caking, Class II.
R VI	Shripur-Toposi-Kenda Chora-Purushottampur.	2—9	Non-caking, Class II.
R V	Bara Dhemo and Raghunathbati-Manoharbahal-Rana-Pariarpur-Satgram-Jote-Janaki-Dobrana-Darula-Mandarboni-Sonpur.	2—9	Semi-cokingto non-caking, Class I to II ; deteriorating to Class III in some areas.
R IVA/IV	Dishergarh-Samla-Dhasul Govindapur.	1—5	Semi-coking to non-coking, Class I to II.
R III	Hatnal-Koithi	2—4 ..	Hatnal semi-coking, Koithi non-caking, Class II.
R IIA/II	Sanctoria-Poniati-Bamanband.	1.5—5.5 ..	Sanctoria is semi-coking and Poniati non-caking, Class I
R I	Taltor-Gangutiya (Bottom most).	1.2—1.8	Weaklycaking, Class II.
	Barakar measures		
B L	Shampur top (local)	2.2	Medium coking, Class III.
B VII	Chanch-Begunia-Shampur	1.0—3.0	Medium coking, Class II.
B VII	Shampur No. 1	3—4	Class II
B L	Shampur No. 2	3.5	——

B VI L	Shampur No. 3—Kharbari	4.3—4.6	Medium coking, Class II to III.
B VI	Shampur No. 4—Ramnagar	4.3—4.6	Medium coking, Class I to II.
B V	Shampur No. 5—Laikdih-Bahira 1—3.	5.5—13	Medium coking, Class II to III.
B IVL	Bahira 4-Salanpur D	1.5	Semi-coking, Class III.
B IV	Gopinathpur-Bahira 5-Salanpur C-Gourangdih-Jamgram-Kasta.	2—13	Semi-coking to non-coking Class III at some places ; near Kasta, Class II.
B III	Bindavanpur-Birsinghpur Salanpur B.	2.7—8.2	Semi-coking, Class III to IV.
B II	Kalimati-Damgaria-Salanpur A.	9—38	Semi-coking, Class III.
B I L	Metadi	2.4	Semi-coking, Class II.
B I	Pusai-Marthadih-Farewell (Bottom most.)	1.5—8.2	Semi-coking Class II.

TABLE 11

Quality of Coal Seams (Raniganj field)

	Barakar Measures	Raniganj Measures Seams Dishergarh Sanctoria, etc.	Raniganj Measures Seams Samla Jambad, etc.
1. Proximate Analysis (%)			
Moisture ..	1.2—1.8	2.2—3.5	3—10
Ash ..	15—30	15—20	13—30
Volatile Matter ..	20—30	35—45	35—45
Fixed Carbon ..	55—65	48—58	48—58
2. Calorific Value (Kcal/kg)	7,000—7,500	6,500—7,000	6,500—7,000
3. Gray King (L.T.) Coke type ..	E—G	E—G1	A—D
4. Hardgrove grindability index ..	50—60	45—55	45—55

TABLE 12 **Reserves of coal in Raniganj coalfield**

Depth range	Reserves of total workable seams (over 0.5 m)	Reserves of thick seams Between 4.5—9.0 m	Reserves of thick seams >9.0 m
m	Mt	Mt	Mt
0—150 ..	7 230	2 859	1 186
150—300 ..	5 925	2 726	966
300—600 ..	6 130	2 046	1 028
600—900 ..	3 628	1 349	862
900—1,200 ..	3 298	1 329	935
Total ..	26 213	10 309	4 977
		15 286	

(i) Percentage of reserves in thick seams to total reserves is 58.3.
(ii) Quarriable reserves within coal/overburden ratio of 1 : 5 are 468 Mt.

Bokaro Coalfield

Named after the river Bokaro flowing through the greater part of the field, Bokaro coalfield in the Hazaribagh district of Bihar State ranks third in importance, next to the Jharia and Raniganj fields.

Extending over a length of 64 km from Siladhar river in the east to Kodme in the west, the Bokaro coalfield has a width of 11.25 km. Lugu hill divides the field in two parts : East Bokaro (occupying an area of 208 sq km) and West Bokaro (extending over 259 sq km) coalfields.

Damodar river with its tributaries Bokaro and Konar provides the drainage. Gomo-Dehri-on-Sone branch of Howrah-Mogulsarai Grand-Chord line of the Eastern Railway and recently Chandrapura-Rourkela line of South Eastern Railway and a number of roads provide the rail and road communications.

The geological work in this field has been done by Williams (1846), Hughes (1867), Fermor (1916-17), A. B. Dutt (1944-51) and Indian Bureau of Mines.

Talchir, Damuda and Panchet series of the Lower Gondwana and Mahadevas of Upper Gondwana systems are exposed

but the rock types of Raniganj stage, Panchet and Mahadeva series are confined to Lugu hill and the neighbourhood while Barren measures and Barakars are met in the East and West Bokaro coalfields.

EAST BOKARO COALFIELD.

Talchirs consist of boulder beds, sandstones and shales. Over the Talchirs are the Barakars consisting of conglomerates, grits, coarse to fine grained sandstones, shales, carbonaceous shales and coal seams, aggregating to a thickness of over 800 m. They contain **all the coal seams** (which are about 15) met in this field. Barren Measures overlying Barakars are met in certain parts of the field and make up a thickness of over 300 m. (Fig. 3)

Sandstone, shales, carbonaceous shales and unworkable coal seams of the Raniganj stage are exposed over a small area between Kodwatanir and Dhendha.

Mica peridotite intrusions appear as dykes and sills in the Barakars and have caused widespread damage.

There are several strike, dip and oblique faults, the important ones being Govindapur-Pichhri fault of a throw of 300 m, Borea, and Dhori-Pichhri faults.

The Barakars dip southernly to the north of the Govindapur fault and northernly to the south. The general dip is 5° to 25° with steeper gradient upto 60° near the faults.

The general sequence of seams in descending order is depicted in Table 13.

These coal seams belonging entirely to Barakar stage are largely low moisture and medium volatile, V.M. (dmf) ranging between 30 and 35%; lower volatiles (22 to 29%) are reported from Phusro, Jarangdih and Govindpur areas. Volatile and moisture contents tend to increase as we proceed from east to west. Coal of Class I is limited to Uchitdih and Uchitdih A seams of Kathara area. At the present reckoning all coals belong to medium coking variety; it is expected that coal of Kargali seam may improve its coking property at depth. After washing, and with suitable techniques of sizing and carbonisation the coal can be used as blend to produce metallurgical coke.

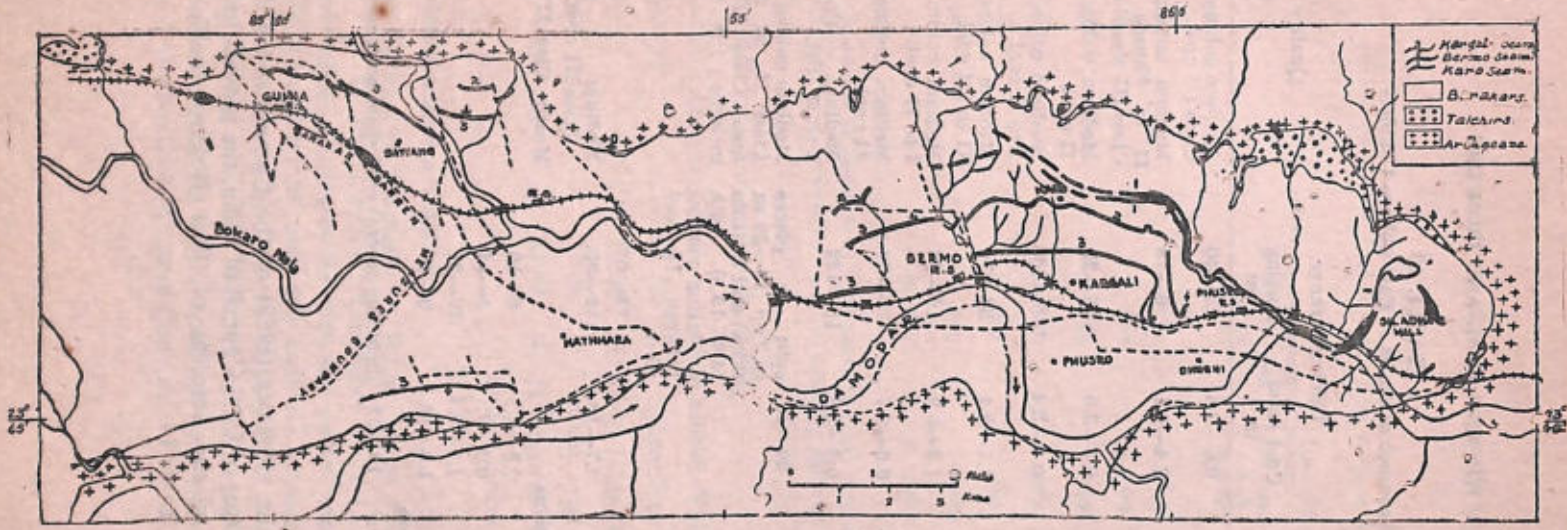

Fig. 3. Geological map of East Bokaro coalfield

TABLE 13

Sequence of seams in East Bokaro Coalfield

Seam	Thickness range		Quality
	Coal seam m	Parting m	
Jarangdih top	0.6—2.4	22—40	Medium coking, Class III coal.
Jarangdih	2.4—6.1	6—18	Medium coking, Class II in Phusro area, Class III elsewhere.
Jarangdih New	0.3—3.0	13—32	Medium coking, Class II coal.
Jarangdih 6′ seam.	0.9—2.4	106—168	Medium caking, Class I to II coal.
Upper Kathara	1.5—2.4	34—76	Medium coking, Class II to III coal.
Kathara-Swang	1.2—5.2	10—36	Medium coking, Class I to II coal.
Uchitdih-Swang-7′ seam-Amlo.	0.6—2.7	5—20	Medium coking, II coal.
Uchitdih-A	0.3—2.4	18—122	Medium coking, Class I to II coal.
Kargali	20—45	split at places in Top (13—20 m thick) and Bottom (4.6—15.5) with a parting varying between 13 and 49 m.	Medium coking, mostly Class II, in Swang area Class III, including dirt bands.
Parting		21—67	
Bermo	3.7—16.8	18—85	Medium coking, Class III coal.
Karo-group of seams			Mostly Class III coal.
Karo I	0.3—2.1	6—16	
Karo II	0.3—2.1	2—17	
Karo III	1.8—9.4	10—21	
Karo-Upper Major/Karo Top.	1.8—18.3	9—46	
Karo lower major/ karo bottom.	18—53	including several coal shale horizons.	

Reserves

Based on the estimates (1963) of the Committee on Assessment of Resources of Coal Council of India, the proved, indicated and inferred *in situ* reserves of East Bokaro coalfield are summarised in Table 14.

TABLE 14

In situ reserves of coal seams in East Bokaro coalfield

Depth range	Total reserves of seams over 0.5 m	Reserves of thick seams	
		Between 4.5 & 9.0 m	Over 9.0 m
m	Mt	Mt	Mt
0—150	1 648	706	783
150—300	1 024	546	572
300—600	1 735	603	679
600—900	1 421	543	693
900—1,200	699	198	424
Total	6 527	2 596	3 351

(i) Percentage of reserves in thick seam to total reserves is 91.1.
(ii) Quarriable reserves within coal/overburden ratio of 1 : 5 are 183 Mt.

Quality of Kargali Seam (Bokaro field).

1. Proximate Analysis (%)		2. Calorific Value (Kcal/Kg)	7,110
Moisture	.. 1.58		
Ash	.. 15.00	3. Gray King (L.T.) Coke type	.. E-G4
Volatile Matter	.. 31.08		
Fixed Carbon	.. 53.92	4. Hardgrove Grindability index.	50-60

The South Karanpura Coalfield

The Karanpura Coalfield in Bihar forms the westernmost patch of the Lower Gondwana Coalfields of the Damodar Valley. Situated between latitudes 23° 38′ and 23°56′ north and longitudes 84°46′ and 85°28′ east, this coalfield extends over an area of 1,425 km². An elongated patch of the Lower Gondwana rocks, covering an area of nearly 200 km² on the south and connected with the major portion of the coalfield by a narrow tongue, is known as the South Karanpura Coalfield. (Fig. 4)

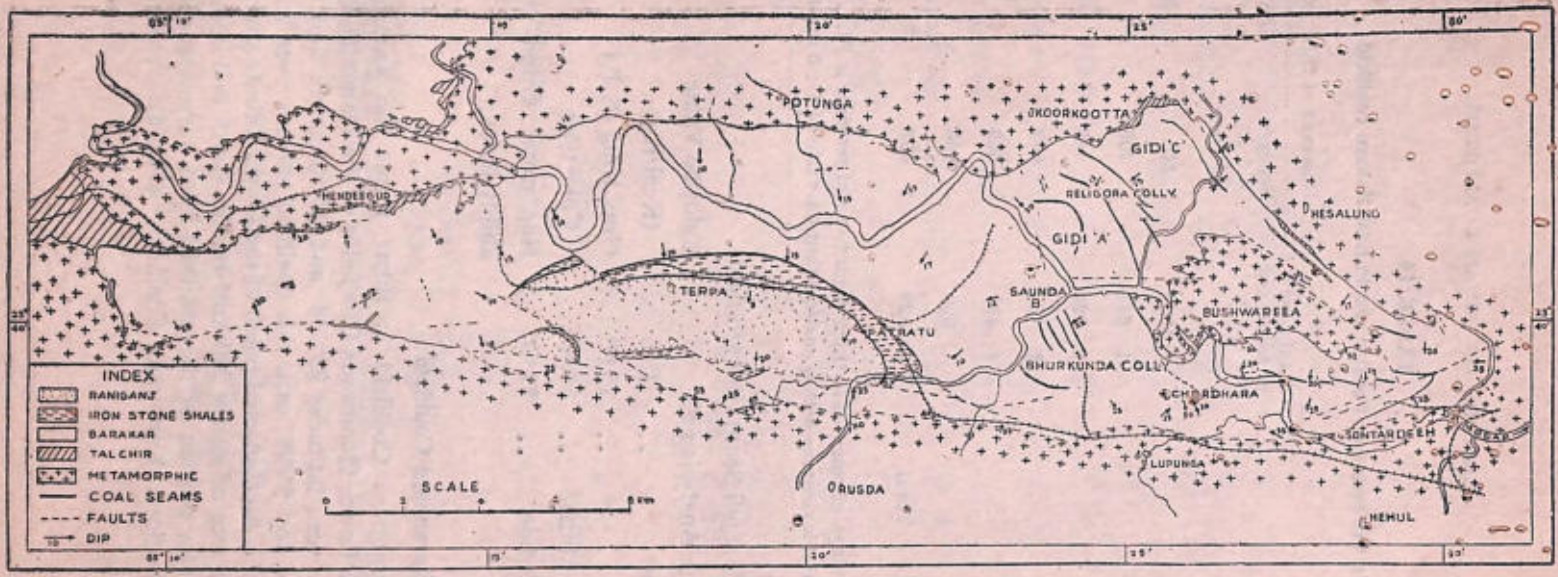

Fig. 4. Geological map of South Karanpura coalfield.

The larger portion of the coalfield on the north, or the North Karanpura Coalfield, is still in an undeveloped state, but the South Karanpura Coalfield is already being exploited and has become a major coal producing field in recent years.

The area comprising North and South Karanpura was surveyed by Williams in 1848, by Hughes in 1868 and by Jowett in 1916. Later on Henry Day did the prospecting and geological mapping in 1934. In 1956-57 and subsequent years, Geological Survey of India and Indian Bureau of Mines carried out detailed prospecting and mapping in some blocks of South Karanpura field.

Rocks belonging to the Barakar, Barren and Ranigunj Measures are present in the South Karanpura area. The Raniganj Measures however appear to be without coal seams, as indicated in the boreholes near Patratu. In the South West area near Patratu, Raniganj have come against Barren Measures and the latter against the Barakars due to erosion of the surface, and deposition of the beds appears to have taken place in a basin. The whole coalfield has in general the structure of a syncline.

Along its southern boundary the area has a major fault known as Chungurha fault with a throw exceeding 125 m, bringing Barakars against metamorphics. The area is highly faulted. There is repetition of outcrops of a number of seams due to strike and dip faults.

There are nearly 36 coal seams most of them outcropping. They are steeply inclined, nearly 1 in 2 to 1 in 5. The area has some very thick seams like Argada (more than 52 m) and Sirka 15 m which are of Grade-I or of Selected-B quality. There are again, seams like Argada A, B, C, D, etc. Sirka seam is available as top and bottom seams. The seams of medium thickness (nearly 4.5 m) like Semana and Kurse are worked at some mines and are of Grade-I. Nakari Seam, Gr. I, worked at some collieries is however only 2 m to 3 m in thickness. Coals of these seams are medium coking and blendable after washing. They generally have high content of moisture (over 2%) and volatiles. The other seams like Bansgarha, Hathidhari Balkudra, Sayal and Saunda are inferior to Grade I.

The mines in these areas are worked mostly at shallow depths. The sequence of the seams is :—

Saunda (top-most)	.. 2 m—5 m thick
Sayal	.. 3 m—6 m
Baikudra	.. 2 m—12.5 m
Kurse	.. 3 m—5 m
Nakari	.. 1.8 m—2.5 m
Semana	.. 3 m—4.8 m
Hathidhari	.. ? m—6 m
Bansgarha	.. 2 m—3 m
Sirka	.. 10 m—15 m
Argada	.. 18 m—52 m

So far drilling in the area has been carried out only upto 500 m depth in some blocks. It is understood that seams as thick as Argada, may be available below the Argada Seam.

Dolerite dykes are encountered in some areas near Sirka and Gidi. Mica peridotite sills appear in Sayal area.

The river Damodar flows throughout the coalfield which is served by a railway line from Barkakana to Dehri-on-sone (Eastern Railway). A number of mines is owned by N. C. D. C.

Proximate analysis of important seams is given below

Seam (colliery)	Moisture %	Volatile matter %	Fixed Carbon %	Ash %	Calorific value, Kcal/kg
Sirka (Sirka)	1.90	31.20	51.20	15.40	6522
Argada (Argada)	3.60	33.12	48.34	14.94	6533

The Pench Kanhan Valley Coalfield (M.P.)

The Pench Kanhan Valley Coalfield (Fig. 5) lies to the north-west of Chhindwara within Chhindwara district extending from Sirgora in the east to Nonkharak in the west, and is nearly 55 Km in length from east to west and 8 to 10 Km in width. It lies between latitudes 22° 9′ N, 22° 24′ N. and between longitudes 78° 19′ E, 78° 55′ E. The main rivers are the Pench and the Kanhan. The general altitude of the valley is 600 m above M.S.I. The western part of the coalfield

as compared to other parts is somewhat hilly. A narrow gauge railway line from Chhindwara to Barkuhi and a broad gauge line from Amla to Parasia are the two railway lines, serving the two coalfields.

Sequence of the rock formations is given below.

Basaltic flows and intrusive rocks (topmost)—Deccan Trap.

Sandstones and clays	LOWER GONDWANA	Motur Stage, (corresponds to Barren Measures of Jharia field).
Sand stones, shales and coal seams		Barakar Stage.
Boulder beds, sand stones and shales.		Talchir Series.
Gneisses, schists & granites		Archaean.

The archaean metamorphic rocks appear in the south of the coalfield. Talchir series rocks are exposed near Khirsadoh railway station. The Barakars occur in an isolated patch near Sirgora in the east and in a long tract extending east-west from Chhinda to 4 Km west of Jamkunda. A strip of the Barakars again extends from Junnordeo railway station to west of Rakhikol. The thickness of Barakars is not fully known, but from drill holes it is expected to be over 115 m. The general trend of the Barakars in the area is W.—E. and the dips vary from 1 in 10 in eastern part to 1 in 5 in western areas. The seams are available near the surface and upto a depth of maximum 150 m.

There are a number of faults in this area. Near Jamkunda is a fault which brings Motur rocks (Barren Measures) against Barakars. A major fault, known as Barkuhi fault, extends in ENE—WSW direction from a Deccan trap hill near Chandameta to Ambara and further upto Jamkunda. Its throw varies from 21 m near Bhamori to 100 m near Ambara. Near Damua is a fault known as Damua fault with 100 m throw which extends from Rakhikol colliery to Panara, throwing Talchirs against Barakars. A few dolerite dykes also exist in the area.

PENCH AND KANHAN VALLEY COALFIELD

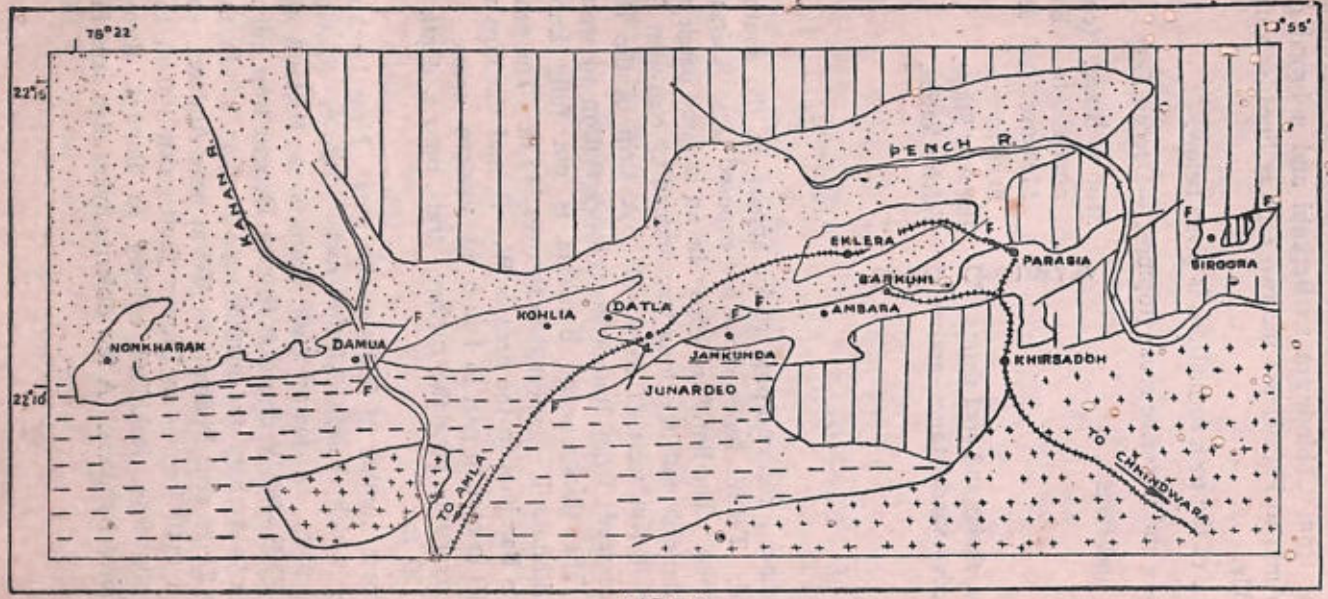

Fig. 5. (Not to scale)

The number of workable seams so far proved is three. The top seam, 1.8 to 6 m. thick, has been worked and practically exhausted in most of the collieries. It is non-coking in the eastern side, but semi-coking in the western region. The middle seam is 1.1 m to 1.5 m thick and the bottom seam, 1.2 m to 1.8 m. The coal seams contain iron pyrites and the surface stocks are liable to spontaneous heating readily. The quality of coal is Grade II and in some areas Grade III.

Typical analyses of Kanhan and Pench valley coals are given below.

Seam	Moisture %	V.M. %	F.C. %	Ash %	Cal. value Kcal/kg
Top or Main (Chandameta)	7.48	31.24	44.24	17.04	5688
Bottom (Palachauri)	4.20	33.10	49.40	17.50	6372

The Pench valley area has been better developed than the Kanhan Valley region.

The important towns in the area are Junnordeo and Parasia. A large number of collieries is under the management of Messrs. Shaw Wallace & Co. Ltd.

Pranhita—Godavari Valley Coalfield

(Covering the field of operation of Singareni group of collieries).

This field, a south-east continuation of Wardha valley coalfield, lies between latitudes, 17°N and 20°N, and longitudes, 78°E and 81°E, and coal-bearing rocks cover an area of nearly 9,000 Km² (Fig. 6). The lower Gondwana rocks developed in the area have the following sequence and thickness :—

Formation		Maximum thickness
Kamthi beds	..	45 m
Barakars	..	610 m
Talchirs	..	60 m
..... unconfirmity		
Vindhyas	Pakhal limestones and shales	
..... unconformity		
Archaens		

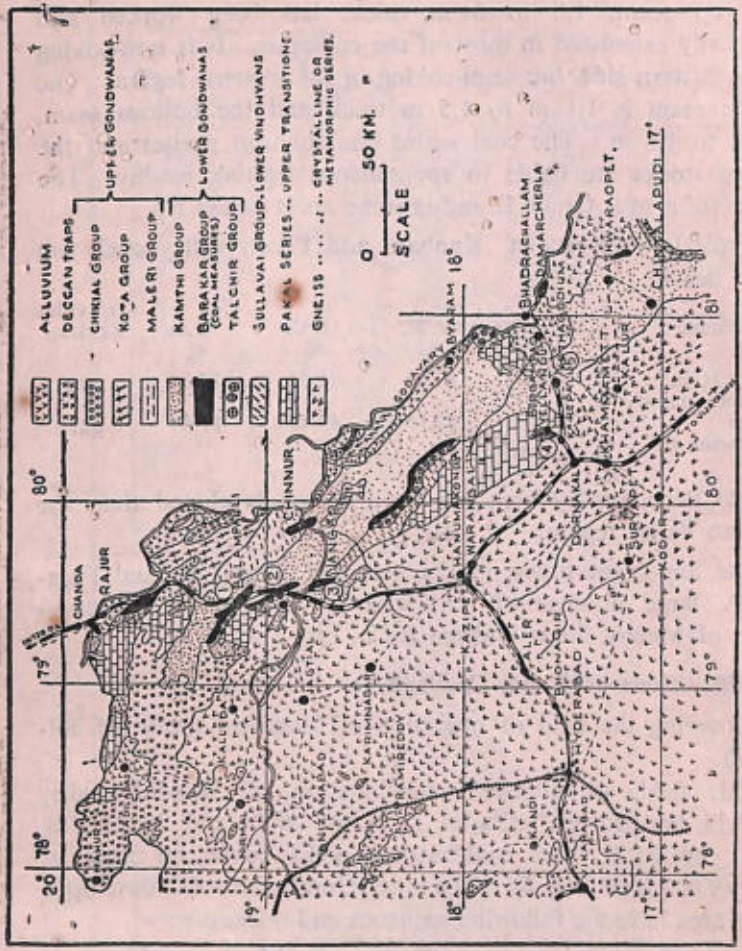

Fig. 6. Geoglogical sketch-map of Pranhita—Godavari Valley coalfield. 1. Tandur and Kanala coalfield. 2. North Godavari coalfield. 3. South Godavari coalfield. 4. Yellandu coalfield. 5. Kothagudium coalfield.

Kamthi sandstones outcrop throughout the field with only a few exposures of Barakars. Barakars must have been eroded considerably before Kamthis were deposited. There is little igneous activity in the area.

The Singareni Collieries Co. Ltd. holds a lease over a considerable area in this field and have prospected and proved the existence of workable coal seams over some 2,000 km² of the area which includes, from the north :—

(1) Asifabad (2) Tandur (3) Kanala (4) North Godavari (5) Sarrangapalli (6) Chinnur (7) South Godavari (8) Kamawaram (9) Singareni (10) Kothagudium (11) Poloncha (12) Siwaram (13) Allapalli. Out of these areas, the geology of the producing areas is given below.

Tandur and Kanala area is 182 km² in extent, lies about 2.4 km to the north-east of Belampalli railway station on Kazipeth Ballarshah railway line. The workable seams in the area are :—

Top seam (or Salarjung seam)	..	8.2 m thick
Parting	..	27 m
Bottom seam (or Ross seam)	..	2.1 m

The bottom section of Salarjung seam, and the whole thickness of Ross seam are being worked. Coal averages 16% in ash content and 7.5% in moisture.

North Godavari area is about 6.4 km south-West of the Tandur area and covers an area of 75 km.². The sequence of workable seams in this area is as follows :—

Name of strata	Thickness
No. 1 seam	2.4 to 3.0 m
Parting	82 m
No. 2 seam	4.0 to 4.5 m
Parting	26 m
No. 3 seam	2.1 to 3.3 m
Parting	7.6 m
No. 4 seam	1.8 to 2.4 m

South Godavari occupies an area of 94.5 km² and lies about 11 km northeast of Ramagundam railway station on the Ballarshah-Kazipeth railway line. The area contains four seams in the following order :—

Name of seam	Thickness
No. 1 seam	6.4 m
Parting	23.0 m
No. 2 seam	3.5 m
Parting	64.0 m
No. 3 seam	7.0 m
Parting	13.7 m
No. 4 seam	4.8 m

Kothagudium has an area of 485 km², 54 km north-east of Dornakal station. The sequence of major seams in a descending order is as follows :

Seam	Thickness
Top seam (or thick seam or Queen seam)	10 to 30 m
Parting	22 to 48 m
Bottom seam (or King seam)	2.2 m in the north to 31 m in the south, better quality.

Situated 38 km north-east of Kothagudium, Singareni area covers 48 km² and contains the following sequence of coal horizons :—

Top seam (or Queen seam) ..	10 to 31 m
Parting	60 m
Bottom seam (or King seam)	2.4 m

Reserves as proved by drilling upto 1961 are estimated as follows :—

Area	Reserves of coal in million tons. Upto 300 m Depth.	610 m depth, probable.
Kathagudium	500	729
Yellandu	30	30
Tandur	100	200
North Godavari	140	170
South Godavari	140	200
	910	1329

The coals are fairly high in moisture (between 6 and 12%, as received) and are non-coking or yield soft coke. A typical analysis of coal of the important seams is given below :—

Seam (Colliery)	Moisture %	Volatile Matter %	Fixed Carbon %	Ash %	Calorific value (dry) Kcals/Kg
King (Singareni)	7.18	28.75	50.30	13.77	5,977
Salarjung (Tandur)	5.15	37.10	49.17	13.73	6,747
Bottom seam (Sasti)	8.92	38.11	41.29	11.68	6,065

REFERENCES

1. LOWRY, H. H.,—"Chemistry of Coal utilization", *Supplementary Volume*, 1963.
2. FOX, C. S.,—"Natural history of Indian coal"—*Memoirs of Geological Survey of India LVII*, 1931.
3. THIESSEN, R.,—Coal Age 18, 1920 ; J. Geol. 28, 1920.
4. STOPES, M. C.,—Proc. Roy. Soc. 90B, 1919.
5. CHANDRA, D.,—"Application of coal microscopy", *Journal of Mines, Metals and Fuels, November*, 1965.
6. GEE, E. R.,—"Bulletins of Economic Minerals—Coal", *Records of the Geological Survey of India*, 1945.
7. FOX, C. S.,—"The lower Gondwana Coalfields of India", *Memoirs of Geological Survey of India—Vol.* LIX, 1939.
8. KRISHNAN, M. S.,—"Geology of India and Burma", *Higginbothams Ltd., Madras*, 1960.
9. COAL COUNCIL OF INDIA,—"Report of the Committee on Assessment of Resources, Vol. I, II & III 1963.
10. MAJUMDAR, S. and GUPTA, S.,—"Resources and qualities of coals in India"—*Journal of Mines, Metals and Fuels, Special issue*, 1964.
11. FRANCIS, WILFRED,—"Coal" *Edward Arnold Ltd., London*, 1954.
12. BAGCHI, S., BAGCHI, D., MALIK, P. C., "Reserves of Indian Coals" *Journal, Mines, Metals, Fuels", Aug.* 1966.
13. MEHTA, D. R. S.,—"A revision of the geology and coal measures of the Raniganj coalfield", *Memoirs of Geological Survey of India, Vol.* 84 *Part I*, 1956.
14. MEHTA, D. R. S., and MURTHY, B. R. N.,—"A revision of the geology and coal resources of the Jharia coalfield", *Memoirs of Geological Survey of India, Vol.* 84, *Part II*.
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8. KRISHNAN, M. S.,—"Geology of India and Burma", *Higginbothams Ltd., Madras*, 1960.
9. COAL COUNCIL OF INDIA,—"Report of the Committee on Assessment of Resources, Vol. I, II & III 1963.
10. MAJUMDAR, S. and GUPTA, S.,—"Resources and qualities of coals in India"—*Journal of Mines, Metals and Fuels, Special issue*, 1964.
11. FRANCIS, WILFRED,—"Coal" *Edward Arnold Ltd., London*, 1954.
12. BAGCHI, S., BAGCHI, D., MALIK, P. C., "Reserves of Indian Coals" *Journal, Mines, Metals, Fuels", Aug.* 1966.
13. MEHTA, D. R. S.,—"A revision of the geology and coal measures of the Raniganj coalfield", *Memoirs of Geological Survey of India, Vol.* 84 *Part I*, 1956.
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15. NARGUNDKAR, S. K.,—"Coal Industry in the South—The Last Decade."

CHAPTER 4

IRON ORE DEPOSITS OF INDIA

1. Introduction. 2. Minerals of Iron. 3. Classification and Origin. 4. Distribution. 5. Types of Ore. 6. Reserves.

1. INTRODUCTION

India is known to possess fairly large reserves of iron ores of good quality for its domestic requirement as well as for export and there should be no alarm far into the distant future of this nation for the shortage of this key mineral of modern industry, even if the home and export demands show upward trend by several folds. In India there are vast regions underlain with iron ore deposits of good quality and quantity in addition to ample reserves of lower grade iron ores, which, at present, are commercially unattractive and their industrial utilization must wait until the favourable factors, such as better methods of metallurgy, cheaper means of transportation, favourable marketing conditions and competitive price in the national and international trade spheres, call for their exploitation.

2. MINERALS OF IRON

The chief minerals of iron are the following :

Mineral	*Composition*	*Iron per cent*
Magnetite	Fe_3O_4	72.4
Hematite	Fe_2O_3	70.0
Turgite	$2Fe_2O_3,H_2O$	66.3
Goethite	Fe_2O_3, H_2O	62.9
Limonite	Fe_2O_3, nH_2O	59.9
Siderite	$FeCO_3$	48.3
Pyrite	FeS_2	46.6
Pyrrhotite	$Fe_n S_{n+1}$	38+
Ilmenite	$Fe TiO_2$	36.8
Greenalite, Chamosite, etc.	Hydrous iron silicates.	35

Among the minerals of iron, four are important as iron ores, and in order of their decreasing industrial utility in this country, they can be arranged as hematite, magnetite, limonite and siderite.

Hematite (Fe_2O_3).

It is the most common iron ore and by far the only ore of iron being presently smelted in domestic blast furnaces. Pure hematite contains 70 per cent of iron and 30 per cent of oxygen by weight. Often, the ore is fed directly into the blast furnaces in its natural state, but at times, it is necessary to upgrade the ore by chemical and physical treatments before despatching to the steel plants. Apart from its chief use in iron metallurgy, the mineral finds application in the manufacture of red paint and red colouring materials.

Magnetite (Fe_3O_4).

It is the richest ore of iron and, theoretically, contains 72.4 per cent of iron and 27.6 per cent of oxygen by weight. India is well endowed with sizable reserves of magnetite ore carrying appreciable quantities of titanium and vanadium and, at places, chromium, the latter mainly in ores from Mysore ; such magnetites are, therefore, of strategic importance and their export is prohibited by the Government of India.

Limonite ($Fe_2O_3.nH_2O$)

This is not an independent mineral. The generic name limonite is often used for mixtures of goethite (α—Fe^{3+} OOH) and lepidocrocite (γ—Fe^{3+} OOH) ; these two mineral species are chemically identical, and crystallise in the orthorhombic system but their crystal structures are different. Pure limonite would contain 59.9 per cent iron, 25.7 per cent oxygen and 14.4 per cent water. Although limonite makes up a substantial part of the total Indian iron ore reserves, it does not find any application for the present in our iron and steel industry because of the abundance of high grade hematite ore in the country. However, small quantities of yellow variety of limonite, called yellow ochre, is used in paints, oil-cloth and lenoleum industry.

Siderite ($FeCO_3$).

It is an iron carbonate and is constituted by 48.3 per cent iron, 10.3 per cent carbon, and 41.4 per cent oxygen. When mixed with clay, sand and organic matter it is commonly called *clay ironstone*, or *blackband*, when black. In some countries of Europe siderite is mined as an ore for smelting iron, but, so far, it has not been of much industrial utility at home. Nevertheless, it had been worked for some time in the past for feeding the local pig iron furnaces of the Barakar Iron Works and its successor, the Bengal Iron and Steel Co., later amalgamated with Indian Iron and Steel Co.

3. CLASSIFICATION AND ORIGIN

Commercial bodies of iron ore in India have been derived mainly from igneous, sedimentary, metamorphic and residual processes of mineral formation. With a view to compare and standardise the Indian iron ore deposits with those of other parts of the world, they may be classified, on the basis of their mineralogy and mode of occurrence, as follows :—

(1) Banded hematite deposits genetically related to the Banded Hematite Quartzite formation of Pre-Cambrian age.

(2) Banded magnetite deposits—highly metamorphosed equivalents of Type 1.

(3) Vanadium-bearing titaniferous magnetite bodies associated with basic igneous rocks.

(4) Apatite-magnetite bodies localized along the Singhbhum Thrust Zone.

(5) Cavity-filled hematite deposits.

(6) Sedimentary iron ore beds of sideritic or limonitic composition in Gondwana and Tertiary formations.

(7) Residual iron ore concentrations.

(8) Surficial deposits of limonite.

(9) Secondarily enriched iron ore deposits formed by circulating waters.

Banded Hematite Deposits.

The iron ore mined from banded hematite type of deposits is the backbone of our iron industry and meets the entire quantum of export commitments and would continue to be the principal source of iron ore for several decades. Geologically, the deposits are associated with Pre-Cambrian banded iron ore formations that, when subjected to low grade metamorphism, are composed of alternating bands of silica and iron oxide ; the former generally comprising quartz, chert or jasper, and the latter, predominantly hematite with minor quantity of magnetite and martite.

In India the banded iron ore deposits are abundantly developed in Singhbhum (Bihar), Keonjhar, Bonai and Mayurbhanj districts (Orissa), Bastar and Drug districts (M.P.), Chanda (Maharashtra), Bellary and Sandur (Mysore), and Bicholim (Goa) together with minor occurrences in the States of Madras and Rajasthan. These deposits are similar in age, stratigraphical relationship, and lithological association, to the Pre-Cambrian banded iron ore formations of Lake Superior region (Jaspilite), Canada, South Africa (Calico rock), Brazil (Itabirite), Venezuela, Rhodesia, South Russia, etc.

A good deal of controversy surrounds the origin of Pre-Cambrian banded iron ore bodies of India and similar deposit in other parts of the world. The main debatable points of the problem are : the origin of banding, the source of enormous amount of iron and the agencies involved in the precipitation of iron from solution. But generally, they are considered special forms of chemically precipitated sediments, the deposited material having been derived from the weathering of adjacent continental masses and transported in solution to the sea by river water.[6,8,10] Notwithstanding the prevailing belief, there are some who maintain that weathering of the average crust of the earth, excepting ferruginous rocks, cannot produce solutions capable of depositing such iron-rich formations and according to them source of iron and silica is to be sought in volcanic exhalations and in the reaction of sea water with submarine lava flows.[20]

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A good deal of controversy surrounds the origin of Pre-Cambrian banded iron ore bodies of India and similar deposit in other parts of the world. The main debatable points of the problem are : the origin of banding, the source of enormous amount of iron and the agencies involved in the precipitation of iron from solution. But generally, they are considered special forms of chemically precipitated sediments, the deposited material having been derived from the weathering of adjacent continental masses and transported in solution to the sea by river water.[6,8,10] Notwithstanding the prevailing belief, there are some who maintain that weathering of the average crust of the earth, excepting ferruginous rocks, cannot produce solutions capable of depositing such iron-rich formations and according to them source of iron and silica is to be sought in volcanic exhalations and in the reaction of sea water with submarine lava flows.[20]

As regards the banding in the ores, it is assumed as a primary sedimentary feature and is usually considered to have originated

from the rhythmic deposition of one or the other constituent (iron or silica) owing to seasonal fluctuations in their supply, iron in the rainy season and silica in the dry season.[18] Some workers have also held that the primary banded feature has been caused by local or general variation in the proportion of iron oxide to silica in the colloidal sediment with almost continuous supply of the former and an intermittent supply of the latter so as to produce bands of pure hematite alternating with bands of ferruginous silica.[7] Possibility of the origin of banded structure by metamorphism and from intermittent supplies of mixed material with more rapid settlement of the iron than the siliceous colloids has also found favour with a few others.[15]

Opinions are also divided on the agency that brought about the precipitation of the chemical constituents from the solutions. Factors, such as presence of organic matter, like algae and bacteria[8], the nature and amount of electrolyte[15], hydrogen ion-concentration[10], oppositely-charged colloids,[4,22] etc. have been postulated to be responsible for the deposition of iron and silica from the solutions.

Although after deposition the sediments were severely deformed into a series of intricate folds with simultaneous production of numerous faults, lamination has since remained well-preserved through the geological ages. In the field close relationship is often discernible between the regional structure and the richness of ore in quality and quantity; commercial bodies of iron ores are localized in the noses of the major fold structures, of which anticlinal portions have been weathered and the ore is at present found preserved mainly in pitching synclinal troughs.

Banded Magnetite Deposits.

In certain areas, the banded hematite formations have been subjected to higher grade of metamorphism, which has locally converted them into banded magnetite-quartzite rocks, occasionally containing small quantities of grunerite-cummingtonite varieties of amphibole. The magnetite bodies in them are regarded as "metamorphosed." Where complete conversion of hematite into magnetite has not been possible, hematite-magnetite association has developed in many locali-

ties of Pre-Cambrian sedimentary formations. Typical occurrences of banded magnetite deposits come from the districts of Salem and Tiruchirapalli, where they are known to form some of the important hills like Kanjamalai, Godumalai, Talamalai, etc.; their constituent rock formations are similar and of the same age as those of Bihar and M.P.

Vanadium-bearing Titaniferous Magnetite Deposits.

In their areal distribution and economic concentrations the magnetite deposits in India are much less restricted as compared to the banded hematite types of orebodies described in the preceding section. They are genetically associated with igneous rocks of basic composition, such as gabbros, norites, and anorthosites. Generally, the ore in them is titanian magnetite containing a fairly high quantity of vanadium; titaniferous magnetite from Tumkur district in Mysore is reported to contain chromium also. Widespread occurrences of magnetite deposits in southern Singhbhum (Bihar), Mayurbhanj and Keonjhar (Orissa), and southern districts of Mysore belong to this class.

To the earlier workers these bodies seemed to have originated by early crystallization and magmatic segregation of the magnetite into concordant masses or stratified sheets at the place of settling within the parent igneous bodies. But since the beginning of the present century arduous research involving careful field and laboratory studies by many investigators has indicated that these ores are extreme differentiates of the basic magma, in which the residual product of differentiation was the iron-rich hydrous solutions of late magmatic or even of hydro-thermal origin[21], the orebodies and the host rock, in part, showing textural and structural evidence of deformation during later stages of consolidation. Field features exhibited by them include transgressive character of the orebodies, control of localization by fractures and other structural breaks in the host rocks, and intensive hydrothermal wall-rock alteration with receding intensities away from the margins of the ore zones,—all of which are convincing evidence in favour of their hydrothermal origin. The orebodies of this class are typical in their mode of occurrence in forms such as fish-hook

bodies, lath, plank, or pod-shaped lenses, veins and irregular masses having cross-cutting disposition with the surrounding rocks, indicating thereby that they were emplaced later than the host rocks.

Apatite-Magnetite Deposits along the Singhbhum Thrust Zone.

The apatite-magnetite orebodies, that are localised along the Singhbhum Thrust Zone, mineralogically form a distinctive category because of the intimate association of magnetite and apatite, with biotite, chlorite and subordinate quartz as chief gangue minerals. The ore is rich in phosphorus and does not carry appreciable amounts of titanium and vanadium. The geologists, who have conducted investigations of these orebodies in the past, generally support the thesis that they are genetically related to the host granite and have originated by the pneumatolytic hydrothermal process of ore formation. The deposits have been exploited for winning apatite for phosphate fertiliser industry. The ore is also technically important for producing pig iron which would obviously contain relatively higher quantity of phosphorus.

Cavity-Filled Hematite Deposits.

To the structurally controlled hematite bodies localized in a ENE-WSW trending fault zone through the rocks of Cuddapah system in Veldurti and Ramallakota areas, Kurnool district, Andhra Pradesh, Krishnan[14] has given a separate identity but they may be logically called as cavity filled deposits and classified with hydrothermal iron ore deposits, as they seem to have originated by the precipitation of iron oxide from highly ferruginous solutions traversing the fault zone. In the area, the orebodies form small hillocks and extend intermittently over a length of about 8-10 km.

Sedimentary Deposits.

Sedimentary deposits of siderite and ironstone (the latter term applied to the iron rich formation) are not only restricted in their geographical distribution but are also economically unattractive in the face of the cheaply available high grade iron ore from other types of iron ore bodies. However, in the past they

were extensively exploited for iron smelting in many parts of the country.

In India fairly good deposits of clay ironstone and blackband varieties of siderite occur in the Barren Measures (Iron Stone Shale Stage) of the Damuda series in the Raniganj coal fields, while their minor occurrences are also found in the red sandstones of Kamthi and Mahadeva age in Auranga coalfield in Bihar. In these formations the orebodies are in the form of layers and irregular lenses of concretionary masses of sideritic composition that are locally rich enough to be used as iron ore. Locally, at or near the surface the ore outcrops are oxidized and enriched giving rise to patches of limonite, while at depth the primary siderite still persists.

Ironstone deposits of sedimentary origin are developed in the Inter-Trappean beds of the Rajmahal hills. Tertiary formations in the States of U. P., Assam, Madras and Kerala are also known to contain ironstone in workable quantity. Most of them were formerly used for the manufacture of iron in their respective areas of occurrence.

Residual Iron Ore Concentrations.

Since geological past the process of differential decay and leaching has been vigorously operative for the formation of residual iron orebodies practically in all the regions characterized by the dominance of Pre-Cambrian banded hematite-quartzite and other ferruginous formations. The intimate positional relationship between such orebodies and the Banded Hematite Quartzite suggest that the ores formed by removal of silica from the parent rock under favourable physiographic and tropical climatic conditions of weathering and the iron residues left behind, developed *in situ* into irregular concentrations or pockets of residual iron ores, which, subsequent to their formation, have possibly been compacted and further enriched with the introduction of more iron probably through meteoric waters. Occurrences of residual type of iron orebodies are abundantly scattered in many districts of Bihar, Orissa, Madhya Pradesh, Goa, etc. that are supported by the banded hematite rock formations.

Surficial Deposits of Limonite

Lateritic type of iron orebodies result whenever highly ferruginous rocks, such as gneisses, schists, basic igneous rocks etc., are chemically weathered for prolonged period of time under tropical climate and over subdued topography. Under such conditions all constituents except alumina and iron may be carried away in solution, leaving a residue composed chiefly of either yellow or red limonite ; and brownish hematite and reddish yellow goethite which at times may carry titanium, nickel, manganese, or chromium as minor constituents in varying proportions. Large stretches of Deccan Trap, the gneisses along the Western Ghats and the Chhotanagpur Plateau in Bihar, and schists in Mysore and Rajasthan furnish fairly large deposits of iron-rich laterites.

Secondarily Enriched Iron Ore Deposits Formed by Circulating Waters :

Discussing the genesis of iron ore deposits of Noamundi in the Singhbhum district of Bihar, and of Joda East in the Keonjhar district of Orissa, some of the TISCO's geologists, including Wazalwar[21] and others,[4,5] have advocated that the iron orebodies in these areas have originated through the process of leaching and enrichment of iron by circulating meteoric waters under shallow depths of about 30 m from the surface, generally in the severely fractured anticlinal and less commonly in the synclinal zones.

These areas are predominantly occupied by huge thicknesses of ferruginous shales interbanded with hematite, banded hematite jasper, and chert. According to these workers the first stage in the development of iron ores began with the solution and vigorous mobilization of iron and to some extent silica from the banded hematite shales by acidic waters mingled with activated oxygen. Later on, downward and lateral migration of the mobilized constituents (iron and silica) followed with the result that considerable quantities of iron were added to the underlying ferruginous shales to form enriched orebodies of iron, and released silica was redeposited in the intermediate zone to bring about partial silicification of the shales.

If the theory of the above-mentioned workers is correct, these bodies of iron ores may be classified with those formed by concentration of substances by means of circulating waters.

4. DISTRIBUTION

Iron ore deposits in India range in age from Pre-Cambrian to Tertiary and in their geographical distribution they are known to occur widely throughout the country, so much so that in a few States the deposits are amongst the world's largest mines noted for their rich hematite orebodies. Worth-mentioning geological and geographical occurrences are listed in Table 1.

TABLE 1

Geological and Geographical distribution of Iron Ore in India

Formation	Type of deposit	State	Locality
Pre-Cambrian		Bihar	— Singhbhum
		Orissa	— Bonai, Keonjhar and Mayurbhanj,
		Kashmir	—Poonch.
		Madhya Pradesh	—Bastar, Durg and Jabalpur,
Banded Iron formation.	Mainly Hematite (Massive, shaly, powdery, etc.).	Maharashtra	Chanda, Dharwar and Ratnagiri.
		Mysore	— Bellary, Sandur, Shimoga and Chikmagalore.
		Goa	— Bicholim, etc.
Banded Iron formation (metamorphosed).	Magnetite quartzites.	Andhra Pradesh.	— Guntur.
		Madras	— Salem and Tiruchirapally.
		Mysore	—Shimoga.
		Himachal Pradesh.	Mandi.
Basic and ultrabasic rocks.	Vanadiferous Titanian Magnetites.	Bihar	— S.E. Singhbhum
		Orissa	— Mayurbhanj
		Mysore	— Southern districts.
Acidic Intrusive (granodiorite).	Apatite-magnetite	Bihar	— Singhbhum.
		Orissa	— Mayurbhanj.

Formation	Type of deposit	State	Locality
Granites	Magnetites (residual).	Assam	Jaintia Hill.
Cuddapah		Madhya Pradesh.	Bijawar, Gwalior, Indore and Rewa.
Bijawar Gwalior Cuddapah.	Hematite and ferruginous quartzites.	Punjab	Mohindergarh.
		Rajasthan	Jhunjhunu, Sikar and Jaipur.
		Andhra Pradesh.	Cuddapah.
Gondwana			
Mahadeva	Ironstone and siderite.	West Bengal Bihar.	Birbhum Auranga coalfield.
Ironstone Shales.	Ironstone and siderite.	West Bengal	Raniganj coalfield.
Triassic (?)			
Sirban Limestone.	Hematite and Limestone.	Kashmir	Udampur.
Jurassic			
Rajmahal trap (intertrappean beds).	Iron-stone	West Bengal Bihar	Birbhum — Rajmahal.
Tertiary			
Miocene and Eocene.	Ironstone	Assam	N.E. district.
		U.P.	Kumaon.
		Kerala.	Travancore, Malabar.
Quarternary	Laterite	Practically in all States.	Widely scattered outcrops in a number of districts.

5. TYPES OF IRON ORE

As stated earlier, India's output of iron ores is derived entirely from the orebodies associated with banded hematite formations. The ores mined from them are of different kinds and show considerable variation in their chemical composition and physical characteristics, in regard to which they are divisible into the following types :—

Massive ore :—It is hard and compact reddish-grey variety of hematite that is often fine-grained and homogeneous near the surface. The ore is usually of high grade quality containing near about 70 per cent iron. As the ore is more resistant to weathering, its floats are often found strewn on and nearabout the peripheral zones around the hill slopes supported by the exposures of iron ore bodies.

Laminated ore :—This variety of iron ores is characterised by biscuity colour, slightly coarser texture and well-preserved bedding planes, often filled with powdery and shaly iron material. The ore may sometimes exhibit folding and crumpling and be siliceous owing to the presence of veins of secondary silica. That it is derived from leaching of silica from the banded hematite quartzites is indicated by several evidences, such as gradual transition of the orebodies laterally into unaltered primary rock, presence of the relict structures, etc.

Shaly ore :—It is purple to chocolate in colour, silky in lustre and shaly in appearance. The iron content of the ore may be low or high but usually with fair amount of silica and alumina. At places it seems to have been derived through the process of enrichment of shales in iron from circulating waters.

Flaky ore :—Presence of fine flakes of hematite interspersed with specks of kaolinous mineral and veinlets of hematite and fibrous goethite are the diognostic features of this type of ore, which is also friable in nature.

Powdery ore (Blue Dust) :—As the name implies, the powdery ore occurs as lens and pockets of iron sands, generally in shades of dark bluish grey; hence it is also named as blue dust. It is known to occur as pockets and lenses practically in all the large field. of iron ores in India. Its concentrations are seen grading laterally into one or the other types of ore,

whereas towards their bottoms *lithomarge becomes increasingly* noteworthy. It has been held by several workers that the blue dust, in all probability, represents the residues from intensive leaching of the banded hematite formations, from which practically all undesirable material (silica and alumina) has been removed by meteoric waters with the consequence that a blue grey powder of almost pure hematite results containing on an average 66 to 67 per cent of iron.

Fig. 1. Important industrial centres for coal and iron ore mining industry in Eastern India (W. Bengal, Bihar, Orissa and Madhya Pradesh)

An important iron ore field in India is Goa which produces about 7 million tonnes of iron ore, *i.e.* nearly 28% of the total iron ore production of India including Goa.* There is very little published literature available about these deposits and, therefore, a brief description has been given below.

* 1966 iron ore production of India—26·51 million tonnes. (Orissa—6·72 Mt ; Goa—6·53 Mt ; Bihar—6·37 Mt ; M.P.—4·14 Mt ; Mysore—3·34 Mt).

6. GOA IRON ORE INDUSTRY

Situated about 350 km south of Bombay on the west coast, the territory of Goa covers an area of 3,600 km² and is 105 km in length and 60 km in width. The landscape is mostly hilly with forests, the hill ranges forming part of the Western Ghats. The altitude varies from sea level to a maximum of 1,700 m above m.s.l.. It enjoys a warm tropical climate (average temperature 27°C) and an annual rainfall of nearly 400 cm during the monsoon months of June to October. The main rivers are Mandovi and Zuari, both flowing into the Arabian sea and navigable by 400 tonner barges upto about 40 km. Transport of ore-laden barges through these rivers has conferred on Goa a unique advantage of cheap transport and competitive ore prices in the export market.

The mining of iron ore dates back to 1908 when a few French mining engineers carried out some prospecting and mining on a small scale. Some of the prospecting tunnels made by the French mining engineers can be seen even today in Bicholim and other mines. The mining activity ceased after some time but revived in 1949 and enjoyed a boom in 1957 and onwards with the keen interest shown by the Japanese in Goan ore.

A geological map of Goa was published in 1958 on the basis of geological work done by a group of foreign geologists from 1953 to 1957. Some of the terms used by them for the rocks are different from those adopted by the G.S.I. and recent mining excavations have indicated some mistakes in the map then prepared. Immediately after liberation the G.S.I. and I.B.M. have been active in the area to assess the mineral deposits.

The iron ore deposits in Goa are hematite, belong to the Dharwar system and extend in N.N.W.-S.S.E. direction from Advalpol in the north to Canacona in the south, occupying nearly 1/5th of the total area of the territory. The general direction of the strike is north-south and the beds dip due east at 40° to 60°. The thickness of the reef varies from 3 m to 20 m A strike fault runs across the country rock from Advolpol to Sanguem but not in the orebody itself. The orebody is disturbed by a number of faults which are mostly dip faults.

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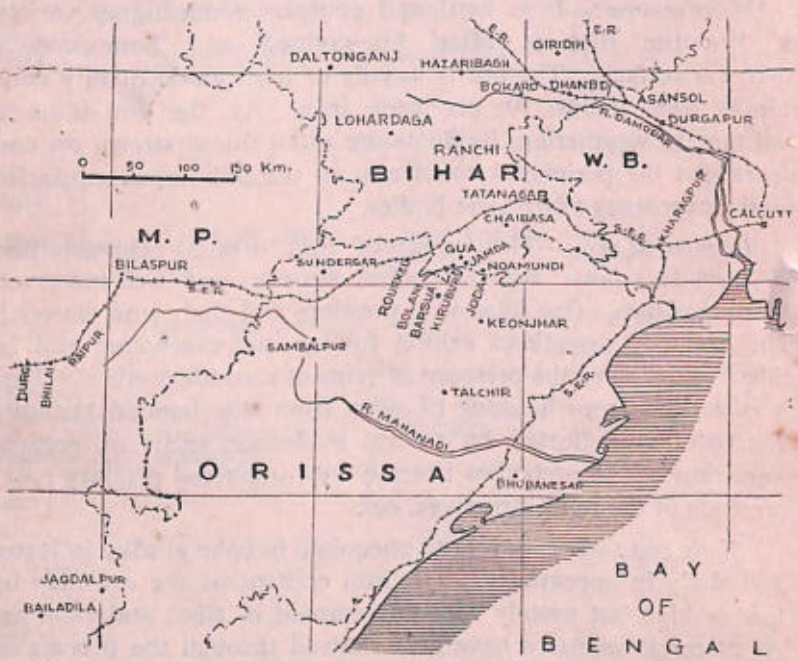

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The surface exposures of the rock formations are all laterite which are of two types :

(1) Laterite bearing bluish flakes of high grade hematite showing a general strike of NNW-SSE.

(2) Laterite with the characteristic vermicularity with argillaceous material indicating neither any strike nor dip direction.

The latter occur as pockets (outliers) over the former on the western slopes of the hill range and appear exposed on a mass scale on the eastern slopes. This is particularly noticeable in the Bicholim Pale area.

The main iron ore bearing zone may be divided into four separate areas which are separated from one another by major faults whose throws and directions are under further investigation. The areas are :

(1) Bicholim Pale area.
(2) Sancordem area.
(3) Sigao-Kalay area.
(4) Sanguem Quepem area.

Throughout the territory the hematite deposit in lumpy form underlies a covering of massive laterites, 3 to 20 m thick. Phyllites appear in the hanging wall as well as in the footwall but in the Bicholim-Pale area the phyllites in the footwall are manganiferous in a few pockets and have kaolinised to black clay. In other areas the phyllites in the footwall have kaolinised to clay which is brown, bluish or black ; for example, in the Betul mine, the footwall is indicated by brown clay which has about 10% of iron and a good percentage of alumina.

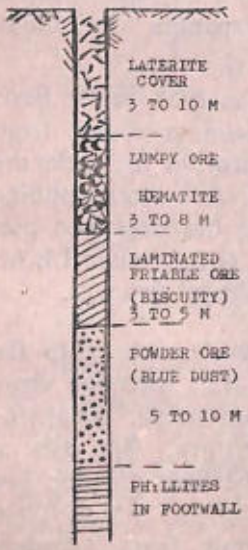

Fig. 2. Typical cross-section in a borehole in Bicholim-Pale area (Goa)

The iron ore in the territory contains FeO and Fe_2O_3 and is feebly magnetic. The lumpy ore, as compared to the ores of Bellary Hospet or Noamundi, is soft and porous with a little moisture.

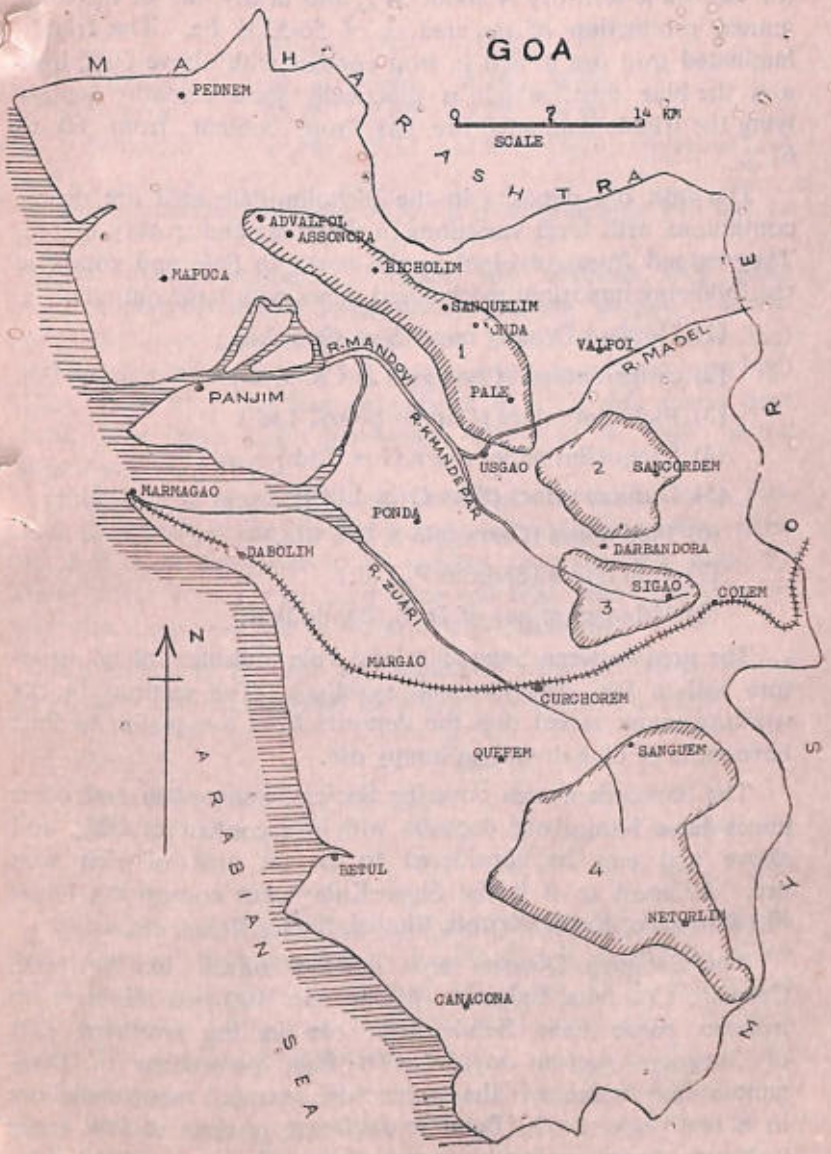

Fig. 3. Iron ore localities in Goa.

The average iron content in lumpy iron ore (+40mm) for the whole territory is about 58% and nearly half of the total annual production of the area is of 56-57% Fe. The friable laminated iron ore is rich in iron content with above 60% iron and the blue dust, which is practically pure hematite underlying the friable laminated ore, has iron content from 60 to 67%.

The iron ore deposits in the Bicholim-Pale area are nearly continuous with local variations in thickness and quality of ore. They extend from Advolpol in the north to Pale and comprise the following important mechanised mines with large outputs.

(1) Horasso Dongar mine (Sesa Goa, Ltd.)
(2) Sirigao mines (Chowgule & Co. Ltd.)
(3) Bicholim mines (Dempo Souza, Ltd.)
(4) Sanquelim mines (Sesa Goa, Ltd.)
(5) Gulliem mines (Sesa Goa, Ltd.)
(6) Pale mines (Chowgule & Co. Ltd.)
(7) Pale mines (Sallitho P. Ltd.)
(8) Velguem mines of D. B. Bandodkar.

The area between Sanquelim and Pale indicates folded structure with a few anticlines and synclines. The sections in the working mines reveal that the deposits from Sanquelim to Pale have more of blue dust than lumpy ore.

The Sancordem area covering Seniem, Sancordem and other mines have lumpy ore deposits with iron content of 60% and above and can be considered to be an area of rich iron ore. Adjacent to it is the Sigao-Kalay area comprising mines like Kirlepale, Kodli, Koshti, Bimbal, Sigao, Kalay, etc.

The Sanguem-Quepem area includes mines like Barazan, Curpem, Colomba, Salginim, Maina, etc. and two mines in an isolated patch near Betul. Iron ore in the southern part of Sanguem-Quepem area has a high percentage of manganese and some of the mines do extract manganese ore in a few pockets. In Betul mines some pockets of low grade iron ore have gradually increased in alumina content ultimately leading to bauxite and mining of bauxite in such pockets is likely to be economic.

A typical commercial analysis of lumpy ore in Bicholim-Pale area is given below (percentages).

	Fe_2O_3	SiO_2	Al_2O_3	Fe	
Sample 1	82.72	1.86	8.18	58.20	
Sample 2	83.72	3.10	7.72	58.90	
Sample 3	69.14	5.72	13.94	48.70	(rejection)

Ore containing less than 54% Fe is not vendible and has no buyers.

The average content of silica and alumina vary from place to place even in a small mine but it may be said in broad terms that ores in Bicholim-Pale area have low silica (1 to 3%) and higher alumina (5 to 8%), ores in mines near Sigao have high Silica (6 to 9%) and low alumina (3 to 4%), and mines near Betul area have silica percentage of 3 to 4, with higher alumina of over 10%.

Blue dust is practically pure hematite. In mines which produce blue dust it has been observed that coarser particles from the dust zone are lower in Fe content than the finer ones, by about 1%. It has also been observed that blue dust in contact with lumpy ore has no water of crystallisation.

Commercially the iron ore grades in Goa are based on the Fe content and are classified as follows for export market.

Fe%	Grade
54 to 57	Low
58 to 60	Medium or normal
above 60	High.

Certain commercial terms are widely used for iron ores in this territory. Ferromanganese (technically it should be called ferrugenous manganese) contains minimum 28% manganese and minimum 22% iron and the total percentage of manganese and iron should not exceed 55.

Black iron (technically it should be called manganiferous iron) contains manganese from 5 to 10% and iron from 50 to 45%, the total percentage of iron and manganese not exceeding 55.

6. RESERVES

Fortunately, the richness in quality, the abundance in quantity, and the regional distribution of the Indian iron ore deposits are such that they could not have been more ideally situated for exploitation and utilisation. As shown in Table 2, our iron ore reserves are estimated at 21,000 million tonnes, which should meet our increasing requirements of domestic consumption and foreign export at a production rate of 100 million tons a year for over 200 years.

TABLE 2

Estimates and Grades of Iron-Ore Reserves by geographical regions

State	Locality of occurrence	Grade (per cent Fe)	Reserves (Million tonnes)	
			Proved	Probable & possible
1	2	3	4	5
Andhra Pradesh	*Anantpur district :*			
	Obalapuram-Siddarampuram, Nalapamnangudu area	60	1.01	
	Cuddapah district :			
	Chabali, Pagadalapalle and Rajampet ..	55—63	0.04	
	Guntur and Nellore districts :			
	Ongole group of deposits	33—37	391.18	
	Khammam and Warangal districts :			
	Area between Bangwan and Tatrayapalli ..	54—65	5.59	11.18
	Krishna and Khammam districts ..	60—65	1.32	
	Kurnool district :			
	Veldurti-Ramallakota	50—65	4.05	
	Valasala-Gurapalli ..	50—60	..	
Total ..	.... ..		403.19	11.18

1	2	3	4	5
Bihar and Orissa	*Palamau district :*			
	Gore and Riwabathan	66—65	0.71	
	Singhbhum district :			
	Noamundi, Thakurani, Gua, etc. extending through Bonai (Sundergarh district) and Joda (Keonjhar district) of Orissa ..	60—68	2,032.09	8,128.38
	Saranda Block ..	60	207.27	
	Saranda N.E. Block	60	17.70	
	Gaudi Buru ..	60	51.82	
	Kiriburu and Dirisum Buru	60	131.58 74.07	
	Cuttack district :			
	Sukinda area ..	60	27.43	
	Mayurbhanj district :			
	Badampahar, Gorumahasani and Sulaipat	60—68	60.96	
	Sundergarh district :			
	Bolani, Taldi, etc. ..	60—68	665.51	
	Horomotoguali Block	60—68	124.97	
	Kandadhar pahar ..	60—68	188.98	
	Malangtoli (Sundergarh and Keonjhar districts) ..	60—68	203.21	
Total ..	....	3,780.20	8,128.38	
Goa ..	Bicholim etc. ..	57	60.96	298.72
Total ..	....		60.96	298.72
Himachal Pradesh	*Mandi district :* Sangalwara ..	35—38	60.96	60.96
Total ..	....		60.96	60.96
Madhya Pradesh	*Bastar district :* Bailadila ..	60—69	1,134.00	3,657.77

1	2	3	4	5
	Rowghat ..	60—69	751.87	
	Kondapal ..	..	60.96	
	Durg district :			
	Dhalli-Rajhara ..	68	116.84	
	Jabalpur district ..			2,438.51
	Agaria, Jauli Kanhwara, Saroli etc. ..		101.60	
Total ..	....	..	2,165.27	6,096.28
Madras ..	*Salem and Tiruchirapalli districts* ..	25—48	309.89	1,016.05
Total ..	....	..	309.89	1,016.05
Maharashtra	*Chanda district :*			
	Asola, Devalgaon Lohara amd Pilalgaon	60—69	22.35	
	Puser area (lateritic ore)	60—62	0.254	0.254
	Ratnagiri district :			
	Aros, Khanvali and Redi	60—62	53.85	
Total ..	....	..	76.464	0.264
Mysore ..	*Bellary district :*			
	Timmapanna Gudi, Donimala ranges, etc. in Sandur Taluk ..	60—67	132.09	304.81
	Chikmagalore district :			
	Gangur, Hirlyur, Kalhatti and Kemmanugundi ..	50—60	772.19	2,032.00
	Dharwar district :			
	Kusalpur ..	40—60		8.64
	Hassan district :			
	Nugguhalli and Uranium ..	60	9.14	
	Shimoga district :			
	Shimoga Block ..	51—56	34.55	

1	2	3	4	5
	Tumkur district :			
	Kunnigal area ..	51—56		32.51
	Total ..	..	947 97	2,378.05
Punjab	*Mohindergarh district :*			
	Chhapri-Jaunpur ..	60	2.03	
	Total ..	..	2.03	
Rajasthan	Alwar, Jaipur ..			30.48
	Jodhpur and Udaipur districts ..		5.08	
	Total ..		5.08	30.48
Uttar Pradesh	Almora, Garhwal and Nainital districts ..	58—61	10.16	
	Total ..	..	10.16	
West Bengal	*Birbhum district :*			
	Damara and Ganpur	28—59		
	Burdwan district :			
	Kulti and Raniganj ..	39—48	508.02	2,032.00
	Total ..	..	508.02	2,032.00
	Grand Total ..	..	7,933.104	11,912.804

*After A. G. Jhingran and B. N. Sinha.

It may be worthwhile to point out that although major sources of high grade ores from the banded hematite deposits, which are by no means unlimited, are being fast developed, little attention has so far been paid to the proper consumption of deposits containing ore with less than 50% iron and also to the good quantity of fines, which constitute considerable proportion of our deposits. There is also ample scope for developing the magnetite ores for domestic steel consumption ; in addition to iron these ores would yield sufficient quantity of titanium and vanadium, the metals in great demand in ordnance applications in the modern times.

REFERENCES

1. DUNN, J. A., "The Origin of Iron Ores in Singhbhum, India", *Econ. Geol., Vol.* 30, 1935.
2. DUNN, J. A., "The mineral deposits of Eastern Singhbhum and surrounding areas, India", *Geol. Surv. Mem., Vol.* 69, 1937.
3. DUNN, J. A., "The Origin of banded hematite Ores in India", *Econ. Geol., Vol.* 36, 1920.
4. DUTT, A. & NANDY, N. C., "Origin of Noamundi and Joda East iron ore deposits" *TISCO Tech. Joranal*, 1965.
5. GILL, J. E., "Origin of the Gunflint iron-bearing formation", *Econ. Geol., Vo.* 22, 1927.
6. GOODWIN, A. M., "Facies Relations in the Gunflint Iron Formation", *Econ. Geol., Vol.* 51, 1956.
7. GRUNER, J. W., "Origin of sedimentary Iron formations", *Econ. Geol., Vol.* 17, 1922.
8. HUBER, N. K., "The environmental control of sedimentary iron minerals", *Econ. Geol., Vol.* 53, 1958.
9. JAMES, H. L., "Sedimentary facies of iron formation, *Econ. Geol., Vol.* 49, 1954.
10. JONES, H. C., "The Iron ores of Singhbhum and Orissa, India", *Geol. Surv. Rec., Vol.* 54, 1923.
11. JONES, H. C., "The iron ore deposits of Bihar and Orissa, India", *Geol. Surv, Mem Vol.* 63, 1934.
12. KRISHNAN, M. S., "Iron Ores of India," Ind. Assoc. for the Cultivation of Science, Jadavpur, 1955.
13. MOORE, E. S. & MAYNARD, J. E., "Solution, Transportation and precipitation of iron & silica", Econ. Geol. Vol. 24, 1929.
14. PARSONS, E., "Indian Iron Ores", Mining Mag., Vol. 26, 1922.
15. PERCIVAL, F. G., "The iron ores of Noamundi", Min. Geol. Inst. Ind. Trans., Vol. 26, 1931.
16. SAKAMOTO, T., "The Origin of the Pre-Cambrian Banded iron Ores", Am. Journ. Sc., Vol. 248, 1950.
17. SPENCER, E. & PERCIVAL, F. G., "The structure and origin of the banded hematite jaspers of Singhbhum, India", Econ. Geol. Vol. 47, 1952.
18. VAN HISE C. R., & LEITH, C. K., "The Geology of the Lake Superior Region, U.S., Geol. Surv., Vol. 52, 1911.
19. VARMA, O. P. "The Geology and Origin of magnetite deposits of the Baula range near the village of Nausahi, District Keonjhar (Orissa) Q. Journ. Geol. Min. Met. Soc. Ind. Vol. XXXVI, 1964.
20. WAGNER, P. A., "The Iron Ore depcsits of the Union of South Africa", Geol. Surv. S. Af. Mem., No. 26, 1928.
21. WAZALWAR, H. V., "Tisco Tech. Journ., 4, 1957.

CHAPTER 5

PROSPECTING AND EXPLORATION

1. Introduction. 2. Legal position. 3. General outline of prospecting programme. 4. Field work. 5. Geophysical methods. 6. Correlation of beds and borehole logging. 7. Drilling. 8. Cost of prospecting and exploration in some projects.

1. INTRODUCTION

The terms *prospecting* and *exploration* are loosely used synonymously ; in the Mines and Minerals (Regulation and Development) Act, 1957, *prospecting operations* have been defined as "operations undertaken for the purpose of exploring, locating or proving mineral deposits". In more precise terminology, *prospecting* operations are those connected with searching for minerals and *exploration* work is that which is undertaken to gain knowledge of size, shape, position, characteristics and value of the mineral deposit.

Before launching a mine project, the prospect must be *thoroughly explored.* The purpose of exploration is to ascertain the extent, shape and structure of the deposit, its tonnage and grade, physical characteristics of the enclosing rocks and the structure of the area. The design of the mine is considerably influenced by the geological structure and market grade of the commercial reserves. Detailed exploration may be costly but nevertheless necessary. If in an unwise bid to save time and money involved in an exhaustive exploration operation, a mine is started on the basis of the preliminary prospecting, the entrepreneur may come to grief. Should the wishful thinking not materialise the capital outlay may prove infructuous. On the other hand, should the operations of exploration confirm the earlier forecast and result in a fruitful mining venture, the sum spent on exploration would amount to not more than a mere insignificant portion of the total cost of the project.

2. LEGAL POSITION

Legal requirements for a firm or agency undertaking prospecting operations are embodied in the Mines and Minerals (Regulation and Development) Act, 1957, and the Mineral Concession Rules, 1960. Atomic Energy Act, No. XXIV of 1948 governs fissionable materials. Main provisions of these enactments may be summarised below for the guidance of prospectors.

Any party wishing to prospect an area for any mineral has to obtain a Certificate of Approval from the State Government. Any Indian citizen, not necessarily technically qualified, can get a Certificate of Approval. The application for such Certificate should be in Form A_1 and accompanied by a fee of Rs. 500/-. The Certificate is valid for one calendar year and can be renewed each year on payment of a fee of Rs. 250/-. With a valid Certificate of Approval and an income-tax clearance certificate, the party must apply to the State Government for a prospecting license in a prescribed form accompanied by a fee at the rate of Rs. 20/- for the first square kilometre or part thereof and Rs. 4/- per subsequent square kilometre or part. In respect of coal and iron ore (which are minerals specified in the First Schedule of the Act) State Govt. has to obtain Central Government's previous approval for grant of prospecting license. Maximum aggregate area for which one or more prospecting licenses can be granted to one party in any one State is 50 sq. miles (130 sq. km), subject to relaxation by the Central Government in the interest of mineral development. The license is valid for a period not exceeding two years but can be extended for one or more such periods with Central Government's approval, if a longer period is considered necessary for completing the prospecting operations. If the applications for grant of mining lease and grant of prospecting license are received for the same area within 30 days of each other, the former has precedence.

Prospecting licensee has to pay, in addition to the above fees, a prospecting fee fixed by the State Government (which may be between 25 p. and Rs. 2/- per year per hectare of land covered by the license) and to deposit a security at the rate

of Rs. 200/- per sq. km of area, against due observance of conditions of the license.

The licensee is permitted to enter upon the land and to search for mineral by digging, quarrying, boring, or otherwise. The licensee is entitled to carry away, for purposes other than commercial sale, a maximum of 50 tonnes of coal and 10: onnes of iron ore without payment of royalty, and three times these quantities, on payment of royalty. He can clear undergrowth and bush-wood and trees with written permission of the Govt. Forest Officer or Deputy Commissioner/Collector, make use of water courses and drains without interfering with other users' rights and erect temporary structures and machinery. With prior approval of the Central Government, the licensee can transfer his license to another person holding Certificate of Approval and consenting to fulfil other conditions, on payment of Rs. 100/- to the State Government.

The licensee must plug all bore-holes and fill up all excavations made by him within six months of cessation of prospecting operations. He must not employ a foreigner except with previous permission of the Central Government in connection with prospecting operations. He must report discovery, within 60 days, of any mineral other than that covered by his license and shall not prospect for it unless he takes out a license for it. The licensee should maintain accurate accounts of his expenses and permit them to be inspected by authorised Government officials. He must submit to the State Government confidentially a report of the work done and disclose all information acquired by him of the geology and mineral resources of the area. He has also to furnish to the Director, Geological Survey of India, Calcutta, Geophysical data obtained by him and to the Secretary, Department of Atomic Energy, New Delhi, all information on radio-active minerals collected by him during prospecting.

While granting license, State Government may prescribe compensation for damage to land, require indemnity to Government against claims by third party, impose restrictions on felling trees, lay down conditions for entry in occupied land, require facilities to be given by the licensee for working other

mineral in the same or adjoining areas, and lay down any conditions in the interest of mineral development.

Central Government may, after consultation with State Government declare that no prospecting license be granted in respect of any land notified in the official Gazette to be reserved for its own prospecting for specific minerals. The Central Government has within recent days shown interest in iron ore bearing areas, and areas containing coking coal.

3. GENERAL OUTLINE OF PROSPECTING PROGRAMME

Geological investigation forms the backbone of any prospecting programme not only for selecting areas for proving minerals but also for interpreting the observations noted in the field. Modern tools coming to the aid of geology are mainly geophysics and geochemistry. The whole programme consists of collecting evidences by direct and indirect methods and making geological maps. Direct methods include the conventional use of pick and shovel, mortar, pestle, and pan of early prospectors, pitting and trenching and/or adit driving, and drilling and laboratory investigations. Indirect methods consist of photogeology, and geophysical and geochemical anomalies.

The prospecting and exploration campaign should be under the overall charge of a mining geologist who should be able to integrate his geological knowledge with the data and information provided by other means, notably by geophysics, geochemistry, drilling, sampling and assaying.

Before actual prospecting is undertaken, all available information that would expedite the work must be collected and studied. The geologist has to collect available maps of the area which may be any of the following : (1) Aerial photograph and a base map prepared therefrom, (2) Topographical map, (3) Geological map, and use the most reliable one as his base map for reconnaisance traverses and geological mapping.

An *aerial photograph* is a bird's eye view of the land surface, the relief and man-made features on the earth's surface and is obtained by a camera fitted to a flying aeroplane.

A *topographical map* shows the size, shape and distribution of features of the earth's surface, showing (a) relief includ-

ing hills, valleys, plains, and the like, (b) drainage and water bodies, such as streams, canals, swamps, seas, lakes, ponds, etc. and (c) cultural features including man-made works such as mouza boundaries, district and state boundaries, cities, roads, railways, etc. Relief is represented by contour lines at definite intervals, and various colours are used to indicate drainage and culture (lines of contour are usually coloured brown, drainage channel, blue and culture, black). In India, topographical maps have been prepared by the Survey of India for almost all accessible parts ; these are available in the following scales :

R.F.	1 : 1,013,760	(1″=	16 miles)
	1 : 253,440	(1″=	4 miles)
	1 : 126,720	(1″=	2 miles)
	1 : 63,360	(1″=	1 mile)
	1 : 31,680	(1″=	½ mile)
	1 : 15,840	(1″=	¼ mile)

A base map in a scale of 1″=¼ miles (1=15,840) is convenient to a geologist for detailed prospecting.

Some of these maps were prepared in the last century and have to be corrected for changes in the river courses, man-made features and read in the light of the indices then prevalent, as some of them have since changed.

A *geological map* shows the distribution of rocks and geological structures and it is prepared by plotting these on the topographical map. Geological maps are prepared for the country by the Geological Survey of India for practically all parts of India. Private parties and State Government agencies have also in their possession useful information and geologimcap maps of areas where they worked.

The prospecting programme may be divided into several stages and it may start off from the first stage in a virgin area and intermediate stage in an area where some preliminary work has already been done. The stages are :

1. Aerial photography of the whole area, accompanied by aeromagnetic survey only for iron ore, if funds permit.

2. Selecting areas of special interest based on aero-geologic and, if available, aero-magnetic map.
3. Survey of the selected area by geophysical and geological mapping.
4. Purposeful detailed investigation of worthwhile anomalies.
5. Pitting, trenching and/or drilling to obtain direct evidence of mineral deposits and to collect samples for qualitative and quantitative assessment.

These stages are grouped under two categories :

(i) Exploratory prospecting.
(ii) Detailed prospecting.

The **exploratory prospecting** is always carried out by the Geological Survey of India leaving the detailed prospecting to G.S.I. itself or to other government or private parties. The object of exploratory prospecting is to delimit the area over which the mineral may exist and to establish broadly the correlation of rocks, the structure of the beds, the major geological disturbances and the reserves. A more correct picture of the latter factors is however available after detailed prospecting which aims at determining the exact correlation of rocks, the structure of the beds, the dykes, the position and the throw of faults and the proved reserves. Without detailed prospecting, it is not possible to undertake the preparation of a project report of a mine. Exploratory prospecting may be dispensed with in areas known to be contiguous to coal/iron ore bearing localities. N.C.D.C., for example, did not undertake exploratory prospecting in Sudamdih or Central Jharia projects. During exploratory as well as detailed prospecting boring (drilling) and recourse to geophysical methods of prospecting/logging become inevitable. For exploratory prospecting the boreholes are placed far apart, the distance in coal measures varying from 1.5 to 4.5 km. Every bore-hole drilled during prospecting leads to calculation of proved, indicated and inferred reserves.

Exploratory prospecting, particularly for important areas, is often assisted by aerial photography. A Geologist or a Mining Engineer in charge of a new project is sometimes required to interpret an aerial photograph and some essential elements of aerial photography are given below.

Aerial Photography

An aeroplane for aerial photography has a camera mounted in its belly and during flight, with the camera pointed vertically downwards, it takes photographs, not continuously but at fixed intervals such that photographs of adjacent areas in a straight flight overlap by about 30%. (Fig. 1). A large area is divided into a number of parallel strips, each strip over-lapping the next and the photographs over-lapping by 30 to

Fig. 1. Taking an aerial photograph.

45%. Tilt of the plane or variation in altitude is avoided as far as practicable. A map prepared by the assemblarge of such photographs is called a mosaic. It provides continuous picture of an area which is too large to be covered by a single photo.

Aerial photographs are of confidential nature. They are normally in black and white and are prepared by the *Survey of India* for civil purposes. The work of aerial photography is

generally entrusted to contractors who are specialised in this art. Recently, the Ministry of Mines & Metals has created a Department of Air-borne Mineral Surveys and Exploration under its administrative control. Its function is to carry out aerial surveys for exploration in certain selected areas of the country.

A subject known as *photogrammetry* deals with correction, rectification and adjustment of aerial photographs for the different scales in mapping, parallax, tilt of aeroplane etc. If the reduced levels of two or three prominent objects in the aerial photograph are known, a photogrammerist can draw contours on the aerial photograph in the office itself without going to the field with the help of an instrument known as Sketch-master. If the reduced level of some prominent object in the aerial photo is not known, field work is necessary to ascertain such reduced level with reference to a bench mark in nearby localities, preferably railway or mines.

Some of the aerial photographs which were prepared for the Damodar Valley Corporation cover the areas of the South Karanpura coalfield. The scale was nearly 1.5 inches=1 mile (R.F. 1 : 42,240). Topographical maps showing contours at 3 m intervals were prepared from the aerial photos on a scale of 6 inches=1 mile (R.F. 1 : 10,560) on matt paper and were useful to geologists for field work and marking details on the map itself.

Study and Interpretation of Aerial Photographs

The study of the air photographs and maps constructed from such photographs for geological information is called photogeology. When studying aerial photographs maps of adjacent areas should be correlated to some prominent reference objects in each sheet and placed together with suitable allowance for side lap.

A three-dimensional view of the aerial photographs is seen through an instrument called Stereoscope. A small folding pocket stereoscope is a good help to a field geologist. A very useful modification of the folding pocket stereoscope is a pair of spectacles with lenses prepared for stereoscopic observation.

INTERPRETATION :

Most roads are very irregular or they may be in rectangular patterns. They usually have sharp angular turns. The railway turns, on the other hand have always smooth curves. As the gradient of railway lines is very low and often level, the railway line can serve as an approximate reference contour in a limited area. Houses and other buildings may be conspicuous by their shadows. Stream courses indicate the general slope of the ground controlling them. Vegetation, woods, grass, cultivated fields, etc. have various appearances on air photographs. These differences are seen as changes in the tonal value on black and white prints. Vegetation is closely related to soil and if the soil is from underlying bed rocks, the distribution of the different bed rocks is deciphered. It is commonly believed that major fault planes are associated with vegetation or passage of streams, rivulets, etc. as a fault plane is a softer zone liable to heavy differential erosion. This is however not always true and fault zones are sometimes indicated by a row of hillcocks as in the Bokaro-Kargali area. Hard rocks like iron ore bodies, ferruginous sandstone, metamorphic gneisses and schists, dykes, etc. stand out as ridges or escarpments and the softer beds of shales and coals are eroded more, forming plain areas, valleys or stream courses. On a level plain interpretation of geology is very much restricted. The area must have moderate relief so that dip of the beds may be determined. An experienced photogeologist, if aided by stereoscope, can make fairly accurate estimates of the amount of dip of stratified rocks. Dips observed under stereoscopes invariably appear steeper than their actual inclination. This is, of course, a help in extremely low dip country but is misleading in areas where dips are steeper. However, dips ranging from 3° to 20° can usually be estimated with a reasonable degree of accuracy, after some actual check of the ground.

Aerial photography has the following advantages over ground-survey methods :

(1) It makes for greater speed in the preparation of topographical map of comparatively large areas. For small areas the method is uneconomic.

(2) It shows up prominent landmarks and relief of ground and can be used to prepare contour maps.

(3) It is of particular use where terrain and trespass restrictions will involve considerable amount of time and money to cover the area by ground survey.

(4) It has multifarious uses, for example, in topographical and geological surveys, forestry, soil conservation, highway engineering, crop estimation, etc.

It is stated that skilled photographic interpreters have on occasion been able to identify objects of 0.5 metre in length in photographs taken from a height of 8 km above the earth. Details are, therefore, not lost in aerial photos.

The geologically promising area which has to be earmarked for exploratory or detailed prospecting is decided upon after a careful interpretation of the aerial photographs.

Armed with all the available information and maps, and equipped with necessary equipment and instruments, the geologist sets out for his field work of prospecting. Transports of varied type are used depending on the terrain ; ponies, elephants and other animals, bullock carts, jeeps, barges, helicopters or aeroplanes.

4. FIELD WORK

Accommodation

After having delimited on the base map the area in which the mineral is likely to occur, the geologist has to plan his programme for detailed work in the field. Duration of the prospecting camp will depend on :

(1) The extent of the field.
(2) The depth to which the economic occurrence is likely to exist.
(3) Nature of geological disturbances.
(4) Facilities of transporation and repair work.
(5) Staff and equipment at his disposal.
(6) The extent of prior investigation.

While the duration of the programme will inevitably influence the choice of the site for his camp, the geologist will have to bear in mind the factors such as proximity to towns for obtaining provisions, facilities of water supply, presence of wild beasts, means of transportation and communications. For obvious

reasons rainy-season is to be avoided for field mapping, though it is ideal for drilling work which requires profuse water. In areas of constant rains such as Cherrapunji there is, however, no choice. In region of high altitude such as the Himalayan areas and Eastern & N.E. India the field season is in the spring and summer months.

Accommodation during prospecting camp poses quite a problem. Tents are generally used ; they last for 2-3 rainy seasons if used with care. The tents normally used have the following accommodation : Swiss cottage, 4.5 m×3 m for officers ; Kabul Pal, 3 m×3 m×2.3 m for 2 men ; Servant's Pal, 3 m×3 m for manual workers.

Though tolerable during winter and summer, the tents are insecure against storms and rains. Dormitories can be constructed of bamboo matting with brick floor and corrugated sheet roof for community living of a number of unskilled labour, particularly if large scale drilling and pitting are undertaken. Such dormitories can endure for 3 to 5 years with a little care. They are also useful for housing stores, workshop, etc. In place of bamboo matting, unburnt bricks can be used for walls and if carefully plastered with earth and cowdung, they can last for 4-5 rainy seasons.

Some trials have been made with prefabricated aluminium hutments which have a useful life of 5 to 6 years and can provide security and protection against rough weather and undesirable elements. However, they are uncomfortable, too hot during summer and too cold during winter nights, require some training for erection and are about ten times costlier than tents.

Water supply to prospecting/drilling camp is often arranged by a water tanker pulled by tractor as the roads are rough and the source of water, away. In the absence of any suitable water supply, vicinity of river or tank receives the first consideration for a camp site.

Looking for Minerals

The geological formations where to look for coal or iron ore are already known. Names of localities and villages are significant pointers, such as Kalipahari (black hill), Angarpathra (stone for fire), Lohardaga, etc. Outcrops and places where rocks are likely to be exposed, such as railway cuttings, trenches,

wells etc. are to receive a geologist's first attention during traverse. The dip of the strata is known from the outcrops exposed in such places and the direction of travel for the younger or older rocks becomes obvious. The structured faults and dykes also reveal themselves during the traverse. Where streams or water courses are encountered, the geologist has to keep his eyes open for blackish material for coal search and for brownish material for iron ore in the water course. In either case, the source has to be traced on the upstream side near the place of highest concentration of minerals.

In coal measures the harder rocks like sandstones, stand out as ridges and softer beds of coal or shale, by differential erosion, form plain level ground or depressed areas suitable for growth of paddy or other vegetation. A geologist in search of coal, therefore, looks expectantly to the low ground in the hope of locating outcrops which can be of shale or coal but rarely of sandstone. The weathering of coal gives the soil a darker hue than that normal for the surrounding rocks and small pieces of coal may be visible in the soil if it has been ploughed. In one case the black soil at the mouth of a rat hole provided to a weary and desperate geologist a clue to the concealed outcrop of a seam in the Bokaro coalfield.

Unlike coal, iron ore (hematite) is to be traced in hills which have stood weathering due to rain water and other atmospheric agencies. The roads and soil in the vicinity are of brownish colour and the hill slopes or outcrop, covered by laterite mixed to a varying degree with float ore. In some instances phyllites occur in the foot wall. The weathered hematite ore and laterite appear blackish. The flaky (or biscuity) hematite ore, characteristic of iron ore deposits practically throughout India, often forms a characteristic stratum whose flakes appear over the outcrops. Prospecting generally starts with search for float or outcrop ore or by recognising some member of the series with which the ore is associated. There is usually some characteristic stratum above and below the ore bed, which outcrops. From measurements of this, an approximate idea of the location of the orebody outcrop along the strike may be got.

If a geologist comes across outcrops of coal or iron ore during his traverse, he confirms his finding by trenching and

pitting. Sometimes drifts are put down at regular intervals to get a better idea of the thickness, grade and inclination of the mineral deposit.

Trenching & Pitting

Trenches are open, horizontal or inclined excavations with small cross-section and comparatively long length

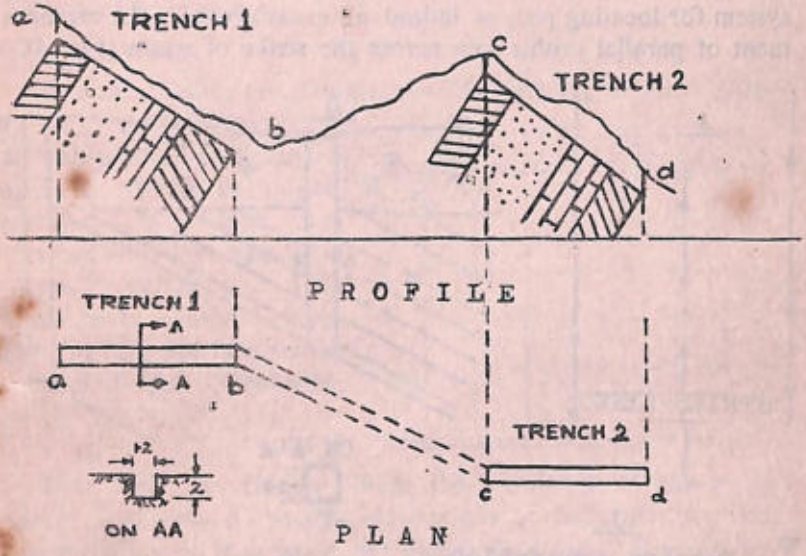

Fig. 2. Location of trenches on slopes for structure.

They are dug in the surface (i) to expose bed rocks in order to find out their lithological characteristics, mode of occurrence and age, or (ii) to establish an outcrop, when the alluvium thickness is not greater than 2 m. Appropriate place for a trench is on a slope inclined in a direction opposite to the dip of the strata (Fig. 2,) or on ground at right angles to the course of outcrop (Fig. 3). The first trenches in a selected area should be dug as wide apart as possible and the intermediate ones dug as few as circumstances would warrant.

Fig. 3. Location of trenches to trace boundaries of a single orebody.

Pits are rectangular, square or sometimes circular, vertical or steeply inclined excavations dug to a depth of over 2—3 m, reaching a depth of 20 m in rare cases. They (1) enable bed rocks to be exposed under thick alluvium cover, (2) afford opportunity to study rocks more closely, (3) serve to determine strata thickness, study contacts, disturbances and (4) provide samples for laboratory investigations. Usually accepted system for locating pits, as indeed all excavations, is the arrangement of parallel profile run across the strike of strata (Fig. 4).

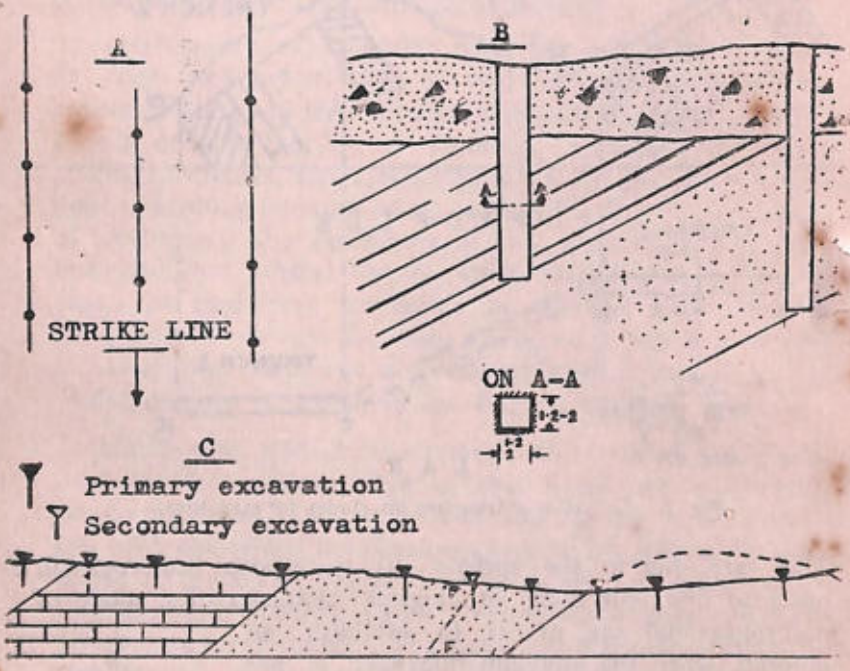

Fig. 4. Location of pits in deciding structure.

Documentation of pits and trenches should include :

1. Description of rocks exposed by excavation.
2. Statement of contacts, disturbances and bedding planes.
3. Sampling results.
4. Sketch of location.

Funds permitting, recourse may be taken to geophysical methods of prospecting which enable one to locate con-

cealed structures with reasonable accuracy. The knowledge of such concealed structures help the geologist to choose drilling sites intelligently. Drilling equipment is a great asset to him but where the bulky drilling rigs cannot be used conveniently sometimes due to hilly terrain or due to water shortage, in the initial stage a jackhammer drill like "Cobra" (Atlas Copco) or "Pionjar" with a built-in petrol engine is quite handy for holes upto 3 m depth.

GRID PATTERN

A major drilling programme, essential during detailed prospecting, consists in putting boreholes on a grid pattern. (Fig. 5). With this object, the area is divided into recttangles, with one side gnerally parallel to the likely strike direction, and pegs for drill sites are put at corners of such rectangles.

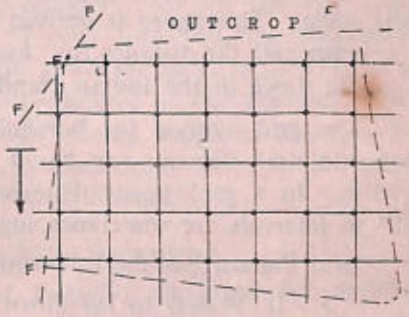

Fig. 5. Siting of boreholes on a grid pattern.

The aim is (1) to obtain the knowledge of the structure of coal/mineral deposits, inclination of beds, faults, folds and variations in magnitude and direction of dip. (2) to get information of overburden : mineral ratio, the quarriable limits, nature of overburden in shallow deposits and floor and roof conditions and nature of wall rocks in the case of deeper horizons. (3) to calculate reserves and assess qualities of the mineral. (4) to correlate coal seams/iron ore beds.

The grid interval may range from 30 to 450 m depending on the following factors :

(1) Nature and extent of disturbances. Faulted and folded area would require closer grid intervals and a few stray holes to check.

(2) Uniform structure. Area which indicates uniformity in the beds may be adequately covered by a few check holes.

(3) Depth of coal seams. Deep holes being costly, every effort is made to be satisfied with the barest minimum

number of holes and grid interval, especially to the dip, is wide. For shallow depth with a promise of an opencast project, closer-interval is planned to get more information of the nature of the overburden and ratio of overbuden : mineral, etc.

(4) Experience and ability of the geologist to interpret the data from the boreholes.

In a relatively undisturbed (geologically) coal bearing area the number of boreholes per km^2 for detailed prospecting is 12 to 16 as the maximum distance between boreholes may be 400m. This figure is derived from the definition of proved reserves and the distance for assuming continuity of coal bed as laid down in the Indian Standards Procedure.

The grid interval for borehole during detailed prospecting of iron ore deposits as based on experience is generally 50 m. In a geologically disturbed area, check boreholes at 25 m intervals are sometimes suggested.

Grid Pattern has the following advantages :

1. It is easy to lay down on the ground by simple surveying methods.
2. Sections across boreholes in any direction can be easily drawn without need for interpolation.
3. Calculation of reserves is simplified,

A borehole, 300 m deep, may take 2 to 3 months to drill and a deep borehole, about 600 m deep, may take 6 to 7 months involving nearly Rs. 130/- per metre depth including interest, depreciation and overheads. Systematic drilling is therefore time-consuming and costly. Where initial funds do not permit of a regular and systematic drilling campaign but a few holes at a shallow depth indicate deposits suitable to undertake a mining project, major drilling programme is sometimes postponed and carried out when mining operations continue in a relatively proved area, yielding some financial return on the investment.

The rock samples and the sludge obtained during drilling are sent to laboratory for chemical analysis and microscopic study. The information gained by field traverse and with the assistance of drilling, geophysical methods, borehole logging,

laboratory tests, etc. is noted down on the base map. Though a base map on a scale of 1 : 15,840 (1"=1/4 mile) is convenient for a geologist for detailed prospecting, some of the details required to undertake a mining project, particularly in respect of iron ore, are preferably marked on a base map or plan of 1 : 2,000 scale. Sometimes isochore maps (a map showing lines passing through points having the same borehole thickness of the mineral) are prepared by the geologists.

When data from all the boreholes are available, mineral samples analysed, and mineral beds/coal seams correlated, the geologist plots sections along different lines on the dip and on the strike. The reserves of each seam/bed are calculated for different portions separately and then added up. The area bounded by faults or dykes is considered as one portion. The full report is then placed before mining engineers who undertake the planning of the mine and decide the points where quarries, inclines and pits have to be located.

5. GEOPHYSICAL METHODS.

The principal geophysical methods employed for specific purposes in exploration are :

(a) Magnetic
(b) Gravitational
(c) Electrical and Electomagnetic
(d) Seismic

Besides there are quite a few special methods such as :

(e) Radiometric or Radioactivity (f) Thermal, etc.

Excepting the seismic method all the methods can be used for exploration without resort to drill holes. The electrical and radioactivity methods have their application for logging of boreholes also.

Magnetic Methods

These methods work on the principle of contrasts in magnetic susceptibility of various types of rocks and on local variation of the compass needle caused by magnetic bodies within the earth. The instruments commonly used are the *Dip Needle*, the *Hotchkiss Superdip* and the *Magnetometers*. The last two are the most modern and fieldworthy.

Readings are taken of vertical and horizontal components of magnetic field across a profile by a dip needle and a note is taken of an anomaly caused by ore bodies containing magnetic minerals, which exert greater influence on the needle than the surrounding rocks bereft of such minerals (Fig. 6). Thus bodies of magnetic iron ore, pyrrhotite, nickel, cobalt, rocks high in iron such as basalt flows, are marked out.

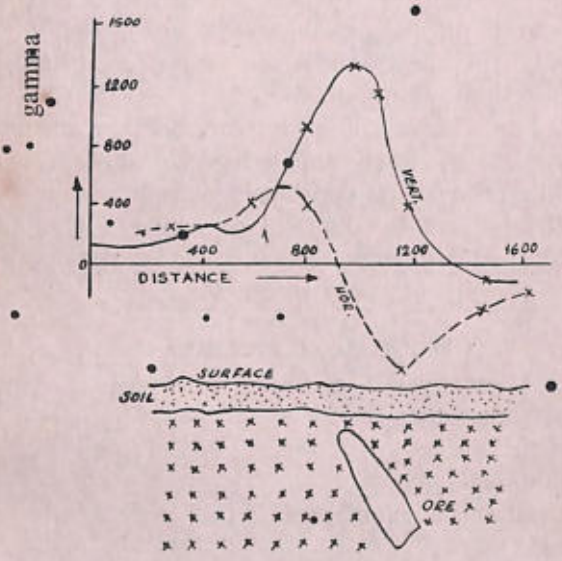

Fig. 6. Magnetic anomalies across a magnetic deposit.

Hotchkiss superdip is a refinement of the common dip needle.

Magnetometer measures the variations in strength of vertical and horizontal components of earth's magnetic field in two balances : vertical variometer and horizontal variometer. Magnetometer surveys are now carried out from the air by fitting a special type of magnetometer below an aeroplane and connecting it electrically with a recording device, inside the plane, which traces a continuous profile of magnetic intensity. The plane flies at an altitude of 300 m or so to avoid the disturbing influence of local magnetic substances. The magnetic susceptibility of basic igneous rocks is 10—100 times greater than that of granite and of most sedimentary rocks, and that of

dolomite is some 10—50 times lower than some shales. The relative differences in both components can be detected in respect of bodies at depths of 500—4,500 m.

Magnetic surveys can, therefore, be used to indicate occurrence of magnetic deposits and bodies associated with magnetic deposits, or structure of a folded sedimentary system, or the strike of a buried diabase dyke, or the depth of bed rock beneath a thick sedimentary cover. When hematite contains some magnetite, magnetometer can be used for hematite search.

Gravitational Methods

These methods apply the principle that a freely suspended plumb bob normally pointing towards the centre of the earth is deflected towards a heavy mass in or on the earth by an amount that is a function of the size of the heavy body and its distance from the body. It is apparently repelled by lighter bodies as the denser surrounding rock exerts greater attraction. There are three main methods for measuring gravitational attraction :

(1) Pendulum
(2) Torsion balance
(3) Gravimeter.

The last-mentioned has almost replaced the other two and measures small variations in the vertical component of gravity directly to the tune of ·01 milligal ; (1 milligal $=10^{-3}$ cm/sec^2, approximately 1 millionth of earth's field) (Fig. 7). The use of gravitational method is usually confined to relatively level ground, corrections being necessitated for a sloping surface and projecting ground masses in a rough terrain. Field data are to be reduced by elevation correction, a terrain correction, latitude correction and in precision surveys, a correction for the effect of diurnal temperature variation and the tidal effects. Ore bodies must be sizable to exert a measurable effect on the instrument.

The method is used mainly for preliminary surveys of major structural features, such as those produced by the dense rocks of a core of an anticline compared with the rocks of the

flanks, low density salt domes, rock faults, haematite and lignite deposits.

Electrical Methods

Electrical methods, used for locating metallic mineral deposits, outlining structure and logging wells, are based on the

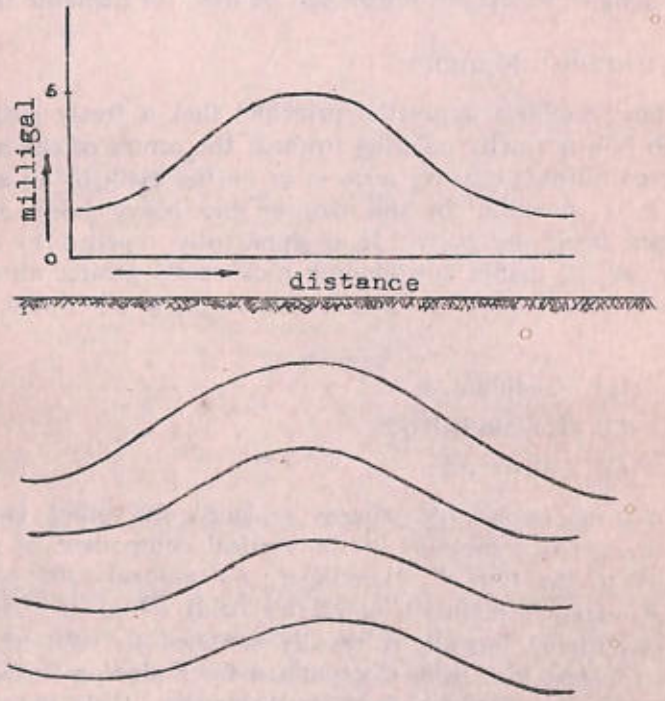

Fig. 7. Nature of gravitational anomaly over an anticline.

contrasts in electrical conductivity and specific electrical resistance of ore-bodies and rocks. In some, natural currents generated by metallic minerals in the ground are measured while in others, external currents are introduced into the ground and differences in potential or resistivity measured. The following main variants are available. (1) Natural current of

self-potential or spontaneous polarisation methods, (2) Resistivity methods, (3) Equipotential line methods (4) Electro-magnetic methods (5) Potential drop ratio (PDR) methods.

In an orebody beneath a soil mantle, difference between chemical activity of the upper part and that of lower part causes

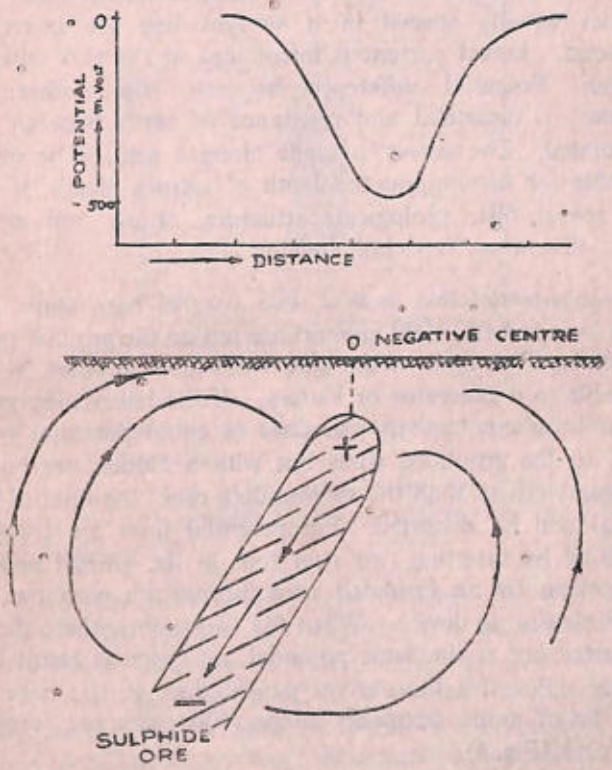

Fig. 8. Self potential over a sulphide ore body.

electrical current to flow downwards through the body and to return upwards through the enclosing rock. These currents are traced on the surface by placing two porous pots in watered holes in the ground and connecting them through a sensitive galvanometer. Lines connecting equal potential readings are plotted on a plan to reveal the centre of the metallic body

(Fig. 8). This method has limited application as it is not suitable for regions marked by arid climate and thick subsoil. Care should be exercised to take note of anomalous results in situations of water bodies, shore lines of lakes, marshes and clay-sand contacts, etc.

Resistivity method is based on the principle that different rocks and ore bodies have different electrical resistivities. Four electrodes equally spaced in a straight line are inserted in the ground. Direct current is introduced at the two outermost electrodes. Potential difference between the other two electrodes is measured and resistance of earth between them is calculated. The survey is made along a grid. The method is suitable for determining the depth of bedrock, depth of water table, gravel fills, geological structure, strata and mineral bodies. (See also "bore hole logging.")

In *equipotential line method*, two parallel bare wires about 1000 m long and 600-1000 m apart are put on the ground, pegged at intervals. The ends of the bare wires are connected by insulated cable to a generator or battery. If the intervening ground is of uniform conductivity the lines of equal potential will be parallel to the grounded wires but with a hidden ore body of better conductivity than the surrounding rock, the lines of equal potential will be distorted. Equipotential lines are traced on the ground by inserting two steel rods in the ground and connecting them by an insulated wire through an earphone or a suitable measuring device. When the two points where the rods are inserted are at the same potential, no signal is heard in the earphone. For this the current passed through the bare wires should be of audio frequency range (500 cycles per second to 1500 c.p.s.) (Fig. 9).

This method is useful for shallow deposits where interference with water bearing beds is absent.

Electromagnetic methods utilise the principle that a current passing along a conductor (orebody) sets up an induced current around the conductor and if the conductor lies within the field, it sets up a secondary induced current around it. There are two main variants : (1) Loop method, (2) Single Cable method.

In the *loop method*, a rectangular loop of insulated cable is laid on the ground and an alternating current of 500-1000 cycles is supplied by a petrol-driven generator. Survey lines are laid at right angles to the longer axis of the loop (also normal to the expected conducting ore body) and readings of induced secondary fields in horizontal and vertical coils are taken at

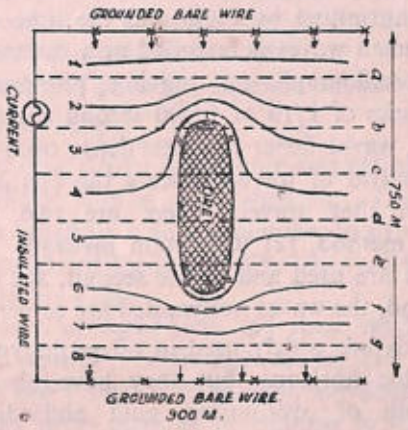

Fig. 9. Distortion or equipotential lines due to presence of a conductor orebody.
1, 2, 3, etc. are actual equipotential lines across the field.
a, b, c, etc. would be equipotential lines if the conductor was not there.

intervals of, say, 15 m on these survey lines. As the observer walks away from the loop on the survey line, he gets decreasing results but the readings increase as hidden conductor (ore body) is approached, falling off as it passes. Similar observations on parallel survey lines would mark off the size, shape and position of the conductor. The method is precise, though slow and is used to check results of other faster methods. Loop may be laid on the surface, and readings taken in underground workings.

In *Single cable method*, a long straight cable (maybe 2-3 km long) grounded at both ends, is laid and measurements taken on straight lines normal to the length of the cable. Cable may be used underground, after removing rails, pipes and wires.

While interpreting results, conducting bodies other than ore-bodies such as water-bearing strata, fault zones, graphite, slates, etc. must be allowed for.

Seismic Method

In the seismic methods, an artificial earthquake is created by detonating a charge of explosives in a shot-hole. The time of explosion is transmitted by radio and the time of arrival of reflected or refracted waves is recorded at a distance of 1000 m or so, by an oscillograph on moving photographic film on which time marks of 1/10 or 1/100 second are marked. Denser rocks transmit waves faster than less dense ones ; hence the relative time of arrival of waves indicate the character of material through which they travel. There are two main variants : (1) Reflection method, (2) Refraction method. In the first, the reflected waves are used and in the second, the refracted waves ; in either method, the waves not wanted are screened out.

The methods are widely used in oil geology to determine the key stratigraphic horizons, but they have also been used to determine depth of overburden, gold and diamond gravels, etc.

Radioactivity Method

Radioactive minerals such as those of Uranium and Radium are detected by measuring the effects of their radioactivity on various devices utilizing ionization chambers. The Geiger-Muller Counter detects surface or underground radioactive emanations but its use for detecting hidden ores is limited as the most penetrating natural radiation is absorbed by a few metres of soil or rock cover. Most rocks are radio-active to the extent that their radioactivity can be measured ; acid rocks are more radioactive than basic ones. Certain ground waters may also contain radon, or soluble radio-active material.

In concluding this section, it is to be stressed that the most efficient geophysical method to be used can only be decided after some knowledge of the geology of the area, and that the geophysical methods are only tools to be selected with care for specific jobs. There is no geophysical method universally applicable for all geological settings.

6. CORRELATION OF BEDS AND BOREHOLE LOGGING

The geologist has to correlate the coal seams and other rocks during prospecting. Very often the bore holes are not vertical but slightly deviated. It is necessary to find out the vertical depth of the beds to know their horizons if the borehole is very deep. For the shallower bore holes, not exceeding 1000 m, drilled in coal bearing and iron ore bearing formations, the bore hole depth is assumed to be nearly vertical and no special devices are used to know the deviation of a hole.

For correlation the geologist depends on the following :—

(1) *Lithologic Characters*

The driller, during drilling, gets approximate idea of the rocks from the rate of penetration of the drill bit. The sludge coming to surface from borehole with the circulation of water also gives idea of the rocks passed through. The cores of the rock act as a positive check on his guess.

The sequence of rock beds and their thicknesses, the sequence of coal seams, their thicknesses, the stone partings in these seams, the presence of marker horizons like sandstone or shale, etc. are most obvious indications to the geologist for correlation.

(2) *Micropalaeontology*

The cores of coal seams are studied for microfossils. A coal seam may contain same fossils over a widely distributed area.

(3) *Heavy Mineral Analysis*

The percentage of heavy mineral content in sandstones, coal, etc. is often characteristic of the particular bed only.

(4) *Analysis of cores and sludge*

The cores and sludge are sent to laboratory for proximate or commercial analysis. Where detailed analysis is likely to take more time, even ash and moisture content give sufficient idea for initial correlation of coal seams as drilling proceeds in the field.

(5) *Borehole logging*

This is used on a large scale for confirming correlation of coal seams and has, therefore, been explained in detail in the following pages.

Borehole logging

(i) RESISTIVITY METHOD

Depending upon the number of electrodes used to measure the resistivity of the rocks, the method is further classified as single electrode system, two electrode system, three electrode system and four electrode system. The single electrode system is simple and therefore widely used ; for example, by G.S.I., in Karanpura field and by I.B.M. at Kathara (Bermo-Kargali field) in 1961-62. The borehole to be logged is filled with water ; only its uncased portion can be logged by this method. Two electrodes are used. One is inserted into the surface, say within 3 m to 6 m from the borehole and the other is lowered by a cable in the bore hole at the rate of 3 m to 5 m per minute. The descending electrode is lowered with the help of a cable drum and the cables of both electrodes are electrically connected with electric circuit of a small petrol-driven generator or battery which passes current through rocks via electrodes. As the electrode descends, an automatic recording pen marks on the right half of a graph paper mounted on a geared rotating drum, the resistivity in ohms. The return (upward) travel of the electrode can be utilised to mark the self potential in millivolts on the left side of the same graph paper. The resistivity curves are more reliable for coal seams. Figure 10 shows the results of electrical logging for some seams of Sayal Colliery. Their comparison with the core logs obtained during drilling is interesting. The high resistivity of coal seams compared to shales or sandstones may be noticed in the log.

(ii) RADIOACTIVITY METHOD

All rocks possess some amount of radioactivity which may be remarkably constant and therefore characteristic for a given stratum over wide areas. Shales as a class, particularly black or dark coloured shales, are by far the most radioactive of all the sedimentary rocks. Coals are very weakly radioactive, and sandstones show low radioactivity. Since radioactivity depends on the parent igneous rocks, many exceptions may be encountered.

Two logging methods involving nuclear properties are in common use. The first utilises the natural radioactivity of the

rocks ; the second is based on the capacity of the nuclei in the rocks to scatter and capture neutrons. Upon neutron capture, most nuclei emit gamma rays through a process known as radioactive capture. Of all the three types of radioactive emana-

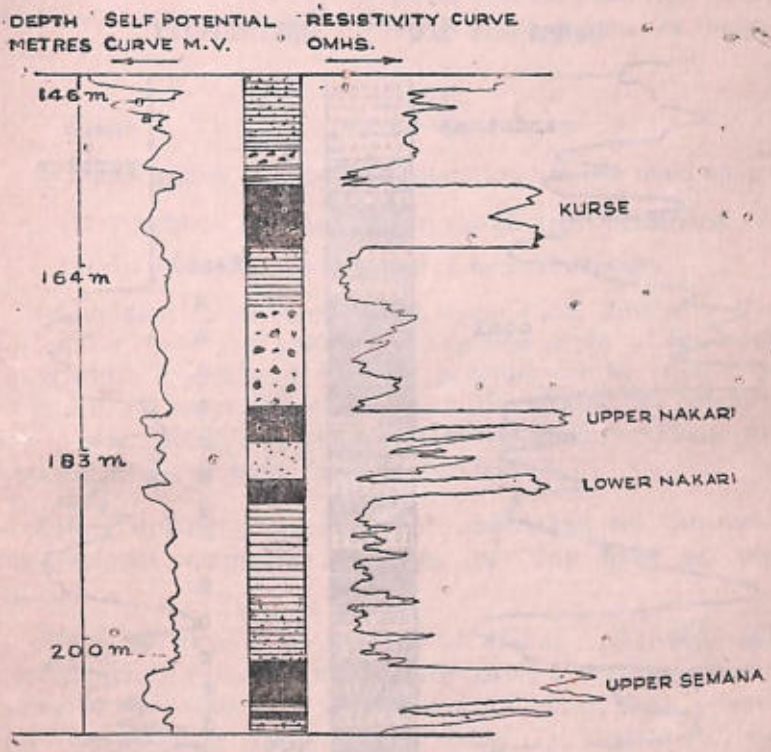

Fig. 10. Electrical log of a borehole at Sayal colliery.

tions, gamma rays are the only ones at all suitable for well logging since they are by far the most penetrating. The apparatus for gamma ray logging consists of a cylindrical "bomb" suspended from the surface by water proof electrical cable. Near the lower end of the bomb is an ionization chamber for detecting the gamma radiation, while the upper end contains a preamplifier.

Unlike the electrical resistivity method, gamma ray logging is possible even for the portion of a borehole having steel casing.

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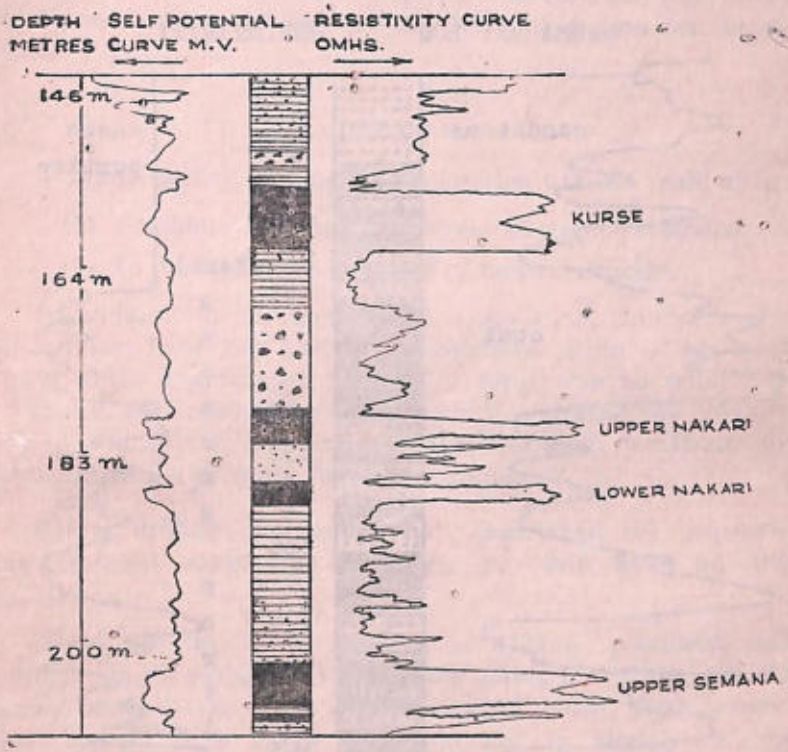

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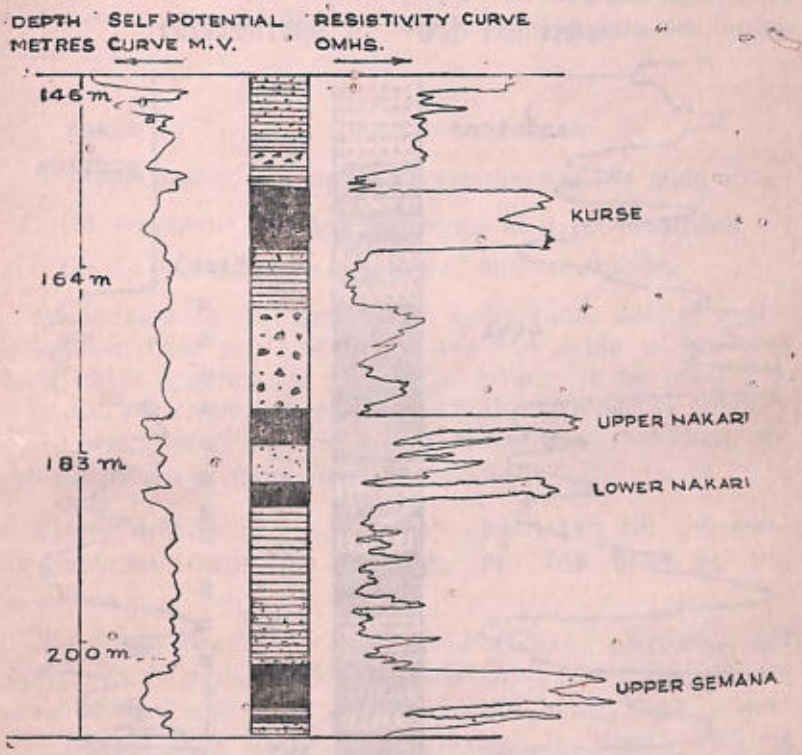

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Unlike the electrical resistivity method, gamma ray logging is possible even for the portion of a borehole having steel casing.

I.B.M. had used gamma ray loggers for boreholes in coal formations. It was observed that such loggers are somewhat complicated and need more skill in handling than the electrical loggers. In appearance gamma ray logs closely resemble electrical logs. (Fig. 11)

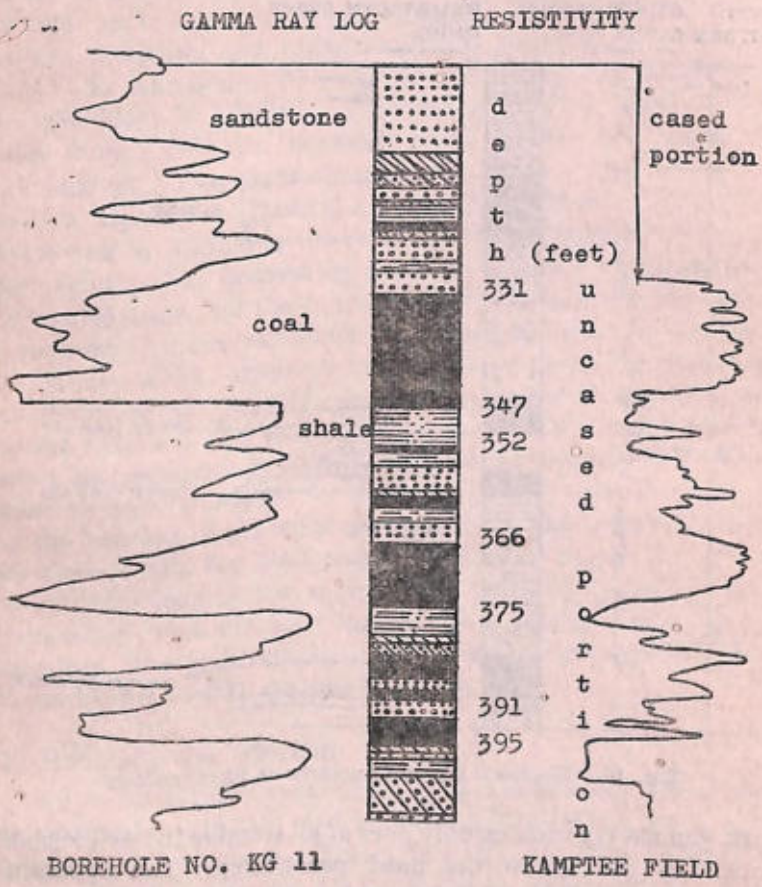


Fig. 11. Gamma ray log of a borehole (Kamptee coalfie l
Courtesy—Indian Bureau of Mines.

(iii) CALIPER LOGGING

This is based on the principle that with the same drill bit a borehole in softer rocks is slightly larger in diameter than in harder rocks. A spring loaded caliper is sent down the bore

hole so that it remains always in contact with the sides and the diameter as measured by the caliper is recorded automatically on a sheet of paper mounted on a revolving drum. The contacts of soft and hard rock, and the depth of the rock in the bore-hole can thus be known and correlated with the other logs. This is at present not practised in India for coal or iron ore formations.

7. DRILLING.

As stated earlier, drilling for exploration has two main aims :

(1) To obtain records and samples of strata penetrated

(2) To determine the structure of mineral deposits.

In addition to its main use in exploration, drilling is also undertaken from the surface to determine depth of bed rock upon which foundations of heavy structure can be safely constructued, for supply of water or sand for stowing, for conducting power cables, for ventilating underground workings, for draining off gas or water from old workings.

Before drilling programme is undertaken the prospecting geologist marks the positions of drill holes on the ground.

There are two main methods of drilling : percussive and rotary. In the percussive method, the boring tool at the end of a string of drill rods gives a succession of blows, fresh ground for the successive blows being obtained by slightly rotating the tool between successive blows. An improvement is obtained by using steel wire rope in place of rigid drill rods and the drilling in that case is known as *cable drilling or churn drilling*.

In the rotary method, the drill tool is rotated in the hole and as it rotates, it cuts, crushes, and abrades the rock. Important aspect of rotary drilling is the possibility of recovery of cores of strata penetrated. There are two main variants in rotary drilling (1) Diamond drilling (2) Calyx or chilled-shot drilling.

Table 1 summarises the field of application of the most important of drilling methods.

TABLE 1.

Uses and limitations of common methods of exploration drilling.

Type of drill	Common sizes of drill holes (mm)	Max. economic depth of hole (m)	Types of bit used	Rock formations where used.
1. Percussive with rods.	50-75	250-300	Chisel shaped	Sand and clay and sedimentary rocks of soft and medium hardness ; in fissured formations: Does not give core.
2. Churn drilling or cable drilling.	75-500	300-600	Steel choppy bits of many styles.	Placer deposits; evenly firm and moderately soft formations.
3. Diamond drilling	38-200	100-3000	Diamond bits of various types : Coring, casing plug types ; tungsten carbide bits of above styles ; tricone bits mostly non-coring.	Any rock formation except fissured. Holes can be drilled from surface or underground working at any angle to the horizontal ; used mainly for coring.
4. Calyx or chilled shot drilling.	75-1800	upto 450	Calyx ; chilled steel or iron shots ; crushed steel.	All rocks except the hardest ; unsuitable for soft and unfissured formations ; can drill at angles upto 35° from the vertical.

Of these methods, diamond drilling is the most widely used, both for proving reserves and assessing the grade of mineral deposits and has the following merits as compared to its closest rival, the churn drilling.

1. It can drill at any direction from the horizontal to the vertical, from the surface or from the confines of underground workings, also upwards. Churn drills can only drill from the surface and only vertically downward holes.
2. It gives core of the strata drilled through except in very friable rocks thus providing valuable information of

bedding planes, dip, lithology and finally the quality and grade and gives an excellent material for diverse laboratory work. In churn drilling, no core is obtained and the only guide is the sludge which is not as informative as the core.

3. Shape and diameter of hole are uniform in diamond drilling ; not so with churn drilling.
4. Diamond drilling is used only for small diameter holes, larger holes being uneconomic. Churn drills are however used for large diameter holes for purposes other than exploration, e.g. for ventilation, drainage, well hole blasting, carrying stowing pipes underground, etc.
5. Diamond drilling is usually slower than churn drilling in average formations but speedier and effective in hard rocks. Bits suffer costly damage or unduly large wear in conglomerates, soft rocks with nodules, and diamond drilling is unsatisfactory in producing samples from unconsolidated formations. Churn drilling is faster in depths upto 300-400 m in average rocks but slow and costly in hard rocks, tackles satisfactorily fissured rocks, conglomerates, etc. unless they are very hard and gives satisfactory samples in sand, clay and gravel.

Opinions vary as to which method causes excessive deviation of vertical boreholes. Diamond drilling, because of the rigidity of the string of rods, may be expected to cause less deviation than that by churn drilling where the tool tends to follow the softer rocks. Its free fall, which always tends to be vertical, counteracts that tendency.

With the introduction of diesel engines and tungsten carbide tipped drilling bits, rotary drilling has expelled percussive drilling with rods for prospecting or exploration purposes.

Diamond Drilling

Diamond drilling being universally used in India for obtaining cores, a few salient points deserve fuller attention. The drilling bit or crown is studded with industrial diamonds and cuts a cylindrical core of rock passed through (Fig. 12).

It is attached to a string of hollow drill rods through which water under pressure flows and flushes the cuttings to the surface leaving the core in a core barrel attached above the bit. Diamond pieces (called "stones") are usually hand-set or machine-set in the cast steel metal constituting the bit or alter-

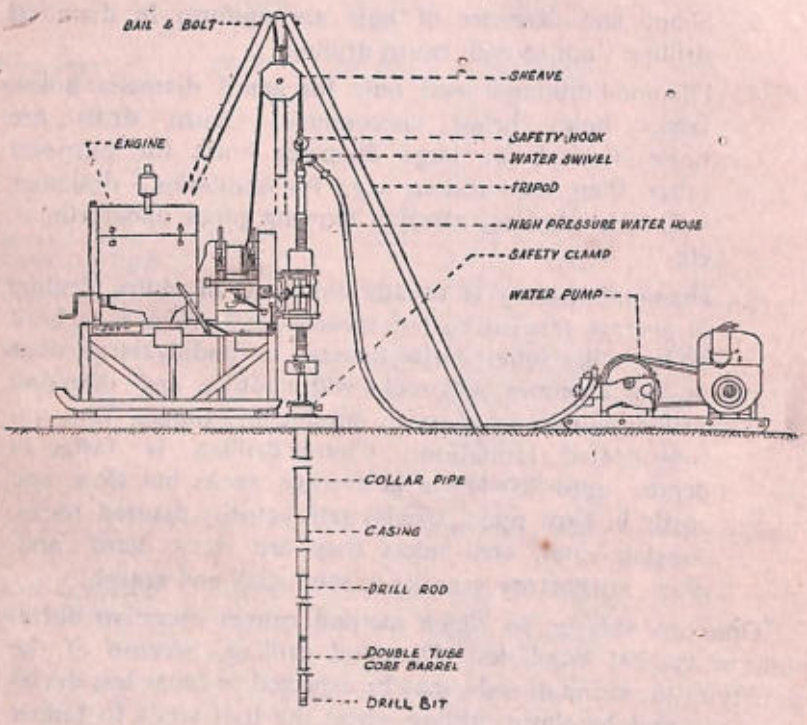


Fig. 12. Diamond drill rig used on the surface.

natively, a mould is prepared under pressure of a heated mixture of powdered tungsten carbide, powdered metallic cobalt and bortz (small carborandum stones mostly from Brazil). Sometimes only tungsten carbide stones are used in place of diamonds.

Proper selection of diamond bits for specific tasks is basic to the lower drilling cost per metre, high percentage of recovery and optimum speed of drilling. The points to note for selection are :

(1) In soft formations, the diamonds can take a deeper bite, producing larger cuttings which require larger

clearance between the rock and the matrix of the bit; hence larger diamonds must be used. Conversely in hard formations, smaller diamonds are used. Diamond size is specified by a figure denoting their number per carat, *e.g.* 16/30 connotes that there are between 16 and 30 stones per carat.

(2) In a hard ground the granule cut by a bit is small; so for obtaining fair penetration large number of small diamonds are required to give more cutting points and maximum penetration; large number of stones also help in distributing the high pressure necessitated in hard rock drilling.

(3) More waterways are needed while drilling through soft sticky formations to clear the face and give better core.

Choice of Correct Diamond Bits in Diamond Drilling

Formation	*Diamond size*	*Matrix*
1. Very soft or soft, such as chalk, fireclay, shales.	8/16	Standard.
2. Medium hard, e.g. sandstone, fractured sandstone.	16/30	Standard or hard.
3. Hard : hard limestone or dolomite.	30/60	Standard or hard.
Broken limestone	16/30	Hard.
4. Very hard quartzite, hematite, jasper, taconites.	60/100, Quality 120	Standard.
Fractured	30/60, Lesser quality	Hard.

Speed in Drilling and Core Recovery

The rate of drilling depends, to a large measure, on the nature and hardness of the rock and the depth of the hole, apart from the skill and experience of the drill crew. With new bit and

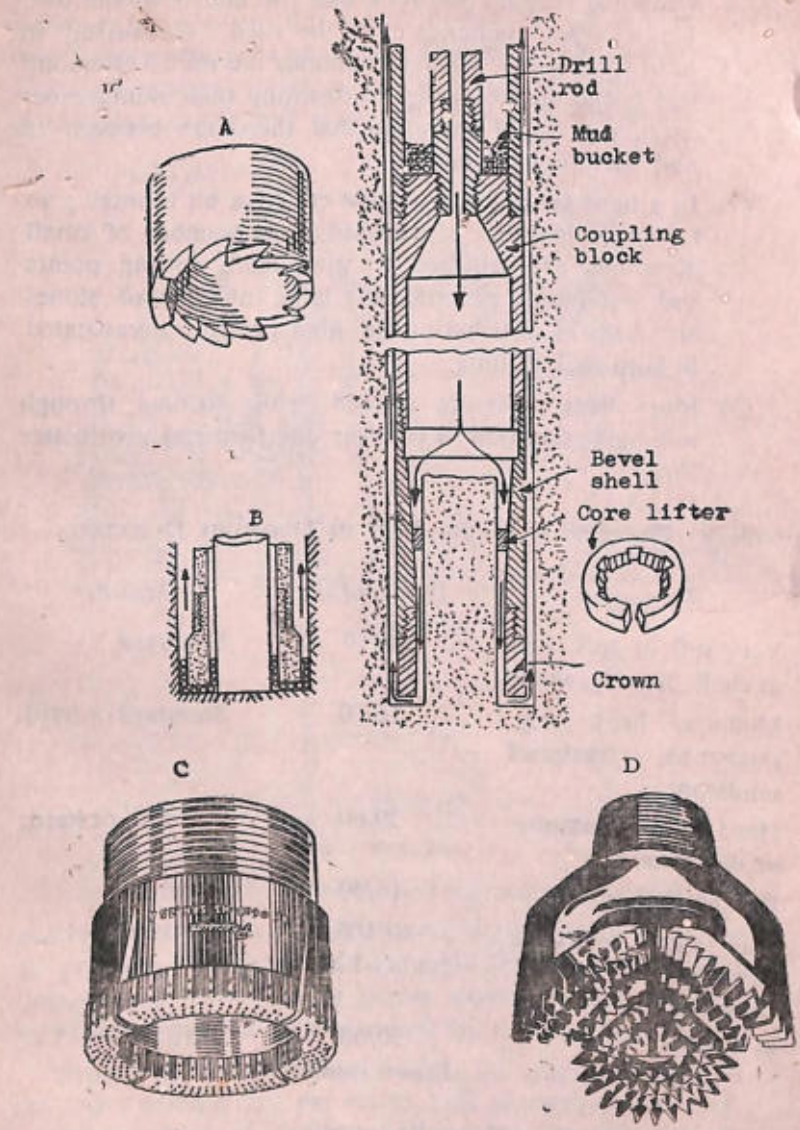


Fig. 13. Illustrating various drill bits and single tube core barrel.

A. Serrated tooth bit, coring type.

B. Calyx or chilled shot drilling, coring type.

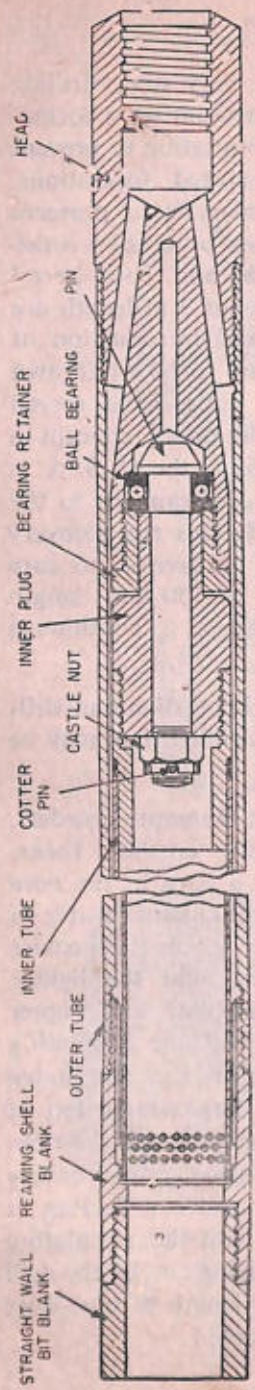
C. Diamond drill bit ; coring type.

D. Tricone rockroller bit ; non-coring type.

high speeds of rotations upto 3,600 r.p.m. very high drilling speeds have been obtained in uniform and medium hard formations. Soft rocks require slowing down of the rotation to prevent pulverising of the core while in the fractured formations, core barrel has to be withdrawn at short intervals to preserve the core from grinding and jamming. In deep bore holes considerable time is wasted in pulling out rods and core barrel when the latter is full. Normally core barrels of 3 m length are employed. Large core barrels upto 10 m and introduction of wire-line core barrel, which enables core barrel to be withdrawn without the necessity of pulling out the rods, have served to reduce this wastage of time. A new revolutionary concept in diamond drilling successfully employed in 1965 in the U. S. A. is the continuous core rig, which delivers the core samples to the surface as drilling proceeds uninterrupted. In this rig, recovery of core in soft formations is claimed at 100%. There is no core barrel but a core breaker breaks the core to 150 mm length and flows it to the surface, giving the geologist a continuous record of strata passed through.

Average speeds of penetration in conventional diamond drilling of the order of 1.2 to 3.6 m per hour of drilling time may be considered fair.

When drilling through a zone of blue dust (hematite powder), friable, flaky or biscuity hematite deposits, crushed rocks, and similar formations, it is difficult to get a core in the core barrel. In a blue dust zone, the column of circulating water in the borehole acts as a differential column in which the heavier particles of hematite remain at the bottom, and the lighter particles of silica and alumina rise to intermediate and higher levels. They are washed away to the surface with the circulating water and overflow the sludge pan. The result is that the sludge settling on the surface in a pan contains a large proportion of the heavier iron ore particles giving a false impression of the richness of the ore. The circulating water also causes caving of the blue dust zone, the caved hematite powder collecting at the bottom of the hole. It is carried away with the circulating water and yields a wrong sample of the rock being cut by the drill bit. Casing of the borehole along the entire length of blue dust zone is not economic but is occasionally adopted.



Typical BX, BWX, NX, or NWX core barrel

Fig. 14. Double tube core barrel. X design. swivel type.

Where the drill employs a screw feed, the drilling bit may not always be in contact with the bottom of the borehole when it is passing a zone of blue dust. The rate of descent of the drill rods depends upon the speed of the engine and the ratio of gearing to the screw and is therefore fixed for a given engine speed. If, however, the pressure of circulating water is high enough to erode the blue dust at the bottom of the hole and carry it away, the drill bit may not be in contact with the bottom of the hole. A hydraulic feed is an advantage in such a case as it will always keep the drilling bit pressed against the bottom of the hole. When employing a screw feed, the driller has to be particular about the pressure of the circulating water.

These difficulties have been overcome to some extent by adopting air flushing instead of water circulation during drilling. One company engaged in drilling boreholes for prospecting in Goa iron ore deposits, has adopted a down-the-hole percussive unit employing compressed air for flushing to secure cores of the blue dust zone. The recovery of core in the weathered rocks near the surface, and even in blue dust zone, is claimed to be 100%.

X-Ray Drilling

This name has not the slightest relation with X-rays as they

are understood in the medical science. The X-ray drill is just a diamond drilling equipment consisting of a skid-mounted portable drill run by a built-in petrol engine. It has no derrick, can be easily carried by 2 persons and is quick to instal and commission. It has a single tube core barrel and gives core of 19 mm dia. Its drilling capacity is 45 m depth, but is generally preferred for drilling upto 30 m depth. In coal measure strata the drilling rate per shift is nearly 20 m with petrol consumption of 6 to 8 litres. Its operation is limited to drilling shallow holes in outcrop region in moderately hard strata as are normally encountered in coal measures. The core recovery is poor, 20 to 50%, for two reasons ; (1) single tube core barrel (2) opera-

TABLE 2
Nomenclature

Equipment	Two-letter names	Three-letter names
Drill rods	EW, AW, BW, NW Current standards.	....
Core barrels and parts.	EX, AX, BX, NX Once standard but now superseded by W group.	EWX, AWX, BWX, NWX Current standards; X design core barrels threaded for W group drill rods.
		EWM, AWM, BWM, NWM Current standards; Core barrels of M design threaded for W group drill rods.
Casing	EX, AX, BX, NX, Current standards.	....

Relation of former E, A, B and N rods, and of present EW, AW, BW and NW rods with EX, AX, BX and NX hole sizes.

Hole size	EX		AX		BX		NX	
Drill rod size	E	EW*	A	AW*	B	BW*	N	NW*
Hole diameter, in.	$1\frac{1}{2}$	$1\frac{1}{2}$	$1\frac{15}{16}$	$1\frac{15}{16}$	$2\frac{3}{8}$	$2\frac{3}{8}$	3	3
Drill rod diameter, in. (outside)	$1\frac{5}{16}$	$1\frac{3}{8}$	$1\frac{5}{8}$	$1\frac{3}{4}$	$1\frac{29}{32}$	$2\frac{1}{8}$	$2\frac{3}{8}$	$2\frac{5}{8}$

*Current standards.

tion in outcrop zone where the strata are somewhat weathered and therefore liable to be washed away with the circulating water. The boring rods experience lot of vibrations and as the machine gets old, core recovery may be no better than sludge.

The machine enables the driller to get a fairly rapid picture of the rocks below the ground surface ; hence, probably, the same X-ray drill. It may not be suitable for the harder rocks of iron ore formations.

Standardisation of Drilling Equipment

The drilling rods, bits and other equipments were standardised by Diamond Core Drill Manufacturers' Association, an international organisation, as early as in 1930. The hole sizes were designated as EX, AX, BX, and NX. In recent years these standards have been revised and the new standards are designated by the letter "W" in the hope that they would become world standards. The nomenclatures are clarified in Table 2.

TABLE 3

Craelius and D.C.D.M.A. (American) Stanards for coring drill bits, cores and casing pipes.

Standard	Coring bit dia (outside)	Hole dia	Core dia (With D & T type bit)	Casing dia	
				O.D.	I.D
Craelieus	mm	mm	mm	mm	mm
	36	36	22	..	..
	46	46	32	44	37
	56	56	42	54	47
	66	66	52	64	57
	76	76	62	74	67
	86	86	72	84	77
D.C.D.M.A.	Inches	Inches	Inches	Inches	
EX	$1\frac{7}{16}$	$1\frac{1}{2}$	$\frac{7}{8}$	$1\frac{13}{16}$	$1\frac{1}{2}$
AX	$1\frac{37}{32}$	$1\frac{15}{16}$	$1\frac{1}{8}$	$2\frac{1}{4}$	$1\frac{29}{32}$
BX	$2\frac{5}{16}$	$2\frac{3}{8}$	$1\frac{5}{8}$	$2\frac{7}{8}$	$2\frac{3}{8}$
NX	$2\frac{15}{16}$	3	$2\frac{1}{8}$	$3\frac{1}{4}$	3

In metric system Craelius Standard is accepted by the Drilling Equipment Manufacturers on the continent of Europe and U.S.S.R. Table 3 gives comparative data on sizes of coring bits, drill holes and casing pipes of drills using the two standards ; Craelius and D.C.D.M.A.

Sampling

The material through which the drill bit has passed is represented by two samples : core and sludge. Core is caught in the core barrel and sludge is collected from the circulating water which brings with it the rock fragments cut during drilling. As core is removed from the core barrel, it is placed in a box having shaped compartments, each run being separated from the next one by a small plate marked with the length drilled. The length is obtained by counting the number of drill rods and correcting for distance between the hole collar and top of last rod. The portion of the core equal to the length from the face of the bit to the bottom of the core barrel, which remains in the hole must be put in the core box before the marked plate, when recovered.

Driller's log generally records the state of softness or hardness of formations and other behaviour of the machine indicating whether the core is being obtained solid or as sludge. Geologists' log should contain the following information :

(1) Length of core recovered from each run, percentage recovery in each run.
(2) Grain size of the rock sample.
(3) Minerals that can be recognised.
(4) Cleavage or schistosity.
(5) Type of alteration.
(6) Angle of dip of strata to the axis of the core.
(7) Veinlets.
(8) Location where broken or crushed core is got indicating disturbance.

In some instances, the core is colour-photographed to form a permanent record supplementing the description in the geologist's log book. After logging, the portion of the core whose assay is required is split up into two halves longitudinally by a

mechanical device (called *core splitter*). One half is examined for external evidences and sent to laboratory ; the other half is retained in the core box for record and future reference, if any.

Sludge forms a greater volume of the sample than the core and should be treated with respect ; it assumes greater importance when core recovery is not satisfactory, which is often the case. In all important exploration drillings when economic deposit is struck, return water and washings should be collected in barrels or sludge boxes and after allowing enough time for sludge to settle (if necessary, the process is hastened by adding alum or lime), clear water is removed and sludge collected for assaying. To obtain full recovery of sludge,

(1) all rods must be thoroughly washed with soap water before commencing to drill in,

(2) hole must be thoroughly washed,

(3) it must be ensured that there is no loss of water by plugging the crevices by cementing or introducing oatmeal, animal manure, etc. or if it cannot be avoided, the loss must be allowed for,

(4) in extreme cases when orebody is important, the portion of hole above may be cased.

Sludge and core samples are assayed separately and their values are combined in proportion of their volumes, to get representative assay of the deposit. In the event of an incomplete recovery of both core and sludge as assessed by comparing the actuals with the theoretical volume of the hole, the following method (after Moehlman) gives more reliable results. Theoretical volume of the hole is calculated by assuming the diameter of the hole equal to the diameter of the bit at outside stones plus 0.75 mm and the diameter of the core is taken at a value equal to diameter of the bit at inside stones less 1.5 mm. Core length is measured and its theoretical volume in percentage of the volume of the hole is computed ; theoretical sludge volume in percentage is 100 minus core volume and these values are placed in column 1 in the following example. Assuming that unrecovered part of core is ground to sludge and should normally have

appeared as sludge, weight for core is taken at 100 and that for sludge is taken at a figure equal to :

$$100 \times \frac{\text{Weight of sludge actually recovered}}{\text{Weight of theoretical volume of sludge that should have been recovered.}}$$

Example : Volume of core recovered = 10% of the volume of hole.

Amount of sludge recovered = 50% of (amount of hole excavation—core weight)

Assay value of core = 60% Fe.

Assay value of sludge = 65% Fe.

TABLE 4

Method of calculating average assay in the event of incomplete recovery of both core and sludge

	Volume in percentage of theoretical volume of hole	Weight in terms of percentage of theoretical volume recovered	Product (1) × (2)	Assay value	Product (3) × (4)
	1	2	3	4	5
Core	10	100	1000	60% Fe	600
Sludge	90	50	4500	65% Fe	2925
Total	100		5500		3525

Average assay : $\frac{3525}{5500} = 64\%$ Fe.

If the recovery of samples through drill holes is of high order, several samples from a large number of places provide a fairly representative idea of the grade of the deposit, except in the case of a precious metal deposit.

8. COST OF PROSPECTING AND EXPLORATION IN SOME PROJECTS

It may be realised that mining areas differ in their geological structure so much that no definite norms can be laid down for the expenses on prospecting and exploration in terms of the percentage of the total cost of the eventual project. Drilling forms most expensive part of the amounts spent on exploration ; in present day prices (1966) it costs as much as Rs. 100-120 per metre of drilling on an average in the normal conditions in coal and much more in deep drilling and in casing the strata drilled through. A big coal mining establishment has been working on drilling with grid intervals of 300 m for proving areas for underground mines and 150 m for opencast mines in a project of comparatively normal structure ; closer intervals have been adopted in geologically disturbed areas. In one instance to prove 240 hectares of lease, as many as 115 boreholes had to be put. In another extensive exploration programme for coal mining an average of 10-15 boreholes were put down per sq.km. for regular deposits and 40-45 holes in disturbed formations. In iron ore formations, boreholes at 50 m intervals on a grid pattern for detailed prospecting have been observed to give good results in geologically disturbed areas.

In project reports, it is usual to express the cost on prospecting and exploration as cost per million tonnes of commercial reserves, commercial reserves being defined as tonnes of coal that can be raised to the surface from the entire leasehold throughout the life of the project. Table 5 gives some data of amounts spent in recent coal mining projects on prospecting and exploration.

TABLE 5

Amounts spent on prospecting and exploration in coal mining projects

		Project A	Project B	Project C
1	Commercial reserves, Mt—	78.0	24.0	38.6
2.	Capital investment on the project, MRs,	73.68	31.74	32.96
3.	Metres drilled	..	9,888	708
4.	Cost on prospecting and exploration, MRs,	1.06	0.75	0.5
5.	Cost of prospecting and exploration, Rs, per Mt of reserves	14,000	31,000	73,000
6.	Cost of prospecting and exploration per cent of total capital investment ..	1.44	2.36	1.52

In Bolani, 353 nos. of 7.5 m deep trial pits were dug on a grid of 150×150 m squares to prove 52.7 Mt of ore. Further reserves of 365 Mt of proved and 134 Mt of indicated reserves were suggested in 61 nos. of 15 m deep pits on a grid of 150 m × 150 m and 24 nos. of diamond drill holes. Trial pits were 1.8 m × 1.8 m upto a depth of 7.5 m, 1.5 m × 1.5 m to a depth of 12 m and 1.2 m × 1.2 m to a depth upto 15 m. A 15 m pit took 22 days at an average cost of Rs. 120 per metre, proving the superiority of pitting to drilling in shallow depths.

REFERENCES

PEELE, R., "Mining Engineer's Handbook. Vol. I"
Land Acquisition (Mines) Act, 1885.
The Mining Leases (Modification of Terms) Rules 1956.
Mineral Conservation Rules, 1960.
...inerals (Regulation and Development) Act 1957.
Minerals Conservation and Development Rules, 1958.
Coal Board "Coal Board Manual, 1961".
VOROBJEV AND DESHMUKH, "Advanced Coal Mining".
SHEVYAKOV, L., "Mining of Mineral Deposits".
McKINSTRY, "Mining Geology" 1962.
BATEMAN, A. M., "Economic Mineral Deposits", 1962.
LAHEE, "Field Geology".

CHAPTER 6

OPENING A MINE

1. General terms and conditions of mining leases. 2. Size and life of mines. 3. Outlines of a mine project report. 4. Preparatory and field work for a new project. 5. Railway siding.

1. GENERAL TERMS AND CONDITIONS OF MINING LEASES

The prospector or any other individual or company intending to undertake mining has to obtain mining lease from the State Government, before commencing work ; to be eligible to obtain mining lease, he has to produce a Certificate of Approval and a certificate of income tax clearance. The prospector has a preference over other entrepreneur who has not prospected the area for which mining lease is sought. The terms and conditions of the mining leases are specified in Mineral Concessions Rules, 1960, made under the Mines and Minerals (Regulation & Development) Act, 1957.

Mining lease in respect of coal and iron ore is granted for a period of 30 years and may be renewed for one or more such periods if authorised by the Central Government in the interest of mineral development. Maximum area for which one or more mining leases may be issued in any one State to any private agency is 10 square miles (25.9 sq km) ; this restriction may be relaxed by the Central Government for reasons to be recorded. The length of the mining lease area should not exceed four times its breadth.

The Central Government after consultation with the State Government may notify its intention to undertake mining operations in any specified area not already held under any prospecting licence or mining lease and in that event no prospecting license or mining lease will be granted to any other operator ; it will be liable to pay the usual prospecting fees, royalty, surface rent and dead rent, etc.

Granted under the Mineral Concession Rules, 1960, the mining lease indenture specifies the location and area of the lease and bestows the powers and privileges on the lessee :

(a) to enter upon the land and search for, mine, dig, drill, win, dress, process and convert, carry away and dispose of the minerals for which the lease is granted,
(b) to sink, drive, make pits, shafts, inclines, etc. in the said lands,
(c) to bring and use machinery, equipment etc. on the said land,
(d) to make roads, ways, etc. and use existing roads and ways,
(e) to get building and road material from the land only for the purpose of mining,
(f) to use water from streams, etc., with the permission from the Deputy Commissioner/Collector,
(g) to use land for stacking, heaping and depositing the mineral or waste,
(h) to beneficiate any ore or carry away the produce.
(i) to make coke from coal produced in case the lease is for coal,
(j) to clear bushwood and fell and utilize trees, subject to payment at specified rates.

The following restrictions and conditions are generally prescribed for the exercise of the above-mentioned powers and liberties.

(1) The lessee shall not build any structure on burial ground, or sacred place or interfere with right of way, well, or tank.

(2) Before using any new land, the lessee must give two months' notice to Deputy Commissioner and, if any objection is raised within the notice period, the land shall not be used.

(3) For felling trees and entering reserved forest, express sanction from District authorities has to be obtained.

(4) To work within a distance of 50 m from a railway line, permission from Railway Administration, and for

reservoir, canal and public works, permission from authorised State Government Officer is required. Similarly for working within 10 m of village road, permission from Deputy Commissioner is to be sought.

(5) Reasonable access facilities which do not amount to hindrance or interference must be allowed to adjoining or enclosed licensees or lessees.

State Government may allow any other lessee or authorised agent to work other mineral or to make and maintain railways or roads, lay pipe line etc., in the area of the mining lease, provided such activities do not interfere with or hinder the work of the main lessee ; the lessee is, however, entitled to fair compensation in the event of loss or damage resulting from these activities.

The lessee has to pay every subsequent year the yearly dead rent or royalty for each mineral mined, whichever is higher. The prevailing rates of dead rent and royalty in respect of coal and iron ore are as follows :—

	Dead rent per hectare	Royalty
Coal	Rs. 2.50 to 12.50	5% of F.O.R. price, subject to minimum of 50 P. per tonne.
Iron Ore	Rs. 2.50 to 12.50	Rs. 1.50 per tonne for ore containing more than 62% Fe. Re. 1/- per tonne for less than 62% Fe.

The Central Government has powers to enhance the rate of royalty only once in four years but not so much as to exceed 20% of the price at pit head. In addition, the lessee has to pay rent and water rates per annum per hectare to the State Government in respect of all parts of the surface of the land occupied and used by him from time to time except that land to which the public have full right of access. To facilitate computation

of royalty correct account shall be maintained and kept for scrutiny of the Government of the mineral produced and despatched, in stock or in the process of exports and of the mode of arriving at the prices of minerals. The State Government has rights to recover the sums due to the lessee as arrear of land revenue.

The lessee has to commence mining operations within one year of the execution of the lease and work in a workmanship manner and indemnify the Government against damage or injury. He must secure and keep in good condition working shafts and pits. He must strengthen, support and maintain safe the railways, reservoir, canal, road or such public works and structures. He must allow and give all facilities for inspection of workings, report accidents causing death or serious bodily injury or serious injury to property. He has to report to the State Government discovery of other minerals with all particulars within 60 days of the discovery and has to allow work for these minerals by authorised leaseholder. He must maintain correct plans and sections of the mines, supply them to authorised Government officers on demand, and give facilities of inspection. The lessee shall furnish to Director-General, Geological Survey of India, Calcutta, all geophysical data relating to mining fields or engineering and ground water surveys collected by him in the course of mining operations; similar information pertaining to radioactive minerals has to be furnished to Secretary, Department of Atomic Energy, New Delhi. The Director of Geology and Mining of the state in which prospecting or mining is being done has to be informed, in addition, in both cases.

The lessee may with previous consent in writing of the State Government acting in concurrence with the Central Government transfer the lease in respect of coal or iron ore or any right therein to a person holding a certificate of approval and income-tax clearance certificate, on payment of Rs. 100/- to the State Government and make him available the original or certified copies of all plans of a belt of 65 m surrounding and including the abandoned workings.

The lessee shall not allow himself to be financially controlled by a trust, syndicate, corporation, firm, or person

without prior consent of the Central Government. On termination of the lease, the lessee must surrender workings together with their equipment and fittings in good order to the State Government. The State Government has the right of pre-emption of the minerals and their products lying under the control of the lessee and is liable to pay fair market price.

A full time mining engineer must be employed if the average employment in a mine exceeds 150 below ground and 400 in all the mine and a part time mining engineer in case of less employment. In addition, if the Controller of Indian Bureau of Mines so desires, a full or part-time geologist has to be employed. Employment of a non-Indian in connection with mining operation is prohibited except with permission of the Central Government. The lessee has to provide facilities of practical training to students of mining or geology of institutions approved by the Central Government. The mine operator working the mine under the land acquired for public purpose or for a Company with a declaration that mines and minerals under the land are not needed as prescribed by Land Acquisition (Mines) Act, 1885, has to give a notice of 60 days of his intention to carry out mining operations under the land. The appropriate Government may cause inspection of mines and minerals to be made and offer compensation in consideration of the restrictions of work imposed by it for not working the mine under the land. In case of no such offer, mine may be worked in such a manner as not to cause damage to the work for which the land has been acquired ; in the event of damage by improper working, the mine operator has to repair the damage or to bear the cost of repair.

The Mining Leases (Modification of Terms) Rules, 1956 gives Controller of Mining Leases power to issue orders to modify and alter the terms and conditions of the existing leases to bring them in conformity with Mineral Concession Rules, 1960. Under Rule 6, he may reduce the area of the existing mining lease by excluding the area, (i) which is not being worked and which is not contiguous to any area which is being worked, or (ii) whose exclusion may render the remaining area more compact and appropriate for working the mine in a workmanship manner and will facilitate conservation of minerals.

2. SIZE AND LIFE OF MINES.

In planning mining enterprises, one has to deal with three main elements : commercial reserves, annual capacity and term of life of the mine.

There are certain terms used in connection with the treatment of proved reserves which require to be explained. Geological reserves (Q_g) are those reserves of all the deposits of a mineral whose existence is proved within the boundaries of the mining lease. Out of these, all deposits cannot be worked profitably, because of low thickness or poor quality in some portions or sections ; those which can be profitably worked are workable reserves, ($Q\omega$). The tonnes that can be raised to the surface for commercial purposes, that is, commercial reserves (Qc), will be less than these workable reserves by the amount of underground losses. Losses are of three kinds :

(1) General losses, resulting from the need for keeping protective pillars either for the safety of overlying surface or in barriers.

(2) Exploitative losses, which comprise the pillars to be left along development workings, slices or bands of the deposit left in the floor or roof and in interslice partings, or broken mineral which is left in the goaf or stopes, and

(3) Losses due to unfavourable geological conditions, such as faulted and folded zones (Fig. 1).

If these losses are estimated as L, Qw—L=Q_c and ratio $\frac{Q_c}{Q_\omega}$ is called co-efficient of recovery or simply recovery and is usually denoted by a symbol, c.

On the basis of the data obtained in prospecting and exploration of the prospect which has been proved worthy of development, decisions have to be taken on the output, size and life of the mine to be projected. Economics of scale operate very sharply in the case of coal mine projects, particularly in thick seam mining, to which the natural resources have put us heir. Table 1 compiled from Russian practice illustrates the

advantages to be gained from projecting large capacity mines ; there is a marked increase in O.M.S. and reduction in the prime cost of winning and in specific investment per tonne of annual capacity. This results from concentration of output per square metre of exploitation area and relatively small aggregate length of development workings, with accompanying reduction in costs of maintenance of roadways, transport and ventilation.

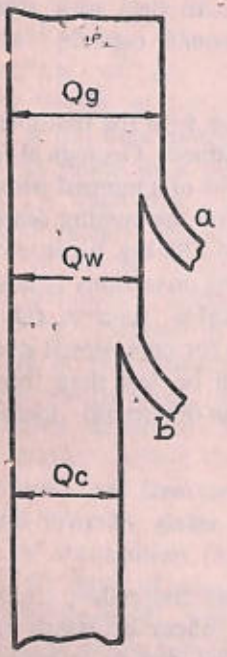


Fig. 1.
Commercial reserves of a mine.
a. unworkable reserves.
b. underground coal losses.

In deciding upon the annual capacity and term of life of a mine project corresponding to the size and reserves of a mine field, two cases arise : (1) Mine field limited in extent by geological disturbance such as faults and folds so that size, reserves and boundaries of the mine field are well-defined to support not more than one mine—limited mine field (2) Mine field extended over large area which can sustain a number of mines, leaving it to the planners to decide on the size, reserves and boundaries to be taken for a specific mine project—unlimited mine field.

Annual capacity that can be planned for a project must, in the ultimate analysis, depend on :

(1) the market for the products raised and prepared,

(2) availability of labour required for getting the proposed output,

(3) degree of priority of the project in the national economy,

(4) expected cost of production per tonne of output.

TABLE 1

Productivity, prime cost and investment per annual tonne in recent Russian coal mining projects

Annual capacity, Mt	Output per man per month, t	Prime cost. Rb./t	Investment per t of annual capacity, Rb.
Donbass			
0.3	50	6.7	26.9
0.6	54	6.5	25.2
0.9	56	6.6	22.4
1.5	61	5.7	21.7
1.8	70	5.4	18.8
Kuzbass			
1.0	68	5.0	17.1
1.5	73	4.5	14.0
Karagandabasin			
0.9	85	4.0	19.5
1.2	83	3.9	15.0
1.5	93	3.8	15.0

Limited Mine Field

Economically the optimum capacity of a mine in a limited mine field will be a compromise between the following conflicting considerations :

(1) The higher the capacity of the mine, the higher is the capital investment for its construction, though it must be conceded that capital investment is not directly proportional to the increase in the capacity of the project but may be on the line depicted by a curve a in Fig. 2.

(2) The higher the capacity of the mine, the lower is the revenue expenditure per tonne of mined coal. Revenue expenditure consists of two groups of expenditure, one which does not depend on the level of production (such

as, payment to face and piece-rated workers) and the other which decreases with the level of production (e.g., expenses on ventilation, drainage, hoisting of workers, etc.). Revenue expenditure per tonne as production level increases is illustrated by a curve e in Fig. 2.

The minimum cost of production for the project under examination is obtained by combining the two curves and finding the minimum range. This is the optimum production capacity of

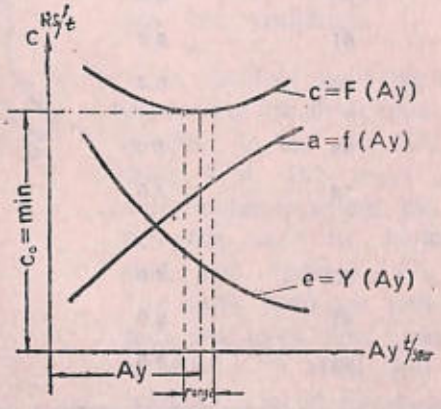


Fig. 2. Optimum size of a mine in case of limited reserves.

the project. The term of life of the project is obtained by dividing the commercial reserves by the optimum production capacity $\left(T=\frac{Q_c}{A_y}\right)$. This optimum life of a project will, it may be observed, not be influenced so much by the useful life of any particular equipment and buildings as by combination of amortisation and revenue expenditure of mining coal.

There is a tendency all over the world to pitch the annual capacity of coal and iron ore mines to as high a figure as possible. Underground coal mines have been projected for a daily output of 20,000 t in favourable mining and geological conditions when commercial reserves are high, giving a mine a useful life of 60 to 100 years. In the U. S. S. R. the coal mining projects are planned

to give the term of life corresponding to the projected annual capacity on the basis of the recommendations in Table 2. Commercial reserves ought to be adequate for the purpose.

TABLE 2

Recommended capacities and life of coal mining projects.

Daily output t/day	Annual capacity Mt/year	Term of life Not less than—years.
1,000—1,500	0.3—0.45	30
2,000—3,000	0.6—0.9	40
4,000—5,000	1.2—1.5	50
6,000—7,000 & higher	1.8—2.1 & higher	60

Average daily capacities of coal mines in some leading countries in 1959 contrasted with the average in India in 1959 and 1965 are stated below :

Average daily capacities of coal mines in 1959

Belgium	— 4,100	France	— 1,860		
Holland	— 3,800	Czechoslovakia	— 1,830		
West Germany	— 3,500	U. S. S. R.	— 1,030		
Poland	— 3,150	India	— 184	(276 in 1965)	

In recent years, in India there is a trend to plan new mines for annual outputs of more than 1·5 million tonnes as far as possible.

UNLIMITED MINE FIELD

When a mining establishment has a choice in an area of unlimited reserves uninterrupted by a natural boundary, problem that arises is to decide on the dimensions to be taken for a mine. Though generally speaking, rich deposits can sustain large-capacity mines, there is a limit to this capacity, set by increased

production cost, unjustified long periods of construction and difficulties of organisation and management. Prof. Popov has suggested an empirical formula (1) based on practice and this together with a logical formula (2) may be used to determine the dimensions of a mine field which could constitute the project.

$$\frac{S}{H} = 7\sqrt{\sin x + 1} \quad \text{..... (1)}$$

x = dip in degrees.

$$S.H = \frac{Q_c}{c\Sigma p} = \frac{T.Ay}{c\,\Sigma\,p} \quad \text{..... (2)}$$

where S —dimension of the mine field along strike, in m

H —dimension along dip, in m

Qc—commercial reserves

T —term of life of the project.

Ay—Annual capacity of the mine

c —co-efficient of recovery, that is, the ratio of commercial reserves to workable reserves. Commercial reserves are those which can be raised to the surface and workable reserves are reserves of seams which can be profitably developed and mined.

Σ p—quantity of coal in tonnes per m^2 of the seam area from all seams in the mine field.

Though there are some instances in practice, where length along the strike is 9,000-12,000 m, it would perhaps be rational to suggest that a mine field should not exceed 7,000 m along strike and 1,000-3,000 m along dip, depending on the severity or ease of gradiant. When the mine field dimension along the strike is considerable, it is expedient to divide it into blocks of 3-4 km length ; each block may be provided with separate ventilation/man-hoisting and stowing and materials pits ; coal may be transported through underground workings to the centrally located main production shaft, in which it is hoisted to the surface for central preparation plant and despatch. Such a mine is called a central mine.

3. OUTLINES OF A MINE PROJECT REPORT

After the major decision on the size of the project is tentatively taken, a mine project report is prepared. It is basically a financial proposal bringing into clear relief the capital and revenue implications and indicating the best possible ways and means of carrying out a progamme of construction and production under the conditions that prospecting and exploration operations have forecast. Among the many important aspects this report has to deal with, the following are the foremost.

MARKETING OR COMMERCIAL ASPECT

1. Grade or quality of the mineral to be produced ; probable demand and least demand.
2. Selling price of the mineral and competition from other dealers ; margin of profit.
3. Location of the probable markets and freight ; approach roads and transport facilities including railway siding near the mine and for long distance despatches.
4. Existing and likely attitude of the Government in controlling production, distribution and sale of certain grades of mineral.

Other technical and financial considerations to receive attention for production are :

1. Depth of mineral bed ; its quality, thickness, inclination ; number of mineral beds and coal seams ; extent on rise-dip and strike ; geological disturbances ; workable reserves ; etc. Technical report on its working.
2. Adjacent mines and their effect on the new mine, e.g. fire in adjacent coal mines may spread to the new mine. If new mine is on the dip side, it may have heavy water percolation from adjacent mines of the rise side.
3. If mechanised quarry is planned,
   (i) necessity of diversion of Public road, railway, stream, power and telephone lines, aerial ropes etc.
   (ii) place for disposal of overburden. The dumping plot should be so located that wind does not carry the dust to residential colony, office, workshop, etc.

(iii) effect of operation of heavy earth-moving machinery, and blasting on underground workings in the area.

(iv) rehabilitation of persons residing within danger zone of blasting range.

4. Power supply.
5. Fuel supply e.g. diesel oil in large quantities if mechanised quarry is planned.
6. Water supply.
7. Availability of sand and large quantities of water if the technical report recommends hydraulic sand stowing.
8. Drainage in the area ; highest flood level of river and its effect ; period of heavy rainfall.
9. Compensation payable to local inhabitants for damage to agricultural land, cocoanut groves, water tanks, etc.
10. Availability of skilled labour and unskilled labour. Their accommodation ; their likes and dislikes to certain methods of work ; wage structure for various categories of workers.
11. Site for residential colony, magazine, washeries, office, etc.
12. Impact of mining legislation and other legislation in foreseeable future.
13. Machinery available ; state of approach roads and strength of bridges on rivers/streams for transport of heavy machinery.
14. Capital required. A new underground coal mine takes 2 to 5 years to come to the production stage, depending primarily upon depth of seam and other factors. Rs. 100/- to Rs. 200/- per ton of desired coal output per annum are required as capital expenditure,
15. Foreign exchange involved in getting machinery.
16. Collaboration of foreign experts. if desired.
17. Washery or ore beneficiation/treatment plant.

The above considerations decide the workability of a new mine. The report is prepared by a team of technical experts, and goes by the name Project Report.

If the report gives an encouraging picture of the new project, the owner decides to open a mine.

Preparation of such detailed project report on completion of drilling operations and geologists' report, takes a long time. If the initial drilling-and-geology report is encouraging and the technical experts feel that the mine will be a profitable undertaking, a flexible plan can be drawn up to open the mine in a reasonably proved block of the leasehold.

At least one month before due date of opening a coal mine, or a seam, an application for permission has to be preferred to the Coal Board in a prescribed form and its permission obtained for commencement of mining operation. For other mineral, a 30-days' notice before commencement of mining operation is to be given to the Controller, Indian Bureau of Mines, Nagpur.

Coal Board generally grants permission if it is satisfied that—

1. The area of the lease is at least 67 hectares and has a length equal at least to twice the breadth.
2. The project will have a capacity of more than 10,000 t.p.m.
3. The propsal for developing the mine is reasonably sound as regards conservation and safety.

4. PREPARATORY FIELD WORK FOR A NEW PROJECT

Time to Start Field Work

It is usual to start the field work in connection with a mine project after the rains ; sinking shafts, driving drifts, and erection of civil structures started during the rainy season have proved costly owing to damage suffered due to collapse. The brick making season starts usually in November and the whole requirement for a year must be ordered and got ready by May.

Intimations

The Project Manager when posted on a project, should send intimations to the following authorities at least one month before commencement of work in connection with the project :

(a) Chief Inspector of Mines (under Regulation 3 of Coal Mines Regulations, 1957).

(b) District Magistrate of the district in which the project is located.

(c) Controller, Indian Bureau of Mines, Nagpur.

He should initiate contacts with, and secure the appropriate facilities from, the following government agencies :

(a) State Forest Department for permission to cut trees,

(b) Divisional Superintendent, Railways for getting railway siding for the despatch of the mineral.

(c) Chief Inspector of Explosives to construct magazines for explosives and storages for kerosene, petrol, carbide etc.

(d) State Electricity Board for supply of electricity.

(e) Electrical Inspector of Mines for approval of site of surface substation and general electrical layout.

(f) Public Works Department or District Mines Board of Health for access roads.

(g) State Police Department to open up a Police Station within 10 km or so of the project.

(h) Public Exchange Telephone for telephone.

(i) Post & Telegraphs department for opening up a post and telegraph office near the project.

(j) Employment Exchange for recruiting the required labour.

(k) Coal Mines Labour Welfare Organisation for arranging recreational and other welfare activities.

Apart from what has been stated earlier, some preparatory work is essential in the field when starting a new project, for example, the following :

For open cast mining

1. Surface rights have to be acquired from the State Government or from private land owners, as mining lease gives only underground rights to the mine owner.

2. Large scale plans (R.F. 1 : 1200) of the area should be prepared showing contours at 1.5 m intervals and surface features like railways, rivers, roads, streams, buildings, boreholes, inclines, aerial ropeways, telephone and power lines, faults, dykes, etc. in the quarriable limit and 30 m on all sides of the

quarriable limit. Residential quarters within 200 m of the proposed quarry boundaries should be shown as they lie within the range of flying rocks after blasting.

3. It is desirable to prepare plans showing seam/deposit isopachs (lines drawn through places having the same *true* thickness of seam/deposit) and overburden isopachs. These assume significance when estimating total quantity of overburden that is to be removed and are also helpful to decide the quarry limit at every phase of working.

General, for all Mining Projects

A permanent base line should be marked in the area. The base line should be nearly NS or EW in direction and between 60 m and 100 m in length, preferably on a level ground not far from the quarry or mine entrance and located in an area, which will be undisturbed by quarrying operations or movement of machinery. On the contour plan grid lines, 30 m to 60 m apart, parallel and at right angles to the base line should be marked and extended 100 m on all sides of the quarriable area. On the ground, the junction of the grid lines should be shown by masonry pillars. The base line should be similarly marked on the ground by masonry pillars. Such a plan showing surface contours and grid lines is necessary for periodical measurement of the excavation of the overburden and mineral.

Selection of site for permanent housing programme is one of the earliest decisions the Project Manager has to take. Some of the residential colonies constructed by a few Public sector undertakings, e.g. by Hindusthan Steel Ltd. at Bhilai and Durgapur, have underlined the advantages of separating workers' colony from officers' colony. At Bhilai the workers' colony is within walking distance of the plant ; the colony for supervisory staff who are slightly better off is within cycling distance. Senior supervisory staff and officers who can afford to maintain scooters or cars have their colonies still farther away from the plant. This has also proved advantageous during times of labour unrest when workers are separated from the senior supervisory staff and officers who cannot be intimidated. In a mine where underground operations are likely to cause subsidence of the surface, the permanent site for houses is selected

where surface is not likely to be disturbed by mining operations, for example,

1. Near river banks, permanent roads, railway,
2. Near dykes, faults, outcrop region (if no quarrying is to follow),
3. Areas containing burnt seams or inferior quality coals not likely to be mined,
4. Common boundary of two mines,
5. Areas enclosed by fault planes, very close to one another.
6. Near the shaft pillar.

Where mechanised quarry is planned, the site for colony and installation of permanent machinery like generators, mineral handling plant, compressors, etc. should be beyond the blasting range (300 m) from the quarriable limit and also away from the dumping yard.

5. RAILWAY SIDING

Railway siding is an expensive item of the capital investment of a mine and is an essential feature on which depends the economic disposal of the produce. The cost of a siding varies very widely. In favourable conditions when very little cutting and filling and no major bridges or culverts are involved, a track km of siding may cost as low as Rs. 4 lakhs but on a rought terrain with large earthwork and major bridges, cost may go up to Rs. 11 lakhs per track km ; Cost of track is generally in the neighbourhood of Rs. 2 lakh per km. It is, therefore, expedient to cut the size of a siding to the barest minimum required at the moment of construction, with adequate provision for future expansions.

Depending upon the financial implications involved in construction of the siding, the following types of sidings are provided by the Indian Railways for collieries :

(1) Railway siding
(2) Assisted siding
(3) Private siding.

Railway siding is constructed by the Railways on Railway land entirely at their cost for their operational and commercial convenience.

In **Assisted siding** the party for whom the siding is constructed acquires the land for the siding and makes it over to the Railways and all rights over the land will then vest with the Central Government ; cost of the earth work, bridges and culverts (excluding girders), ballast, and buildings is borne by the party, while the railways bear the cost of all works which would be removed by the Railway administration in the event of siding being closed, namely, sleepers, rails, fastenings, points and crossings, girder work of bridges, fencings, signalling, interlocking and other machinery. Assisted siding terms are offered by the Railways on being satisfied of the operational and commercial necessity, justified when new business gives a return of 15% on the capital investment by the Railway (provided funds are available).

In **private siding** all cost of the siding beyond the Railway premises is borne by the party and this is the most common form available to the mines at the moment.

Application for a siding in a prescribed form is addressed to Chief Operating Superintendent of the concerned Railway with copies to Dy. Coal Controller (P) and to Divisional Superintendent. The application should state the details of :

(1) present siding, if any,
(2) pilot working the area,
(3) present output and output for the past five years,
(4) plan of the leasehold, showing all the present railway lines and the proposed siding alignment.

After preliminary scrutiny, the Dy. Coal Controller (P) forwards the proposal for a joint investigation by the Coal Superintendent and Mining Adviser. This joint investigation report submitted by the Coal Superintendent to Dy. Coal Controller (P) and by Mining Adviser to the Divisional Superintendent, should contain (i) the recommendation of the loading accommodation determined on basis of past and present output, stocks and approved development scheme, and (ii) an alignment and layout of the siding that ensures adequate underground support, involves the least earthwork and blocks the minimum amount of mineral reserves.

On the receipt of this joint report, Dy. Coal Controller (P) and Divisional Superintendent send their reports to the Chief

Operating Superintendent, who, if he approves of the proposal, sends the case to the Chief Commercial Superintendent, Financial Adviser and Chief Accounts Officer, for concurrence.

After the proposal is finally approved by the Chief Operating Superintendent, he advises the Chief Engineer and Divisional Superintendent to prepare a detailed plan and estimate on realisation of the requisite survey fee (which at the moment is Rs. 1,000/- for the first km and Rs. 350/- for the subsequent half km or part) from the party. When the plan and estimate and other terms are accepted by the party, the construction is undertaken by the Engineering Department of the Railway concerned.

In planning for the sidings, a few basic requirements of the Railways must be kept in view. In broad gauge line sidings, Railways use crossings of 1 in 8½ (or 1 in 12) and curves of a minimum radius of 10°. In recent days, the Railways supply wagons in full rakes at a time. A full rake consists usually of 70 wagons of 4-wheeler (each of a pay-load of 22 t and with a buffer to buffer length of 8.4 m) or 28-30 box wagons (each of 55 t capacity and 13.5 m length) with one locomotive and 42-45 wagons with two locomotives in tandem. These three types of rakes require track lengths of 675 m, 450 m and 675 m respectively and railways would require these lengths provided at train reception and departure ends of the siding yard. The minimum centre to centre distance between two parallel tracks is 4.6 m ; a crossing of 1 in 8½ requires a length of 61.5 m and minimum distance apart is to be maintained beyond the fouling board of the crossing. Near the place where structures are to be installed, this distance between tracks is increased to 6 m and when two weighbridges are to be installed on adjacent tracks, to 7.5 m. In designing structures and buildings over the siding, standard clearances are to be observed. A clear height of 2.1 m over the track is to be kept in normal cases, 4.8 m, when locomotive is to pass beneath them and 5.4 m, when electric traction tracks are laid under them (Fig. 3).

The average rolling resistance of box wagons is 2 kg per tonne or 1 in 500 and, therefore, track gradient towards the load standage should be 1 in 400 to 1 in 500 ; when wagons are to reach buffer end, reverse gradients of 1 in 100 to 200 over a

distance of 60-100 m is kept, or the tail half of the load lines is kept level. Under a weigh-bridge a level length of 30 m is generally provided. A provision is also made for a derail point (trap point) or isolation siding (slip siding) to arrest runaway wagons going over to the main line.

The railway siding is aligned with respect to the loading point. There may be one loading point or two points or numerous chutes beneath an overhead bunker. The weigh-bridge may be located beneath the loading point or some distance ahead, the former position being preferred to avoid large adjustment of loads against over-or under-loading. The loading can be done when the wagon is moving, being pulled by a slow-moving haulage rope or such other contrivance, or when it is standing; a movable shuttle conveyor can be employed to load it uniformly. The yard should have good drainage away from the loading point and should involve the least cutting and filling. High filling at the loading point will enhance the cost of permanent foundation for related structures and weigh-bridge.

A simple and economic type of layout is that in which a full empty track and a full load track lie on two sides of the loading point. Depending on the economical direction of slope of the yard, either of the two tracks may lie towards the buffer. One typical alignment with one loading point is illustrated in Fig. 3,

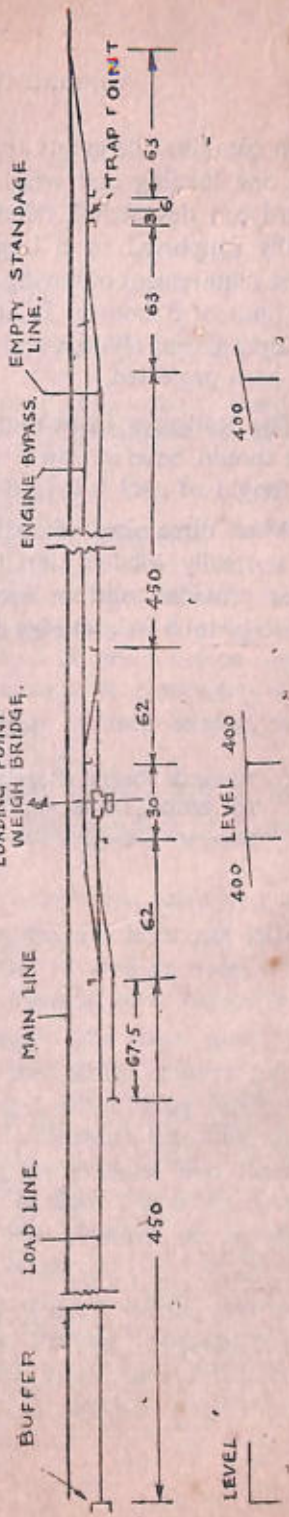


Fig 3. Railway siding at a mine.

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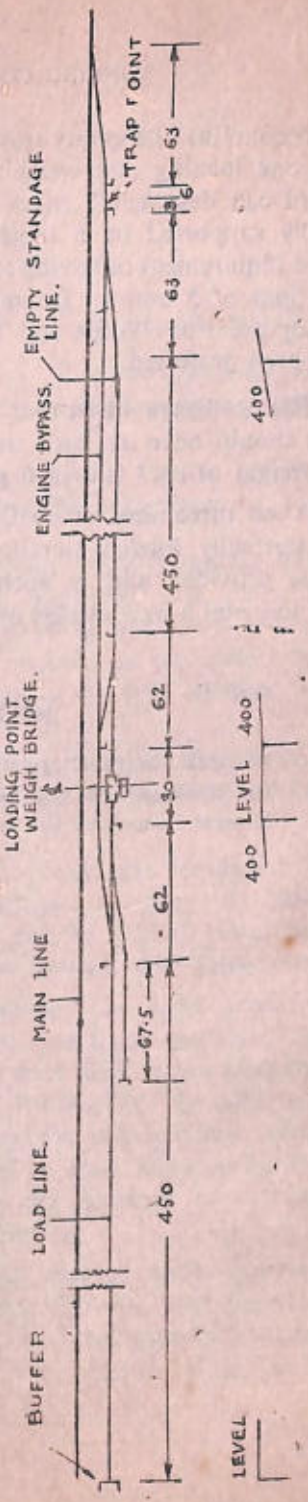

Fig 3. Railway siding at a mine.

which contains one empty track, one load line, one engine by-pass line, one loading and weighing point and a store spur. Such a yard can despatch 2 rakes a day. A single point loading is usually supported by a large capacity storage bunker to cater to the requirement of having to load wagons within the allowable time limit of 5 hours. In some recent high capacity sidings, a loading point of 600 tph and a ground bunker of 6,000 tonnes have been projected.

The Railways insist that a colliery raising 39,000 t.p.m. of coal should have its own weighbridge and the rebate available on freight of coal is then 6 paisa per tonne.

When three sizes of coal, namely, steam, rubble, and slack, are normally loaded simultaneously, three loading points are to be provided and in such circumstances, it may be necessary to instal a weighbridge away from the loading points.

REFERENCES

1. Mines & Mineral (Regulation & Development) Act, 1957.
2. The Mining Leases (Modification of Terms) Rules, 1956.
3. Mineral Concession Rules, 1960.

CHAPTER 7

OPENCAST MINING : WORKING AND LAYOUTS

1. Classification of quarrying methods. 2. Formation of benches. 3. Working and layouts in manual and semi-mechanised quarries. 4. Layouts of mechanised quarries.

1. CLASSIFICATION OF QUARRYING METHODS

Opencast mining is much simpler and more economic than mining of minerals by underground methods. The alluvium and other rocks below which the minerals lie are removed after ripping or blasting and the exposed mineral is completely extracted. Opencast mining is also known as **strip mining,** the latter term being in common use in the U.S.A.

The quarrying operations may broadly be classified as follows, depending upon the methods used and the extent of mechanisation adopted :

(1) **Manual quarrying**—Manual workers are employed to load the rock of overburden and coal/iron ore after the latter is fragmented into convenient sized pieces by drilling holes into it with the help of compressed air and blasting with explosives.

(2) **Semi-mechanised opencast mining**—The drills used for drilling blast-holes are of a larger size than those employed for manual quarrying. Employment of wagon drills is the common practice. The manual labour is used for loading the rock and mineral or sometimes only for sorting the mineral and stacking it. Tractor shovels are employed to load the overburden into trucks. For the same output the element of human labour is much less than in manual quarrying.

(3) **Mechanised opencast mining**—Heavy earth moving machinery like draglines and dipper shovels, rear-dumping trucks like Euclids, and well-hole drills for blast-holes of 125 mm dia. and above are commonly employed. Bucket wheel exca-

vators are also sometimes used for soft rocks as at Neyveli. The element of human labour employed is the least but the capital outlay is the highest.

Quarriable limit

The cost of removing overburden to extract mineral lying below it increases as the quarrying operations extend to the dip side of the property and the thickness of overburden increases. The stripping ratio, thickness of overburden : thickness of mineral deposit, therefore, decides the economic working limit of quarrying. The softer the rocks, the less is the expense of overburden removal and higher is the ratio. The wages of labour, the selling price of mineral and the margin of profit are other major considerations in deciding the limiting ratio which is as follows :—

Limiting ratio of mineral : overburden

	Coal formations	Iron ore formations
(i) Manual quarrying	1 : 1 to 1.5	1 : 1 to 1.25
(ii) Semi-mechanised quarrying	1 : 2	1 : 1.5
(iii) Mechanised quarrying ; with dipper-shovel-dumper combination :	1 : 4 to 5	1 : 2 to 2.5
with draglines :	1 : 8 to 10	....
with bucket wheel excavators :	1 : 3 to 4	....

The ultimate depth of the quarry is also a consideration as haul elevation will increase out of all proportion and transport may not be feasible in certain cases ; at any rate it will be more costly than underground winning. The stripping ratio must therefore be related to the maximum depth of the quarry which is economically profitable. This maximum profitable depth of the quarry is the level at which the costs of opencast and underground mining are the same.

In British mines the ratio is nearly 1 : 10 in coal formations so that a seam 1.5 m thick is extracted by removing 15 m of over-

burden. In Russia it is as high as 1 : 20. In the U.S.A. the figure ranges between 1 : 10 and 1 : 15.

Iron ore in this country is extracted only by opencast mining and nowhere by underground operations. In foreign countries underground mining of iron ores is not uncommon.

ADVANTAGES OF QUARRYING

(1) There is no problem of roof control or ventilation.

(2) Full extraction of mineral is possible. No mineral is blocked as in shaft pillars, support of main roadways, etc.

(3) Quick return on capital and early extraction are possible without the need to wait for a long period of development work or unproductive work like shaft sinking, etc.

(4) Concentration of output. Large output is available from a small area. Supervision is easy owing to concentrated work.

(5) Artificial lights are necessary only after dark.

(6) Dangers and hazards are less as compared to underground mining. There is no risk of gas explosion.

(7) Better sanitary conditions can be maintained.

(8) High efficiency of workers due to ease in loading the tubs in unconfined space and natural light. High O.M.S. is possible.

(9) Female labour can be employed. All the members of a family are often employed and the accommodation problem is simplified.

(10) Once mineral is exposed, output can be easily varied to meet wagons supplies.

(11) Very few stringent mining regulations are applicable to quarries. For example, compared to underground mining, less number of competent persons are to be appointed and less number of statutory inspections are necessary.

(12) Training of operatives is easier.

(13) Large scale mechanisation is possible as there is no restriction on the dimensions of machines to be used.

DISADVANTAGES

(1) Work is affected by weather. During winter nights and summer mid-days, efficiency of workers is very low. During rainy season unless effective steps are taken to dewater the mine, mineral which is at the lower levels, cannot be worked and mining comes practically to a stand-still.

(2) Surface land is destroyed and is rendered unfit for agriculture and residential purposes.

(3) Mining lease gives only underground rights. Surface rights have to be acquired for quarrying.

(4) The method is uneconomic for working mineral beds at depth.

(5) Where quarrying aims at quick return on capital, outcrop mineral is also mined. As it is inferior in quality due to weathering and percolation of water, the overall quality available to the consumers is affected in earlier phase of mining.

In some foreign countries, the mining law requires that the quarried area should be filled up with overburden and restored to the pre-quarry state fit for agriculture. Marshy land, after extraction of underlying mineral and restoration of surface has, in a few cases, resulted in a good agricultural area. Such law requiring surface restoration to pre-quarry stage does not exist in India.

A property with extensive area on the strike and containing a thick mineral bed moderately inclined, lying at shallow depth, is ideal for quarrying. In India coal seams with inclination as steep as 1 in 3 have been worked by mechanised open cast mining methods in Karanpura field. Seams at shallow depths which are gassy, liable to spontaneous heating or with bad roof should preferably be extracted by quarrying as the mining legislation for underground working of such seams is stringent.

Iron ore mines, which are situated mostly in hills and plateaus, have been worked with the ore deposit inclined at as much as 70° from horizontal, provided the ratio of OB : mineral deposit is within economic limit.

2. FORMATION OF BENCHES

The whole thickness of overburden, and in the case of a thick coal seam or iron ore bed, the entire mineral bed is extracted

by forming benches or steps. The bench in a mine has two dimensions, the face (also called **bank** or **highwall**) and the floor. The mining regulations relating to benches are more or less the same for coal mining and for iron ore mining. In alluvial soil, morum, gravel, clay, debris or loose soil, which is likely to slide the bench sides should not be vertical but sloping at an angle of 45° or less from the horizontal. Alternatively, the slopy sides may be formed into benches such that the height of the bench does not exceed 1.5 m and the breadth is not less than the height. In the rainy season, there is a possibility of land-slides and it is a safe practice to keep the breadth of the bench more than the height.

The overburden in coal formations generally consists of sandstone and shale and in iron ore formations, it is laterite. If this overburden is hard and compact—as it generally is, excepting the top weathered surface—the height of benches in it is not prescribed by regulations which simply state that "the sides shall be adequately benched, sloped or secured so as to prevent danger from fall of sides". The relative stipulations in the case of coal and iron ore mining are :

Coal—In coal the bench height should not exceed 3 m ; width should not be less than the height ; if sides are kept sloped, their angle should not exceed 45° from the horizontal.

Iron ore—In iron ore of "float" type, bench height should not exceed 6 m ; width should not be less than the height ; if sides are kept sloped, their angle should not exceed 60° from the horizontal.

If the orebody is of hard rock, e.g. massive hematite, the bench height may be upto 7.5 m and the width not less than 6 m, with permission from Regional Inspector of Mines. Such bench height is desirable for semi-mechanised and mechanised quarries.

In mechanised quarries using tractor shovels or dipper shovels the bench heights are generally equal to the maximum height of the bucket or boom over the quarry floor ; If the bench height is more, the top layers of the fragmented rock roll down as the shovel digs, resulting in burial of shovel crawler chains or damage to the shovel or dumpers standing nearby.

If, to avoid this danger, the shovel is worked from a distance, the operation not only strains the dipper stick but also delays the time for each pass (loading of bucket) and the bucket may not be fully loaded. For full loading of the bucket it is desirable to keep the shovel as close to the face as practicable. With high benches there is also the possibility of formation of large size boulders after primary blasting. These need secondary blasting for loading by shovels.

If on the other hand, the bench height is small, the blasted rock is exhausted within a short time and the shovel has to move from place to place to feed itself. The shovel bucket may not be completely filled in one pass on a small bench. Smaller bench height means blast-holes of smaller depth, and for a given volume of rock yield by blasting, more holes to be drilled. This involves wastage of time in shifting drilling equipment frequently from hole to hole. For each bench, there has to be an access road, a set of drilling equipment, loading shovel, dumpers, and supervising staff. The general tendency, therefore, is to have high benches. Where the mine workings are approaching the mine boundary or quarriable limit, there is a tendency to have higher benches and narrower bench floors to prevent freezing of mineral in the otherwise wide benches. If the vertical cross-section of mineral at the face shows marked difference in quality, it is an advantage to divide it into smaller benches. This enables loading of nearly uniform quality from each bench.

The drill used for blast-holes may sometimes influence the bench height e.g. if the drill rod is of 6 m length, holes deeper than 6 m would necessitate withdrawal of the rod and its replacement by a longer rod. This waste of time can be avoided by having bench heights suitable for 6m deep holes. A suitable bench height would therefore be 5 m as the blast holes have to be slightly deeper than the bench height. If the drilling equipment has more than the required drilling capacity, this aspect may be ignored, and higher benches, suitable for the loading shovels should be the aim.

Considering the height of boom above ground level, a suitable bench height for a 2 m^3 shovel would be 8 m and that for a 3.5 m^3 shovel, 12 m.

The face of the bench is rarely vertical, though it is the aim in working mechanised quarries having consolidated rock and the angle of the bench is that formed by travel of the bucket during loading. This is nearly 70° from the horizontal. In some cases the jointing and bedding planes of the rock, and the extent to which it is saturated with water influence the face-slope stability and may necessitate a flatter face angle (Fig. 1).

Bench floors are generally level or with very mild slope for drainage of water. If a number of benches are worked simultaneously, the bench floor should be level as far as practicable and also wide. Otherwise a boulder released from a higher face may roll down and gathering enough momentum and without stopping on the narrow width of the bench may

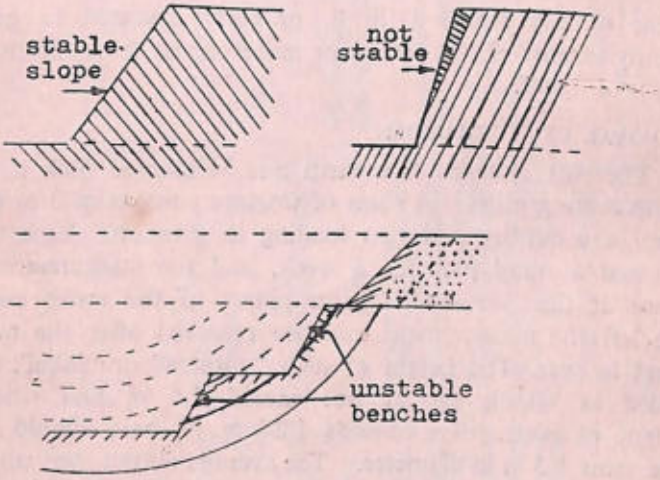

Fig. 1—Top-effect of bedding planes on the stability of a sloping bench-face.
Bottom—Unstable benches in a syncline.

land on the lower bench with dangerous consequences. Width of bench floor, less than 15 m, is uncommon where dumpers are employed. The turning circle over front tyres of a Euclid dumper (15 tonner) is 19.8 m.

3. LAYOUTS OF FACES AND WORKING IN QUARRIES

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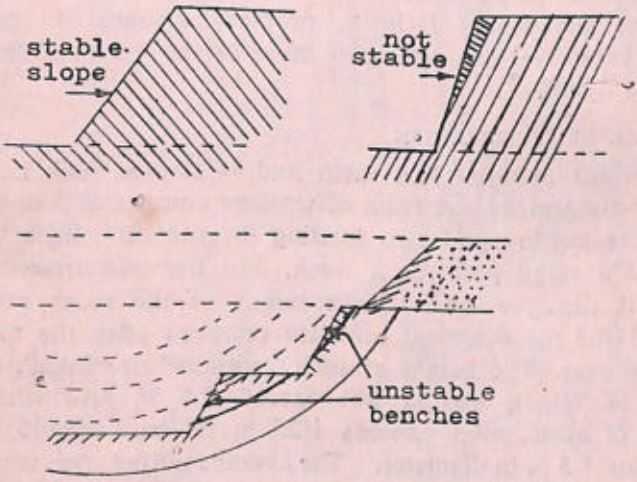

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1. Manual quarrying (coal).

Fig. 2 shows the layout where the seam thickness is 6 m, gradient of the seams 1 in 8, property assumed to be sufficiently extensive, say, 600 m or more on the strike and free from faults or dykes.

REMOVAL OF OVERBURDEN

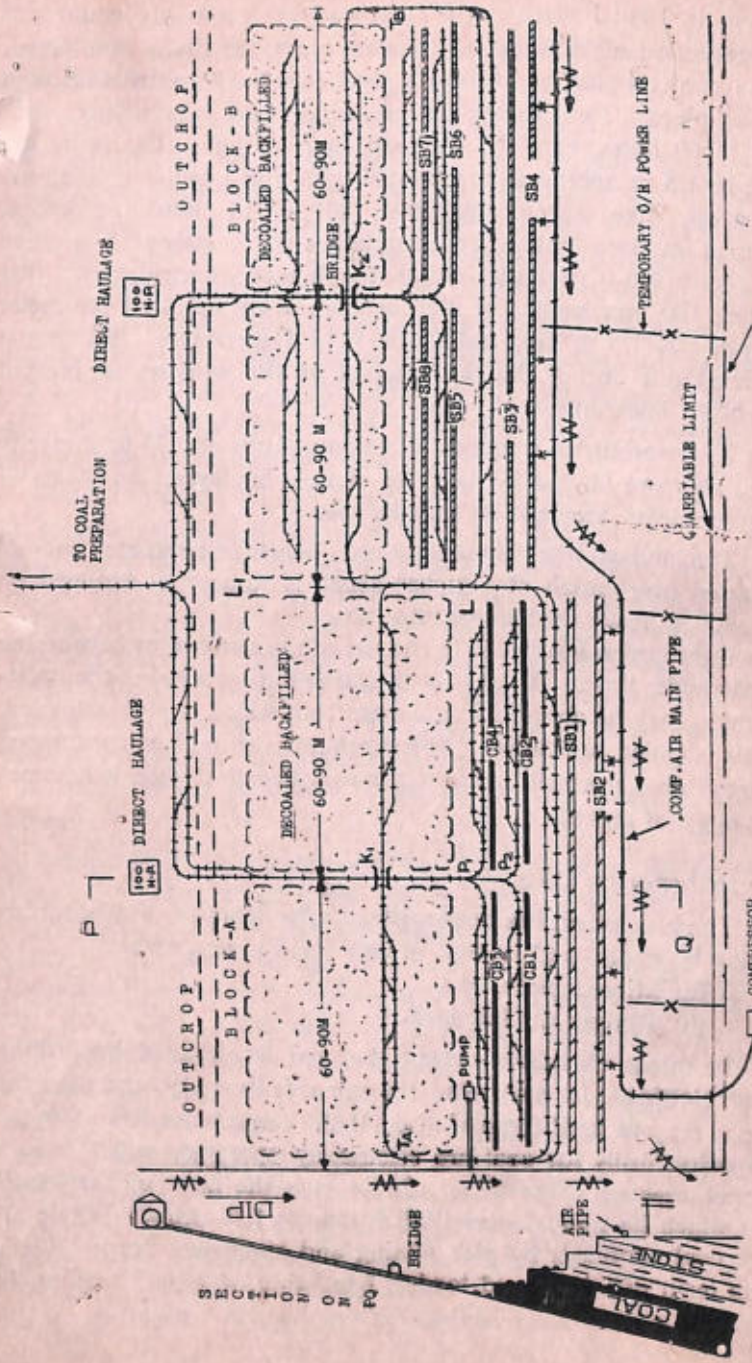
The soft material like earth and weathered rock is cut by earth cutting picks. A team of workers consists of 3 or 4 members, one cutting and two loading in rotation. Each team is allowed a small plot for a week, and the measurement takes place at the week-end. Small pillars of the rocks excavated are left for measurement and are removed after the measurement is over. The height of such "witness" or 'Sakhi', as it is called in Hindi, should not exceed 2.5 m and where the height of such pillar exceeds 1·25 m its base should not be less than 1.5 m in diameter. The average output per worker in soft rock or earth is about 2.8 m^3 per day when the lead (distance to which the debris has to be carried) is about 45 m. The work starts from the outcrop end.

In the hard rocks which need blasting holes are drilled :

(a) Manually with the help of hexagonal steel rod with chisel end, or

(b) by compressed air hammer drills (Jack hammer drills).

Method (a) is used in a small quarry where about 50 workers are employed in a shift, electricity or compressed air is not available and labour is cheap. A hole is 1 m to 1.5 m deep, 25-30 mm dia. A worker can drill 8 to 10 holes, each 1.2 m



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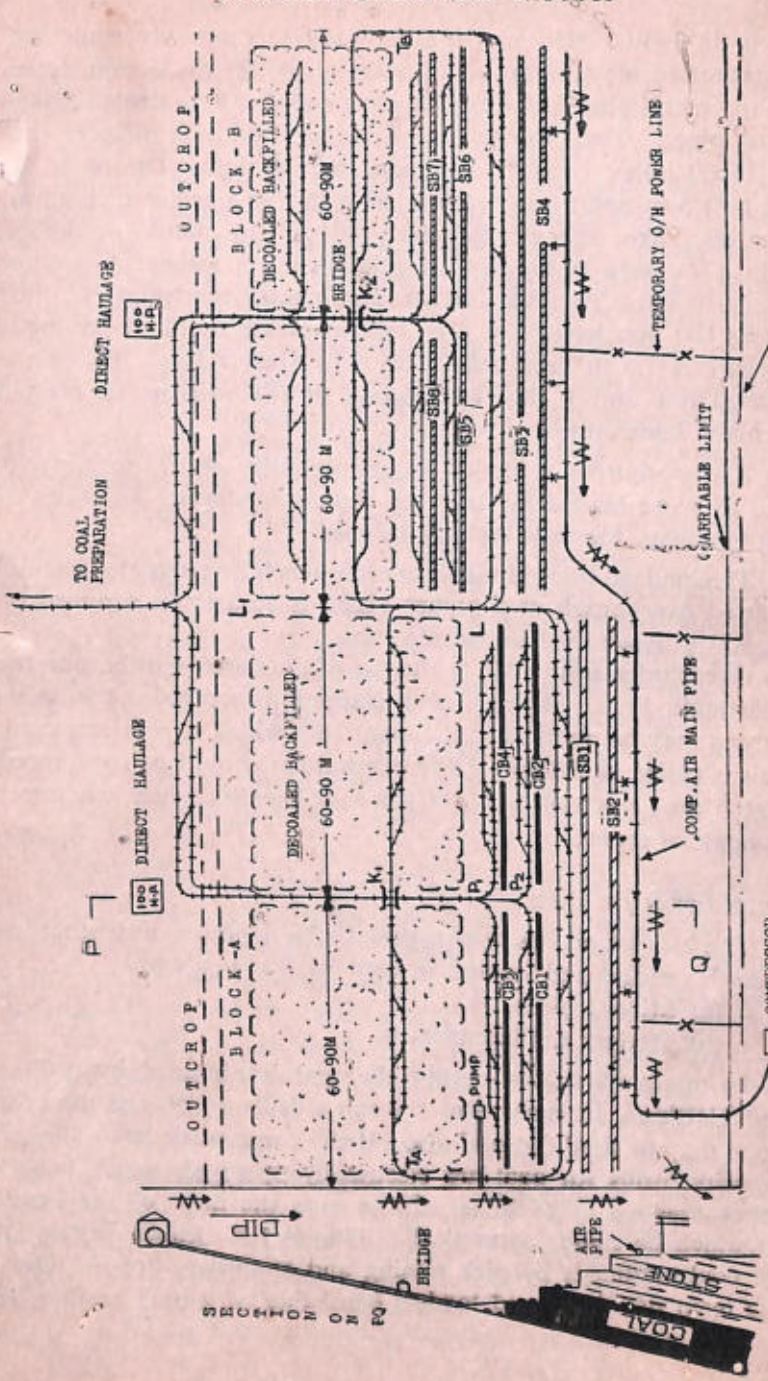

Fig. 2—Layout of a manual quarry (coal). W with arrow indicates water drains.

deep, in a shift. Method (b) is nowadays commonly employed. Compressed air is supplied by steel pipes, 50 to 75 mm diam., up to central places and branch pipes supply air to drills through hose pipes. Two workers (one driller and one helper) drill 50 to 60 holes, each 1.5 m deep, in one shift. The holes are 1.2 to 1.5 m apart and are blasted with gun powder or high explosives, like special gelatine (80%). In hard sandstone, with 1 m deep holes, special gelatine (80% strength) required is nearly 220 g per hole. As a thumb rule 3 cartridges (25 mm diam, 100 mm length) of high explosives are required per metre depth. A 1.5 m deep hole takes 4 to 5 cartridges. Blasting is carried out during the rest interval of the workers to prevent frequent interruption of work.

The overburden is loaded into tipping tubs of 0.75 m^3 capacity and they are hauled by direct or endless haulages. In seams of flat gradients, locomotives may be used.

The haulage track is taken to each bench or alternate benches. Blasted overburden of a higher bench is sometimes dropped on the lower bench for loading into tubs.

Overburden is dumped to the rise of the outcrop or beyond the quarriable limit. As the coal extraction proceeds, the overburden may be dumped in the area from which coal is extracted. This is called **backfilling**. The overburden should be so dumped that it does not roll down at the coal benches when it assumes its angle of repose.

Extraction of Coal

The coal which has been exposed after removal of overburden is cut by miners and loaders, in very small mines, with

(a) Miner's picks, or

(b) Wedges and hammers.

In mines of sizable output the coal is obtained by drilling and blasting. In such cases the same drills which are used for stone may be used for coal also. If the compressor has a limited capacity, drills for coal are electrically operated, and this is a better practice. The miner always cuts the floor of the bench on which he stands as vertical drilling is less tiring. Where all the coal cutting is by pick mining and explosives are not used one team of miner and loaders consisting of 2 or 3 workers is

allotted 1.5 m of length on the coal bench. Cutting and loading of coal is done by members of the team in rotation.

Where coal is extracted by drilling and blasting, these operations are carried out by departmental workers and the loaders are paid on piece-rate system only for loading coal into tubs. The spacing of holes in coal is 1.5 m to 2.4 m and the depth may vary from 1.2 m to 2m. The coal available per kg of high explosive is generally 12 tons, but only 5 to 7 tons per kg of gunpowder.

Where the miner has to cut the coal and load it into tubs, the average tub loading per head of the team of miners and loaders is 2 to 3 tubs (tub capacity 0.85 to 1 m^3), depending upon the lead and the availability of tubs. Where the coal is blasted, this performance is easily 1½ to 2 times.

Fig. 2 shows the layout of a large manual quarry, after it has advanced some distance along the dip. The entrance to the working places is by steps and inclined roads. The permanent installations like haulages, compressor, power transmission lines, etc, are installed in such places that their frequent shifting is not necessary when the stone benches or coal benches advance.

On a level track hand pushing of empty tubs is not uncommon for a distance of 120 m. If the same track is used for loads as well as empties, the gradient of the track should be nearly 1 in 80 in favour of loads. These factors limit the length of benches.

The property is divided into blocks, each 120 to 180 m long along the strike. Each block has a direct or endless haulage in the middle and the benches are formed along the strike on either side of the haulage. Block A, which is shown more advanced than block B, raises coal from the coal benches CB_1, CB_2, CB_3, CB_4. Each bench is 3 m high and has a haulage track on the bench floor. From the clipping point P_1 to the junction P_2, the haulage track is either level or slightly rising inbye at 1 in 80 or so. The stone benches SB_1, and SB_2, are ahead of the coal benches so that the exposed coal lasts for at least two months. Stone from the higher stone bench SB_2, is dropped on the lower stone bench SB_1 which has haulage tracks on its floor. The stone is taken in tipping tubs along a level track TA by hand pushing for dumping in the decoaled area. The track TA is along the barrier. The stone tubs may

be taken across the central haulage track over a bridge K_1 to fully utilise the decoaled area for dumping. As the stone benches advance, the position of the bridge K_1 has to be shifted to the dip side. Another level haulage track is taken from the stone bench SB_1, to the decoaled area over a ledge LL_1. This ledge is 3 m wide at the top and is a solid barrier of coal and stone left between blocks A and B. The coal of the ledge is not recoverable. In block B, there is emphasis on removal of overburden. The central haulage track is along the floor of the coal seam and more stone benches than in block A are provided for employment of a large number of workers on overburden removal. The stone benches SB_5, SB_6, SB_7, and SB_8, are served by a level track TB and another level track passing over the ledge LL_1. Stone is taken over these level tracks, in tipping tubs for dumping in the decoaled area in the same manner as described for block A. The bridge K_2 serves the same purpose as bridge K_1 in block A.

During rainy season, coal raising may be suspended in block A which is on the dip side, and only overburden removal may take place from the higher stone benches which are free from water. From block B, only coal-raising may take place during rains, and the block can be kept free from water by making a through connection 1.8 m high in the ledge LL_1, for the water to gravitate from block B to block A.

Drains for water, as shown in the figure, are cut to minimise inflow of surface water into the quarry. Pumps are installed at places where there may be heavy accumulation of water. The installation of compressed air pipes and the overhead power lines is as shown in the figure.

2. Manual quarrying (Iron ore)

Fig. 3 depicts the layout of a manual quarry situated on a hill slope, and using portable air compressors, jackhammer drills and tipping trucks upto 6 tons carrying capacity for transport of mineral/rejection. The approach road to the quarry is at the slope of the hill from where roads lead to different benches, at gradients not exceeding 1 in 5. As manual workers are employed to work at the faces and load the overburden, mineral or gangue (rejection) into tipping trucks, the bench heights are only 3 to 4 m and the bench widths are kept more than 6 m to

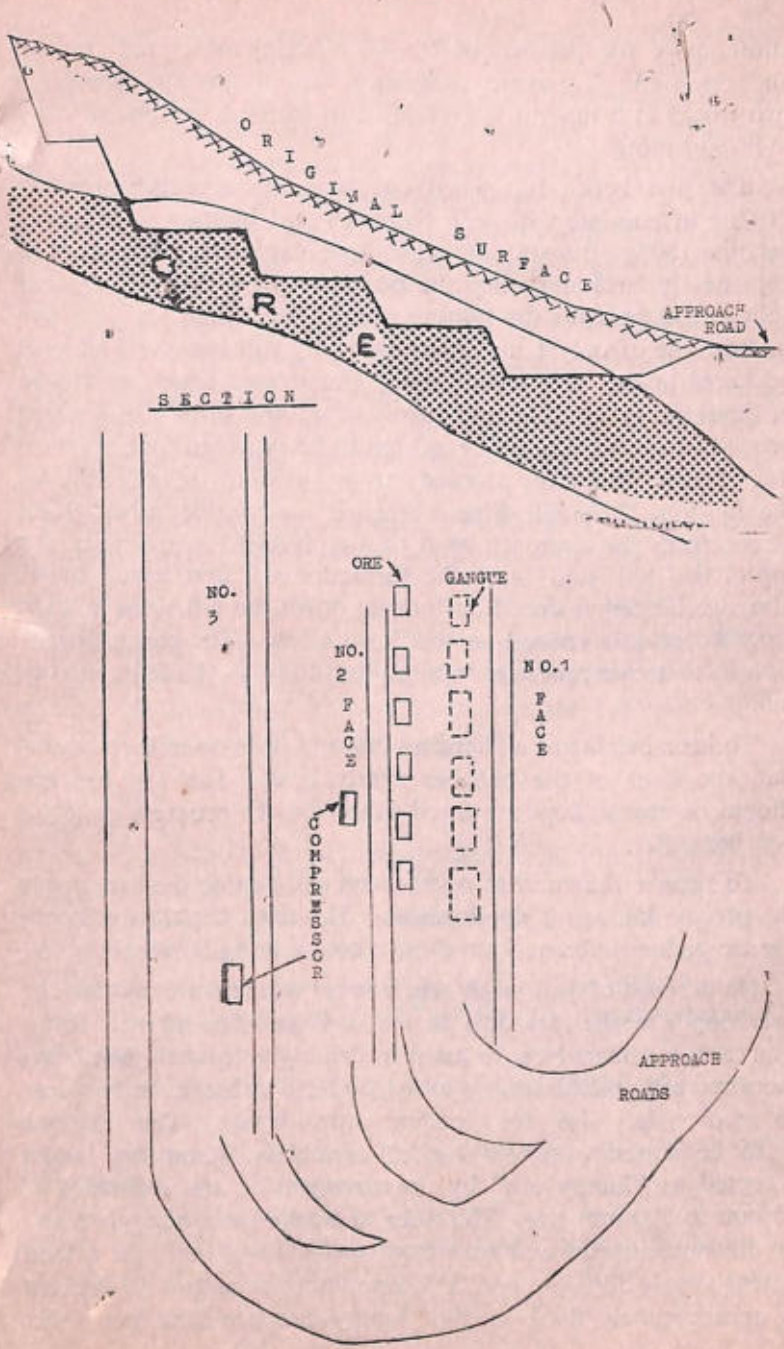

Fig. 3—Layout of an iron ore quarry (manual) on a hill slope.

allow space for passage of trucks, stacking of sorted mineral and rejection. To avoid congestion and to permit passage of two trucks at a time, it is preferable to increase the bench width to 9 m or more.

The first bench is opened out near the approach road by drilling in lateriate with jack hammers and blasting with special gelatine (80% strength) or other high explosive. The floor is kept nearly level and when the bench height is nearly 4 m, it is split into 2 benches the bottom one of full height i.e. 3 m and the top one of only 1 m which ultimately will become 3 m as it advances in the direction of rise. The second bench, in course of development, is split into two benches and gives rise to third bench or face and a branch road has to be provided for each face. The bench formation proceeds from the foot of the hill to the hill-top. If the hill slope is steep, it is advisable to branch off a road from the approach road to a convenient place near the top of the hill and start the formation of first bench there. The overburden is dozed or dumped down the hill slope and the second bench is created on the down slope. The bench formation in that case proceeds from hill-top towards the foot of the hill.

To keep heights of all benches uniform, it is essential to maintain the floor of the benches nearly level. The top benches should be always kept advanced otherwise the progress of lower benches suffer.

To remove the laterite overburden and expose the iron ore is the process known as **development.** The term has thus a meaning altogether different from that in coal mining terminology.

On a bench of 3 m height the manual workers are allotted the face length at the rate of 1 m for each member of their team. The team members have to assist in drilling holes with jack hammers and after the blasting is over they have to break the boulders to convenient size for loading into trucks. The mineral is to be sorted out and stacked separately. Iron ore lumps accepted as "lumpy ore" by the management are normally of 75 mm to 200 mm size. The rates to workers are not governed by any wage award and vary from owner to owner. At a face having nearly 50% ore and 50% rejection, the output of workers is approximately 3—4 tons of lumpy ore per head per shift.

This includes little assistance to jack hammer driller, sorting out, stacking and loading into trucks.

The compressed air for jack hammer drilling is supplied by portable compressors usually of 7 m^3 capacity. Only two jack hammers can be operated by 1 compressor of that capacity. As the compressor has to move from place to place in a shift a truck or a jeep with towing arrangement is essential on level bench floors, and a heavy duty truck or bull dozer for shifting it from one bench to another *via* steep roadways.

The jack hammer holes are generally 1.2 to 1.5 mdeep, 1 to 1.2 m apart. One driller and his helper drill about 60 holes in a shift. The charge of special gelatine (80%), 25 mm × 100 mm cartridge in 1.5 m deep hole is 5 to 6 cartridges i.e. 350 to 420 g for laterite or iron ore, depending upon hardness, and the yield per kg of explosive is 7 to 8 tons.

Where the topography permits, the laterite and rejection of lower most bench may be dozed down the slope by a bulldozer.

Example

A Manually worked quarry of iron ore working 2 shifts a day is to produce 1,000 tons per day of saleable ore. Give a layout of the quarry and state the machinery and labour strength required, assuming your own conditions.

Answer

Assumptions :

1. Ore body is in a hill practically following its slope and overlain by laterite cover of 6 to 12 m.
2. Thickness of orebody varies from 7 to 10 m and has sufficient extent, nearly 300 m, on the strike.
3. The approach road is at the foot of the hill.
4. Portable compressors of 7 m^3 capacity and jack hammer drills are available.
5. No shortage of manual loaders or tipping trucks (6 tons payload).
6. Special gelatine (80%) in 25 mm diam × 100 mm long cartridges are used for blasting.
7. Development work is partly over.
8. Blasted rock in the ore body exposed after development yields 50% to 70% as saleable ore.
9. Bench dimensions are : 3m high × 6 to 8m wide.

Production faces.

1,000 t.p.d. ore i.e. 2,000 t.p.d. blasted ore-mixed rock are to be dealt with in 2 shifts=1,000 tons rock/shift. Blasting per shift is required for 1,000 tons of ore-mixed rock and it yields 500 tons of ore/shift.

With o.m.s. of saleable ore at 4 t for face workers, $\frac{500}{4}$= 125 workers per shift are needed for face work of sorting, stacking and loading.

Face length for 125 workers=125×1 m=125 m. This may be arranged on 2 to 4 faces depending upon the length available along the strike. We may employ this strength on 3 productive faces, each about 50 m long.

For 1,000 tons of ore and mixed rock per shift, at 8 tons/kg, explosive consumption is 125 kg.

1 shot-hole 1.5 m deep, takes about 350 g of explosive.

$$\therefore \text{ holes per shift } = \frac{125 \text{ kg}}{350 \text{ g}} = \text{say, 400 holes.}$$

One jack hammer drill gives 60 holes per shift ∴ 7 to 8 jack hammers are required. For their operation 4 compressors of 7m³/min. capacity are to be employed. Each compressor needs one operator and each drill needs 1 driller and one helper.

Development work

To balance the expenditure on development work 2 faces may be kept advancing as development faces unless the development work is all over. The two faces, each 50 m long, need nearly 100 persons per shift. As no sorting of mineral is to be done on development faces, rock handled per head is nearly 5 tons in a shift i.e. 500 tons per shift consuming $\frac{500}{8}$ =63 kg of explosives (approx.).

$$\text{Shot-holes for 63 kg explosives } = \frac{63\text{kg}}{350\text{g}} = 180 \text{ (say, 200)}$$

Jack hammers required for 200 holes=4 Nos, and compressors required=2 (each of 7 m³/min. capacity).

Total Equipment required

(A) Compressors ; 4 for production+2 for development =6 in operation ; 1 as spare ; Total 7.

(B) Jack hammers ; 8 for production+4 for development =12 in use ; 3 as spare; Total 15.

(C) One heavy duty truck for shifting compressors and supplying diesel, lubricants and other materials.

(D) Transporting Trucks.

(1) Production faces.

Ore and gangue to be handled per shift =1,000 tons.

Therefore trips per shift $= \frac{1000}{6}$ =say, 170.

(2) Development faces

Rejection rock to be handled per shift =500 tons

Therefore trips per shift $= \frac{500}{6}$ = say, 85.

Total trips of trucks in a shift=170+85=255.

Number of tipping trucks to complete 255 trips per shift will depend upon the haul distance to processing plant and the rejection dumping plot, and the arrangement to load the rock. The same fleet can be used in the second shift. 10% of the number of trucks so calculated should be provided extra for repairs, maintenance and breakdowns. If one truck makes 10 trips per shift, 26 trucks are required every shift and another 3 to 4 as spare. A fleet of 30 trucks is, therefore, necessary.

(E) JEEPS—one or two for supervisory staff ;
one for repairs and maintenance crew.

(F) A diesel supplying tanker.

The above gives only the field equipment. The equipment required in the garage is extra and is apparently not required by the question.

Fig. 4. Shows another layout of a manual quarry in an iron ore mine. For simplicity only the face layout is shown as formation of benches may be carried out in the same manner as described earlier. The laterite overburden is stripped by drilling deep holes with churn drills (250 mm dia. hole), blasting with ammonium nitrate and special gelatine and loading by $2m^3$ dipper shovel into 14 tonner Euclids. The removal of overburden is therefore fully mechanised. The lumpy mineral is extracted by jack hammer drilling and blasting. The bench height in mineral is nearly 3.5 m. The jack hammer is air-leg mounted, drilling 35 mm dia. holes, 3 m deep, spaced 1 m apart. Maximum height of air leg above floor level is 2 m when fully

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(B) Jack hammers ; 8 for production+4 for development =12 in use ; 3 as spare; Total 15.

(C) One heavy duty truck for shifting compressors and supplying diesel, lubricants and other materials.

(D) Transporting Trucks.

(1) Production faces.

Ore and gangue to be handled per shift =1,000 tons.

Therefore trips per shift = $\frac{1000}{6}$ =say, 170.

(2) Development faces

Rejection rock to be handled per shift =500 tons

Therefore trips per shift = $\frac{500}{6}$ = say, 85.

Total trips of trucks in a shift=170+85=255.

Number of tipping trucks to complete 255 trips per shift will depend upon the haul distance to processing plant and the rejection dumping plot, and the arrangement to load the rock. The same fleet can be used in the second shift. 10% of the number of trucks so calculated should be provided extra for repairs, maintenance and breakdowns. If one truck makes 10 trips per shift, 26 trucks are required every shift and another 3 to 4 as spare. A fleet of 30 trucks is, therefore, necessary.

(E) JEEPS—one or two for supervisory staff ;
one for repairs and maintenance crew.

(F) A diesel supplying tanker.

The above gives only the field equipment. The equipment required in the garage is extra and is apparently not required by the question.

Fig. 4. Shows another layout of a manual quarry in an iron ore mine. For simplicity only the face layout is shown as formation of benches may be carried out in the same manner as described earlier. The laterite overburden is stripped by drilling deep holes with churn drills (250 mm dia. hole), blasting with ammonium nitrate and special gelatine and loading by 2m³ dipper shovel into 14 tonner Euclids. The removal of overburden is therefore fully mechanised. The lumpy mineral is extracted by jack hammer drilling and blasting. The bench height in mineral is nearly 3.5 m. The jack hammer is air-leg mounted, drilling 35 mm dia. holes, 3 m deep, spaced 1 m apart. Maximum height of air leg above floor level is 2 m when fully

expanded. The topmost row is inclined at 20° to horizontal and its blasting brings down the rock upto the full bench height, (Fig. 5). The mineral, blasted by special gelatine is

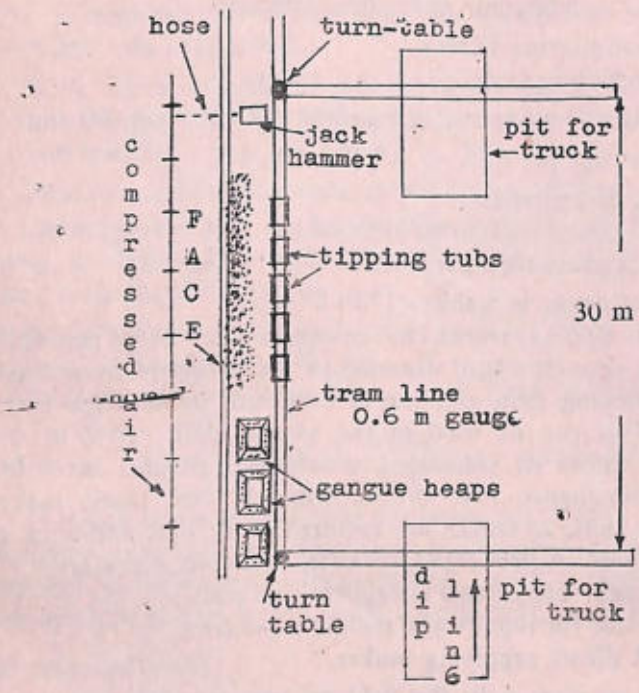

Fig. 4—Layout of an iron ore quarry (manual) with tipping tubs.

broken down to convenient size by the manual labour, sorted and loaded into tipping tubs of 0.75 m^3 capacity moving on 0.6 m gauge track. The tubs unload into tipping trucks standing below the track in a sloping pit. If any patch of the face has large proportion of gangue this is loaded by a tractor shovel into tipping trucks. If the proportion of gangue is small it is sorted out by the same manual workers and its stock pile is disposed of by the tractor shovel. The

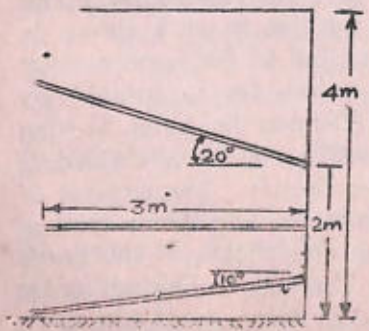

Fig. 5.—Pattern of shot-holes in an iron ore quarry (manual working).

output per worker in a shift of 8 hours is 4 tons including the time spent on breaking and loading.

3. Layout and working of a semi-mechanised mine (Iron ore).

The benches are higher than those for manual quarrying, the usual height being 6 m suitable for drilling by wagon drills and for loading by tractor shovels of 2 m^3 bucket capacity, (e.g. Michigan 85 or Caterpillar 977 Traxcavator). Width is dictated by the need for manoeuvrability of tipping trucks, tractor shovels, space for wagon drills and for stacking mineral and rejection. Width less than 15 m is inconvenient and therefore uncommon (Fig. 6).

Compressors are usually of high capacity like Rotair 600 (600 cft. or 17 m^3 capacity) which can operate 2 wagon drills of 50 mm dia drill bits at a time and one or two jackhammers.

A wagon drill normally drills holes of 50 mm to 75 mm dia and has a crew of 1 operator and 1 helper. An additional helper works as a common hand for two wagon drills for pushing them from one hole site to another. A 6.5 m deep hole is essential for a 6 m high bench for effective blasting of toe. A wagon drill using 50 mm drill bit gives about 20 holes, each 6.5 m deep, in a shift of 8 hours with a little incentive to the driller, and the burden and spacing are nearly 2 m and 2.5 m respectively.

The charge per hole is calculated as follows :

Hole dia 55 mm (50 mm drill bit)

Rock yield per hole = depth × burden × spacing
= 6 × 2 × 2.5
= 30 m^3
= 90 tons nearly (1 m^3 = nearly 3 tons)

Ammonium nitrate required $= \frac{90}{6} = 15$ kg (rock yield, 6 tons per kg)

Or, Special gelatine required $= \frac{90}{7} = 13$ kg nearly (rock yield, 7 tons per kg).

A little special gelatine or high explosive is to be used as a primer for ammonium nitrate so that 2 kg of high explosive and 12 to 13 kg of ammonium nitrate should suffice per hole.

The blasted ore is sorted by manual workers to whom the face length is allotted on the basis of 1 m length per head. The gangue is loaded by tractor shovel into tipping trucks for unloading at the rejection dump. Manual loaders, who are paid on the basis of truck trips, load the mineral into trucks.

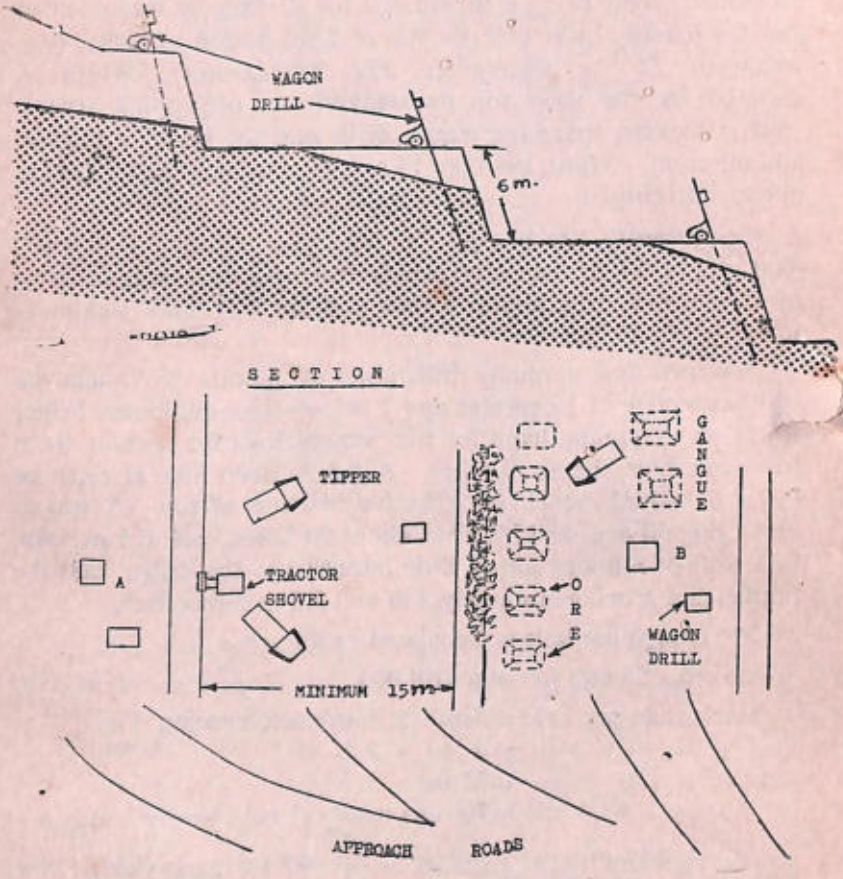

Fig. 6—Layout of a semi-mechanised quarry (iron ore).
A, B—Compressors.

4. LAYOUTS OF MECHANISED QUARRIES.

In a mechanised quarry working with heavy earthmoving machines the management has to send to the Chief Inspector of Mines byelaws governing the working of the mine for his

approval. Some of the important provisions of the byelaws in force in the quarries are as follows :

Bye-Laws for Mechanised Quarries.

Height and Width of Benches.

(a) Height of benches in overburden, ore body or other rock formation shall not exceed by more than 3 metres the maximum reach of the machine used for digging, excavation or removal.

(b) Unless otherwise permitted by the Regional Inspector of Mines by an order in writing, the width of a bench shall not be less than five times the length of the largest machine working on the bench.

Provided that, where there is one way traffic over the benches, the width of a bench may be reduced to not less than twice the length of the largest machine working on the bench.

Provided further that, as a final operation the width of a bench may be reduced to not less than 3 metres.

Fencing around opencast workings.

(1) The top edge of the opencast working shall be kept fenced with wire ropes, wire rope strands or barbed wire, supported by movable post stands (wooden, iron or concrete). The gap between the adjacent rope strands or wire shall not be more than 0.30 metres, and the bottom-most rope strand or wire shall not be more than 0.25 metre and the top-most rope, strand or wire not less than 1.0 metre from ground level.

(2) At the finishing stage, opencast working shall be fenced with a masonry wall using lime mortar, not less than 0.40 metre thick and not less than 1.2 metre high, with a parapet top.

Spoil Banks

(1) (a) The slope of a spoilbank face shall be determined by the natural angle of repose of the material being deposited but shall in no case exceed 37½ degrees from the horizontal. The spoil-bank face shall not be retained by artificial means at an angle in excess of its natural angle of repose.

(b) Any spoil-bank exceeding 30 metres in height shall be benched so that no bench exceeds 30 metres in height and the general slope does not exceed 1 vertical to 1.5 horizontal.

(2) (a) The toe of a spoil-bank face shall not be permitted to approach a railway or other public works, public road or building or other permanent structure not belonging to the owner of the mine closer than a distance equal to the vertical height of its face.

(b) A suitable fence shall be created between any railway or public works or road or building or structure and the toe of an active spoil-bank so as to prevent unauthorised persons from approaching the spoil-bank.

(3) No person shall, or shall be permitted to, approach the toe of an active spoil-bank where he may be endangered from material rolling down the face.

Roads for trucks and dumpers etc.

(1) Where practicable, all roads leading from the opencast workings to surface shall be arranged to provide one-way traffic. Where this is not practicable, no road shall be of a width less than three times the width of the largest vehicle plying on that road unless, definite turnouts and waiting points are designated.

(2) All corners and bends in roads shall be made in such a way that the operators and drivers of vehicles have a clear view for a distance of not less than 30 metres, along the road.

(3) Except with the express permission of the Regional Inspector, no road shall have a gradient steeper than 1 in 10 at any place.

(4) Where any road exists above the level of the surrounding area, it shall be provided with strong parapet walls (*matty* bunds) to prevent any out-of control vehicle from getting off the road.

Protective Equipment.

Every person working in the opencast working shall be provided with, and shall use, a hard hat and re-inforced safety boots of a type approved by the Chief Inspector of Mines.

Precautions against dust.

Adequate arrangements to allay dry dust by wetting shall be made on roads and benches where trucks and dumpers operate.

Precautions while drilling.

(1) (a) No drilling shall be commenced in an area where shots have been fired, until the blaster has made a thorough examination at all places, including remaining butts of old deep holes, for unexploded charges that the drill may strike.

(b) No drill or bore rod or pick shall be inserted in butts of old deep holes even if an examination under clause (a) has failed to reveal presence of explosives.

(2) Drilling and charging of deep holes shall not be carried out in the same area at the same time.

Duties of drill operators.

All finished drill holes are properly plugged so as to avoid possible leg injuries to any one accidentally stepping into the hole.

Precautions during short firing.

(1) (a) Shots shall not be fired except during the hours of daylight. All holes charged on any one day shall be fired on the same day.

(b) As far as practicable the shot firing shall be carried out either between shifts, or during the rest interval, or at the end of work for the day.

(2) The danger zone shall be distinctly demarcated (by means of red flags properly arranged and supported) at least 30 minutes before firing of holes is to commence.

Layouts

The laycut for coal as well as iron ore mine is nearly the same. The quality of coal/iron ore fed to the washery/beneficiation plant should be nearly uniform and this may necessitate working of mechanised quarry at 2 or 3 sites so that good quality of one site and inferior quality of the other, would give mineral of a blended quality suitable for feed to the washery/beneficiation plant.

The benches formed are sometimes designated as Bench No. 1, Bench No. 2 etc., or by the reduced level of the floor of the bench, e.g., 500 metre bench, 510 metre bench, 520 metre bench etc.

Dipper shovel—dumper combination

We shall consider layout of a mechanised coal quarry, using dipper shovels and dumpers. Let us assume that the surface is nearly level, seam is 18 m thick, dipping at 1 in 6, and the property is 900 to 1000 m along the strike and 900 to 1000 m along the dip, free from major geological disturbances. Output desired is 1 million tons per annum.

Such a long property along the strike is divided into convenient areas so that one area is worked at a time. Each area of 300 m length along the strike gives 3 areas, so that overburden removal and coal extraction in one section gives quick return on capital. Moreover, an area after coal extraction can be used for backfilling when overburden is extracted in adjacent area.

The outcrop is rarely available at the surface and usually has a cover of alluvium 6 m to 9 m thick which is removed by scrapers or dozers. The top 3 m layer of coal near the outcrop is not considered despatchable due to poor quality. It may be scraped and stocked separately for brick burning in the colliery.

The *quarriable limit* on the dip side, considering 4 : 1 as overburden to coal ratio for economic quarrying, is :

18 m (thickness) $\times 6$ (gradient 1 in 6) $\times 4$ (economic ratio) $=432$ m plan distance from the outcrop, say, 440 m.

The initial layout of the quarry is shown in Fig. 7. A boxcut (a wide trench) is excavated by a scraper along the strike near the outcrop if the soil is not hard. If hard rock is present, it may be ripped with a ripper or it may be blasted and removed by a small shovel of 2 m^3 capacity and dumpers. The shovels for overburden removal are usually large, of 3.5m^3 or 4.5 m^3 capacity because the quantity of overburden removal is heavy and the large openings of the dippers of such shovels permit big boulders to be loaded without the need of secondary blasting. Shovels for coal may be of 2 m^3 or 3.5 m^3 capacity.

We consider here 3.5 m^3 capacity shovel for overburden removal and 2 m^3 capacity shovel for coal extraction.

LAYOUT NO. 1

After the boxcut is made, the overburden is removed by shovels and coal is exposed in area A till the overburden thickness is nearly 12 m. A bench height of 12 m in overburder is the maximum height for 3.5 m^3 shovel. Beyond that height 2 benches should be formed and each equipped with a shovel.

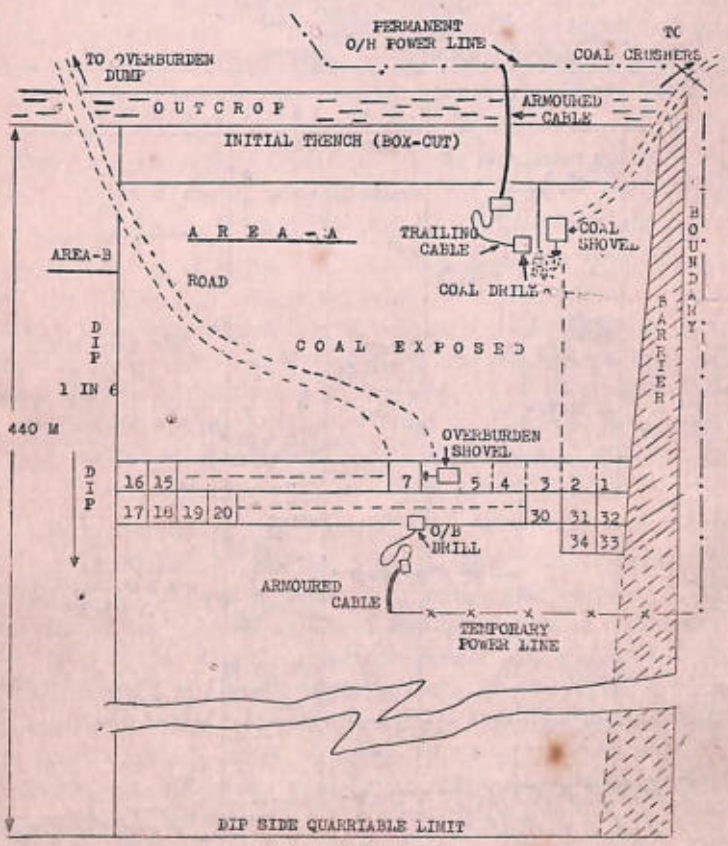

Fig. 7—Mechanised quarry in the initial stage. (Dipper shovel and dumper combination).

In the overburden, the rectangles 1, 2, 3, etc., represent blocks, one each to provide one day's extraction. The shovel advances daily in the strike direction, and the general advance considered over a quarter or a long period is towards the dip.

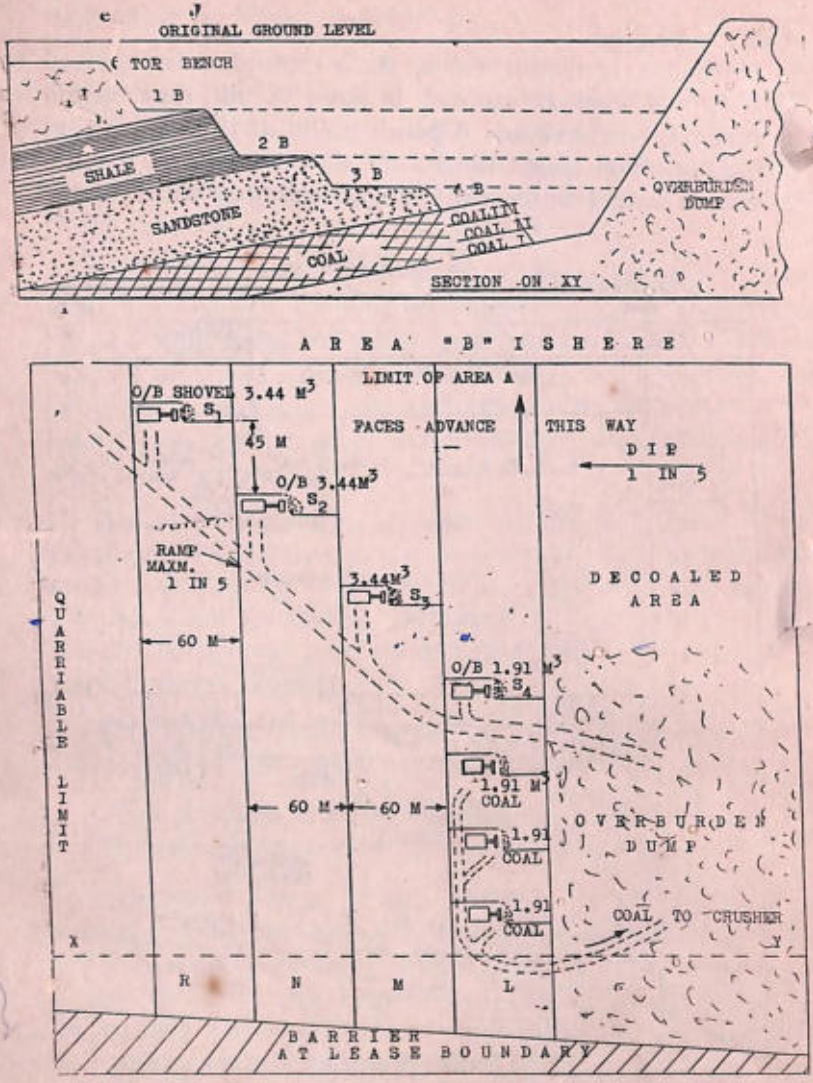

Fig. 8—Mechanised quarry at an advanced stage. (Dipper shovel-dumper combination).

The overburden shovel thus removes the overburden upto the limit of area A in 16 blocks, which takes 16 to 20 working days, making allowance for some breakdowns and stoppages for unavoidable reasons. Blocks 17, 18, 19, etc., are then extracted.

In this manner the top 12 m overburden in area A is removed upto quarriable limit and then the shovel is shifted to area B. In area A, more shovels are put into operation on the basis of 1 shovel for every 12 m thickness of overburden.

An *alternative arrangement* is shown in Fig. 8. The seam dips at 1 in 5. The area A may be divided into strips of 45 to 60 m width along dip rise. A width of 45 to 60 m permits easy movement of dumpers, the minimum width required being 18 metres. Each strip provides a bench, which is equipped with a shovel and a blast hole drill. Daily progress of shovel in the strips L.M.N.R. is along dip rise or rise dip direction, but the general advance of the bench is along strike when considered over a quarter or half year. The quarry excavation proceeds towards the dip to the quarriable limit.

As the seam is 18 m thick, 3 benches each with 6 m height, can be formed and each equipped with a 2 m^3 shovel. In the initial stage of coal extraction, a boxcut is made in the exposed coal to use a shovel. The shovels S_1, S_2, S_3 and S_4 work on stone benches and the others on coal benches. The overburden may be taken to dumping yard away from the quarry, or dumped in the decoaled area. Suitably graded roads with ramps between the strips are provided as shown in the figure.

The water accumulates on the floor of lowest coal bench and is pumped out by pumps, some of which are permanently installed and the others, portable with flexible suction and delivery to dewater local patches and feed the main pumps. It will be observed that this layout is suitable where the make of water in the quarry is not heavy and the seam is not steep. During the monsoons, the lowest bench of coal cannot be worked for 3 to 4 months unless heavy pumping capacity is available. The bottom coal of the lowest coal bench is mixed up with muck if sufficient care is not taken.

MAIN ITEMS OF MACHINERY REQUIRED FOR THE ABOVE PRODUCTION (2 PRODUCTION SHIFTS/DAY)

(*i*) *Shovels*

(a) Coal—Annual output of coal :—1 million tons =7,64,000 m^3 solid.

Number of shovels of 2 m^3 capacity for coal =3

A spare shovel is not necessary when we consider annual output as 2,53,000 m³ per shovel. This figure leaves reasonable margin for breakdowns.

(b) Overburden—Annual overburden removal in the final stage (1 : 4 ratio) = 30,56,000 m³

Number of shovels 3.5 m³ capacity for overburden in the final stage $= \frac{30,56,000}{4,00,000} = 8$

As we consider 4,00,000 m³ per shovel per annum, a figure which makes sufficient allowance for breakdowns, a spare shovel is not necessary. In the initial stage, only 4 shovels should suffice.

(ii) *Well hole drills* (150 *to* 200 *mm drill bit, rotary*)

1 drill for 3 coal shovels	=2 for coal.
1 for each O.B. shovel	=4 in the initial stage.
Total	=6 well hole drills in the initial stage.

(iii) *Tractor dozers, crawler mounted :—*

With the coal shovels	=2
With the O.B. shovels, 1 for every 2 shovels	=2 (initial stage).
At the dumping yard, 1 for every 2 shovels	=2 (initial stage).
For road making and towing materials	=2
	8
As a dozer has a heavy duty to perform, one spare dozer for every 4 dozers is necessary. Spare dozers	=2
Total dozers	=10 in the initial stages.

(*iv*) *Dumpers for rock, 35 ton capacity* :—
This depends upon the haul distance.

Considering 5 dumpers per O.B. shovel (if haul distance is 1 Km), for 4 shovels (initial stage)	=20
Spare dumpers, 10%	=2
	22
Dumpers for coal (22 tonners) considering 5 per shovel	=15
Spare dumpers, 10%	= 2
	17
(*v*) *Motor grader* for maintenance of roads	= 1
(*vi*) *Road roller* (10 ton capacity)	= 1
(*vii*) *Water sprinkler* tankers, each 9,000 litres capacity)	
for O.B. roads	= 1
for Coal roads	= 1
Total :	= 2

If water supply source is away, tankers should be of 13,000 litres capacity.

(*viii*) (*a*) *Crane 20-ton capacity*	= 1
(*b*) *Mobile crane 5 ton* capacity at 3.5 m radius	= 1
(*ix*) *Jack hammer drills.* This depends on secondary blasting that may be necessary, say 4+1 spare	= 5
(*x*) *Portable compressors* 7 m^3/min. (free air) capacity	= 3

(xi) *Portable electric welding* sets = 1

(xii) *Portable gas welding* sets = 1

(xiii) *Portable field-repair-workshop on truck* = 1

(xiv) *Scrapers* = 2 } in the initial stage only.

(xv) *Heavy duty ripper attachments* = 2 }

A well equipped workshop is necessary to run a mechanised quarry producing 1 million tons of coal per annum. That machinery is not included here.

The large number of overburden shovels required when the quarry is in full production stage can be minimised if the overburden removal is started much in advance of coal extraction, say by 6 months. It is to be noted that each machine has to be replaced when its useful life is over.

LAYOUT NO. 2 WITH A PONTOON-MOUNTED PUMP (DIPPER SHOVEL-DUMPER COMBINATION).

This layout is shown in Fig. 9 and resembles the one suggested to N.C.D.C. by Russian experts a few years ago for a mine in M.P. The coal and overburden benches are parallel to the strike throughout in Section A and are so arranged that the lowest bench is in stone and not in coal. The benches form a stepped trough. The coal is not extracted at the outcrop to start with. During the dry season the coal is exposed and the stone benches are formed with one drain trench in stone but nearly in contact with the coal roof. The trench has a minimum width of 18 m at the bottom for movement of dumpers when the former is to be excavated.

To start with, halfway between the outcrop and the quarriable limit, a box cut is made with the object of forming benches. From the box cut the excavation proceeds in the direction of the dip and also in the direction of the rise in such a manner that the bench length extends in the strike direction from one end to another end of section A. The formation of benches follows the same method as stated for Layout No. 1.

The figures 1, 2, 3, 4, etc. represent slices in coal or stone benches. Each slice has a height suitable for dipper shovel operation and length upto the limit of the Section A. The width of the slice (in the dip rise direction) may be 18 to 20 m.

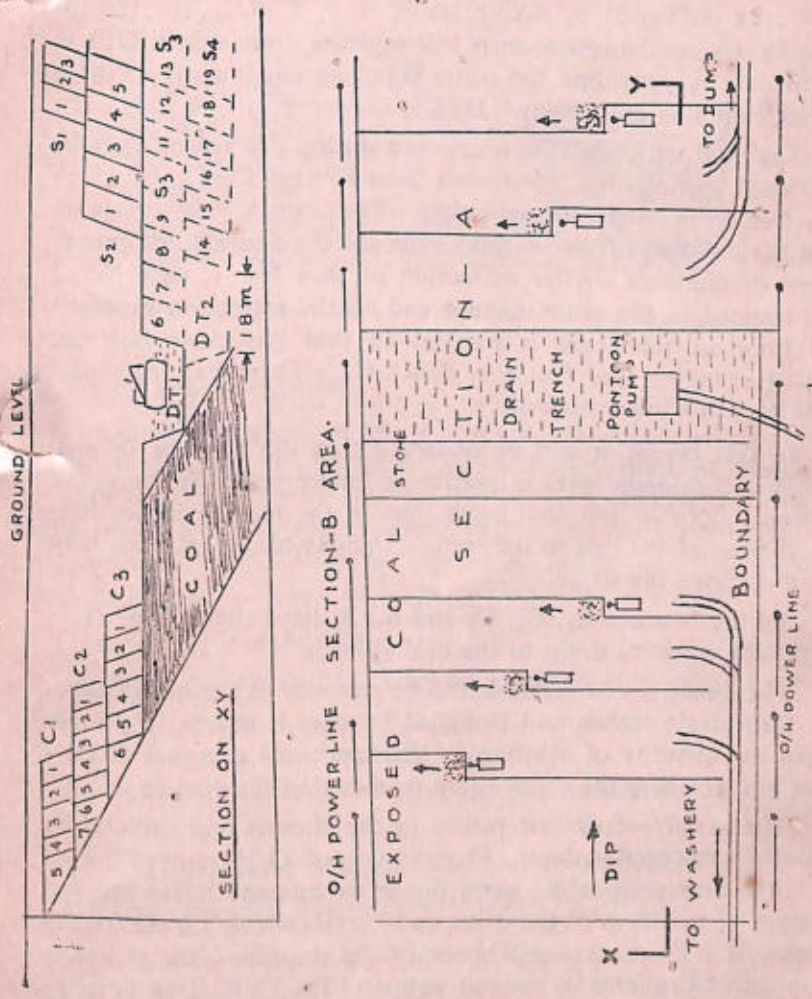

Fig. 9—Mechanised quarry layout with pontoon mounted pump (shovel dumper combination).

The day-to-day advance of the shovel is along the strike to remove a slice and the general progress of the bench extraction, considered over a quarter or longer period is in the direction of dip or rise.

The shovels work on stone benches S_1, S_2 taking slices in a definite horizon in the order, 1, 2, 3, etc. After the end of monsoons, the drain-trench DT2 has to be formed at a lower level in preparation for the next monsoon and overburden shovel of S_2 works on bench S_3 taking slices, 6, 7, 8, 9.....12, 13. During dry season, pontoon is idle and the drain-trench DT1 is excavated by providing for water standage and pumps at some small local area in the trench DT1.

The coal which has been exposed during dry season may be extracted partially by $2m^3$ shovels from benches C_1, C_2, C_3 which are free from water accumulation. Each bench is of adequate width so as to provide working area for the shovels and movement of dumpers. After extraction of slice No. 1, slice No. 2 is extracted in the same manner and so the extraction proceeds till floor of the seam is reached in that particular horizon, e.g. after taking slice No. 5 in Bench C_1. Thereafter the shovel is shifted to a lower horizon.

In this layout it will be observed that the advance of coal benches considered over a quarter or longer period is from roof towards floor of the coal seam, but in the stone benches it is from floor of the bed to the roof. The day-to-day advance is of course, along the strike.

All the benches C_1, C_2, C_3 and S_1, S_2 have slight slope (1 in 150) for water to drain to the drain-trench.

The quantity of overburden to be removed in the initial stages to form drain trench and troughed benches is nearly 50% more than the quantity of overburden that has to be removed in Lay-out No. 1, where the water sump is in coal at the coal floor.

The supply of electric power to the shovels and drills is by pliable armoured cables. From poles of O/H power line of 3.3 kV, armoured cables carry power to gate-end boxes situated within 90 to 120 m of the drills and shovels which are electrically operated. From gate-end boxes to the machines, the power is supplied by pliable armoured cables. The O/H lines have to be shifted every 3 to 4 months as the benches advance.

Flood lights for work during dark hours are mounted on 220 volt lighting poles, which run nearly parallel to the overhead power line.

Fig. 10 shows a layout of a mechanised quarry with a stripper shovel used for overburden, dipper shovel for mineral, and dumpers.

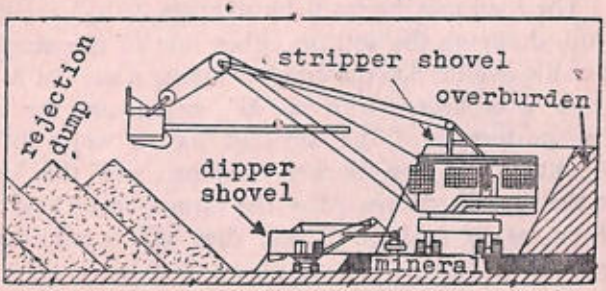

Fig. 10—Quarry working with stripper and dipper shovels.

Layout for Dragline Working

Draglines are used in some mechanised coal quarries of India for removal of overburden and exposing the coal, which is then extracted by a shovel dumper combination. Draglines are not used in iron ore quarries due to hard overburden and mineral. The following layout is nearly on the same lines, as adopted at South Balanda Colliery some years ago.

The *working conditions are* :—

(1) Seam is almost flat (1 in 20 gradient) and nearly 6 m thick.

(2) Overburden is soft, mostly alluvium.

(3) Cover is nearly 24 m and ground surface is practically level.

(4) Angle of repose of blasted overburden is about 38°.

(5) Proposed limit of quarrying is upto OB/coal ratio of 7.

The following layout is modified and can be used in any quarry with slight modifications to suit individual conditions.

The property is divided into two sections West and East, each about 1,220 m along strike, with a central rib 30 m wide at the top. This rib serves as access road to workshop and stores situated centrally for the two sections on the outcrop side. A

sub-station is also situated on the rib for supply of power to the draglines. (Fig. 12).

An access road is made for dumpers in the middle of each section. The road may be made by scrapers or by dragline itself in soft soil, dumping the soil on either side of the access road. To remove the overburden the dragline makes a key cut as shown, casting the overburden with a 90° swing on the outcrop side from position A. The key cut has a width of nearly 4.5 m, suitable for the bucket to pass, and the length is equal to the digging radius. After the formation of key cut, the dragline moves to position B and digs the overburden commencing from the key cut and casts it on the outcrop side. When

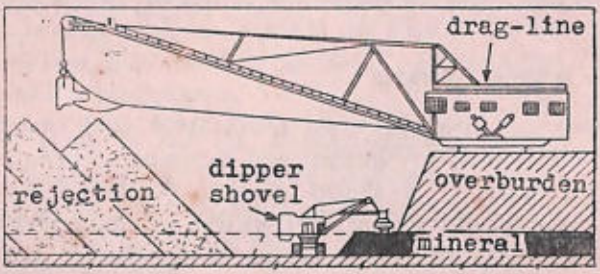

Fig. 11—Quarry working with dragline and dipper shovel.

the overburden within the reach of dragline is exhausted, it moves to position C, and forms another key cut. The process is thus repeated. A, B, C, D, E indicate the operation-positions of the dragline.

The general movement of the dragline is from the access road away towards the rib or the barrier. In West section when the dragline reaches the limit i.e., the rib, it travels to the other side of the access road to work in Section F_1 and works towards the barrier, commencing from the access road.

Overburden or debris scattered on the exposed coal is removed by a dozer before blasting. The coal is blasted after drilling holes by well hole drills. Shovels of 2 m^3 or 3 m^3 capacity may be used depending upon bench height available in coal and the coal production desired.

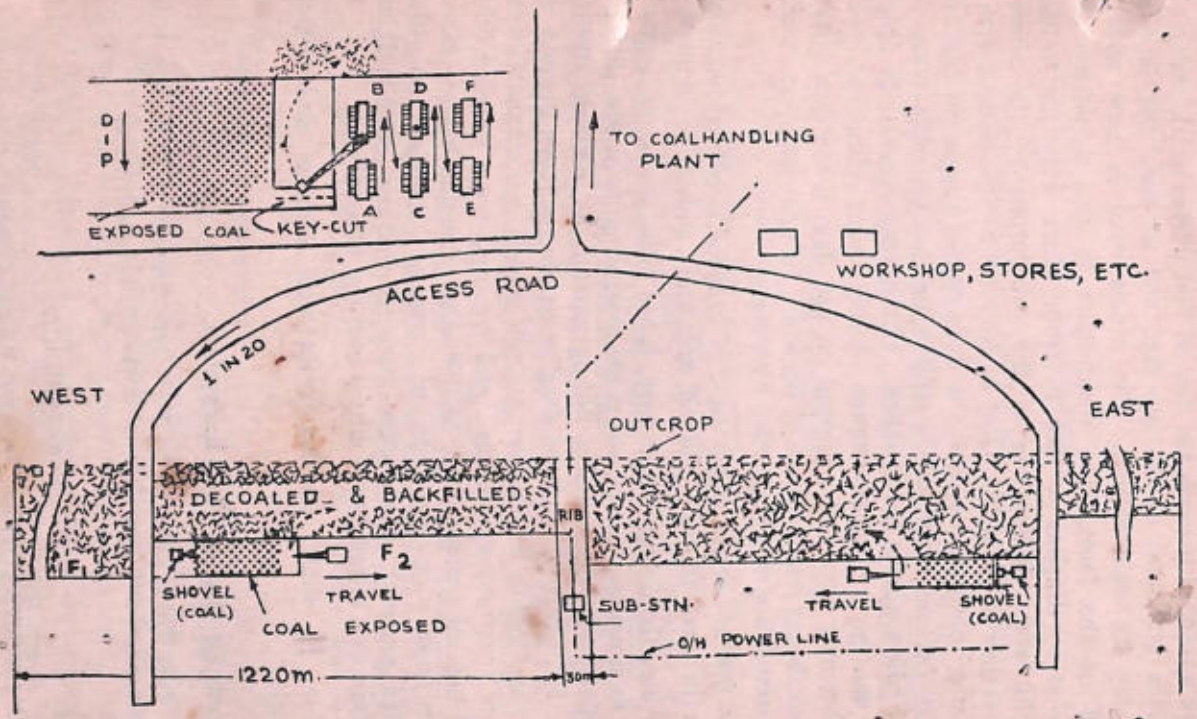

Fig. 12—Quarry layout for dragline working.
Top left—A, B.....E, F, indicate operation position of dragline

Each section has two faces, F_1 and F_2 and one of the faces is kept advanced towards the dip, so that during rains, water of rise face gravitates to the dip face and at least one face is available for coal raising. The access road is kept always ahead of the faces. The overburden in the access road is removed by dragline to face F_1 from where it is cast to the back-fill area. Rehandling of the overburden of the access road is thus unavoidable. Coal is extracted by shovels and dumpers after blasting, so that the front end of the access road is always the lowest area in the section and has the main sump of water where pumps are installed. Overburden removal by dragline can go on uninterrupted during rains also, but coal loading may have to be stopped on bad days if the haul roads become boggy due to nature of the soil. In a flat seam, the roads at the coal face may be flooded with water.

In this layout the travel route of dragline is entirely separate from the travel routes of the well hole drills, shovels or dumpers.

Where the overburden is hard, it has to be blasted for extraction by dragline. Sandy soil is the most suitable overburden for dragline. Alluvium makes flitting conditions difficult during rains for a crawler mounted dragline as the roads get boggy. A walking dragline is, however, suitable under such circumstances. Where overburden is blasted and removed by a dragline, the blast hole drill precedes it so that the dragline has to move on blasted rock and has to operate, standing on the blasted material. Such case warrants the use of a walking dragline as it is capable of flitting over the blasted rock.

Fig. 11 shows another layout with dragline and dipper shovel.

Layouts for Bucket Wheel Excavators.

These have been given in a separate chapter dealing with construction and operation of bucket wheel excavators.

REFERENCES

1. SHEVYAKOV, L. "Mining of Mineral Deposits".
2. HERBERT L. NICHOLS, Jr. "Moving the Earth".

CHAPTER 8

COMPRESSORS AND DRILLING EQUIPMENT

1. Compressed air as power. 2. Types of compressors. 3. Care and maintenance of portable compressors. 4. Drilling and types of drills for blast-holes. 5. Health Hazards from rock drilling ; Pneumonoko

1. COMPRESSED AIR AS POWER.

Rock drilling in opencast mines and also in underground mines is efficiently and conveniently carried out by drills operated by compressed air. Even in small opencast mines a few jackhammer drills operated by portable compressors is a common sight. An air compressors may be stationary or portable. The stationary compressor may be rigidly mounted on concrete foundation sufficiently away from the blasting site while a poartable compressor may be temporarily fixed in a shed beyond the unsafe blasting zone. Fixed compressors are usually of large capacity so as to provide air for a number of drills and this necessitates large dia. steel pipes for air transmission from the compressor to a central point in the quarry beyond which rubber hose pipes carry the air to the drills.

Compressed air provides power somewhat similar to steam but unlike steam it can be used cold.

Capacity of a compressor

The capacity of a compressor is always expressed in cubic feet or cubic metres of free air per minute and this refers to the actual volume of air compressed and delivered, expressed in terms of free air at the inlet temperature and pressure.

The heat generated during compression of air may raise the air temperature to 150°C to 200°C if the final gauge pressure is 7 kg/cm^2, the common pressure in air receiver for mining work. Such high temperature has a detrimental effect on the

working parts and the lubricants which become more fluid and greatly lose their lubricating properties.

The work which a prime mover of a compressor has to do for compressing the air from atmospheric to final pressure and also increasing the air temperature in the process, should be available back in the form of power in the compressed air when the latter is utilised in the air motors or other equipment if the air leaves them at atmospheric air temperature and pressure. The heat of compression is however largely wasted as the compressed air cools by radiation in the receiver and also in the transmission pipe lines if they are long. The aim of compressor designers is therefore to achieve compression which is, as far as practicable, isothermal.

The foregoing would explain the need to dissipate heat of compression from a compressor. On small compressors, used on tippers and other trucks, this is achieved by a number of radiating fins on the compressor cylinders. For large compressors the normal practice is :

(1) To achieve the final pressure by compression in two or more stages and leading the compressed air from one stage to another through intercoolers having cold water in circulation,

(2) Water-jacketing the compressor cylinders,

(3) Arranging flow of comparatively cooler atmospheric air by radiator fan over the compressor.

Cool intake air is essential for compressors. The higher the inlet air temperature the greater is the work done per kilo of air during compression and the higher is the final temperature. If a compressor is installed in a shed or room, such room has a higher temperature than the outside due to compressor operation and cool air should enter the compressor from outside the room. The intake pipe should be small in length and of large dia. to reduce friction and its mouth should be covered by a wire screen to prevent entry of coarse dust particles.

Condensation in compressed air system

Atmospheric air contains some quantity of water vapour. The amount of water vapour required to saturate a given space is the same whether that space is a vacuum or filled with air or

other gas. It should also be noted that the capacity of the space to contain water vapour depends only on the temperature and is independent of the quantity of air present or its pressure. If a given volume of air is compressed, temperature remaining unchanged, its content of water vapour will saturate the space, reduced due to compression, and the excess water vapour is then deposited as water. As the temperature of air rises during compression, the compressed air is able to retain the moisture in the form of water vapour at the high temperature inspite of the reduced volume but on cooling, the water vapour reaches a saturation point and the excess water vapour is deposited as water in the compressed air system including intercooler, after-cooler, air receiver and the transmission pipe line. The condensed water, if not removed before using the compressed air in air motors or other equipment, causes water hammer and leaky pipe joints. It also accumulates at low points in the pipe line resulting in restricted air passage, more friction to air flow and thus, loss of power. It is therefore essential to drain water from the compressor cylinders, intercoolers, air-receivers and air transmission pipes before start of work.

The prime mover for a compressor may be an electric motor, or an engine driven by steam, diesel or gas, though diesel engine is the most common arrangement on portable compressors. The prime mover may be coupled to the compressor direct, through gears or through a clutch arrangement. The compressor and the engine may be mounted on one bloc as in the case of Atlas Copco, Monobloc-250, reciprocating compressor, which has a common crankcase and a common crankshaft, housed in one cast bloc for the engine as well as the compressor.

Portable Compressors used on the mines rarely have a capacity higher than 17 m^3/min of free air. Higher capacity compressors are bulky and unwieldy for towing.

2. TYPES OF COMPRESSORS

Based on the types of construction, compressors are classified as follows :—

(1) Reciprocating compressors,

(2) Rotary compressors.

(a) Rotary vane type

(b) Rotary screw type

(c) Centrifugal or turbo compressor.

The reciprocating compressor

The reciprocating compressor is by far the most commonly used compressor for mining operations. It consists of a cylinder in which a piston is operated to and fro by a prime mover. The cylinder is provided with inlet and delivery valves. In a single acting compressor, air is sucked through the inlet valve into the cylinder in one stroke and compressed and delivered through the delivery valve during the next stroke (Compression stroke). In a double acting compressor, suction of free air and its compression take place during each stroke of the piston. The valves are spring-loaded and they open automatically by pressure difference but return to their seats due to spring tension.

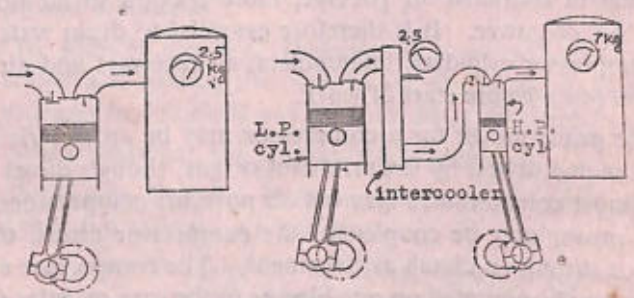

Fig. 1—Single stage and two-stage reciprocating compressor.

The cylinder is surrounded by a water jacket through which cold water is circulated in a closed circuit by a water pump. Only small compressors are air-cooled.

In a single stage compressor, the final pressure is achieved by compression in one cylinder only, and it is convenient for gauge pressure of upto 4 kg/cm^2. For higher pressures, the air is compressed progressively in two or more cylinders and it is then known as two-stage or multi-stage compressor respectively. (Fig. 1). For pressures less than 5 kg/cm^2, two stage compression is normally not practised as the cost of construction of such compressor is high compared to the advantage gained in 2-stage compression.

The compressed air is discharged into a large cylindrical vessel, called **receiver** where the pressure fluctuations even out. The receiver thus acts as a reservoir of compressed air with uniform pressure. The lubricating oil carried with the compressed air and the moisture contained in it are also deposited in the air receiver and they can be drained away when desired. A safety valve or pressure relief valve is a standard fitting on the receiver. In two or multi-stage compressors, a relief valve is fitted on each intercooler also. The safety valves are set to operate when the pressure exceeds 0.35 kg/cm^2 more than the working pressure of the system controlled by it. An air receiver is equipped with a main delivery valve and in the case of one ore more outgoing air pipes, with shut-off valves for each air pipe. A smaller air pipe goes to a pressure gauge and the automatic controls which are operated by air pressure. An air receiver of different design is incorporated in a compressor, model VT5PD(1) manufactured by Atlas Copco in India. This reciprocating compressor with a free air delivery of 7m^3/min. has a tubular air receiver incorporated in the chassis frame, providing a low centre of gravity and good road holding.

Unloading Arrangements

The demand of compressed air from the receiver is not always constant. When it is low, working of the compressor for its normal output is simply wasteful. An arrangement has, therefore, to be provided to adjust the output according to the requirements. An automatic device that varies the amount of air being pumped is called an **unloader.** With a constant speed compressor, for example, the one having an A.C. synchronous motor as prime mover, the unloader permits the driving unit to run at full speed without delivery of more air than is required. An unloader also facilitates starting a compressor, permitting a short-period operation at start without doing full amount of work required for normal compression.

Fig. 2 illustrates the basic operating principle of an unloader on air inlet line. On a spindle are mounted a valve b and a piston c which is acted upon on one side by the air pressure from the receiver and resisted by spring a from the other side.

When the pressure is low in the receiver, the spring pushes the piston c and valve b, and opens the air inlet. The free air is then drawn into the cylinder during the suction stroke and the compressor pumps up the pressure in the receiver. When the pressure in the receiver builds up to a predetermined value, the piston c overcomes the spring tension and forces valve b on its seat thereby closing the air inlet of the compressor. When this happens the compressor, though working, cannot pump air into the receiver. As the air from receiver is consumed in the air motor and its pressure drops below a predetermined working pressure the unloader valve functions again to admit air into the cylinder.

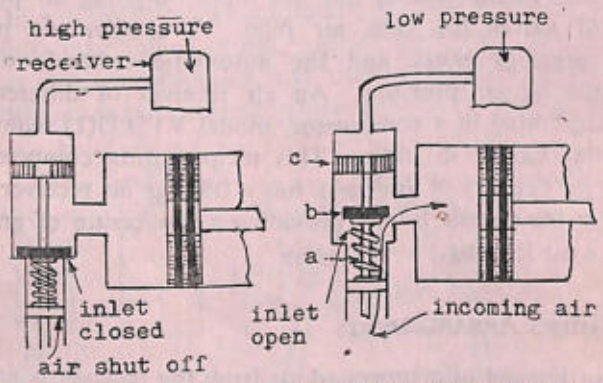

Fig. 2—Unloader valve arrangement in reciprocating compressor.

Another method of unloading, widely adopted, is to provide an automatic arrangement, operated by receiver pressure, which holds open the inlet valve when the pressure reaches 7 kg/cm^2 or a predetermined higher limit. The air which enters the cylinder during suction stroke is thrown out of the same inlet valve during compression stroke and only negligible work is done by the compressor. A governor on the prime mover reduces its speed at the same time when the unloading takes place. This continues till the receiver pressure comes down to 5.5 kg/cm^2 or a predetermined lower limit when the inlet valve again functions in its normal way and air is compressed.

During the unloading operation, the engine and compressor continue to run, though at a lower speed.

On the Holman Rotair 600 and Kirloskar-Broomwade rotary compressors the speed control and progressive off-loading system controls the operation of the compressor unit entirely by the demand for air. The output controller operated by the air receiver pressure regulates the engine speed from full speed to half and thus regulates the air demand down to half capacity. Should the air demand still fall, the inlet is progressively closed to meet with demands of less than half the machine capacity down to zero.

Rotary compressors

ROTARY VANE TYPE COMPRESSOR

In the Rotary vane type compressor a vane rotor or its equivalent is mounted eccentrically in a stationary casing. (Fig. 3).

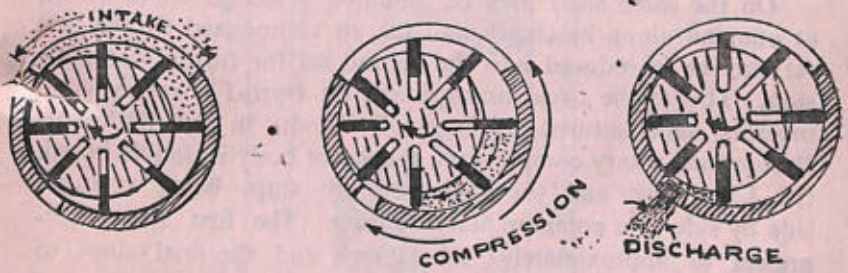

Fig. 3—Principle of working of a rotary vane type compressor.

Sliding vanes are fitted into radial lengthwise slots in the rotor and centrifugal force keeps them in contact with the casing wall whenever the rotor is turning rapidly. The vanes divide the space between the rotor and the casing into a series of compartments. The cylinders are of considerably larger diameter than the rotors which are placed off centre, so that the air pockets formed by the vanes between the cylinder and the rotor gradually increase and then decrease in size during each revolution. Air enters the compartments where their area is the largest and as the area gets smaller during rotation, the air is compressed and eventually forced out of a delivery passage just before the point of closest contact between rotor and casing. Cooled oil is injected directly into the air during compression. This atomised oil mixes with the air and performs three functions.

(1) It cools the air by absorbing the heat of compression.

(2) It forms an oil film between the vanes and the casing and prevents slippage of air from one pocket to the next across the vane.

(3) It lubricates the sliding vanes. The oil which mixes with the air during compression, passes out through the discharge into a receiver cum oil separator. The bulk of it drops out when the air stream loses velocity. The balance oil then passes on to a secondary separator, where, by means of baffles and a special filter, it is removed from the air before the latter goes into air hose.

The oil is cooled by passing it through a radiator-type oil cooler located in front of the engine radiator. It is then pumped back to bearings and compressor. (Fig. 4).

On the same shaft may be mounted a second set of vanes, at one end along its length, and the air compressed by the first set may be introduced into the second set for further compression. This is the usual arrangement on Gyro-Flo rotary compressors manufactured by Ingersoll Rand. In the Kirloskar-Broomwade rotary compressors, the stator body is double bored, the L.P. stage and the high pressure stage being arranged side by side in a common stator casing. The first stage compresses to approximately 2.5 kg/cm^2 and the final stage to 7 kg/cm^2.

The main features of such rotary vane type compressor are :

1. Lubricating oil absorbs the heat of compression. No intercooler or other air cooling device is required.

2. Flow of air is steady dispensing with air receiver and the air can be taken directly from the separator into the transmission hose pipes. Automatic controls can regulate the air output.

3. Before starting the engine the vanes are deep in the rotor slots. This provides automatic unloading at start. It is therefore not necessary to disconnect the compressor to start the engine.

4. During idling of the engine, the insufficient centrifugal force cannot keep the vanes in contact with the casing. This results in leakage around the vanes from one compartment to

another causing automatic unloading and economy in fuel consumption.

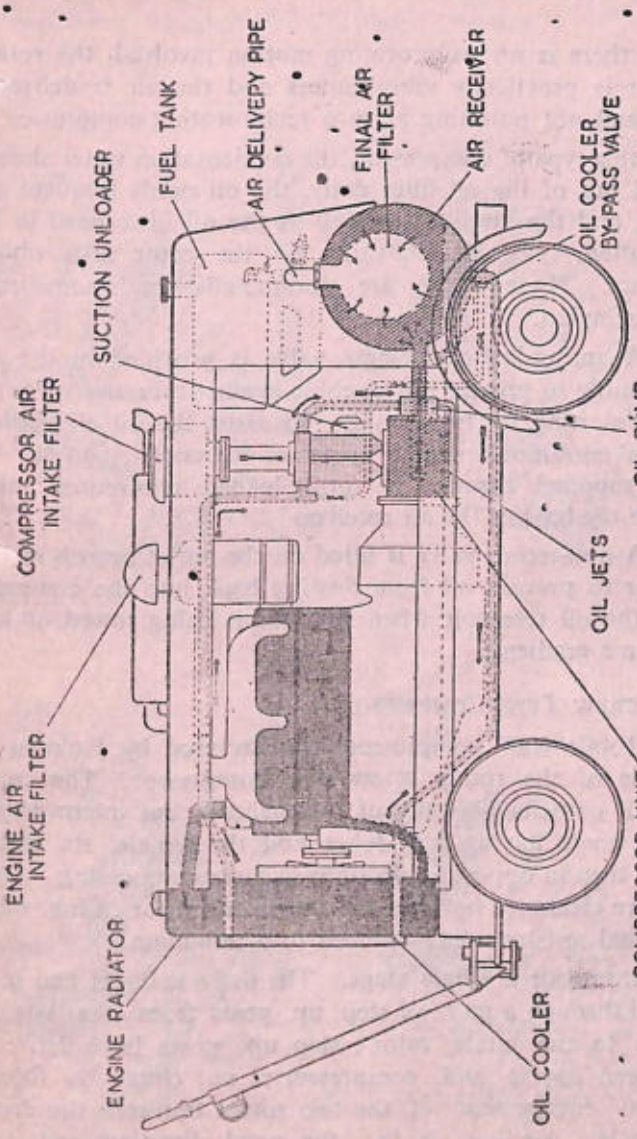

Fig. 4—Kirloskar Broomwade compressor ; rotary vane type. *Courtesy of Kirloskar Pneumatic Co. Ltd.*

5. The discharge temperature of a rotary compressor is about 40°C above ambient and therefore about 40°C cooler

than a 2-stage reciprocating compressor. This results in efficient operation and prolonged life of the drills and other pneumatic tools.

6. As there is no reciprocating motion involved, the rotary compressor is practically vibrationless and the air is delivered smoothly and not pulsating as in a reciprocating compressor.

7. In this type of compressor, the condensation water should be drained out of the oil filter daily, the oil needs frequent replacement, and the air filter as well as the oil filter have to be changed often. The set of vanes on the rotor lasts about 2,000 hours. These vanes are non-metallic and sometimes of plastic or nylon.

8. A spring-loaded by-pass valve is provided in the oil pump assembly to protect the machine against excessive oil pressure when starting up from cold. To assist the oil circulation under these conditions, a pipe by-passes the cooler, and the oil pump is supplied through a spring-loaded non-return valve direct from the base of the air receiver.

9. A non-return valve is fitted on the outlet branch of the H.P. stator to prevent oil from flowing back into the compressor from the oil reservoir when the unit is being towed or left standing on a gradient.

Rotary Screw Type Compressor

The "Rotair 600" compressor manufactured by Holman is an example of the rotary screw type compressor. The compressor unit consists basically of two helically cut intermeshing rotors, the male having four lobes and the female, six flutes. The rotors turn in opposite directions in a common casing. Due to a positive clearance between the rotors and rotor casing, wear and frictional resistance are reduced to a minimum.

The compressor is single stage. The drive is direct and it is transmitted through a pair of step up gears from the engine crankshaft to the male rotor; step up gears have 2.17 : 1 ratio between engine and compressor; no clutch is fitted. The helical engagement of the two rotors transmits the drive to the female. Air flow is in the axial direction and the compressor has a built-in compression ratio. Oil which is used as a cooling and sealing medium, also lubricates the bearings

and driving gears. The large quantity of oil injected directly to the rotors, results in a low final air delivery temperature which is in the region of 38°C above ambient.

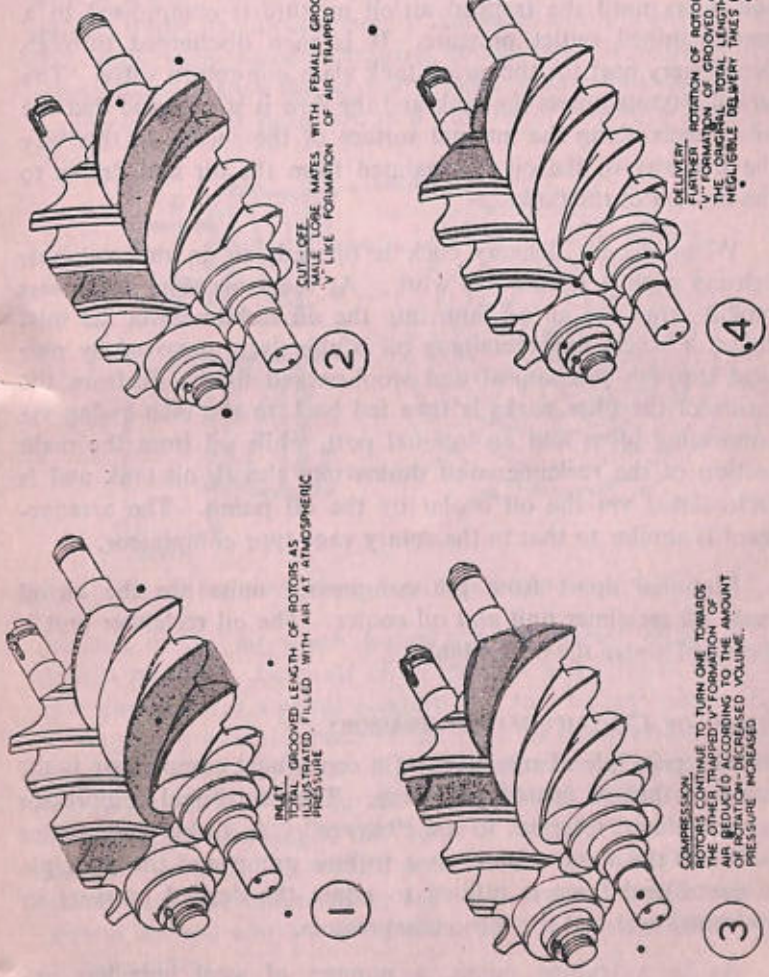

Fig. 5—Principle of air screw compression. *Courtesy of Holman Bros, Ltd.*

As the rotors turn, air is drawn in via the air cleaner and the inlet port until the complete effective length of the space between

the lobes on the male rotor and the flutes on the female rotor is filled with air at atmospheric pressure (Fig. 5). This air is carried within the male and female rotors around the casing. As the male lobe intermeshes with its corresponding female flute, air is trapped, compression commences and oil, which is being continuously injected, mixes with the air. Further rotation of the rotors reduces the effective length of the meshing lobes and flutes until the trapped air/oil mixture is compressed to a predetermined outlet pressure. It is then discharged through the delivery port to the air/oil tank via a non-return valve. The air/oil mixture enters the tank and the flow is so directed that the oil spreads along the internal surface of the shell. In this way the majority of the oil is separated from the air and drains to the bottom of the tank.

When the air delivery cock is opened, air in the reclaimer delivery pipe is released to work. At the same time, air passes rapidly from the air/oil tank into the oil reclaimer via the inlet deflector where any remaining oil is effectively removed by passage through mechanical and wool-packed filters, oil from the inside of the filter packs is then fed back to the inlet casing via connecting pipes and an internal port, while oil from the main section of the reclaimer unit drains into the air/oil tank and is recirculated via the oil cooler by the oil pump. The arrangement is similar to that in the rotary vane type compressor.

Mounted apart from the compressor units are the air/oil tank, oil reclaimer unit and oil cooler. The oil reclaimer unit is mounted above the air/oil tank.

Turbo or Centrifugal Compressors.

The principle of operation of a centrifugal compressor is the same as that of centrifugal pump. The centrifugal compressor is sometimes referred to as "blower". A turbo compressor works in the same manner as a turbine pump and the principle of centrifugal force is utilised to attain the desired pressure in the centrifugal and the turbo compressors.

As in a trubine pump, a number of steel impellers are mounted on the shaft. The impellers have backward curved blades (also called vanes). The shaft, and alongwith it the im-

pellers, rotate at a high speed within a fixed casing, known as stator. The stator is provided with diverging channels DD known as *diffusers* which open out into the return channels RR which have progressively increasing cross-section (Fig. 6). As the rotor revolves, air enters the first impeller which, by virtue of centrifugal force, throws it to its periphery with a higher velocity and pressure. The air then enters the diffuser where its kinetic energy is converted into pressure energy, and with such higher pressure energy and a little kinetic energy the air enters the second impeller where the same process is repeated. Such repetition in a number of impellers raises the

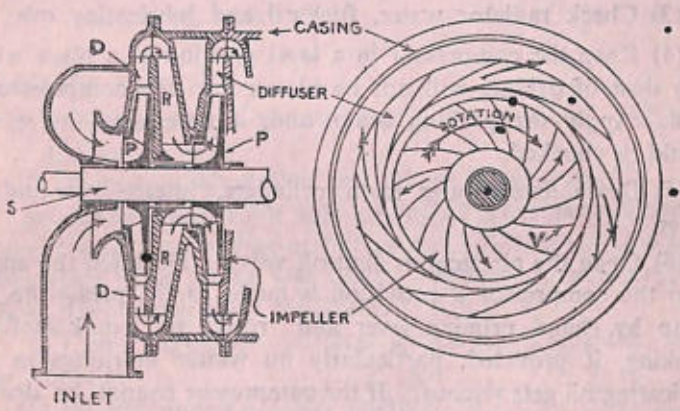

Fig. 6—Turbo or centrifugal compressor.

pressure of the air which leaves the compressor outlet at the desired pressure. Each set of an impeller and diffuser is known as a **stage** and for a gauge pressure of 5 to 8 kg/cm^2, about 10 to 15 stages are essential because the ratio of compression in each stage does not exceed 1.2. Such large number of stages necessitates a long shaft. As the air is compressed during its passage through successive stages, the radius of the impellers is gradually reduced towards the outlet end of the compressor. Water jacketing of the outer casing and cold water circulation is the general method adopted for cooling the compressor, there being no intercoolers.

Turbo compressors are not used as portable equipment in mining practice. They are suitable for large quantities of air

but small pressures ; minimum quantity is nearly 169 m^3 of free air per minute and the maximum pressure attainable is about 11 kg/cm^2. Reciprocating compressors are suitable even for small quantities and for high pressures.

Care and maintenance of portable compressors (diesel operated).

Daily (Before start of work).

(1) Visually check tyre pressure and general condition of compressor including leakages of lubricating oils, fuel, water, etc.

(2) Check battery connections and fan belt adjustments.

(3) Check radiator water, fuel oil and lubricating oils.

(4) Keep the compressor in a level position at a place where rock dust of drilling will not be blown into the compressor by wind. Apply the parking brake and, if necessary, the wheels should be chocked.

(5) Drain away water from cylinders, intercoolers and receiver.

(6) Open the air receiver shut-off valve. Declutch the engine from the compressor if provision is made for it, prime the fuel pump by hand priming lever and rotate the crank/shaft by cranking, if provided, particularly on winter mornings as the lubricating oil gets viscous. If the compressor cannot be declutched from the engine, unloading arrangement on the compressor can be utilised at the start.

(7) Start the engine. When the lubricating oil pressure builds up, engine warms up and attains good speed, operate the clutch to engage the compressor and close delivery cock on the receiver.

(8) When the compressed air pressure builds up, test manually safety valve on the intercooler and the receiver and see if all the meters are functioning well.

Check meter readings from time to time. When work stops for the day, compressed air from the receiver should be released.

The number of hours a compressor has run at its rated speed is indicated by an hour meter which is a standard fitting on most of the compressors. The hour meter on Rotair 600 is based on

engine running at 1,800 r.p.m. If the engine runs at 900 r.p.m. for 2 hours, the hour meter will record only 1 hour.

50-hour check up.

1. Clean the air cleaner on compressor as well as on the engine. Clean lubricating oil filters.

If the air cleaner is of oil bath type, change the oil ; clean the fuel filters.

2. Top up lubricating oils.
3. Check oil level in injection pump.
4. Check level of electrolyte in batteries, top up if necessary and clean battery terminals.

250-hour check up.

(1) Change the lubricating oil filter elements. Change also the lubricating oils.

(2) Examine fuel oil filter and change if necessary.

(3) Remove dust from radiator and all other parts of engine and compressor by blowing compressed air on them. This may be necessary after 50 to 100 hours' running if the compressor is working under dusty conditions.

(4) Clean the exterior surfaces with a jet of washing soda-water solution.

(5) Grease nipples of water pump.

The 50-hour check up has naturally to be carried out during the 250-hour check up also.

500-hour check up.

(1) Open out the exhaust manifold and remove carbon deposit from exhaust ports. This is required only for 2 stroke engine e.g. on Atlas Copco Monobloc-250 compressor, and not for 4-stroke engines.

(2) Renew oil in the fuel injection pump.

(3) Check up fuel injection nozzles on the engine and valves on the compressor.

(4) Change fuel oil filters.

1,500 to 2,000- hour check up.

(1) On Holman Rotair-600 compressor and similar rotary

compressors drain the air/oil tank and oil cooler and refill with clean oil.

(2) Dismantle oil sump and clean it with kerosene.

After 3,000-hours of running it is advisable to dismantle the engine and compressor, overhaul and clean all parts. The injection pump should be overhauled only by the dealer.

Transmission of compressed air.

Air is carried from receivers of portable compressors to drills or other equipment by rubber hose pipes. The hose pipes are oil resistant and specially made for compressed air which invariably carries oil particles in suspension. Hose pipe suitable for water under pressure cannot be used as a substitute.

The hose pipes are available generally in lengths of 18 m (25 mm bore) for a wagon drill and 30 m, 19 mm bore for a jackhammer. The compressed air from the receiver is hot and the combined effect of heat and oil is to soften the hose pipe and peel off the internal rubber which is carried in pieces with air to the inlet air ports of the tools resulting in their choking. The 25 mm diam. hose, generally used for wagon drills, lasts about 100 shifts and the 19 mm diam hose, used for jack hammers, lasts about 200 shifts. Two or more pieces of hose pipes can be joined by quick release couplings. The pressure drop in the hose pipes is heavier than in equivalent size of steel pipes and it is an advantage to connect a large dia. long steel pipe to the receiver and fix up a manifold at the other end for hose pipe connections. This prolongs the life of hose pipes and enables operation of the compressor from a place comparatively more distant from the dust-raising drills resulting in a beneficial effect on the filters and the cylinders of the engine and compressor.

4. DRILLING AND TYPES OF DRILLS FOR BLAST-HOLES

In opencast practice, a firm hard and stable part of the earth's crust has to be blasted after blast-holes are drilled. Great strides have been made in drilling to keep pace with advancements in blasting techniques. The main considerations in choosing a drilling equipment and pattern of blast-holes that satisfies the given situation are :

(1) daily tonnage of the material to be moved and the equipment for such removal.

(2) height of the bench.

(3) proposed hole dia, depth, burden and spacing.

(4) type of explosive to be used.

(5) breaking characteristics of the rock : tonnage obtainable per kg of explosive after the experience of the type of rock.

(6) angle of drilling.

(7) drilling characteristics of the rock.

(8) skill of drilling crew.

(9) availability of the drilling equipments and compressors.

(10) funds available.

The main systems of drilling are :

(a) Rotary

(b) Percussive.

In very general terms it can be said that in rock of hardness of less than 4.0 on Moh's scale, rotary drilling has decided advantage, except when the rock is abrasive. In harder rock, percussive drilling is preferred. This generalisation must however, be checked with the experience obtained in each individual situation by calculating the cost of drilling per ton of the material released after blasting. The improvement in rock roller bits have made possible many incursions in the hitherto known field of percussive drilling.

Rock Drills

Rock drill is the term applied to all machines using compressed air for drilling holes into rock by combined percussive and rotary action. The hole diameter is normally upto 100 mm.

To meet the various conditions rock drills are available in various types and weights and each rock drill can be provided with alternative speeds of rotation by changing the rifle bar and rifle bar nut, or alternative speeds of blows by changing the valve.

The rock drills are classified mainly as follows :—

(1) Jack hammers (also called *Sinkers*).

(2) Drifters.

(3) Stopers.

(4) Wagon drills.

A jack hammer, so familiar to mine workers, is a hand-held and unmounted drill used for vertically downward drilling. It weighs from 15 to 25 kg and is used for drilling upto a depth of 3 m ; hole dia. is generally 25 to 37 mm and rarely 50 mm. In a few cases a jack hammer may be mounted on an air leg. Though ordinarily used for dry drilling, it can be adopted for wet drilling as well.

A drifter is a mounted drill, generally designed for horizontal drilling. It is heavier than the jack hammer and is used extensively for mining and tunnel driving. The widely used mounting is the column and arm and the drill may be used for wet as well as dry drilling.

A stoper is a drill for drilling upward and derives its name from its widespread use in mine stopes. It is used normally for wet drilling.

A wagon drill is essentially a drifter type drill capable of movement up and down a vertical guide and mounted on a portable frame fitted with wheels. The hole dia. is from 50 to 100 mm and the depth drilled ranges from 3 m to 15 m.

Jackhammer

Fig. 7 shows a jackhammer in section. Compressed air enters the drill through a hose pipe (19 mm bore) and a curved metal tube with swivel connection. The speed of the drill is manually controlled by the throttle valve which regulates the quantity of air. The air passage is past the pawls on the rifle bar into an automatic valve which directs it alternately to the top and bottom of the cylinder. As the piston is forced down by the air pressure, its lower portion, called the stem, strikes the upper end of the drill steel through the medium of a chuck. In the top of the piston is a rifle nut having splines which match with those of the rifle bar or twist bar. These splines are not straight but slightly twisted.

The upward movement of the piston is also by compressed air entering the bottom of the cylinder through the automatic

valve located around the upper part of the rifle bar and as the piston moves up, the rifle nut splines exert a twisting force on both the rifle bar and the piston. The rifle bar is held by the ratchet, and thus compels the piston to turn. On the downward stroke of the piston, the rifle bar turns. The piston, therefore, experiences slight rotation during the complete cycle of

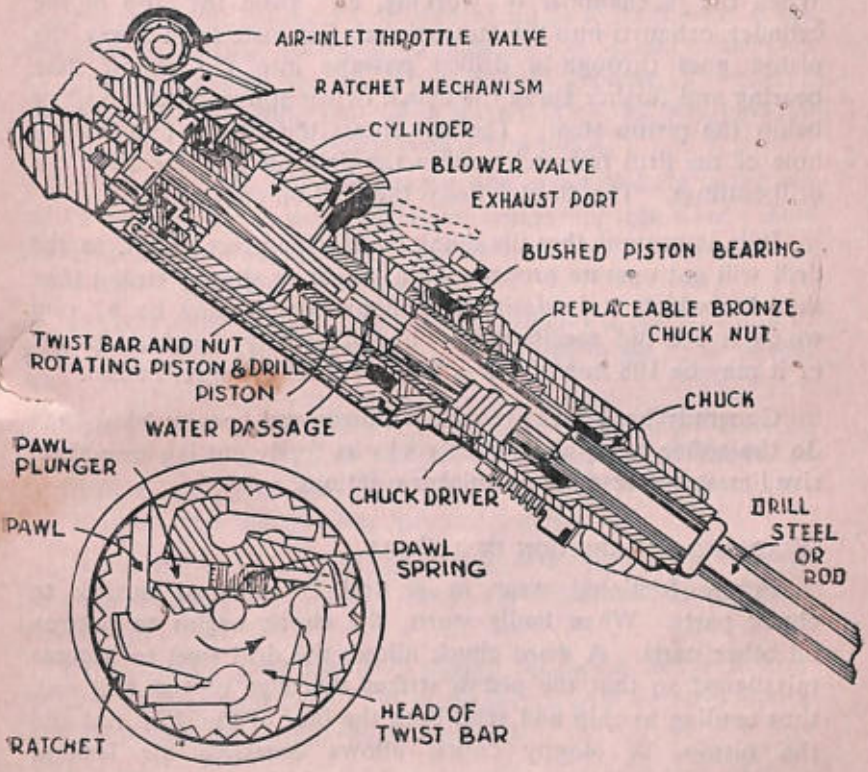

Fig. 7—Jackhammer drill.

one up and down stroke. The extent of rotation is dependent on the twist in the rifle bar and the spacing of the ratchet teeth.

The stem of the piston has straight splines corresponding with those of the chuck driver. This chuck driver is threaded into the chuck which has, at its lower end, a socket to receive the shank of the drill steel. The twist which the piston undergoes in each cycle of up and down movement is transmitted

through stem to the chuck driver ; through it to the chuck, and through the chuck socket to the drill steel. The drill steel, therefore, receives hammering during each down stroke of the piston and between successive strokes, it receives a slight twist as well so that the drill bit strikes new surface of the rock during each stroke. The number of strokes per minute are about 2,200. When the jackhammer is working, air from the top of the cylinder exhausts into the atmosphere. Exhauste from below the piston goes through a drilled passage into the piston stem bearing and further along the chuck driver splines into the space below the piston stem. There it enters the central longitudinal hole of the drill rod and reaches the drill bit to clear away the drill cuttings. The air also keeps the drill bit cool.

It is important that the shank be of the correct length, as the drill will not operate properly on a longer or shorter stroke than that for which it is designed. Shank length may be 82 mm which is the old specification, but still widely used in India ; or it may be 108 mm which is the international specification.

Generally hard rocks require a slower and heavier blow than do the softer ones, while such rocks as freely cutting non-abrasive limestones respond to quicker rotations.

Wear and Lubrication in a Drill

Purely frictional wear in a drill is confined mostly to chuck parts. When badly worn, the chuck begins to destroy all other parts. A worn chuck allows the drill steel to become misaligned so that the piston strikes the edge of the drill rod, thus tending to chip and spall both the face of the drill rod and the piston. A sloppy chuck allows excessive air leakage in the front end, with consequent loss of blowing power, which adds to rotational difficulties.

An oil bottle or line oiler is used on the air line to supply lubricating oil to the drill when it is working. The oil bottle is placed between the air receiver and the drill but it should be as close to the drill as possible, as the air and oil may separate almost completely in 3 to 5 m. Oil from oil bottle is blown into a fine spray when compressed air passes through the hose pipe and the spray is carried to the drill, alongwith the

compressed air. The oil bottle has an arrow marked on it and it should be so connected to air hose pipes that the air moves in the direction of the arrow. The oiler is automatic and actuated by the flow of air.

To know whether the drill is getting enough oil, hold a smooth surface in front of the exhaust port while the drill is running. It will collect a thin film of oil in a short time if the machine is getting adequate lubrication. A surer method is to observe the drill shanks, which should become oily after a short period, indicating that oil is travelling all the way through the machine. A full oil bottle lasts 3 to 4 hours.

It is well to add about a quarter cup of oil directly to the air intake connection of the drill before connecting the hose. This takes up the time lag between starting the drill and the moment when the oiling device takes over.

The oils suitable for lubrication are : Shell Clavus 27, Tona F, Shell A.11, Esso grade Arox 65, Caltex Capella oil B, or Mobil Almo oil No. 3.

The drill rod is hexagonal in cross-section, suitably shaped at one end to form the shank and the other end is so shaped as to form a non-detachable single chisel bit with a tungsten carbide insert. The whole thing is called a drill steel.

Drill rods may also be equipped with detachable ×-type tungsten carbide drill bits. They are available in different lengths upto 3 m.

Two workers hold the drill during operation and can drill about 60 holes, each 1.2 to 1.5 m deep in a shift of eight hours in laterite, sandstone, shale or similar rock.

Wet Drilling With Jack Hammers

To suppress the dust produced during drilling some jack hammers are equipped with arrangement to force water into the hole when the drilling continues. The drill cuttings mixed with water come out of the hole in the form of sludge. Fig. 8 shows the arrangement of water tank and pipes with a jack hammer mounted on an air leg as practised in an iron ore quarry. The water tank is of 24 litre capacity, and holes drilled are of 37 mm diam. (35 mm dia. detachable tungsten carbide tipped × bit),

3 m long. The drill rods are 18 mm hexagonal cross-section, hollow, in lengths of 2 m and 3 m. In a shift of 8 hours, nearly 75 m of wet drilling is possible if the rock

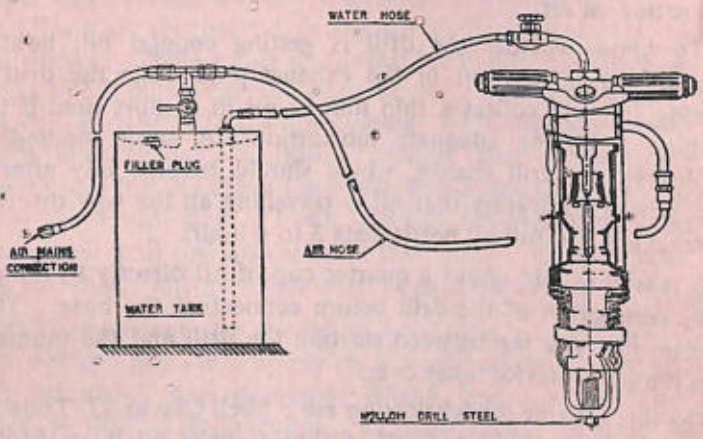

Fig. 8—Wet drilling arrangement with jackhammer.

is moderately hard. Nearly 18 litres of water suffice for 6 m drilling. If the rock is soft even 3 m drilling may consume 18 litres of water. It is necessary to have water-filled barrels

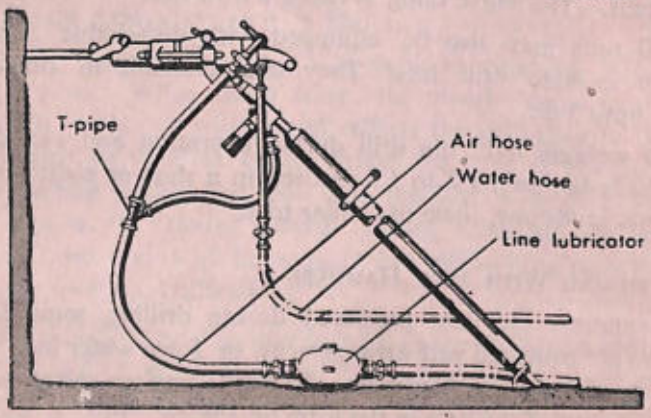

Fig. 9—Air leg.

nearby to fill up water tank from time to time. If the water for wet drilling is supplied from an external source, its pressure should not exceed the air pressure at the drill.

A jack hammer designed for wet drilling can be used for dry drilling by plugging the water inlet. A dry drill can be converted to wet drill by replacing some parts and the kit costs about Rs. 400/-.

Jack hammer drills are manufactured in India by Kirloskar Pneumetic Co., Atlas Copco and a few other companies for dry as well as wet operation. Some specifications of Jack hammer, model CB 600 manufactured by Kirloskar Pneumetic Co. Ltd., are :—

Bore	66	mm
Stroke	50	mm
Length overall with cross-bar handle ..	50	cm
Net weight	21.77	kg.
Blows per min.	2500	
Maximum diameter of bit. ..	63	mm
Free air per min. at 5.6 kg/cm^2 ..	2.12	m^3

Petrol Driven Rock Drills

A portable rock drill, not dependent upon external compressed air but operated by a built-in petrol engine, is handy in isolated places, specially for prospecting. One such rock drill in the market goes by the trade name **Pionjar rock drill** manufactured in Sweden. This is a drill for drilling holes upto 50 mm dia. with a built-in petrol engine. The jack hammer drill requires a compressor and compressed air range, both of which are dispensed with when drills with built-in petrol engine are used for secondary blasting. One man can carry the drill from place to place and operate it without any extra device. These drills have been found to operate successfully at 1400 m altitude. The important specifications are : weight-30 kg ; length 730 mm ; fuel tank capacity 1.9 litres to last for approximately 15 m of drilling ; drilling rate in standstone is nearly 0.3 m per minute, with 25 mm drill bit. The petrol engine is single cylinder, two-stroke, air-cooled and mounted on top of the drill. The engine is started with a cord arranged for automatic rewind.

Such drills have been found to give better results in limestone, where the grains are finer, than in sandstone with coarser

grains. As the engine is single cylinder the driller has to be careful in operation otherwise frequent stoppages take place necessiating starting again and again, which proves irksome with single-cylinder engines.

It is suitable only for vertically downward drilling.

Another rock drill of nearly similar construction is **Cobra** manufactured by Atlas Copco in Sweden. It has a built-in single cylinder petrol engine and a small built-in compressor. (Fig. 10). The total weight is only 25 kg. It can be used as a rock drill or as a rock breaker by manipulation of a selector handle. A floatless carburetter permits drilling in all positions from straight down to 45° above horizontal. The starting is by a totally enclosed recoil starter using ignition coil and condenser. The revolving exhaust pipe on silencer is an advantage when working in a confined space from which exhaust gas must be piped. It is operated by one man and the net drilling rate is 225 mm/min. in hard granite with a 28 mm tungsten carbide bit. It is the lightest self-contained rock drill on the market. Maximum drilling depth is 4 m ; fuel consumption is 0.17 l/metre, and the capacity of the fuel tank is 1.5 litres.

Fig. 10—"Cobra" portable drill. *Courtesy of Atlas Copco.*

Wagon Drills

Compressed air drills mounted on mobile frames, are known as wagon drills. The frame may be wheel-mounted or crawler chain mounted. They are used to drill holes of diameters varying from 50 mm to 100 mm for a depth of 3 m to 15 m. The drifter on a wagon drill works in the same manner as the

jack hammer but it is provided with a lever which can reverse the rotation, or keep it steady when the lever is in neutral position. Another lever provides for movement of the drifter up or down a mast (or boom) which is hinged to a U frame. The mast is about 3 m long so that the range of travel of the drifter is 3 m. The mast is capable of swivelling from vertical to a horizontal position and it can be kept fixed at any angle between the horizontal and the vertical thereby facilitating vertical, horizontal or inclined drilling upto 40°. In some models the mast can be raised or lowered through a vertical distance of upto 1 m by a hand operated cranking lever and rope arrangement. The U frame forms part of the wagon drill chassis which is mounted on 3 wheels. (Fig. 11). Out of the 3, two front wheels on either side of the mast, are large. The third is small, free to swivel and provided with a drawbar. The drill cuttings are removed to the surface from the hole by compressed air which is blown into it by an independent lever provided to regulate the quantity of air.

The travel of the drifter up and down the mast is by a chain, operated by air motor. The feed during drilling can be varied to suit the hardness of the rock by a lever which controls air to the air motor.

The drill rods for wagon drill are round or hexagonal, 35 mm to 37 mm dia, in lengths from 1 to 4.5 m, and they are provided with rope threads at the ends for coupling by a coupler (also called socket). As in the case of drill steels for jack hammer the drill rods have a longitudinal hole throughout for air passage. During drilling the drill rod is engaged with the drifter by keeping the rod steady and running the drifter for a little feed. It is disengaged by reverse operation. The rod rotates, while drilling, in anti-clockwise direction as viewed from the drifter towards the drill bit. Rods of standard length are coupled as the hole depth increases, the hole diameter remaining the same throughout the depth. Some drillers may use drill rods of different lengths, each fitted with a drill bit, so that the longer drill rods replace the shorter ones as the hole depth increases, as in the case of jackhammer drilling. The hole diameter, in such arrangement, gradually decreases with depth. An operator and a helper form the crew of

the drill and they can conveniently push the wagon drill, suitable for 50-75 mm drill bit, from one hole to another on a level ground.

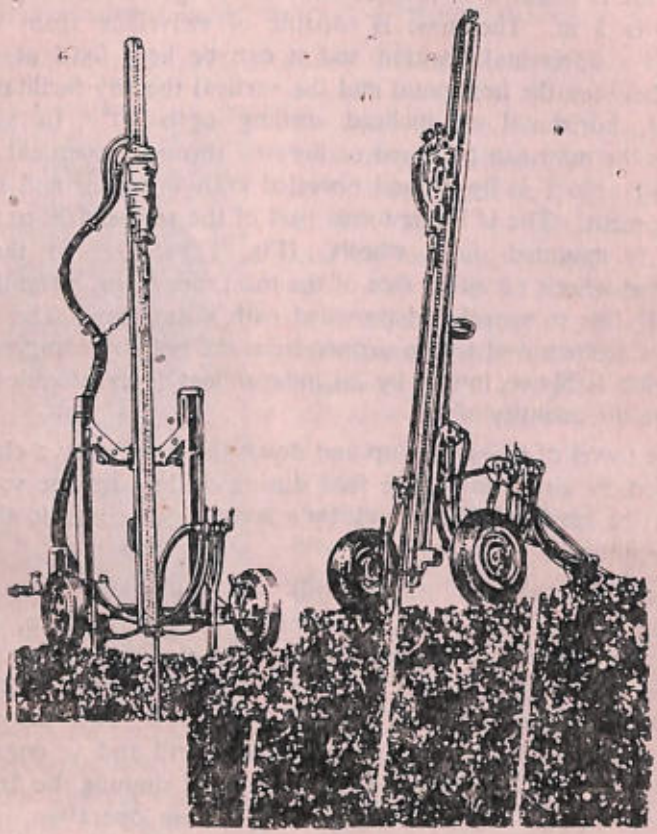

Fig. 11—Wagon drills ; left : model BVB 23 ; right : model BVB 14
Courtesy of Atlas Copco.

The total meterage drilled by a wagon drill suitable for 62-75 mm drill bit, may be about 75 m in a shift of 8 hours. This includes the time spent in shifting the wagon drill from hole to hole and the performance covers rocks like laterite, sandstone, shales and iron ore. Normal air consumption of a wagon drill (62-75 mm dia. drill bit) is 7 m^3 of free air per minute exclu-

ding flushing. Where the ground is uneven shifting of wagon drill by manual labour becomes difficult and towing by a jeep/truck involves construction of road. The hole site has also to be nearly level. For these reasons a wagon drill is suitable for drilling on nearly level ground. In hilly areas, it is convenient to use some other type of drill to form benches e.g. *a track drill*, which is a self-propelled drill mounted on crawler chains, or a *column drill*. Once the benches are formed a wagon drill can be used advantageously on the level floors.

Following are *specifications of a wagon drill model BVB* 14 manufactured by Atlas Copco :

Height	.. ..	1100 mm
Length	.. ..	2115 mm
Width	.. ..	1500 mm
Front wheel dia	..	600 mm
Steering wheel dia	..	300 mm
Height above ground to horizontal hole	..	1220 mm (maximum) 590 mm (minimum)
Weight	.. ..	280 kg.

Chainfeed is suitable for 25 mm hexagon, or 32 mm round drill steel, and is designed for feed lengths of 1.8 or 2.4 m.

A separate 4-cylinder high torque feed motor, mounted on the mast, provides for feed and quick return. The feed pressure exerted on the drill bit is 400 kg.

Drilling angle : vertical upto 40° above horizontal.

Drifter suitable with the above wagon drill has following specifications :

Drifter BBC 100A : Piston dia 100 mm ; Piston stroke 60 mm; overall length 770 mm ; Net weight 65 kg ; Maximum air consumption including air blowing at 6 kg/cm^2, 10 m^3/min ; blows/min. 2150 ; suitable for 32 mm dia. drill steel and 62 mm dia. drill bit.

Track drills.

Large size wagon drills capable of drilling 80 mm to 120 mm dia. holes are generally equipped with crawler chains and pro-

pelling air motors operated by an external compressor. Such drills usually are classified as "track drills" and they are suitable for iron ore and limestone quarries. One such track drill marketed by Ingersoll Rand is CM130 D475 capable of drilling 100 mm dia. holes. With such drill, in the iron ore quarries, the burden is 3 m, spacing of holes is 2.5 to 3m, and the drilling performance in a shift of 8 hours yields nearly 2,800 tons of ore after blasting as against nearly 1,500 tons of ore available from a wagon drill with 62 mm dia. drill bit which permits burden and spacing of about 2 m only.

Joy Manufacturing Company has recently brought in the market a track drill under the trade name "Ram". The external compressor, suitable for it is of 17 m^3/min. capacity. The crawler mounted drill can tow the compressor also so that both the units travel together from one drilling site to another. The machine can be operated by only one person. Each track has an independent control. It can be used for wet drilling as well, and a 140-litre water tank is available as an optional extra.

Some *Specifications of Joy Ram Track Drill* (model 450A-DR)

Net wt.	4,267 kg.	Bore dia of drill	115 mm
Width	2.2 m	Stroke of drill	85 mm
Length, chassis	2.8 m	Standard rifle	40 upstroke
Length, boom lowered	6.2 m	bar rotation.	
Height, boom lowered	1.5 m	Hose size I.D. :	
Height, max. horizontal hole	3.3 m	Drill	38 mm
Ground clearance	280 mm	Blow	25 mm
Motor H.P., total	23	Rev. rot	19 mm
Tramming speed	45 m/min.	Hole size range-bit dia.	64-90 mm
No Levelling jacks		Air pressure	6.4 kg/cm^2
		Air-consumption (Drill & blow)	16.4m^3/min.

MUSTANG HEAVY WEIGHT TRACK DRILL.

A recent heavy duty track drill marketed by Joy Manufacturing Company under the trade name "Mustang" has no level-

ling jacks. It is equipped with a heavy duty reverse rotation drill (Joy 600 RR) of 150 mm bore, 94 mm stroke and uses a down-hole drill (Joy DH-650) and carset bit of 150 mm diam. An external compressor of 17 m^3/min capacity, 6.5 kg/cm^2 air pressure, is suitable for operation of the drill, but a 25 m^3/min compressor is often recommended.

Specifications

Weight (chassis, mast & drill)	8,172 kg.	Drill feed, advance	39.6 m/min.
Width	3 m.	Drill feed, retract	30.5 m/min.
Length, mast raised	3.7 m	Tramming motor H.P.	28
Length, mast down	11 m	Feed Motor H.P.	20
Overall height, mast down	3.7 m	Chain Pull	4570 kg.
Width, track tread	280 mm	Min. Toe Hole Angle	12°
Track gauge	2.7 m	Drill hose size, I.D.	50 mm
Tramming speed	45 m/min.	Blow hose size, I.D.	25 mm
		Hole dia (bit size)	90 to 140 mm.

Drill bits

Drill steels used for jackhammer drills are so shaped at the drilling end that it forms a chisel with tungsten carbide insert embedded in it. The single chisel is most suited for rapid drilling of solid rock. A cross bit with T.C. insert does not form an integral part of the drill steel, but is available as a detachable bit, and is used where the rock is abrasive and hard. It is an all purpose bit. Single chisel bit is also available as detachable one (Fig. 12).

Wagon drill rods are normally provided with detachable cross bits having T.C. inserts. Such bit may have a male thread or a female thread for attachment to the drill rod. A 62 mm cross bit on a wagon drill drills nearly 4,000 metres in laterite and

hema- tite, but about only 2,000 metres in sandstone which is abrasive.

T.C. tipped bits have to be ground to their correct shape, after 2-3 shifts, on a grinding wheel. When they are worn, they cannot be reconditioned in the mine unless a costly machine

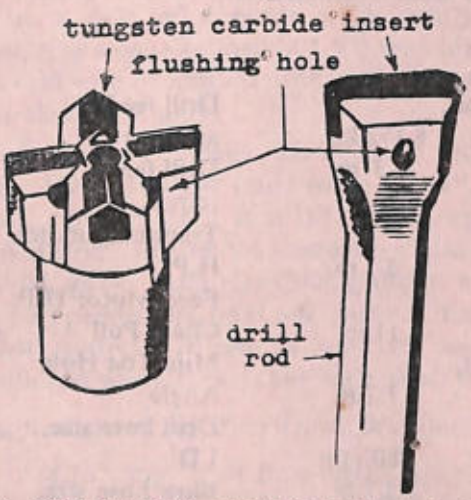

Fig. 12—Drill bits ; left—cross bit, right—single chisel bit.

is installed as at Kolar Gold field. The drill steels and the bits can be sent to manufacturers (e.g. Sandvik Asia, Poona) for re-conditioning.

Drill bits for down-the-hole percussive drills are also ×-type, but slightly different in design and have been described in the section on those drills.

Well hole drills

Drills capable of drilling holes of 100 mm to 250 mm dia are grouped in a class known as well hole drills. These are of the percussive as well as rotary type. The churn drill, now obsolete, but still used in some quarries, is an example of the percussive type. Speed of the drilling in percussive drill is 1.8 to 2.4 m per hour and of the rotary drill is 9 m to 24 m per hour, depending upon rock hardness. It can be said that for coal, sandstone, soft limestone and similar rocks a rotary drill is economical and gives good progress but its per-

TABLE 1

Rock drill trouble-chart

Trouble	Cause
Rapid wear of rifle nut and/or rifle bar.	Usually due to faulty lubrication. May also be due to grit in machine or contamination of oil.
Breaking pistons and/or rifle bars.	Usually due to heat crack ingcaused by faulty lubrication. May also be caused by bad steel shanks or excessive chuck wear.
Spalling of piston face	May be due to steel shanks having improperly shaped striking faces. May also be caused by badly worn chuck allowing steel to be struck at an angle.
Breaking side rods	May be caused by uneven tension on rods or by loose rods. Usually caused by piston and/or spacer bushing worn beyond limits allowing piston to strike front end.
Broken pawls	Invariably caused by the operator turning the drill steel in the wrong direction with a pipe wrench in an effort to free stuck wheel.
Broken or battered water tubes.	Shanks improperly punched. Badly worn chucks.
Broken steel shanks	Usually caused by worn chucks, or striking faces not square and flat. Frequent breakages in shank might also be due to uneven steel quality.
Drill refuses to start	May be plugged exhaust ports, valve stuck by gummy lubricant, or chocked air passages due to dirt or rubber from worn hose lining.
Drill refuses to rotate or weak rotation.	May be caused by bad drilling ground, mudor clay; may be due to worn chuck, chuck nut, piston, rifle nut, rfle bar.
Drill does not have standard hitting power.	May be due to short shanks, short piston (through wear or regrinding), low air pressure (at source, long runs of small dia. hose or pipe, or chocked air passages in drill). If an air line filter is used, it may be chocked.
Drill heats	New drills may heat, particularly at spacer, due to close fits and heavy work load or lack of oil. Heating in other than new drills is always caused by faulty lubrication, although hot air from an overloaded compressor is a contributing factor.

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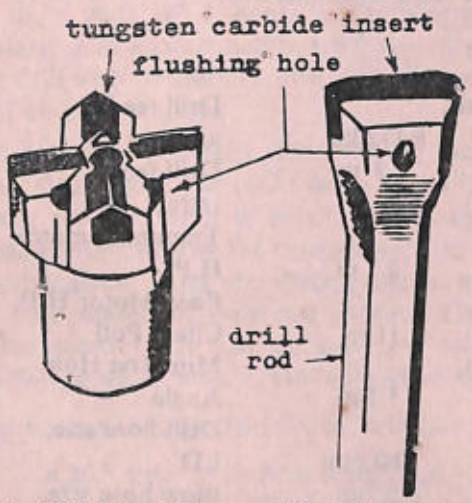

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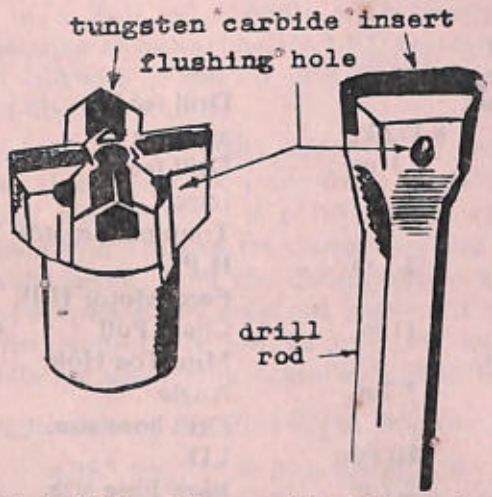

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formance is not satisfactory in laterite or iron ore deposits, which are comparatively harder.

A rotary well hole drill can drill in a shift of 8 hours nearly 20 holes, 200 mm diam. each 9 m deep, in sandstones, coal, shale and similar rocks and its operation is restricted practically to vertical drilling. It is mounted on crawler chains or pneumatic tyres, though the former is the common practice for quarries. It is operated by a diesel engine or an electric motor receiving A.C. supply at medium voltage through trailing cables from an external source. The drill carries a mast to accommodate the drill rods during drilling. The mast is collapsible and the drill should not travel over an appreciable distance with the mast raised. The drill carries hydraulically operated jacks, usually two in the front and one in the rear, to keep the machine level while drilling. When levelling, the two front jacks should be operated first and then the rear one. To lower the machine back to ground, the rear jack should be operated first. The machine should be resting on jacks and approximately level before raising the mast. Some models provide for drilling arrangements at one end of the machine, others, in the middle. A well hole drill having the drilling arrangement at one end, instead of at the middle, is suitable when drilling near the edge of a bench as required in cases where the holes are of small diameter (nearly 125 mm) permitting limited burden. With drilling arrangement in the middle, the hole has to be 3.5 to 4.5 m away from the bench edge.

The drilling tool is a tricone rotary bit. The teeth on the cone are hard faced to give resistance to abrasive wear. In general, cone bits for soft or sticky formations have long widely spaced teeth while those for drilling harder formations have shorter and more closely spaced teeth. The r.p.m. of the drilling bit is variable to suit various rock hardnesses. During drilling, the bit is kept pressed against the rock by hydraulic pressure and the feed is also hydraulic. The pressure in consolidated rocks is 35 kg/cm^2, and less in fissured strata. The life of the rock rollerbit in sandstone is nearly 800 m after which it has to be discarded as it cannot be repaired. The drill rods are in lengths varying from 3 m to 6 m.

Some of the familiar wellhole rotary drills in mechanised coal quarries are Joy 58 BH and Joy 56 BH. Joy 58 BH is suitable for holes of 160 mm to 200 mm dia.; Joy 56 BH is suitable for holes upto 170 mm diam. 48 m depth.

Both drills are available with either electric or diesel drive and use tricone rock roller bits as the drilling tools. They are suitable only for vertical drilling. Joy 58 BH can be adopted to work with down the-hole tools. These models have already become out of date in foreign countries, but as many of them are still used in the coal quarries in this country, their specifications are given below. In appearance they resemble the Drillmaster, depicted in Fig. 13.

Specifications ; Joy 58 BH Drill

Height, standard derrick lowered	5.2 m
Width, standard overall	4.9 m
Wt. Electric machine	25,000 kg.
Wt. Diesel machine	27,000 kg.
Hole Size	160 mm to 200 mm
Rated Depth	75 m
Bit speeds	24 to 109 rpm.
Transmission	4-speed.
Hydraulic feed,	stroke 1270 mm
Drill pipe, standard 100mm×100 mm square	
Levelling Jacks, number	3
(hydraulic) stroke	1270 mm
Compressor, volume	10 to 25 m^3/min.
Compressor pressure	3.5 to 8.5 kg/cm^2
Hoist rope capacity 16 mm dia-hoist rope, each drum	42 m.
Hoist rope speeds (4)	16 to 75 m/min.
Crawlers, tread width standard	558 mm.
Tramming speeds (4) forward	7.3 m to 33 m/min.
Tramming speeds (4) reverse	8 m/min.
Bit pressure, standard model	20,000 kg.
Power, diesel	60 h.p. at 1,400 rpm.
Electric, standard	50 h.p. 1,160 rpm. at medium voltage.

A 4-speed transmission from main motor or engine to drill pipe enables the operator to choose the correct speed suitable for rock hardness. On electric models the dust collectors & air compressors are powered by separate motors. The 50 H.P. motor provides power for rotation, tramming, hydraulic pumps and hoisting. Where power supply is at high voltage, stepdown

transformers are mounted as an optional fitting to bring down the voltage to 550 or 440.

On diesel powered models, the main engine is of 60 H.P. providing power for rotation, tramming, hydraulic pumps and hoisting of drill rods. The dust collector is powered from the main engine via a belt drive. There is a separate diesel engine for air compressor and its size depends on the compressor used.

Where mine conditions require drilling holes very close to the edge of the bench or highwall, the 58-BH rotary mechanism and derrick may be located at the end of the drill instead of at its normal center-drive position. Maximum down-pressure for this arrangement is 18,000 kg.

Joy 58-BH Champion drills can be adopted to drill holes of 225 mm dia. by use of a conversion kit, available as optional extra.

The recent rotary well hole drills which have replaced Joy BH 56 and BH 58 are BH 625 (for 150 mm diam. holes) and BH 735 (for 175 mm diam. holes). They are suitable for vertical as well as inclined drilling. Joy 965, a drill in the same series, drills 230 mm diam. holes.

Down-the-hole-percussive drills

Down-the-hole-percussive drill, of which the Drill-master manufactured by Ingersoll Rand (Fig. 13) is an example, can be grouped in the class of well-hole drills.

In large size wagon drills the drill is on the surface travelling up and down a mast and a considerable portion of the drill-energy is utilised in overcoming the inertia of the drill string making up the column of drill rods, and in rotating them. Such loss of drill-energy increases with depth. In the churn drill and similar percussive drills the drill has to spend a significant proportion of its energy in overcoming the inertia of the drill string. A concept has, therefore, been introduced in which the percussive drill is situated in the hole just above the drill bit, and the blow of the piston in the drill is delivered against the bit shank without any steel or rod in-between. The rotation of the drill rods is provided independantly by a rotary head

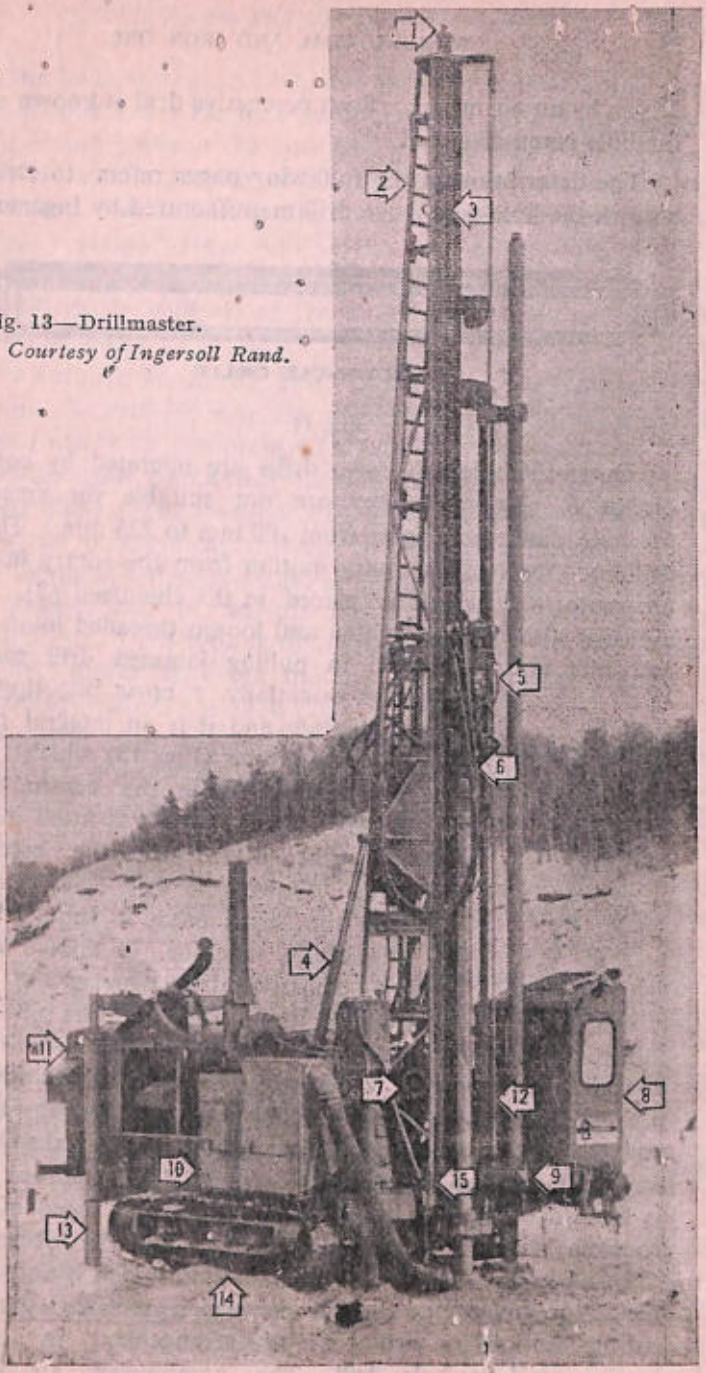

Fig. 13—Drillmaster.
Courtesy of Ingersoll Rand.

driven by an air motor. Such percussive drill is known as down-the-hole-percussive drill.

The description in the following pages refers to Drillmaster a down-the-hole-percussive drill manufactured by Ingersoll Rand.

DOWNHOLE DRILLS

Fig. 14.

Down-the-hole-percussive drills are operated by compressed air at 6.7 kg/cm^2. They are not suitable for small holes, the hole diameter ranging from 100 mm to 225 mm. The hollow drill rod receives its rotary motion from the rotary head of an air motor (*5*) which is placed in the chainfeed (*3*). The air motor is also used to tighten and loosen threaded joints on rods and bits and is helpful in pulling jammed drill rods. The drill bit is essentially a cross bit, tipped with tungsten carbide and it is an integral part of a unit called carset bit (Fig. 15) which has five air cleaning holes, one in the centre and four in the sides. The carset bit is fitted in a 1.5 m long pipe-like device and the assembly is known as "Down-hole drill". To avoid confusion in the terms used, it should be noted that the whole machine is known as down-the-hole-percussive drill whereas the assembly consisting of the drill bit and the pipe-like container is known as down-hole-drill (Fig. 14). The down-hole drill contains a piston and valve arrangement. In operation, compressed air passing down the hollow drill rods operates the piston and the piston blow is delivered against the bit shank without any drill steel or drill rod in between. The number of blows is from 500 to 2,500/min. and the rotational speed of the drill rods is nearly 15 to 25 r.p.m. It will be observed that the drill rods have only a rotary motion and it is only the carset bit at the end of the drill rods which receives the reciprocating motion by virtue of the arrangement in the device known as down-hole drill. The compressed air exhausting

CARSET BITS

Fig. 15.-

from the holes in the drill bit, and the drill exhaust going down the drill rods, carry the drill cuttings to the surface through the annular-space between the hole and the drill rods. The exhaust air should have a velocity of 900 m/min or more if the cuttings are coarse ; with deep holes, the velocity of exhaust air laden with cuttings becomes slower and this is one of the factors which limits the drilling depth. Another factor which limits the drilling depth is the capacity of the rotary head to carry the weight of the drill string.

The Drillmaster, model DM3-DHD, 325A, used in some iron ore mines to drill 165 mm dia. (drill bit size) holes has a diesel-powered rotary air compressor (*11*) of 17 m^3/min capacity. The engine and the compressor have a common intake with dry type air filters. The traction is by two crawler chains (*14*), each having an independent driving air motor. Tramming speed is 3.2 km/hr and the maximum gradient negotiable is 15° (26%). The rotary head of an air motor rotates the drill rods, and the feed to the rods is by a chain operated by a 12.5 H.P. reversible piston type air motor (*7*). The chain tension is adjusted by a crown spring at (*1*).

The drill rods are hollow, 132 mm outside diameter in lengths of 7.6 m. The assembly of downhole drill and bit is 1.8 m long so that the rod and the down-hole drill together is 9.4 m long and it is possible to drill 9 m deep hole with only one rod. For deeper holes additional rods can be coupled together. The total mast (*2*) height is 12 m above the crawler level. Four hydraulic jacks, two in the front and two in the rear, are provided to level the machine for drilling operation. A hydraulic pump driven by the diesel engine through gears supplies hydraulic powers for—

(1) dust collector blower (*10*),

(2) levelling jacks (*13*),

(3) hydraulic wrench used for coupling and uncoupling drill rods and down-hole drill (*15*), and

(4) mast raising cylinders (*4*).

The drill cuttings are collected by a dust collector which is a blower operated by a hydraulic motor. They are carried through tube to a dust separator provided with a large hopper.

The cuttings stored in the hopper can be used as a stemming material.

The Drillmaster can be used for inclined drilling also for maximum hole inclination of 30° from vertical.

The machine drills 8-9 holes, each of 9 m depth in a shift of 8 hours in medium hard iron ore, and 12 holes in laterite. In hard iron ore of Noamundi, Joda, etc. the meterage drilled in one shift of 8 hours is from 25 to 45 m. These are the results obtained in practice. Hole depth of 45 m can be reached, though holes of 9 m to 15 m, are common in iron ore mines. Where the rock formations are soft, and only rotary drilling would be suitable the down-hole drill can be substituted by conventional tricone rock roller bit for hole sizes upto 160 mm dia. For drilling in wet ground, the drill can be provided with a built-in automatic check valve that keeps water out when the air is shut off. The life of the drill bit may be as low as 200 m and may be as high as 3,000 m or more depending upon rock abrasiveness and hardness. In one instance after 5,800 m of drilling in laterite and hematitic iron ore in Goa it was observed that the drill bit was still in good condition, but it had to be changed due to wear on the splines of the carset bit. Normal life of the bit is 600 to 1,600 m in iron ore drilling in Bolani range.

Down-the-hole-percussive drills have been extensively used in the iron ore mines e.g. at Noamundi, Joda, Kiruburu, Bicholim (Goa) etc. where the rock formations are hard. Their use is not recommended in the softer rocks of coal mines where rotary well-hole drills have established themselves due to their speed and comparatively low cost.

Inclined drilling

Where the overburden consists of soft rock which can be conveniently removed by ripper and scraper combination an alternative to ripper is the method of drilling nearly horizontal blast holes and blasting them. Vertical blast holes (or *nearly*-vertical blast holes) have to be drilled where the overburden consists of hard rock like sandstone, limestone, laterite, etc.

The face of a bench is rarely vertical in a quarry. Where shovels are used, the face slope is practically the path followed

by the bucket of the shovel. This is generally 20° off vertical. A vertical hole on such face results in more burden at the foot of the face and causes inadequate blasting of the toe. Unless steps are taken to remove this toe, the bench floor gradually rises and the effective bench height decreases. The two can be removed by extra drilling of short length holes only in the toe and blasting them, or by resort to inclined drilling of the main blast holes. In vertical as well as inclined blast holes for the face, it is always essential to extend the hole slightly beyond the level of bench floor to secure proper fragmentation of the toe if the hole is terminating in hard rock.

Wagon drills normally drill at an angle of 10° to 20° off vertical. Many of the well hole drills, capable of only vertical drilling are nowadays being replaced by drills suitable for inclined drilling and manufacturers are, therefore, concentrating their attention on producing drills for inclined holes. 30° off vertical may be considered to be the limit for inclined drilling of *nearly*-vertical holes on a bench. Larger angle increases the length of the hole, difficulties in charging it with explosives of fixed-shaped cartridges, proportion of stemmed section of the hole and gives face inclination unsuitable for travel of the shovel bucket.

Structure of the rock may influence the angle of inclined drilling.

The *advantages of inclined drilling* are :

1. Toe problem is considerably reduced.
2. Easier digging for the buckets of shovels, specially at the toe.
3. Higher benches can be blasted ; this reduces the number of benches and the number of approach roads.
4. Better rock fragmentation and less secondary blasting.
5. Difficulties caused by back break in using vertical holes can be eliminated.
6. Less vibrations during blasting.
7. An inclined face is safer for men and machines than a vertical face.

It has been observed in practice that the explosive cost is also less because, (1) the rate of available explosive energy utilised

for fragmentation is much higher ; (2) the rock resistance against blasting at the toe of the bench is much smaller ; and (3) the part of the borehole which can be loaded with explosive is longer.

Jet piercing

A novel idea of drilling called "jet piercing" has been developed in the U.S.A., specially for taconite, an iron ore containing appreciable quantities of silica. The process employs a high velocity, high temperature flame of paraffin in oxygen for spalling the rock which is broken off with a water jet impinging on it. Sudden expansion of the surface layer in contact with the flame breaks it off from the cooler rock behind it. The broken particles are then blown out with the force of the steam thus formed.

The hole diameter may be 100 mm, 150 mm or 300 mm with the additional facility for enlargement by reaming action of the jet flame. Thus the bottom of the hole can be enlarged at will for the advantage of a greater concentration of the explosive charge. However, (a) this process can only be utilised in rocks which are spallable, and (b) the hole sides are irregular and may not be suitable for explosives in fixed shape cartridges. A technique of using some free flowing explosives with or without shaped-cartridge-explosives is being developed for the jet pierced holes. The speed of drilling by jet piercing varies from 4.5 m to 7.5 m per hour in formations where percussive drills would average only 150 mm, and rotaries only 1 to 1.2 m. Average piercing speed in Jasper was found to be 5 m per hour. The process is applicable to such siliceous rocks as are highly resistant to conventional drilling, such as taconites, granites and syenites. Operating costs are high chiefly due to the price of oxygen.

New Concepts in Drilling are :

Laser activated drills
Transferred and nontransferred arc drills,
Electronic beam drills,
Nuclear drills,
Electrical disintegration drills, and
Terrajet shockwave vibratory and thermal drills.

The other sources of drill power are : plasma, rocket exhaust, friction and liquid heat. Use of chemicals as rock softener is also suggested.

Prices of commonly used compressed air equipment are given in Table 2.

TABLE 2 Prices of compressed air equipment (February 1967)

1.	Air compressor, portable, diesel engine driven, 4.5 m^3 free air, 7 kg/cm^2	Rs.	32,000/-	
2.	-do- for 7m^3, free air	Rs.	42,000/-	
3.	Wagon drill BVB 14 (Atlas Copco) with mast and drifter	Rs.	22,900/-	C.I.F. *
4.	Wagon drill BVB 23 (Atlas Copco) with mast and drifter (more suitable than BVB 14 for continuous drilling)	Rs.	31,240/-	C.I.F.*
5.	Drillmaster for 165 mm diam holes ; (Ingersoll Rand)	Rs.	800,000/-	
6.	Jackhammer, dry drilling	Rs.	1,870/-	
7.	-do- wet drilling	Rs.	1,925/-	
8.	Air leg	Rs.	925/-	
9.	Air line lubricator	Rs.	210/-	
10.	Jackhammer modified for long hole drilling (upto 15 m) ; dry as well as wet drilling possible ; separate air and water passages	Rs.	2,100/-	
11.	Drill steels 800 mm × 40 mm (bit diam)	Rs.	140/-	
	1600 mm × 39 mm	Rs.	150/-	
	2400 mm × 38 mm	Rs.	170/-	
12.	Cross bits, t.c. tipped			
	$\frac{7}{8}$" (22 mm) × 38 mm rope thread	Rs.	165/-	
	$1\frac{1}{4}$" (32 mm) × 63 mm rope thread	Rs.	270/-	
	2" (50 mm) for wagon drill	Rs.	221/-	
	$2\frac{1}{2}$" (63 mm) -do-	Rs.	270/-	
13.	Carset bit 152 mm dia	Rs.	4,500/-	

* Customs duty and incidental expenses are extra, about 30% of c.i.f. value.

5. HEALTH HAZARDS FROM ROCK DRILLING. PNEUMONOCONIOSIS AND ITS PREVENTION.

Mechanisation of mines produces dusts which, when airborne, are dangerous to workers who have to work under dusty conditions over long periods. The dusts cause diseases of the lungs and *Pneumonoconiosis* or *Pneumoconiasis* is the general term used for the disease of the lungs caused by the habitual inhalation of minute particles of dust of any type.

Pneumonoconiosis is a combination of the two words from the Greek "pneumon" (lung) and "konis" which signifies dust. The term is applied to all conditions in the lungs resulting from the inhalation of dust. The most important of the pneumonoconioses are :

(1) *Silicosis*, the disease of the lungs caused by the inhalation of silica or quartz ;

(2) *Asbestosis*, which results from the inhalation of silicate of magnesium. Silicatosis is the term applied to the group of lung conditions caused by the inhalation of mineral dust in which silica is combined with various bases ;

(3) *Anthracosis*, a condition characterized by the deposit of black carbon pigment in the lung and usually seen in coal workers ;

(4) *Siderosis*, the lesion occuring in workers with iron, particularly oxides which colour the lungs varying shades of red or brown ; and

(5) *Berylliosis*, the disease caused by the inhalation of beryllium either in the form of dust or fumes.

Pneumonoconiosis may exist for considerable periods of time without producing symptoms or imparing physical efficiency. This is especially true of siderosis, chalicosis and anthracosis. It is known that the dangerous particles are those under 5 microns in size which are too small to be seen by the naked eye. These can evade the natural barriers in the nose and bronchial tubes which arrest and eliminate the larger particles.

Classification.

The classification of the pneumonoconioses may be made according to the physical nature of the dust or by the type of tissue response to the particular agent. Others consider the clinical and roentgen features of the particular dust disease. The inorganic dusts producing the disease are classified as :

A. *Fibrosis producing.*—Silicosis and asbestosis are the most important in this group. These dusts are slightly soluble.

B. *Non-Fibrosis producing.*—Anthracosis is the chief example in this group. These dusts are inert and become encapsulated in the tissues or absorbed but do not produce fibrous tissue.

C. *Toxic and/or irritant.*—This group comprises lime, dichromate compounds, lead, mercury and other heavy metals. These are corrosive and cause severe local reactions. If absorbed into the body in sufficient amount, generalised toxic effects appear. The dust and also the vapour of these substances are harmful.

SILICOSIS.

Silicosis is caused by the inhalation of free silica or quartz particles into the lungs. It is the most frequent and disabling of all the dust diseases and is characterized by the production of extensive fibrosis in the lungs. Silicosis is common in the metal grinding and polishing industry, in iron and steel foundries, and also in occupations involving mining, and tunnelling, where rock has to be drilled through or crushed.

Normally the protective mechanisms of the nasopharynx and bronchi are sufficient to keep the lower air passages free from dust. It has been found that particles of silica measuring 0.5 to 2.5 microns in diameter are most apt to produce damage. Particles larger than 10 microns in diameter seldom gain admission to the alveoli. In asbestosis, particles as large as 200 to 400 microns in diameter may be found in the bronchi. The important factors in the production of silicosis are :

(a) the size of the particles of dust,
(b) their concentration in the atmosphere,
(c) the percentage of free silica, and
(d) the length of exposure.

The presence of lung infection or other underlying pulmonary disease may be an important contributory factor. Mouth breathers are more apt to acquire the disease than those with a normal condition of the nose and pharynx.

Alkalies increase the effect of the dust while coal dust, aluminium and other minerals apparently retard the development of the disease. The principle of aluminium therapy depends on the fact that particles of metallic aluminium or amorphous hydrated alumina when engulfed in the same phagocyte cell along with silica dust, neutralise the effect of silica and thus arrest further progression of fibrotic tissue reactions. Aluminium ordinarily has no harmful effect on lung tissue but heavy doses make the lung more susceptible to tuberculosis infection.

British Standards lay down that the concentration of airborne stone dust of particle size 0.5-5 μ must not exceed 450 particles per c.c.

Asbestosis.

Asbestosis is a form of pneumoconiosis which results from the inhalation of hydrated magnesium silicate. An important feature of this disease is the presence of asbestos bodies in the lung and sputum. Fibrosis of the lungs developes faster in asbestosis than in silicosis and in extreme cases a person may die of asbestosis within 5 years of the onset of symptoms.

Talc Pneumoconiosis.

The dust in talc workers is made up of 50 per cent. hydrous magnesium silicate occuring as tremolite and slipstone. No free silica is present. There is a morphologic similarity between the fibres of talc and those of asbestos. The clinical manifestations of talc pneumoconiosis comprise dyspnea cough, cyanosis, clubbed fingers and diminished vital capacity and are similar to those in all types of pneumoconiosis.

Anthracosis.

Anthracosis occurs in coal miners and in city dwellers exposed to dusty atmosphere containing large amounts of smoke. The term means black lungs. The particles of carbon which find their way into the lung produce a phagocytic reaction similar

to that in silicosis. However, the carbon particles are inert chemically and do not cause local injury to the lung tissue. The foreign material may be carried to the liver, spleen, bone marrow, lymph nodes and other organs. The affected organs are dark in colour but show no local reaction unless silica is also present. In mine workers subjected to the inhalation of coal and silica dust mixtures, anthrocosilosis developes characterized by extensive silicotic fibrosis and marked emphysema. These patients show a high degree of tuberculosis. Miners in underground workings having poor ventilation are prone to this disease and the symptoms may be observed only after 15 to 20 years of working. According to British standard, the concentration of coal dust of particle size 1 to 5 μ should not exceed 850 particles per c.c. The usual symptoms of miner's pneumokoniosis are dyspnoea cough with coal black sputum and emphysematous chest.

SIDEROSIS.

Siderosis is caused by the inhalation of iron dust. The pathologic changes are due principally to the presence of silica in iron ore. Electric arc welders also are apt to develope this type of disease due to the fact that the electrodes used in the process are composed of approximately 99% ferrous material. The fumes arising from the electrodes contain inorganic substances, particularly finely divided iron oxide. Iron ore dust, though it produces pneumokoniosis, usually does not show fibrosis and is generally non-progressive and non-disabling though the alveolar walls indicate their pigmentation and consequent thickening. If, however, iron ore dust contains some free silica, fibrosis will result.

BERYLLIOSIS.

Berylliosis results from the inhalation of finely divided beryllium compounds. The disease is characterized clinically by pulmonary insufficiency, the major involvement being in the lung.

Dust suppression during drilling.

Where water supply is adequate, wet drilling is the practice

normally followed for dust suppression and it has been illustrated in Fig. 8.

Dry drilling with arrangements of collecting the dry dust has to be resorted to where wet drilling is difficult due to shortage of water or due to difficulties in transporting it to drill site *e.g.* on hill-tops, or places having bad access roads. One of the dry dust extractors in the market is the CP—Hemborn Dust Extractor manufactured by Consolidated Pneumatic Tool Company, Ltd., (U.K.). It can be used for opencast as well as underground drilling. The system makes use of a vaccum created in the collection unit which positively sucks all dust *actually from the tip of the bit*, through the drill rod and into the collector. Air is continually sucked into the hole during drilling, and the dust cannot escape from the mouth of the hole as sometimes happens in other methods using dust seals at the mouth of the hole. As no hood or seal covers the hole, the latter is always visible to the operator. Moreover the drill operator cannot avoid using the dust extraction unit as drilling mechanism blocks without it. A triple filter traps the dust praticles which fall to the bottom of the container and only clean dust-free air passes through the filter and is ejected to the atmosphere. The container may be filled upto $\frac{1}{3}$ of its depth with water so that the dust falls into it to form a sludge. Alternatively the dust may be trapped in the container in a dry paper bag.

A simple and inexpensive dust extractor manufactured by Indian Rock Drill (P), Ltd., Ponda, Goa for opencast mines is illustrated in Fig. 16. The exhaust air from the drill, laden with rock dust, passes to the surface through the space between the drill rod and the sides of the hole. It is carried to a wet jute bag through a pipe in which the suction effect is created by a venturi to which a regulated supply of compressed air is admitted. Rubber seals at the mouth of the hole and the suction effect inside the dust box prove sufficiently effective to prevent the dust from entering the atmosphere. This indigenously manufactured device has proved quite popular in Goa.

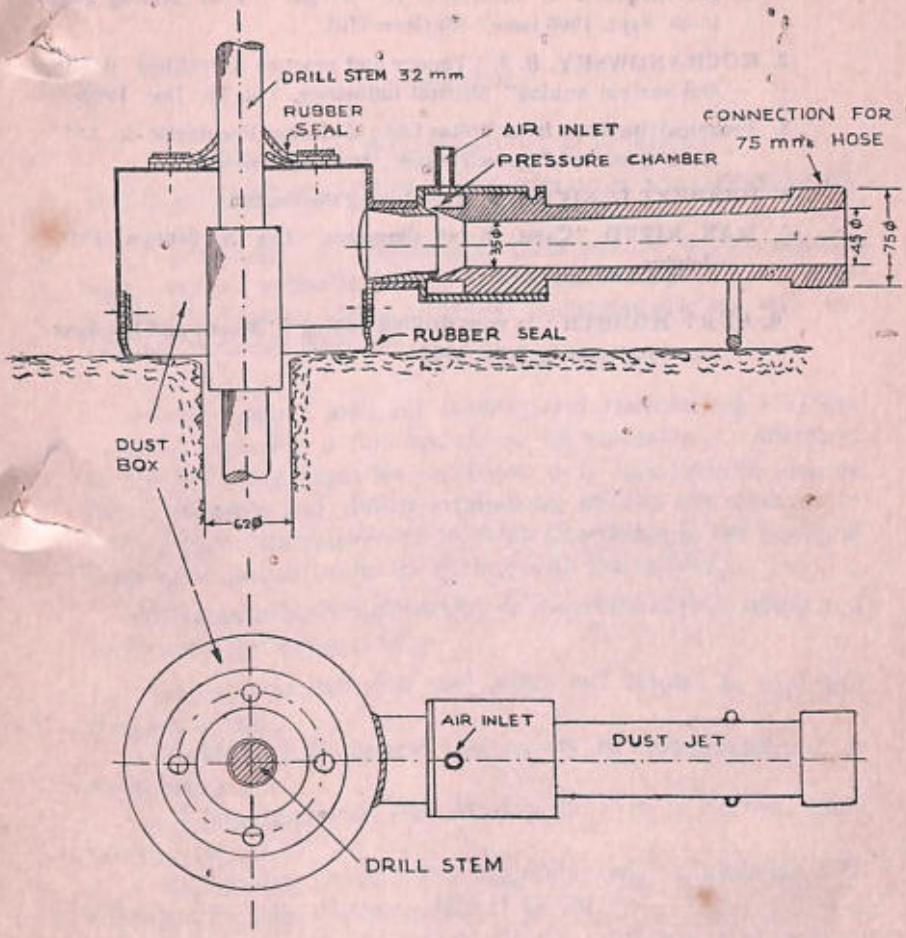

Fig. 16. Dry dust extractor for wagon drills.
Courtesy of Indian Rock Drill (P), Ltd. Ponda, Goa.

REFERENCES

1. ENGINEERING & MINING JOURNAL "1966 Mining guide book, Sept. 1966 issue" McGraw Hill.
2. KOCHANOWSKY, B. J. "Theory and practice of inclined drilling and surface mining" Mineral industries, Vol. 30, Dec. 1960.
3. Technical literature from Voltas Ltd.; Kirloskar Pneumatic Co. Ltd. ; Atlas Copco.; Ingersoll Rand ; Holmas Bros. Ltd.
4. HERBERT L. NICHOLS, Jr. "Moving the Earth."
5. MAX RITVO "Chest X-ray diagnosis" Lea & Febiger, Philadelphia.
6. BURT HONETH ; Is your drilling paying ?" Roads and Engineering Construction, Sept. 1962.

CHAPTER 9

HEAVY EARTHMOVING MACHINERY
BASIC PRINCIPLES FOR OPERATION & MAINTENANCE

1. Power unit. 2. Transmission (Mechanical). 3. Transmission (automatic) and hydraulic system. 4. Steering and suspension. 5. Brakes. 6. Traction ; crawler chains and tyres. 7. Tipping gear and other controls. 8. Fuels and lubricants.

The equipment used for loading and transporting rock/ore in mines calls for a fair knowledge of automobile machinery. In the following pages we shall give only fundamental idea of the automobile and similar equipment, leaving the students to acquire more theoretical and working knowledge in the field and from more exhaustive books dealing with the subject.

The commonly used equipment in mines for earth moving and heavy transport consists of :—

(1) Scrapers in a few coal mines and rippers in coal and iron ore mines,

(2) Drilling equipment run mostly by compressed air or hydraulic motors,

(3) Dipper shovels like Marion 93, P & H shovels, Lima shovels, etc.,

(4) Loading shovels like caterpillar 966, Caterpillar 977, Michigan 85, Allis Chalmers HD 11 G, etc.,

(5) Bull dozers like Euclid C6, Caterpillar D8, International TD 24, etc.,

(6) Road graders,

(7) Tippers like rear dumping Euclid trucks, Macks, etc.

Diesel engine is the prime mover on most of the equipment, petrol operated equipment being rare. Only in a few cases electric power from external source is supplied to dipper shovels or to heavy drills which move only within a limited distance

during the working shifts. High horse power machines, generally in excess of 250 H.P., are diesel electric, i.e. a diesel engine mounted on the machine drives a generator which provides electricity for the various controls.

The basic principles and guidelines for operation and maintenance have been explained in this chapter keeping in view a diesel operated truck or dumper, the most familiar equipment in the mines, and these basic principles have been elaborated at places to illustrate their sphere of application to other equipment. The study can be conveniently grouped under the following headings :

1. Power unit,
2. Transmission—Mechanical,
3. Transmission (automatic) and hydraulic system.
4. Steering and suspension,
5. Brakes,
6. Traction.
7. Tipping gear and other controls.

The chassis frame on which the power unit, gearbox, body and other controls are mounted in a tipper, loading shovel, dozer, etc. consists primarily of long heavy beams of I section, box section or channel section of rolled steel, joined by cross members. Brackets are welded to the frame for mounting the engine, the body and various other fittings. The chassis frame demands very little attention except that it should be washed periodically to keep it free from mud to prevent rusting. Any cracks or breaks observed after washing should be repaired by electric arc welding or by riveted reinforcements. Gas welding is not recommended.

1. POWER UNIT

The petrol and diesel engines are mostly straight line engines with vertical cylinders. They have many features in common. Each of these burns a mixture of air and finely sprayed fuel (petrol or diesel) in a closed cylinder. The heat of combustion causes expansion of gases resulting in a thrusting force which pushes a piston in the cylinder. The linear movements of the

piston are converted into rotary motion of crank-shaft with the help of a connecting rod. As the combustion of air-fuel mixture takes place inside a cylinder, the petrol and diesel engines are known as *internal combustion engines*. They convert the heat energy stored in the fuel into mechanical energy.

FOUR-STROKE ENGINE

In the larger power units used on cars, trucks and other heavy equipment, the engines are of four-stroke type, i.e. out of the four strokes of the piston completed in two revolutions of the crankshaft, only one stroke is the power stroke or working stroke and the force of gases resulting from combustion of air-fuel mixture is transmitted by the movement of piston to the crank-shaft during that power stroke.

In the four-stroke petrol engine, the cycle is as follows :—

1. *Suction stroke or induction stroke*

It is the down ward stroke. An inlet valve opens just as the piston commences its downward movement from top dead centre (T.D.C.) and atmospheric air, mixed with petrol in the form of mist, is sucked in through the inlet valve opening throughout the stroke. Exhaust valve remains closed throughout.

2. *Compression Stroke*

It is the upward stroke commencing from bottom dead centre (B.D.C.). As the piston commences its upward movement the inlet valve closes, the exhaust valve continues to remain closed and the air-fuel mixture is compressed. In ordinary petrol engines, the compression ratio is usually between 6 : 1 and 9 :1 and the pressure at the end of compression is about 6.5 to 8.5 kg/cm^2 with full throttle opening.

$$\text{Compression ratio} = \frac{\text{Volume swept by the piston in full travel} + \text{clearance volume}}{\text{Clearance volume}}$$

As the piston nears the top of the cylinder (top dead centre), a spark jumps across the electrodes of a spark plug. This is timed through an electrical component known as "distributor".

The spark causes combustion of the air-fuel mixture resulting in a sudden rise of temperature and pressure.

3. *Power Stroke*

The gases of combustion expand and the sudden rise of pressure forces the piston on its downward travel. Both valves remain closed.

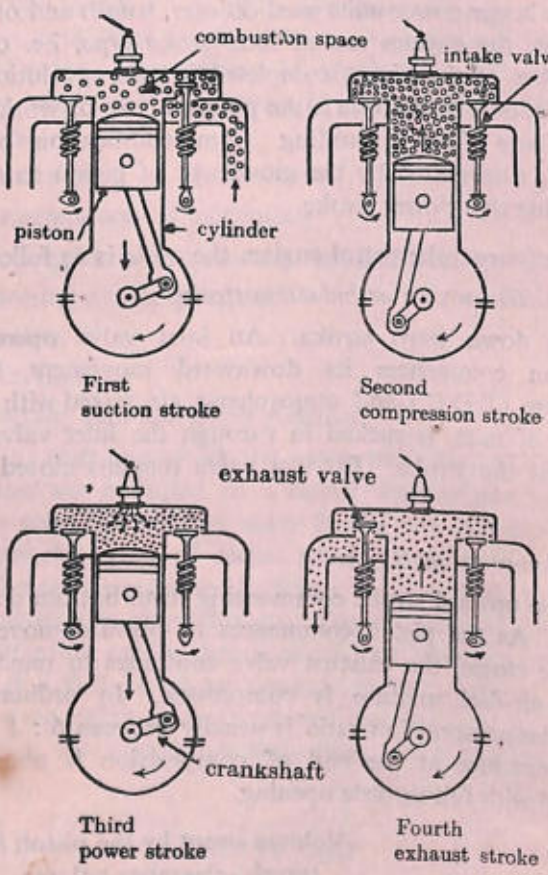

Fig. 1. Four stroke petrol engine

4- *Exhaust Stroke*

As the piston starts its upward travel, the exhaust valve is opened, the inlet valve remaining closed and the gases of com-

bustion escape to the atmosphere through the exhaust pipe and the silencer.

These four strokes constitute a cycle which is completed in two revolutions of the crankshaft and during the two revolutions the cylinder imparts power to crankshaft only during $\frac{1}{2}$ revolution yielding a very irregular motion. To attain smoothness in motion and performance a heavy flywheel is quite effective. For a regular turning effect (torque) to the crankshaft, 4, 6 or 8 cylinders cast in one block are commonly used in an engine. The sequence in which the power stroke in each cylinder takes place depends upon the sequence in which electric spark occurs in each cylinder when the piston is at the end of compression stroke. This sequence is called *firing-order* which is generally as follows depending upon crankshaft designs.

Four cylinder in-line engine : 1-3-4-2, or
1-2-4-3

Six cylinder in-line engine : 1-5-3-6-2-4, or
1-2-4-6-5-3.

In the 4-stroke diesel engine, the suction stroke sucks in only atmospheric air which is compressed to a high pressure during the compression stroke. The compression ratio is generally 14 : 1 to 18 : 1 and even 20 : 1 in some engines. The pressure at the end of compression is nearly 30 kg/cm^2 and the heat of compression raises the air temperature to nearly 500°c. This high temperature is sufficient to ignite diesel or other fuel oil which has self ignition temperature between 350°c and 450°c in air at atmospheric pressure, but lower at higher pressures. At the end of the compression stroke, a metered quantity of diesel oil is injected at high pressure in the cylinder in the form of an atomised spray. The high temperature of the compressed air ignites the fuel spray and the combustion results in rise of temperature and heavy pressure which forces the piston on its downward stroke (power stroke). The fourth stroke is upward and as in the petrol engine, it exhausts the gases of combustion. As the ignition of fuel is caused by compressional heat, the diesel engine is also known as a *compression-ignition engine*. The pressure in the cylinder after the combustion is nearly 45 kg/cm^2 and the high temperatures and pressures resulting in diesel engines demand robust

struction from high quality materials. For this reason, a diesel engine is costlier than a petrol engine, but it is more efficient. Moreover, cost of diesel oil is less than that of petrol.

Two-Stroke Engine

In a 2-stroke engine, the operations of suction, compression, expansion and exhaust are completed in two strokes. The crankshaft therefore receives a power thrust once in each revolution. One may expect that size for size, a two stroke engine develops twice the h.p. available from a 4-stroke engine, but this is not the case due to exigencies of design and the

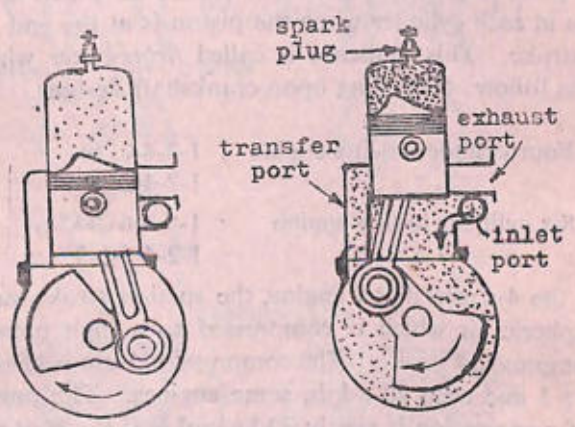

Fig. 2. Two-stroke engine (petrol)
left—end of power stroke.
right—end of compression stroke ; about to fire

increase of power in a 2-stroke engine may be only from 10-30%. A two stroke engine has no valves, but ports, cut in the cylinder wall. Compared to 4-stroke engine it is mechanically simpler, easier to manufacture, gives a smoother turning effect to the crankshaft, and needs a less heavy flywheel. Fig. 2 illustrates the working of a 2-stroke petrol engine. The piston top (crown), it may be noted, has a special shape.

The inlet port remains open partly during the upward stroke and partly during the downward stroke and admits a mixture of petrol vapour and air from the carburettor, not into the cylinder, but into the crankcase. The descending piston, near the end of its downward stroke, closes the inlet port and the

mixture is pushed from the crankcase into the cylinder via a by-pass port which is uncovered slightly later than the exhaust port. The piston, during its upward stroke first closes the by-pass port and then the exhaust port. Further upward movement of the piston compresses the fuel-air mixture and the ignition spark takes place when the piston is at top dead centre. The resultant combustion and expansion of gases thrusts the piston on its downward travel, imparting power to the crankshaft. The descending piston uncovers the exhaust port and the burnt gases exhaust to the atmosphere. The cycle is thus repeated.

The typical piston shape prevents the fuel-air mixture from passing straight out through the exhaust port since the mixture is deflected up into the cylinder. During the upward stroke, the piston shape assists in scavenging out on any burnt out gases which might have remained in the cylinder. During this scavenging process it is unavoidable that part of the compressed fuel-air mixture enters the exhaust. This results in slightly increased fuel consumption as compared to that in a 4-stroke engine.

There is no oil in the crankcase. Engine lubrication is achieved by adding lubricating oil to the fuel (about 25 litres fuel and 1 litre oil). There are no valves and therefore no need for camshaft, tappets etc. The piston rings are kept in their position by lugs; otherwise, the ends (joints) of the piston rings may expand into the ports and cause serious damage. The big-end bearing in a 2-stroke engine is not shell bearing, but roller type. A two-stroke engine requires more frequent decarbonising of cylinder top, combustion chamber, exhaust port and manifold, and by-pass port than in a 4-stroke engine.

The two-stroke diesel engine works in the same way as a petrol engine but there is no spark for ignition. In a diesel engine, there is not as much loss of fuel during the scavenging action as in a petrol engine, because only air is sucked in and not fuel-air mixture.

The two stroke engine is not as efficient as the four-stroke engine and its use is limited to engines of small horsepower, e.g., for motor cycles. In quarries, among the machines with high h.p., the Atlas Copco Monobloc-250 compressor is equipped with a 2-stroke diesel engine.

Fuel Supply

In petrol engine a fuel pump, either electrically operated, or mechanically operated by a cam in the engine, sucks petrol from the petrol tank and delivers it to a carburettor which feeds it to the cylinders. In a diesel engine a mechanically operated fuel pump sucks diesel from the diesel tank and feeds it to injection pump which delivers metered quantity of diesel to each injector at the appropriate time. Fuel filters are provided as shown in fig. 4. Fuel supply by gravity feed is sometimes practised for stationary engines, but very seldom on high h.p. engines of moving machines.

CARBURETTOR

Its function is to supply petrol and air mixture to engine

(i) in the approximate ratio of 1 : 15, and

(ii) in requisite quantities to cope up with load requirements.

During the suction stroke, the engine sucks in atmospheric air which passes through an air cleaner to the intake pipe of the carburettor. A constriction (technical term *venturi*) is provided in the intake pipe so that velocity of intake air increases at the narrow passage. A little opening (carburettor jet) connected to fuel supply at this narrow passage (venturi) enables the flowing air to suck the fuel which then mixes with the air in the form of a very fine mist. In this way the carburettor supplies fuel-air mixture to the cylinder and the quantity is regulated by controlling the air passage in the induction pipe (intake pipe) by a throttle valve operated by the accelerator pedal. When the accelerator pedal is pressed by the foot, the valve (called butterfly valve) increases the opening in the air intake pipe, and larger the quantity of air passing, more is the petrol sucked in from the carburettor. And with more fuel : air mixture burning in the engine the latter runs faster yielding more horsepower. When the foot is off the accelerator pedal, the throttle valve is closed and it allows just enough air-fuel mixture to pass to the cylinders for idling speed. A float chamber forms a part of the carburettor. A float in the float chamber maintains the fuel level always at level with the jet tip and it closes

the inflow of fuel from the fuel pump by blocking the passage to the float chamber with a needle when the petrol level rises to the height of the jet (fig. 3).

A punctured float gets filled with fuel and ceases to maintain correct fuel level in carburettor, resulting in overflow of fuel and faulty engine operation. The jets need replacement if their orifice (jet diameter) becomes large.

CHOKE

A choke is provided for initial starting in the day or for cold starting. It is essentially an arrangement to supply enriched fuel-air mixture to the engine at the time of starting. This is achieved (i) by enabling the petrol flow to bypass fine passage of the jet, or (ii) by restricting the air passage in the intake pipe. In the latter case, a choke may not work satisfactorily if the

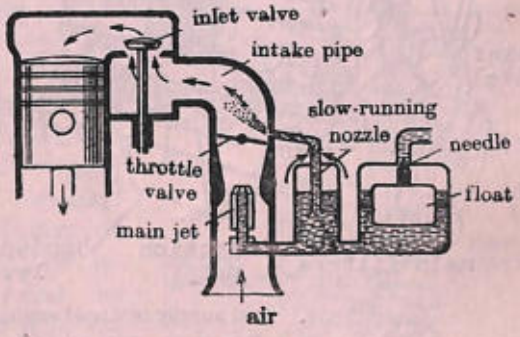

Fig. 3.
Carburettor (Schematic)

accelerator pedal is pressed when the choke is pulled. It is to be noted that a choke should be pulled only to start the engine when cold and it should be pushed back after the engine has warmed up, i.e. after a few seconds.

Fuel injection pump

In a diesel engine, the injection pump and the injection nozzles :

(i) Control the pressure at which fuel is injected into the cylinder,

(ii) Control the quantity of fuel required for varying loads, and

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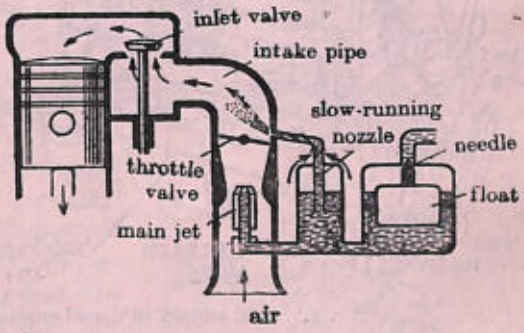

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In a diesel engine, the injection pump and the injection nozzles :

(i) Control the pressure at which fuel is injected into the cylinder,

(ii) Control the quantity of fuel required for varying loads, and

(iii) inject the metered quantity of fuel into the cylinder at the predetermined moment.

The fuel injection takes place in the form of atomised spray when the piston is at the end of its compression stroke. The heat generated during compression of the sucked air is sufficient to cause combustion when the diesel is injected as atomised spray in the combustion space filled with hot compressed air. No electric spark, as such, is necessary. When a number of cylinders are used in an engine, the sequence in which the injection takes place to the different cylinders is known as firing order, and is controlled by the injection pump which is an essential

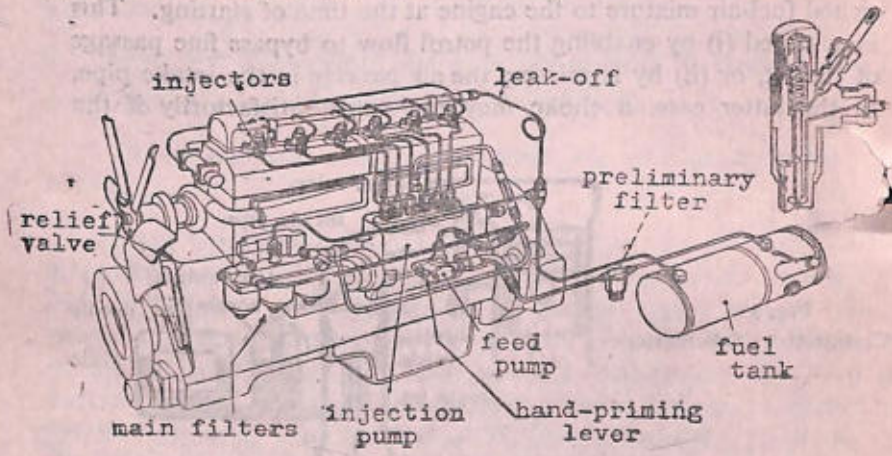

Fig. 4. Fuel supply in diesel engine (Schematic)
Top right—Injection nozzle.

unit of a diesel engine (Fig. 4). An injection pump suitable for a firing order of, say, 1-3-4-2 in a four cylinder engine, will not be suitable for another engine having a different firing order, say, 1-2-4-3. An injection pump has the same number of pump elements as the number of cylinders in the engine and each pump element consists of a pump barrel, the pump plunger and a delivery valve. All the elements are combined in a common housing. The plungers arranged in a line are operated by a camshaft in the sequence of the firing order. It is essential that all the cylinders in an engine receive the same quantity of fuel. The plunger stroke in the pump element is fixed so that the quantity of fuel pumped during each stroke is also

fixed but the quantity of fuel supplied to the cylinder is variable due to the milled "helicoid" provided at the top end of the plunger which slides in the barrel. Whenever the plunger is turned about its longitudinal axis, the control edge varies the feed of the fuel to the fuel pipe. The quantity of the injected fuel is metered by the pitch of the plunger's control edge and the extent of its turning. The surplus quantity of fuel flows back to the diesel tank *via* an excess-fuel-return-pipe. Simultaneous turning of all the plungers is performed by a toothed control rack controlled by the accelerator pedal.

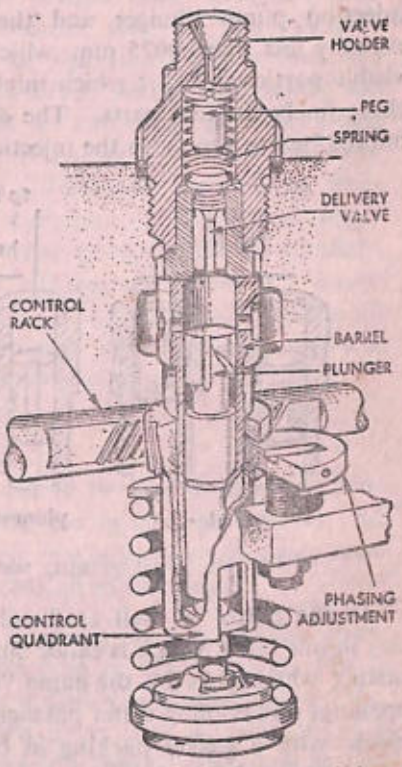

Fig. 4a. Pumping elements of the C.A.V. N-type injection pump

A **governor** which is a device that keeps the engine speed constant regardless of changes in the engine load, is an essential component of every fuel injection pump. The governor automatically increases or decreases the fuel supply to the cylinders in accordance with the load requirements—the more fuel, the more power ; the less fuel, the less power. It performs this function by turning the fuel pump plunger about its longitudinal axis and this is achieved by a system of levers actuated by the governor mechanism. The governor may be (i) pneumatic and centrifugal type, or (ii) mechanical type.

When the operator presses the accelerator pedal with his foot, the butterfly valve in the intake air pipe of suction manifold increases the opening for air passage and the increased suction effect actuates the diaphragm of a pneumatic governor. The diaphragm is connected through levers to the toothed

rack which turns the plungers in the pump elements and the turning results in more fuel supply to the cylinder as already described.

The fuel injection pump is a very precision equipment and should never be tampered with. The clearance between the fuel injection pump plunger and the pump barrel is very small, actually less than .0025 mm, which makes it evident that the invisible particles of dirt which might pass with diesel can damage these finely finished parts. The diesel should, therefore, be well filtered before supply to the injection pump.

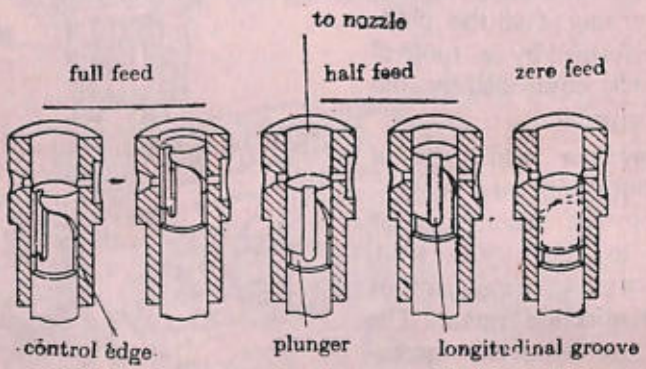

Fig. 4b. Fuel quantity metering by plunger rotation

In the petrol as well as diesel engines, all the cylinders are cast in one block which is called simply "engine block". Another casting which goes by the name "cylinder head" contains valve openings and cooling water passages. It is bolted to the top of the block with a gasket packing in between to make the assembly air tight and water tight. A crankcase accommodating a lubricating oil sump is bolted to the lower portion of the block.

Piston and Piston Rings

The piston has, at the top end, two or three grooves to accommodate piston rings and one or two grooves below them to accommodate oil scraper rings. Some pistons have, at the lower end, one more groove for a scraper ring. The piston rings ar required to seal the gas pressure resulting from the combustion and to avoid its leakage into the crankcase. The face of the

piston rings has to contact the cylinder wall uniformly and closely. In course of use the rings become worn due to usage or abuse. Poor performance (efficiency) coupled with excessive engine oil consumption is indicative of the need for piston and/or rings replacement. It is also evident from increased smoke emerging from the silencer pipe. Oversize rings of 10 oversize may be fitted without reboring the cylinder. (10 oversize means outer dia. of the ring equals standard dia. of piston+.010 inch). With more wear, the cylinder has to be rebored and a new set of matching pistons, rings, gudgeon pins has to be used. Reboring may be done only thrice, i.e., upto 60 oversize rings. Thereafter the cylinder walls become too thin. Some engine cylinders are fitted with liners or sleeves which can be replaced when worn so that the standard size piston and rings can be used. If for any reason it is essential to change only one piston, it is a good practice to change all the pistons in the cylinder to maintain a proper dynamic balance. The same applies to change of sleeves and rings.

Air cleaners and filters

Air, lubricating oil and fuel should be free from dirt, particularly dirt of abrasive nature, as it causes wear of cylinder and other moving parts. Earth moving machines work in dusty conditions which demand frequent attention to this aspect.

Air cleaners used for petrol or diesel engines and on compressors are of two types : (1) Wet type, or (2) Dry type. Fig. 5 shows a *wet type air cleaner*. Air is drawn in through fins on the underside of the precleaner which gives it a whirling motion throwing the heavier dust particles out through the slots in the dome of the precleaner. The air then passes down the centre tube, striking the oil in the base, and throwing out further dust as the air flow direction and momentum is reversed. The air then passes through wire gauzes which clean off the dust laden oil as the clean air flows towards the outlet and then to the engine. The oil and dirt deposited on the screens drain into the outer portion of the oil cup which is undisturbed by the airflow. The dirt then settles at the bottom of the oil cup leaving the surface oil clean. All the cleaning is done in the oil cup and lower screens ; the upper screens always remain clean.

Maintenance of air cleaner after100 to 200 hours' operation under dusty conditions is essential. The oil container should be cleaned and filled with fresh lubricating oil upto the level marked on it but not more. The lower screens should be washed by kerosene or petrol and thoroughly dried.

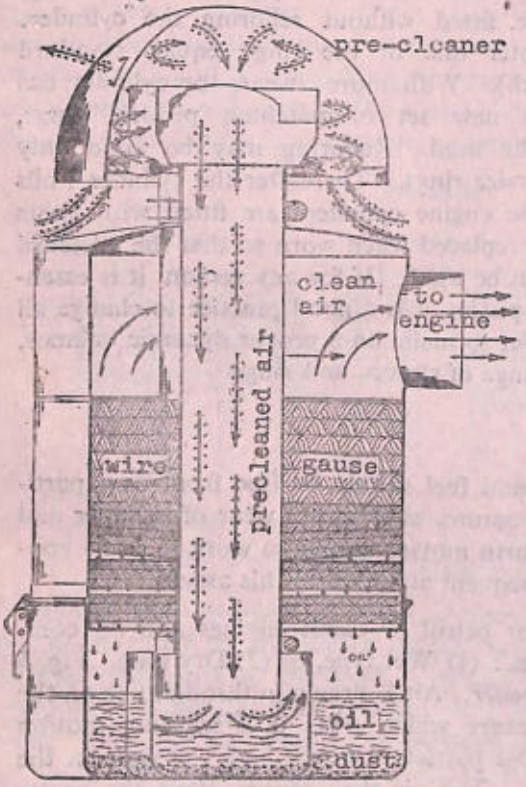

Fig. 5.
Air cleaner ; wet type or oil bath type

In a *dry type air cleaner* (Fig. 6) the air enters the cleaner body (*1*) through the inlet tube (*2*) fitted with a wire mesh to prevent entrance of coarse particles. It then enters a number of funnel shaped nylon tubes (*5*), having spiral vanes (*3*). The vanes give the air a whirling motion and as the air rotates down the tube, dust particles are thrown out and fall down the tube and through the funnel shaped tip into the dust cup (*4*). The cleaner air passes the central column of each funnel shaped tube into the filter element which consists of resin impregnated cellu-

lose (commonly called paper type). It removes the last traces of dirt from the air which then passes through a hose to the engine.

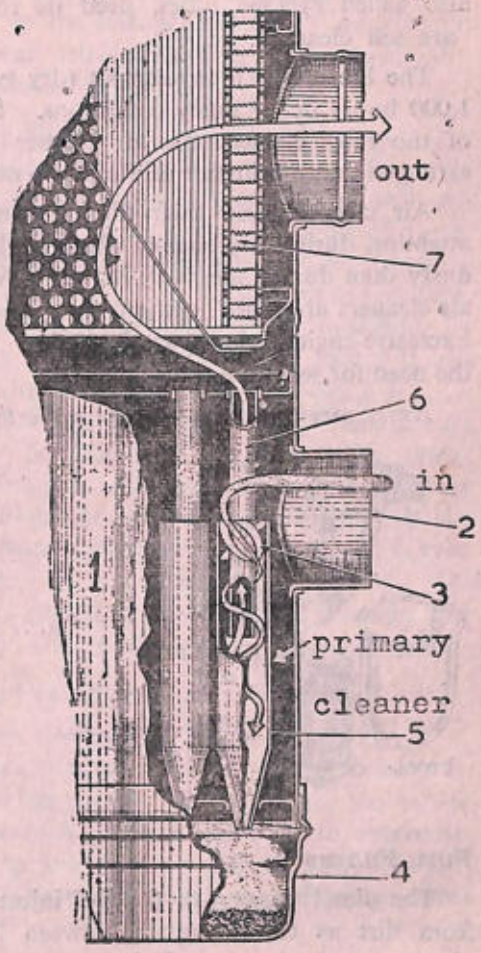

Fig. 6.
Air cleaner dry type

The dry type filter needs more frequent maintenance, though simpler, than the wet type, After 20 to 50 hours of operation under dusty conditions remove the dust cup (4), empty and clean it dry. The paper filter element is cleaned after every 100 to 200 hours by a jet of compressed air at a pressure not exceeding 3.5 kg/cm^2. No oil should be placed in

the dust cup. The filter element can also be cleaned with clean water but it should be thoroughly dried before re-use. It should not be washed with petrol. The funnel shaped tubes, also called cyclone tubes, need no routine attention as they are self cleaning.

The life of the filter element (dry type air cleaner) is 500 to 1,000 hours under dusty conditions. To extend the service life of the filter element the air cleaner inlet pipe should be so arranged that exhaust or oil fumes do not enter it.

Air cleaners and their filter elements need more frequent attention during dry season when working conditions are more dusty than during the rainy season. Wet type i.e. oil-bath type air cleaners are more efficient for dusty conditions in India. Excessive engine exhaust smoke and/or loss of power indicates the need for servicing air cleaner.

Never service the air cleaner while the engine is running.

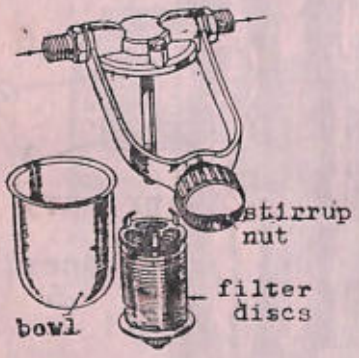

Fig. 7. Disc type primary fuel filter

Fuel Filters

The diesel passing to the fuel injection pump should be free from dirt as the clearance between the fuel injection pump plunger and the barrel is very small, actually less than 0.0025 mm (.0001 inch). There are usually two filters, a primary filter usually between the fuel tank and the fuel feed pump, and a second or final filter in the delivery line of fuel feed pump. The primary filter element consists of a number of perforated brass discs (Fig. 7) and is also called edge-type filter. It should be cleaned by petrol after 50 hours of operation and the filter

housing drained of sediments. The primary filter element sometimes consists of a felt tube, e.g., in the Mico dual filter which consists of two bowls, one containing a felt tube insert as the primary stage filter and the other containing a resin impregnated paper element as the final stage filter. The felt tube insert should be cleaned with petrol after every 250 hours and discarded after 800 to 1,000 hours.

The final or secondary filter is always a paper element. The filter housing is provided with a drain plug. After every 50 hours of operation, open the drain plug to remove sediment and water which settles at the bottom. After 250 hours of operation, the filter element (paper) should be discarded though dirty fuel often necessitates its cleaning before the 250 hours servicing. A filter element clogged with dirt particles is indicated by refusal of engine to attain full speed or take full load. It can be washed once or twice with petrol and re-used though it is not a recommended practice. The performance of fuel filters should be checked after every 50 hours of running, the check being made when the engine is running on load, by loosening the air release screw on the filter and watching the fuel jet. If the flow is uniform and free from air bubbles and the engine is running smoothly, the filter is operating satisfactorily. On the other hand, if the oil is ejected in small spurts it is a sign that the filter element is clogged and needs cleaning or replacement.

Expelling Air from the Fuel System (Air Bleeding)

For the injection pump elements to function properly, it is necessary that the elements, fuel oil pipes and filters be entirely free from air. Should air be present in the system, the pump elements cannot attain the high pressure necessary to overcome the spring tension on the sprayer needless and lift them off their seats. To remove air from injection system the following procedure should be followed—

Check that there is enough oil in the fuel tank and all connections secure. Loosen the top air release screw on the cover of the fuel oil filter. Operate the fuel feed pump by hand priming lever, till the fuel coming out is free from air bubbles and then tighten the air vent screws whilst still pumping. Next vent the injection pump by loosening the air vent screws

provided on the top of the main pump body, operate the hand priming pump and close the air vent screw as soon as frothing ceases.

Note—When the engine is cool, the fuel oil pipe unions at the injection valves should also be loosened, to ensure complete air expulsion. Secure the air release screws (and fuel oil pipe unions, if loosened) when the fuel oil at all discharge points is entirely free from air.

TURBOCHARGER & SUPERCHARGER

In a diesel engine if the air entering the cylinder during the suction stroke contains more oxygen, naturally more diesel can burn in it and yield more power. The same object is achieved by admitting air under higher than atmospheric pressure to the cylinder and a turbocharger or a supercharger is used for this work. A turbo-charger consists of a set of turbine vanes driven by the high velocity exhaust gases and a compressor or blower by the shaft of the turbine vanes. The blower forces air into the cylinder at a pressure higher than atmospheric. (Fig. 8). The waste energy of the exhaust gas is thus utilised to increase the efficiency of the engine and yield more H.P. At higher altitudes the atmospheric air is rare and it has less oxygen to combine with the fuel which diminishes the engine power. In general power available drops by 1% for every 100 m altitude above sealevel in the case of a 4-stroke engine. An engine equipped with a turbo-charger is an advantage in such cases.

A turbocharger runs at very high speeds of 15,000 r.p.m. or more and has very close limits of balance and clearances. It is lubricated with oil supplied from lub oil crankcase of diesel engine. The oil strainer provided for the lub oil should be removed and washed with petrol after every 1,000 service hours.

In the earthmoving machinery some of the high h.p. machines are equipped with turbo chargers, e.g. Caterpillar tractor shovel, 977; Interna ional TD25 bulldozer.

Timing—Timing refers to

(a) Valve timing, and

(b) Ignition timing in the case of petrol engines and fuel injection timing in the case of diesel engines.

The *valve timing* is controlled by the camshaft, push-rod, rocker arm and the tappet clearance. This is set during the initial engine assembly and subsequent engine overhaul and needs no attention in the day to day operation or periodical maintenance. Only the tappet clearances should be checked after every 1,500 km or so. The correct clearances are always stated in the maker's instruction book. Too big a clearance results in excessive noise and wear but if the clearance is too small, the valve cannot close properly and the gases, escaping during the working stroke, will soon burn the valve head and the seating. If the clearance of the intake valve is too much, the rate of filling the cylinders with fuel-air mixture is in-

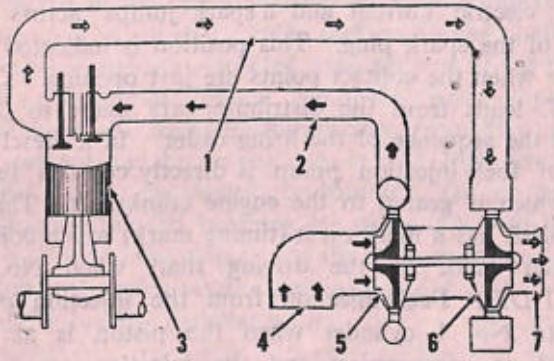

Fig. 8. Turbocharger (Schematic)
1. Exhaust air from engine. 2. Compressed air to engine. 3. Cylinder. 4. Atmospheric air. 5. Compressor blades. 6. Turbine. 7. Exhaust to atmosphere.

sufficient. Too much clearance of the exhaust valve considerably slows down the escape of hot exhaust gases. Too little clearance results in insufficient closing of the valves.

The ignition timing is controlled by the setting of distributor in a petrol engine. The ignition spark in each cylinder takes place when the piston is approaching the end of the compression stroke. This generation of the ignition spark, known as firing, takes place in a definite sequence for the different cylinders and the sequence, as already stated, is called firing order. On most of the engines, there is a mark on the crankshaft pulley,

or on the flywheel which coincides with another fixed mark on the body when No. 1 piston (nearest to fan) is at T.D.C. It can be at T.D.C. on two occasions, once when it is at the end to compression stroke, and again when it is at the end of exhaust stroke. The piston position at T.D.C. at the end of compression stroke is indicated by the position of tappets when the tappet cover is opened as the inlet and exhaust valves are both in closed position. Alternatively, when the spark plug is removed and the thumb placed on the hole, the air pressure due to ascending piston gives a pressure on the thumb and this indicates when the piston is approaching the end of the compression stroke. At that moment the high tension wire leading from the distributor to the spark plug of No. 1 cylinder carries electric current and a spark jumps across the electrodes of the spark plug. This position is indicated in the distributor when the contact points are just opening. Connections of H.T. leads from the distributor are made to other cylinders in the sequence of the firing order. In a diesel engine the shaft of fuel injection pump is directly coupled to a driving shaft which is geared to the engine crankshaft. The injection-pump-shaft has a mark on it (timing mark) which coincides with a similar mark on the driving shaft when No. 1 piston is at T.D.C. Fuel injection from the injection pump takes place to No. 1 cylinder when the piston is at T.D.C. at the end of compression and the injection pump is always set by the dealers for such timing.

If the distributor in a petrol engine is so set as to cause a spark earlier than the T.D.C., the ignition timing is said to be "advanced". If, however, the distributor is so set as to cause a spark later than the T.D.C., the timing is said to be "retarded". The same terms are used for diesel engine with the difference that fuel injection takes place instead of spark.

Engine Lubrication

The methods by which various moving and rubbing surfaces receive lubrication in a four-stroke engine are :

1. Force-feed lubrication.
2. Oil splash lubrication.
3. Mist and oil vapour lubrication.

The main bearings and big-end bearings in a 4-stroke engine are of shell type and are lubricated by force feed system of lubrication. A gear type oil pump, located in the oil chamber and driven by the crankshaft delivers oil under pressure through oil passages in the engine block to the main bearings of the crankshaft. The journals of the crankshaft resting in these main bearings are drilled and connected by internal passages in the crankshaft to the crankpins. The oil emerging at the crankpins lubricate the big-end bearings of the connecting rod. The same oil is circulated after it passes through a filter. The cylinder walls receive oil which is splashed by the centrifugal effect of the crankshaft rotation. For this purpose a small hole for oil splash is drilled in the upper surface of each connecting rod bearing housing. Cam shaft, valve rockers, timing chain and its sprockets receive through restricted passages part of the oil which is pumped up by the oil pump. Gudgeon pins, tappets and the cylinder walls receive lubrication from the oil mist which is present in the crankcase at the high temperatures which arise when the engine is running. (In a two-stroke petrol engine the big-end bearings are roller type and the lubricating oil is added to the petrol. As the mixture enters the crankcase the lub. oil settles on the connecting rod and main bearings and also lubricates cylinder walls).

When the engine runs, the lub. oil pressure is generally 2 to 6 kg/cm^2. Some vehicles use oil pressure indicator lamp (tell-tale lamp) in place of the oil pressure gauge. The lamp burns when the oil pressure is low, but extinguishes when it builds up above the minimum pressure specified for adequate lubrication. If the engine fails to build up the pressure within seconds of starting, the engine should be stopped and the cause investigated.

A lub. oil sump usually carries a dipstick having two marks, the higher one indicates normal oil level and the lower one, the danger level, below which the oil level should never be allowed to fall. It is to be noted that the lubricating oil assists the cooling of the engine and oil level midway between the danger mark and "normal level" mark, though satisfactory for lubrication, is not helpful for the cooling. The oil level should be checked when the engine is cool, or if it has run, after minimum of

15 minutes' stoppage, as the circulating oil takes some time to come back to the oil sump.

The lub. oil is used up partly by the influence of the temperature and partly by abrasion of the bearing surfaces. In course of use it becomes slightly blacker and somewhat diluted. due to H_2O (steam) formed during the combustion of hydrocarbons. In the case of new or reconditioned engine, the oil should be changed after 500 km of run ; in other cases after 1,500 km of operation or after 200-250 hours. The suitable engine oils have been described under the section "Fuels and lubricants" at the end of this chapter.

High oil consumption is due to wear on the cylinder and piston rings. Such state of engine cylinder and piston rings is indicated by low oil pressure when the engine warms up and is the time for engine overhaul.

Engine lubrication is badly affected if the lub. oil filters do not receive proper care and maintenance. They should be changed after every 200-250 hours of operation, when changing the lubricating oil. The filter element is usually of resin impregnated paper type, pleated and encased within a perforated metallic cover like the secondary fuel filter. On diesel engines usually two or three filters are placed in series and a by-pass valve located in the top cover of filter housing is provided to pass oil direct to the engine if the filter element gets clogged with dirt.

Cooling Arrangements

The fuel combustion in an engine raises the temperature of the components. To prevent their overheating it is essential to cool the engine in the region of combustion space and cylinder. Air cooling is practised only on small engines, *e.g.*, on motor cycles, and some cars like the Volkswagen. But the larger engines are always cooled by circulation of water from the radiator. The coefficient for the heat transfer from metal to air is only 1/80 of that for metal to water. A water pump circulates water of the radiator in a closed circuit and as the water passes through the water jacket surrounding the cylinder and its components, it absorbs the heat which is conducted to the radiator tubes. A belt-driven fan behind the radiator

sucks air through the spaces between the radiator tubes which dissipate their heat to the air. The passage of this air over the engine also assists cooling of the latter.

Dirty water should never be used to fill the radiator as its use over some period results in choking of the radiator tubes. When the engine is so hot that radiator water is boiling, cold water should not be poured into the radiator as this may cause a crack in the engine block due to the stresses of large temperature variation. The water may be poured slowly with the engine running or when the engine has cooled down.

After 200-250 hours of operation, the radiator should be cleaned of any accumulated sediments by opening its drain cock.

The cooling water may reach unduly high temperatures or may boil due to any of the following causes.

1. Insufficient quantity of cooling water in the radiator ;
2. Foreign deposit in the radiator ;
3. Slipping of the fan belt ;
4. Incorrect timing of ignition or valves, wrong carburettor adjustment ;
5. Clogged exhaust ;
6. In the case of new engine or overhauled engine, racing it before the running in period is over ;
7. Engine badly in need of decarbonising.

In very cold weather, to prevent the radiator water from freezing and thereby causing damage to the radiator and engine, antifreeze mixture (chiefly consisting of glycerine) is added to the cooling water.

Starting an Engine

An engine is started by one of the following methods :

1. By cranking handle
2. Direct by starter motor
3. By compressed air start
4. By auxiliary engine start
5. By firing a powder cartridge.

1. Small diesel engines, particularly those of portable compressors, may not be equipped with a dynamo or battery and are often started by a cranking handle.

2. Larger diesel as well as petrol engines are started with the help of a starter motor which receives current from 6V, 12V or 24V storage battery when the starting switch is operated. As the starter motor rotates, a movable pinion on its shaft engages with the flywheel of the engine and rotates it. When the engine starts running, the starter switch should be put off. This causes the pinion to move away from its meshing position as the starter motor slows down and comes to rest. The starter motor should not be allowed to run a moment longer than is necessary to start the engine. Car engines, truck engines, heavy duty dumper engines (like NH 220, Cummins, 220 H.P. at 2,100 r.p.m. on Euclid dumper) and even tractor shovel engines upto 250 H.P., are started in this way. Some engines of higher H.P. may also be started in this manner but a decompression lever is utilised at start to reduce the heavy compression ratio which would otherwise demand high starting current to set the revolving engine parts in motion against the heavy compression.

3. Large H.P. stationary diesel engines are started by admitting compressed air at about 16 kg/cm^2 from a special receiver into the cylinder of the diesel engine. By suitable air-starting valve arrangement the intake manifold is utilised for such admission and the compressed air sets the pistons in motion. Starting air is not admitted to all the cylinders but generally to only 3 cylinders of a six-cylinder engine.

The diesel injection is then allowed to take place and the engine starts in the normal way. When the engine starts running, the air-starting valve lever is moved to the "run" position ; this stops the admission of compressed air and permits atmospheric air (or air from supercharger/turbocharger, if provided) to enter the intake manifold. The compressed air, once the receiver is filled, lasts for 6 to 7 starts. A separate compressor driven by the main engine is not always essential as one of the cylinders of a multicylinder engine can be utilised, by suitable arrangements, to work as compressor for a short while when the remaining cylinders work the engine.

4. Heavy duty diesel engines are sometimes started by an auxiliary petrol engine, *e.g.* on Caterpillar Traxcavator 977 model, 235 H.P. crawler bulldozer of Bharat Earthmovers, Ltd., etc. To start it, the main diesel engine is declutched, the auxi-

liary petrol engine mounted on the same chassis is started with the help of a battery and starter-motor and when the petrol engine attains full speed, it is engaged with the flywheel of the main diesel engine through a clutch arrangement. After the main engine is started, the petrol engine is declutched and stopped.

5. A simple method to start a diesel engine is by firing a powder cartridge and is adopted on the Atlas Copco Compressor Monobloc-250. The method is however getting obsolete.

In the powder cartridge method, no cranking handle is used. Specially impregnated ignition paper rolls, detonating caps and starting cartridges are required to start the engine. These materials, as they are classed as explosives, are supplied by the manufacturers in sealed tins. To start the engine, turn the flywheel with a lever so that "Start" mark on it comes in line with "Start" mark of the body; charge the cartridge starting device and ignition paper roll holder and place it in position on the cylinder ; then strike the firing pin sharply with a light hammer. This causes the powder charge to explode ; the piston is depressed and the crankshaft rotates thereby starting the engine.

The impregnated ignition paper rolls, detonating caps and starting cartridges are imported and difficult to get. The machines started by this method are also replaced by the manufacturers by other machines which are started by storage-battery powered starter-motors.

Decarbonising

The burning of fuel and some of the lubricating oil (4 stroke engine) results in formation of carbon deposit on the crown of the piston, the valve stem, the combustion space, inlet and exhaust passages and the exhaust manifold. As these deposits reduce the combustion space, and the passages of intake air and exhaust gases, the engine performance gradually goes down. Excessive accumulation of carbon on valve stem causes sticking of valves. The carbon deposit in the combustion chamber may glow with the heat of compression as the compression ratio is increased and in a petrol engine, the heat may ignite the compressed fuel-air mixture even before the ignition

spark is produced by the spark plug. This is called pre-mature ignition (or pre-ignition).

Somewhat similar phenomenon sometimes results when the engine continues to run even when the ignition key is switched off. It is, therefore, essential to decarbonise at intervals those parts in the engine which accumulate carbon deposits. This interval is difficult to generalise, but in a 4-stroke petrol engine, decarbonising may receive attention after a run of about 20,000 km if the engine performance is gradually going down. A surer way is to test compression of the engine by proper compression gauge. Valve seat cutting and valve grinding is an operation generally undertaken during decarbonising. In a two-stroke engine, the lub. oil burns with the fuel and the carbon deposit builds up much faster than in a 4-stroke engine and calls for decarbonising at shorter intervals.

A diesel engine does not need decarbonising as frequently as a petrol engine. An average engine may need this attention after 30,000 to 40,000 km but the valve grinding should not be delayed so long.

2. TRANSMISSION (MECHANICAL)

The transmission covers the following—

1. Clutch
2. Gearbox
3. Propeller shaft
4. Differential
5. Rear axle
6. Wheels and hubs.

The flywheel is attached to the engine crankshaft at the rear end. It has teeth on the circumferance and the pinion of the starting motor engages with it whenever the latter is set in motion by the starter switch to start the engine. Immediately after the engine starts, the pinion should be disengaged from the flywheel by stopping the starter motor.

CLUTCH

A clutch is a device by which two shafts turning on the same axis *i.e.* in line with each other, can be connected and discon-

nected. It is mounted behind the flywheel. It disconnects the engine from the gearbox and enables the former to run without moving the vehicle. The clutch used on trucks, dumpers, etc. is of the dry disc or single plate type. The driving member of a single-plate clutch is actually the machined face of the engine flywheel. The driven disc is the clutch disc, lined on both sides with a special friction material (rather like a brake lining). It is mounted on a splined shaft which drives the gearbox. (Fig. 10). The rear part of the clutch disc can be brought

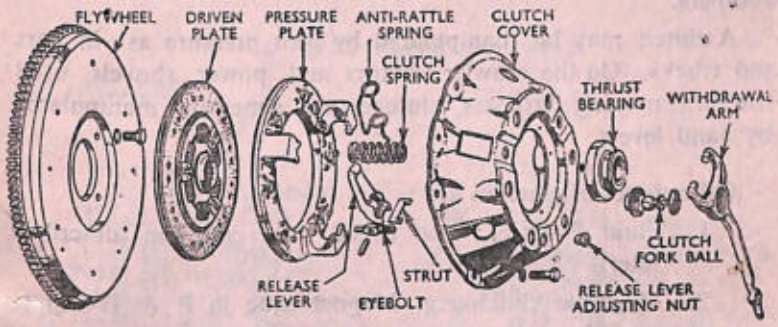

Fig. 10. Friction clutch components

into contact with a hardened steel plate called the pressure plate, attached to the flywheel rim but also able to move a little towards or away from the flywheel face. A series of strong springs normally forces the pressure plate on to the clutch disc, pushing this in turn against the flywheel, thus engaging the drive. A slight clearance must exist between the clutch release levers and thrust bearings when the clutch is engaged and the pedal is against its stop. This ensures that the pressure plate can move fully forward. With the wear of the linings on the clutch disc, the pressure plate will move closer to the flywheel and reduce the clearance. Clutch adjustment which involves readjusting the linkage between the clutch pedal and the withdrawal arm, therefore, becomes necessary from time to time to prevent clutch slip.

Any oil or grease on the clutch disc reduces the friction which keeps the clutch disc in contact with the pressure plate. The clutch facings on the clutch disc wear out in course of use and have to be renewed.

There is also another method of withdrawing the pressure plate when the clutch pedal is pushed down and this consists of a clutch "Fork" and levers. On some cars (*e.g.* Standard Herald) and trucks the mechanical connection between the clutch pedal and the withdrawal fork is assisted by hydraulic pressure system as in the case of the familiar hydraulic braking and this requires the use of a separate master cylinder for clutch. In some trucks and dumpers equipped with air brakes, the clutch is sometimes operated by compressed air, *e.g.*, some Euclid dumpers.

A clutch may be manipulated by foot pressure as in cars and trucks. On the crawler tractors and power shovels used for earth moving purposes, clutches are generally manipulated by hand levers.

Other types of clutches are :

1. Fluid flywheel ; also called fluid coupling (described later)
2. Magnetic clutch *e.g.* Magnetorque in P & H dipper shovels.
   (Described in the next chapter).

The earthmoving equipment which has to work under severe conditions is often equipped with clutches which are internal expanding bands or shoes and resemble the band or shoe brakes used on some mining haulages. The main difference between the two is that the mining-haulage band brakes are external to the drum and are operated to slow down or stop a revolving drum whereas in a clutch system, the band or shoe is internal to the drum, rotates with the driving member and is engaged to set in motion another drum (driven member.) Also see under "Industrial wheel tractor."

Gear Box

The function of the gear box is to enable the operator to select the proper speed and power to meet changing driving conditions. It also provides for reverse driving of the truck and enables engine to be used as a brake on steep down gradients. Fig. 11 shows the manner of varying speeds. The jack shaft (also called the input or primary shaft) is the "driven"

shaft of the clutch and the mainshaft is the one connected to the propeller shaft. The small gear wheel on the jack shaft (pinion), and the large gear wheel on the layshaft are always meshed as shown in the fig. 11. The gear wheels on the mainshaft are free to slide on it and each of it can be engaged with a corresponding gear wheel on the layshaft. The ratio between the teeth of these two wheels would determine the speed of the main shaft. In common practice, there are 3 or more gear wheels with varying number of teeth on the layshaft

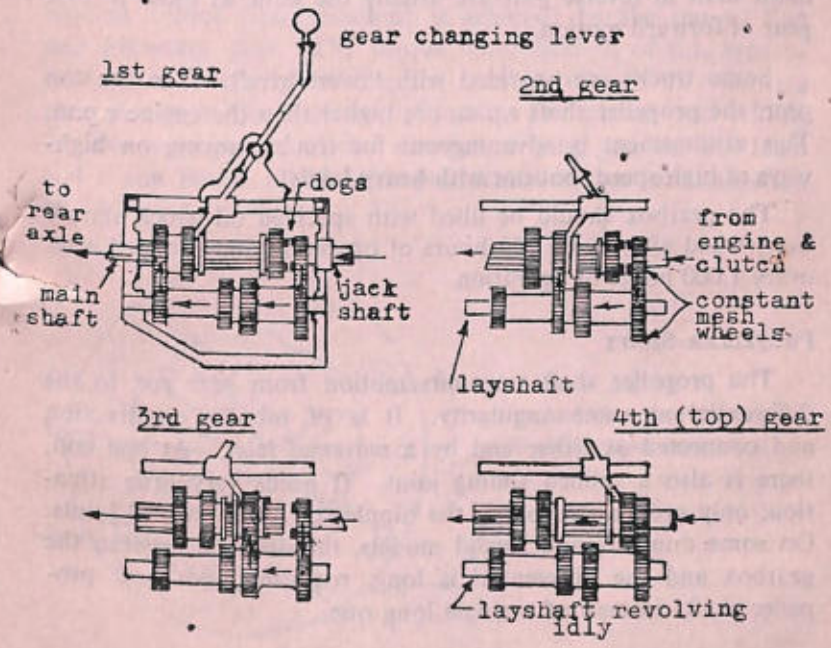

Fig. 11. A four-speed gear box

firmly attached to it and corresponding mating gear wheels on the main shaft which are free to slide along the splines so that at one time one wheel on the lay shaft and the corresponding wheel on the main shaft are engaged (except in top gear or reverse gear). Fig. 11 shows position of different gear wheels in a four-speed gearbox in the first, second, third, and fourth (also called top) gear. The arrows indicate the manner in which motion is transmitted from the jack shaft to the main shaft.

In the top gear the engine r.p.m. and the propeller shaft r.p.m. are the same as they are directly engaged by the dogs, and no gear wheels of the main shaft are engaged, the layshaft and its gear wheels revolving idly. For neutral running of the engine neither the dogs nor any of the gears of the main shaft are engaged and the constant mesh wheels run idly.

For reverse gear, a fourth shaft fitted with a gearwheel is mounted inside the gear box. This extra wheel is interposed between the lay shaft and the third motion shaft. The r.p.m. of main shaft in reverse gear are usually the same as those of first gear of forward motion.

Some trucks are provided with "over drive" *i.e.* in the top gear, the propeller shaft r.p.m. are higher than the engine r.p.m. This arrangement is advantageous for trucks moving on highways at high speed, but not with heavy loads.

The gearbox should be filled with specified oil which should be checked after every 100 hours of operation and changed after every 1,000 hours of operation.

Propeller Shaft

The propeller shaft transmits motion from gear gox to the differential at some angularity. It is of tubular construction and connected at either end by a *universal joint*. At one end, there is also a splined sliding joint. It needs very little attention, only greasing or oiling the nipples on the universal joints. On some dumpers and Euclid models, the distance between the gearbox and the differential is long, requiring two short propeller shafts instead of a single long one.

Differential

The differential performs three functions :

1. It transmits motion of the propeller shaft, through stepdown gears to the axle shafts. The speed reduction is usually in the ratio 5 to 1, resulting in corresponding increase in torque.

2. It turns the direction of drive through a right angle through the medium of a pair of bevel gears called a crown wheel and pinion.

3. Where a curve has to be negotiated the outer wheel has to move slightly faster than the inner wheel and this is achieved by the differential, working on the principle of a balanced beam.

From the differential, the motion is carried to the rear wheels on either side. The axle on either side is called half axle shaft. Normally the differential needs no attention except renewing its lubricating oil during the normal 250 hrs. servicing. Revolutions of the half axle shaft and of the rear wheel are the same. But in Euclids a further speed reduction (and consequent torque multiplication) is achieved by the use of Sun and Planetary gear. The torque multiplication of this type of gear set is found by dividing the number of teeth in the ring gear by the number of teeth in the sun pinion and adding 1. For example, in Euclid dumper a ring gear of 30 teeth and a sun pinion of 10 teeth produces a torque multiplication of $\frac{30}{10}+1=4$.

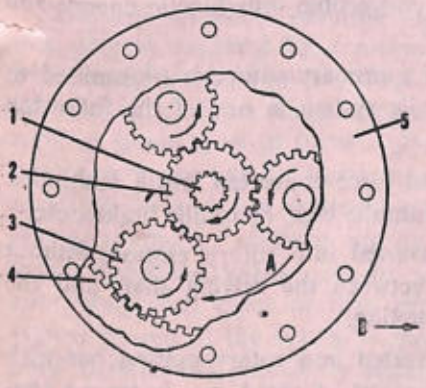

Fig. 12.
Sun and planetary gear on Euclid dumper rear wheel

The engine power is transmitted to the rear wheels which are sometimes referred to as drive wheels. The front wheels are the rolling wheels. Where a four-wheel drive is provided as in jeeps, a front differential is used to transmit torque from the engine to front wheels also. The front differential can be engaged whenever required, *e.g.* for climbing steep gradients; driving on sand, mud or loose ground. The wear on tyres is heavy when four-wheel drive is used for long and it should therefore be disengaged immediately after its purpose is over. The fuel consumption also is high with the four-wheel drive.

In the case of shovels, tractors, etc. which move on crawler chains the engine torque is transmitted to rear sprocket wheels which are known as bull wheels and the front wheels are known as idler wheels.

3. TRANSMISSION (AUTOMATIC OR POWER SHIFT)

The automatic or power shift transmission uses hydraulic fluid (an oil) and before we describe it, it is essential to understand how force or power is transmitted in a hydraulic system, so commonly used on earthmoving equipment.

The principle governing the transmission of force or power by hydraulic system is based on the following facts :

1. Liquids are incompressible except under extreme pressures of laboratory experiments.
2. A liquid in a confined space exerts the same pressure on all parts of the container in contact with it.
3. Pressure energy is convertible into kinetic energy and vice versa.

The power or force from a primary source is transmitted to another member in a hydraulic system in one of the following ways :

(a) Mechanically applied force converted into a multiplied mechanical force, *e.g.* in a hydraulic jack, hydraulic brakes, etc.

(b) Rotary motion converted into rotary motion without any mechanical connection between the driving shaft and the driven shaft, *e.g.* in a fluid coupling.

(c) Rotary motion converted into rotary motion but with multiplication of torque at the cost of speed, *e.g.* in torque converters.

(b) and (c) are covered by the term "fluid drives".

(d) Rotary motion converted into force causing linear movements, *e.g.*

i. hydraulic rams on bulldozers to manipulate the dozer blade, on the dumpers to tip the body, on tractor shovels for manipulation of the bucket in various positions, etc.

ii. Hydraulically boosted steering on heavy duty dumpers.

iii. Down pressure for ripper tooth.

(e) Force converted into rotary motion, *e.g.* on the Drill-master for dust collector blower.

The hydraulic jack is a very familiar example of transmission of power by hydraulic system. The manual effort is multiplied due to the (i) mechanical leverage, and (ii) the different sized pistons or rams, the smaller dia. piston where the manual effort is applied and larger dia. piston where the load has to be lifted. In the hydraulic lifts, used at car service stations, compressed air forces the smaller size piston downwards and the force is transmitted through hydraulic medium to the large size piston (or lift).

This arrangement of using compressed air to boost the hydraulic pressure in a hydraulic system is usually spoken of as air-over-hydraulic and finds an application in some earth-moving equipment, *e.g.* on P & H shovel for the hoist drum and also for digging drum.

The manual effort required for operation of the hydraulic jack may be replaced by a reciprocating pump which operates with more force and speed where heavy loads are encountered. On moving machines for construction and earthmoving purposes a rotary pump (gear or turbine type) is used in place of the reciprocating pump and its power availed of to transmit force. A hydraulic system for converting kinetic energy into pressure energy ordinarily includes an oil reservoir, a pump, a relief valve, directional valves, one or more cylinders or motors and connecting steel pipes or high pressure rubber hose pipes. A typical lay-out of the system is shown in figure 13. It is a standard practice to provide a separate hydraulic pump and oil reservoir for each control. A common pump and oil reservoir may mean inadequate pressure for each controlling system if two or more hydraulic controls are operated simultaneously. Moreover the control requiring lower pressure will operate first. Some hydraulic pumps work all the time an engine is running, *e.g.*, for the steering booster, or the front end pump in a tractor ; others can be pressed into service whenever desired, *e.g.*, pump for the tipping gear of a dumper. Hydraulic systems usually have paper type filter in the oil reservoir or in the oil pipe connected to the pump, and a filter on

the air breather through which air enters and leaves the reservoir as the oil level rises and falls. The oil used is generally of S.A.E. 10W viscocity though some companies have achieved equally good results by using used-up engine lub. oils of diesel engines. It is important to keep oil at the proper level. Too little will allow the system to suck air and perform jerkily ; too much may cause oil to spill out or result in building of damaging pressure, *e.g.* in a tractor shovel when lowering a loaded bucket.

Common replaceable parts are the O-rings (oil seals) on the piston rods and the hose pipes. Dust, particularly of the abrasive nature, causes scratches on the piston rod (ram) resulting into leakage of oil and this should be guarded against.

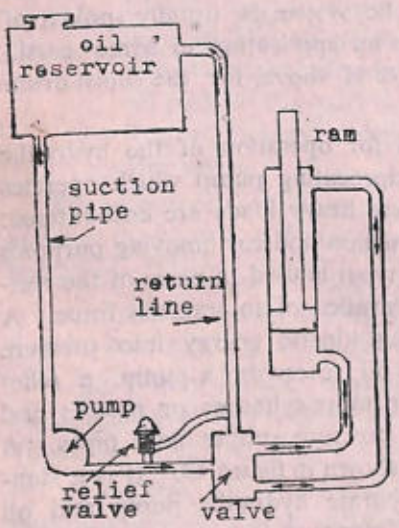

Fig. 13.
Hydraulic system for power transmission

Hydraulic system adopted for fluid drives is employed for automatic transmission and is generally used on engines having speeds of 1,800 r.p.m. or more. Automatic or power shift transmission is one in which the full engine power is transmitted to the drive wheels irrespective of load and without any manual shifting of levers by the operator.

Automatic Transmission (Power shift transmission)

On some of the latest high h.p. cars and some earth moving equipment like dipper shovels, draglines, Caterpillar Trax-

cavator, Allis Chalmers HD 21 shovel, etc., the transmission of engine torque to the vehicle (or some other driven parts) is automatic and without the type of the mechanical gearbox described earlier. The automatic transmission is also known as power-shift transmission, and is usually teamed with torque converters which absorb jerks that would otherwise result by sudden changes of gear ratio. Automatic transmission employs (i) a fluid flywheel (fluid coupling) or a torque converter as the first stage in the transmission of power from the engine to the rear wheels or bull wheels, (ii) an epicyclic gearbox, and (iii) some form of hydraulic valve mechanism controlled by throttle opening.

FLUID FLYWHEEL

A fluid flywheel works like a coupling which transmits the rotation of the driving member (flywheel) to the driven member through the medium of a fluid and not through solid friction between the driving and driven members. There is no mechanical connection between the driving shaft and the driven shaft. Its advantages are :

(1) The drive is engaged without any shocks which normally exist when connecting a revolving engine with a stationary (or much slower) vehicle.

(2) The engagement is not dependent upon driver's skill but is controlled by the accelerator only.

(3) It works without practically any slippage in the high speed range.

Its principle of operation may be found in the familiar water wheel. When water impinges on the vanes of the wheel, the latter turns its shaft and the motion is utilised to drive other machinery. Power available from the shaft can be greatly increased if water impinges on the vanes under a high pressure or if the number of jets and corresponding vanes is increased. This is the principle of working of a fluid flywheel. (Fig. 14).

An impeller with radiating vanes is attached to the shaft of the gear box and it runs in a sealed housing (stator) connected to the crankshaft. The stator is also equipped with radial vanes inside, and is filled with fluid. At start when the

crankshaft revolves, beyond a certain speed the oil in the stator is pushed towards the periphery by the centrifugal force. As the driven member is stationary the fluid will flow back from the periphery through its channels to the centre, to be again

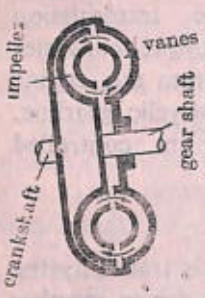

Fig. 14.
Fluid flywheel (Schematic)

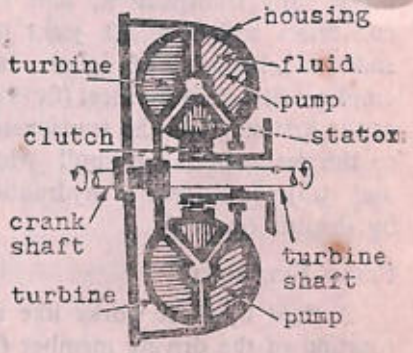

Fig. 15.
Three-element torque converter

thrown out through the channels of the driving member. As a result of this the vanes of the driven member are dragged along. The more the speed of the crankshaft the higher is the dragging force acting on the turbine (impeller).

Torque Converter

A torque converter is a device used to increase the torque while reducing the speed. In this respect it performs a function similar to that of a gear box, but whereas a gear box provides only a small number of fixed ratios the torque converter provides a continuous variation of ratios from the lowest to the highest upto a certain limit. A torque converter consists of three principal parts, viz.

(a) The driving element or impeller or pump which is connected to the engine shaft. It has a ring of metal blades which rotate in the fluid with which the converter is filled.

(b) The driven element or rotor which is connected to the propeller shaft. The rotor is also called *turbine*.

(c) The fixed element or reaction member which is fixed to the frame. It is the stator or housing containing two stationary sets of blades. (Fig. 15).

The action of the converter is somewhat similar to that of a fluid coupling, described earlier, and as follows :

As the impeller is driven by the engine, the fluid flung out of the periphery of the impeller by centrifugal force impinges on the first ring of blades of the rotor and is deflected by those blades. The fluid having given initial rotation to the turbine is then redirected by the first set of reaction blades in the stator. The fluid then impinges against the second ring of turbine blades and in this manner it is directed and re-directed through all three sets of turbine blades.

Under normal running conditions the action of the impeller causes the turbine blades to turn almost freely and the fluid passes through the converter easily and quickly, striking each blade on a very slight angle. But when a load is encountered, the turbine slows down since it is directly connected to the propeller shaft, and the fluid strikes the turbine blades at a sharper angle. As the force of the fluid is relayed from the first to the second, and from the second to the third ring of turbine blades, the output torque accumulates and it is increased upto as much as six times the input torque in a three stage converter.

The twisting force that accumulates as the fluid is directed and redirected from one turbine blade to another is the output torque of the converter. It becomes greater with each increase in the angle at which the fluid successively strikes each set of blades in the turbine and housing. Thus the torque converter selects the proper output torque for any load. Under heavy load, or at low engine speed, the converter slips and the heat generated is dissipated by circulation through a radiator. In operation the engine runs essentially at constant speed driving the impeller also at constant speed whereas the turbine runs fast or slow just like an electric D.C. series motor adjusting its speed automatically to the load. The turbine shaft speed varies inversely as the torque so that with increase in torque, the turbine shaft speed decreases until a limit is reached beyond which the shaft simply stalls, without causing any overload of the engine.

In some ways a torque converter resembles a fluid coupling. In both of them oil is accelerated by rotating vanes in the

impeller and is thrown into a vaned driven member which therefore rotates by the reaction of the moving oil. The mechanical energy of the driving shaft is converted into hydraulic energy and back to mechanical energy of the driven shaft. The two devices, however, serve different purposes. The fluid coupling transfers the engine torque to the gearbox, after certain speed, whereas the torque converter multiplies the engine torque in a ratio of nearly 2 to 2.5 : 1 and in extreme cases, 6 : 1, as in a three-stage converter. Some torque converters have twin turbines with rotating housing *e.g.* on Allis Chalmers tractor loaders with pneumatic tyres of model TL 545, TL 645, TL 745. At the time of start only one of the turbines which is designed for high torque and low speed, becomes effective, whereas during the run, the other turbine designed for high speed and low torque, comes into play automatically without any attention from the operator.

Torque converters are not used for small engines and engines of H.P. from 40 to 1,000 are considered suitable for their utilisation. Torque converters have a reasonably good efficiency over only a narrow range of rotor speeds. The loss of efficiency arises mainly due to slippage inside the fluid drive and results in heat which has to be dissipated by an outside cooling system, commonly a radiator. A good operating range for a converter fluid is between 83°C and 104°C. They are therefore, used on tractor shovels, dipper shovels, heavy duty trucks and other high powered equipment.

4. STEERING & SUSPENSION

Steering

The front wheels in a truck are the rolling wheels and they enable the driver to steer the truck in the desired direction. The arrangement of steering essentially consists in providing a means to swivel the front wheels, each about an axis close to its hub, and of linking it to a steering wheel. The turning of the steering wheel operates a worm gear which steers the front wheels through rods and linkages and is quite familiar to a car driver. The manual effort which a driver has to exert in steering a large vehicle is heavy and tiring. It is, therefore, a

common practice to assist the manual effort by hydraulic pressure in heavy vehicles of above 6-ton carrying capacity. Such steering is said to have a "hydraulic booster" or said to be power assisted and the arrangement is shown in Fig. 16. A gear type pump driven by the engine, draws oil from the verti-

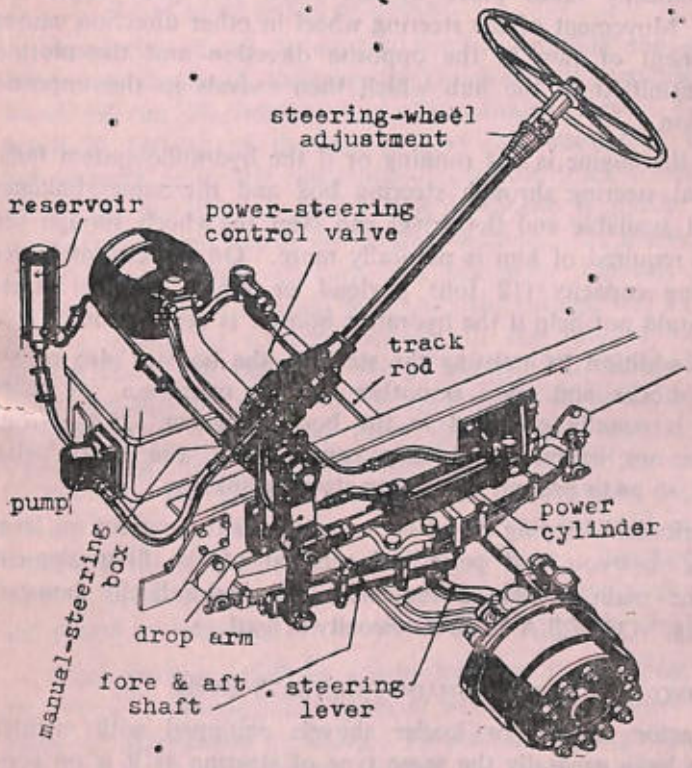

Fig. 16. Hydraulically boosted steering on a Thornycroft truck.

cal reservoir and delivers it under pressure to a control valve, housed in the manual steering box and column. The hydraulic system is of the continuous flow type and when the steering wheel is not turned by manual effort, the oil passes straight through the control valve back to the reservoir. If, however, the steering wheel is turned by manual effort, a spring fitted in the control valve is compressed and the valve directs oil under pressure to one end of the power cylinder which

is a double acting hydraulic ram. The rear end of the power cylinder is anchored to a frame bracket and the oil forces out a ram connected to the drop-arm. The movement of the drop-arm causes swivelling of one hub to which it is connected by the steering lever. Swivelling of the other hub automatically takes place by the track rod (also called tie rod). Movement of the steering wheel in other direction causes movement of ram in the opposite direction and the motion is transmitted to the hub which then swivels in the opposite direction.

If the engine is not running or if the hydraulic system fails, manual steering through steering box and the same linkages is still available and the driver can steer the wheels, though the effort required of him is naturally more. On vehicles of larger carrying capacity (12 tons payload or more), manual steering would not help if the hydraulic booster is not working.

In addition to assisting the steering, the booster also resists road shocks and gives smoother steering operation. A relief valve is usually provided in the booster system. If the front wheels are jammed and they cannot turn, the relief valve opens, so as to prevent damage to steering linkages.

Periodical topping up of oil to maintain the correct oil level in the reservoir and periodical renewal of the filter element are the main points to remember for hydraulically boosted steering. Oil of S.A.E. 10W viscocity is used.

Steering on Crawler-Chain Tractor Shovels

Tractor shovels or loader shovels equipped with crawler chains have generally the same type of steering as it is on conventional crawler tractors. Each crawler chain can operate independently and has a steering clutch. Its lever is hand operated and when in the forward position, engages the crawler ; if it is pulled back, the track on that side will run idle and the other will continue to drive. On heavier models the steering clutch controls are assisted by hydraulic pressure which minimises the effort required to release the clutches during operation of the machine. The crawler track can be braked by a brake pedal which is known as *steering brake* and each track has an independent brake pedal which can be kept locked

when fully braked. In general five factors are involved in steering such machines, viz., effect of :

1. Steering clutches.
2. Steering brakes.
3. Foot work.
4. Gradient.
5. Load being pulled or pushed.

When the shovel is moving forward, pulling the steering clutch lever back will disengage the clutch of that particular track which will run idle, but the other will continue to drive. This will result in turning of the shovel, and the amount of turning will depend upon the resistance of the idle track to dragging caused by the driving track. The resistance is partly by internal friction and partly by external friction. On smooth hard surface the machine will turn very gradually. On soft ground which causes more external resistance, the turning will be sharp. A tree stump or a large boulder in the way of the non driving track will cause an abrupt swing. The same effect of resistance can be produced by application of brake, and the turning effect will depend upon the pressure on the brake. The steering effect produced under various conditions is shown in Fig. 17. For an even turning both the controls, *i.e.* clutch release lever and the brake pedal, should be handled smoothly otherwise the turn will be in a series of jerks. Just before the turn is completed, release the brake, then engage the steering clutch by releasing the control lever quickly but gently.

When moving uphill the gravity increases the tendency of the free track to remain idle ; in other words the resistance of that track to movement increases. Therefore, when going uphill, less braking is required to produce the steering effect. When going down a steep slope, the machine has a tendency to roll faster than the engine drives it. The free track will move faster than the driving track which is under a sort of "braking" effect as it is engaged with the engine. The result is, steering of the machine towards the driving track. For example, to turn to the right when going down a steep slope, release the steering clutch on the left, but do not apply the brake. When going downhill, place the range selector lever in low and the speed selector lever in first : *Never place the speed selector lever in*

neutral for travelling downhill. The speed selector lever has 3 positions, neutral, 1 and 2 for forward as well as reverse travel.

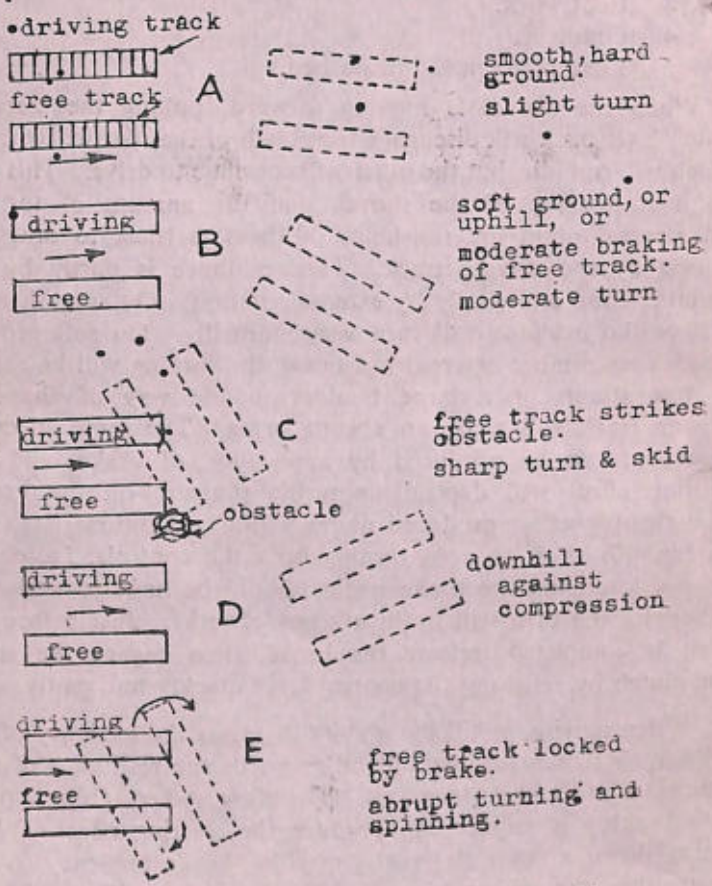

Fig. 17. Steering of a crawler mounted machine

When both the steering clutches are engaged, application of either steering brake can slow down or stop the machine, but will not produce any steering effect. The pedals must be in the fully released position during the normal operation of the machine. Keep the feet off the pedals except when it is necessary to use the brakes.

Suspension

The chassis frame is connected to the road wheels through the medium of springs and axles. The term suspension covers springs, and shockabsorbers and is closely linked with steering, which would not be smooth if the suspension system is defective. The familiar leaf spring is commonly used on heavy duty trucks and dumpers. All the spring leaves are held together by a centre bolt and the assembly is attached to the axle by U bolts and to the chassis by silencebloc type rubber shackle bushings. Long springs give a softer ride as compared to shorter springs, but tend to oscillate for a longer period, a tendency kept under check by the use of shockabsorbers, which should more aptly be called *dampers*. The longest member in the leaf spring assembly is known as the main leaf. The number of leaves in the spring assembly are sometimes increased to give a better shock dampening effect. In course of use the spring leaves loose their shape and have to be tempered and reshaped by blacksmiths specialised in the job.

Normal maintenance of the springs consists in occasional lubrication between the leaves and is carried out by jacking up the vehicle so that the leaves separate to some extent.

Side play in the shackles should be avoided as such play adversely affects steering performance.

A vehicle fitted with axles and leaf springs may lean over to one side when either of the wheels experiences a bump which is imparted to the other wheel also. This affects the smoothness of steering and the difficulty is overcome by eliminating the axle connecting the two wheels and providing an independent spring for each wheel. The wheels are then said to have independent suspension. Such independent suspension, though gaining ground for cars, is generally not adopted for heavy duty trucks. Independent suspension uses coil springs for each wheel and such coil springs are easier to maintain than leaf springs. Torsion bar and coil springs are now coming in vogue for cars.

Shockabsorbers nowadays are usually of the hydraulic type. In principle a hydraulic shockabsorber consists of a piston or a rotating vane moving in a cylinder or case during the

rebound of the spring. The oil in the cylinder is forced through small holes or valves and this has a damping effect. In hydraulic shockabsorbers, special oil has to be used to top up from time to time. If the shockabsorber is allowed to run dry, air may enter it and this air has to be expelled when refilling.

Shockabsorbers are single acting or double acting. Single acting shockabsorbers are effective during rebound of the springs.

Telescopic shockabsorbers are getting popular now-a-days. They hold more oil, facilitate sealing and dispense with special operating rods. Normally they are not repaired and have to be discarded after some mileage.

5. BRAKES

A brake is a mechanical device for retarding or stopping the motion of a vehicle. Its operation is dependent on the friction created between the rotating brake drum and the brake lining mounted on stationary brake shoes. The kinetic energy of the moving brake drum is converted into heat energy by the friction and this heat is dissipated. If the brake linings are so placed that the revolving drum is outside and the brakes, when applied, expand to grip the drum, the brake is said to be "internal expanding". If the brakes are outside the revolving drum and squeeze inward for braking effect, the brakes are "external contracting". The brakes used on cars and trucks are internal expanding. The typical brake assembly includes brake shoes on which brake linings are mounted, and a spring to return the shoes to their original position.

Hydraulic Brakes

In the system of hydraulic brakes, a system commonly adopted on cars and smaller size trucks, the brake shoes are forced to expand, on application of brake, by hydraulic pressure. The braking system (Fig. 18) consists of a cylinder called master cylinder, containing special oil, connected by pipes to small cylinders at the wheels (called wheel cylinders). The master cylinder is a cylinder in which a piston, connected to the brake pedal, can move to and fro and it also includes a small oil reservoir. On the front wheels there are two wheel cylinders for

each wheel, and for the rear wheels, one wheel cylinder for each wheel. A wheel cylinder carries a small piston which is connected with a brake shoe. The master cylinder, pipe lines and the wheel cylinders are full of oil all the time. When the driver presses the brake pedal, the piston in the master cylinder exerts pressure on the oil in the latter and this pressure is transmitted through pipe lines to the pistons in the wheels cylinders. The pistons are forced out under pressure and this causes the brake shoes to expand against the force of the springs to contact the brake drums. When the driver releases the pres-

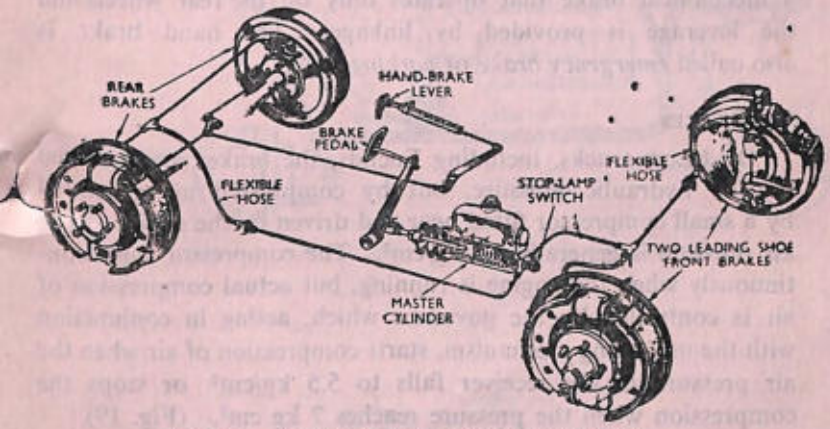

Fig. 18. Hydraulic brake of tyred equipment

sure from the brake pedal, the brake shoes come back to their original position due to spring tension, the pistons in the wheel cylinders travel back to their normal position and in so doing, some of the oil is pushed back to master cylinder. The last connection of the piping to the wheel cylinders on front wheels is by hose pipes which need to be changed in course of use as they become old and develope leaks. The brake linings also wear out during use but arrangements provided to adjust the brakes can, compensate for some weaf and when the final limit is reached, the brake linings have to be renewed. These brake linings are made principally of asbestos and are riveted on brake shoes. They are effective when dry ; oil or grease on them reduces their braking capacity appreciably. If there is a small leak anywhere in the piping

system, it may be necessary to "pump" the brake pedal twice or thrice before the brake has its effect. Presence of air anywhere in the oil renders the brakes ineffective and to bleed the air from the system is essential to make them useful. The rubber components of a hydraulic braking system should never be washed with kerosene, petrol, carbon tetrachloride or any mineral oil, otherwise they will soften and swell and become useless. Only clean hydraulic brake fluid should be used for cleaning.

A hand brake is provided on all cars and trucks. It is a mechanical brake that operates only on the rear wheels and the leverage is provided by linkages. The hand brake is also called *emergency brake* or *parking brake*.

Air Brakes

On heavy trucks, including Euclids, the brakes are operated not by hydraulic pressure, but by compressed air generated by a small compressor fitted near and driven by the engine. The air pressure is generally 6.5 kg/cm^2. The compressor runs continuously when the engine is running, but actual compression of air is controlled by the governor which, acting in conjunction with the unloading mechanism, starts compression of air when the air pressure in the receiver falls to 5.5 kg/cm^2 or stops the compression when the pressure reaches 7 kg/cm^2. (Fig. 19).

A valve operated by the conventional brake pedal, and known as the *treadle valve* controls the movement of an inlet valve and exhaust valve which, in turn, control the air pressure in the brake cylinder or brake chamber. (Fig. 19a). In the brake chamber the air pressure acts against a diaphragm that moves rods and levers which apply the brakes. The farther the driver presses the brake pedal, the higher the air pressure delivered to the brake chambers and the stronger the braking action. Unlike the hydraulic brake, air brake cannot be applied by "pumping" the brake pedal which should be applied only once to produce the braking effect. It is essential for the driver of an air-braked vehicle not to start driving unless the minimum pressure specified in operating instructions is reached. When garaging the vehicle at the end of the day, air and water from the receiver should be drained.

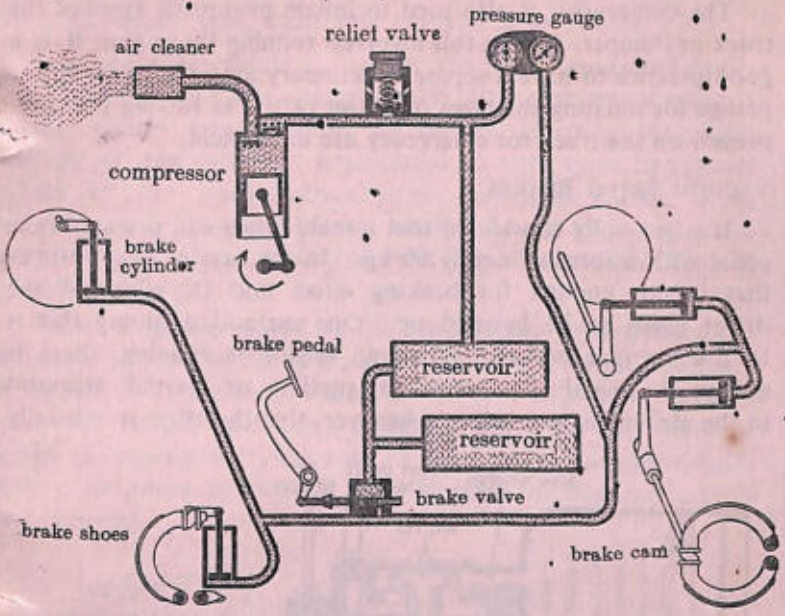

Fig. 19. Compressed air brake on tyred equipment.

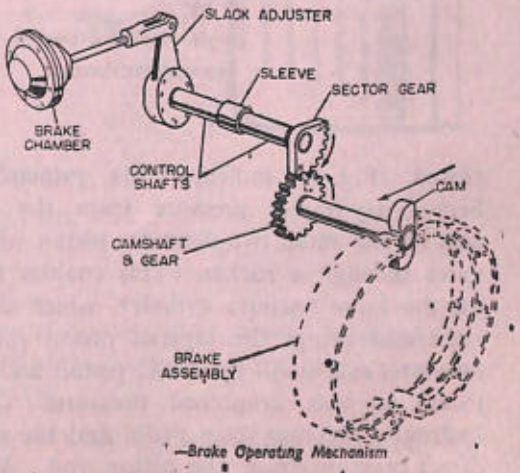

Fig. 19a. Brake operating mechanism on Euclid dumper

The compressor is also used to inflate pneumatic tyres of the truck or dumper, but as this involves running the engine, it is a good practice to have a separate stationary air compressor in the garage for inflating the tyres of a fleet of trucks leaving the compressor on the truck for emergency use in the field.

Vacuum Servo Brakes

It is generally considered that a male driver can press a brake pedal with a force of nearly 60 kg. In the case of heavy trucks that is not enough for braking effect and the effort of the driver needs to be boosted up. One method of doing this is to use vacuum brakes. When an engine is running, there is always a negative pressure (a suction or partial vacuum) in the air induction pipe whenever the throttle is partially

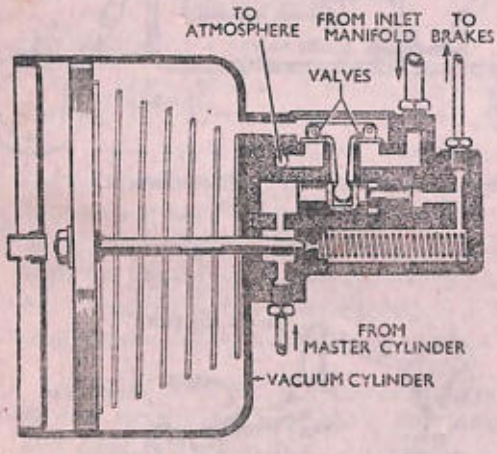

Fig. 20.
Vacuum servo brake.

closed. Fig. 20 indicates the principle of Girling Vacuum Servo. Hydraulic pressure from the brake master cylinder acts on the small two-diameter piston which operates a vacuum valve through a rocker. This enables manifold suction to act on the large vacuum cylinder, which draws the piston to the right and brings the tapered piston rod into contact with an aperture in a small hydraulic piston and the piston is then subjected to two combined pressures ; one resulting from the hydraulic pressure from pedal and the other resulting from the direct pressure from the piston rod. Vacuum servo brake is not as powerful as the compressed air brake.

6. TRACTION. CRAWLER CHAINS AND TYRES

A machine or vehicle moving by the power generated by its own engine or motor is equipped with either pneumatic tyres or crawler chains on practically all the earthmoving equipment and the driving torque is transmitted to such tyres or crawler chains by the transmission system described earlier. Four wheel drive is uncommon for trucks though it is the standard practice on jeeps, tractor shovels like Michigan 85, 125, 175, and others.

When a vehicle moves on the road it has to overcome the resistance offered by the road, and if going upgradient, the force of gravity. The *rolling resistance* is the resistance between tyre (or crawler) and level ground that must be overcome before the tyre (crawler) can roll. Table 1 shows the approximate rolling resistance of various road surfaces to pneumatic tyres and crawlers.

TABLE 1

Rolling resistance

Type of surface	Pull in kg per 1,000 kg of gross weight.	
	Lower pressure tyres	Crawler chains
Smooth concrete	18	28
Good macadem	30	35
Earth roads, dry, dusty	40	43
Unploughed earth	60	55
Earth road, rutted, uneven	90	70
Loose sand and gravel	110	85
Construction haul roads or roads in loose soil	145	112

It will be seen that the advantage of pneumatic tyres recedes rapidly as the ground conditions worsen and for rough roads a crawler unit is at an advantage.

A resistance of 9 kg per 1,000 kg is roughly equivalent to that offered by upgradient of 1%, i.e. road rising 1 m vertical through 100 m horizontal.

Crawler Chains

In tractor shovels, bulldozers, etc. the crawler chain consists of grouser shoes (Fig. 21) or plates bolted on to a sprocket chain.

For heavy duty work as in a dipper shovel, dragline etc. the crawler chain is made of a number of shoes or plates, drilled at their ends so that they can be fastened together by pins. Wedge shaped projections cast into the inner surfaces of the shoes

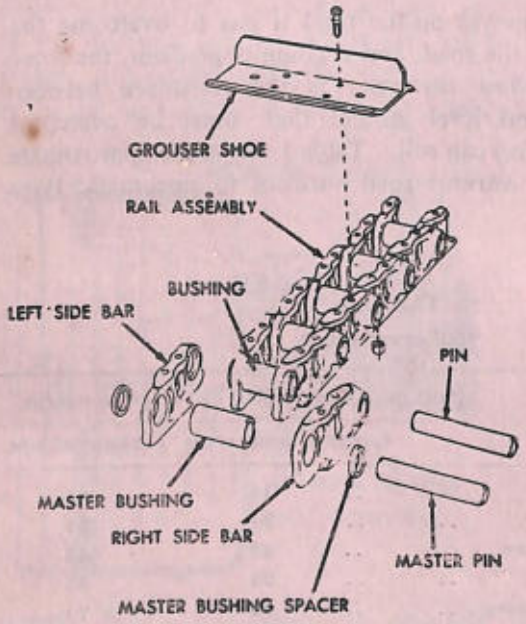

Fig. 21.
Crawler chain.

provide a grip for the sprocket or bull-wheel and help to keep the track centred on the idler and the rollers. Wider tracks are suitable for soft ground, but their use increases the effort for steering. Long tracks increase stability and are desirable for machines handling heavy loads or having long booms like stripper shovels, draglines, etc.

The axle which turns the sprocket or bull wheel is known as the live axle. The idlers are smooth flanged wheels similar in size to the drive sprockets and are mounted on yokes backed by heavy springs. The track frame extending from the bull-

wheels to the idler, has two or three support rollers mounted on it to prevent excessive sagging of the upper track chain. The frame rests on track rollers.

The crawler chain should be under proper tension and not too loose or too tight. A loose chain may come off the wheels ; a tight chain will cause heavy wear of the sprocket teeth, bushings, etc. The idler is held in position by a sliding yoke backed by a spring and the spring tension is adjusted by an adjustment screw which is threaded into the idler yoke and turns freely on the spring *e.g.* in Allis Chalmers HD 11 G tractor shovel.

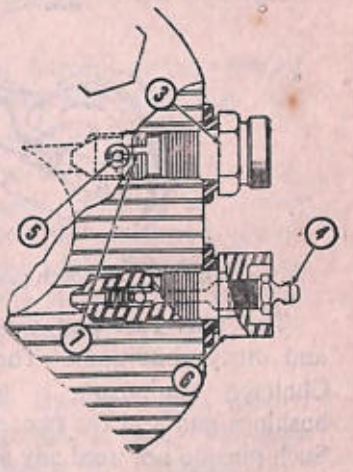

Fig. 22.
Crawler chain tension adjustment arrangement by grease pressure on a Traxcavator

(3) Relief valve,
(4) Fitting,
(5) Ventholes,
(6) Ball check assembly,
(7) Slot.

On some models, *e.g.* on Caterpillar 900 models, the track tension is adjusted hydraulically. Hydraulic track adjusters usually consist of a piston in a tight cylinder filled with grease. The piston rod is an extension of the idler yoke. To tighten the track, force grease with a grease gun into a nipple on the cylinder. This forces the idler outwards. To slacken a tight track, bleed grease of the cylinder through a fitting on the relief valve. The arrangement (Fig. 22) is much simpler than the earlier one of adjusting spring tension by an adjustment screw. Another method, adopted on Euclid C6 bulldozer, is the use of a cylinder filled with nitrogen gas under a pressure of 100 kg/cm^2 (fig. 23). The cylinder replaces the recoil spring mentioned for Allis Chalmers shovel track adjustment. The leakage of gas necessitates re-filling the cylinder in course of use but oxygen

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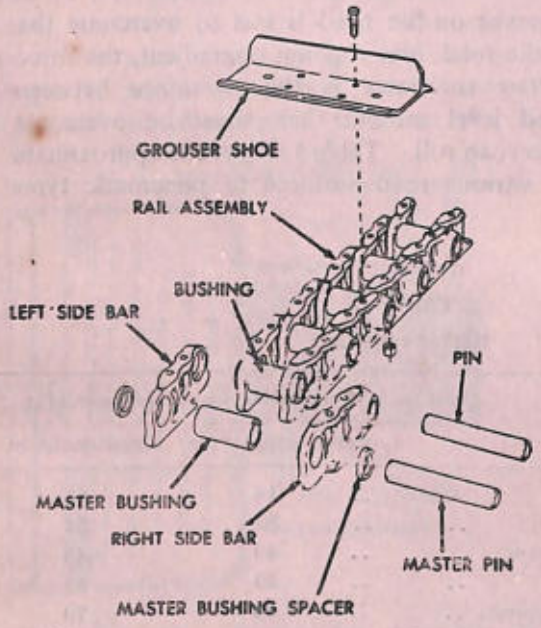

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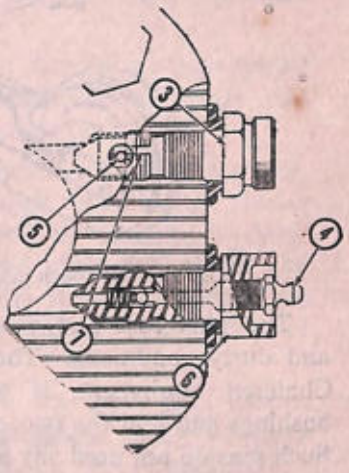

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should never be used as a substitute for nitrogen. In one instance the use of oxygen resulted in bursting of the cylinder due to the high pressure resulting from high temperature during the oxidation of grease.

Normal wear is on the shoes or plates, bushing and the sprocket wheel. The bushing, when one side is worn out, can be rotated through 180° ; when the other side is also worn out, replace the bushing or the pin will start wearing.

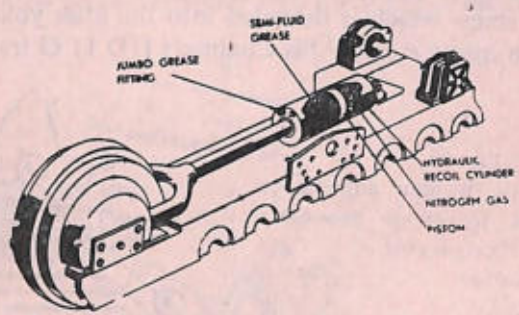

Fig. 23. Nitrogen gas cylinder on Euclid C 6 dozer

The pins rust and wear out due to operation in muddy and dusty conditions. The latest trend, for example, on Allis Chalmers equipment, is to place the pins in well greased bushings and seal the two ends of bushing with neoprene plugs. Such pins do not need any attention for lubrication and last practically the life of the grouser plates. The grouser plates can be built up with hard surfacing welding rods, but help only a slight extension in their life compared with the cost of repairs. They should better be replaced. The sprocket wheel, when worn, should be changed. In some models the sprocket teeth form part of a ring which is attached by studs to the drive wheel like the toothed rim on the flywheel of a motor car.

Machines using crawler chains are not allowed to cross public roads which are damaged by the cleats of grouser plates and the latter should be replaced by plain plates, preferably with rubber soles, for crossing a public road. Speed of crawler chains is slower than that of tyres. Where public roads have to be crossed often, or where the flitting is frequent, it is an advantage to use pneumatic tyred equipment. In the field, the

crawler mounted equipment scores over the tyred one for climbing steep gradients, and has better traction on rough roads and better gripping capacity on bench floor when the machine has to dig hard.

Tyres

Tyres support the whole weight of the vehicle and on most of the vehicles, the rear tyres, in addition, transmit the driving torque to it, as already stated. When the brakes are applied, it is the tyre again which transmits the whole of the retarding force. The construction of tubed tyres, so familiar to car and truck owners, is as shown in *fig.* 24.

1. BEADS. These are formed of bundles of high tensile wire and act as the tyre anchor. The plies (see below) are tied in to the bead to preserve the shape and fit on the rim.

2. PLIES. Successive layers of cord make up the casing. This is the framework to which the tread and sidewall rubber compound are applied. Each cord is adhesive-solution-dipped and surrounded by a compound of resilient rubber and each ply layer is cushioned from the next by rubber.These plies contain the inflation pressure, support the load and cushion the shock of impacts.

3. BREAKER STRIPS or tread plies are extra plies located under the tread. They act as shock absorbers and give extra resistance to blows from rough surface or obstacles.

4. SIDEWALLS must flex and bend without cracking, and the rubber compound of which they are made must combine this quality with resistance to damage.

5. TREAD. This is the contact surface of the tyre and the actual power-transmitting surface of a power-driven wheel. It must provide adequate traction and give maximum resistance to wear and damage.

The tube contains the air under pressure. The flap is placed between the rim and the tube to protect the latter from the rim or from damage that might be caused by excessive brake drum heat.

The tyres used for heavy earth moving equipment are of the pneumatic type, that is, air under pressure is held inside the tube or inside the tyre if it is of tubeless type Most of the

tyres are the conventional tubed type in which an inner rubber tube of a size suitable to fit the tyre is placed within and it holds air under pressure. In the other type, known as the "tubeless tyre", no rubber tube is used, but the tyre has an air-tight inner layer on the casing that seals in the pressure. The

Fig. 24. Pneumatic tyre construction *Courtesy of Good Year India Ltd.*

rim on which the tyre has to be mounted has also to be airtight. Where a tubed tyre is used the tube has the valve through which air is filled, but in the case of a tubeless tyre, the valve is fitted to the rim. The rim used for a tubeless tyre can be used for a tubed tyre also by drilling a hole for the valve but not

vice versa. The rim for a tubeless tyre is 10 mm wider than that for a tubed type. A damaged or out-of-shape rim of a tubeless tyre will not serve the purpose of retaining air pressure and therefore needs more careful attention than the rim of a tubed tyre. In appearance the tubed and the tubeless tyres are alike. During driving if a tubed tyre develops a puncture, the air leaks away fast accompanied by a hissing sound and the vehicle must stop. With a tubeless tyre, the air will be blown out very slowly so that the vehicle can reach its destination, if not far off, where the punctured tyre can be repaired. This is possible because the shape and thickness of the cover tread is such that the inner side of it is in compression and therefore closes a hole whereas in a tubed tyre, the tube is stretched when inflated so that a small hole is self-enlarging. The tubeless tyres are somewhat cooler than the tubed type during running, a factor particularly important in high ambient temp. and on long hauls. Most of the garages have no facilities to repair a puncture on a tubeless tyre and truck owners therefore prefer to go in for tubed tyres.

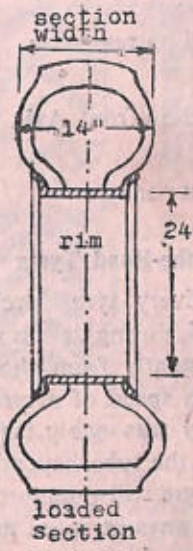

Fig. 25. Denoting tyre size 14.00-24;

Size of a tyre. Tyres are specified by 2 numbers *e.g.* 14.00—24, for rear tyre of Euclid, or 5.90—15, for Ambassador car. The first number 14.00 represents the section width in inches (*fig.* 25) of a tyre inflated, but not under load, and the second number, the diameter across the bead in inches. The maximum load which a tyre can carry depends upon its size, ply rating and the maximum inflation pressure recommended by tyre manufacturers. The size and ply rating are always marked on a tyre. The plies are the individual fabric layers in the casing. They are usually woven of cotton, rayon or nylon cords surrounded by rubber. Nylon plies are stronger and the present trend is for them though the tyres are somewhat costlier than those with cotton or rayon plies.

More number of plies indicate a tyre of stronger construction which can therefore carry more air pressure and thus a heavier load. The number of ply rating is an index of tyre strength but does not necessarily represent the number of actual plies of material in the tyre.

LOAD DISTRIBUTION

The distribution of load on tyres when standing on a level surface is calculated as shown below :

If A = distance from front axle to centre of payload

B = distance from rear axle to centre of payload

C = wheel base.

Distribution of Payload per Axle

$$\text{Load on rear Axle} = \frac{A}{C} \times \text{payload.}$$

$$\text{Load on front Axle} = \frac{B}{C} \times \text{payload.}$$

$$\text{Load per tyre} = \frac{\text{Proportion on axle}}{\text{No. of tyres on axle}}$$

Off-the-Road Tyres

Every tyre develops heat, mainly by friction against the road, during its run and by flexing of the sidewalls. It is radiated constantly from the tyre surface and also through the wheel. High speed of a tyre will result in heat which, if not dissipated away reasonably fast, will build up the temperature of the tyre and the tube beyond their vulcanising temperature and the tube or tyre will puncture or burst. Tyres which have to operate on highways at speed are therefore of a different construction than those which are used on vehicles off the highways as such off-the-road tyres are designed to carry heavy loads comparatively slowly over abnormally uneven surfaces for limited distances. Off-the-road tyres have special treads to ensure grip on the road and heavy traction, and their speed should not exceed the maximum recommended by manufacturers, usually 50 km per hour.

The tread pattern on a tyre depends upon whether it is a rolling tyre as on the front wheels or a driving tyre as

on the rear wheels. In general, high, widely separated diagonal cleats in various forms or V-shaped lugs are used for drive tyres and wide but closely spaced bars around the circumference are common on free rolling tyres. Tread patterns which lack cleats or lugs do not provide sufficient traction in soft or loose ground. Each tyre manufacturer has his own tread patterns for the tyres. It should be noted that a tread pattern on an off-the-road tyre which may give a good milage on a rough mine road is not suitable when the road is well maintained and the tyre life available may be actually low.

The main tread patterns on off-the-road tyres are usually as shown in fig. 26 or slight variations thereof.

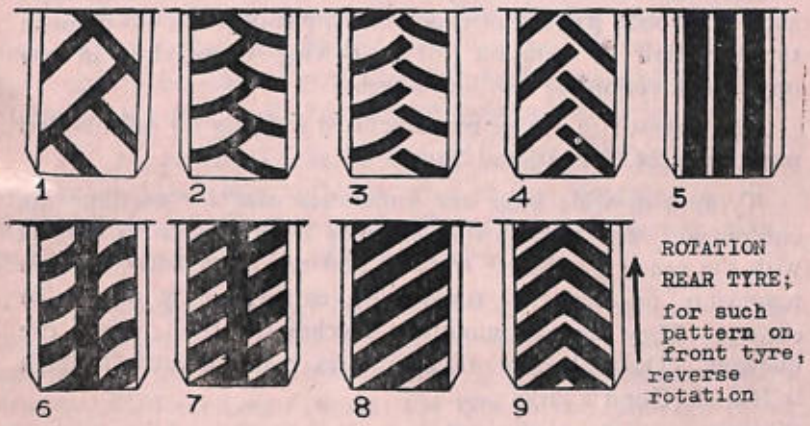

Fig. 26. Tread patterns for off-the-road tyres

Patterns 1, 2 ; *Closed centre* pattern. For tyres *not* engaged mainly on field work. Suitable for operation on shallow soil with a stony sub surface.

Patterns 3, 4 ; *Open centre* pattern for soft ground ; gives better grip.

Pattern 5 ; Circumferential rib suitable for front or free rolling wheels. High centre rib reduces side slip when turning.

Patterns 6, 7 ; For hard surfaces where little traction is required. Lugs on these patterns should be as close as possible.

Pattern 8 ; Diagonal bar design for hard rock conditions. Also suitable for road-type vehicles operating mainly under condi-

tions which require a degree of traction. Not suitable if the vehicle is operating mainly on roads.

Pattern 9 ; Suitable for general earth moving work where traction is not of primary importance. Not suitable for use on road-type vehicles. Note the direction of rotation for drive wheel. The direction of rotation should be opposite if the tyre is fitted on the front wheel.

A tread pattern which can be adopted for the driving as well as the rolling wheel irrespective of the direction of rotation is called *non-directional pattern*. Some tread patterns are called *directional patterns* and the direction of rotation of the driving wheel is marked on the tyre. A mistake in fitting the tyre the wrong way results in poor tyre life and bad results. Fig. 26(9) shows one such pattern in which the appex of V should point in the direction of rotation for a driving wheel, but in the opposite direction for a rolling wheel.

The largest size of tyre manufactured in India for off-the-road purposes is 24.00-25 (Good Year).

Tyres with wide base are sometimes used for earthmoving equipment. These, by virtue of their large surface in contact with the road, give better traction and are particularly used on temporary uncompacted roads with earthy, sandy or muddy surface. They are uncommon in mechanised coal or iron ore quarries. The only size of such tyres manufactured in India is 26.5-29 (Good Year).

Hydroflated Tyres

Drive tyres can be utilised to provide increased traction by filling them partly with a fluid and partly with air. The weight of the fluid gives additional weight to the tyre and hence the increased traction.

The suitable fluid is a solution of calcium chloride in water. 0.8 kg of $CaCl_2$ in one litre of water gives the proper fluid. It is not harmful to rubber. Not more than 75% of the tyre volume should be filled with the solution. A special valve with a sealed-in base is necessary to prevent separation between the rubber and the valve body. When checking pressures the valve must be in the lowest position and the vehicle should be under full load. Corrosion resistant pressure gauge should

be used for checking purpose. Hydroflated tyres are particularly useful in tractor, tractor loader and grader drive wheels. Tubeless tyres can be used for hydroflation but not the ordinary tubed type.

Care and Maintenance of Tyres

Tyres wear away in course of use and have to be replaced at intervals. Tyre cost is a sizeable recurring expenditure e.g. a rear tyre on Euclid truck, with a size 14.00—24, 20 ply, costs Rs. 2,700/- and on original tread it runs about 12,000 km on well maintained roads before it is removed in a condition fit for retreading. Attention has to be given to tyres right from the stage of storage and the following points need attention ;

1. Tyres should be stored in places where they are protected from light, heat, oil, dirt, moisture and ozone. Tubeless tyres should be stored vertically as horizontal stacking may compress the tyre walls and reduce their air sealing property.

2. Do not use hooks, cables or chains in contact with the tyre beads when lifting tubeless tyres as any damage to the bead will result in faulty air seal. For the same reason take particular care of the rims of tubeless tyres.

3. Maintain correct inflation. With over inflation, tyre centre of tread is subjected to excessive wear. To the regular operator of the vehicle over-inflation is felt by bumps during driving. Under-inflation causes the tyre to flex at every turn of wheel resulting in high internal heat. Under-inflation gives better absorption of road shocks, but results in heavier steering, lack of control on corners, increased fuel consumption due to the added rolling resistance and increased wear of tyre. Tyre pressure should be checked when it is cool, and slight rise of pressure due to heat during a run is not harmful. Table 2 gives tyre inflation pressures for some sizes.

4. Prevent tyre contact with oil or petroleum products as it makes the tyre rubber soft. Never let a tyre stand on oily or greasy spots overnight.

5. Maintain good haul roads. The investment on a road grader is well justified if the haul roads are long.

6. Match all dual tyres. Twin tyres should be of the same size, same ply rating and inflated to same air pressure, so that they share the load equally. Unequally worn out tyres are to be avoided. There should be a difference of not more than 13 mm in diameter or 38 mm in circumference between the two duals for tyre sizes above 8.25 inch section. If there is a difference, even within these limits, the smaller tyre should be on the outside position for earthmoving operations. To tyres on trucks for highway running, the reverse applies since roads are often highly cambered and the smaller diameter tyre should be fitted to the inside position, or alternatively, fit the more worn tyre to the inside position.

TABLE 2

Tyre inflation pressures and loads.

Tyre size	Ply rating	Air Press p.s.i.	Load Max. lbs.	Remarks
5.90—15	4	26	900	Ambassador Car.
do.	6	30	980	do.
6.00—16	6	28	1,225	Jeep.
7.50—20	10	80	3,090	Truck, highway.
8.25—20	10	70	3,330	do.
do.	12	85	3,730	do.
9.00—20	10	70	3,960	Tata Mercedez Benz truck.
do.	12	85	4,480	do.
10.00—20	12	75	4,580	Truck, highway
do.	14	90	5,210	do.
7.00—24	10	60	2,870	Road grader, front.
9.00—24	10	50	4,015	do. do.
10.00—24	10	40	3,945	do. do.
12.00—24	8	30	4,180	do. rear.
13.00—24	10	30	4,910	do. do.
14.00—24	10	30	5,780	do. do.
12.00—24	16	70	7,430	Euclid (FD model), front.
14.00—24	20	75	10,690	do. , rear.
16.00—25	24	70	13,490	Mack (M-25X), front.
18.00—25	28	70	17,500	do. , rear.
17·5—25	12	85	III,	Michigan tractor shovel
26·5—25	24			Scraper

7. Prevent overloading. Excessive loading will overstress both the vehicle and tyres and shorten the life of both

8. Inspect tyres regularly for cuts, bruises, fabric also aks, rocks lodged between duals, excessive or uneven wear, imbedded foreign matter, etc. Excessive or uneven wear on tyre may sometimes be due to mechanical irregularities such as wheel misalignment, excessive wheel camber, badly adjusted brakes, loose steering linkages, sagging or bent axle, worn wheel bearings, twisted frame, etc.

9. Remove tyres in time for retreading.

It is pertinent to remember that no tyre will give satisfactory service, if placed in the hands of a careless and erratic driver.

Remoulded Tyres

When the tread of a tyre is worn out, it loses appreciably its capacity to provide adequate traction and grip on the road. Beneath the tread is the undertread which is of comparatively softer rubber and the wear of the tyre is fast once the tread is worn out and the tyre is plain. A new tread can be fixed on a worn out tyre by processes which go by various names such as relugging, recapping, topcapping, remoulding and retreading.

The proper time to remove a tyre for retreading is when the tread depth is so much worn out that some rubber is still left on the plies. In some cases a tyre in which the top ply has just exposed at a place or two is accepted by retreaders but if the top ply is much worn out, the retreader will accept the tyre for retreading only at the risk of the tyreowner as the retreaded tyre will have much reduced life or early failure. Other considerations before a tyre is accepted for retreading are :—

(1) There should be no cuts on side-wall or crown so that the canvas or cord is intact.

(2) Bead is not damaged.

(3) Rubber of sidewall or shoulder is not separated from canvas plies.

(4) Tyre has not run under punctured condition as such run separates the cords and causes irreparable damage to them.

(5) The tyre is not too old. An old tyre is indicated by cracking on the sidewalls and the "deadness" of the rubber.

(6) Whether the life of the tyre after retreading justifies the expenses on retreading and other repairs.

How often is retreading possible? On tar roads and metalled roads in good condition a standard pattern tyre on trucks for highway running gives 25,000 to 28,000 km on original tread before the tyre is removed for retreading. 75% of this life is reasonable to expect on well maintained roads after first retreading till the tyre is removed for second retreading. Life of the tyre progressively decreases with each retreading as the ply strength decreases with use. Normally 2 retreadings on truck tyres and 3 on car tyres, if operating on good roads, should be considered as economic limit. Thereafter the life remaining in the tyre may not justify the cost of retreading and other repairs which the tyre will invariably require after such prolonged running. Off-the-road tyres have to undergo more rigorous service and only one retreading is normally considered economic for them. For all vehicles, the rear tyres which are the drive tyres, have less life than the front ones. In practice, the rear tyres of Euclid trucks, of size 14·00—24, 20 plies, have given the following life on well maintained metalled roads in iron ore mines.

On original tread till rejected	..	15,000 km.
-do- till removal for retreading	..	12,000 km.
Life after first retreading till rejection ..	..	10,000 km.

A new tyre of the above mentioned size costs Rs. 2,700/-. The retreading charges cost about Rs. 1,200/-. It should however be noted that only very few tyre retreading shops have facilities for retreading off-the-road tyres. The cost of transport to send the tyre to a distant place and the time involved are factors to be considered before deciding to have the tyre retreaded.

It is possible to retread tubeless tyres in the same manner as the tubed type.

Good-Year is marketing off-the-road tyres with steel wire under-tread, and such tyres have the letters S.W.U. marked on

them. The under-tread consists of metallic wires embedded into rubber. The tyre has the obvious advantage of more life due to high resistance to wear of the S.W.U. Such tyres can also be retreaded.

Michelin metallic tyres which were hitherto imported for trucks are of special construction. The cords are of metallic wire fabric, though the under-tread is of rubber. The steel cords provide very strong resistance to tread distortion under cornering loads. If however the cord near the tube comes out the latter is easily punctured. The tyres are difficult to retread and nowadays not available due to import restrictions.

A tyre on which a new tread has to be fixed should be entrusted only to reliable shops. The process of relugging, recapping or top capping replaces only the tread rubber compound which has been worn away. The process is simple; the tyre is rasped over its worn surface which is then sprayed with a rubber solution, and the new tread, consisting essentially of a band of rectangular sectioned rubber, is fitted over the prepared area of the tyre. The tyre is then subjected to heat treatment, known as "curing", which causes a good, adhesion between the tyre surface and the rubber, and also hardens the new rubber. The process uses approximately 50% of the quantity of rubber compound required for the original tread. It can however be adopted if the lug depth is significant and the pattern of tread is simple *e.g.* on tractor tyres. For the tyres of off-the-road vehicles, the process lends itself to free rolling tyres of front wheels, and the drive tyres having straight lugs of V shape cross lines, or other shape. It is not suitable for standard pattern treads of highway truck tyres.

Remoulding and retreading are synonymously used terms for the process of fixing up rubber on worn out tyre surface and subjecting the rubber to heat treatment ("curing") after putting it under a cast iron frame. The pattern of the C.I. frame is reproduced on the newly fitted rubber giving the tyre its tread pattern. In this process, the breaker strips, undertread and tread rubber are replaced and it is therefore costlier than the process mentioned earlier.

Retreaded tyres sometimes prove a failure due to casing fatigue. Casing is the term used for the tyre body from which tread and undertread have worn out. The new tread may separate out from the casing due to defective or negligent retreading operation and this is often the trouble noticed with tyres retreaded by inexperienced retreaders having little equipment. The rapid wear, sometimes observed of the new tread, may be due to :

(1) Poor quality of rubber for tread.
(2) Insufficient rubber in the tread.
(3) Under-cure.
(4) Unsuitable pattern.

When on the subject of tyre retreading a word of caution on tyres with nylon cord plies is desirable. Nylon cords, when cooled in the process of retreading, shrink much more than rayon or cotton cords and create problems in retreading. Tyres with nylon cord plies should therefore be entrusted only to retreaders having experience of those tyres.

7. THE TIPPING GEAR.

All dumpers/tippers are provided with arrangements to lift the loaded body by utilising hydraulic pressure to force a ram out. The body swings from its horizontal position round a fulcrum through nearly 70° to dump its load and the hydraulic system also functions to pull the body back on its seat i.e. the chassis. Fig. 27 shows the hydraulic system layout for the tipping gear of a Euclid dumper. From an oil tank oil flows by gravity to a hydraulic pump. When the driver engages the power take off (P.T.O.) control lever, power from the engine is transmitted from the transmission countershaft to the power take off which drives the pump. The oil under high pressure from the pump goes to the control valve and the control valve lever can be manipulated for 4 different positions.

(a) *Raise position* : High pressure oil goes through the hose pipes to the bottom of the hoist cylinder and the ram is then forced out. Oil at the top of the hoist travels back to the control valve through the hose connected to the piston rod.

(b) *Hold position* : Both the passages between the control valve and the hoist are closed so that oil at the bottom and top of

the hoist is at a stand still and the latter is unable to move in either direction.

(c) *Float position* : Both hose passages between the control valve and the hoist are open so that oil at either end of the hoist

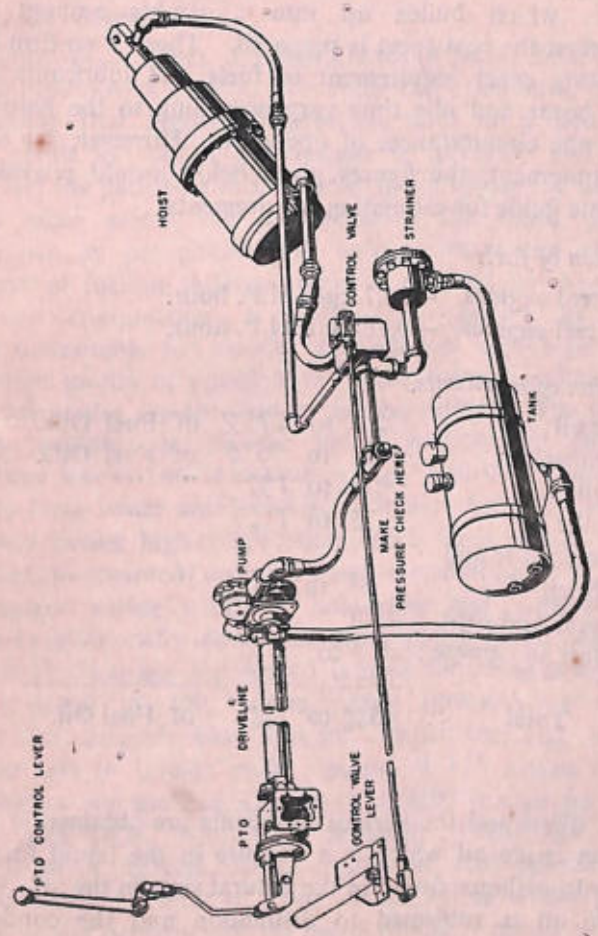

Fig. 27. Tipping gear on a Euclid truck

can flow either way. The hoist can then travel in either direction depending upon the direction in which the force is applied.

(d) *Lower position* : High pressure oil goes to the top of the hoist which then telescopes itself by the oil pressure and the

oil at the bottom of the hoist travels back to the tank via the control valve. The body is thereby lowered on to the chassis.

8. FUELS AND LUBRICANTS

The expenses on fuels and lubricants form a recurring expenditure which builds up into a sizable amount over a month when the cost sheet is prepared. There is no firm basis of calculating exact requirement of fuels and lubricants since operating hours and idle time vary according to the nature of work and the circumstances of operation. However, for earth-moving equipment, the figures given below would provide an approximate guide for estimating requirements.

Consumption of fuel.

1. Petrol engines —0.27 kg/B.H.P. hour.
2. Diesel engines —0.18 kg/B.H.P. hour.

Consumption of lubricants.

Engine oil	: 2	to 2.75%	of	Fuel	Oils.
Air cleaner oil	: .5	to .75%	of	Fuel	Oils.
Gear oil	: .75	to 1%	"	"	"
Hydraulic oil	: .75	to 1%	"	"	"
Air compressor and Air tool oil	: .5	to .75%	"	"	"
Open gear and rope lubricants and grease	: .5	to .75%	"	"	"
Total ..		5% to 7%	of	Fuel	Oil.

Fuels

Petrol, diesel and the various lubricants are obtained in a refinery from crude oil which is a mixture in the liquid form of various hydrocarbons, found in the natural state in the oil wells. The crude oil is subjected to distillation and the condensation at different stages yields various products like the incombustible heavy asphalt, furnace oil, diesel oil, kerosene, petrol and the familiar gas used for domestic heating.

The value of a fuel, petrol as well as diesel, lies in its heating qualities. Petrol should possess another important quality,

volatility, so that it can readily evaporate in air and burn easily inside an engine. As a general rule, addition of benzole, a byproduct from coke ovens, increases the heating value of petrol while addition of alcohol decreases it. Petrol marketed at petrol pumps by various oil companies in this country has practically the same properties as far as motors and trucks are concerned.

There are, however, different grades of petrol differing mainly in their *anti-knock quality*. To understand this term, one has to appreciate that higher compression ratio in a petrol engine may result in such high increase of pressure and temperature that the petrol-air mixture may ignite before the timed spark takes place across the electrodes of the spark plug. This is known as pre-ignition. In extreme cases the rapid combustion of fuel-air mixture, may be replaced by explosion due to high compression, a phenomenon known as *knocking*. It is undesirable for smooth and efficient engine performance. Different grades of petrol have different anti-knocking qualities and the grades are specified by octane ratings. The petrol normally available at all the petrol pumps and suitable for cars and trucks has 78 octane rating. Petrols of higher octane rating have better anti-knocking qualities and they are used in engines having high compression ratios. Such petrols have internationally accepted distinguishing pigmentations, *e.g.*, petrol of 73 octane rating which is colourless and without lead, is used in aviation for some aircrafts ; petrol of 80 octane rating which is colourless but leaded is used for some costly imported cars ; petrol of 100 octane rating (greenish in colour) is used for aircraft like Dakota, Skymaster and also some costly cars of foreign make ; petrol of 115 octane rating, the highest octane number, (purple in colour) is used for some aircraft like DC6. An engine normally would run well with petrol of a definite octane rating as recommended by the manufacturer. Petrol of lesser octane rating would give somewhat poorer performance but petrol of higher octane rating would be satisfactory.

The diesel oils used in compression-ignition engines are specified as Gr. A, Gr. B and Gr. C in the Indian Standard specification No. 1460-1959. Gr. A and Gr. B oils are high speed

diesels but Gr. C is low speed diesel used only for some stationary diesel engines running at speeds not exceeding 700 r.p.m. Gr. A and Gr. B diesels have the same specifications, the only difference being in their cetane number. The diesel marketed by various companies at the diesel pumps for trucks and other high speed machines is of Gr. B quality.

There is no octane rating for diesel which is rated by cetane number.

LUBRICANTS

A lubricant prevents metal to metal contact between two metal surfaces rubbing, turning or scraping against each other, by providing a very thin film of lubricant between them. It thus reduces friction, wear and heat, and prolongs the life of the working parts. The lubricating oil also serves as a cooling medium and dissipates appreciably the heat produced by friction. The waste of power in overcoming friction is also much reduced.

Lubricants used for machines are oils and greases. They loose their lubricating properties to some extent in course of use and get contaminated with external dust sucked from air, through leaky oil seals or other openings, small though they be. They have, therefore, to be changed periodically as recommended by the machine manufacturers.

The nature of lubrication required of a lubricant and its qualities depend upon the part of a machine for which it is used. Engine oil, for example, should remain chemically stable at high temperatures produced inside the engine and should have viscocity which will be nearly the same for all the ranges of temperatures. A too viscous oil may not get access to the fine clearances, *e.g.*, between bearings and journals. A viscous oil is somewhat difficult to circulate and may require more power for the oil circulating pump. A too thin oil may flow away easily from the surface it is required to lubricate and its thin film between rubbing surfaces may break under high loads. Oil used for gearbox is more viscous and it is not required to withstand the high temperatures which an engine oil has to encounter but some gears have to with-

stand high pressures. Slow moving parts require viscous oil or grease.

Viscocity of an oil is specified by S.A.E. (Society of Automobile Engineers, an American Organisation) numbers which are internationally accepted. The higher the number, more viscous is the oil. Thus S.A.E. 30 oil is more viscous than S.A.E. 20 oil. All the oils change their viscocity with temperatures. Higher temperature results in less viscocity and more fluidity ; lower temperature causes more viscocity and less fluidity. Most of the oils are marketed by only one number of viscocity *e.g.* S.A.E. 30 ; S.A.E. 10W (W stands for winter grade). Some of the engine oils are however marketed as multigrade oils. The change in their viscocities with variation of temperatures is very low so that the same oil can be used during summer as well as during winter. Such oils are specified by two numbers such as S.A.E. 20W-40 which means that the oil is free flowing during winter to comply with S.A.E. 20 viscocity but it will be sufficiently viscous to comply with S.A.E. 40 viscocity during summer. Multigrade engine oils are used only for petrol engines and not for diesel engines.

Engine oils contain detergent additives which wipe off the dirt and unwanted combustion products from inside the engine and keep them afloat in the oil until they are trapped by an oil filter. Diesel engines tend to produce sludge and varnish depositing compounds as byproducts of combustion. They, therefore, need heavy duty detergent oils which keep the byproducts of combustion in suspension and engine oils suited for petrol engines are, therefore, unsuitable for diesel engines. Oil companies market diesel engine oils having varying percentages of detergent additives. ESSO Oil Company, for example, markets diesel engine oils with prefix HD to indicate normal detergent quality ; HDX, to indicate more detergent qualities ; and D 3, to indicate the maximum detergent qualities. Burmah Shell's equivalents for HD are Rotella ; for HDX, Rotella T ; and for D3, Rimula. Some of the machinery manufacturers recommend only D3 or similar engine oils (having maximum detergent qualities) for their latest equipment *e.g.* Caterpillar Tractor Co., International Harvester Co., and a few others

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Engine oils contain detergent additives which wipe off the dirt and unwanted combustion products from inside the engine and keep them afloat in the oil until they are trapped by an oil filter. Diesel engines tend to produce sludge and varnish depositing compounds as byproducts of combustion. They, therefore, need heavy duty detergent oils which keep the byproducts of combustion in suspension and engine oils suited for petrol engines are, therefore, unsuitable for diesel engines. Oil companies market diesel engine oils having varying percentages of detergent additives. ESSO Oil Company, for example, markets diesel engine oils with prefix HD to indicate normal detergent quality ; HDX, to indicate more detergent qualities ; and D 3, to indicate the maximum detergent qualities. Burmah Shell's equivalents for HD are Rotella ; for HDX, Rotella T ; and for D3, Rimula. Some of the machinery manufacturers recommend only D3 or similar engine oils (having maximum detergent qualities) for their latest equipment *e.g.* Caterpillar Tractor Co., International Harvester Co., and a few others

Such oils are specified for diesel engines operating under conditions of extreme severity, notably on high sulphur diesels. They are also specified for many transmission, gear units, clutch housings, hydraulic systems, etc. which by design, are integral with diesel engine units so that the oil used in the crankcase serves those systems also and the need to stock a variety of gear oils or hydraulic fluids is eliminated. It is, however, not advisable to fill highly detergent oil into an old engine which may have deposit of dirt or other undesirable combustion products inside ; these products will be released with a few hours' run by the highly detergent oils, and the comparatively sudden release can interfere with oil circulation.

Upper cylinder lubricants are oils added to petrol for reducing wear of cylinder walls. These upper cylinder oils are used only for petrol engines and not for diesel engines. They are added in the proportion of 0.03 litre for every 10 litres of petrol. Their advantage for reducing engine wear is a controversial subject and in any case, use of upper cylinder lubricating oil is recommended by some authorities only for the running in period of a new or reconditioned engine.

Table 3 lists the oils and lubricants marketed by some leading Oil Companies.

TABLE 3 Lubricants for Various Purposes in Opencast Mines

Purpose	*ESSO Oil Company*	*Burmah Shell*	*Remarks.*
Engine Oil			
1. Diesel engine	ESSO lube HD 30	Rotella 30	For winter.
	-do- 40	-do- 40	For summer.
	-do- 50	-do- 50	For very high temp.
			Shell rimula oil is used for tractor engines.
2. Petrol engine	ESSO Motor Oil 30	Shell X 100 —30	As above ; Alternatively
	-do- 40	-do- —40	ESSO extra motor oil 20W-30-40.
	-do- 50	-do- —50	

Purpose	*ESSO Oil Company*	*Burmah Shell*	*Remarks.*
Upper cylinder lubricant.	ESSO top lube	Donax U	For petrol engines only.
Air cleaners	Normally same as engine oil, or slightly less viscous engine oil.		
Gear box & differential	Esso Gear Oil GP 90	Spirax 90 EP	Dumpers, trucks, cars, Michigan tractor shovel etc.
	-do- GP 140	-do- 140 EP	For heavy loads. *e.g.* For Bulldozers, Motor graders, Scrapers, etc.
Steering box	ESSO gear oil GP 90	Spirax 90 EP	
Torque converter	ESSO Automatic transmission fluid.	Automatic transmission fluid A, or Shell X100-20W.	
Fluid coupling	Esso Motor Oil 10	Shell X 100-10W	
Hydraulic power lift.	Esso Motor Oil 10	Shell X 100-10W	Hydraulic steering booster, tipping gear, hydraulic rams.
Turbocharger or Supercharger	Tesesso 43	Tellus 27	
Hydraulic brakes	Standard Brake fluid	Donax B.	
Shockabsorbers	ESSO shockobs oil	Heavy Donax A4	Heavy dumpers
	-do-	Medium	Trucks, Buses
	-do-	Light. Donax A1	Motor cars.
Compressor (cylinders and bearings).	Estor 30 (napthanic base) Teresso 52 or 56 (paraffinic base).	Talpa 30 Turbo 33	
Air line lubricator.	Arox EP 65	Tona F Clavus 27	Rock drills.
Open chains & sprockets.	Coray 65 or Millcot 60	Trochus 45	Normally chead oil is used.
Open gears and wire ropes.	Surett 1,000 or surett fluid 30	Cardium compound D or F	Grease
General grease lubrication	ESSO Multipurpose grease H	Shell M.P. grease.	
Wheel bearing	ESSO W.B. grease	Shell W.B. grease.	

REFERENCES

1. THE MOTOR MANUAL ; Temple Press Ltd., London.

2. ABBEY, STANTON : "Practical Automobile Engineering", Asia Publishing House.

3. HERBERT L. NICHOLS, jr. "Moving the Earth", 2nd edition, 1962.

4. AUDEL, "New Automobile Guide", D.B. Taraporewala Sons & Co.

Chapter 10

TYPICAL EARTHMOVING EQUIPMENTS & THEIR PERFORMANCE

1. Industrial wheel tractor. 2. Bulldozer. 3. Ripper. 4. Scraper 5. Tractor shovel. 6. Motor grader or road grader. 7. Dumpers or tippers. 8. Excavators—dipper shovel, pull shovel, dragline. 9. Working costs. 10. Bucket wheel excavators ; Neyveli Lignite Project.

A number of machines used in opencast mining consist essentially of the familiar tractor with attachments for special purposes, *e.g.* bulldozer, ripper, tractor shovel, road grader. The work done by three different machines is sometimes carried out with only one machine which is then known as 3-in-1 machine and is equipped with detachable attachments each of which can be brought into use to serve a definite purpose. Power of these machines, especially of those fitted with torque converters, is often expressed in terms of drawbar pull in kg at a given speed or in terms of flywheel horsepower. There is another set of machines which is widely used in opencast mines and which consists essentially of a revolving shovel equipped with a suitable boom and attachment, *e.g.* a dipper shovel, crane, dragline, clamshell, and back hoe. Bucket wheel excavator, a unique machine of its own kind, is not a common sight in mechanised quarries in India and is used only at Neyveli lignite mine.

Dumpers have found an extensive application in quarries on earth moving jobs.

1. INDUSTRIAL WHEEL TRACTOR.

The familiar tractor is with a petrol or a diesel engine, though the latter is more common. The clutch, mechanical transmission and steering are on the same lines as in an ordi-

nary truck. The wheels are on pneumatic rubber tyres. The rear axle is longer and the rear wheels, which are the drive wheels, are larger having tyres with road gripping tread pattern.

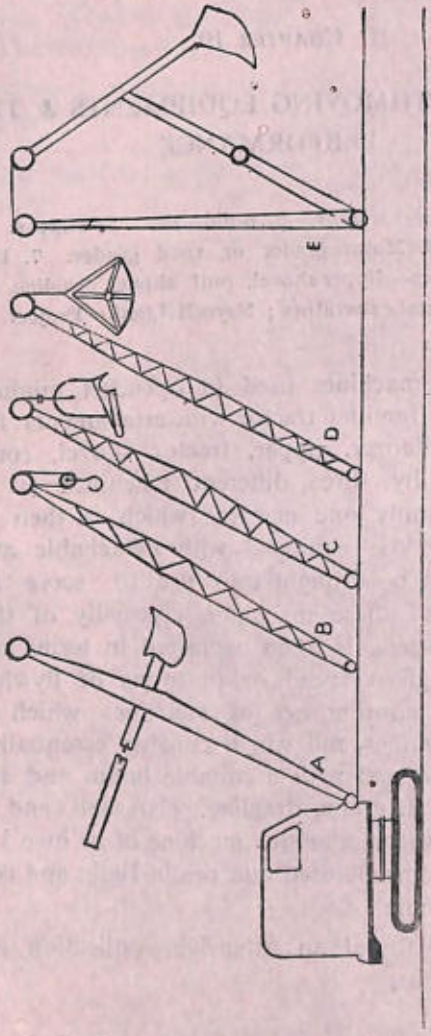

Fig. 1. A revolving shovel can be used for different purposes by suitable boom and attachments
A—Dipper Shovel B—Crane C—Dragline D—Clam Shell
E—Back Hoe or Pull Shovel.

The front axle is shorter and the wheels are of smaller diameter with tyres of free rolling tread pattern.

The maximum size of tractor with pneumatic tyres manufactured in India is 50 H.P. Such small tractors are not

much used in quarries except on a limited scale for towing stores, lubricants and other materials in a trailer. They are, however, used on a large scale for agricultural and municipal purposes.

The two-wheel drive pneumatic tyred tractors are manufactured by such companies as Hindusthan Tractors Limited, Escorts, Fergusson, and a few other companies.

Four-wheel drive tractors are finding increasing acceptance in quarry work for dozing and shovel loading.

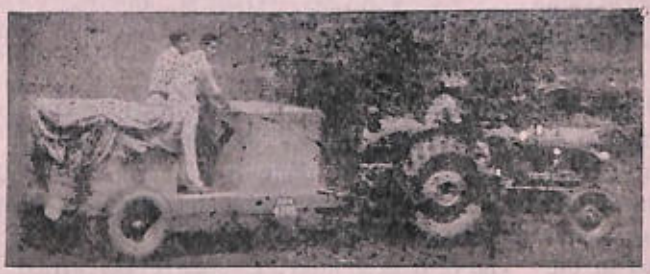

Fig. 2. A tractor towing a trailer of liquid oxygen cartridges.

Some important *specifications of Hindustan Tractor* (model HWD-50, diesel) manufactured by Hindusthan Tractors, Limited at Baroda are as follows :—

Engine	..	Four stroke, 4-cylinder, diesel, water-cooled.
Bore/stroke	..	105/120 mm.
Engine power output		50 B.H.P. at 1,650 r.p.m.
Compression ratio	..	15.6 : 1.
Clutch	..	Dry, double clutch, one for tractor, and the other, disconnectable, for power shaft operated equipment.
Speeds	..	Without reduction 4 front, maximum is 26.6 km/hr and one reverse 4.66 km/hr ; with reducer, additional 4 front, maximum speed is 6.47 km/hr and one reverse, 1.13 km/hr.
Overall length	..	3,570 mm.

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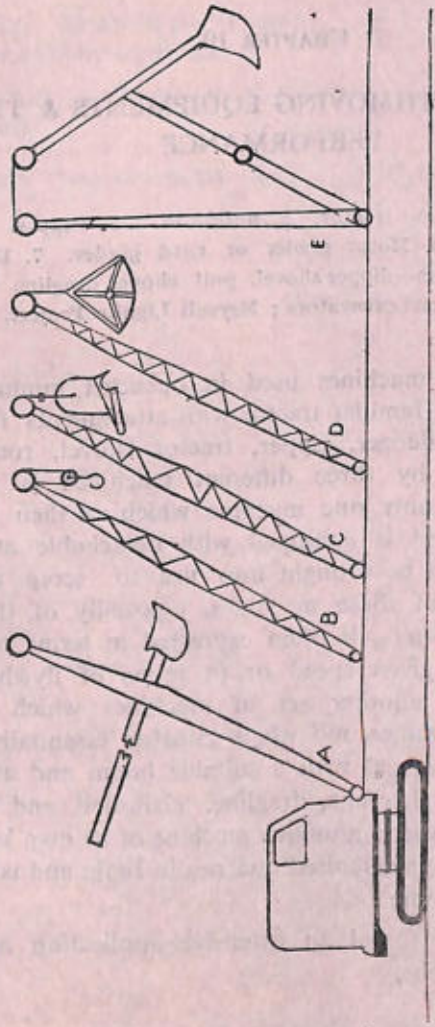

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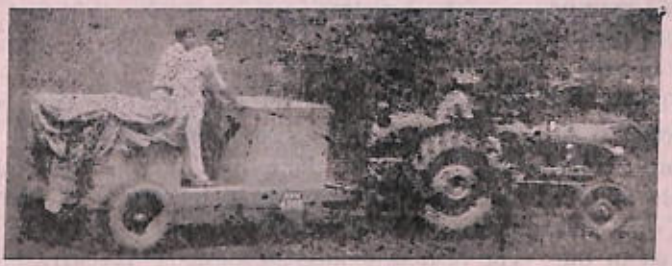

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Speeds	..	Without reduction 4 front, maximum is 26.6 km/hr and one reverse 4.66 km/hr ; with reducer, additional 4 front, maximum speed is 6.47 km/hr and one reverse, 1.13 km/hr.
Overall length	..	3,570 mm.

Wheelbase	..	2,200 mm.
Gearbox ground clearance		470 mm.
Weight in operating condition with hydraulic power lift	..	2,740 kg.
Front axle	..	Independently sprung wheels. Wheel tread adjustable 1,280—1,730 mm.
Rear axle	..	Adjustable wheel tread stagelessly 1,380—1,820 mm.
Turning radius	..	3 metres.
Starter motor	..	12 V, 4 H.P.

Heavy duty tractors that carry shovel dozers or rippers are sometimes equipped with clutches that operate in oil. For example, in Caterpillar dozers there is a double plate clutch which operates in oil kept in circulation by a pump. Linings are of a special type which have good gripping power when soaked in oil. The heat generated by slippage is carried away by circulating oil.

A clutch is manipulated by foot as in cars and trucks. The foot pressure releases the clutch which is re-engaged by a spring on the clutch pedal after the foot is taken off. On some shovels the foot pressure is sometimes assisted by hydraulic pressure where the clutch is hydraulic, as such control arrangement minimises the effort required to release the clutches.

Some heavy duty equipment sometimes uses "shoe clutches" which are internal expanding bands similar to band brakes with the exception that the bands and linkages revolve. This type of clutch is preferred for severe conditions where there are rapid changes in load.

2. A BULLDOZER.

A bulldozer (referred to as "dozer") is a tractor with a pusher blade attached to the front portion. The tractor is the diesel operated power unit, which is now-a-days generally equipped with hydraulic steering, and torque converter drives. Dozers are equipped with crawler chains or rubber tyred wheels. The

pusher blade can be raised or lowered by hydraulic arrangement or by cables and is used for pushing loose material or for digging. A bull dozer is not a primary dirt mover in the same sense as a scraper. On most earthmoving jobs a bull-dozer is considered an auxiliary equipment for levelling or spreading or compacting in dumping yards with very little attention given to its earth moving capabilities. Angling dozers are bull dozers, which have blades that can be set at angles to cast dirt, loose rock or small boulders to either side when the tractor moves forward. The pusher blade is rectangular at the bottom edge, concave shaped and sloped backwards. The cutting edge of the blade wears most and is detachable for replacement. A hydraulically controlled blade is more suitable for digging hard surfaces than a cable controlled one but the latter has a faster drop and lift. In case of cables grading is done keeping the cables tight, but digging may be done with a few inches of slack in the cable, allowing the latter to find its own depth. The cables are liable to break often causing stoppages and need replacements.

A bull dozer is generally known as a *maid of all works*. It is used for clearing bush and surface rubbish, to excavate surface soils, to consolidate and grade temporary roads, for levelling of rock spoil in dumping yard, for pushing coal into bunkers through grizzly at ground level, for towing dumpers, etc. It is often used as a necessary adjunct to a shovel to level, grade and clear the bench of scattered rock after blasting or spillage. This reduces wear and tear on tyres. The toe of a bench, which is not effectively dug by a shovel, is dug and scraped by a dozer and collected into the rock heap for loading by the shovel. It is used sometimes for shifting heavy cables. For upkeep of dumper track it is indispensable. It can dig 1.2 m to 1.5 m below ground.

The only crawler bulldozer manufactured indigenously is that by Bharat Earthmovers Ltd., Bangalalore, in collaboration with Komatsu Manufacturing Co. of Japan. With a flywheel H.P. of 235, it is available as hydraulic as well as cable type. Its *main specifications* are as follows :

Engine :—Model Komatsu 6D155-2 ; 4-stroke ; diesel ; 6 cyl ; 55 mm bore × 200 mm stroke ; 22.6 litre piston displacement ; water cooled ; max. torque 142 kg. m.

Fuel consumption :—190 g per H.P. hour.

Starting :—By auxiliary petrol starting engine.

Flyweel H.P. :—235 H.P. at 1300 r.p.m.

Operating wt :—Cable type, 27420 kg ; hydraulic type, 27220 kg.

Travel Speeds :—5 front and 3 reverse.
Max. forward speed 8.6 km/hr.
Max. reverse speed 7.4 km/hr.

Blade :—4220 mm wide × 1160 mm high ; turning angle each side 25°.

Dimensions :—Overall length 6435 mm × overall width 4220 mm × overall height 3160 mm.

Track :—Gauge 2060 mm; ground clearance 390 mm ; width of track shoe 560 mm ; tension adjustment by grease gun system.

Main clutch :—Wet type with hydraulic booster.

Steering clutch :—Hydraulically operated wet type.

3. RIPPER.

A Ripper is a machine which cuts, as it travels, 0.6 to 1 m deep furrows in the ground, and it can be well compared with the farmer's plough. The Ripper is essentially a crawler mounted heavy duty diesel tractor with a ripper attachment as shown in fig. 3. Like a farmer's plough, the ripper with the ripping tool thrust into the ground by hydraulic pressure, travels along close parallel paths, about 1.2 m apart, and during the travel rips open the ground. The broken ground or rock can be dozed to form a stockpile for loading or can be loaded by a scraper. If the overburden or mineral is suitable for ripping, its breaking is possible with the help of a ripper and the process of drilling and blasting can be dispensed with.

Soft rocks and medium hard rocks, below hardness 5, which are laminated and stratified, provide suitable material for ripping. The alluvial surface deposits, weathered sandstones and shales underlying them in the coalfields can be easily ripped. In the iron ore areas, laterite covering in a few places lends itself to ripping, as also the laminated hema-

titic deposits and the blue hematitic powder underlying them. Rippers are used in some coal mines in Raniganj coalfield to break weathered sandstone in the overburden, and coal. They are also used in limestone quarries in Sourastra and some quarries of hematitic iron ore in Goa.

Certain conditions are conducive to ripping *e.g.* fractures and faults in the rocks, weathering, brittle and crystalline nature, high degree of stratification or lamination, large grain size and low compressive strength. The extent of consolidation of the rock, however, is the main factor to decide its rippability and a

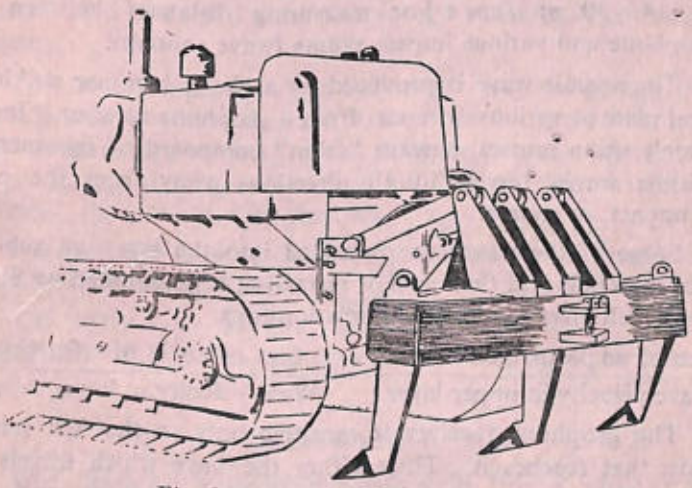


Fig. 3. A ripper attachment

Refraction Seismograph is the instrument which is used to give the relative rippability of the ground or rocks to be ripped.

The Refraction Seismograph operates on the principle that "Sound waves travel through subsurface material at different velocities, depending upon the degree of consolidation of the material". It is believed that the same factors that affect consolidation also affect rippability. Thus poorly consolidated material with low seismic wave velocities could be ripped easily, while highly consolidated material with high velocities would be difficult to rip.

Equipment needed for seismic analysis includes, a source of a sound or shock wave, a receiver, an electric counter, and a set of cables.

The main items are :

1. *Refraction Seismograph* : An electronic counter that determines the time interval between the strike of the hammer and the arrival of the seismic wave at the geophone.

2. *Geophone* : Receiver of sound waves.

3. *Sledge Hammer and Impact Plate* : Source of sound wave transmits sound through earth and also through an impact switch having direct connection with seismograph, through a connecting wire.

4. 30 *m Tape* : For measuring distances between the geophone and various impact points (wave sources).

The seismic wave is produced by a sledge hammer striking a steel plate at various distances from a geophone receiver. Immediately upon impact, a wave "front" composed of innumerable seismic waves travels in all directions away from the point of impact, or source.

Some of the waves are refracted into the layers of sub-surface materials and the angle of refraction is determined by Shell's law which gives the following relationships.

$$\frac{\text{Size of angle of incidence}}{\text{Wave velocity in upper layer}} = \frac{\text{Size of angle of refraction}}{\text{Wave velocity in lower layer}}$$

The geophone receiver is sensitive only to the first seismic wave that reaches it. Thus, either the wave which travels the shortest distance, or one which travels a longer path but which includes a high velocity segment, arrives first at the geophone.

In addition to determining the degree of consolidation or rippability of each layer, it is also possible to determine the depth of each layer.

In iron ore areas of Goa the practical results obtained with seismograph were as follows : Seismic velocity in overburden (practically laterite) was 600 to 1,200 m per second. In iron ore it was 1,050 to 1,500 m per second but in some cases velocities as high as 1,800 to 2,100 m per second were also recorded.

When selecting a tractor for ripping purposes, it is necessary to consider (1) the down pressure on the tooth to determine

whether penetration can be accomplished, (2) the tractor H.P. which should be capable of advancing the tooth through the rock and break it, (3) tractor weight which provides traction for full use of the H.P. in advancing the tooth.

The main replaceable part is the detachable cutting tooth on the digging shank. The tractor suitable for ripper attachment is D8 Caterpillar, TD25 International and similarly powered models. The latest models are D9, TD30 and others which have higher H.P. and are, therefore, suitable for heavier duties in iron ore mines. The shank may be rigid or hinged ; the former type is preferred for positive digging.

The tractor speed is 0.8 to 2.5 km/hr during ripping. If the rock is soft it is advisable not to increase the speed but to add one or more ripping teeth. The distance between adjacent furrows during ripping may be 1 to 2.4 m. The harder the material, the closer are the furrows.

In some rock formations ripping is possible after sparse blasting of widely spaced charges.

The heavy duty tractors may be expected to rip and doze about 250 to 350 tons per hour, depending upon hardness of rock.

4. SCRAPERS.

This machine is diesel-operated with rubber tyred wheels and has, between the front and rear tyres, a bowl fitted with a cutting blade at the bottom (See Fig. 10). The blade is reversible and can be replaced when blunt. Its working may be compared to that of a lawn mower. As a scraper is pushed forward by a dozer, its blade cuts a thin slice of earth usually between 75 mm and 230 mm thick over a distance of nearly 30 m and the earth is automatically collected in the bowl located at the centre of the machine. The bowl capacity ranges from $3m^3$ to $22m^3$ and takes nearly half to one minute for loading. The bottom discharge opening of the scraper is closed or opened by wire ropes or hydraulic system controlled by the operator. After the bowl is loaded it is lifted clear off the ground by the same cables or by hydraulic control, before the

scraper moves onward to the dumping yard where its contents have to be unloaded. At the dumping yard, as the scraper moves, the bottom opening of the bowl is opened and the contents are unloaded in a layer 150 mm to 250 mm thick, over a distance of 30 to 70 m. The bowl is always bottom discharging.

A scraper's power and traction can be substantially increased by mounting a second engine at the rear to get the advantage of 4-wheel drive. The extra power also enables the scraper to self-load without the help of a dozer, to move faster and climb steep gradients.

A wheeled scraper is commonly used. It is fast, turns easily round curves and can negotiate gradients of upto 20 to 25°. A scraper may take 5 to 6 minutes for a complete cycle of loading and unloading if the total up-and-down distance is nearly 1,000 m. One way traffic of loaded and empty scrapers is essential for good results.

One dozer is normally sufficient for every two scrapers used and it is required only for pushing the scraper when the latter is cutting.

A scraper is not only a convenient machine for bulk earth moving but also a levelling or finishing equipment. Scrapers are unsuitable in soils with tree stumps, large boulders and hard rocks. When the ground is hard, it is necessary to rip the surface with the help of a ripper before loading by a scraper. Sandy soil is best suited for a scraper. In alluvium, such machines have to be stopped during rains. Scrapers are not used in iron ore mines due to the hardness of the laterite and iron ore. They have been used in coal mines for cutting and transporting soft sandstone as well as coal. The coal excavated by it is however of smaller size.

The main wearing parts are the cutting blade and the wire ropes (cables).

In practice, a scraper with a struck capacity of 9 m^3 loose, or a heaped capacity of 13.5 m^3 gave 10.5 m^3 loose as an average load. One dozer was sufficient for 2 scrapers, which together gave the same output as that of a 2 m^3 shovel, when the dumping yard was nearly 60 m away.

5. TRACTOR SHOVEL

It is essentially a tractor with a bucket as the front attachment. A tractor shovel is sometimes called a shovel dozer, dozer shovel, or tractor loader. The machine is mounted on crawler chains or pneumatic tyres and in the latter case, the front wheels are sometimes of smaller diameter than the rear ones for easy manoeuvrability and steering. The circumstances in which a crawler chain shovel or a pneumatic tyred shovel enjoys preference have been stated in the earlier chapter. The tyred tractor shovels are sometimes fitted with steel chain netting on the pneumatic rubber tyres. These chains ensure a

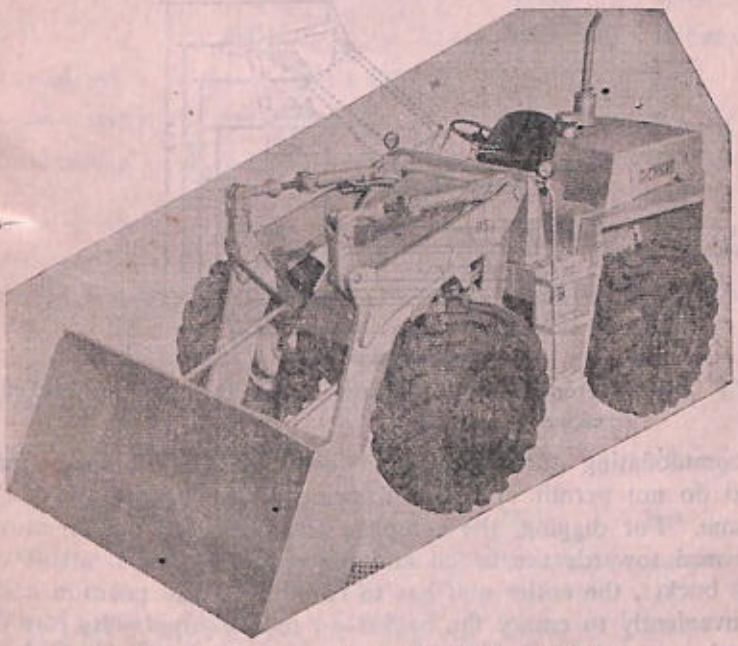


Fig. 4.—A tractor shovel, model Michigan 85 III.
(Courtesy of United Provinces Commercial Corpn.)

firm stand on all types of ground so that the shovel can perform somewhat harder digging at the benches than is possible with only rubber tyred equipment. The chains also minimise the wear on the rubber tyres and prolong their life nearly three to four times. The tractor shovel attachments consist of a push frame and a bucket that can be pushed forward into a rock pile,

raised, lowered and dumped hydraulically or mechanically. Some models are designed for unloading of the bucket in the front; others, for unloading sideways. Capacities of the buckets range from 0.5 m^3 upwards and 2 m^3 buckets are quite common in coal as well as iron ore mines.

The bucket is of simple box construction and usually the same width as the outside of the tracks. Cutting teeth are standard equipment on machines used for rock digging. The shovel, usually has a pusher fan so that the dirt falling from the bucket will not be sucked back towards the operator and engine air intake. The bucket, the push frame and the chassis

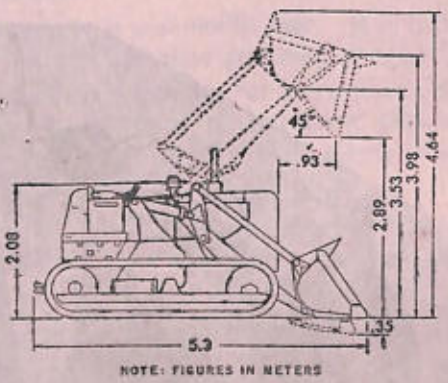


Fig. 5—A tractor shovel, Allis Chalmers, HD 11 G, 2·25 m^3 bucket capacity.

accommodating the engine and controls are all in a line and do not permit of a lateral swing of the bucket and push frame. For digging, the complete tractor shovel has to move forward towards the bench and for unloading the contents of the bucket, the entire unit has to come back and position itself conveniently to empty the bucket on to a dumper. Its rate of digging and loading cannot, therefore, be as fast as with a revolving shovel, described later.

Tractor shovels have been employed in some coal mines to load the stacked coal at the siding into railway wagons. In the iron ore mines they are used to load blasted rock at the bench into dumpers. In a shift of 8 hours a 2-m^3 shovel can normally load at the bench 1,000 tons of iron ore into dumpers of 6 to 10 tons carrying capacity, but the performance

may be about 1,500 tons for large capacity dumpers of nearly 12 to 15 tons capacity as the total time lost in positioning the dumper near the shovel is somewhat reduced.

TABLE 1

Leading specifications of some tractor shovels

	Michigan Model 85 III	Caterpillar 977-H	Allis Chalmers HD 11 G.
Engine Make	Cummins C-160-CL	CAT D 333	A.C. 10000
H.P.	160(Max).	150(rated)	111(rated)
Bore, stroke	4.44" × 5" ; 6 cyl	6 cyl	6 cyl
Torque(max), lbs-ft.	519 at 1400 rpm	N.A.	N.A.
Transmission	Torque converter Clark, powershift.	Powershift	Manual shift.
Travel speeds	4 front (28.6 mph, max)	4.7 mph, max. front.	5.7 mph, max. front.
	4 reverse (-do-)	5.7 mph, reverse max.	4.4 mph, max. reverse.
Bucket capacity (cyd) heaped.	2.25	2.5	2.25
Max. dumping height inches.	109	100	114
Weight, operating	21,000 lbs.	N.A.	N.A.
Traction	Pneumatic tyres 17.5—25,12 ply	Crawler	Crawler
Ground contact, of crawlers (Sq. in.)	—	3820	3416
Gauge of tracks (inches)	—	74	74

6. ROAD GRADER

This is a machine for levelling the road surface by smoothening out the ups and downs and for casting aside the boulders on the road. It is always pneumatic-tyre-mounted with only rear-wheel drive and the front wheels are small. The grading

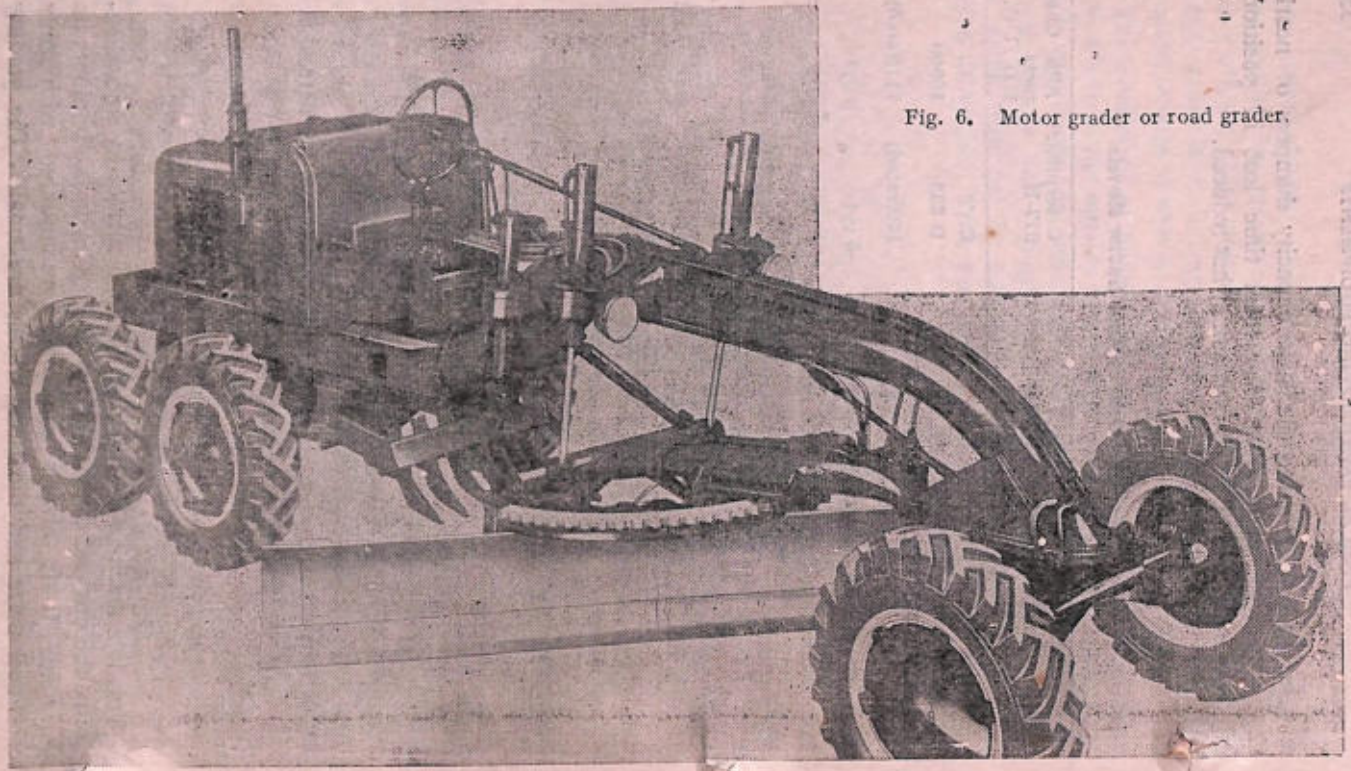


Fig. 6. Motor grader or road grader.

blade is attached to a circle that is hung from the overhead frame and pulled by a drawbar fastened to the front of the frame. The circle is a toothed ring having gear teeth on the periphery. The blade is usually 3.5 to 4 m long having replaceable edges on sides and bottom, and an extension, 0.6 m long, may be obtaining for bolting to either end of the blade. Power control for the blade operates through the circle but the angle of blade with the road is changed by rotating the circle. The brakes are usually hydraulic and act only on the rear wheels. Steering is direct-connecting mechanical by a hand wheel, though a hydraulic booster is fitted on some models (Fig. 6).

A set of teeth known as **Scarifier** are mounted in front of the blade and used for breaking up surfaces too hard for easy penetration by the blade. It can be lifted and lowered by means of a pair of cranks on the ends of a cross shaft. It is possible to attach on the front of the frame a bulldozer blade operated by the scarifier control. This attachment proves quite useful in spreading piles of material.

The transmission provides 6 forward speeds from 2 km to 32 km per hour and two or three reverse speeds from 5 km to 12 km per hour.

7. DUMPERS OR TIPPERS.

The constructional features and other details of heavy duty dumpers have already been explained in the previous chapter and only the latest developments which characterise some of the dumpers will be mentioned here.

The indigenously manufactured dumpers are of smaller size. Tata Mercedez Benz manufacturers dumpers capable of side tipping as well as rear tipping with a payload of 5 tons. Other truck manufacturers like Bedford, Dodge and Fargo also market rear dumpers of 5 to 7 ton payload. T.M.B. dumper, 5 tonne payload capacity, costs about Rs. 55,000/- including chassis, body and tipping gear, ex-Jamshedpur. Ashok Leyland manufactures the following sizes of rear dumpers. Prices are ex-Calcutta including sales tax (10%).

Comet, 7·5 tonne payload, 6 tyres, Rs. 81,200/-
Beaver, 10 tonne payload, 6 tyres, Rs. 1,52,000/-
Hippo, 16 tonne payload, 10 tyres, Rs. 2,11,520/-

The tipping gear suitable for 5 to 7 tonne payload is manufactured in India by Telehoist (Calcutta) and Allwyn (Hyderabad). Larger capacity tipping gear is imported.

Following are some of the specifications of Euclid and Mack dumpers in use in the mining industry.

	B7FD Euclid	Mack M 25X
Engine	Cummins four cycle, diesel NH 220, 6-cylinder, 220 H.P. at 2,100 r.p.m. Bore and stroke : $5\frac{1}{8}'' \times 6''$; Maximum torque at 1,300 r.p.m. : 613 lbs. ft.	Mack END 864, diesel, 4-cycle, 6 cyl., 255 B.H.P. at 2,300 r.p.m. ; torque at 1,700 r.p.m. is 639 lbs. ft.
Clutch	Single plate, manual, dry ..	Two-plate, dry, manual.
Tyres	Front—12.00 × 24-16 Ply, 70 psi	Front—16.00 × 25, 24 ply.
	Rear—Twin 14.00 × 24-20 Ply, 75 psi.	Rear—18.00 × 25, 28 ply.
Brakes	Internal expanding, compressed air.	Internal expanding compressed air.
Steering	Hydraulically boosted ..	Hydraulically boosted.
Carrying Cap. (quarry body)	10.6 cyd struck .. .. 13.0 cyd heaped	16 cyd struck 20 cyd heaped
Wheelbase	4 m	4.2 m
Turning circle over front tyres	19.8 m	18 m
Tare weight (Unladen)	14,300 kg (quarry body) ..	19,160 kg
Payload	13,000 kg (quarry body) ..	22,650 kg
Electrical system	24 volts	24 v.
Travel speeds	5 front gears (max. 25 m.p.h.) 1 reverse gear (3 m.p.h.)	9 forward 2 reverse.

Some of the Euclid models are equipped with hydrotarders. On all the models the compressed air is used to operate brakes, horn and windscreen wipers. On some models the clutch operation is also assisted by compressed air for reasons which will be evident from the following.

In order to completely disengage the clutch, the pressure exerted by the clutch spring is with a force of approx. 375 kg. To obtain the necessary 375 kg force, a driver would have to exert approx. 47 kg of force on the clutch pedal to disengage the clutch. This is an excessive amount of effort, and would very soon result in driver fatigue.

An air assist mechanism has been incorporated in the clutch linkages to produce greater part of the required force and prevent driver fatigue.

The air assist mechanism consisting of an air valve and air cylinder exerts a force of 194 kg to one end of the rocker arm when an air reservoir tank pressure of 6 kg/cm^2 is supplied to the air valve. A pedal force of 23 kg is multiplied by the clutch linkage so that a force of 180 kg is applied to the opposite end of the rocker arm. The combined air cylinder and pedal forces (194 kg and 180 kg respectively) are transmitted by the rocker arm to the release bearing cross shaft.

THE HYDRAULIC RETARDER (HYDROTARDER)

This is a device used on some trucks and dumpers, to prevent the speed from exceeding certain limits when travelling a steep downslope and also to produce a braking action on the vehicle. In a way, it acts as a governor. It uses the hydraulic friction to produce the braking action. Unlike the regular brakes, the hydrotarder will not completely stop the vehicle but will slow it down preparatory to stopping with the familiar friction brakes, operated by compressed air or hydraulic pressure.

The retarder essentially consists of a vane type rotor turned by the drive shift, a fixed casing or stator fitted with vanes and an oil circulation system. On some of the Euclid dumpers (FD models), the hydraulic retarder is known as hydrotarder.

There are two types of hydrotarders in use on Euclid, water operated and oil operated. Both types have the same principle

of operation but use different speed reducing fluids in their systems. The water operated unit uses water from the vehicle's engine radiator as the speed retarding medium whereas the oil operated hydrótarder uses oil from its own supply as its speed retarding medium. Its construction is similar to the water operated unit but it uses different components in its system. Water operated hydrotarder systems in use on Euclid equipment are of two types ; the pump system and the gravity system. The difference between the two systems has no bearing on the operation of the hydrotarder.

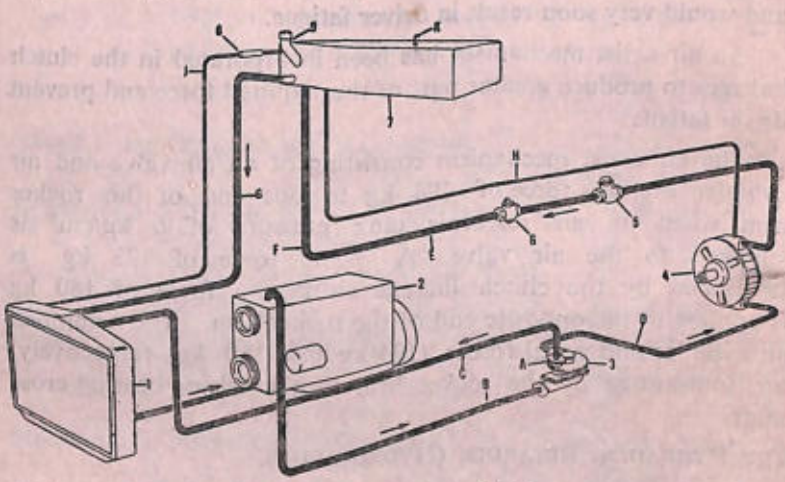


Fig. 7. Hydrotarder on a Euclid dumper.

In fig. 7 it can be seen that there is a continuous flow of water in the system. From radiator the water flows through line *A*, then through *B* to the control valve *3*. If the hydrotarder is not in use as a brake, the water is diverted into the bypass line *C* and return line *F* in which it is carried to the surge tank *7* and through the radiator supply line *G* to the radiator *1*.

Assuming that the hydrotarder is not in use, the water flow is from engine *2* through the valve supply line *B* to the control valve *3* which is closed.

When the hydrotarder is in operation the hydraulic friction heats the oil or water in the hydrotarder and the heat in the fluid is then dissipated by the radiator or oil cooler. As the vehicle speed is reduced the rotation of the hydrotarder rotor

decreases resulting in less retarding action until the point is reached when the pumping action is cancelled. At this point the service brakes must be applied to further slow down or stop the vehicle.

The hydrotarder is especially useful when a heavily loaded vehicle is required to descend long steep slopes. Its use promotes safer operation of the vehicle, reduces driver fatigue and prolongs the life of the vehicle's tyres & brakes.

A coal hauler in use in the U.S.A., is CAT 630 with 60 tons struck capacity. It uses a new concept that automatically adjusts the machine to grade and road conditions. There are 9 different speed variations, but the operator need concern himself with only 3 speed ranges. Controlled by one lever, each range has 3 automatic shifts—torque divider drive, direct drive, and over drive. A dial indicator tells him when to change his speed range for the best travel speed. On good roads, the vehicle hauls heaped loads up 7% grade at 10 km/hr.

Haulpaks

One of the rear dumpers used in the mechanised quarries is the *Haulpak*. It has been used at Kathara Colliery for the first time in India and has many interesting features as follows :—

(1) There are no springs for suspension. Four gas cylinders filled with nitrogen under pressure serve the purpose of springs.

(2) It has short wheel base, resulting in great manoeuvrability and negotiation of sharp curves.

(3) Front and back tyres are of same size, so that ground clearance at the front and back is equal. Moreover, all tyres are interchangeable.

(4) The chassis is trough-shaped to accommodate the trough-shaped body and the weight of the body is evenly distributed over the chassis.

(5) The body is heated by exhaust gases so that when wet rock is carried the rock does not stick to the body and all the contents are unloaded.

(6) Since the wheel base is small the propeller shaft and the drive line are of smaller lengths.

(7) The steering is operated by hydraulic booster. To facilitate the hydraulic steering in case of engine failure, there is an electric motor to assist the steering and also to facilitate the easy movement of the dumper from place to place. The motor is started by a battery.

(8) There are two brakes, one on the wheels, and the other is the torque converter brake.

(9) There is no clutch.

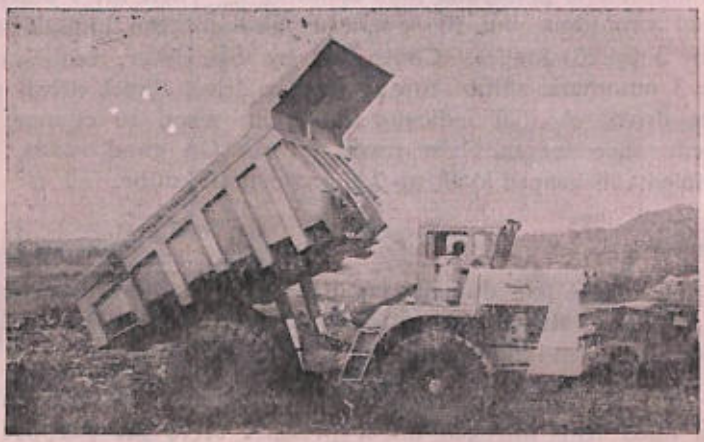


Fig. 8. Haulpak.

(10) Eight speed ranges are possible, but the driver has to manually operate only 4 gear shifts. For each manual gear shift, there is one automatic gear change possible (fluid control) Allison transmission is used.

(11) The most interesting aspect is the maintenance. There are only 4 or 5 grease nipples, which have to be greased only after 500 hours. The other bearings are grease packed.

(12) The body of the dumper is made of a special light weight alloy steel, reducing the unladen weight of the dumper to a great extent, thereby increasing the contents of the rock moved.

(13) The cylinders for the hydraulic rams which lift the body are of larger diameter on the body and smaller diameter on the chassis, so that chances of choking of the larger dia. cylinder are practically eliminated.

The haulpaks are available in 22 ton and 32 ton carrying capacities.

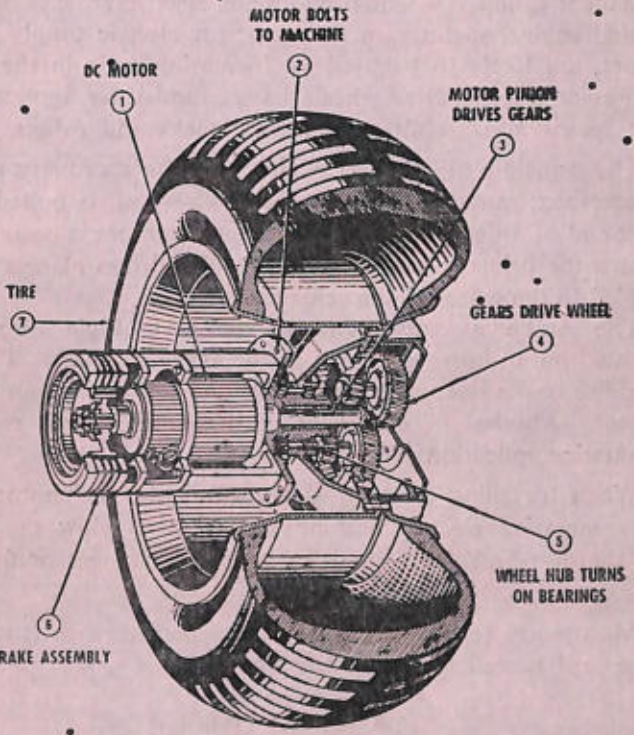


Fig. 9. Tyred wheel for electric traction.

Tyred electric wheel.

An interesting method of communicating torque to the drive wheels is shown in fig. 9 and represents the greatest departure from conventional design in eliminating the complexity of high power mechanical transmission systems by incorporating an electric motor (D.C.) within the hub of each wheel. 400 H.P. wheels have been available for sometime and prototype 1,200 H.P. trucks, incorporating these, are in use. The interest shown in this novel form of traction has encouraged

development of a smaller 100 H.P. electric wheel, which might well find a more general application in opencast haulage equipment in due course. Dumpers, so equipped and receiving electric supply from overhead trolley wires, operate only on a fixed permanent route and are similar to the trolley wire buses in Bombay. For quarries, flexibility of operation on non permanent routes is essential and diesel-electric vehicles, in which a diesel engine operating a generator for electric supply to hub-motors, are likely to be used on increasing scale in the future. At present electric drive wheels have found their application in most recent all-wheel-drive scrapers, trucks and rollers.

The principle of working is simple. The stationary member or gear box, mounted close to the drive wheel, is bolted to the outer end of a dead axle. The rotating armature is geared down to turn the primary reduction shaft which drives planetary gears coupled to the wheel of the vehicle.

The brake assembly consists of a multiple disc brake mounted on an inner extension of the armature shaft. The multiple disc brake is applied by springs and released by an electro-magnet. Absence of electricity in the brake circuit results in automatic application of brake.

When travelling on steep down gradients, the motor assists in regenerative electric braking, sufficient to slow the vehicle, and the disc brakes are used for sudden stoppages or for parking.

Motors are reversible and the same performance is available in forward as well as reverse direction.

8. EXCAVATORS.

An excavator, technically speaking, is any machine which excavates the rock or earth and swings or transports it, within narrow limits, to an adjacent place or dumps it on to a receptacle like a dumper or railway wagon. In this sense, a tractor shovel may well be considered an excavator—the name 'Traxcavator' of a Caterpillar machine aptly conveys the meaning—but, in, earthmoving terminology, the term "excavator" covers machines of the following type :

1. Power shovels like dipper shovels, stripper shovels and back-hoes or pull shovels.

2. Draglines.
3. Bucket wheel excavators.

Power Shovel

A power shovel is a shovel using electric or diesel motive power for its operation, as distinct from a hand-operated shovel. The functions of power shovels are very simple. Basically, these machines lift fragmented rock, and swing it to a different location such as dumpers or spoil heaps.

The main components are :

1. Propelling arrangements consisting of either crawler chains or pneumatic tyres.

2. A deck or cab mounted on a turn table and housing the prime mover, all the controls for operation, cable (wire rope) drums and the operator's seat. The deck or cab can swing through 360° independently of the propelling crawler chains or tyres.

3. Deck swinging mechanism. When the deck swings, all the equipment mounted on it, including the boom and the bucket also swings.

Crawler chain arrangement is invariably adopted for excavators of 1 m^3 capacity and over. The propelling with crawler chain is in the same manner as with bulldozers, tractor shovels etc. and is described in earlier pages.

The capacity of a shovel is specified by the capacity of the bucket it carries and varies from 0.2 m^3 to 50 m^3 and over. In U.S.A. a shovel with 88 m^3 bucket is now under construction. It will have a reach of 140 m and will be powered by 50 motors ranging from $\frac{1}{4}$ to 3,000 H.P. It will be able to move 2,300,000 m^3 of rock per month. The shovels, indigenously manufactured, have capacities of 1.15 and 2 m^3 (Tata-P & H). 2 and 2·3 m^3 (Hind Marion) and 4.6 m^3 (Heavy Engineering, Ranchi).

Diesel and Electric Shovels

Shovels are diesel operated, or electrically operated when the electricity is supplied from outer source. Some shovels are diesel-electric, *i.e.*, all the controls in the shovel are electrically operated and the electricity is supplied from a diesel generator mounted on the deck of the shovel. The smaller capacity

shovels upto 3.5 m^3 are usually diesel operated. Shovels with higher capacities are powered by electric motors, with alternating current at 2,000 to 6,000 volts, the common voltage being 3,300. The current is supplied through trailing cables. Electric shovels are costlier than diesel ones in the first cost but they have few moving parts, no fueling problem, need less maintenance and, therefore, number of spares to be stocked is less. Electric shovels have fans or blowers in the deck to keep the motors cool. Diesel and electric shovels both have small exhaust fans for ventilation of operator's chamber. The electric shovel can operate only within a restricted area due to limiting length of the trailing cable. The trailing cable, which is quite heavy, needs men to handle when the shovel is moving.

Steam operated shovels are now-a-days out of date.

A Dipper Shovel

Different makes of dipper shovels are in use in the quarries. One such shovel of 2 m^3 capacity, Tata P & H make, is described below.

The shovel has a *bucket* (also known as *dipper*) in the front. (Fig. 10). The *boom* is a massive beam, usually wider in the middle. Its lower end is hinged to the deck by means of a pair of brackets called "boom foot." The boom can be moved up and down, but has no side movement and it is supported by a 4 part boom line, *i.e.*, 4 lines of a wire rope called the *boom hoist rope*. One end of the rope is anchored on the '*A*' *frame* ; the other end passes round the pulleys on the boom top, the 'A' frame and finally to the boom hoist drum. The manner in which a rope is passed on the pulleys and finally to the drum is known in technical language as "Reeving the line". During digging operations the boom rests at an angle from 40 degrees to 45 degrees with the horizontal, as per manufacturer's specification.

The middle of the boom accommodates a shipper shaft with a sprocket wheel for the endless crowd chain and a pinion. The dipper stick, having teeth on the lower side, rests on the pinion and can be moved to and fro like a rack and pinion combination.

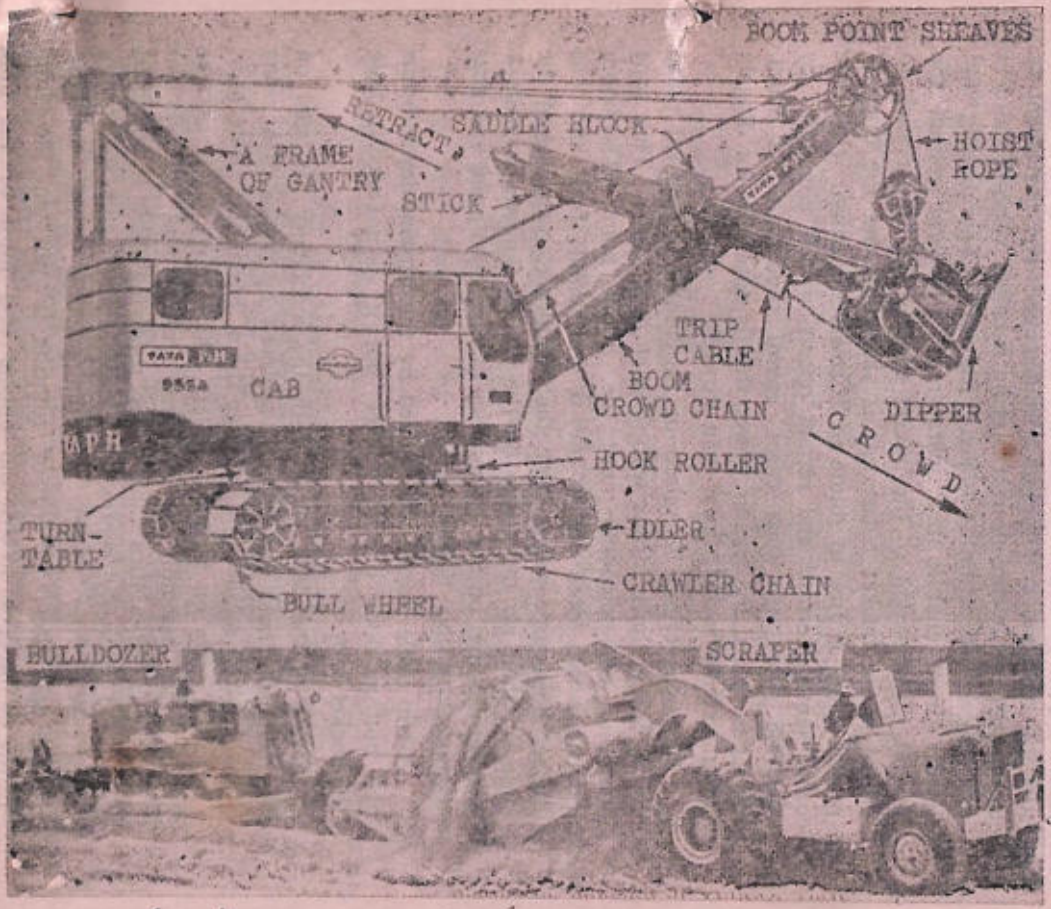


Fig. 10.

Top—Dipper shovel

Bottom—A scraper pushed by a bulldozer

To one end of the dipper stick, is hinged the bucket (or dipper) and the other end is bumper-shaped, so as to prevent slipping of the dipper stick through the *saddle block* or the boom. The shipper pinion acts as a fulcrum for the dipper stick. The saddle block is attached to the boom and acts as a guide to the dipper stick which passes through it. The rotation of the shipper shaft pinion causes, in turn, the travel of the dipper stick by the rack and pinion arrangement. The movement of the bucket in the direction of digging is called "Crowd" and in the opposite direction is called "Retract". The digging and retract drum enables the operator to crowd or retract the bucket. The weight of the bucket enables the dipper stick to crowd if the bucket hoist rope is slackened and the digging/retract drum is free to rotate in "Crowd" direction. (In some other shovels, crowd and retract operations are controlled by separate ropes. In such cases, crowd rope is on a drum, which is geared for greater power that is necessary for digging and the retract is 50% faster than crowd.)

The bucket can be raised or lowered in a vertical plane by a bucket hoist drum with *bucket hoist cable*. This rope has one end anchored near the boom point. The boom point has two loose pulleys on a horizontal shaft and the bucket hoist rope passes from the anchorage, over one of the pulleys ; then *via* the pulley on the bucket (known as *padlock sheave*) it comes on the other pulley of the boom point and goes to the bucket hoist drum.

A trip cable (rope) when pulled by the tripping lever, opens the bottom door of the bucket but it is not used to close the bucket door.

Swing of the deck, along with the machinery mounted on it, the bucket and the boom is achieved by means of a swing pinion, which meshes with a large diam. horizontal *swing gear*. The deck rests on tappered rollers which move on a circular track. The assembly of tappered rollers and the circular track is called a turn table. The revolving deck is stabilised and held properly centred by hook rollers and the centre shaft. With the object of counteracting tilting tendency during operation centre of rotation is put in the front centre of the deck, so that the engine and most of the deck

machinery are behind it. Counterweights are usually placed in the rear of the revolving unit in the deck.

The shovel is stationary during digging and loading at the face, only little movement at intervals being necessary to be near the fragmented rock. The movement is possible with the help of *crawler chains* to which motion is transmitted through the *bull wheel*.

Never lower the boom tip to a point below the plane on which the machine is resting. The angle of pull on the boom hoist cables will be such that the boom may collapse before lifting off the ground.

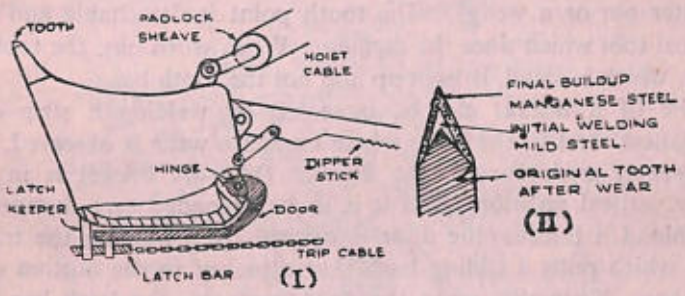


Fig. 11. Bucket of a dipper shovel.

Diesel shovels have a small petrol-driven electric generator (110 V) for lighting. Electric shovels have a small transformer, about 5 KVA, for lighting at 110 V. Some electric shovels are, however, equipped with petrol driven electric generator for emergency lighting.

Bucket of a Shovel

The bucket is a cast or welded steel box open at the top and closed at the bottom by a hinged door. The front top edge of the bucket has a set of four or more detachable teeth which can be fixed in position by wedges. The teeth which are of manganese steel are subjected to maximum abrasion during digging. When one side is worn they can be removed and reattached to the bucket in reverse position. When worn out they can be rebuilt by welding with an initial build up of mild steel and a final surface of hard manganese steel. Building it to correct size and edge is essential. The life of the

original tooth, including the life after reversal of position, is 50 to 70 hours in sand-stone. Thereafter one re-building gives a life of another 30 to 40 hours. Re-building is uneconomical after a total life of 200 to 250 hours. For coal and iron ore the total life may be nearly 800 hours, including life after re-building. Blunt teeth would retard progress of digging and would be heavy on the engine. Spare sets of teeth are always essential.

The digging tooth of some big shovels (3.5 m^3 and above) is made in two parts. The major part is called the *tooth base* and is fixed in position on the lip of the bucket by a wedge as described above. To the tooth base is attached a tooth point by a cotter pin or a wedge. The tooth point is detachable and is the real tool which does the digging. When worn out, the tooth point, which is small, is built up and not the tooth base.

Life of a bucket can be increased by welding a strip of manganese steel to the parts where excessive wear is observed.

Opening and Closing the Bucket Door—A bucket is in a nearly vertical position when it is to be unloaded on a dumper. To unload a bucket the door is opened by a pull on the trip cable which pulls a sliding latch bar attached to the bottom of the door. Normally, when the door is closed, the latch bar is held securely in a latch keeper. As the latch bar is pulled out of the latch keeper by the pull of the trip cable, the heavy hinged door opens due to its own weight as also the weight of the load in the bucket. When the bucket is lowered by the hoist cable, so that the teeth are nearly in a horizontal position, the door closes due to gravity forcing the latch bar into the latch keeper.

With the door closed, the bucket can be used to tidy up the load on the dumper.

A 2 m^3 shovel can be used for coal loading by using a 3.5 m^3 bucket, if necessary, as coal is light.

Clutch, Transmission and Controls

Fluid coupling in place of the mechanical clutch and torque converter for transmission is the normal practice on most of the shovels. On the P & H shovels an arrangement known as magnetorque is provided to transmit mechanical power from one rotating shaft to another driven shaft. A magnetorque may be considered to be a magnetic coupling. This magnetic coup-

ling transmits power to the swing gear train. It has two concentric rotating members. The outer member is a drum carried on a supporting spider and is driven by a motor or engine. On the inside circumference it has field coils which are excited with direct current and produce magnetic lines of force. The inner member is essentially a circular steel rim supported by sturdy fins. A uniform air gap exists between the outer member and

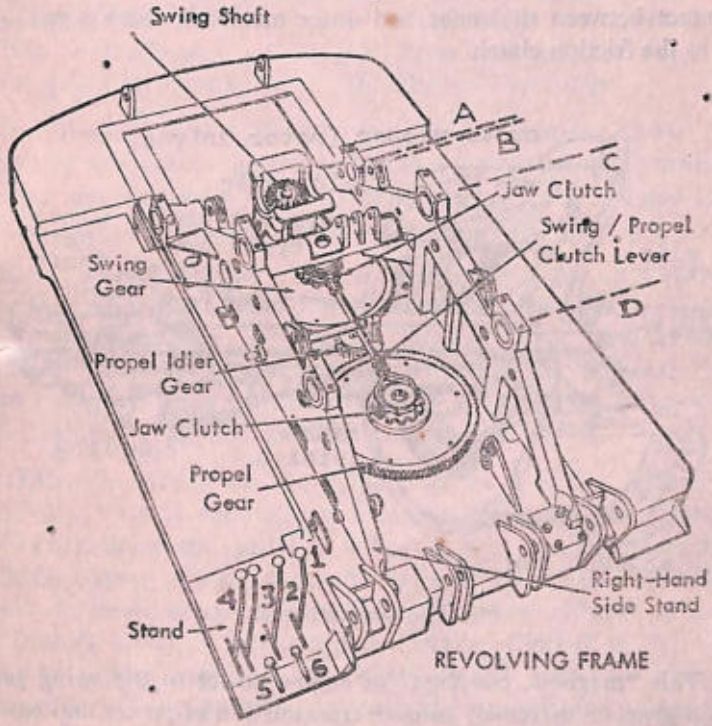


Fig. 12. Major assemblies on the deck of a dipper shovel. (Tata P & H) Bearing blocks, A for jackshaft ; B for intermediate hoist shaft ; C for hoist drumshaft ; D for digging drumshaft.

the rim of inner member and there is no mechanical contact between the two. When excitation is applied to the rotating outer member, voltage is induced in the steel rim of the inner member and this causes eddy currents to flow in the rim. This establishes a magnetic field which tends to follow the rotating magnetic field of the outer member. Hence the inner

member rotates in the same direction. The greater the D.C. winding current (excitation current) becomes the greater will be the magnetic force and the driven member will rotate faster, approaching (but never equalling) the speed of the driving member. The *slip* or the difference in R.P.M. between the inner member and the outer member is dependent upon the amount of excitation supplied to the outer member and the load on the inner member. Since there is no mechanical contact between the inner and outer members, there is no wear as in the friction clutch.

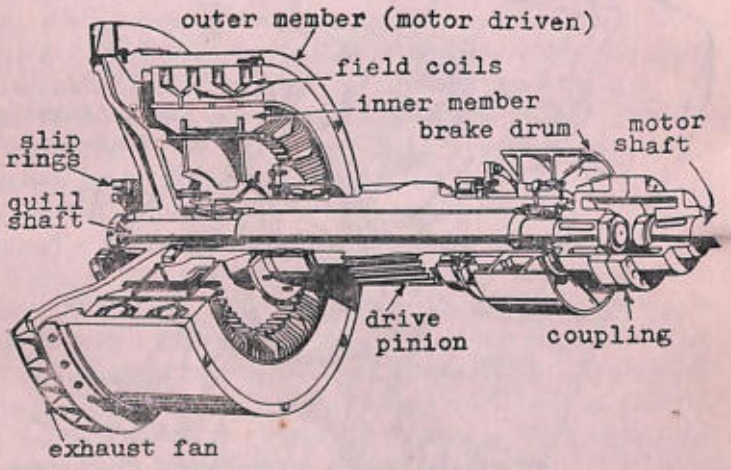


Fig. 13. Magnetorque.

This "magnetic coupling" of engine power to the swing gear train gives an extremely smooth transmission of power and easily controlled speed to the swing motion. No mechanical braking is required to stop the swing motion. With the machine swinging in one direction, moving the swing control lever in the opposite direction cuts off power to the coil of one unit and energizes the coil of opposite unit. This action (called "plugging") results in the smooth action of the "magnetic coupling" to act as a brake in stopping.

Ward-Leonard control is the standard practice on electric and diesel-electric shovels of 3.5 m^3 and higher capacities.

The controls in a shovel are as follows :

1. Swing Clutch Lever
2. Digging Lever
3. Hoist Clutch Lever
4. Boom Hoist Lever
5. Hoist Brake Pedal
6. Digging Brake Pedal
7. Hoist Brake Lock
8. Digging Brake Lock
9. Propel Brake Control
10. Steering Control
11. Swing and Propel Shifter Lever.
12. Swing Lock Lever
13. Engine Clutch Lever
14. Boom Hoist Pawl Control
15. Pressure Regulating Valve
16. Dipper Trip Lever

Out of these, during operation at a bench when the shovel is stationary and has to dig the rock, the important controls which need to be frequently handled are nos. 1, 2, 3, 4, 5, 6 and 16. All the levers are hand-operated and the pedals, foot-operated.

Each gear train (or motor in the case of an electric shovel) is equipped with a powerful brake, operated by a brake pedal. In the case of electric shovels, such brakes are normally in the "On" position, but can be released by electro-magnets when the motor or gear has to work. If power fails brakes are applied automatically.

Hydraulic system on the same lines as used for hydraulic braking of cars is adopted for control-stand levers and pedals. The work done by master cylinder in the familiar hydraulic braking system of cars or trucks is done by a similar device called *compensator* and the work done by wheel cylinder in the car braking system is performed by "brake cylinder" or "clutch cylinder". Movement of the control-stand levers or pedals connected to the compensators causes the compensator piston to move. Movement of this piston forces hydraulic brake fluid through the tubes connecting the compensators with the various brakes and clutches. The pressure of the hydraulic fluid in the connecting tubes moves the piston in the brake or clutch cylinder. This movement of the piston causes the clutch or brake to engage the drum and the force applied to the brake or clutch varies with the travel of the lever or pedal. Springs in the compensators and cylinders return the pistons to their normal positions as soon as the control-stand lever or pedal is released.

Air-over-hydraulic control system is employed for the hoist brake, the digging (crowd) brake and for some other controls. Compressed air is used as a booster in the hydraulic system and provides the advantages : (1) It enables the operator to apply more pressure to the brake than would be possible with the hydraulic control alone ; (2) Brake band travel is increased, permitting not-too-accurate adjustment of the former.

LIFE OF CABLES (ROPES)

In practice, the life is observed as follows on most of the shovels in coal mining areas (Figures in bracket for 1.9 m^3 Tata P. & H. shovel).

Boom rope (19 mm × 106 m)	..	1,000 hr ;
Hoist rope (28 mm × 33 m)	..	80 to 100 hr ;
Dipper trip rope (10 mm × 8 m)	..	30 to 40 hr.

The life is much influenced by the make of the ropes.

CYCLE OF LOADING OPERATIONS OF A SHOVEL

After a bucket is emptied into a dumper,

(1) Slacken bucket hoist cable (generally known as hoist cable) with the crowd drum braked. The bucket swings down on the hinge due to gravity and the bucket door closes ;

(2) Retract the bucket slightly ;

(3) Swing the boom back to the digging position ;

Operations 1, 2 and 3 are generally carried on during the swing of the bucket to minimise time.

At the digging position,

(4) Slacken the hoist and crowd the bucket simultaneously into the material ;

(5) Hoist the bucket. It moves through an arc in the rock or earth and fills up. As the bucket is filled, retract ;

(6) Hoist the bucket to the required height and swing ;

(7) Crowd a little, slackening hoist cable slightly and position the bucket over the dumper ;

(8) Pull the trip cable. The bucket door opens and load falls through.

That completes the cycle of loading by a shovel.

Working Conditions Suitable for a Dipper Shovel

A dipper shovel works in the quarry standing on the floor of the bench to be excavated. As a shovel digs satisfactorily at a bench, it is necessary to provide the latter by cutting a box cut or a trench before a shovel can be used for digging. The rock is loaded on dumpers which have, therefore, to travel in the quarry. A dipper shovel is, therefore, suitable,

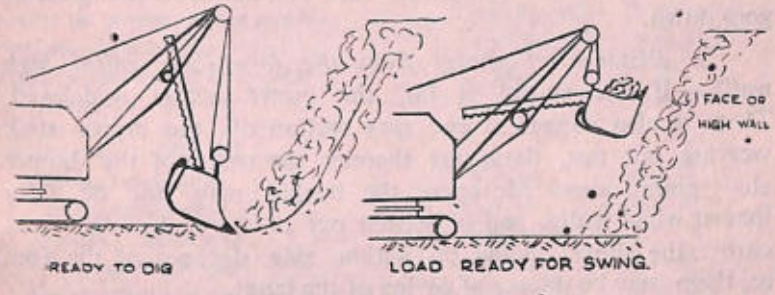


Fig. 14. Dipper shovel at the face.

(i) For nearly dry working conditions ;

(ii) Where compact roads with mild gradients can be provided in the quarry ; and

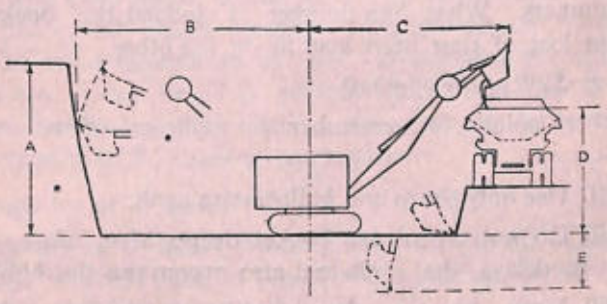


Fig. 15. Working ranges of a dipper shovel.

(iii) Where the dumping yard is far away, requiring use of dumpers.

A dipper shovel is not suitable for cutting below its floor level.

PERFORMANCE OF A SHOVEL

The performance of a shovel depends on the following factors :—

1. *The nature of fragmentation of rock* :—If the rock is well fragmented, the crowd and digging is fast and bucket is fully loaded. Bucket fill is low when cleaning up the remains of blasted rock near the back break after the bulk of the blasted rock is removed and the overall bucket filling factor goes down.

2. *Distance of shovel from the face* (*also called high wall*) :—If the shovel is far, the crowd action is delayed. There is also danger of end rack section of the dipper stick wearing out fast, damaging thereby the teeth of the shipper shaft pinion also. Moreover, the bucket may not be fully loaded when it digs and is hoisted out of the rock. As stated earlier the shovel should be within safe distance of the face as there may be danger of caving of the latter.

3. *Swing of the deck* :—Large angle of swing should be avoided. An ideal angle is 45 degrees but even 90 degrees is good enough for efficient performance. For quick cycle of operations, it is advantageous to position the dumpers on either side of the shovel, and a traffic man is necessary to guide the dumpers. When one dumper is loaded the bucket can, without loss of time. start loading on the other.

4. *Skill of the operator.*

Other points to remember *for efficient shovel operation* are :—

(i) Use only sharp and built-to-size teeth.

(ii) Do not crowd the bucket deeper than what is necessary. It delays the cycle and also overstrains the hoist rope.

(iii) Move up the shovel while waiting for dumpers. Frequent movement of shovel should not be necessary.

(iv) Keep the benches dry.

(v) Use a dozer at intervals to scrape and pile up the fragmented rock near the high wall for convenience of bucket loading. The spillage during dumper loading should also be pushed by dozer to high wall.

TABLE 2

Leading specifications of some dipper shovels

	Tata P & H 955A	EQC—4.6
Shovel capacity (dipper size)	1.9 m^3	4.6 m^3
Boom length	8 m	10.5 m
Dipper stick length	5.8 m	7.3 m
Lifting (hosting) force	34 t	45 t
Pressure on ground during motion ..	0.88 kg/cm^2	2 kg/cm^2
Travel speed on level surface	1.25 km/hr.	0.45 km/hr.
Weight (without counterweights) ..	68 t	160 t
H.P. (electric, 3.3 kv)	125	250 kw.
Working ranges (Fig. 15)		
A. Cutting height max.	9 m	10.2 m
B. Cutting radius max.	11.5 m	14.4 m
C. Dumping radius max.	9.8 m	12.7 m
D. Dumping height max.	6 m	6.3 m
E. Cutting depth below crawler level ..	2.7 m	3 m

The special features of EQC-4.6 include a hoisting mechanism for its boom, by means of which the latter can be raised or lowered as and when required without calling for a separate crane to do this job, as is usually done with other machines. The main motor of the M.G. set is fed from 3.3 or 6.6 kv 3 phase, 50 cycles supply through an oil curcuit breaker. The main mechanism of the excavator-hoist, crowd and swing are driven by D.C. motors fed from the generators of the M-G set. The main motor has a short time overload capacity of 2.2 times of rated capacity.

Loading capacity of a shovel

The capacity of a bucket is always expressed in m^3 of loose material and is fixed. Let us consider the loading capacity of 2 m^3 shovel in *well fragmented rock*.

Time for the cycle of loading one bucket .. .. $= \frac{1}{2}$ minute.

Number of buckets loading normally into a dumper (depending upon carrying capacity) $= 5$ (and sometimes 6).

Total time for loading a dumper ..	=3 minutes.
Total buckets (also called passes) loaded in 1 hour of 45 minutes' loading time.	=75 to 85.

80 *passes in* 1 *hour* is a good performance.

Passes per shift of 8 hours (7 working hours)=$7\times80=560$.

500 passes in a shift of 8 hours is a good performance. Considering 5 buckets per dumper, it gives 100 dumper loads. Performance is slower during dark hours and in a day with 2 working shifts, each of 8 hours, the total passes are nearly 500+450=950 approximately, or $\frac{950}{5}=190$ dumper loads. In the following calculations a day is considered of 2 working shifts.

The minor stoppages of a shovel in a shift are due to moving the shovel to a safe place when blasting, ordinary moves upto the face periodically, taking fuel on board, and shifting trailing cables, etc.

Shovel Output

A 2 m^3 bucket may load slightly more than 2 m^3 of earth in a few cases. But, generally it loads slightly less than 2 m^3 when rock is to be loaded. *On an average*, the bucket fill factor is 0.8 in well fragmented rock and goes down considerably with a bad operator and bad operating conditions.

Loose m^3 per bucket on an average	$=2\times0{\cdot}8=1.6$ m^3
Considering 100 m^3 solid to become 140 m^3 loose on blasting, 1.6 m^3 loose	$=\frac{1.6\times100}{140}=1.14$ m^3 solid.
Solid m^3 per pass	$=1.14$ m^3
Solid m^3 per shift (500 passes)	$=1.14\times500=570$.
Solid m^3 per day (950 passes)	$=1.14\times950=1083$ m^3, say 1070 m^3 (1400 cyd.)

Considering some time lost on repairs during the week, a 6 day week should produce, $1070\times5\frac{1}{2}$ days=5885 m^3 solid.

Weekly performance of 5800 m^3 is reasonable.

Monthly performance :

25 working days minus equivalent of 1 day, say, lost on major repairs gives 24 working days or 4 working weeks :

Solid $m^3 = 5800 \times 4 = 23200\ m^3$.

An average of 23,000 solid m^3 (nearly 30,000 cyd.) per month is a good performance.

Yearly performance :

During 8 dry months, the performance is good, but during 4 monsoon months the performance is nearly 75% of the usual dry period performance.

Performance of 8 dry months is $8 \times 23000 = 1,84,000\ m^3$ (solid)

do 4 wet months is $4 \times 23000 \times \frac{75}{100} = 69,000\ m^3$

do for one year is $1,84,000 + 69,000 = 2,53,000\ m^3$ (nearly 3,31,000 cyd.)

The foregoing assumes rock of good fragmentation and shovel in well maintained condition. Under the circumstances met with in a quarry, a 2 m^3 shovel should give nearly 253,000 m^3 (solid).

In foreign countries where standard of maintenance is good and availability of spare parts is not as serious a problem as it is in India, the annual performance of a 2 m^3 shovel is taken as 3,82,000 m^3 (0.5 million cyd.) solid.

To SUMMARISE, under conditions existing in India, the loading capacity of a 2 m^3 shovel in good condition and well fragmented rock is as follows :—

per hour	..	80 passes.
per shift	..	500 passes.
per day (2 shifts)	..	950 passes or 190 dumper loads or 1070 m^3 (1400 cyd.) solid.
per week	..	5,800 m^3.
per month, dry	..	23,000 m^3.
per year	..	2,53,000 m^3.

Loading capacities or performance of other shovels may be considered as follows :

3.5 m^3 capacity	..	400,000 m^3 per year.
4.5 m^3	..	550,000 m^3 per year.

Pull Shovel or Hoe

Fig. 16 shows a pull shovel, which is also known as a hoe, back hoe, drag shovel or ditching shovel. This is not used in our coalfields but one such machine manufactured

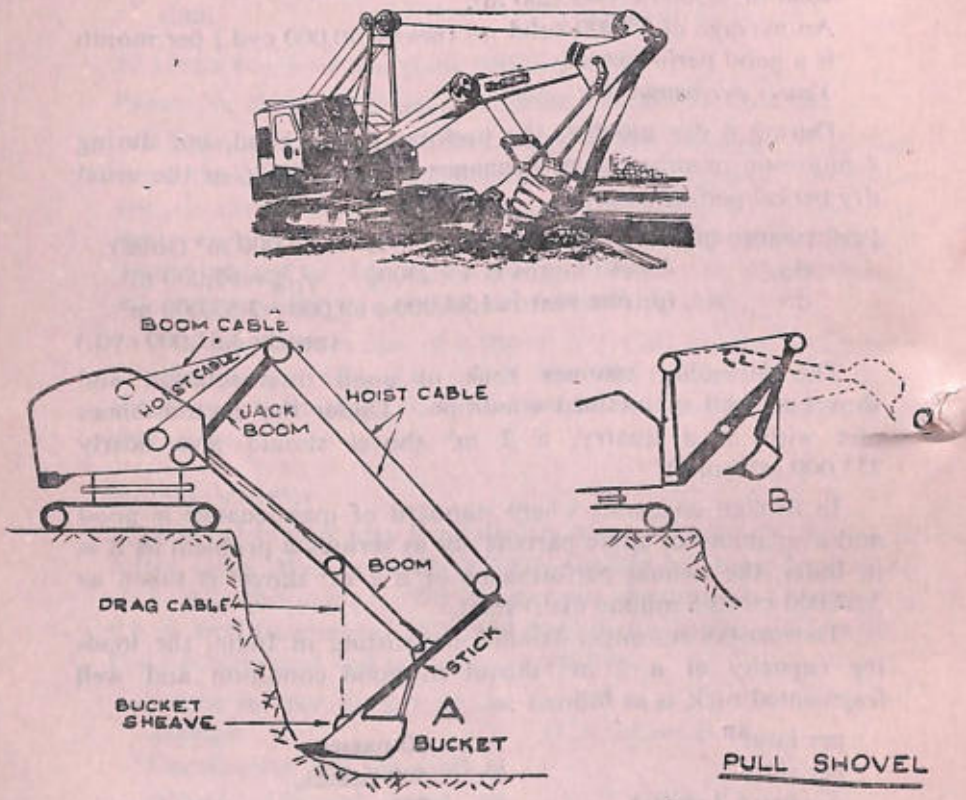


Fig. 16. Back hoe or pull shovel.

by Thomas Smith & Sons has been in use in a manganese mine in Supapita area (Mysore). It is used for loading dumpers and its best application is for digging below the level on which it stands. The shovel and the dumpers can stand at a higher level free from water and mud on the quarry floor. As the attachments to the bucket are by dipper stick and not

by cables, the bucket is under positive control of the operator and, therefore, suitable for hard digging.

The shovel is used for stripping top soil, and making shallow cuts and trenches upto a depth of 3.5 to 6 m. Compared to dipper shovel, the hoe is slower in digging, and less efficient for loading trucks.

In deep digging, the face should be kept fairly straight and the shovel should be as far back from the edge as possible, otherwise there may be danger of caving of the edge.

Clam Shell

This works in the same manner as a crane, but has a bucket instead of the hook of a crane. Its controls and attachments are basically the same as those of a shovel and crane but the bucket is of special construction. It consists of two jaws hinged on a movable bar, the main shaft, and pulleys on brackets. The bucket jaws have teeth which are detachable and reversible as in a shovel bucket. Some buckets have flat or curved lips, with or without side cutters. The more resistant the earth to be dug, the heavier the bucket must be in relation to its capacity. The bucket digs in the material by its weight. When the hoist rope is coiled on the hoist drum, the jaws close and the bucket, filled with the material grabbed in it, is lifted up by the hoist cable. The material is shifted to another site or dumped on trucks by a swing of the boom.

WORKING CONDITIONS FOR CLAM SHELL

(1) Clam shell can dig below, at, or above the level of the machine.

(2) Clam shell is generally used for rehandling loose material such as coal, sand, ash, gravel, etc.

(3) In alluvium and unconsolidated rocks, it can dig deep, narrow, straight sided excavation and is useful for cutting foundation, trenches or pier holes. In recent years this has been used for removal of muck and blasted rocks in shaft sinking.

(4) It is also used for handling bulky objects, such as boulders, stumps and logs, and chaining of the object to the bucket in such cases is necessary.

(5) Loading and unloading of trucks is often done by a clam shell.[a]

It does not pull or push loose material to other position as it digs. It is slower in action than a shovel or a dragline. The digging is done directly under the boom point and the radius of digging is limited by the swing of the boom.

Clam shell is not used as an excavator in the mechanised quarries in our coalfields.

Dragline

Kurasia was the first colliery in India to use dragline in mechanised quarry of coal. The dragline, 11.5 m^3 capacity, was put into commission in 1960. Draglines have not been used in iron ore quarries.

A dragline is an excavator consisting essentially of a revolving deck, travelling mechanism, a long light crane boom, and a special type of bucket held in position and controlled by cables. One end of the boom carries a large sheave for the rope (cable) of the bucket. At the foot of the crane is a fair-lead which is a device to line up cables, so that the latter can wind smoothly on a drum. The boom is of lattice construction of angle iron, but very large booms have sometimes aluminium sections near the top end. Additional sections, usually 1.5 to 3 m long each, can be coupled in between the upper and lower sections to get extra length (Fig. 17).

Dragline buckets are of different construction as compared to the dipper shovel buckets. They are light duty type for soil or rehandling a loose stock pile and mud, but heavy duty type for blasted coal and sandstone. Sometimes the buckets are perforated, so that water drains out from the load as the loaded bucket is being swung. Bucket is held in position by cables, and when lowered in the earth, it is dragged by cables for loading. Hence the name "Dragline."

A dragline is electrically operated at high voltage A.C. Diesel operated draglines are of small capacities, generally upto 3 m^3. The controls on electric draglines are always of Ward Leonard type.

WALKING DRAGLINE

A dragline is crawler mounted or has a "walking mechanism" for its movements. The walking mechanism is operated by cams or cranks, and the machine moves by eccentric movement of shoes on each side. The process is similar to that by which a man would move on the ground with his hands and knees. He

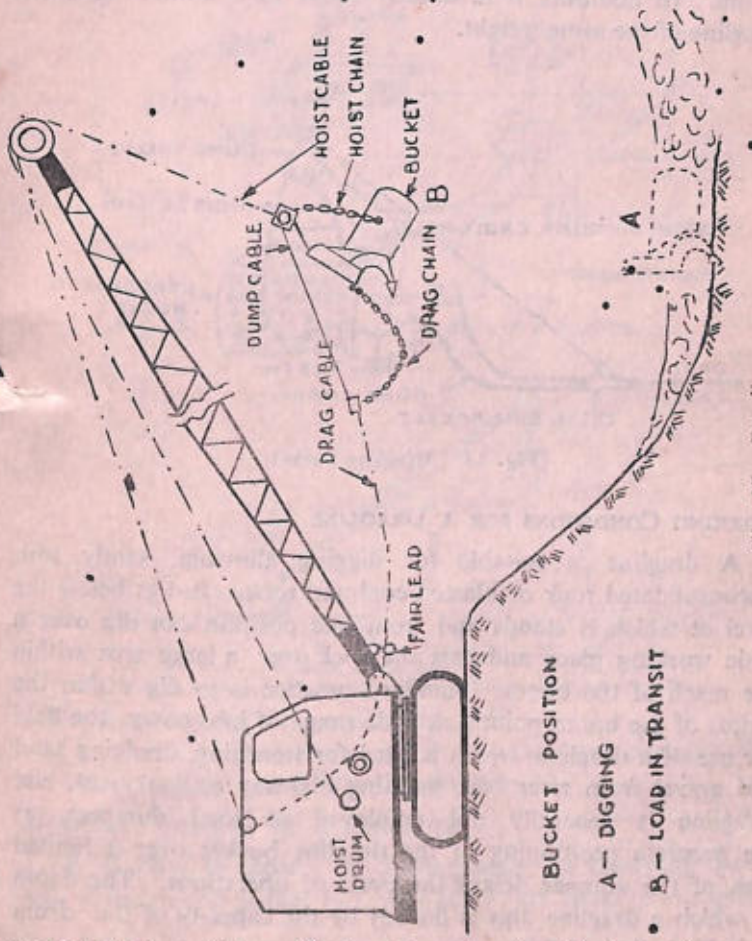


Fig. 17. Dragline.

would rest his weight on his hands and swing his body forward with feet dragging; then he would rest on his knees and toes while he moves his hands forward again. Mechanical movements similar to this process propel the dragline. The degree of mobility

of a walking dragline is nigh and gradients upto 8% are easily overcome. Large draglines usually have walking mechanism. Such dragline can travel safely on loose banks and blasted rock because its ground pressure is low and it can turn to walk without exerting any side thrust. Walking dragline is not suitable for use on quarry floors because it can only walk away from the boom. In addition, it is usually wider than crawler mounted dragline of the same weight.

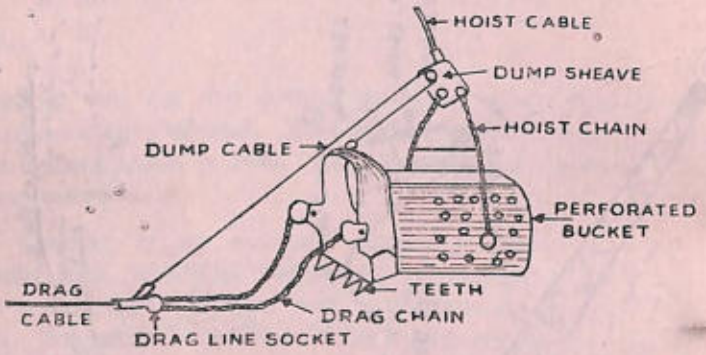


Fig. 18. Dragline bucket.

Working Conditions for a Dragline

A dragline is suitable for digging alluvium, sandy soil, unconsolidated rock or blasted coal and rock. It digs below the level at which it stands and from one position can dig over a wide working place and cast the rock over a large area within the reach of the boom. Standard practice is to dig within the radius of the boom point. A wide range of jobs covers the field for use of a dragline, which is used for trenching, dredging sand and gravel from river bed, building highway embankment, etc. Dragline is generally not employed to load dumpers, as the accurate positioning of the dragline bucket over a limited area of the dumper delays the cycle of operations. The depth to which a dragline digs is limited by the capacity of the drum to hold the hoist cable.

If the trench or area over which a dragline has to work is very wide, and the boom is not long enough, the earth has to be rehandled. Alternatively, two or more draglines are used

in tandem and the machines stand on benches having different levels, so that a lower-level machine feeds one at the higher level.

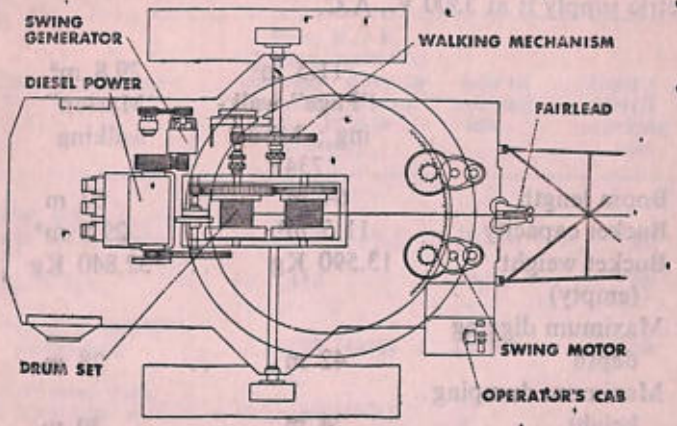


Fig. 19. Typical layout on the deck of a dragline.

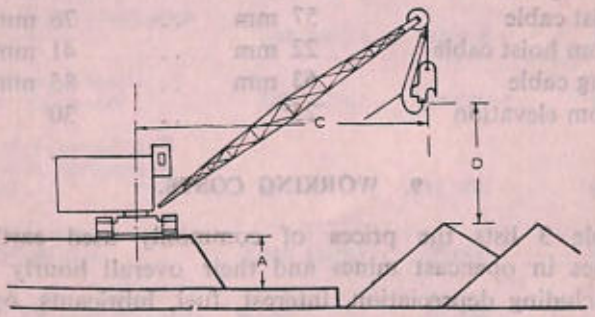


Fig. 20. Working Ranges of a Dragline
A—Digging depth. C—Dumping and digging radii,
D—Dumping height.

Loading Capacity of a Dragline

A dragline is capable of dealing with the following quantities of rock/earth (solid) in a year (12 to 14 hours' work daily).

Bucket Size	Million m^3
4.5 to 7.5 m^3 (6 to 10 cyd) ..	0.25 to 0.75
11.5 to 15 m^3 (15 to 20 cyd) ..	1.5 to 1.7
23 to 30 m^3 (30 to 40 cyd) ..	3 to 4.5

Important specifications of 11.5 m^3 *and* 29.8 m^3 *draglines used in some coal quarries of N.C.D.C.* Both are electric and the electric supply is at 3300 V., A.C.

	11.5 m^3 "Page" walking ; Model 734E		29.8 m^3 "Marion" walking
Boom length	64 m	..	67 m
Bucket capacity	11.5 m^3	..	29.8 m^3
Bucket weight (empty)	13,590 Kg	..	32,840 Kg
Maximum digging depth	42 m	..	28 m
Maximun. dumping height	24 m	..	30 m
Dumping radius	66 m	..	61 m
Driving motors	1,000 H.P.	..	2,500 H.P.
Hoist cable	57 mm	..	76 mm
Boom hoist cable	22 mm	..	41 mm
Drag cable	63 mm	..	85 mm
Boom elevation	25°	..	30°

9. WORKING COSTS.

Table 3 lists the prices of commonly used earthmoving machines in opencast mines and their overall hourly working cost including depreciation, interest, fuel, lubricants, operator's wages, cost of repairs and spares, etc. Here the hourly depreciation (D) is calculated as total cost of machine divided by the total life in *working hours*. But the depreciation accrues even if the machine is not working for want of repairs or for any other reasons. Depreciation equivalent to 15% D is recommended by some authorities for the non-worked hours to calculate the cumulative depreciation and arrive at the depreciated value of the machine. The figure of cumulative depreciation is significant when preparing cost sheet at the month-end or for selling away the machine before its total life in terms of working hours is over. The practice of calculating depreciation however varies with individual operating Companies.

TABLE 3
Working cost

Name of machine	Price F.O.R. Calcutta including customs duty in thousand rupees. (duty rate in bracket)	Life in working hrs.	Hourly overall working cost
Motorgrader 125 H.P. with scarifier and 3.7 m blade ..	247 (43%)	10,000	80
Road roller 10 ton ..	115	12,000	35
Bulldozer crawler mounted, 230 H.P., power-shift ..	470 (43%)	10,000	180
Ripper attachment with shanks for above dozer ..	50 (15%)	..	..
Rear dumper 15 t payload with indigenous body ..	403 (80%)	12,000	120
Rear dumper 25t payload with indigenous body	565 (80%)	12,000	168
Crane, 20t lifting capacity, tyre mounted, 9m boom ..	587 (43%)	18,000	116
Dipper shovel, capacity 2m^3 crawler mounted, indigenous elec. or diesel	826	15,000	200
Dipper shovel, capacity 3.5m^3 crawler mounted, elec. ..	2,640 (15%)	20,000	460
Dipper shovel, elec., 4.6m^3 model EQC—4.6 of H.E. Corpn.	2,000	20,000	
Scraper or Payscraper, 13.5m^3 heaped capacity, 260 H.P. ..	562 (43%)	12,000	164
Dragline 11.5m^3 bucket capacity		..	1,300

10. BUCKET WHEEL EXCAVATORS ; NEYVELI LIGNITE PROJECT

A Bucket Wheel Excavator (Fig. 21) is perhaps the most spectacular of all the excavating machines. It is used for the excavation and removal of soft or unconsolidated overburden, coal, and soft ores in benches of upto 90 metres.

The bucket wheel excavator essentially consists of 3 units :—

(1) The bucket wheel,

(2) The connecting belt to the main crawler mounted carriage, and

(3) The loading belt which disposes of the excavated material continuously into rail wagons or conveyor belts through hopper cars.

THE BUCKET WHEEL

This may be compared with the Persian wheel used on large wells and is a large wheel known as bucket wheel with 6 to 12 buckets. The bucket has a lip with teeth for cutting the rock. The teeth are of manganese steel or high carbon steel faced with tungsten carbide for hard rocks. The maximum life of teeth with t.c. inserts is 250 hours in soft sandstone at Neyveli. The teeth can be brazed 6 to 8 times ; thereafter the parent metal gets diluted and does not permit of further brazing. The bucket capacities range from 50 litres in small wheels to 3,600 litres and more in large wheels. The wheel is mounted at the end of a boom which can be (a) pushed out or pulled back (b) lowered or raised (c) swung through 90° in a horizontal plane depending upon design. The dumps/min. by the buckets vary from 70 in small models to 27 in large models. The hourly capacity of an excavator is equal to the total volume of all buckets × wheel speed per hour. The hourly capacities of small models is only 100 m^3, but in large models it is nearly 4,000 m^3. The bucket wheels are fitted with shock absorbers.

The excavators are all crawler mounted. Fig. 21 shows the relative positions of the various units of a bucket wheel excavator

ROCK SUITABLE FOR BUCKET WHEEL EXCAVATOR

The most suitable material for bucket wheel excavator is sand. For this reason these excavators of small sizes are used in South West Africa in alluvial strata of diamond deposits. Cost per m^3 of rock removal is lowest by bucket wheel excavator as compared to dragline or shovel. The bucket wheel excavator is not suitable for abrasive rocks. The American practice now-a-days is to use bucket wheel excavators for the soft weathered upper portion of overburden and use the shovel or dragline in lower tougher portion after blasting.

Methods of working with a bucket wheel excavator

A bucket wheel excavator is employed to operate on (1) full block method, or (2) the lateral block method.

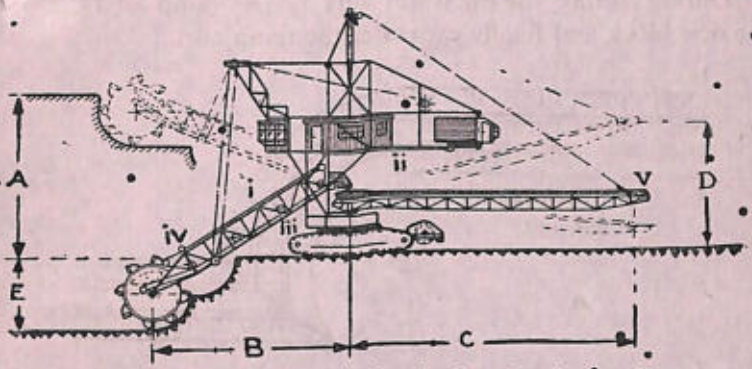


Fig. 21. General arrangement of a bucket wheel excavator. (i) Boom which can be raised and lowered. (ii) Superstruc-ture with machinery houses, ballast and winches. (iii) Under carriage with crawler travel gear, control equipment, cabins for switch gear, workshops and crew. (iv) Operation cabin, (v) Discharge end.

Working Ranges

A—Cutting height B—Cutting radius C—Dumping radius

D—Dumping height E—Cutting depth,

FULL BLOCK METHOD

This is commonly adopted and is shown in Fig. 22. The position of the excavator is shown by the numerals 1, 2, 3. Fig. 22A shows the full block method as practised with a bucket wheel excavator having a boom that can be pushed out and pulled back, *i.e.*, capable of thrust forward. Fig. 22B shows the full block method as practised with an excavator having a bucket wheel boom that can be lifted and lowered but has no thrust movement.

Where selective winning of minerals is not necessary, full block method is employed so that the bucket wheel cuts in full blocks and in parallel operation with cuts which are made as wide as possible. The blocks removed by giant bucket wheel excavators have a width of 45 m to 100 m. This large width of cut which can be maintained along the whole face, not only reduces

the travel way for the crawlers, and consequently, the time required for this displacement, but also the shifting work required for the transport system used to dispose of the mineral. The method is therefore simple and economical.

During cutting, the excavator cuts its own ramp of 1 : 20 into the new block and finally excavates the ramp also.

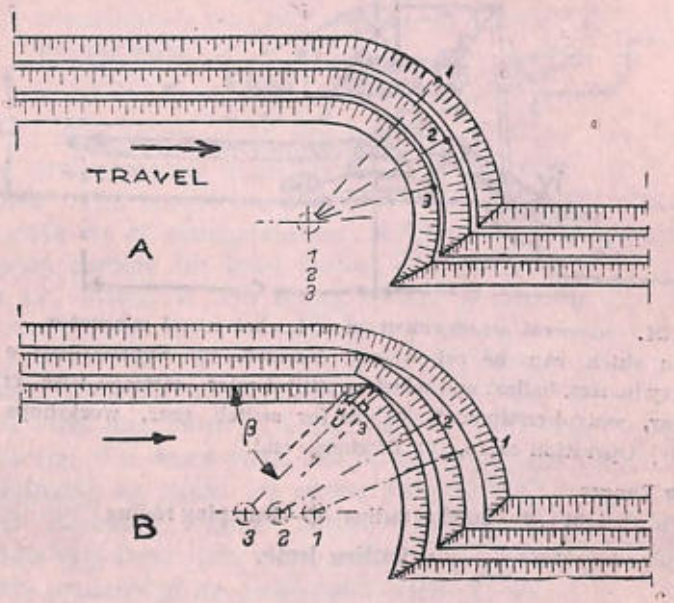
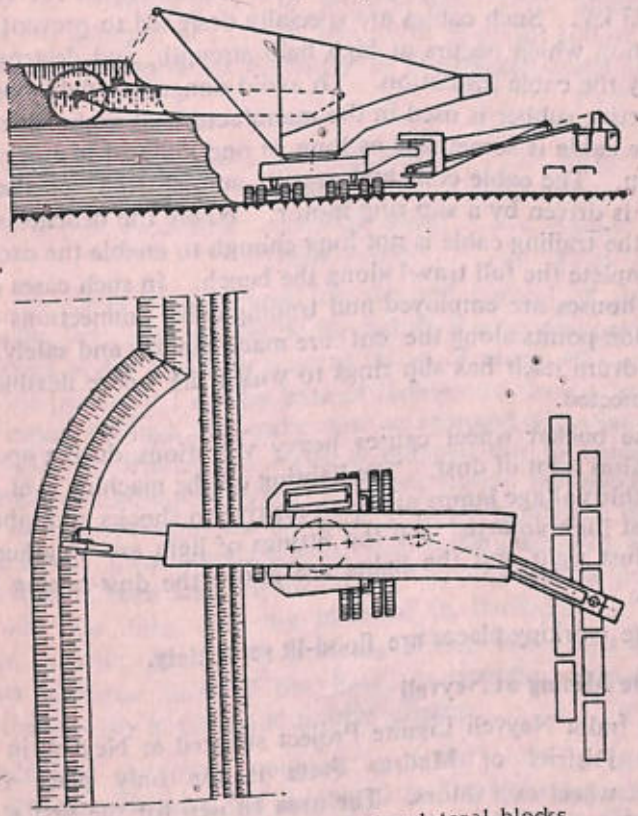


Fig. 22. Full block method.

Lateral Block Method

In order to remove continuously waste partings or mineral seams of minor thickness selectively, the excavator adopts the method of cutting lateral blocks. Both bucket wheel excavators—with thrust or without thrust—can dig the necessary lateral slopes. The width of the block depends, however, always on the length of bucket wheel boom (Fig. 23). Bucket wheel excavators with a short bucket-wheel boom are not suitable for cutting lateral blocks. Should the level of the selected seam lie high above the travel-way of the machine, and taking into account the necessary slope angle, a "super" bucket wheel would be required, i.e., with a diameter larger than that which would actually be needed for the output required.

In any case, the amount of crawler travel increases with decreasing width of blocks. The travel-way is reduced if an excavator with thrust is employed.



Method of cutting lateral blocks

Fig. 23.

The excavated rock is transported by a system of belt conveyors mounted on the machine to hopper cars and thence to belt conveyors or rail wagons. The loading belt of the machine is also mounted on a boom.

Power for Excavators

The horse power required for small machines is nearly 100 but for large models the power requirement is nearly 7,500 H.P.

or more. Power is supplied at A.C. voltage as high as 25 kV though motors operate at 3300 V, and 500 V. Machines of earlier days were fed from trolley wires and used to run on rails. Now-a-days special trailing cables are manufactured for voltages upto 25 kV. Such cables are specially designed to prevent ozone formation which occurs at high field strength, and destroys very quickly the cable insulation. To avoid dangerous field strengths, conductive rubber is used in the manufacture of such cables. The flexible cable is sometimes as long as one km and is mounted on a drum. The cable coils and uncoils automatically on the drum which is driven by a slip ring motor. Where the benches are too long, the trailing cable is not long enough to enable the excavator to complete the full travel along the bench. In such cases mobile stope houses are employed and trailing cable connections to distribution points along the cut are made quickly and safely. The cable drum itself has slip rings to which the 4-core flexible cable is connected.

The bucket wheel causes heavy vibrations during operation and raises a lot of dust. The lighting on the machine is at 110 V, as at this voltage lamps are less sensitive to shocks and vibrations than at high voltage. All the fittings of light are weather-proof and dust tight and the lights are above the dust-raising bucket wheel.

The working places are flood-lit separately.

Lignite Mining at Neyveli

In India Neyveli Lignite Project situated at Neyveli in South Arcot District of Madras State is the only place to use bucket wheel excavators. The area chosen for the first stage of operation comprises 14.25 Sq. Km. north of the Cuddalore Vriddhachalam railway line. It is estimated that this area contains 200 million tons of lignite with the following analysis.

Moisture—50%, ash—3%, V.M.—25%, F.C.—22%; sp. gr. 1.1€ (in-situ), Cal. Val.—2,450 Kcal/kg.

The lignite seam is 16m thick, dipping 1 in 100, as against an overburden of 54m to 55m. The ratio of overburden to lignite is thus 3.4 to 1 and quite favourable. The overburden comprises soft rocks except for a patch of hard rock at 12.5 m to 22 m level. The hard rocks are blasted by O.C.G.

and all the overburden is removed by bucket wheel excavators. Lignite being soft needs no blasting.

The mine is divided into 3 overburden benches of 21 m, 22 m and 12 m, totalling 55 m of overburden. The lignite bed exposed has a length of 1,500 m and width of about 100 m. The bench dimensions for operation of the bucket wheel excavators are :

350 litre b.w. excavator	—average width	— 25 m
	max. height	— 12 m
700 litre b.w. excavator	—average width	— 38 m
	max. height	— 22 m.

The project is situated near the sea and control of artesian water under high pressure below the lignite is a serious problem. The artesian water exerts an upward thrust of nearly 80 tons per m^2 (8 tons per sq. ft.) on the base of lignite and there is risk of water bursting through the lignite seam on removal of overburden. Nearly 220,000 litres/min of water is pumped out by means of 48 or more bore-hole pumps, most of them around the area being worked out, to maintain the pressure of water below the lignite at a safe level. Each borehole pump is of 4,500 litres/min. capacity with 102 m head and 150 kw rating. These borehole pumps are of 400 mm diam. and are installed in boreholes of 1 m or 1.5 m diameter lined with a casing of 500 mm. The large quantities of water pumped out have, incidentally, helped to irrigate the vast dry areas in the nearby villages.

In addition to these, pontoon mounted floating pumps (18,000 litre/min capacity) are employed during the rainy season on the bottom-most bench to minimise flooding of the benches.

The machinery used in the mine for production consists of:

700 litre bucket wheel excavators for overburden;

350 -do- for lignite and overburden;

Spreaders (model 213 and model 181/182);

Mobile transfer belt conveyors;

Belt conveyors of 1500mm, 1200 mm and 1000 mm width;

Transfer feeder;

Bulldozers, etc.

or more. Power is supplied at A.C. voltage as high as 25 kV though motors operate at 3300 V, and 500 V. Machines of earlier days were fed from trolley wires and used to run on rails. Now-a-days special trailing cables are manufactured for voltages upto 25 kV. Such cables are specially designed to prevent ozone formation which occurs at high field strength, and destroys very quickly the cable insulation. To avoid dangerous field strengths, conductive rubber is used in the manufacture of such cables. The flexible cable is sometimes as long as one km and is mounted on a drum. The cable coils and uncoils automatically on the drum which is driven by a slip ring motor. Where the benches are too long, the trailing cable is not long enough to enable the excavator to complete the full travel along the bench. In such cases mobile stope houses are employed and trailing cable connections to distribution points along the cut are made quickly and safely. The cable drum itself has slip rings to which the 4-core flexible cable is connected.

The bucket wheel causes heavy vibrations during operation and raises a lot of dust. The lighting on the machine is at 110 V, as at this voltage lamps are less sensitive to shocks and vibrations than at high voltage. All the fittings of light are weather-proof and dust tight and the lights are above the dust-raising bucket wheel.

The working places are flood-lit separately.

Lignite Mining at Neyveli

In India Neyveli Lignite Project situated at Neyveli in South Arcot District of Madras State is the only place to use bucket wheel excavators. The area chosen for the first stage of operation comprises 14.25 Sq. Km. north of the Cuddalore Vriddhachalam railway line. It is estimated that this area contains 200 million tons of lignite with the following analysis.

Moisture—50%, ash—3%, V.M.—25%, F.C.—22% ; sp. gr. 1.1€ (in-situ), Cal. Val.—2,450 Kcal/kg.

The lignite seam is 16m thick, dipping 1 in 100, as against an overburden of 54m to 55m. The ratio of overburden to lignite is thus 3.4 to 1 and quite favourable. The overburden comprises soft rocks except for a patch of hard rock at 12.5 m to 22 m level. The hard rocks are blasted by O.C.G.

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The machinery used in the mine for production consists of:
700 litre bucket wheel excavators for overburden;
350 -do- for lignite and overburden;
Spreaders (model 213 and model 181/182);
Mobile transfer belt conveyors;
Belt conveyors of 1500mm, 1200 mm and 1000 mm width ;
Transfer feeder ;
Bulldozers, etc.

The machines are supplied electric power at 11 KV. through T.P.S. cables and transformers in the machines step down the voltage to 3.3 KV for some motors and to 400 V for other motors.

Two large bucket wheel excavators dig the overburden in various steps at different levels from faces having a length of 1,500 m and deliver the material through travelling transfer hoppers on to conveyors, which in turn, transport the material on the spreader at the spot where the material is to be dumped. The exposed seam of the lignite is worked by the small bucket wheel excavators and the lignite conveyed to the power station or the briquette factory. The place from where lignite is excavated is used for back filling of overburden. A layout of the quarry is shown in Fig. 24.

Following are the details of *bucket wheel excavators.*

	(700 litre cap.)	(350 litre).
Cutting height above crawlers	20m	12m
Cutting depth below crawlers	3m	5m
Effective output loose material/hour	2700 m^3	1350 m^3
Width of belts	1.8 m	1.4 m
Service weight metric tons.	1300	475
Track speed/min.	5 m	8 m
Number of motors including transformers	32	
Power installed kW	1873	740
Supply voltage	11,000 V	11,000 V
Motor voltage (Standard)	400 V	400 V
Wheel diameter, measured over knife circle	8 m	6.2 m
Number of buckets	9	8
Number of discharges per min.	59—72—85	48—60—72
Bucket capacity...litres	700	350
Number of revolutions of bucket wheel....per min.	6.5—8—9·4	
Circumferential speed, measured at knife circle metres/sec.	2.74—3.35—3.95	1.95—2.44—2.92

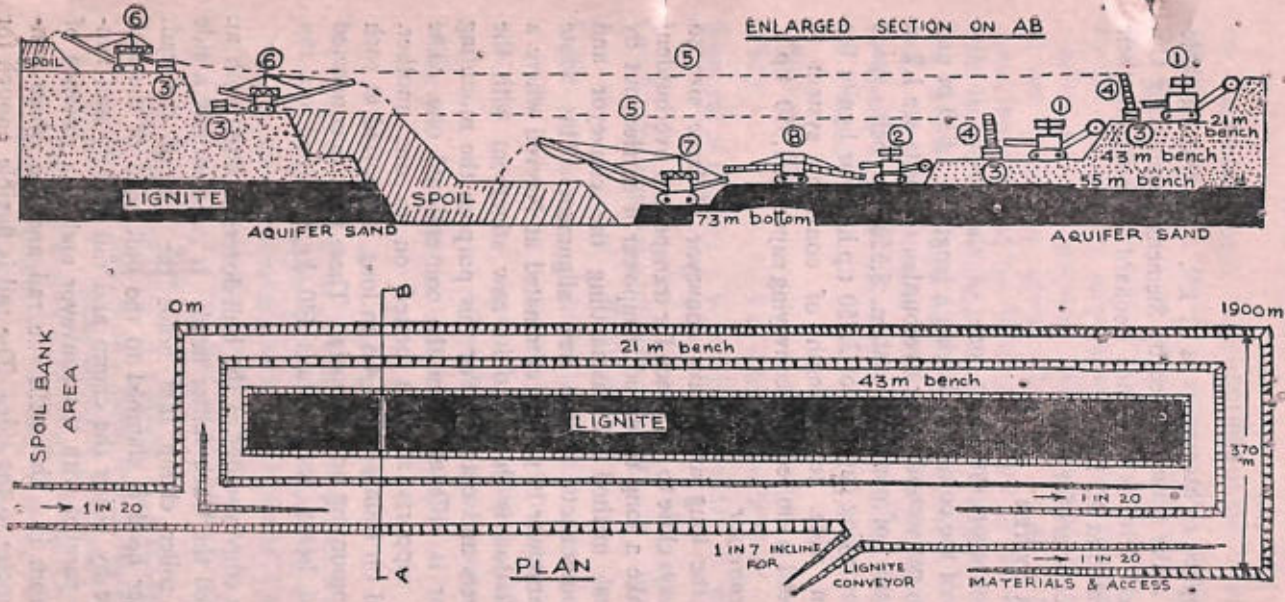


Fig. 24

(1) Bucket wheel excavator, 700 litre. (2) B. W. excavator, 350 litre. (3) 1·2 m OB conveyors. (4) Bench lift conveyors. (5) Rail-mounted shuttle conveyor. (6) 35m spreader. (7) 60m spreader. (8) mobile transfer conveyor.

Driving power kW (for bucket wheel)	.. 650/120	.. 200/50

Power is supplied to excavators at 11 kV. The trailing cable is "OXONEX" make manufactured by Siemens Engineering Co. The bucket wheel drive is by Ward Leonard system, and their motors are operated at 3.3 kV.

There are 6 crawlers for travel.

BELT CONVEYOR SYSTEM

The belt conveyor system is one of the largest in the world with one of the conveyors having a length of 1,830 m and a speed of 4.8 m per second. For overburden the belts are 1.2 m wide, total length of conveyor system, 8,350 m, belt speed 4 m/sec. and conveying capacity of 2,750 t.p.h. For lignite the belts are 1 m wide, total length of conveyor system is 6,450 m, belt speed is 4m/sec. and conveying capacity, 1,350 t.p.h.

Shifting of belt conveyor

Shifting of the long and heavy conveyor of large size so as to keep it always close to the bench for transport of overburden/lignite would take a long time and manpower if carried out by the conventional method of dismantling the conveyor and shifting the components to the new alignment. This waste of time and manpower is greatly eliminated at Neyveli where a conveyor installation is shifted to its new alignment with the help of bulldozers and cranes. For this purpose the mounting of the conveyor is different from the conventional one. The conveyor idler supports are not bolted on to one another, but are grouped on frames about 4.5 m long, supported at each end by a pivot mounting on a crosstie. These ties are connected by one or two heavy section rails (40 kg/m) fitted on rail sleepers.

To shift the conveyor one of the bull-dozers is fitted with an attachment called the track shifter head. It is essentially a side boom with a roller clamp that holds the top of the rail. The track shifter head is clamped to the bull head of the rail. Shifting is done by lifting the clamp and rail slightly and running the dozer parallel with the conveyor and at a distance of about 1.5 m from it. This pulls the rail and the whole conveyor about a metre to the side. The rail is flexible enough for

the bending involved. A return trip of the bulldozer moves the conveyor through another metre. The process is repeated till the conveyor is at its new alignment. The drive head and the tail end is correspondingly pulled every time gradually to suit the snaking movements of the conveyor till it finally reaches its new position. The conveyor is laterally shifted to new alignment about 30 to 40 m at a time depending upon the width of the block excavated by the excavator. The belts are normally 350 to 500 metres in length and are shifted in less than 2 to 3 shifts at the rate of about 2,500 sq. metres per hour, so as to ensure minimum loss of production hours. Normally 3 to 4 bull-dozers and a crane are employed for this operation.

PRODUCTIVITY

O.M.S. —3 to 4 tonnes of lignite ;

Removal of lignite : overburden : water is in the ratio of 1 : 8 : 25 ;

Overburden removal —40,000 m^3/day, to be increased to 80,000 m^3/day.

Lignite extraction —8,000 to 12,000 t.p.d. or 3.5 million tonnes per annum ; proposed to be increased to 18,000 to 20,000 t.p.d. or 6.3 million tons per annum.

Water pumped out —170,000 litres/min.

REFERENCES

1. ZEINDLER, R. W., "Economic comparison and detailed discussion of dragline VS. shovel for removal of overburden". Opencast Mining/Quarrying, and Alluvial Mining (1964 Symp). Inst. Min & Met. London.
2. NAIDU, B. S., "Neyveli lignite mining", paper read before 1.M.M.A. 28th Nov. 1965.
3. CENTRAL WATER & POWER COMMISSION, GOVT. OF INDIA "Hand Book of Earth Moving Machinery".
4. MISRA, G. S., "Some aspects of mechanised opencast mining" Trans. Min. Geo. & Met. Inst. of India, Vol. 57, June 1960.
5. FLACHSENBERG, P., "Loading & haulage in quarries" Opencast Mining/Quarrying and Alluvial Mining 1964 symposium, Inst. Min. & Met. London.

CHAPTER 11

EXPLOSIVES, DETONATORS & ACCESSORIES. TRANSPORT & STORAGE

1. Explosives. 2. Permitted explosives. 3. Detonators. 4. Exploders. 5. Transport and Storage.

1. EXPLOSIVES.

In the mining industry, explosives are used to break coal and rock in the underground mines, quarries and shaft sinking. In the non-mining industry explosives are used for providing the stone from quarries to make roadways ; preparing dam foundations ; for demolishing buildings, chimney stacks, bridges, etc. ; for deepening rivers and harbours ; for blasting wrecks or under-water obstructions ; for seismic prospecting ; for tree stump removal, ditching, boulder breaking and many other purposes. Gunpowder or black powder used in fireworks on festive occasions is the explosive with which a layman is generally acquainted.

An explosive is a solid or a liquid substance, or a mixture of substances, which change themselves instantaneously into a large volume of gases at high temperature and pressure when subjected to combustion or detonation.

The chemical energy in an explosive is released and converted suddenly into heat and mechanical energy during blasting and this conversion is by chemical reactions initiated by combustion or detonation. *Detonation* is the process of giving sufficiently violent shock to the explosive to bring about an almost instantaneous re-arrangement of atoms. In other words, it is the process of propagation of a shock wave through an explosive, accompanied by a chemical reaction that furnishes energy to maintain the shock propagation in a stable manner ; combustion or deflagration is the process of burning with extremely rapid rate. The chemical reactions

involved in both the processes are oxidations and an explosive contains, by virtue of its constituents, all the oxygen necessary for complete oxidation. It also contains, apart from the explosive substance or explosive mixture,

(i) Combustible matter, such as woodmeal of bulky type, fibre, flour, sulphur, etc.

(ii) Oxidising agents, such as sodium nitrate, ammonium nitrate, etc.

(iii) Stabilisers, such as magnesium and calcium carbonates.

The disruptive strength of an explosive depends on the speed with which the gaseous products are formed and explosives are generally grouped in two types, depending upon the speed with which explosive effect is produced :

(1) Low explosives,

(2) High explosives.

Low explosive, when blasted, does not produce a shock wave in the explosive but burns with extremely high rate, producing in the process some solid residue and a large volume of gases that exerts a slow pushing (heaving) effect on the walls of the confining material. High explosive detonates when it is blasted ; it is completely gasified and through rapid production of gases exerts a shattering effect on the confining walls. Gunpowder is a common example of a low explosive and nitroglycerine, T.N.T., special gelatine, etc., are high explosives. The speed at which explosion wave travels through an explosive is called the velocity of detonation and ranges from 1,800 metres per second to 5,000 metres per second, while speed of deflagration in gunpowder is about 450 m/sec.

The **properties of explosives** of interest to the engineer are :

(1) *Strength* : This is a measure of the amount of energy released by an explosive on detonation and hence its ability to do useful work. The relative weight strength of explosives is determined by measuring the amount of recoil as compared with that produced by an equal weight of blasting gelatine. Blasting gelatine is taken as the standard because it is the most powerful commercial explosive with characteristics that are well-

known. In the U.S.A. the explosive used as the standard for comparison is T.N.T.

(2) *Velocity of Detonation* : It is the rate at which the detonation wave passes through a column of explosive and this is of considerable importance as the shock energy of detonation increases rapidly with this velocity. The shock energy does most of the work when the explosive is used unconfined, *viz.*, in plaster shots.

(3) *Density* : The density of an explosive is important and in selecting an explosive this property has to be given due consideration. With a high density explosive the energy of the shot is concentrated—a desirable feature in tunnelling and mining operations in hard ground. On the other hand when the output of lump coal from a mine is important, a low density explosive, which distributes the energy along the shothole may be needed.

(4) *Water Resistance* : Explosives differ widely in resistance to water and moisture penetration. Some explosives deteriorate rapidly under wet conditions, but others are designed to stand water long enough to enable the work to be done. When blasting is to be done under wet conditions a gelatinous explosive is preferable. The higher the nitroglycerine content of an explosive, the better its water resistance properties.

(5) *Sensitivity* : An explosive is required to be insensitive to normal handling, shock and friction, but must remain sufficiently sensitive to be satisfactorily detonated and capable of propagating satisfactorily cartridge to cartridge and even over short gaps such as may occur in careless loading.

(6) *Oxygen Balance* : Explosives which are to be used in conditions where ventilation is restricted, must be such that will give the minimum of harmful gases in the products of detonation.

Tests for Explosives

The tests carried out to determine the weight strength and other characteristics of an explosive are :

1. Heat test or stability test to determine effect on che-

mical stability or the degree of deterioration after prolonged storage.

2. Velocity-of-detonation (V.O.D.) test or the Dautriche test.
3. Ballistic Mortar test for weight strength.
4. A.D.C. (Ardeer Double Cartridge) test to determine the sensitivity of an explosive to propagation of detonation across a maximum gap between adjacent cartridges.
5. Continuity test.

These tests are carried out by the manufacturers at Gomia factory of Indian Explosives Ltd., and also at C.M.R.S. Dhanbad, which issues the certificate. The details of the tests are beyond the scope of this book and can be studied from a text book on mining explosives.

Comparison Between High and Low Explosives

Low explosive	*High explosive*
1. It is a mechanical mixture ; individual components are not necessarily explosive.	1. It is a chemical explosive compound mechanically mixed with other ingredients.
2. It is fired by ignition	2. Action is initiated by detonation.
3. Decomposition is slow from grain to grain. Gases attain maximum temperature gradually.	3. Decomposition is at a high speed. Maximum temperature is attained by gases instantaneously.
4. Pressure is gradually built up ; heaving effect.	4. Pressure is built up instantaneously ; shattering effect. This can be modified by suitable ingredients.
5. Velocity of propagation of flame is low ; a few metres per second.	5. Velocity of detonation is a few Kms per second.

Low Explosives

Gunpowder

This was the earliest known explosive. Its constituents by weight are nearly of the following proportions :

Charcoal	..	..	15%
Sulphur	..	..	10%
Potassium nitrate (KNO_3)	..	..	75%

The Charcoal used should be from hard wood. Charcoal and sulphur are combustible and KNO_3 is a supplier of oxygen. Sodium nitrate may be used in place of KNO_3 but the latter is less hygroscopic. Gunpowder is cheap, stable, safe to handle, does not adversely affect roof due to its heaving effect and is easily manufactured. It is available from local supply-contractors in all the coalfields and as the stocks can be replenished from week to week, large magazines are not necessary. It is available in paper cartridges, and for use in wet places, in cartridges of thin tinned sheets. Volume of gases produced is nearly 2,000 times and the pressure developed is between 1,550 kg/cm² and 2,100 kg/cm². Cartridges of various sizes can be prepared at an approved place on the colliery premises, to suit blasting requirements.

Gunpowder has the following disadvantages :—

(1) It is spoilt by water.

(2) It is bulky for a given explosive effect.

(3) It has a flame of long duration and is dangerous in gassy mines.

(4) Its burning particles are liable to remain in contact with the surrounding atmosphere for some considerable time.

(5) It produces a lot of carbon monoxide and other noxious gases and after blasting a good deal of time is lost in allowing the gases to clear.

The length of flame produced by gunpowder when it explodes can be reduced by including 10% starch in the composition. For disadvantages No. (3) and (4) use of gunpowder is dangerous in gassy mines and also in dry and dusty non-gassy mines. Its use in such conditions is prohibited by the Coal Mines Regulations.

The efficiency of gunpowder cannot be decided at the colliery before use and the efficiency may be different in the different supplies made by the same contractor. A good quality gunpowder is granular and when a handful of it is burnt on a clean stone, green flame is indicative of good quality. (Never burn gunpowder in a confined space, otherwise it will explode). A poor quality gunpowder is fine and would burn with yellowish flame. The quality may also be judged by the residue left after burning ; if it sticks to the stone, the quality is upto the mark ; if the residue can be easily scratched by hand, the quality is to be discarded. Moist gunpowder should not be used and should not be accepted from supplying contractor. To know whether the gunpowder is moist or dry, open the cartridge, pour the contents on the palm of the hand and press it in the fist. If it is difficult to press and the powder spreads easily when the fist is released, the powder is dry. If the powder can be easily pressed and retains some shape when the fist is opened, it is moist.

In a newly opened mine, if it is not gassy or dusty, gunpowder may be used till such time as the supplies of other suitable and permitted explosives are available.

SAFETY FUSE

Gunpowder is fired by safety fuse which consists of a core of fine grained gunpowder wrapped with layers of a tape or textile yarn and waterproof coatings to provide protection against mechanical damage and moisture penetration. The rate of burning is normally about 600 mm per minute. Safety fuse is supplied by some firms who manufacture it in the coalfields, e.g., Sanyal Hasanjee Explosive Works, Nagpur and Inchex Co. (Private) Ltd., at Mohoda (Jharia field). Indian Explosives Ltd. manufacture a range of safety fuses to suit various conditions ; Bull brand safety fuse for dry conditions, Blue Sump for damp conditions and W.C.P.S. for wet and very rugged conditions.

CORDTEX—(See under "Blasting in Quarries").

PLASTIC IGNITER CORD

This is used in quarries, non gassy mines and metalliferous mines for lighting any number of safety fuses in the desired se-

quence. It is cordlike in appearance and when ignited, the flame passes along its length at a uniform rate. Two types of igniter cords are available, a "fast" cord burning at a rate of 3 sec/metre and a "slow" cord burning at 30 sec/metre. Both are resistant to water and rough handling. The burning material consist of paper strings coated with gunpowder in the case of 'fast' igniter cord which has overall diam. of 2.5 mm.

The explosives other than gunpowder used for mining operations in this country are those manufactured by Indian Explosives Ltd. at Gomia and by the Indian Oxygen Ltd. The latter manufactures liquid oxygen and absorbent cartridges only and these are described in the next chapter. The explosives of Indian Explosives Ltd. are described in the following pages.

High explosives

The chemical energy of a high explosive is released by a process called detonation. Blasting gelatine, special gelatines, opencast gelignite, hectorite and G.N.1 are all high explosives which explode when subjected to detonation. Most high explosives do not detonate when ignited, but simply burn away. They are not permitted to be used underground in gassy mines.

Important constituents of high explosives which confer explosive properties on them are nitroglycerine, ammonium nitrate, collodion cotton and T.N.T.

NITROGLYCERINE

This is produced by the reaction between glycerine compounds and nitric acid. In its pure state nitroglycerine is an oily fluid, with a freezing point of 13°C and a specific gravity of 1.6, so sensitive to explosion by shock of any nature that it is dangerous to handle it when pure. The sensitivity to shock increases appreciably when nitroglycerine freezes. Its temperature of detonation is very high, about (3150°C). When placed on the skin, it is slowly absorbed causing a violent headache. To render it suitable for industrial use, it must either be absorbed by some inert material or be gelatinised. Explosives containing NG are available in 3 consistencies ; gelatinous, semi-gelatinous and powdery. Higher NG content renders the explosive gelatinous ; lower NG content upto 10%, gives it powdery consistency. The gelatinous explosives are

therefore, better water resistant. NG when frozen is more sensitive to explosion but loses explosive efficiency. All explosives containing NG have a highly shattering effect. NG is insoluble in water. Of all the high explosives only NG contains in its own composition the whole of the oxygen required for complete combustion. NG, on explosion, produces harmless gases if reaction proceeds perfectly.

$$4C_3H_5(NO_3)_3 \rightarrow 12CO_2 + 10H_2O + 6N_2 + O_2.$$

All explosives containing ordinary NG are liable to freeze when the temperature falls to 8°C or less and are then more sensitive to friction and impact. To avoid this, a low freezing agent, usually di-nitro-glycol which is itself as powerful an explosive as NG, is used. Di-nitro-glycol freezes at −22°C. Low freezing explosives are usually designated by such trade names as "Polar", e.g., Polar viking, Polar special gelatine. In low temperature regions where explosive cartridges are likely to freeze, use explosives with prefix "Polar".

AMMONIUM NITRATE (NH_4NO_3)

It is an interesting compound. It is a high explosive, an oxidising agent, and a cooling agent at one and the same time. It is very difficult to detonate by itself and is therefore safe to handle. To use ammonium nitrate as an explosive, it is necessary to mix it with some other high explosive such as NG or T.N.T. as sensitiser. Ammonium nitrate is hygroscopic ; it must, therefore, be kept wrapped in water proof wrappings of waxed paper.

If subjected to great heat under confinement or direct detonation of high explosives, ammonium nitrate decomposes explosively into nitrogen, hydrogen and water and the simplest reaction is

$$2NH_4NO_3 = 2N_2 + 4H_2O + O_2$$

Ammonium nitrate has a low temperature of detonation (about 1,150°C as compared to 3,150°C for NG), has a surplus supply of oxygen and it is less violent than NG explosives so that it has a heaving effect. Prilled ammonium nitrate of fertiliser grade mixed with diesel oil, is used for well-hole blasting.

Collodion Cotton

A reaction between cellulose compounds and nitric acid yields collodion cotton. To render it safe to handle, collodion cotton is subjected to a treatment known as "gelatinisation". This consists in dissolving collodion cotton in alcohol and ether and allowing the solvent to evaporate so that the nitro-cotton remains as a glue-like mass which is more stable and much safer to handle than the original explosive substance.

T.N.T.

Reaction between nitric acid and benzene or toluene compounds yields Trinitrotoluene (T.N.T.) which is highly explosive.

Dynamite, a powerful explosive with intense shattering effect and containing NG as a main constituent is no longer marketed in India.

Blasting Gelatine

A gelatinous explosive, it contains nearly 92% of NG and 8% of collodion cotton. It is the strongest of all the known commercial explosives and is unrivalled for blasting in the hardest rocks, such as quartz, granite, etc. It is also used for plaster shooting (mud capping). Weight strength of an explosive is expressed as a percentage of the strength of blasting gelatine, the most powerful commercial explosive produced.

Seismic Explosives

Special forms of blasting gelatine, and ammonia gelatine dynamite are developed to give consistent rate of detonation under heavy water pressure. They are used in geophysical prospecting by seismic method.

Other high explosives like special gelatine, opencast gelignite, G.N.1 etc. which find an application on a wide scale in opencast mines have been described under the heading "Blasting in Quarries" in the next chapter.

2. PERMITTED EXPLOSIVES.

Explosives for use in places where inflammable gas may occur have to pass certain tests. These tests are designed to determine whether an explosive is likely to ignite gas or coal

dust when used in charges upto a specified weight (which may vary in different countries depending upon the nature of coal and other prevalent conditions) e.g. in India the normal charge limit in one shot-hole is 795 g in coal in the case of sheathed and non-sheathed permitted explosives and 1020g for eq. sheathed explosives.

An explosive which passes the tests stated below is known as "permitted Explosive". The safety of "Permitted Explosive" depends primarily upon the (i) low temperature, and (ii) duration of flame produced, the flame lasting only 1/1000th second. It has been observed that weaker the explosive, (i.e. the smaller the energy released per mass unit), the more readily would it pass any gas ignition test. A cooling agent is an essential constituent of a permitted explosive. It is considered that when an explosive detonates, the inert particles of the cooling agent travel ahead of the decomposing explosive particles and form a dust barrier which prevents access of flame to surrounding firedamp. Sodium chloride is a suitable cooling agent.

Only permitted explosives should be used in the following places (Coal Mines Reg. 172) :

(1) A gassy mine.
(2) A mine having fire in any district, whether such fire is sealed off by stoppings or not.
(3) In dry and dusty places in a mine.
(4) In a mine where gas is likely to be liberated by a shot, e.g. near a fault plane.

Permitted explosives are categorised in the following groups and the group to which a cartridge of explosives belongs is printed on the wrapper of the cartridge.

Group

P1 —Unsheathed explosives, such as Ajax G, Viking G.

P2 —Sheathed explosives.

P3 —Equivalent sheathed explosive (Eq.S) e.g. Unisax G.

P4 —Explosives approved for special purposes, such as for delay firing, firing in rippings, etc. ; hitherto known as ultra-safe explosives. These are not produced in

India. British examples are : Carribel, Nobel's explosive 1235.

P5—Off-the-solid-explosives (For solid blasting):

Tests for inclusion in Permitted List

In India C.M.R.S. tests for explosives for inclusion in Permitted list and the tests carried out are similar to the ones prescribed in the U.K. Before testing, the explosive must have passed tests for being placed on authorised list of explosives under the Explosives Act, must be certified by manufacturers as having cleared the "continuity of detonation" (COD) test, and satisfy the normal requirements of safe storage and practical use.

Official tests for inclusion in the Permitted List are :

(i) firing shots into explosive mixtures of fire damp and air containing combustibles equivalent to 9.0 ± 0.25 per cent of methane,

(ii) firing shots into coal dust containing 36% volatile matter on dry ash-free basis and of fineness such that 85% of it by weight passes through 200 mesh (BSS) sieve.

Different groups of permitted explosives have to pass tests prescribed for that particular group.

Unsheathed explosives have to pass the following tests :

(i) Five shots, each of 8 oz, consisting of cartridges of $1\frac{7}{16}''$ dia, (tolerance $+\frac{1}{16}''-0''$) directly initiated, are fired without stemming into the gas mixture.

(ii) Five shots, each of 28 oz, directly initiated and stemmed by 1" (25 mm) standard dry clay plug pushed down to the charge are fired into the gas mixture. The explosive must not ignite the mixture. The composition of gas mixture is checked for each shot.

(iii) Five shots, each of 28 oz, directly initiated and stemmed as above are fired into coal dust of the specified fineness. Prior to test, 5 lb. of coal dust are scattered along the first 18 ft of gallery. For first shot, 40 oz. of coal dust are laid along first 6′ of plank ; for subsequent shots, 20 oz are used.

The explosive must not ignite coal dust.

Equivalent sheathed explosives have to pass the following tests :

(i) 26 shots, each of 14 oz, consisting of cartridges of $1\frac{7}{16}$" dia, (tolerance $+\frac{1}{16}$"-0") inversely initiated, are fired without stemming into the gas mixture.

Not more than 13 ignitions are allowed.

(ii) Five shots, each of 36 oz, directly initiated, stemmed by an 1" plug are fired into the gas mixture.

None should ignite.

(iii) Five shots, each of 20 oz, inversely initiated, are fired without stemming into a pre-formed coal dust cloud. To form the cloud, two paper bags containing 20 oz of coal dust of prescribed fineness and nature and one No. 8 instantaneous copper-cased detonator are suspended near the top of the gallery, $2\frac{1}{2}$' and 7' respectively from closed end ; and detonators are connected in series with detonator of the test charge, which is No. 1 half-second delay detonator. In addition 5 lbs of coal dust are scattered along 18' length of the gallery.

The explosive must not ignite coal dust.

After the explosive has passed the gallery tests as above, its composition will be determined by analysis to find that proportions of ingredients are within permissible limits.

The explosive is then tested for Continuity of Detonation test. For this 1 m length of explosive is built up by laying in loose contact cartridges of the smallest manufactured dia. and weight. The column is wrapped in three-thickness roll of the wrapping paper and laid on a flat steel rail or plate. It is detonated by a detonator of the strength recommended by manufacturers for use with the explosive being tested. This should detonate the column throughout the 1 m length.

Gr. P_1 and P_3 explosives are manufactured in India.

Ajax G, a gelatinous type of permitted explosive, is suitable for drilling and blasting in hard coal and stone blasting in wet places.

Viking G, a powdery explosive with N.G., has application medium hard rock and breaking coal.

SHEATHED EXPLOSIVES

These are not manufactured in India. A cartridge of a sheathed explosive is surrounded all round its length by a sheath of sodium bicarbonate ($NaHCO_3$), about 3 mm thick, but the ends of the cartridge are not so covered. The $NaHCO_3$ may be either in powder form held in position by a paper wrapper, or (perferably) in the form of a felt (like thick blotting paper) secured to the cartridge by adhesive composition. When the explosive is blasted, the sheath liberates CO_2 and water vapour which blanket the flame and hot gases of the explosive along the length of the hole. The decomposition temperature for sodium bicarbonate is 70° to 120°C. The decomposition absorbs considerable heat from the hot gases produced.

Advantages of Sheathed Explosives

(1) There is less danger of igniting firedamp, whether by an over-charged hole, or by a blown-out shot, or due to the presence of breaks in a shot-hole. The increased safety is due to :

(a) Cooling action of the sheathing material (a physical action)

(b) Volatilisation of the sheathing material (a chemical action)

(c) Formation of an extinctive blanket of CO_2 and H_2O around the detonating explosive.

$$2NaHCO_3 = Na_2CO_3 + CO_2 + H_2O.$$

(2) The charge is enclosed along its whole length, so affording protection against breaks unlike stemming which closes only the mouth of the shot-hole.

(3) Sensitivity and efficiency of the explosive are maintained.

(4) It is claimed that there is a reduced volume of noxious fumes by such explosives.

(5) More round coal is produced, presumably because of the cushioning action of the sheath.

Disadvantages of Sheathed Explosives

(1) Larger diameter of shot-hole is required to accommodate the sheath, involving heavier drilling equipment, greater weight of stemming and increased work for the shot-firer.

(2) Possibility of incomplete detonation exists if some of the sheathing material gets between successive cartridges due to the sheath being damaged.

(3) Increased cost of manufacture.

Precautions in Use

(1) Do not roll sheathed explosives in the hand, for this will loosen the sheathing material.

(2) Do not open the cartridges at the ends, but use a pricker of soft metal to pierce the cartridge for insertion of the detonator.

(3) Do not bump sheathed cartridges with the stemmer when charging.

It is desirable to use only one sheathed cartridge for one shot hole.

Equivalent Sheathed Explosives

In this type of explosive an inert material is incorporated direct and uniformly throughout the explosive composition, so that the products of explosion render the explosive as safe as a sheathed explosive. Some of the dangers associated with the use of sheathed explosives, if the sheath is displaced, are therefore eliminated. The permissible weight is correspondingly increased. The inert material may be sodium chloride plus ammon. nitrate or ammon chloride plus sodium nitrate. Eq. sheathed explosives are now available equivalent to all the different types of sheathed permitted explosives. With Eq. sheathed explosives, the permissible charge per hole is 1,020 g. UNISAX G is the only equivalent sheathed explosive manufactured in the country by Indian Explosives Ltd. It is a gelatinous explosive suitable for blasting in coal and stone in gassy mines in watery places.

Cooling Agents

A cooling agent is incorporated in all permitted explosives. The cooling agent is usually sodium chloride (NaCl), potassium chloride, sodium fluoride, or ammonium chloride.

TABLE 1

Properties of explosives

Explosive.	Gelatinous or powdery. %	Relative Wt. strength. %	Relative bulk strength. %	Density gm/c.c.	Velocity of detonation m/sec with No. 6 detonator.
Blasting gelatine ..	gelatinous	100	100	1.5	2500
Special gelatine					
90% ..	-do-	91	91	1.5	2500
80% ..	-do-	78	78	1.5	2500
60% ..	-do-	66	66	1.5	2500
Polar special gelatine		These are low freezing types with properties similar to those of special gelatines.			
80% ..	-do-				
60% ..	-do-				
Hectorite ..	Powder	75	50	1.0	2500
GN1 ..	Powder	80	51	0.95	3000 (with strong primer).
Open cast gelignite	Gelatinous	84	84	1.5	2500 in small cartridges. 4000 in large cartridges.
Permitted explosives :					
1. Ajax G	Gelatinous	56	56	1.5	2500
2. Viking G	Powder	60	40	1.0	2000
Eq. Sheathed 3. UnisaxG	Gelatinous	41	45	1.65	2500
Ultra safe: 1. N.E. 1235	Powder	28	21	1.10	1700
2. Carribel (N.E. 1331)	..	44	..	0.95	1500
Hydrobel	Gelatinous	48	55	1.70	5000

TABLE 2

Approximate composition of permitted explosives. (% by wt.)

	Ajax G	Viking G	Unisax G	Carribel
Volatile Matter ..	1	1	1	2
Nitroglycerine ..	26	10	28	8.5
Mononitrotoluene ..	2	..	1	
Nitrocellulose ..	2	..	1	
Ammonium Nitrate ..	40	64	24	9.5
Sodium Chloride ..	26	14	42	
China Clay	2	1	2	
Paraffin Wax	1	..	..	2
Woodmeal	1	8	1.5	5
Rosin	1	..		1
Anti setting agent ..	.01	.01	.01	.01
Ammonium Chloride ..	..	..	..	28
Sodium Nitrate ..	..	..	..	45
Calcium Stearate ..	..	..	..	1
Diammonium Phosphate ..	..	..	..	1

ULTRA SAFE EXPLOSIVES

These have been designed to give a greater measure of safety than Eq. sheathed explosives. Such explosive has incorporated in its composition a high percentage of very finely ground sodium chloride and nitrochalk which presents a large surface area in contact with any gas into which the explosive flame may be projected. Nobel No. 1235 is an example of this type.

Deterioration in quality during storage.

The chemical composition of explosives may change slightly after storage under humid and wet conditions. The explosive may become insensitive to detonation or unsafe to use if the change in chemical composition is excessive. At the mines no laboratory tests are carried out to decide the extent of deterioration, but visual examination is the only guide. When the wrapper of a gelatinous nitroglycerine explosive is removed carefully an explosive in a good state would permit the wrapper to come off cleanly from the cartridge and there would be no evidence of discoloration, incrustation, softening or exuda-

tion of liquid in the explosive itself. A gelatinous explosive slightly darkens during storage, particularly at the ends, due to absorption of moisture from the air. Such discoloration is usually accompanied by softening and dampening of the explosives. The moisture absorbed by the explosive may cause a white incrustation of ammonium nitrate or sodium nitrate on a gelatinous explosive. A higher degree of deterioration is indicated by exudation of nitroglycerine or aqueous solutions of sodium or ammonium nitrate and the cartridges becoming too plastic. Such explosive should not be used but destroyed.

Powdery nitroglycerine explosives deteriorate much faster than the gelatinous quality as the former absorb moisture more readily. The deterioration is indicated by discoloration at the ends, softening and dampness. If there is any exudation of liquid the cartridges should be destroyed.

Ammonium nitrate explosive mixed with T.N.T. cake and become solid due to absorption of moisture during storage. If the cartridges become so solid that they cannot break easily between the finger and the thumb, they should not be used but destroyed.

Destruction of Unserviceable Explosives

Explosives which have become unserviceable for blasting should be destroyed and never dumped into a garbage heap. The methods adopted for destroying are as follows :

Gunpowder—It should be thrown into water, preferably hot water, which dissolves out the saltpetre and renders the explosive harmless.

Ammonium nitrate explosives—These are hygroscopic and may be buried in a damp ditch where they become harmless with the absorption of moisture.

Explosives with nitroglycerine—Not more than 25 kg of explosives should be destroyed at a time. A clear space of ground free from dry grass, about 100 m all round, should be selected and a line of shavings of dry straw or grass laid down. On this the cartridges should be placed in a continuous line not more than two abreast with the cartridge wrappers below them and with an interval of about 25 mm between two cartridges. The line of shavings of straw or grass should extend some distance,

say 6 m, beyond the explosives. Kerosene should then be sprinkled over the shavings of straw or grass.

The straw should be lit with a short length of safety fuse and the operator should withdraw immediately to a safe distance. The direction of wind should be given due consideration when burning the straw.

The degree of confinement caused by the cartridge wrappers in high explosives is sometimes sufficient to cause explosion. It is, therefore, essential to open the wrappers and unroll them.

Safety fuse—It should be destroyed by burning in length in the open under suitable precautions.

Detonators—They should be thrown into a deep river or alternatively, they may be soaked in mineral oil for 48 hours and then destroyed, one at a time under suitable precautions, by burning.

3. DETONATORS

A detonator is a small copper or aluminium tube containing essentially a small auxiliary charge of a special explosive. A chemical reaction initiated by a flame in the special explosive can build up very rapidly into an explosion of sufficient intensity to project a detonation wave throughout a high explosive enclosing the detonator.

Detonators are of the following types :—

1. *Plain detonators* :—These are fired by safety fuse, the spark or "spit" from the fuse causing the detonator to explode. These can be used only in nongassy mines and other places where use of permitted explosives is not required under the Coal Mines Regulations.

2. *Ordinary electric detonators* :—These are fired by passage of electric current through the detonator. They are further subdivided as :

(a) Low tension detonators, and

(b) High tension detonators.

Ordinary electric detonators are of instantaneous type, i.e., without any delay element.

3. *Delay detonators*;—These are essentially low tension electric detonators with delay element, the object in their use being to phase the firing of shots, so that time and effort are saved in charging and firing several successions of shot-rounds. These are subdivided as :

(a) Half second delay detonators, and

(b) Milli-second delay detonators (also known as short delay detonators).

The word "ordinary detonator", sometimes used is vague and may mean "plain detonator", or "ordinary electric detonator" though the context may clarify the meaning. An ordinary electric detonator is generally referred to as "electric detonator".

Regulations now require that only detonators of approved type must be used in places where permitted explosives are required to be used. They must be of copper & pass certain tests

A plain detonator, *i.e.* non-electric, consists simply of an aluminium tube 6 mm dia., 37 to 50 mm long, filled 1/3 with A.S.A. composition and penta-erythritol tetra-nitrate (P.E.T.N.). The A.S.A. composition consists of a mixture of lead azide (A), lead styphnate (S) and a little aluminium powder (A). The A.S.A. is the priming composition in which lead azide is the main explosive ingredient and lead styphnate, which is also an explosive, is added to improve the sensitiveness to ignition by flame and to help bind the composition in the tube. Aluminium powder is used to give the composition free flowing properties during manufacture. The A.S.A. priming composition initiates the base charge of P.E.T.N. which is a much more powerful explosive. No. 6 detonator contains about 0.35 g of the A.S.A. composition and 0.25 g of P.E.T.N. Stronger detonators are manufactured by increasing the quantity of P.E.T.N. base charge. No. 8 detonator carries larger charges, 25% more than No. 6, and is used for blasting in hard rocks. It produces much stronger pressure pulse than No. 6. When it is directly in contact with the explosive charge, *e.g.* for breaking boulders and breaking hard toes in quarry benches, it produces better blasting results than No. 6. If, however, cordtex fuse is used for initiation No. 8 detonator has the same effect as No. 6.

No. 6 detonator produces 0·17% CO and No. 8 produces 0.08% CO.

In an ordinary electric detonator, *i.e.* non-delay type, the priming charge and base charge are the same as for plain detonator, but they are fired, not by ignition of a safety fuse, but by passing electric current through a fusehead. The current ignites a flashing composition in the fusehead, which, in turn, initiates the priming charge. (Fig. 1).

In the upper portion of the tube is a neoprene plug assembly. This consists of (a) the electric fuse head, (b) the leading wires and (c) the neoprene plug through which the leading wires pass. The electric fuse head may be of the low tension type as in a L.T. detonator or high tension type as in a high tension detonator. The high tension detonators are not used for mining work in India and will not be described here.

In pulsed infusion shot firing, Hydrostar electric detonators are used ; these have thicker walls and greater initiating power, and are generally designed for special under-water characteristics.

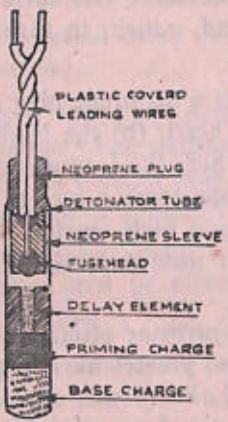
The L.T. fuse consists of two thin strips of brass foil separated by a sheet of insulating board and connected diagonally at the lower end by a fine wire bridge of nickel chromium alloy which is immersed in a bead of ignition composition. The leading wires are soldered to the brass foils at the upper end so that there is a continuous metallic circuit through the fuse. The fusehead consists of lead-mono-nitrate—resorcinol (LMNR), potassium chlorate, a little nitro cellulose and charcoal.

The leading wires are tinned iron wires, 24 SWG, 1.8 metres long and plastic (yellow) covered for wet conditions or enamelled. The sealing plug is made of neoprene which is impervious to water and oil.

The internal resistance of a low tension detonator is 1.3 ohms ; with a 45 m long shotfiring cable the resistance is about 7 ohms. Current required for ignition of the fusehead is 0.5 amps so that a single detonator can be blasted with a minimum voltage of 3.5 V. The circuit continuity of a L.T. detonator can be tested by a galvanometer and simultaneous shotfiring of a number of detonators with series connections is possible.

DELAY DETONATORS

In appearance and composition, this is like the L.T. detonator. The L.T. detonator described above is of instantaneous type; the moment voltage is applied to it, the detonator explodes and along with it the enclosing explosive. A delay



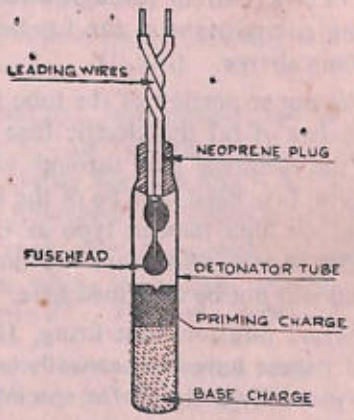


Fig. 1 (right). Section of ordinary electric detonator.
Fig. 2 (left). Section of delay detonator.

detonator has a delay element introduced between the fusehead and the priming charge. (Fig. 2). The delay element consists of a copper or brass sleeve filled with a special composition, which is as follows—

(i) *Half second delay series* :—a mixture of antimony and potassium permanganate.

(ii) *Short delay series* :—a mixture of red lead and silicon.

The special composition burns at a specified rate and the delay is obtained by varying the length of the sleeve containing the special composition. For delays of higher number the delay element is longer and the detonator is therefore longer than one with lower delay number. This is specially noticeable in Czech delay detonators. The products of combustion of the delay element do not produce gas at pressure sufficient to burst the detonator prematurely. All delay detonators are of No. 6. strength, have copper leading wires 3 metres long, bearing a tag of delay number.

The half second delay detonators are available in the range from No. 0 delay to No. 10 delay detonator. The last one, No. 10, has a delay interval of 5 seconds. Milli-second or short delay detonators have a much faster burning delay composition and the minimum delay interval is 25 milli-second (1/40 second). Both the leading wires are covered with red plastic. The delay interval is 0 second for No. 0 delay and increases by 25 milli second delay. Beyond No. 4 delay detonator the nominal delay between successive numbers is 50 milli-second from No. 4 to 12 and 70 milli-second from No. 12 to No. 15. A new non-incendive type namely the Carrick lead core short delay detonator is now available. This new series is much safer than the type just described and has the following nominal delay intervals in milli-seconds.

No. 0-nil ; No. 1—30 ; No. 2—55 ; No. 3—80 ;
No. 4—105 ; No. 5—135 ; No. 6—165 ; No. 7—195 ;
No. 8—230 ; No. 9—265 ; No. 10—300

For short-delay firing in pulsed infusion shotfiring, Hydrostar short delay detonators, having a delay upto 0.1 second are used.

Permission from the chief Inspector of Mines is essential for using delay detonators in gassy mines and other places where permitted explosives have to be used. British Regulations are more specific about the use of delay detonators and the important provisions are given below :

1. Delay detonators should not be used in a gassy mine or in places where permitted explosives have to be used, except with the previous permission from the Mines Inspectorate.
2. Leading wires of delay detonators should be of sufficient length to allow connections to be made without adding extra wires.
3. Delay detonators and ordinary electric detonators should not be placed in the same case.
4. When charging a hole, a primer cartridge having delay detonator should be inserted first into the hole with the detonator base pointing towards the main charge of explosives (inverse initiation).

8. permission to be obtained from D.G.M.S. for firing with M.E.C ~ M.E. 12 Explosives, the delay detonators.

5. The estimated period of delay between the first and the last shot of the round, calculated by references to the markings on the detonators should

(a) not exceed 0.1 second while firing in coal or ripping farther than 4.5 m from coal face, provided P4 explosives are used.

(b) not exceed 0.75 second while firing in drifts containing 0.3 m of coal seam, or within 4.5 m of waste, the places being stone-dusted and tested for gas,

(c) not exceed 5 seconds in other cases.

Delay detonators are used where holes charged with explosives have to be fired in a definite sequence. This is of advantage in stone drifts, sinking pits and mechanised quarries where a number of holes is charged and blasted at one time.

It may be noted that an ordinary electric detonator (L.T.) which is essentially of instantaneous type should not be used as a substitute for a delay detonator with zero delay, as the former may have some "tolerance" before detonating.

Advantages of Delay Detonators

(1) Reduced consumption of explosives as blasting is more efficient due to availability of a free face for each row of shots *e.g.* blasting due to No. 1 delay detonator gives a free face for the blasting effect of shots fired by No. 2 detonator.

(2) Increased fragmentation and ease of loading the rock or coal. Broken rocks from successive shots collide in air, thereby increasing the fragmentation.

(3) Considerable time is saved in that the whole round of shots is fired in a fraction of a second. This is the chief advantage. If individual shots, or even groups of 5 or 6 shots simultaneously by L.T. detonators are fired, the time required for inspection and clearance of fumes and gases between successive firings is considerable. In steep seams, the exertion involved in frequent trips for such examinations and connections is saved.

(4) The millisecond delay (short delay) detonators have been observed to produce less ground vibrations than the half second delay detonators and are therefore used in mechanised quarries where blasting of large diameter holes containing heavy

charges is likely to produce excessive ground vibrations and cause damage to nearby surface buildings and other important engine foundations, etc.

Plain detonators and electric non-delay detonators are manufactured by Indian Explosives Ltd., at Gomia. Electric detonators of non-delay and delay type are imported and marketed by Indian Detonators Ltd., Hyderabad whose plant for indigenous manufacture of detonators is likely to go into production shortly.

DETONATING RELAY : (See under "Blasting in Quarries").

4. EXPLODERS

The portable apparatus which provides the current necessary for firing electric detonators is called an exploder. (Fig. 3) It is illegal to take current from electric power, lighting, or signal lines to fire the detonators. The exploder has to be of a type approved by Chief Inspector of Mines for gassy mines.

The exploders are of 3 types :—

1. Magneto Exploder
2. Battery Exploder
3. Dynamo Exploder.

MAGNETO EXPLODER

The magneto exploder consists essentially of a permanent steel magnet between the poles of which revolves an armature rotated through gearing by a rotary handle or by a rack and pinion. The value of the voltage depends upon the speed at which the armature revolves and the flux created by the magnets. A low tension exploder gives a voltage of about 15 volts. A H.T. exploder gives about 125 volts. The L.T. armature winding has a smaller number of turns of a thicker wire than the H.T. type. The magneto exploder fires only 1 or 2 shots at a time with single shot exploder and upto 6 shots in series with a 6-shot exploder.

The exploder should be intrinsically safe. The armature is actuated by a special twist action detachable key which should always be in the possession of the shotfirer.

The "Little Demon" marketed by the I.C.I. Ltd. is a single shot exploder of the magneto type operated by a twist

handle. Although rated as "single shot exploder", sufficient voltage can be generated by turning the handle to fire six detonators in series.

The "Drake" is a 6-shot magneto type exploder. It is operated by a twist-action detachable handle.

BATTERY EXPLODER

The current is provided from a battery of 10 or 12 dry primary cells connected in series. It has an e.m.f. of 14 volts (1.5 V.

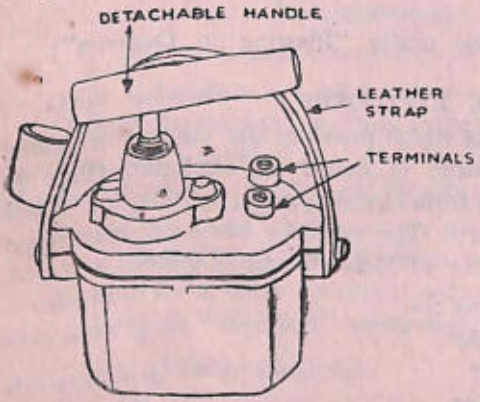


Fig. 3.

EXPLODER

per cell). It is operated by a detachable key. Rotation of the handle through half a revolution first winds up a time switch against the tension of a spring and then trips the mechanism which is controlled by a centrifugal governor so that the battery is connected to the firing leads during a predetermined interval of time (a small fraction of a second). At the end of this time the battery is automatically disconnected from the circuit. The operating handle returns to its original position when released. The voltage supplied to the circuit is not dependent upon the speed of rotation of the handle.

The battery exploder has the main advantage that owing to the low voltage and the absence of inductance, it is inherently incapable of producing under any circumstances a dangerous spark that can ignite fire damp.

M.E. 6 is a common type of exploder used in gassy mines. It is a battery-operated condenser discharge type and is

capable of firing six shots under normal field conditions. As an aid to the shotfirer this exploder contains its own firing circuit tester in the form of a circuit resistance indicator. It can be used in places requiring the use of permitted explosives. It weighs 4.5 kg complete with batteries and is only 164 mm in height and 184 mm in length. The high-tension battery is used to charge a 150 microfarad electrolytic condenser which is subsequently discharged through the external firing circuit. The latest exploder in the market is M.E. 12 which is capable of blasting 12 shots at a time and can be used in a gassy mine.

BEETHOVEN DYNAMO CONDENSER EXPLODER

This is not approved for use in gassy mines. It is a powerful multishot exploder. It comprises a generator which is operated by turning a handle. The alternating voltage generated by the dynamo is stepped up by transformer, rectified and used to charge a 6-microfarad condenser to a potential of not less than 1,200 V, (D.C.). A neon lamp indicates when the condenser is charged. When the firing button is pressed the condenser is discharged through the firing circuit, firing the shots.

The exploder is capable of firing upto 200 detonators connected in series and is widely used for blasting in stone drifts and shaft sinking. It may be used in mechanised quarries where heavy blasting is necessary.

It is a legal requirement to test the apparatus once in 3 months. A practice, highly objectionable, with some shotfirers is to use ordinary torch cells for firing a shot if the exploder is defective. Two torch cells give enough current to fire a detonator, and sometimes even two detonators in series, if the cells are new. Such practice is illegal and dangerous since any person can fire the shot with torch cells when the shot firer is busy with connections or is returning to a safe place after making all connections.

SEQUENCE SWITCH

A sequence switch is one which supplies electric current from a battery to the various branches of a shotfiring circuit in a definite sequence. In this respect it serves the same purpose as the use of delay detonators. Its use is permitted only for opencast mines. In one such switch which is of mechanical

Types of exploders for multishot firing are indicated in the following table :

Energy source	Name	Shot	Remarks
Dry battery ; Condenser discharge.	ME/6	6	Approved (P) May be used in any place where shotfiring is permissible.
Electronic ..	ME/12	12	Approved (P).
Hand-driven ; Direct generator discharge.	Schaffler 350	25	4 millisecond cut off. Limited approval (L) i.e. in places where non-permitted explosives are allowed.
Spring driven ; Direct generator discharge.	Shaffler 750	100	(L) -do-
Hand driven generator ; Condenser discharge.	Beethoven (modified)	100	(L) -do-
-do-	Beethoven	100	(N) used where non permitted explosives are allowed.

type the switch provides for 14 delays including the instantaneous shot and the electric current from a battery is supplied to the cordtex branches in sequence by an arm which sweeps over a series of contacts at a predetermined speed, which is a delay interval of 0.025 second.

Circuit Tester

In electric shot firing before any attempt is made to fire the shots, the circuit is sometimes tested to make sure that there is no open or shot circuit. Such testing should be done by approved apparatus and it is important that the current passed during testing should be limited, so that there is no possibility of accidental explosion of the detonators. In addition, all testing must be done from a safe place and at a safe distance from the blast. Two types of ohm-meters are available—The Pocket Safety ohm-meter and the Evershed and Vignoles safety ohm-meter. The Evershed and Vignoles safety ohm-meter

consists of a hand driven generator and a direct reading ohmmeter mounted together in plastic cases, the whole weighing 1.34 kg. When testing the circuit, firing cable is connected to the terminals of the apparatus. The generator handle is rotated at about 150 r.p.m. and the resistance of the circuit is read directly on the scale. This may then be compared with the calculated expected resistance, and it will indicate if the connections (giving high resistance) are bad or on open circuit (giving infinite resistance).

Such circuit testers are rarely used in mines before blasting except for important blasting in shaft-sinking and tunneling involving heavy charges.

Another type of circuit tester is a simple galvanometer with one torch cell in the meter. This is used only to know the continuity of the circuit of the firing cables and detonators. If the circuit is continuous, on connecting firing cables to the terminals, the galvanometer needle will show slight deflection. The testing is done from a safe distance just before actual blasting. It is used sometimes in mechanised quarries.

Shotfiring Cables

These are of following specifications as per Circular No. 70/1966 of the Chief Inspector of Mines.

(*a*) *Cable for single shot-firing.*

Two-core cable, each core consisting of a conductor of at least four copper wires of not less than 0.46 mm (0.018 inch) in diameter. For insulation and further protection against mechanical wear, the core should be covered by p.v.c., minimum 0.64 mm thick (0.025 inch) or by vulcanised rubber. The two cores should be twisted and covered with a sheath of cotton braid, tough rubber or p.v.c. External colour should be yellow.

(*b*) *Cable for Multi-shot firing.*

Single-core cable consisting of a conductor of at least three copper wires of not less than 0.92 mm (0.036 inch.) in diameter insulated to withstand at least 250 volts, and further protected against mechanical wear and damage. External colour should be white.

(c) Cable for shot-firing in shafts.

Two-core cable, each core consisting of a conductor of at least seven copper wires of not less than 0.74 mm (0.029 inch) in diameter and insulated to withstand at least 250 volts and adequately protected against mechanical wear and damage.

In these cables, the resistance of every 100 m of single conductor is as follows :

Twin core ..	4/0.46 mm	2.4 ohms.
Single core	3/0.92 mm	0.8 ohms.

5. TRANSPORT AND STORAGE OF EXPLOSIVES

Transport

Indian Explosives Rules, 1940, formulated under Indian Explosives Act, 1884, as modified upto 15th August, 1934, lay down certain restrictions regarding transport and storage of explosives.

"Rule 65—No explosive, exceeding 906 kg (2,000 lbs.) in weight, shall be transported by land otherwise than by rail, except in a locked vehicle of a type approved in writing by the Chief Inspector (Explosives). According to rule 63, no transport of explosives in any street or public place within municipal limit should be done, except under a permit of a District Authority."

The quantities of explosives, which can be transported by different modes of transport are :—

(a) Railway wagon (not more than 22,650 kg. in one train).	..	4,530 kg. (10,000 lbs.)
(b) Van (by road)	..	1,812 kg. (4,000 lbs.)
(c) Boat	..	22,650 kg. (50,000 lbs.)

Explosive van approved by an Inspector of Explosives may carry upto 3,624 kg. (8,000 lbs.) of explosives by road. No transport, carrying passengers, is permitted to carry explosives. The van used for transporting explosives more than 906 kg (2,000 lbs.) should be so constructed that iron or steel does not come in contact with the explosives. With this object iron and steel should be covered with wood, cloth, lead or similar

materials. The carriage should be marked EXPLOSIVES in bold letters. It should be locked and closed from all sides during transport and stoppages in transit near populated places should be reduced to the minimum essential. The transport van, carrying explosives, should be in charge of a competent person experienced in handling explosives.

Loading and unloading of explosives is prohibited between sunset and sunrise. On the mine premises, such van should never be parked at any other place, excepting the magazine, if it is loaded with explosives.

Small quantities of explosives used for underground mines can be carried by explosive carriers. One carrier is permitted to carry upto 5 kg of explosives at a time but the Chief Inspector of Mines may permit larger quantities (C.M.Reg. 163). A carrier should not carry detonators and explosive cartridges at the same time. The case or container from magazine in which the carrier carries explosives should be numbered and locked, preferably by a lock which is operated by the exploder key. The explosive carrier should not smoke when carrying the explosives, and should not use open type of lamp. Larger quantities can be carried by bullock carts, or on horse back in bad terrain if the distance is small. Still larger quantities need explosive vans.

It is illegal to transport explosives by jeeps or trucks, and only properly constructed explosive vans approved by the Inspector of Explosives have to be used.

The requirement of explosives for one blast in a mechanised quarry may vary from 100 kg where wagon drills (50 mm dia. bit) are used, to 1000 kg where wellhole drills (200 mm dia. bit) are employed and explosive vans are used for transport of such large quantities.

The restrictions upon transport of large quantities of explosives imposed under the Metalliferous and Coal Mines Regulations can be waived partly by the Director General of Mines Safety. The **main conditions** under which some of the large mechanised mines have been given such permission to carry large quantities of explosives are stated below.

[1. Transport of explosives from the magazine to the 'priming station' of the site of blasting shall not be done

except in day light and in the original wooden packing cases. The quantity of explosives transported at one time to the site of blasting shall not exceed the actual quantity required for use in one round of shots. The explosives shall be transported to the site of blasting not more than 15 minutes before the commencement of charging of the holes.

2. (a) No mechanically propelled vehicle shall be used for the transport of explosives unless it is a type approved in writing by the Chief Inspector of Explosives. Provided that a Jeep or a Land Rover may be used for the transport of detonators from magazines to 'priming stations' subject to the following conditions :

(i) Not more than 200 detonators are transported in a vehicle at a time.

(ii) The detonators are packed suitably in a wooden box.

(iii) The wooden box containing detonators is placed inside an outer metal case of a construction approved by the Chief Inspector of Explosives.

(iv) The outer metal case shall be suitably bolted to the floor of the vehicle or otherwise fixed in a wooden frame so that the container does not move about while the vehicle is in motion.

(v) No person shall ride on the rear portion of the vehicle.

(b) Every vehicle used for the transport of explosives shall be marked or placarded, on both sides and ends, with the word "EXPLOSIVES" in white letters not less than 15 centimetres high on a red background.

(c) Every mechanically propelled vehicle transporting explosives shall be provided with not less than two fire extinguishers (one of carbon tetrachloride type for petroleum fire and the other of carbon-dioxide-under-pressure type for electrical fire) suitably placed for convenient use.

3. (a) Explosives and detonators shall not be transported in the same vehicle.

(b) No person other than the driver and his helper (not below 18 years of age) shall ride on a mechanically propelled vehicle used for the transport of explosives.

(c) A vehicle transporting explosives shall not be driven at a speed exceeding 25 Kilometres per hour.

(d) A vehicle loaded with explosives shall not be taken into garage or repair shop and shall not be parked in a congested place.

(e) A vehicle transporting explosives shall not be refuelled except in emergencies and then only when its engine is stopped and other precautions taken to prevent accidents.

4. All operations connected with the transport of explosives shall be conducted under the personal supervision of a foreman solely placed in charge of blasting operations at the mine, hereinafter called 'the blasting foreman'. No person shall be appointed as a blasting foreman without the prior written approval of the Joint Director of Mines Safety.

The blasting foreman shall not allow priming of explosives on the approach of a thunderstorm.]

Storage

A building where explosives are stored is known as *magazine.*

Under the Explosives Rules the following can be stored together in the same magazine :—

Gunpowder	..	(Class 1)
Nitrate mixtures	..	(Class 2)
Nitrocompounds	..	(Class 3-Div. 1)
Chlorate mixtures	..	(Class 4)
Safety fuse	..	(Class 6 Div. 1)
Plastic igniter cord, connectors and Cordtex	..	(Class 6 Div. 2).

On no account must detonators be stored together with the above. Normally detonators are stored in a separate building, but if the number of detonators does not exceed 44,000, it is permissible to store them in a properly constructed annexe attached to the nitrocompound magazine. Cordtex, it will be noted, should not be stored in the same room as ~~main explosives~~ detonators.

Magazines are situated in places far away from dwelling houses, other places used or visited by men, railway lines, etc. so that if any accidental explosion takes place in the magazine,

the damage caused to public or property is the least. For these reasons isolated places in jungles or hills are preferred. An ideal situation for a magazine would be an area shielded by natural hills on all the sides, but this is not always possible and an area shielded on one or two sides by hills is considered good enough, the idea being that, if a bullet or cannon is fired at the magazine from a distance in a straight line, the hills should protect the magazine. Alternatively, the magazine can be surrounded on all sides by artificial earth mounds. The factory of Indian Explosives Ltd. at Gomia has taken advantage of the hills to locate the plants and magazines.

Apart from the main consideration stated above in selecting a site for a magazine the following factors also should be borne in mind.

1. Magazine should be on a well drained sloping ground. Underground magazines should not be built as these are invariably damp resulting in deterioration of explosives during storage.

2. The site should be accessible by road throughout the year and not only during the dry season.

3. The distance between the magazine and overhead power transmission lines, if any in the area, should be not less than 90 m.

The safe distances at which a magazine should be located away from places visited by men are specified under the Indian Explosives Rules and are given in *table* 3. It will be noted that the distance should be more for high capacity magazine, as the effect of accidental explosion in the large magazine would cover a wide area.

A magazine is to be constructed after receiving approval of the Chief Inspector of Explosives about site, safe distances and building plan. The licence for storage of explosives is to be renewed every year.

If the quantity of explosives to be stored is very small an application can be made direct to the local authorities, who are empowered to issue a 'J' form licence which covers the storage of up to 5 kg of nitrocompounds, 200 detonators and reasonable quantities of safety fuse.

TABLE 3

Table showing distance which should ordinarily be kept clear in and around magazine premises :

Quantity of Explosives.		(a) Inter-magazine distance		(b) Distance between magazine and magazine Keeper's or Chowkidar's quarters, public or private roads.	(c) Distance between magazine and any dwelling houses and private or public buildings.
		Gunpowder or High Explosives :		Gunpowder or High Explosives :	Gunpowder or High Explosives :
Kg.	Lbs.	Mounded m.	Unmounded m.	m.	m.
907	2,000	23	34	70	140
2,268	5,000	31	47	122	244
4,536	10,000	39	59	172	344
9,072	20,000	50		227	454
13,608	30,000	57		263	526
18,144	40,000	62		290	581
22,680	50,000	67		314	628
45,359	1,00,000	85		396	792

NOTE :—

(a) This table furnishes the basis on which applications for licences are considered.

(b) Detonators may be kept in an annexe near to or adjoining a magazine under the following condition :—

The amount of explosives contained in the detonator must not exceed 45 Kg. reckoned in the proportion of 1 Kg. of explosives per 1,000 detonators (This proportion represents roughly about 44,000 detonators of "sextuple" strength.)

If the quantity of explosives to be stored does not exceed 90 kg the application for a storage licence is simplified in that the applicant can obtain a "no objection" certificate direct from the District Authority. This document can then be forwarded to the Regional Inspector of Explosives together with the application form 'C' and the necessary plans as is the case for larger magazines.

Service Magazines

In a large mine, having 2 or 3 raising inclines/pits, service magazines are used. These are small rooms near the pits/inclines, where small quantities of explosives can be deposited for short periods during the working shift. Where a shot-firer has to use a large quantity of explosives in the mine, he need not carry all of it underground, but may keep part of the explo-

sives issued to him at the service magazine and draw them as and when required during his shift. A person is deputed to be in-charge of a service magazine whenever it is used. This arrangement is convenient to the shot-firers and the explosive-carriers whose travel time is considerably saved. Moreover, the

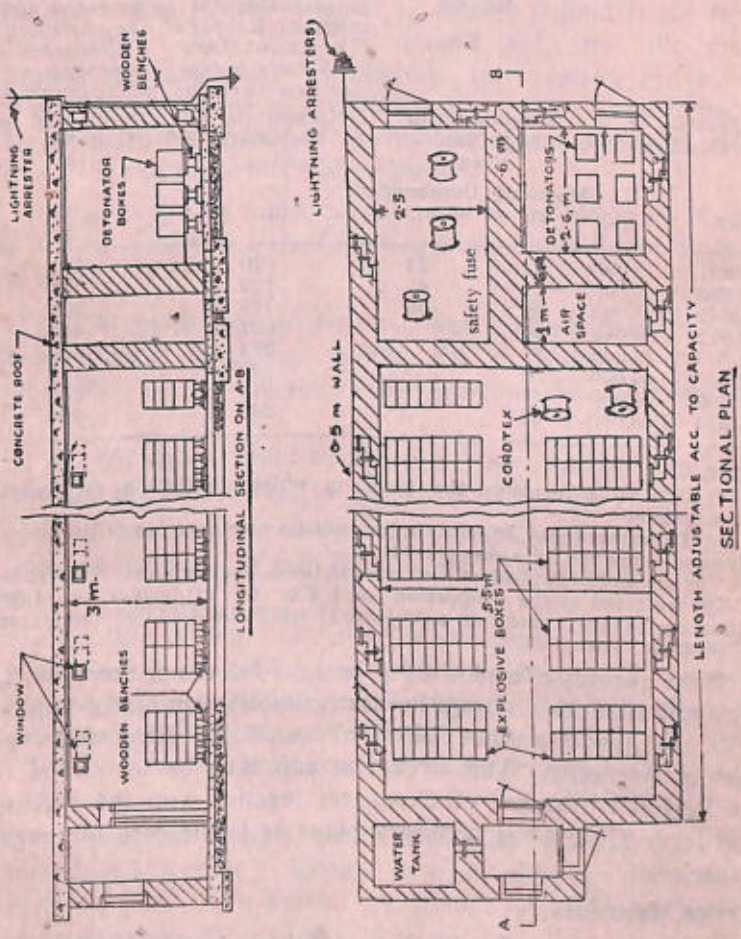


Fig. 4. Typical magazine for large quantities of explosives.

necessity to open the main magazine now and then does not arise.

Explosives should not be stored in underground workings and unused explosives should be returned to the magazine without delay.

A magazine is to be guarded round the clock and a daily record kept of the explosives received in stock, issued to blasters, and unused explosives returned to the magazine by the blaster.

Capacity of a Magazine

A mine with high output, e.g., a mechanised quarry, needs a magazine with very high capacity. Generally two months' consumption is stocked in the magazine. On this basis a magazine should have the following capacity :

1. *An Underground Mine*

Monthly production is, say, 20,000 tons of coal. Assuming 8 tons of coal is produced per kg. of explosive, 2,500 kg is the monthly consumption ; for 2 months, consumption is 5,000 kg for coal ; making a provision for blasting in drifts or stone partings, etc., a magazine with nearly 6,000 kg. capacity is desirable.

2. *Mechanised Coal Quarry*

Assuming yearly production of coal at 1 million tons, the quarry should be planned to produce 1 lakh tons of coal monthly.

1 kg. of explosive gives 12 to 13 tons of coal in a quarry.

Explosive consumption for 1 lakh tons of coal, 8,000 kg nearly. The volume of overburden to be removed for 1 lakh tons of monthly coal production will depend upon the ratio of overburden to coal. Assuming this ratio at 2.5, and the sand stone yield about 6 m^3 solid per kg of O.C.G., the explosive consumption for overburden will be about 30,000 kg monthly.

Monthly consumption of O.C.G. for coal and overburden will be 8,000 plus 30,000=38,000 kg.

Two months' consumption=76,000 kg.

If all the blasting in mechanised quarry is to be done by O.C.G. only, a magazine with nearly 75,000 kg capacity has to be constructed. It will be a huge magazine and at the same time will require large dimensions of safe distances, which may not be possible in the mine lease-hold. Moreover, the Inspector of Explosives may not approve of one magazine to stock such heavy quantity of 90,000 Kg. The problem can be tackled in the following ways :—

1. Part of the blasting should be done by L.O.X. which is not to be stocked in the magazine. The coal blasting may be done by O.C.G. and overburden blasting by L.O.X. or *vice versa*. That will reduce the required magazine capacity to 1/3 or 1/2 of the above. L.O.X. is, however, not available to quarries away from L.O.X. field depots.

2. Instead of one big magazine in the mine area, 2 or 3 small magazines should be constructed, all separated wide apart, in 2 or 3 remote corners of the mine lease-hold. The staff required to look after the magazine will be more in this case but the safe distances of the small magazines would be small.

3. The magazine will not be required to stock the explosives to its full capacity in the initial stages of the quarry as the production will not be heavy. For the smaller quantities to be stocked in the beginning a small magazine of 1/4th or 1/3rd the above capacity, which will, therefore, need smaller safe distances, may be constructed. As the quarry progresses, the overburden should be dumped on all sides of the magazine to make an artificial protection. In that case for the same safe distances the Inspector of Explosives can permit higher storage capacity of the magazine when the artificial mound is ready. This extra storage capacity can be arranged for by extension of the magazine building, or preferably, by providing for it during the initial construction of the magazine.

4. If a number of mines is situated close to one another, such mines can have small magazines of nearly 2,000 kg capacity or so, and a big central magazine to stock two months' requirements of individual collieries from the central magazine.

REFERENCES

1. McADAM, R., & WESTWATER, R., "Mining Explosives".
2. TAYLOR JAMES, & GAY, P. F., "British Coal Mining Explosives".
3. BHOSE, P., & WESTWATER, R., "Recent developments in shot-firing in coal mines". Journal of Mines, Metals & Fuels, Mechanisation of Mines-Special Issue, 1962.
4. PAL, B. N., "Explosives Manual".

CHAPTER 12

BLASTING PRACTICE IN UNDERGROUND AND OPENCAST MINES

1. Blasting at coal faces underground. 2. Blasting and drilling patterns in stone drifts. 3. Blasting in shaft sinking. 4. Blasting in quarries.

An explosive does its work by the sudden conversion of the explosive into large volumes of gases under pressure and at a high temperature. A detonating explosive is capable of doing work not only by the expansion of gases resulting from the explosion, but also by the shock wave which follows the process of detonation. The latter property is taken advantage of in plaster shooting in open-cast mines. If the pressure finds a weak path of escape, the gaseous products under pressure escape through such path, leaving only a little proportion of the gaseous products to do the work of breaking coal or rock. Such path is provided by a shot-hole if it is not properly stemmed. It is desirable that the resistance to the gaseous products should be the same in all directions. This ideal, however, cannot be achieved in stemming a hole, but the stemming material should be stemmed as hard as possible in the shot-hole when charging the latter.

If the explosive charge is more than what is required to do the work of blasting, the excess charge may eject itself out from the shot-hole without doing any work ; if the explosive charge is less than the required charge, it will be incapable of blasting out the rock and may eject itself out from the shot-hole by ejecting the stemming which can be more easily dislodged than the solid rock. The excess unburnt explosive, as well as the insufficient explosive, often in a smouldering state, coming in contact with gas or coal dust in an underground coal mine is a source of danger.

1. BLASTING AT COAL FACES UNDERGROUND

For blasting underground a shot firer has to carry the following tools with him.

1. A suitable electric lamp or torch which may be fitted with a red plastic cover over the glass at the time of the blasting.
2. A Stop-watch.
3. In the case of a gassy mine, a flame safety lamp for gas testing.
4. A wooden stemming rod, which is 2 m long and 25 mm diameter.

The wooden rod should not be shod with iron, brass or copper ring at the stemming end. As the stemming rods get broken often, it is necessary to keep a few spare rods in the underground store.

5. A scraper made of brass or wood suitable for cleaning out shot-hole.
6. Where fuses are used, a knife-cum-crimper for cutting off fuses and for crimping the detonators.
7. Where detonators are used, prickers made of wood or of a non-ferrous material for making an axial hole in the cartridge to prime it with a detonator.
8. A tool for detecting cracks in the hole.

In addition he should carry the cable for shotfiring and exploder. The cable should be well insulated and at least 18 m long.

Position of Shot-Holes in a Coal Face.

The position of shot-holes at a face in coal mines is governed by the following factors :

1. *Position of cut* :—It is observed that the best results are obtained in even ground when the distance between the shot hole and the nearest free face is not more than, and reasonably near to, 4/7th of the length of hole.

Free face : If the coal face is given a cut the coal, when blasted, is free to dislodge itself in the direction of the cut. The explosive charge is, therefore, able to dislodge more coal for the same charge than when the coal face is not given a cut. The cut acts as a "free face".

2. *Depth of cut* :—In gassy seams, a hole should be at least 150 mm shorter than the depth of the cut.

3. *Type of roof* :—If immediate roof is shale, the hole should be 0.3 m below the roof ; if the roof is sandstone, the hole should terminate 150 mm below the roof.

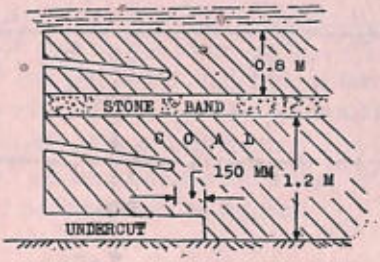


Fig. 1. Position of shot-holes (stone band present).

4. *Type of bands and their position* :—The hole should be drilled near the band. If the band is of hard material, the hole should be drilled towards it, slightly below or above, so that the inclined hole terminates near the band where the explosive force is maximum. The hole should be drilled below the band if it is near the roof, specially when the roof is weak or friable. Fig. 1 shows the position of holes at an undercut face with a stone band.

5. *Cleavage in the coal* :—Shot holes should cross the cleavage planes as near to right angles as possible.

6. *Strength of explosives and hardness of coal* :—Harder coal requires more density of holes or explosives.

7. *Dimensions of the gallery* :—A gallery of large cross-section gives better yield of coal per kg of explosives. A gallery with more height, say between 1.5 m and 2.4 m, requires two rows of holes, but only one row is sufficient for less height.

Fig. 2 shows the position of holes at an undercut coal face. It may be noted that the bottom row holes point towards the undercut. The holes nearer the cut are the first to be blasted, so that a free face is available to the next round of holes. The top row of holes points towards the roof, so that the coal parts from the roof in the blast. Where the face is not cut, the position of holes may be as shown in fig. 3 for solid blasting.

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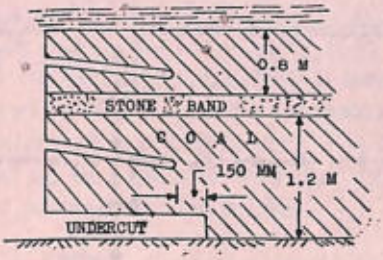

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A coal face which is blasted without cutting is said to be solid blasted. The pattern of holes for such face is any one of those described for stone drifting though wedge cut is quite common.

Where the strata are fissured, quick acting explosives, like the high explosives, should be used.

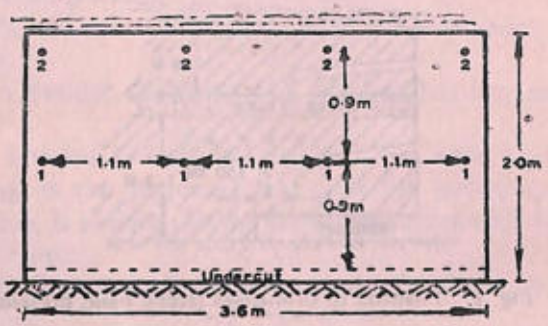


Fig. 2. Position of shot-holes ; undercut coal face.

In a heading generally 8 to 11 tons of coal is produced per kg of high explosives if the face is given a cut. A gallery 4 m wide and 2 m high would require 6 to 8 holes depending upon the hardness of coal.

The blasting at the face may be done by explosives or suitable substitutes for explosives.

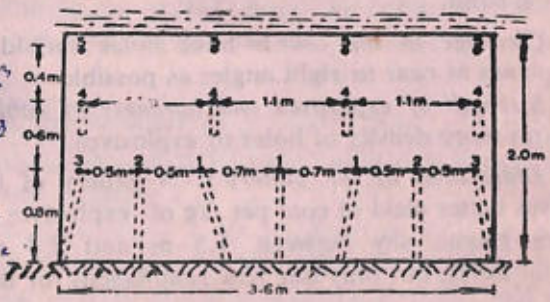


Fig. 3. Shot-holes for solid blasting of coal.

Preparation of a Charge

The charge for blasting in a hole may consist of one or more cartridges. It is desirable to have the least number of separate cartridges as far as possible, commensurate with the work to be done.

If gunpowder is used, one end of a safety fuse is freshly cut and inserted in the powder throughout the length of the cartridge and a length of 1.2 m to 1.5 m is cut for ignition.

For high and permitted explosives, one of the cartridges should have a detonator inserted into it. Such cartridge, equipped with a detonator, is called a "primer cartridge". If a plain detonator is used, saw dust should be removed from the detonator by turning it upside down, but *never* by tapping it on some hard surface. A freshly cut end of a safety fuse is inserted in the plain detonator as deep as it can go and the detonator mouth crimped on the fuse by crimpers, but not by teeth. The detonator is then ready for insertion in an explosive cartridge by pricking a hole in the latter with a pointed wooden pricker.

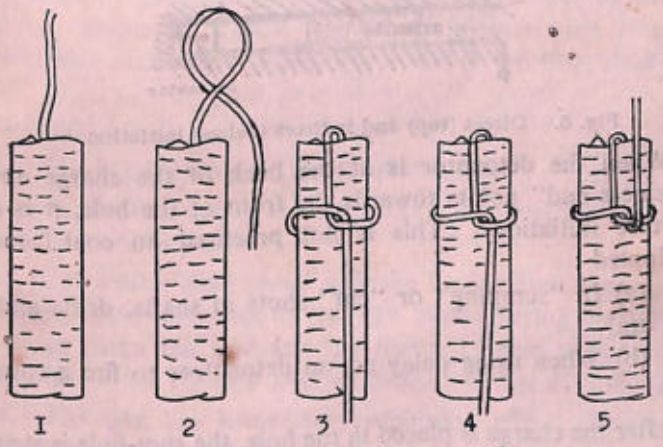


Fig. 4. Preparation of a primer cartridge.

If an electrical detonator is used to prepare a primer cartridge, open the cartridge at one end, make hole with a pricker of brass or wood, insert the detonator until it is completely buried in the explosive, put back the flap of the cartridge and hitch the leading wires around the cartridge to prevent the detonator being withdrawn accidentally during charging. Fig. 4.

Charging a Shot Hole

A cartridge of non-combustible stemming material is first pushed in the hole. The charge is then placed in the hole and the primer cartridge pushed last of all, so that the "business end"

of the detonator points towards the main body of the charge. This position of primer cartridge is termed "direct initiation". With this position of the detonator, the strongest detonation wave is directed towards the back of the hole and the chances of all the cartridges being properly exploded are a maximum. Direct initiation is best to prevent ignition of fire damp, reduces risk of blown-out shots, and gives maximum yield of coal having a free face. (Fig. 5).

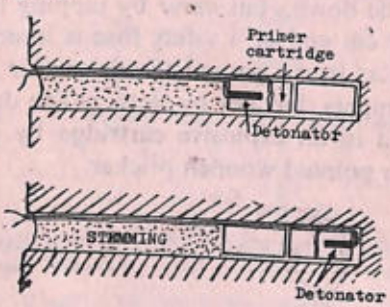


Fig. 5. Direct (top) and indirect (below) initiation.

When the detonator is at the back of the charge and the "business end" points towards the front of the hole, it is called "inverse initiation". This is not practised on coal faces, but is adopted :

(a) In "sumping" or "cut" shots in shafts, drifts and tunnels, and

(b) When using delay action detonators to fire a round of shots.

After the charge is placed in the hole, the shot-hole is stemmed with stemming material, keeping the leading wires of detonator or the safety fuse out of the hole. The stemming material should be compact but plastic, consisting of sand and clay in 3 : 1 proportion, mixed with a little of calcium chloride, and dried at the surface in the form of cartridges 152 mm to 203 mm long and 25 mm diameter. The first 2 or 3 stemming cartridges near the charge should be tamped lightly by the stemming rod and the rest should be tamped hard.

P.V.C. Water Filled Bags as Stemming.

In some mines water filled p.v.c. bags have been used for stemming. These bags are non-inflammable and produce less

coal dust and fumes, as the water from the bag, which is ruptured during blasting, allays dust and the fumes. The P.V.C. bag is 0.45 m long and 30 mm dia., filled with water to within 76 mm of the open end, and a knot is given at the empty end. Self sealing type of bags are also available and when filled with water such bag need not be given a knot. The water filling is usually done at the face, though it may be done on the surface as well. The bag is used as stemming material with normal explosives used in the mine.

When charging a shot-hole, a sand-clay plug of normal sand-clay stemming material is first inserted in the shot-hole as usual. The explosive charge is then inserted. This is followed by insertion of one or two water filled bags. Last of all, the sand-clay stemming material is inserted upto the mouth of the shot hole. A minimum length of 0.3 m of sand-clay stemming as the final insertion is desirable. Shot firing procedure is as usual.

Advantages

1. The bag is completely non-inflammable.

2. No capital outlay is necessary. The cost per bag may amount to only 20 paisa.

3. The visible dust cloud and the fumes from the explosive are drastically suppressed as the bag bursts during blasting and the water allays the dust and the fumes. There is 50% reduction in the lethal 1/2 to 5 micron content of the dust cloud.

4. The bag has somewhat cushioning effect, thereby producing more round coal.

5. Blasting efficiency is improved and nearly 15% saving in explosives may be achieved.

6. The bags may be used for blasting in drifting or tunnelling also.

COMPLETE PROCEDURE FOR THE SHOT FIRER

The whole procedure, which the shot firer has to follow to fire shots using electric detonators, is outlined as follows :—

1. Test shot-holes for breaks. If a crack is found, the hole shall not be charged.

2. Test for gas from the hole and within 18 m of the hole.

3. See that the shot-hole is 150 mm less than the depth of cut.

4. Mark direction of the shot hole on roof or side where practicable.

5. Charge the hole with explosives ; insert the primer last of all.

6. Stem the hole, first lightly and then hard, upto the mouth of the hole.

7. Spray stone dust or water within 18 m of the area.

8. Warn the workers to clear up and post helpers at suitable places 27 m away in approach roads to prevent worker inadvertently entering the area.

9. Lay out the shot firing cable.

10. Test again for gas within 18 m.

11. Couple the firing cable to detonator wires. If more than one shot is to be fired, all connections of detonator leads should be in series.

12. Take shelter.

13. Couple shot firing cable ends to the exploder.

14. Shout a warning again ; ensure that workers have taken shelter, and fire the shots by a sharp twist of the exploder key. If the charge does not explode, try again with a sharp twist of the exploder key.

15. Allow the fumes and gases to clear.

16. Return to the shot hole, examine the roof, sides and timber supports and shout "all clear" for the workers to return to their work if the place is safe. Otherwise, have it dressed by dressers, and supported by timbermen before workers enter the place.

In non-gassy mines, where a number of shots is fired by safety fuse at a time during one round, the shot-firer and another person should count the number of shots as they explode one by one.

At the end of the shift the shotfirer should write a report about the quantity of explosives blasted and the place of blasting.

2. BLASTING AND DRILLING PATTERNS IN STONE DRIFTS

Speed of drifting depends on the depth of "pull" (or advance per round) of blasting ; the greater the pull, the faster the speed as 'mucking' and roof supporting time remains almost constant.

After the alignment and gradient is fixed, the drifting starts with drilling shot-holes according to a specified form, called "drilling pattern", which consists of three groups of holes, namely,

(a) *Cut holes* or *sumpers* which are fired first to produce a free face for subsequent shots. They make the initial opening in the solid face and determine the depth of pull.

(b) *easers*, which are placed around cut holes to enlarge the excavation. They are in two or more rings depending on the size of the drift and are known as *first* or *second easers*.

(c) *trimmers* or *ring* shots, which are fired last to trim, dress and thus complete the excavation to the planned size.

Maximum depth of pull obtained in a normal size of drivage has been of the order of 2 metres in the recent past, but, of late, with "burn cut" pattern, a pull of 3 metres has been achieved.

Drilling Patterns

Drilling patterns are named after the type of cut holes used, and principal patterns are :

1. Pyramid cut or cone cut.
2. Wedge cut.
3. Drag cut.
4. Fan cut.
5. Burn cut.

In general, each hole in a round covers an area of 0.3 to 0.5 sq. metres ; cut holes are located about 0.45 metres vertically apart, and easers 0.5 to 0.6 metres ; trimmers are drilled at intervals of about 0.6 metres round the perimeter of the drift.

Pyramid cut or cone cut ;—Pyramid cut consists in drilling holes (in the centre of the drift axis) at corners of a square 0.7 to 1 metre *almost* to meet at a point at the back of the round. In a modified design known as cone cut, illustrated in fig. 6, 6 to 8 holes are drilled forming corners of a polygon with a centre hole,

all nearly meeting at a point at the back of the cut. The depth of pyramid cut is generally restricted to 50% to 60% of the width of the drift.

Wedge Cut :—Wedge cut takes the form illustrated in fig. 7.

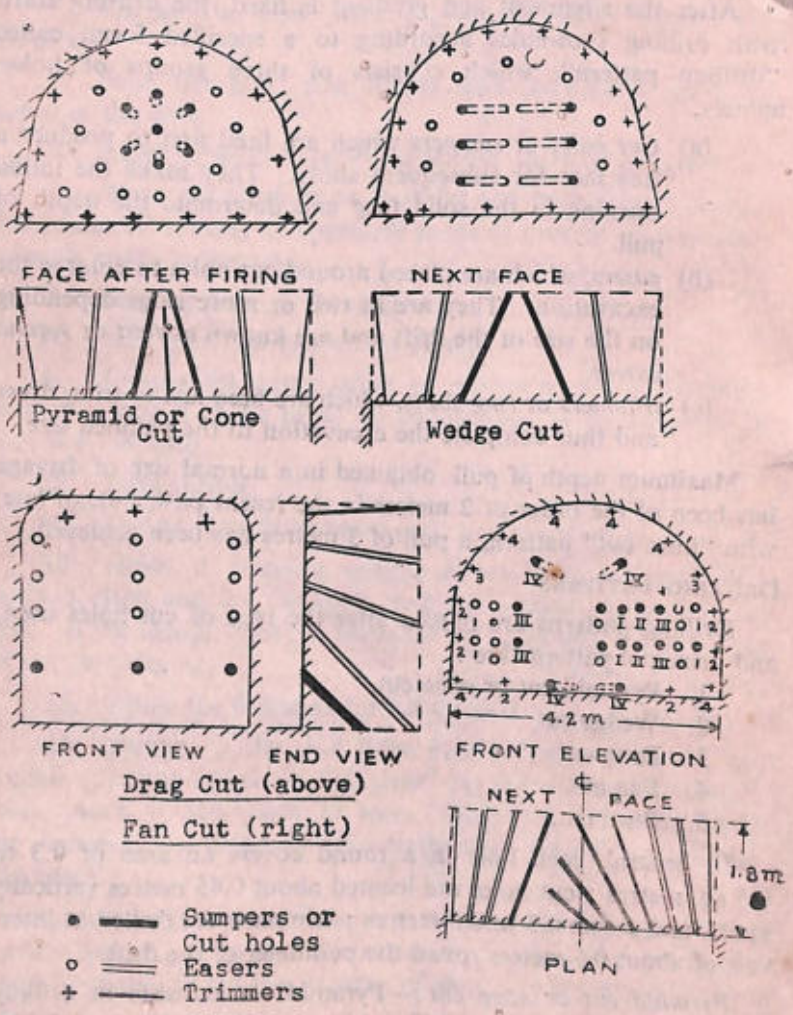
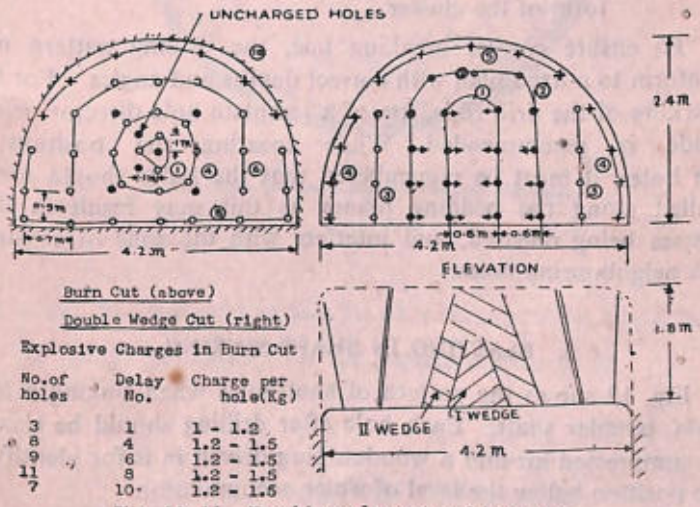


Fig. 6, 7, 8 & 9—In Fan cut, cut shots are fired first. Delays used are indicated by O, I, II, III, IV below hole position. Remaining holes are fired next. Delays used are shown as 0, 1, 2, 3 and 4 below hole position.

in which 2 to 4 pairs of holes are drilled to form a wedge, each pair starting from two sides of the drift centre and inclined at an angle less than 45° towards the centre *almost* meeting at the back of the cut along a line. A double wedge cut is shown in fig. 10 and the two wedges formed, one in the centre and the other next to it; may be noted.

Pyramid and wedge cuts, the most commonly used forms, are suitable for uniform, thickly bedded and hard rocks. They consume the least total quantity of explosives, but the depth of pull is restricted by the width of the drift.



Burn Cut (above)
Double Wedge Cut (right)

Explosive Charges in Burn Cut

No. of holes	Delay No.	Charge per hole (Kg)
3	1	1 - 1.2
8	4	1.2 - 1.5
9	6	1.2 - 1.5
11	8	1.2 - 1.5
7	10	1.2 - 1.5

Fig. 10, 11—Double wedge cut and burn cut.

Drag cut :—Drag cut, used for small drifts, 1.8 to 2.4 metres wide, in well-cleated rocks, consists in drilling holes at an angle to the cleat so that strata break along the cleavage plane (Fig. 8). This pattern, being dependent on the direction of cleavage planes, calls for frequent changes, which are detrimental to systematic work and the pattern is, therefore, not used for large excavations.

Fan cut :—Fan cut, favoured for laminated strata, mostly soft, covers the face with a fan-like pattern as shown in fig. 9. As each shot has to act for itself, charge in each hole is heavy. This cut is not recommended for hard ground.

Burn cut :—In Burn cut, parallel holes at right angles to the face are drilled in a cluster which may take several forms one

of which is illustrated in fig. 11. Some of the holes (which are sometimes of larger diam. than the rest) are left uncharged to give relief to the heavy concentration of explosives in the charged holes. This cut is effective in hard, brittle, homogenous ground which breaks evenly but cannot be used in springy plastic ground. The advantages claimed for this cut are :—

(1) drilling time is considerably reduced and supervision in drilling is easier as holes are straight,
(2) depth of pull is independent of the size of the drift, and
(3) the quantity of material projected is less with suitable form of the cluster.

To ensure correct breaking line, the drilling pattern must conform to a fixed plan with correct depths and angles. For this, marking of the drill rods, use of a template hole director or drill guide is recommended. While locating the position of the holes, it must be remembered that the holes should not be drilled along the bedding planes as this may result in large masses being released, and interfere with the zone of action of the neighbouring holes.

3. BLASTING IN SHAFT SINKING

Fig. 12 shows the pattern of shot-holes when sinking a large diam. circular shaft. Each hole after drilling should be cleaned by compressed air and a wooden plug driven in it for identifying the position below the level of water accumulation.

The cut or sumper shots, when blasted, provide the free face for other shots. Pyramid, wedge or burn types of cut are commonly employed. Explosives which are of high strength, water resistant and of high density are used as conditions a e always wet.

The primer cartridge should be inserted into the hole first with the detonator pointing towards the main charge. Stemming is with sand-clay mixture. Joints of detonator leads should be kept clear of water in the shaft ; with this object they may be allowed to rest on sticks stuck in the collars of the holes. The blasting cable after making connections with the detonators of the circuit, which are always in series, is taken to the surface and connected to exploder.

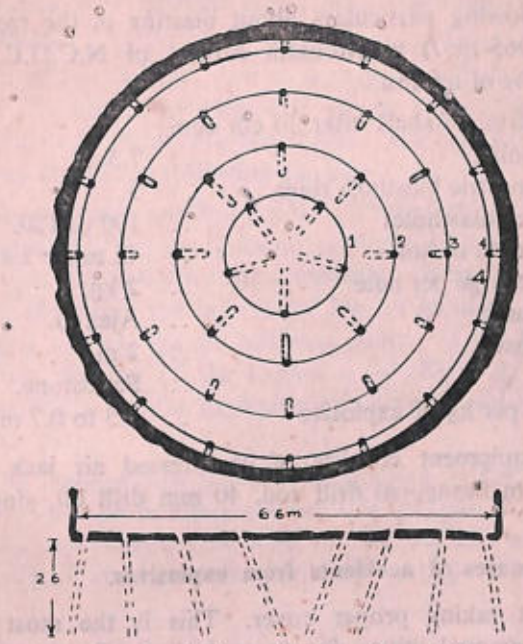


Fig. 12. Shot-holes in a sinking pit 6.6 m dia. and 2.6 m deep holes.

The details of blasting for the pattern shown in fig. 12 are as follows :

Ring	No. of holes per ring.	Charge per hole.	Total Charge per ring (kg).	Delay No. ½ Second Delays.
Sumpers	5	1.32	6.60	1
Inner Easers	8	1.10	8.80	2
Outer Easers	12	0.88	10.56	3
Trimmers	16	0.88	14.08	4
	41		40.04	

Explosive used 90% special gelatine
Nature of rock .. coal and shale
Volume of rock .. 88 m^3
Charge ratio .. 2.2 m^3 per kg.

The following particulars about blasting in the recent shaft sinking (1965-1967) at Monidih Project of N.C.D.C. (Jharia field) may be of interest.

Finished diam. of shaft after 30 cm concrete lining	..	7.5 m
No. of concentric blasthole rings	..	5
Total No. of blastholes	..	100 to 120.
Dia. and depth of holes	..	42 mm × 2.4 m
Explosive charge per hole	..	2 kg.
Explosive used	..	Ajax G.
Pull per blast	..	2 m.
Rock	..	Sandstone.
Rock yield per kg of explosive	..	0.5 to 0.7 m^3 (solid)

Drilling equipment consists of compressed air jack hammers with 25 mm hexagonal drill rod, 40 mm drill bit, single chisel, t.c. tipped.

Common causes of accidents from explosives.

1. Not taking proper cover. This is the most common cause of personal injury due to explosives. It is essential that the shot-firer shall himself take adequate cover and see that all workmen in the vicinity of a shot are removed to a safe place. No place in direct line with a shot can be regarded as safe and every person should be protected by at least one right angled corner. All approaches to the danger zone should be guarded by sentries or otherwise so as to prevent anyone entering inadvertently.

2. Failing to warn persons in an adjoining place into which the shot may explode.

3. Carelessness in handling detonators, causing them to explode, or to be lost in the mine.

4. Carelessness whilst charging a hole, *e.g.*, tamping too forcibly in the neighbourhood of the detonator, or ramming the primer cartridge into a hole of insufficient diameter.

5. Firing a shot when persons are at the shot-hole due to instructions being misunderstood, or lack of proper sentries.

6. Returning to the face too early after firing a round of shots, one of which is a "hang-fire" (*i.e.* a delayed ignition) or before authorised to do so by the shot-firer.

7. Dealing with misfired shots, otherwise than in the prescribed manner.

Ignition of gas or coal dust is a major accident which may arise from the use of explosives if proper precautions, as laid down in the Regulations, are not taken. The ignition may take place by any one of the following causes :

(1) By the flame and hot gases :—Though this is a common cause, because of "lag on ignition", a flame, coming in contact with gas, does not ignite it instantaneously. If the flame is sufficiently cooled during the interval of lag on ignition, which is a fraction of a second, the gas will not ignite. If the heat, generated during blasting of the explosive, is effectively converted into mechanical work, the explosion products are not hot enough to ignite the gas.

(2) By incompletely detonated explosive :—This may continue to burn like an ordinary combustible material.

(3) By incandescent particles coming out of the shot-hole after blasting if such particles come in contact with coal dust or gas.

(4) By the compression wave of the blast which may compress the gases in the cracks connected with the shot hole and raise the temperature of the compressed gas to such an extent as to ignite the gas. Some 20-fold compression is known to be sufficient to ignite all inflammable mixtures of fire-damp and air. The gas in the breaks and fissures connected with the shot-hole and leading to a fire damp pocket cannot be easily removed and the presence of breaks and fissures is one of the commonest causes in explosions of fire-damp due to shot firing. If any break is found, the shot must not be fired.

BLOWN OUT SHOT

A blown out shot is one which has not done any useful work of blasting coal, but has ejected itself out of the shot-hole. An undercharged hole will result in blown out shot. Similarly, in an overcharged hole, where only part of the charge is utilised in doing the useful work of blasting coal, the excess charge may still be burning when the coal is broken, and it has the same effect of igniting gas or coal dust as an undercharged blown out shot.

Misfired shots

When a detonator fails to explode, or after exploding fails to blast the charge, it is known as "misfire"

The reasons for misfire are as follows :—

1. Defective shot-firing exploder.
2. Defective detonator or bad quality explosive either due to bad manufacture or due to deterioration during storage.
3. Broken detonator leads, defective or broken shot-firing cable, bad connections between exploder and cable, between cable and detonator leads or at other places.
4. Short circuit of the cable or detonator leads due to poor or broken insulation.
5. Where fuse blasting is done, it may be due to (i) wet fuse (ii) improper timing of fuse, so that blasting by one fuse may cut the fuse of another hole, and (iii) the fuse being drawn out during stemming.

To guard against misfire, the cable should be checked before blasting, the exploder should be examined once every three months by competent persons, and only good quality of explosives and detonators, not spoilt during storage, should be used. It is better to have two single core cables for shot-firing separated by a good distance, instead of a twin core cable, which may be liable to short circuit if the insulation is bad. During use, all the connections should be carefully made. For important blasting work, the circuit should be tested for continuity by a galvanometer, described earlier.

DEALING WITH A MISFIRED SHOT UNDERGROUND

No person should enter the site of blasting for 30 minutes after firing. If elec. detonators are used, this time may be reduced to 5 minutes after cable is disconnected from the exploder.

All the entrances to the place should be fenced.

Another attempt should be made to blast it by making proper connections of the cable. If the misfired shot does not explode, the shot should be dislodged by drilling another relieving hole at least 0.3 m away from the misfired hole and blasting. The new hole should be drilled in the presence of the shot-firer who knows the direction of the misfired shot-hole, so that during drilling the drill bit does not touch the misfired charge.

After a relieving shot has blasted the coal, a careful search for misfired cartridges and detonators should be made in the presence of the shot-firer in the material brought down by the shot. If the misfired explosive is not traced, the material should be loaded in a separate tub, distinctly marked, for further search on the surface.

The misfired explosive and detonator, when traced, should be destroyed on the surface.

4. BLASTING IN QUARRIES

Blasting of large diam. holes drilled by wagon drills or well hole drills is known as *primary blasting*. If the boulders formed by primary blasting are too large to handle by the bucket of a shovel, they are further fragmented by small charges of explosives and this blasting is known as *secondary blasting*.

The explosives used in opencast mining are :

1. Opencast gelignite (O.C.G.)
2. Special gelatine.
3. Hectorite (under development).
4. Ammonium nitrate—fuel oil.
   (i) GN1 (ii) Anomex.
5. AN-T.N.T. slurry.
6. Liquid oxygen explosives.

1. Open Cast Gelignite

A ready-to-use explosive, unlike liquid oxygen, O.C.G. has to be stored in the mine magazine. Its main ingredients are nitroglycerine (about 60%), nitro cotton and ammonium mitrate. It is a high density, gelatinous explosive, well suited for watery holes due to its high water resistance. With a weight strength of 84, the explosive finds a wide application in hard rocks for open cast mining. The cartridges are wrapped in water proof waxed paper, and are marketed in sizes of over 75 mm dia. only.

The standard sizes are

83 mm dia.	× 355 mm	long weighing	2.5 kg.	–
125 do	× 305	do	5.0 kg.	–
175 do	× 270	do	8.3 kg.	–

for use with 3½" D hole
" " " 8" D hole.
8" D hole

2. Special Gelatine

A gelatinous explosive, it contains in addition to nitroglycerine, ammonium nitrate as one of the main ingredients. It is available in 3 strengths, 60%, 80% and 90%, these figures representing the weight strength. Cartridges of 60% strength are used for soft and medium hard rocks like coal whereas cartridges of 80% and 90% strength are reserved for hard and tough rocks like hematite. Special gelatines are widely used in metal mines, non-gassy coal mines, quarries, tunnels and other surface excavations. Polar special gelatines which have the same properties as special gelatines, can be used safely in temperatures down to —20°C as they incorporate ethylene glycol dinitrate which lowers their freezing point. Special gelatine is widely used for secondary blasting in quarries and is available in the following standard sizes for all strengths.

25 mm dia.	×	100 mm	long weighing	70 g.
25	do ×	200	do	140 g.
32	do ×	200	do	220 g.
50	do ×	220	do	625 g.

3. Hectorite

This explosive is still under development by I.E.L. and not available for sale. A powdery explosive, its main constituent is nitroglycerine. Its specific gravity is 1.0 and the explosive is not, therefore, suitable for watery holes. Its use is likely to be confined to coal and soft or medium hard rocks in dry conditions.

4. **Ammonium nitrate**

Ammonium nitrate, mixed with diesel oil, is used on a large scale in the iron ore mines of Goa and other quarries. It is the cheapest and also the safest explosive, but the ammonium nitrate which should be prilled and of fertiliser grade, is not produced in the country in sufficient quantities though Rourkela is likely to spare large quantities in the near future.

Two types of ammonium nitrate explosives are coming into general use, namely, AN-fuel oil mixture suitable for dry conditions and AN-TNT slurries for wet conditions.

The AN-fuel oil mixture can be prepared in a factory or at the blasting site. The mixture causes irritation of the skin and the workers should, therefore, wear gum boots and rubber hand gloves. The mixing should be done with wooden shovels avoiding contact with iron. The most effective mixture is produced by adding 5 to 6% of diesel oil by weight. In the dry season, 7 litres of diesel oil for 100 kg of ammonium nitrate suffice, but in wet season, the quantity should be increased to 9 litres. Percentage of diesel oil in excess of 10% tends to lower the sensitiveness. The AN-fuel oil mixture is much less sensitive to detonation than O.C.G. It is, therefore, safe to handle and can be mechanically loaded into blast-hole. As the mixture cannot be initiated in the normal way by a detonator it is necessary to prime it with a small quantity of O.C.G. or other high explosive. It is a good practice to use high explosive primer cartridges at the top as well as at the bottom of the hole and also for the deck charge. To break the "toe" of a bench satisfactorily, it is often advantageous to use a gelatinous explosive as a base charge. The primer is nearly 20% of the total charge. The AN-fuel oil explosives are difficult to sink in water due to low density of AN, and should preferably not be used in watery holes. If however, AN-fuel oil mixture has to be used in watery holes it should be packed in polythene bags and forced down the hole with the weight of a high explosive and the stemming above it. Bags may tear and allow water to ruin the contents, thereby giving poor blasting results.

AN-fuel oil mixture has a sp.gr. of 0.8 to 1, weight strength of 75-80 and velocity of detonation, 3,500-4,000 m/sec.

The AN-fuel oil mixture, as it has explosive properties, should be handled and transported with the same precautions as regular explosives and it should not be kept in the mixing rooms for more than 1 or 2 days, and should not be allowed to come in contact with carbonaceous material. The AN-fuel oil mixture gets solid during storage and it is better to use it early, within 10-15 hours after preparation.

For use in holes of 62 mm diam or less the AN-fuel oil mixture is normally not recommended as the propagation of explosion dies out much more quickly in small holes than in

large ones. For holes of smaller diam. special gelatine or other high explosives are superior.

Imperial Chemical Industries are marketing "Anomex" which is ammonium nitrate of a particular grade suitable for mixing with diesel oil at blasting site. They are also marketing GN1, an explosive which consists of ammonium nitrate, diesel oil and certain antisetting agents, in the form of cartridges wrapped in polythene. The use of GN1, therefore, entails the same precautions as for AN-fuel oil mixture and wet holes should be avoided.

5. AN-T.N.T. SLURRY.

Use of AN-T.N.T. slurry as blasting medium is of comparatively recent origin in foreign countries and is not so far introduced in India due to shortage of T.N.T. The slurry is usually a mixture of an explosive such as T.N.T. which is a reducing agent, ammonium nitrate which is an oxidising agent, and water. Sodium nitrate also may be used in place of ammonium nitrate. The T.N.T. is coarse and of −4+30 mesh size. AN/TNT/water in the slurry is in the ratio of 65/20/15. The addition of metallic powder to the slurry enables the explosive to reach very high strength. The slurry explosive has a density of about 1.4 and can be poured directly into watery holes. Like the AN-fuel oil mixture slurry explosive also cannot be detonated by a detonator and primer charge of high explosive is essential.

The slurry explosive is highly water resistant and blast holes have been fired weeks after charging with no ill effects on the explosive strength. The explosive has better blasting power and 50% higher detonation velocity than the AN-fuel oil mixture and is slightly costlier than the latter. The high density of slurry (1.4 to 1.6) compared with AN-fuel oil permits use of smaller diam. blast holes or wider spacing for the same results and the higher price of slurry may be more than offset by reduced drilling cost. They may prove cheaper than O.C.G. or similar high explosives. The blast hole diameter should be minimum 62 mm.

6. Liquid Oxygen (L.O.X.).

Liquid oxygen was used as explosive in India since 1930 in the quarries of Bokaro and Kargali which were then manually worked. Extensive use of L.O.X. in mechanised quarries started since 1956. In quarries L.O.X. can be used for coal also, though its use is prohibited underground. A given volume of liquid oxygen, when gaseous, is equivalent to 840 times at NTP ; *i.e.*, it has as much oxygen as would be available from

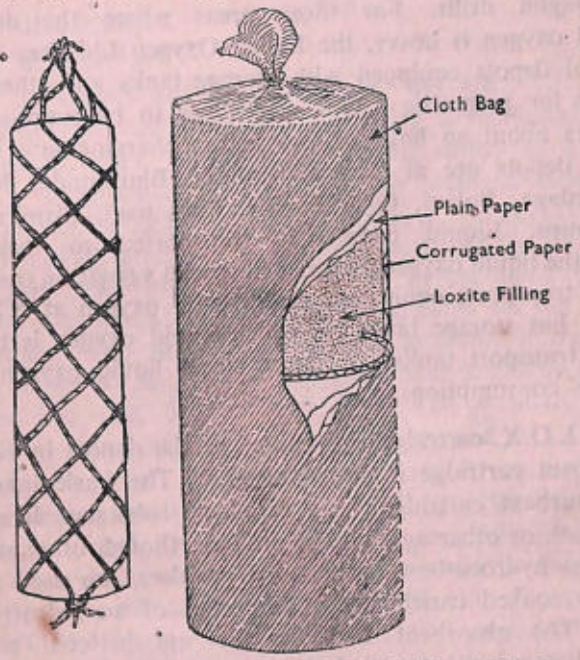


Fig. 13. O.C.G. (left) and L.O.X. (right) cartridges.

4,000 times its volume of atmospheric air. If a combustible ingredient, made in the form of a cartridge, is soaked in liquid oxygen and then subjected to reaction of combustion, the reaction takes place with such terrific speed that large volume of gas is instantaneously released at high temperature so as to cause explosion. The velocity of detonation under suitable conditions of confinement can be faster than 5,000 metres/sec. This is the principle behind the use of L.O.X. as explosive.

Liquid oxygen boils at—183°C. It is manufactured at the works of Indian Oxygen Ltd. at Jamshedpur, Burnpur, Calcutta

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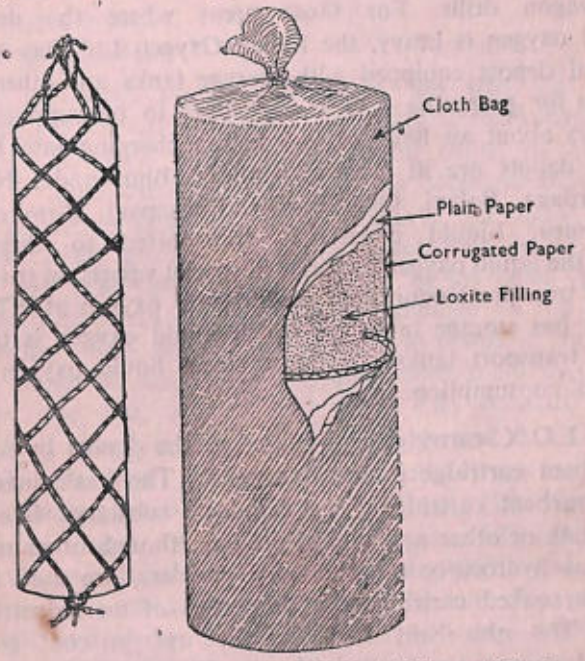

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Liquid oxygen boils at—183°C. It is manufactured at the works of Indian Oxygen Ltd. at Jamshedpur, Burnpur, Calcutta

and other industrial centres. From there it is supplied to small consumers within 300 km in special containers of 25 litres by passenger trains. During transit some evaporation, 5 to 10%, takes place; hence this limit of 300 km. The small consumers have to obtain loxite absorbent cartridges, soaking boxes and other equipment to prepare the L.O.X. cartridges on the spot. Only small cartridges are prepared at the small quarries in this manner for use in holes drilled by jack hammers or wagon drills. For those areas where the demand of liquid oxygen is heavy, the Indian Oxygen Ltd. has established central depots equipped with storage tanks and other arrangements for preparing L.O.X. cartridges to be supplied to consumers about an hour or two before charging into blastholes. Such depots are at Bermo, Kathara, Bhurkunda, Noamundi, Lohardaga, Bolani, Gua, Barsua, Purnapani, Birmitrapur and Kiruburu. Liquid oxygen is transported to these depots from the liquid oxygen factories in special vessels on trucks. One truck transports equivalent of 2,970 m^3 oxygen at NTP. Each depot has storage tanks to which liquid oxygen is transferred from transport tanks. A depot stores liquid oxygen for 3 to 6 days' consumption.

A L.O.X. cartridge is prepared at the depots by soaking an absorbent cartridge in liquid oxygen. The basic ingredient of an absorbent cartridge is a cellulosic substance like crushed jute-stalk or other agricultural product though other substances, such as hydrocarbons or metallic powders, are used to impart to the soaked cartridge the properties of an industrial explosive. The absorbent cartridge for use in coal contains a special composition called "Loxite-C", with a view to reduce the flash of flame sometimes seen in the coal dust cloud formed in the air after blasting. The absorbent cartridge consists of the above ingredients wrapped in paper covered by cloth. Absorbent cartridges are manufactured at Loxite factory at Ranchi and are kept in stock at the depots.

Soaking of absorbent cartridges is carried out in soaking boxes which are double walled, insulated rectangular boxes with internal dimensions of 1,000 mm×750 mm×1,000 mm depth and are constructed on the principle of ice box to reduce evaporation of liquid oxygen. To prepare L.O.X. cartridges, the

absorbent cartridges are arranged in the soaking boxes and liquid oxygen is poured from the storage tank by pressure syphon. It takes nearly 1½ hours to fully saturate them. L.O.X. cartridges are not stored in colliery magazines, but are supplied by depots on 2 to 3 hours' prior intimation of the exact requirement for blasting.

The L.O.X. cartridges are of two groups :

(i) small cartridges,
(ii) large cartridges.

Small cartridges are those of diam. less than 100 mm and are used for secondary blasting or blasting of small diameter holes drilled by jack hammers or wagon drills, as stated earlier. The large cartridges are used for wellhole blasting. For small as well as large cartridges there is a "Standard" cartridge. For small group, the standard cartridge is 38 mm dia. × 300 mm long and other cartridges of the small group are accounted in terms of the standard cartridge on the basis of the volume. The small cartridges are available in various diameters, minimum is 25 mm and maximum is 90 mm and the lengths are 150, 300 or 600 mm. The standard cartridge of large group is called Full Cartridge and is 190 mm dia. × 600 mm long. The large cartridges are of the following diameters : 120 mm, 133 mm, 172 mm, 190 mm, 210 mm and the length is either 300 mm or 600 mm. The cartridges are specified in terms of the volume of the standard cartridge *e.g.*, one cartridge of 120 mm × 300 mm is specified as 0.20 full cartridge'; a cartridge of 170 mm × 600 mm is specified as 0.80 full cartridge. In explosive performance a full cartridge, according to L.O.X. dealers, is considered equivalent to 18 kg of conventional explosive.

One soaking box contains upto 22 full cartridges *i.e.*, 22 × 18 = 396 kg. The soaking box is mounted on a trailer and the soaking boxes containing ready-to-use cartridges are transported to blasting site by tractors. One tractor carries 2 soaking boxes *i.e.*, a maximum of nearly 750 kg. The exhaust of the tractor should not face towards the soaking box, but away from it, to prevent accidental explosion due to hot carbon particles of exhaust coming in contact with L.O.X.

The L.O.X. cartridges should be used in the field without delay within 1/2 hour to prevent loss of absorbed liquid oxygen. Grease or oil should not come in contact with the cartridges at any stage. If the oxygen of a L.O.X. cartridge evaporates, the absorbent cartridge can be used again to prepare L.O.X. cartridge at the depot. The "Life" of a L.O.X. cartridge in open is about one hour.

The L.O.X. cartridge is net handled barehanded but with gloves. The cartridge loses its oxygen by gradual evaporation if it comes in contact with water. For use in watery holes, the cartridge is wrapped in a polythene bag before lowering in the hole. The toe will not be sufficiently loosened, if L.O.X. is used without such precautions in watery holes.

L.O.X. can be fired with or without the help of a detonator. A hole charged with L.O.X. can be fired with safety fuse alone like gunpowder. In small-hole blasting, detonators are not much used as safety fuse is economic. Holes deeper than 3 m are fired by a detonating fuse.

India has not to depend on foreign countries for L.O.X. which is completely indigenous. In case of misfire, there are no worries as the liquid oxygen evaporates within 2-3 hours and the area is safe. Its use is however limited to large quarries lying within 50 km of the L.O.X. field depots. In the case of small quarries which receive the L.O.X. in containers of 25 litres by passenger trains, the use of L.O.X. is limited to quarries lying within 300 km of the oxygen manufacturing plant. Price of L.O.X. is Rs. 36/- per cartridge of 18 kg and varies slightly with consumption.

CORDTEX

O.C.G. and L.O.X. have to be detonated for heavy blasting. No. 6 detonator which is commonly used is inserted in the cartridge itself and detonated by ignition of a safety fuse, or in the case of electric detonator by an exploder. For the large number of holes blasted at a time in mechanised quarries, electric detonators are used. The depth of hole needs long length of detonator leads and to avoid this it is common to use a detonating fuse known as 'Cordtex'. This fuse consists of a core of pentaerythritol tetranitrate (P.E.T.N.) enclosed in a tape which is wrapped with cloth. The fuse is then completely enclosed

in a tubular cover of white plastic material. The external diameter of cordtex is 5.0 mm and the weight 21 g per metre length. Each metre of cordtex contains 11 g. of explosive. It is safe to handle and will not deteriorate even after several years of storage. It has a high velocity of detonation (6,500 metres/sec.). For detonating an explosive, it is necessary that the cordtex fuse should be in close contact with the former. Cordtex needs a detonator to detonate, and No. 6 detonator is suitable. Cordtex has high tensile strength, 54 to 58 kg and high resistance to abrasion, which reduces the chances of failures caused by rough handling. It is supplied by the Imperial Chemical Industries, Ltd.

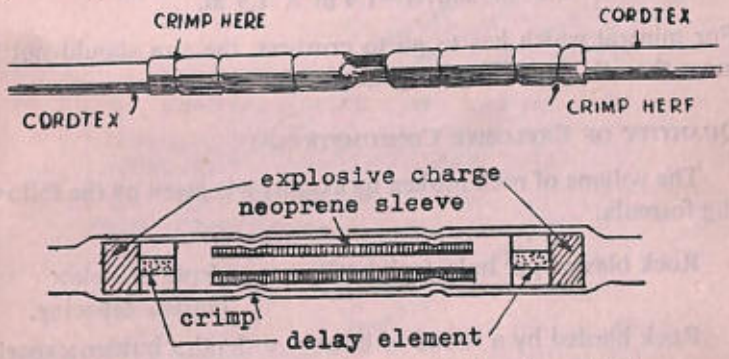

Fig. 14. Detonating relay (above) and below (in section).

Detonating Relays

In quarries and opencast workings, detonating relays using 'Cordtex' fuse for initiation provide an excellent means of obtaining short delay periods in blasts. This method avoids the electrical connections which are required when using delay detonators. A Detonating Relay is essentially an assembly of two open-ended delay detonators coupled together with flexible neoprene tubing in an aluminium sleeve suitable for crimping into a 'Cordtex' line. Inside the detonating relay, the construction is symmetrical with the delay element at either end so that the detonation wave may pass in either direction. The delay interval for each detonating relay is nominally 15 millisecond with a "scatter" of ± 3 millisecond. In use, the Cordtex main or branch line is cut at the point where a delay is

required, and the detonating relay is then crimped between the two cut ends of the line. By judicious selection of the points at which the detonating relays are inserted, any delay firing sequence can be arranged.

Efficient blasting should give such rock fragmentation as to eliminate the need for secondary blasting. The fragmented rock should be easily handled by the bucket of the shovel. Size of bucket mouth is usually as follows :—

2 m^3 shovel—1.2 m × 1.4 m.
3.5 m^3 shovel—1.4 m × 1.7 m.
4.5 m^3 shovel—1.4 m × 1.9 m.

For mineral which has to go to crushers, the size should not be more than the input size for primary crusher.

Quantity of Explosive Consumption

The volume of rock broken by explosive is given by the following formula.

Rock blasted per hole, (solid m^3) = depth of hole × burden × spacing.

Rock blasted by a round of holes = depth × burden × spacing × number of holes in the round.

The volume of rock blasted per kg of explosive, as observed in practice by the authors and also by other experienced mining engineers, is given below. The manufacturers of explosives sometimes claim to have obtained better results which are not impossible as there are a number of factors influencing the blasting performance.

	Solid m^3		tonnes	
	Coal	Sandstone	laterite	hematite
per kg of O.C.G.	6.5-10	5-6.5	9 to 11	7-8
do. L.O.X.	5-8.5	3.4-5	7-8	6-7
do. AN-diesel...		not much used	7-8	6-7

The following table gives an idea of explosive consumption in quarries (O.C.G. or 80% special gelatine) for various rocks under Indian conditions (opencast mines).

	Rock	Explosive consumption, kg/m³
1.	Bituminous coal	0.1
2.	Medium hard shales, sandstones, gypsum	0.2
3.	Sandy shales, sandstone	0.3
4.	Massive sandstone, limestone, laterite	0.4
5.	Very hard shales, marbles, dolomites, limestones, magnesites, hematite	0.5
6.	Extremely hard limestone, conglomerate	0.7
7.	Gneiss, granite, amphibolite, schist	0.9—1

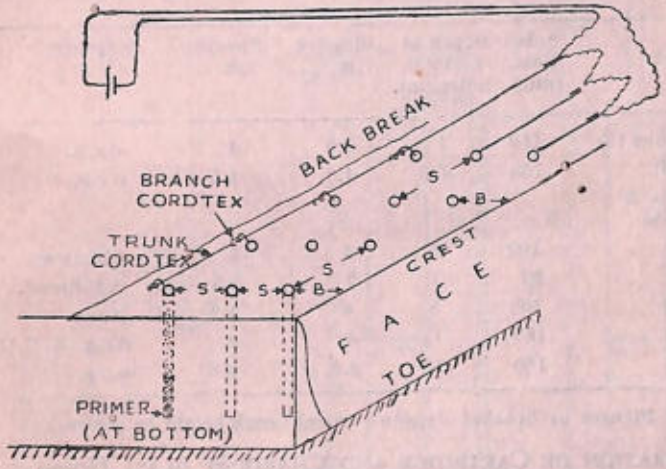

Fig. 15. S=Spacing B=Burden.

Detonator nearest the crest has lowest delay and farthest from the face has the largest delay number.

Depth and Pattern of Blast Holes

The depth of vertical blast holes in coal is generally equal to the height of the bench. In hard rock like sandstone, laterite, hematite, etc. the depth should be 0.5 to 1 m deeper below the level of bench floor. This loosens the toe. In any case, the hole should terminate in hard rock and not in the soft one ; otherwise the force of explosive is wasted.

Burden is the minimum distance from the face to the blast hole nearest to it. (Fig. 15).

Toe is the projection of the bottom of a face beyond the crest. Sometimes the bottom edge of the bench is referred to as toe.

In hard rocks like laterite and hematite the spacing and burden vary from 0.3 to 0.4 times the height of Bench. In less hard rocks like coal and sandstone, the spacing and burden vary from 0.5 to 0.8 times the height of bench. The exact diamensions depend upon the hole diameter, type of explosive used, type of the rock, nature of rock consolidation and the angle of cleats or laminations with the blast hole. These dimensions have to be decided by experiments for the particular rock and the locality and the following practical observations may serve as a guide.

Rock	hole diam. (mm)	Depth of vertical holes (m).	Burden m	Spacing m	Explosive
Sandstone (6)	150	7	4.2	4	o.c.g.
Coal (6)	150	5.5	4.5	4.5	o.c.g.
Laterite & hematite :					
(6)	100	7	3	3	AN-diesel.
(6)	62	7	2	2.5	AN-diesel.
(7)	160	8	4	4.2	o.c.g.
(10)	160	11	4	4	o.c.g. & G.N.I
(14)	160	15	4.5	4.6	o.c.g.

Figures in bracket denote vertical bench height in metres.

Preparation of Cartridge and Charging Blast Holes.

For blasting a number of deep holes simultaneously cordtex fuse is used. The primer cartridge is prepared by pricking a through hole in O.C.G. cartridge with the help of a brass or a aluminium pricker and threading cordtex fuse through it. After threading, a knot is made at the end of the fuse in contact with the cartridge to prevent the fuse from coming out. This is the primer cartridge. No hole is pricked in L.O.X. cartridge, but the cordtex fuse is coiled at the neck of the cartridge in the form of a loose knot, so that some length of cordtex fuse is in close contact with the explosive along its length.

The primer cartridge is always placed at the bottom of the hole in case of O.C.G., and also in case of L.O.X. for deep hole blasting. It is lowered by the cordtex fuse tied to it. O.C.G. cartridges are supplied with a separate net covering and the

cartridge is lowered by attaching a self-detaching hook to the net. L.O.X. cartridges have a loop of copper wire round the neck. The hook is attached to the copper wire loop and the cartridge lowered in the hole. After the cartridge rests in its place in the hole, the hook comes out of the net or the copper wire loop when the rope is slack. Cartridges may be dropped down the hole on a cushion of, say, a special pad, drill cuttings or stemming or even water, if the depth is small, say upto 13 m.

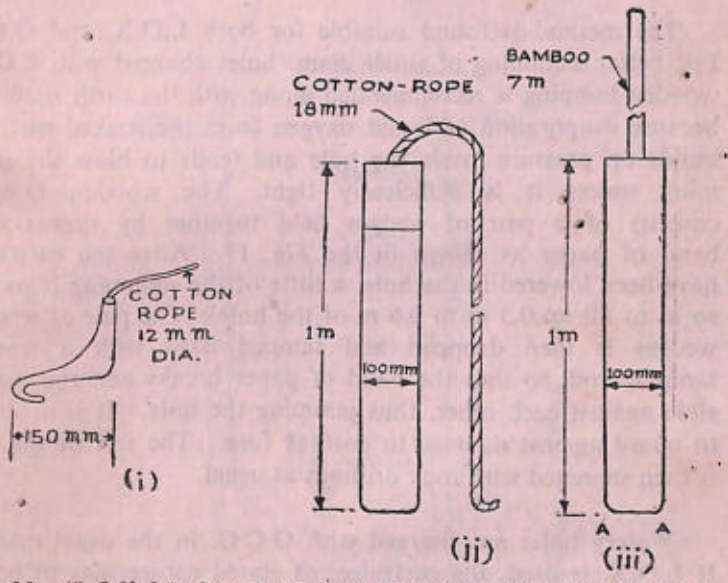

Fig. 16. (i) Self detaching hook. (ii) Tamping dolly. (iii) Stemmer.

It is a good practice to restrict the length of the explosive column in the hole to less than one third of its depth in overburden (one fifth of the depth of the hole in case of coal). Excessively charged holes tend to shatter at the neck region, the fragments being projected to considerable distances.

STEMMING

The drill cuttings in overburden are used for stemming. They are made moist with water and pushed into the hole. The stemming is packed by tampers which are of two types. One type has a wooden log 1 m long and of diameter 25 mm less than the hole diameter. It is pulled 0.6 to 1 m at a time

by the rope attached to it and allowed to freely fall on the stemming material, so that the stemming is packed by the impact. The other type of tamper has a wooden log nearly one m long. A long bamboo is attached to it. This latter type is suitable for holes nearly 6 m deep. For deeper holes, the tamping dolly with rope is convenient. The edges AA are rounded to avoid cordtex being cut by sharp edges of the tamper. The tamping of stemming material is done at intervals till the complete hole is stemmed.

The method is found suitable for both L.O.X. and O.C.G. For better stemming of small diam. holes charged with L.O.X., wooden tamping is recommended along with the earth stemming because evaporation of liquid oxygen from the soaked cartridge builds up pressure inside the hole and tends to blow the stemming unless it is sufficiently tight. The wooden tamping consists of a pair of wedges held together by means of a band of paper as shown in the *Fig.* 17. After the cartridges have been lowered in the hole, a little of the stemming is pushed so as to fill up 0.3 m to 0.6 m of the hole. The pair of wooden wedges is then dropped and tamped hard with a wooden tamping rod, so that the band of paper breaks and the wedges slide against each other, thus jamming the hole. It is necessary to guard against damage to cordtex fuse. The rest of the hole is then stemmed with rock drillings as usual.

Watery holes are charged with O.C.G. in the usual manner. If L.O.X. is used, the cartridge, as stated earlier, has to be enclosed in a polythene bag before lowering. After the calculated number of polythene covered cartridges are lowered down the hole, sand bags of an appropriate size are slipped in on top of them to anchor them at the bottom of the hole and to prevent them from floating to the surface as the liquid oxygen evaporates. No stemming is used as water is considered an effective stemming by itself. Gas bubbling through the water should not distract attention as the blasting efficiency is not affected if blasting is not unusually delayed.

The cordtex fuse is cut after leaving nearly 1 m length outside the hole for attachment to trunk fuse.

Deck Loading

Where the overburden consists of intermittent bands of hard and soft rock, concentration of all the explosive charge at the bottom of the hole does not give desired fragmentation. Hard bands of rock at intermediate depths are not well fragmented and secondary blasting becomes necessary. It is

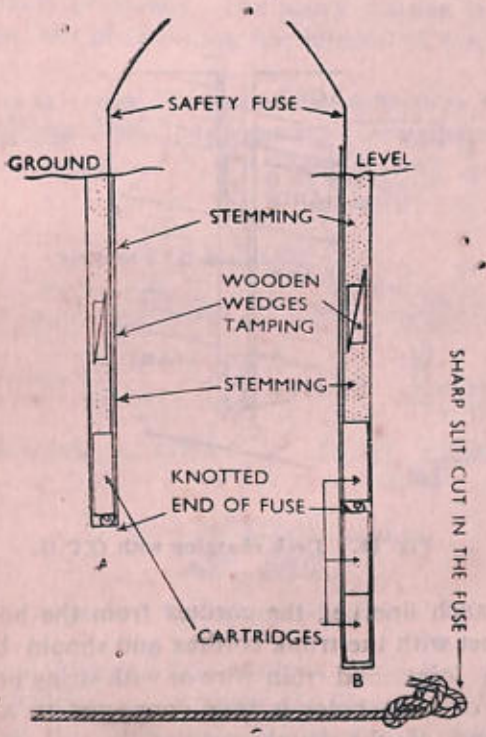

Fig. 17. Short Hole Blasting with L.O.X.

A—When only one cartridge has to be used the knotted end of the fuse is inserted first, then the cartridge.

B—When more than one cartridge is used, the knotted end of the fuse is inserted before dropping the last cartridge.

C—A typical knot in the fuse (enlarged).

useful in such cases to separate the blasting charge of the hole by stemming material into sections, so that some charge is in contact with hard bands at intermediate depths. The driller

should note during drilling the depth and extent of the hard band and inform the shot-firer. Cordtex fuse of the primer used for the bottom-most charge is used for the deck charge also as shown in Fig. 18. It is sometimes preferable to use a separate primer cartridge, but the same cordtex line, for each deck charge. The primer cartridge in the deck charge in such cases should be at the bottom.

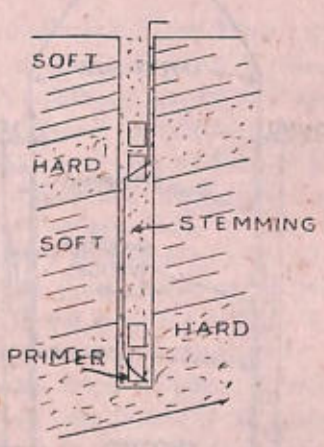

Fig. 18. Deck charging with O.C.G.

The branch line i.e., the cordtex from the hole should be in close contact with the trunk cordtex and should be secured by a clove hitch joint and thin wire or with string or adhesive tape. Each row of blast holes is thus connected to a trunk cordtex. To one end of the trunk cordtex is tied a detonator, (or two, to take no risk of a misfire) by a thin wire or adhesive tape, so that base charge of the detonator is in close contact with the side of the cordtex. The detonator should point along the cordtex towards the explosive charge. Where two or more rows of blast holes are to be fired in one round, delay detonators are used. Short delay detonators or milli-second detonators give good fragmentation and reduce ground vibrations, the latter advantage being important where quarries are situated near areas having buildings and costly engine foundations.

The blasted rock pieces in mechanised quarries fly upto 200 m. All the workers within 250 m of the blasting site should withdraw to safe places. Warning is given by the shot-firer half an hour before the blasting and again, 5 minutes before the blasting by an electric siren, or alternatively a bugle. All the machinery like shovels, drills, etc., should be withdrawn to a place where it is not likely to be damaged by blasted rock or should be suitably protected. The heavy blasting in mechanised quarries should take place during rest interval of workers.

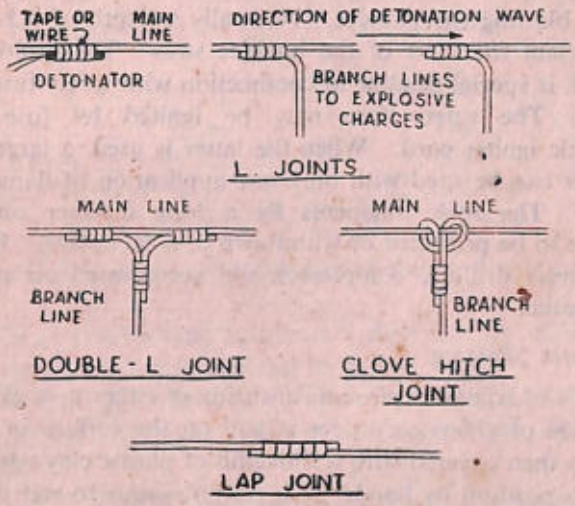

Fig. 19. Cordtex connection.

The electrical circuit of detonators is tested for continuity by a circuit tester which is essentially a galvanometer. When everything is in order, the cable from detonator leads is connected to an exploder and the explosives are blasted. The "Davis Kerby" and ME6 exploders capable of firing 6 detonators in series serve the purpose in most cases, as the number of detonators fired at a time is only 4 or 5.

After blasting of a heavy round of L.O.X. in coal is over, the blasting foreman should depute one of the blasting crew to watch for sign of smoke emanating from the blasted coal for an hour after which he should declare the place safe. If smoke is seen to be persistently coming from the blasted coal, arrangement

should be made for pouring sufficient water on the site to ensure quenching the trapped heat if any.

Secondary Blasting

Secondary blasting is carried out in two ways :

1. pop shooting.
2. plaster shooting.

Pop shooting

A hole is drilled by jackhammer for charging with explosive and blasting the boulder. Normally a depth of 0.3 to 0.6 m is sufficient for most of the boulder sizes. The explosive, widely used, is special gelatine in conjunction with safety fuse or detonators. The safety fuse may be ignited by fuse igniter or plastic igniter cord. When the latter is used, a large number of shots can be fired with only one application of flame to the circuit. The rock fragments fly a long distance and machines have to be protected or withdrawn to a safe place. For the jackhammer drilling, compressor and compressed air pipe line are essential.

Plaster Shooting

A charge of explosive consisting of either a single primed cartridge or a few cartridges is laid on the surface of the boulder. It is then covered with a shovelful of plastic clay which is pressed into position by hand. It is advantageous to wet the surface of the stone before plastering and the clay should be well pressed down for good contact with stone round the explosive. Special gelatine, polar viking or polar ajax can be used for the blasting I.C.I. has developed an explosive known as "Plaster Gelatine" for this purpose. It is a high velocity, high strength gelatine type explosive, and suitable for such work.

It may be noted that a detonating explosive is capable of doing work in two ways : (a) by the shock wave which accompanies its decomposition and (b) by the expansion of gases resulting from the explosion. It is this shock wave effect which is widely used for plaster shooting. The shock from an explosive can induce a system of cracks in the stone and the amount and severity of cracking depends on the area of contact and velocity of detonation.

Plaster gelatine is detonated with No. 6 detonator. It has high water resistance and good storage properties. It is available in cartridges and also in slabs 88 mm×63 mm×12 mm. They are readily moulded to give a good contact with the surface of the boulders.

The explosive used in plaster shooting is nearly four times that required for pop shooting, because of the manner of confinement. The following table gives a guide to the weight of explosive required for different boulder thicknesses :—

Thickness.	Charge (Grams.)
0.3 m to 0.45 m	110
0.45 m to 0. 6 m	170
0.6 m to 0.75 m	225
0.75 m to 1. 0 m	280-350
1 m to 1.15 m	350-450

Plaster shooting is more popular these days. Its *advantages* are :—

1. Material is broken in-situ and does not fly or scatter over wide area. Machines need not be moved far.
2. No drilling is required. This is important where rocks are hard and abrasive. Compressors and compressed air pipes are not required.
3. Plaster shots are quickly prepared and a large number of boulders can be blasted in a short time.

Its disadvantages are :—

1. It makes more noise.
2. Much explosive is required

Plaster shooting is used for demolition of brickwalls, concrete, cutting of steel plates in ship breaking, usually under water, etc.

REFERENCES

1. MISRA G.S. "Blasting with Liquid Oxygen Explosives" Trans. Min. Geo. & Met. Inst. of India, Vol. 52.
2. INDIAN OXYGEN LIMITED a Brochure (Jun, 1962), on "Liquid Oxygen Explosives."
3. I.C.I. LTD. "Blasting in Quarries.

CHAPTER 13

SPECIAL BLASTING TECHNIQUES AND SUBSTITUTES FOR EXPLOSIVES

1. Pulsed infusion shotfiring. 2. The air breaker system. 3. Airdox shells. 4. Hydraulic coal burster. 5. Cardox. 6. Hydrox. 7. Chemicol system.

1. PULSED INFUSION SHOT-FIRING

This method of blasting was introduced in a few mines like Ranipur, Chinakuri and others of Raniganj coalfield in 1960-61. No definite conclusions were possible as the trials were of a short duration and had to be suspended due to difficulties of foreign exchange.

The blasting may be carried out on an undercut face or on solid face but solid blasting is more common.

The technique, in brief, is as follows :—

1. Holes are drilled in the coal face with usual coal drills.
2. The hole is fitted with a pipe and water seal, and water by a pressure pump is pumped into the hole till the coal is infused.
3. The pressure is released and the hole is charged with Hydrobel, a special water resisting type of explosive.
4. The infusion tube and the seal are inserted into the mouth of the hole.
5. The water under pressure is again infused into the hole and with the water under active pressure the charge is fired.

No earth or clay stemming is used, and the water under pressure acts as the stemming.

TYPES OF EXPLOSIVES AND EQUIPMENT REQUIRED.

1. *Hydrobel explosive* :—This is a special type of water resisting permitted explosive capable of efficient detonation under a high pressure of 70 kg/cm^2. Under test, it has been found

to detonate reliably after immersion in water at 70 kg/cm² for 24 hours.

The maximum charge in a shot hole is 794 g.

2. *"Hydrostar" electric detonators* :—These are of submarine type and operate reliably after prolonged exposure to the high hydrostatic pressures required for this technique of blasting. These are fitted with 2.4 m or 3 m long tinned copper leading wires insulated with a plastic covering. Instantaneous type, as well as short delay type detonators are available.

3. *Cordtex detonating fuse* :—This is required for long hole blasting only. Like the cordtex fuse used commonly in blasting in mechanised quarries, this fuse also has a P.E.T.N. (Penta-erythritol tetranitrate) core, but the construction has been altered to prevent water penetration along the core during the period between charging and firing the shot.

4. *Drilling equipment* :—For long hole blasting special drilling rig is employed.

5. *Suitable pump and plentiful supply of water* :—Normally the pressure for water infusion is 70 kg/cm² and more. In deep mines, water at the desired pressure may be obtained direct from the shaft rising main. For shallow mines mobile tanks may be used with a three throw ram pump to deliver 55-90 litres/min. For flank holes of 1.8 m depth, some coals require 13-18 litres per hole and others may require nearly 90 litres.

6. *Armoured hose pipe* :—12 mm internal diameter pipe to withstand high pressure is used. Spring loaded automatic valves are fitted on the pipe line at intervals of 18 to 22 m. The back pressure during blasting is arrested by such valves and does not reach the water main or the pump which is thus protected.

7. *Infusion tubes* ;—In these the pressure of the water is utilised automatically to expand the rubber seal and so seal the hole.

8. *Manifold* :—Where rounds of shots are to be fired, a simple manifold is required. Each infusion tube is attached to the manifold by about 4.5 to 7 m of pressure hose, and a separate valve is fitted in each branch of the manifold.

9. *A pressure gauge and flow meter* :—These are used at the firing point, and if rounds are being fired, also at the manifold.

10. *Charging rods* :—For flanking holes of 1.8 m to 2.4 m length wooden charging rods of suitable length for pushing home the explosive charge are employed. A "Stemming rod" would be a wrong term; as no stemming is involved.

Method of Blasting

The blasting may be done on an undercut face or on solid face. Blasting on solid face is common.

(A) FIRING IN GALLERIES OR HEADINGS WITHOUT UNDERCUTTING.

Fig. 1 shows holes drilled at a coal face on a wedge cut pattern as in a stone drift. The spacing and number of holes depend upon the hardness of coal and has to be decided in the first few experiments. After the drilling is completed, water

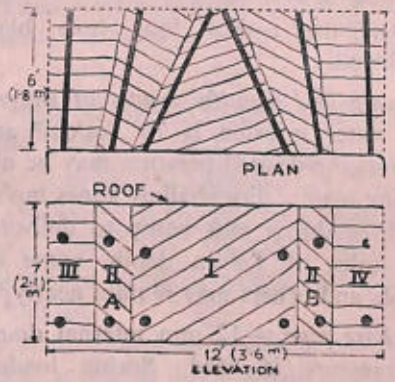

Fig. 1. Position of shot holes for solid blasting by pulsed infusion shot firing.
Blasting Sequence : (1) Portion I (2) Portion IIA (3) Portion IIB (4) Portion III (5) Portion IV.

is to be infused in two or more holes at a time with the help of a manifold. The procedure of water infusion is as follows :—

1. The water infusion operator inserts the infusion tube in two or more holes as deep as possible, plugs the mouth of the holes by water seals and then connects the infusion tubes to the manifold. The manifold is connected to the delivery pipe of a pressure pump (Fig. 2). When manifold is used, the valve in each arm of the manifold is turned on separately, while the

others remain closed. This is to ensure that each hole receives sufficient pressure and water.

2. The pressure pump is operated and water is allowed to infuse the coal till the water is seen to come out at the face. The infusion pressure may vary from 10 kg/cm² to 35 kg/cm². This is known as initial infusion.

3. The pressure is then released and the tube withdrawn.

4. The shot-firer primes a hydrobel cartridge with "Hydrostar" electric detonator. Hydrobel cartridges are inserted into the hole, the primed cartridge last of all. No stemming by sand or clay is necessary.

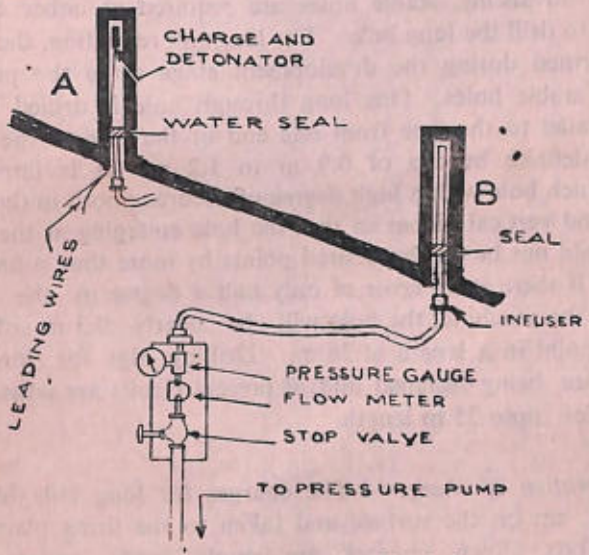

Fig. 2. Water infusion in heading holes.
A—Hole charged after water infusion.
B—Water being infused under pressure.

5. The infusion tube is then re-inserted into the hole, so that the sealing gland is approximately 450 mm from the collar. The water is turned on, and a second infusion given.

6. The shot-firer fires the shot, while the water infusion pressure is still maintained.

Upto six shots can be fired simultaneously.

In all these operations the water infusion operator forms a member of the shot-firing team. The time required for blasting

a face is slightly more than in the ordinary method of blasting and the shot-firer can fire 6 to 8 shots per hour. The rigid inspections for gas at the face are not obligatory and the time required for gas examination. stone dusting or water spraying prior to blasting is eliminated.

(B) Firing in Long Holes

The long hole method of pulsed infusion shot-firing presents a complete departure from the conventional methods of shot-firing. Fig. 3 shows the general arrangement. For longwall advancing, stable holes are required at either end of the face to drill the long hole. For longwall retreating, the roadways formed during the development stage serve the purpose of such stable holes. One long through hole is drilled in the coal parallel to the face from one end of the face to the other with a definite burden of 0.9 m to 1.2 m. It is important to drill such hole with a high degree of accuracy both in the horizontal and vertical planes so that the hole emerging at the other end should not be off the desired points by more than a few centimetres.If there is an error of only half a degree in the alignment of the machine, the hole will be nearly 0.3 m off the desired point in a length of 38 m. Drilling rigs for long-hole drilling are being designed and at present results are satisfactory for holes upto 35 m length.

Preparation of charge :—The charges for long hole blasting are mad up on the surface and taken to the firing place in a special box. Such charges are rarely made underground. The cordtex detonating fuse is cut to a length equal to the total hole length plus a minimum of 0.6 m extra at either end and the ends are sealed by crimping on empty detonator tubes. An axial hole is pierced through each cartridge of hydrobel and the cordtex is threaded through the cartridges, so that the cartridges are spaced at suitable intervals of 1.5 m to 3m, either singly or in groups of two or more cartridges. Alternatively the hydrobel charges may be taped to the cordtex at definite intervals. The cordtex is threaded through the primer cartridge also but the detonator is not put in position till before the charging of the hole.

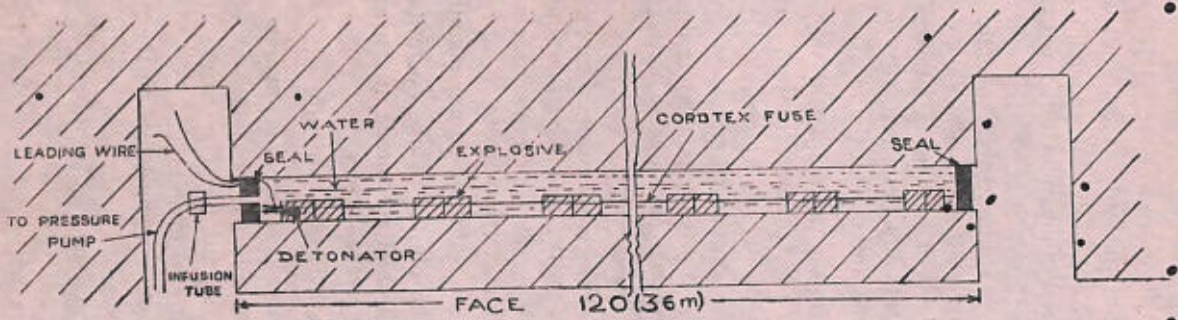

Fig. 3. Longhole blasting by pulsed infusion shot firing.

An example of L/h Blasting in a British Mine:-

Retreating face in a seam of 4½' thickness with 1' dirt band at the top.

Hole dia → 3⅞" (98.4 mm) & length 114' (34.2 mt)

Charge was 12 oz. hydrobel every 2'-6" apart on Low Creep cordtex.

Infusion pressure was 150 p.s.i. & about 600 gallons of water was used for each infusion.

The **procedure of water infusion and blasting** is as follows :—

After the hole is drilled, a strong cord is attached to the drill rods and drawn into the hole as the rods are withdrawn, leaving the cord in the hole. The remote end of the hole is closed with a screw type sealing plug fitted with an air release cock for the purpose of releasing air from the hole. At the drilling end of the hole, a standard hydraulically sealed infusion tube is inserted. Seals at both the ends are 0.4 m to 0.6 m from collars of the hole. The coal is then infused by water under pressure in the manner already described, till water is seen to come out at the other end and also at the face.

When the initial infusion is completed, the pressure is released and the tube and the seal are removed. The cord in the hole is now used to draw the prepared charge into its desired position. In the primer cartridge, a special "Hydrostar" elec. detonator is inserted, and the leading wires taken outside the hole. The primer cartridge is 0.9 m to 1.2 m from the mouth of the hole.

The infusion tube and the seals are then replaced, so that the sealing gland is 0.6 m from the mouth of the hole. The coal is again infused. When the water pressure reaches the desired level, the leading wires are coupled up to the firing cable. The shot is then fired, while the water pressure is maintained.

The whole face can be blasted in one operation. Long hole blasting is likely to be of chief advantage in thin seams in which coal is hauled outbye by scraper boxes.

ADVANTAGES OF PULSED INFUSION METHOD

1. It affords improved safeguards against gas ignition. Cracks leading from a hole to gas pockets have often been contributory causes of gas ignition. The cracks are water filled and the gas flushed out in this method.
2. The water has rapid cooling effect on the products of the explosion.
3. It maintains a high degree of dust and fume suppression as all the dust is wetted.
4. No additional stemming is required. The shot-firer is relieved of the transporting and preparation of sand-clay stemming.

5. The explosive energy is more effectively utilised, because the high pressure impulse, produced by the shot, is transmitted throughout the water distributed in the cleavage planes. The shock is thus spread over a wide area away from the cartridges, instead of being concentrated round the shot hole.

6. Production of round coal is more compared to that in conventional "Dry" blasting.

7. It is easy to deal with a misfired shot by removing the infusion tube and inserting a primer cartridge in the hole.

8. Solid blasting is permitted. The coal yield in solid blasting in the mines of Ranigunj field was from $3\frac{1}{2}$ to $4\frac{1}{2}$ tons per kg of explosive. The hazards due to ignition of gas by sparks of cutter picks during coal cutting are, therefore, eliminated.

LIMITATIONS

The method cannot be applied to the faces that have cracks and fractures since adequate infusion of coal cannot be carried out as the water comes out from the cracks and fractures. If under-cutting induces such cracks or fractures, it is preferable not to under-cut the face. It cannot be used on a longwall face more than 35 m long due to limitations on drilling accurately with the present day rigs. It is open to misuse in absence of water.

2. THE AIR BREAKER SYSTEM

The air breaker system of blasting coal employs compressed air at very high pressure, nearly 600 to 800 kg/cm^2. It has been developed in the U.S.A. in early 1930's and has been introduced in India at Girmint Colliery (Raniganj coalfield) in 1965. At present it is the only colliery using this method of blasting coal. The equipment and other accessories required and the method of blasting are as follows :

(1). *An air compressor* capable of compressing upto 800 kg/cm^2, installed at the surface or underground where an adequate volume of dust free air is available for cooling purposes.

At Girmint the compressor is with horizontally opposed cylinders, 6-stage, piston type, compressing air to 800 kg/cm^2. It is installed underground ; motor is 90 H.P., 1,470 r.p.m. ; compressor r.p.m. 415 and capacity 2.5 m^3/min of free air.

(2). *Receiver*—This consists of a bank of five steel cylinders each 130 mm outside diam. and 1.4 m long tested to 3,000 kg/cm.^{2} It may be dispensed with where the delivery range is long, exceeding 1 000 m e.g. at Girmint.

(3). *Air transmission pipes*—The main line is of 1% chrome molybdenum steel 12 mm bore, 25 mm outside diam. tested to withstand 1 200 kg/cm^2. It is in 6 m long sections joined by screwed couplings. Branch lines in the district are of copper, 4.5 mm bore and 9 mm outer diam., supplied in 16 m lengths.

At the face, high pressure double wire braided hose pipes 3 mm bore, are used to carry air from the copper pipe to the air breaker shell.

(4). *Valves*—(i) Blow-down valve. This is a main valve fitted on the main steel pipe range and controls 2 or 3 branches.

(ii) Shooting valve—This is placed between the copper tubing and the hose pipe near the face. It has a removable key whereby the shotfirer may charge the shell with air from the pipe line or shut it off completely. This key should always be with the shotfirer.

(5). *The air breaker shell*—It is a steel tube with flexible hose connected to one end and the discharge head at the other. Its components are shown in fig. 4. The shells vary in length from 1 m to 4 m, in outer diam. from 31 mm to 63 mm, in capacities from 3,600 c.c. to 4,250 c.c. and in weight from 19 kg to 21.5 kg. The size chosen depends on the seam thickness, manoeuvring space, and blast effects required. The tube has an inlet port at one end and a spring loaded piston valve at the other. Before placing the shell into the blast-hole, a steel shear plate is inserted into the shell. The steel shear plates are designated by numbers as No. 0, No. 1, No. 2 etc. No. 0 is the thickest and the strongest. The shells used at Girmint are of 63 mm O.D., and of 2.3 m length when assembled. The shear plates normally used in coal are of No. 1 and No. 2 sizes. When air from the shooting valve is admitted to the air breaker shell, the shear plate ruptures at a pressure varying between 140 kg/cm^2 and 600 kg/cm^2, depending upon the thickness of the shear plate selected. Average operating pressure

of 390 kg/cm^2 to 425 kg/cm^2 is observed to be adequate for koithi seam coal at Girmint colliery.

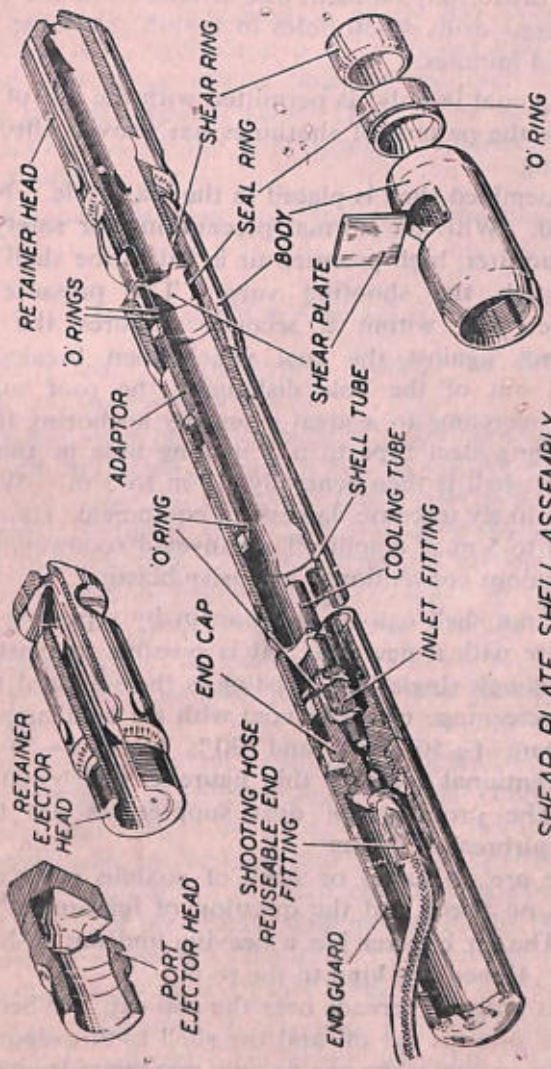

Fig. 4. Armstrong Air breaker.

(6). *Coal drills*—The shell needs a large hole of 70 mm diam. The normal coal drill of 1.25 H.P. is adequate, but the drill rod has to be of turbine section and fitted with a pilot bit (which is

the usual T.C. tipped drill bit for 42 mm diam. holes) and a reamer bit for 70 mm diam. hole. The drill gets hot readily and two drills may be used, one drilling while the other cooling down. A crew drills 40-50 holes in a shift ; average time to drill a hole is 4 minutes.

Blast ng coal in solid is permitted with the use of air breaker. Fan cut as the pattern of shotholes has proved effective in solid coal blasting.

The assembled shell is placed in the bla t hole. No stemming is required. With the normal precautions for safety of persons and the shotfirer, high pressure air is fed to the shell by the shotfirer through the shooting valve. The pressure builds up inside the shell within a second, ruptures the shear plate and presses against the coal which then breaks. The shell is ejected out of the hole dislodging the roof supports. This has been overcome to a great extent by anchoring the shell with a 1.5 m long steel rope to a 1 m long hole in the floor. The ejection of shell is then generally 1.5 m to 3 m. Where ejection of shell is likely to cause damage to equipment, e.g. when driving the first 3 to 5 m of a split off a conveyor roadway, it is advantageous to adopt conventional explosive blasting.

The same shell can be used again by replacing the ruptured shear plate with a new one. It is possible to blast 2-3 holes at a time, though single hole blasting is the standard practice.

The percentage of steam coal with air breaker is high, nearly 70% steam (+50 mm) and 30% slack (−50 mm). With the conventional blasting this figure is nearly 60 and 40 restively. The problems of dust suppression are therefore less with the airbreaker system.

There are no fumes or smell of noxious gases after blasting. There is no flame and the question of ignition of gas does not arise. The air breaker has a heaving and not a shattering effect on coal. Hence it is kind to the roof.

Shells bulge at threads near the end cap. When this happens a 75 mm piece is cut off and the shell re-threaded. In this way the shells gradually shorten to only one metre length.

Cost

The initial cost of installation is nearly Rs. 5 lakhs. The spares to last for two years cost Rs. 2 lakhs. For such heavy

investment, the production of 1,000 tons per day from the air breaker district is the minimum to render it economical. With solid blasting, the coal yield per hole of 1.5 m depth is nearly 2 tons. Under favourable conditions one compressor enables 50-70 holes per hour to be blasted i.e. nearly 400 holes per shift in 6 hr. effective time, equivalent to 800 tons per shift. At Girmint, the average tonnage of coal obtained per hour of compressor working was 108 tonnes (Dec. 1966). The cost of blasting is less than that of the conventional explosives by 25 to 40 paisa per tonne.

Air breaker has not been tried for stone blasting so far, in our coalfields.

3. AIRDOX SHELLS

These shells are of three types, namely, the piston shell, receiver shell and the shear strip shell.

In the piston type of shell, there is a leather-cup piston at the outbye end connected through a rod to a valve at the other end, which can close the discharge ports. As compressed air enters the shell, it forces the piston back thereby closing the valve at the discharge end. Air then bypasses the leather-cup of the piston until pressure inside the shell is equal to supply pressure. Then the main supply valve is closed and the outbye end is open to atmosphere. This causes the piston to move out due to the high air pressure at its back inside the shell which opens the discharge ports.

The receiver shell (Fig. 5) is similar in operation to that of a piston type shell. Here compressed air is charged into a 50.8 cm × 15.2 cm dia. receiver cylinder, which remains outside the shot-hole, rather than into the shell tube itself which is placed in the shot-hole. A discharge tube is connected to this cylinder to carry the ejected compressed air to the back of the shot-hole. This enables the tube to be made of smaller diameter requiring smaller dia. shot-holes to be drilled.

In the shear type of shell, there is a spring loaded sliding sleeve at the inbye end which covers the discharge ports. At the outbye end of this sliding sleeve, there is a 9.5 mm orifice closed by a thin shear strip of steel held in position by a block.

having a corresponding 9.8 mm dia. passage. When the shell is charged with the compressed air, the shear strip bursts as soon as the required pressure is reached within the shell. The pressure of air then forces the spring loaded sleeve back, thereby uncovering the discharge ports. This shell is considered more powerful due to its more rapid discharge.

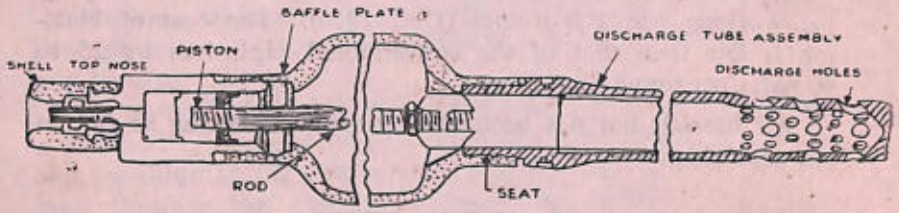

Fig. 5. Airdox shell.

Advantages of the air-breaker system are :

1. Dangers inherent in shot-firing, such as ignition of gas, injury by flying debris, dust, fumes, etc. are eliminated.
2. Large coal is produced.
3. Cost is less than in case of cardox or hydrox.
4. Firing cycle does not take long time.
5. Loading can be started immediately after blasting.

Disadvantages

1. Hazards due to burst of pipes, hoses etc. under high pressure do exist.
2. The system is not as flexible as cardox or hydrox, as it is practicable only within the limits of pipe line.
3. If the air compressor is situated underground there may be a risk of fire.
4. Initial installation cost is heavy, though the operating cost is low.
5. There is difficulty in handling the shells and the hose on steep gradients and in thin seams.
6. Charging of holes in advance of coal cutting is not practicable, though this is normally practiced with cardox and hydrox, because a single shell is used again and again and these shells project outside the hole, causing obstruction in the outer track ; nor can the boreholes be drilled in advance of coal cutting, as the movement

of coal after cutting frequently causes sufficient distortion of the shot-hole for re-use.

4. THE HYDRAULIC BURSTER

This requires holes of larger diameter than those drilled by ordinary coal drills. It consists of a stainless steel barrel 50 to 90 mm in diameter, having holes at intervals to receive closely fitting pistons or plungers. The barrel is connected by a high pressure armoured hose pipe to a high pressure pump (nearly 945 kg/cm^2). The pump may be manually operated or operated by an electric motor. The plungers are forced out at right angles to the cylinder by water pressure. In order to distribute the pressure exerted by the plungers uniformly, a steel liner 50 mm wide and slightly longer than the cylinder is placed along the plungers, when the burster is inserted into the hole. (Fig. 6).

To use the burster the face should be undercut, and the shot holes have to be of large dia. (70 mm to 100 mm). Clean water, mixed with a small quantity of oil, should be used.

The method is likely to give good results in soft coal.

The *advantages* are : (i) no danger of ignition of gas (ii) no flying coal particles (iii) no fumes (iv) no shattering effect on roof (v) no coal dust formation and (vi) silent operation. Workers have not to be removed from the face, and large coal is produced.

The method is slow where a large number of holes is drilled. The coal blocks produced are too large to handle in some cases.

The hydraulic burster is not at present used in India.

5. CARDOX

This is a short name for the Carbon-di-oxide cartridge. The principle behind the use of such cartridge is that entrapped liquid CO_2 when heated, expands to 500 to 600 times its volume in a fraction of a second. (Fig. 7).

The Cardox shell consists of three parts, made of tensile steel : (1) the tube which holds the CO_2 charge, and (2) & (3) two heads which seal the tube at each end. These heads are known as the firing and discharge heads respectively, the function

of the discharge head being to provide an outlet for the gas contained in the tube, while that of the firing head is to provide a means of connecting the heater to an outside source of low tension firing current. The firing head is also furnished

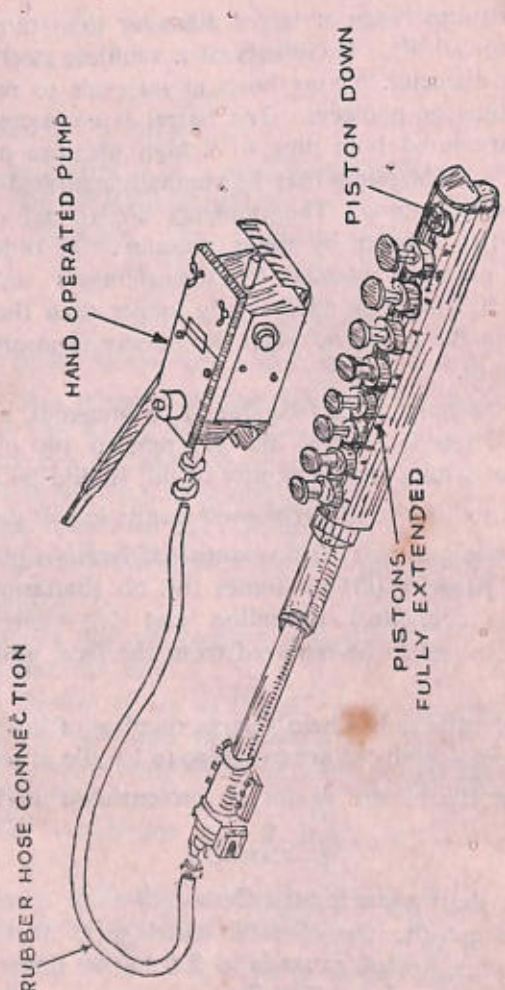

Fig. 6. Hydraulic burster.

with a valve for filling the tube with CO_2. At the discharge end of the tube a soft steel disc acts as a temporary closure, and also regulates the pressure of the gas discharge. The standard discharge head is furnished with four venting holes and

two movable pawls actuated by the gas release. The pawls anchor the tube in the hole.

The heater rests on a shoulder in the tube and makes contact with the firing head when the latter is screwed into the tube. The heater consists of oil paper container, filled with potassium perchloride, ammon. oxalate, salicylic acid and castor oil. With this heater an electric powder fuse is incorporated, which ignites the heater mixture under pressure. An ordinary L.T. shot-firing exploder and cable are used for firing the powder fuse ; it ignites the heater cartridge which generates enough heat by its reaction to convert the liquid CO_2 into gas at a pressure high enough to shear the disc in the discharge end.

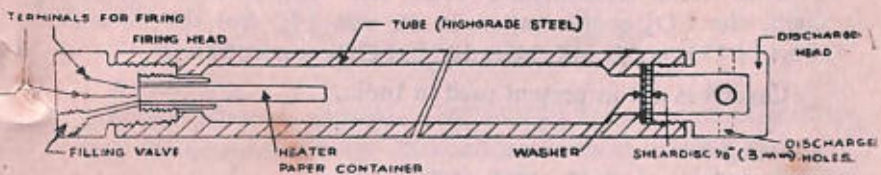

Fig. 7. Cardox Shell.

After firing, the empty shells are recovered and returned to the charging plant, located on the surface. The shell is dismantled and the consumable parts, viz. disc, heater and washers, are renewed. Now-a-days the shell is filled with liquid CO_2 by a bottling process. The original charging plant was similar to the conventional refrigerating plant and consisted of a number of separately filled units, but the present day machine is a small compact outfit capable of filling upto eighty shells per hour.

The dimensions and capacities are as follows for various types of Cardox shells.

Chrome-molybdenum steel shell	0.45 m to 1.67 m long 38 mm to 63 mm diam. (external).
Charge of liquid CO_2	113 g. to 2.6 kg.
Shearing disc of soft steel	3.2 mm to 6.4 mm.

Corresponding rupturing pressure of shearing disc.	1,240 Kg/cm² to 2,748 kg/cm².
Weight of heater	27 grams to 240 grams.
Electric powder fuse.	5 grain low tension type.

The usual precautions necessary for blasting by explosives are observed when using Cardox also. The stemming has to be as usual, but it is a safe practice to erect a prop in front of the shot-hole as a safeguard against violent ejection of the cartridge. In case of misfire, the shot-firer can remove the shell from the hole, after disconnecting it from the cable, and send it to surface for examination. If the shell cannot be removed from the hole, a relieving shot must be fired.

Tests have shown that, even after firing large capacity shells, the CO_2 concentration in air was 1%, but the narcotic effect of CO_2 is felt only after 10% of CO_2 concentration.

Cardox is not at present used in India.

Advantages

1. No danger of ignition of fire damp ;
2. Percentage of large coal produced is high ;
3. Holes may be charged, even before coal cutting is done. This practice is prohibited for ordinary explosives.
4. Reduced disturbing effect on the roof.

Disadvantages

1. High cost of blasting as compared to explosives ;
2. Transport of shells-both before use and after use—is not as simple as with ordinary explosives.
3. In hard coal its results are not very satisfactory.

6. HYDROX

The latest and improved hydrox cartridge resembles the Cardox cartridge and consists of a steel tube (generally 1.2 m long and 44 mm outside dia.) fitted with a firing head and a discharge head (Fig. 8). The disruptive medium, however, consists of granulated powder, having as its main constituents, Ammon. nitrate 50%, Magnesium nitrate hexahydrate 34%, and wood-meal 16%. The hydrox shell, therefore, can be

charged more conveniently than the liquid CO_2 which needs a costly CO_2 plant. The charge used in improved hydrox shells is a composite prepacked charge, which is incapable of igniting gas, even if exposed and fired in the open, or if the cartridge is wrongly assembled e.g. with the shear disc omitted). The complete charge consists of (a) an inherently safe initiator, enclosed in a copper tube (b) a safe primer charge, consisting chiefly of Guanidine nitrate 60%, Ammonium persulphate

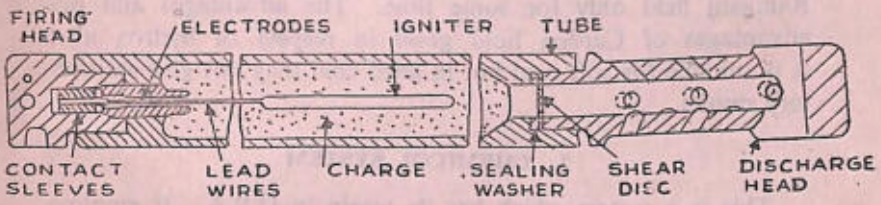

Fig. 8. Hydrox Shell.

36% and Cuprous chloride 4%, and (c) a main charge consisting of Ammon nitrate, Magnesium nitrate hexahydrate, and woodmeal, as mentioned earlier. The main charge is incapable of sustained reaction under atmospheric pressure, but is capable of rapid reaction at slightly higher pressure than the atmospheric. All three components of the charge are packed in paper 30 mm dia. and 0.5 m long, with the leading wires extending from the initiator, for coupling to the firing head. Loading of the shell is done from the firing end, and the discharge end is semi-permanent. Instead of the steel shearing disc used in Cardox, the hydrox shell has a renewable plastic truncated cone plug which forms an integral part of the composite cartridge, and fits in the end face of the discharge head and is self sealing.

When the charge is fired, the initial pressure developed is of the order of 1,900 kg/cm². The discharge head has several rows of bent back holes to aid the venting velocity of the gases to give an increased area of pressure application and to function as a retaining device. The cartridge explodes 2 or 3 seconds after operating the exploder. The products of the combustion are non-inflammable, non-toxic and efficient suppressors of flame.

Corresponding rupturing pressure of shearing disc.	1,240 Kg/cm² to 2,748 kg/cm².
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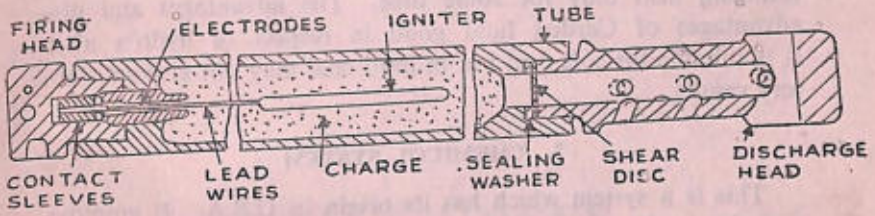

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The hydrox cartridges may be re-charged at or near the face. This reduces the transport problem and the number of shells required to be kept in use at a time. Because of the delay in explosion of the charge after operating the exploder, hydrox is unsuitable for simultaneous firing. After a shot is fired, there is practically no residue in the tube, and the firing head can easily be removed by hand for reloading.

Hydrox cartridges have been used in India at a deep pit in Raniganj field only for some time. The advantages and disadvantages of Cardox hold good in respect of hydrox also. It is suitable for soft coal but in hard coal may not give satisfactory results.

7. CHEMICOL SYSTEM

This is a system which has its origin in U.S.A. It employs a steel tube 58 mm dia. and 1.52 m long closed at one end by an electric plug and at the other by a rupture disc held in its place by a discharge head. Inside the tube there is a gas generating unit consisting of a starter mixture, pressure mixture, and a coolant. The reaction is started electrically by means of a nichrome ribbon embedded in the starter mixture and heated for 7 seconds by a current of 6 to 9 amperes supplied by a 36 volt accumulator. When enough gas is generated to build up sufficiently high pressure, the disc is ruptured. This device cannot be fired accidentally from any other source as the rush of current would fuse the heater wire without starting the reaction. Automatic devices are included in the shell for limiting the flow of current, for cutting off the supply when the reaction is started and before the blast takes place, and to throw off the firing head, as the shell discharges, to avoid damage by falling coal.

These tubes are charged underground. The charging operation is very simple. This system has the same advantages and disadvantages as the new Hydrox. Owing to the time lag, this shell also cannot be used for simultaneous firing. Another disadvantage with this system is that the battery is both heavy and bulky and in the present form is not intrinsically safe.

APPENDICES

APPENDIX I

GEOLOGICAL TIME SCALE

Era	*Period*	*Age in million years*	*Life development*
QUATERNARY	Recent	0.1	Living animals.
	Pleistocene	1	Advent of man.
	Pliocene	12	Flowering plants dominate vegetation.
TERTIARY OR KEINOZOIC	Miocene	25	Primitive horses and other ungulates.
	Oligocene	40	Apes appear.
	Eocene	60	First Placental animals.
	Cretaceous	110	Giant reptiles and ammonites, but become extinct at close of period.
SECONDARY OR MESOZOIC	Jurassic	150	First flowering plants ; first birds.
	Triassic	180	Ammonites, amphibia and reptiles in abundance. First dinosaurs and primitive mammals.
	Permian	215	Rise of reptiles. Trilobites disappear.
	Carboniferous	275	Large non-flowering plants ; first reptiles.
PRIMARY OR PALAEOZOIC	Devonian	325	Age of fishes. First amphibia, abundant corals and brachiopodes.
	Silurian	360	First land plants. Graptolites disappear at end.
	Ordovician	450	Abundant trilobites and graptolites.
	Cambrian	600	Abundance of trilobites.
PRE-CAMBRIAN	Precambrian	1500	Soft bodied animals and plants.
ARCHAEANS	Archaeans	4000	Lifeless.

Appendix II

CURRENT SELLING PRICES OF COAL (MAY 1967)

In pursuance of clause 4 of the Colliery Control Order, 1945, as continued in force by section 16 of the Essential Commodities Act, 1955, Central Government fixes from time to time the prices at which coal or coke may be sold by colliery owners free on rail at the colliery or loading point nearest the colliery.

S.O. 1125 dated the 1st April, 1967 fixes the prices in Rs. per tonne as under :

For Collieries situated in the States of West-Bengal and Bihar

Non-coking coal		*Coking coal*	
Grade	Steam, rubble & Nuts	*Grade*	Steam, rubble & nuts
Sel. Grade A	31.39	A	33.22*
Sel. Grade B	29.92	B	32.24*
Grade I	27.56	C	31.26*
Grade II	Not exceeding 25.89	D	29.78*
Grade IIIA	24.34	E	29.29*
IIIB	23.19	F	28.55**
		G	28.06**
		H	27.83**
		HH	Not exceeding 25.89
		Soft coke of < 45% ash	36.29

*plus Rs. 1.25 per tonne if owner agrees to accept the grade determined by sampling and analysis at destination.

**plus Rs. 0.75 if graded at destination.

For outlying fields in :

Gujrat, Madhya Pradesh, Maharashtra and Orissa.		Assam coals		Andhra Pradesh		
Grade	Steam, rubble & nuts	Collieries	R.O.M, dust	Type of coal	Yellendu	Other collieries
Sel. Grade	31.49	Margherita	37.63	Round coal and nuts (25-50 mm)	32.66	33.64
Grade I	29.26	Dilli, Jeypore & Bimalapore	28.09			
Grade II	28.17			Nuts (12-25 mm)	31.18	32.17
Grade III	27.19	Koilajan and Sheelveta	31.98			
		Laitryngew	19.75			
		Thangjinath	16.80			

For collieries in the states of W. Bengal, Bihar and other outlying fields except Assam the prices of r.o.m., dust and slack coal is approximately Re. 1 to 1.06 less than the price of steam coal.

Appendix III

This appendix was intended for classification of Indian coals on the lines of the international classification. As the classification by Indian Standard Specifications is not ready at the time of printing this portion, it is not included here.

APPENDIX IV

CURRENT LEGISLATION ON MINES AND MINERALS

Legislation in India is broadly divided into two groups ; one for development and conservation of minerals and the other for safety and welfare of persons engaged in mining. For the purpose of development and conservation, minerals are classified as

1. *Atomic minerals*—that is, those minerals which are used for development and control of atomic energy, such as graphite, minerals of uranium, thorium, radium, beryllium and lithium, deuterium, plutonium and neptunium, minerals which contain the above metal as accessories, such as, uraniferous allanite, triplite, columbite, tantalite, pitch-blende, samarkite, monazite and uranium-bearing tailings left-over from ores after extraction of copper, gold, ilmenite, zircon, rutile or beryl.

2. *Mineral oils*—Petroleum and natural gas

3. *Minor minerals*—minerals of local importance which can be developed on small or cottage scale for benefit of local people, such as, building stone, marble, boulder, shingle, gravel, chalacdony pebble, limeshell, kankar, limestone for lime burning, murrum, brick earth, fuller's earth, bentonite, ordinary clay, ordinary sand used for non-industrial purposes, road metal, rehmatti, shale and slate

4. *Major minerals*—other than those above.

Current legislation comprises the following enactments :

(1) Mines and Minerals (Regulation and Development) Act, 1957. Act came into force on 1st June 1958 and extends to whole of India including the State of Jammu and Kashmir. It does not apply to mineral oils and minor minerals. Under this Act the following enactments have been framed :

(a) The Minerals Concession Rules, 1960, framed under Section 13 of the Act, which came into force from 11 November, 1960.

(b) The Mining Leases (Modification of Terms) Rules, 1956, framed under Section 7 of the Act, which came into force on 15th September, 1956.

(c) The Minerals Conservation and Development Rules, 1958, framed under Section 18, which came into force on the 1st of June, 1958.

(2) The Oil Field (Regulation and Development) Act, 1948—This Act came in force from 25 October, 1949 and regulates development of mineral oil in the whole of India except the State of Jammu and Kashmir. Under Section 5 of the Act are framed the Petroleum Concession Rules, 1949 which came into force on 5th January, 1950 and maintain intact the provisions of Petroleum Act, 1934.

(3) The Atomic Energy Act, 1962, for development and control of atomic energy and for purposes concerned therewith. It came into force on 15th September, 1962 and extends to whole of India.

(4) The Mines Act, 1952, for safety and welfare of persons engaged in mining. It came into force on 1st July, 1952 and extends to whole of India except the State of Jammu and Kashmir. Under the Act, the following enactments have been framed.

(a) The Mines Rules, 1955, in force from 2.7.67.
(b) The Mines Creche Rules, 1963.
(c) The Coal Mines Rescue Rules, 1959.
(d) Mines Vocational Training Rules, 1966.
(e) The Mysore Gold Mines Rules, 1952, in force from 17th September, 1953.
(f) The Coal Mines Regulations, 1957.
(g) The Metalliferous Mines Regulations, 1961.
(h) The Mysore Gold Mines Regulations, 1953.
(i) Oil Mines Regulations, 1963—not in force yet, *i.e.* in May, 1967.

(5) The Coal Mines (Conservation and Safety) Act, 1952.

It came into force on 4th March, 1952 and under its Section 17, the Coal Mines (Conservation and Safety) Rules, 1954, have been framed which came into force on 2nd October, 1954.

APPENDIX V

DIGEST OF COAL MINING STATUTES

Abbreviations used in these tables are :

DGMS	—Director-General of Mines Safety
JDMS	—Joint Director of Mines Safety
DM	—District Magistrate
CMWC	—Coal Mines Welfare Commissioner
CMPC	—Coal Mines Provident Fund Commissioner
MA	—Mines Act, 1952
MR	—Mines Rules, 1955
CMR	—Coal Mines Regulations, 1957
MMR	—Metalliferous Mines Regulation, 1961

A. Notices & Returns to be submitted to various authorities

Reference		*Particulars of notice/ return*	*To whom sent*	*Period of notice*
1		2	3	4
Under Mines Act/Rules				
MA	16	Notice of commencing mining operations.	(a) DGMS, (b) Director IBM (c) DM	One month bef.
MA	23	Notice of Accidents	Special notice board.	Keep for 14 days.
MA	23	Returns of minor injuries.	DGMS	20th January.
MR	47	Notice of weekly days of rest.	Notice Board in Form A.	
MA	36	Notice—hours of work.	Notice board, First page of registers in Form B, C, D & E.	
MR	48			
MA	36	Notice of change in hours of work.	Notice Board DGMS	07 days bef.
MR	49	Notice of compensatory days of rest.	Notice Board	1st of every month.

1	2	3	4
MR 72	Notice of appointment & dismissal of Welfare Officers.	DGMS	Within 7 days.
MR 73	Annual report of Welfare Officer through Manager.	DGMS	End of year.
MR 79	Abstract of byelaws & notices.	Notice board.	
Maternity Rules			
Rule 16	Annual returns in Form L, M, N, O.	CMWC	21st January.
16	Notice of abandonment or sale of mine.	..	Bef. year end.
Land Acquisition (Mines) Act, 1885. 1	Notice of working minerals under any land.	Appropriate Government.	60 days bef.
C. M. Regulations, 1957 & *M. M. Regulations*, 1961.			
CMR & MMR 3	Notice of actual date of opening, in Form I	DGMS JDMS	**Forthwith.**
CMR & MMR 4	Monthly returns, in Form II.	DGMS JDMS	14th of each month.
CMR & MMR 5	Annual returns in Form III.	DM DGMS	20th February.
	Annual returns in case of change of ownership/abandonment.	DM DGMS	Within 30 days.
	Annual return for abandonment.	DM DGMS	Within 60 days.
CMR & MMR 6 (and also under Coal Mines Conservation & Safety Rule, 1940).	Notice of intention to abandon/discontinue.	DGMS JDMS Coal Board.	40 days bef.
	Notice of intention to discontinue/abandon the mine when public property involved.	DGMS JDMS	30 days bef.
	Notice of abandonment/discontinuance.	DGMS JDMS DM	Within 7 days.

1	2	3	4
CMR & MMR 7	Notice of intention to reopen.	DM DGMS JDMS	30 days bef.
	Notice of date of re-opening	DGMS	**Forthwith.**
CMR & MMR 8	Notice of change of address/Ownership	DGMS JDMS	Within 7 days.
	Transfer of records	DGMS, JDMS	**Forthwith**
	Notice of appointment/termination of agent, manager, engineer, surveyor, under-manager, Assistant Manager.	DGMS JDMS	Within 7 days.
CMR & MMR 9	Notice of serious accident	JDMS	Telephone/ Express Telegram.
	do Form IV A	JDMS, DGMS, DM CMWC	Within 24 hours.
	Accident resulting in serious body injury/ death.	Notice Board	Keep 14 days.
	Electrical accident	JDMS (Electrical)	Telephone/ Express Telegram/messenger.
	Death from serious injury already reported.	DM, DGMS JDMS, CMWC	Within 24 hours.
	Particulars in Form IV B.	DGMS	Within 7 days.
CMR 10	Notice of disease. Form V	DGMS, JDMS DM, CMWC	Within 3 days.
CMR 31 MMR 34	Authorisation to act as Manager	DGMS, JDMS	**Forthwith.**
	Resignation of Manager.	do	do.
CMR 33 MMR 36	Appointment of Engineer	JDMS	Within 7 days.
	Officiating Engineer	JDMS	**Forthwith.**
CMR 35 MMR 38	Appointment of Surveyor	JDMS	Within 7 days.
CMR 60	Plans under CMR 59(1) (b & c)	DGMS	31st October.
MMR 62	Plans under MMR 61	DGMS.	30th April.

1	2	3	4
CMR 61	Plans of abandonment	DGMS	Within 30 days.
MMR 63	Plans of discontinuance	DGMS	Within 30 days. (MMR— 5 months).
	Plans in case of change of ownership while discontinuing or abandoning.	DGMS	**Forthwith.**
CMR 72 MMR 79	Notice for use of new winding installation.	JDMS	60 days bef ; 30 days (MMR).
CMR 100(6)	Returns of material stowed & coal raised.	JDMS	10th of every month.
CMR 101 MMR 107	Gallery heightening ; crosscut drivage for haulage, Ventilation.	JDMS	14 days bef.
CMR 102	Crush of pillar	JDMS	**Forthwith.**
CMR 125 MMR 126	Notice of recovery of fire area.	DGMS, JDMS	14 days bef.
CMR 127 MMR 128	Unusual heavy seepage of water.	DGMS, JDMS	**Forthwith.**
CMR 128 MMR 129	Notice of intentional flooding of mine workings.	DGMS, JDMS & Owners of adjacent mines.	14 days bef.
CMR 129 MMR 130	Notice of construction of u/g water dams.	JDMS	14 days bef.
CMR 132	Notice—Installation of u/g fan.	JDMS	7 days bef.
CMR 145	Notice—when gas exceeds 1¼% in mines using electricity.	JDMS	**Forthwith.**
Coal Mines Labour Welfare Fund Rules			
Rule 31	Returns—(a) Statement of coal & coke despatched in 6 months.	CMWC	July & January.
	(b) Expenditure incurred in dispensary in the year.	CMWC	January.
Rule 32	Return in Form D, duplicate on monthly despatch of coke & coal.	CMWC	Bef. last day of each month.

1	2	3	4
Coal Mines Provident Fund Scheme Rules			
Rule 38	Return—particulars of employees qualified for membership of the fund, in Form H & A.	CMPFC	Within 15th of every month.
Rule 42	Return of contribution cards.		
Rule 44	Return of contribution cards of absentee members.	CMPFC	Within a month.
	Return of contribution cards of members ceasing to be employed ; in Form W.	CMPFC	Within 15th of each month.
Coal Mines Bonus Scheme Rules			
Rule 10	Return of Bonus to be paid, Form I & II.	RLC, CMPFC	Within a month of the last day by which any bonus is required to be paid.
Rule 11B	Notice—Change of ownership/closure of mine, Form IV.	RLC	Within one month.
Electricity Rules, 1960			
ER 111	Return of size & type of electrical apparatus in Annexure X.	JDMS (Electrical)	1st February of each year.
	Notice—use of new installation.	JDMS (Electrical)	7 days bef.
ER 126(2)	Notice—when gas exceeds 1¼ % annexure XII.	JDMS (Electrical)	Forthwith.

B. Occasions which require prior approval of Office of the Director-General of Mines Safety.

Reference	*Occasions*	*Authority of approval*
CMR 31 MMR 34	Appointment of manager with lower qualification than that of manager of previous 12 months.	DGMS

	One manager for two or more mines	DGMS
	Grant of permit to manage a small mine.	DGMS
	Acting manager to continue for more than 30 days.	DGMS
CMR 33 MMR 36	Appointment of unqualified engineer for a mine raising more than 5,000 tons.	DGMS
	One engineer for two mines	JDMS
	Authorisation to act for an engineer for more than 30 days.	JDMS
CMR 35 MMR 38	Permission to survey two or more mines or work in any other capacity in the same mine.	JDMS
CMR 58 MMR 60	Removal of statutory plans from the mine office.	JDMS
CMR 80 MMR 88	Permission to use a winding rope for more than 3½ years.	JDMS
CMR 88 MMR 96	Permission to use a haulage for man riding.	DGMS
CMR 92 MMR 100	Installation of a belt conveyor underground.	DGMS
CMR 95 MMR 103	Use of locos underground	DGMS
CMR 99 MMR 107	Galleries to be of dimensions exceeding the limits prescribed.	JDMS
CMR 100	Commencing of depillaring operations ; working contiguous seams	DGMS (also Coal Board)
CMR 104	Working a seam in two sections or working contiguous seams.	DGMS
CMR 105 MMR 109	Workings approaching within 45 metres of railways etc.	DGMS
CMR 118 MMR 119	Permission to use a welding apparatus in a gassy mine.	JDMS
CMR 126 MMR 127	Workings extending within 15 m of surface water, reservoirs or streams.	DGMS
CMR 127 MMR 128	Workings extending within 60 m of old water logged workings.	DGMS
CMR 137(4a)	Installation of more than one auxiliary fans in the same ventiliating district in gassy and fiery mines.	JDMS

CMR 146(5)	Change in ventilation system of gassy mines.	JDMS
CMR 160(3) MMR 154	Provision of underground magazines	DGMS
CMR 175(6)	Use of delay detonators in gassy mines	DGMS
CMR 181(1)	Use of internal combustion engine or boilers underground.	DGMS

C. Bound paged books to be kept in the mine office

Reference of enactment	*Name of the book*
MA 23(3) : MR 76	Minor injuries causing 48 hours of enforced absence in Form J.
MA 48(1) : MR-77	Register of employees in Form B.
	Register of daily atteadance ; Form C for underground Form D for opencast and Form E for above ground.
MR 34	Record of whitewashing of urinals and latrines.
MR 44	List of First Aid persons incharge of first aid stations.
MR 49(4)	Compensatory days of rest in Form F.
MR 53	Leave account during the calendar year in Form G.
	Register of leave wages account during the calendar year in Form H.
MR 59	Overtime register in Form I.
CMR 36 MMR 39	Appointment of officials and competent persons.
CMR 41 MMR 44	Manager and Assistant Managers' diaries. Report to owner/agent for material supplies. Accident inquiry.
CMR 43 MMR 46	Overman's/foreman's daily report.
CMR 47 MMR 50	Hourly reading of speed & pressure of fan.
CMR 49 MMR 52	Surveyor's Record of workings approaching within 75 m of mine boundary, disused working, water logged area, etc.
CMR 63 MMR 65	List of plans, instruments, field books, etc.

CMR 70 MMR 72	Weekly examination of outlets by overman/foreman.
CMR 76 MMR 84	Particulars of man winding installations.
CMR 80 MMR 88	History of winding rope in use.
CMR 81 MMR 89	Examination and treatment of suspension gear.
CMR 83 MMR 91	Examination of winding equipment, daily, weekly, monthly and annual.
CMR 89	Examination of buffer, drawbars, coupling chain (14 days).
MMR 97	do do (Weekly).
CMR 91 MMR 99	History and examination of haulage rope.
CMR 93 MMR 101	Examination of haulage engine (daily) ; track, weekly.
CMR 94 MMR 102	Weekly examination of haulage and travelling roads by overman.
CMR 113 MMR 116	Daily Report of Sirdar/mate.
CMR 114 MMR 117	Withdrawal of workers in emergency.
CMR 117 MMR 119	Weekly examination of surface for fire-hazard by Sirdar/mate. Permission to light fire in quarry, or within 15 m of entrance.
CMR 119 MMR 120	Outbreak of fire in a mine ; subsequent steps.
CMR 120 MMR 121	Report of quarterly maintenance of fire extinguishers.
CMR 122 MMR 123	Weekly examination of fire-stoppings by overman.
CMR 123	Monthly report of analysis of mine road dust.
CMR 133	Particulars of fan stoppage and alteration. Weekly examination of fan.
CMR 136 MMR 135	Fortnightly exam. of airways, aircrossings, etc. Monthly exam. of airways, aircrossings, etc.
CMR 133 MMR 132	Fortnightly air measurement. Monthly do

CMR 145	Determination of methane percentage.
CMR 162 MMR 156	Issue and return of explosives.
CMR 169 MMR 163	Overhaul of exploders.
CMR 177 MMR 167	Report of undetected misfire.
CMR 179 MMR 169	Blaster's/Shotfirer's report book.
CMR 183 MMR 173	Report of maintenance and test of air receivers.
CMR 186 MMR 176	Weekly maintenance of all machinery.
CMR 187 MMR 177	Weekly examination of surface fences.

Maternity Act and Rules

Rule 3	Muster Roll—particulars of women workers in Form A.

Creche Rules

10	Particulars of children and their dates of birth, etc.

Coal Mines Labour Welfare Fund Rules

32	Register of monthly despatch of coke and coal in Form D.

Coal Mines Provident Fund Scheme

38A	Abstract attendance register in Form S.

Coal Mines Bonus Scheme

11	Employment below ground in Form III, opencast workings in Form IV and V. Not necessary if Forms, C, D, E of MR are being maintained.
	Bonus register in Form X.

Electricity Rules

116	Record of working of earth fault detectors and fault protective devices.
	Quarterly testing of fault detectors in Form XII.
131	Log book in Form XII.

D. Authorisations to be made by Owner/Agent/Manager

Reference	*Particulars of authorisation*
MA 17 : CMR 31 MMR 34	Mine Manager.

MA 21 : MR 41, 42	Incharge of first aid box/cupboard.
MR 43	Medical practitioner/compounder/cum-dresser for ambulance room.
MR 72	Welfare Officer/Additional Welfare Officers.
CMR 32 MMR 35	Asstt. Manager and under manager.
Circular	Safety officer
CMR 33 MMR 36	Engineer.
CMR 34 MMR 37	Overman/Foreman.
CMR 35 MMR 38	Surveyor.
CMR 52 MMR 55	Banksman/onsetter.
CMR 53 MMR 56	Haulage attendants.
CMR 54+95 MMR 57	Loco driver and helper.
CMR 53 & 87	Set rider.
CMR 71 MMR 78	Winding engine man.
CMR 79 MMR 87	Comp. person to inspect signalling appliances in a sinking shaft.
CMR 80 MMR 88	Engineer to examine, half-yearly, recapping of winding rope.
CMR 81 MMR 89	Comp. person to examine gauge of bell plate once every month.
	Comp. person to examine suspension gear half-yearly.
	Comp. person to examine weekly winding rope and engine.
CMR 83 MMR 91	Comp. person to examine Winding gear Competent person to examine Winding rope.
	Comp. person to examine winding engine yearly.
CMR 87 MMR 95	Signaller.
CMR 89 MMR 97	Comp. person to examine once fortnightly, buffer, couplings and drawbar.
CMR 91 MMR 99	Comp person to examine track and safety appliances weekly.

CMR 93 MMR 101	Comp. person to inspect haulage gear.
CMR 94 MMR 102	Overman for weekly examination of haulage and travelling roads.
CMR 96 MMR 104	Wagon shunters.
CMR 113 MMR 116	Mining Sirdars/mates. Chargeman incharge of sinking.
CMR 117	Sirdar for weekly examination of surface for fire precaution.
CMR 119 MMR 120	Overman for dealing with fire in a district.
CMR 120 MR 121	Comp. person to examine quarterly the fire extinguisher.
CMR 122 MMR 123	Overman to examine weekly the fire stoppings.
CMR 123 and Circular 34/59	Dust In-charge.
CMR 124 MMR 125	Sirdar to drill for draining out gas.
CMR 127 MMR 128	Sirdar on boring to drain out water when approaching old workings.
CMR 131 MMR 132	Fan Khalasi.
CMR 133	Comp. person to examine fan weekly.
CMR 136 MMR 135	Overman to examine airways, air crossings, etc.
CMR 141 MMR 140	Sirdar in-charge of dewatering and re-opening.
CMR 143 MMR 141	Overman to inspect unused workings, weekly.
CMR 145 MMR 144	Overman to determine gas % and environmental conditions.
CMR 147 MMR 143	Body searcher for contrabands.
CMR 148	In-charge of re-lighting stations.
CMR 149 MMR 145	Ventilation Officer.

CMR 155 MMR 149	Comp. person to clean and trim lamps.
	Comp. person to examine each lamp before it is taken underground.
	Comp. person to examine each lamp after it is brought out of mine.
	Manager/Asstt. Manager/Overman to examine each lamp, weekly.
CMR 156 MMR 150	Comp. person to maintain record of issue and return of lamps.
CMR 162 MMR 156	Magazine in-charge.
CMR 163 MMR 157	Explosive carrier.
CMR 166 MMR 160	Shot firer and assistant/Blaster.
CMR 169 MMR 163	Comp. person to overhaul exploders quarterly.
CMR 186 MMR 176	Comp. person to examine all machinery weekly.
CMR 187 MMR 177	Comp. person to inspect all surface fencing.
Circular 35/59	Sampling in-charge.

Pit-head bath Attendant (male and female)

Rule 8 *pit-head baths*

Creche Rule 9	Creche-in-charge.

Electricity Rules—

ER 121	Switch-gear operator.
ER 125	Operators of electrical apparatus including portable or transportable apparatus.
ER 131	Electrician.

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